



Minto Phase VI Preliminary Feasibility Study Technical Report



**Minto Mine
Yukon, Canada**

NAD 83; UTM Zone 8 coordinates 6,945,000 m N, 385,000 m E.

**Prepared by
Minto Explorations Ltd.
Effective Date: January 1, 2012**

Phase VI Preliminary Feasibility Report Minto Mine

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1. Summary

1.1. Introduction

Minto Explorations Ltd. (“MintoEx”), a wholly owned subsidiary of Capstone Mining Corp. (“Capstone”), owns (100%) and operates the Minto Mine; a 3,750 tonne per day (tpd) copper-gold mine approximately 240 km northwest of Whitehorse, Yukon. In 2011, the mine processed circa 1,258,308 tonnes of ore at a grade of 1.61% Cu, 0.68 g/t Au and 5.9 g/t of Ag.

After a successful exploration program in 2006, a preliminary feasibility study and technical report (2007 PFS) was completed by SRK for the Main and Area 2 deposits. Additional exploration since that time, increased resources by pursuing targets in the Ridgetop, Area 118, Minto North, and Minto East deposits that were eventually drilled to resource-quality levels. Additional drilling also expanded the Area 2 resource. In 2009 SRK issued the Phase IV Technical Report and in 2011 the Phase V PFS Technical Report, both of which increased reserves due to the successful exploration program.

The purpose of this technical report, the Phase VI Preliminary Feasibility study (“PFS”), is to build on the SRK 2010 Phase V PFS. This technical report includes the following modifications:

- New or updated mineral resource and/or reserve estimates including:
 - Minto Main;
 - Minto South Deposit (“MSD”). MSD is an amalgamation of the deposits formerly called Area 2/118, Copper Keel and Wildfire. These names continue to designate regions within the larger MSD model for detailed discussions but are not considered to be separate deposits; and
 - Minto East
- New life-of-mine plan that incorporates additional underground mineral reserves from the Copper Keel and Wildfire deposits;
- Updated cost and economic analysis estimates;
- Processing plant capacity improvements; and
- Exploration upside opportunities with recent discoveries at Fireweed A and Fireweed B plus Inferno.

There were no modifications to the Minto North and Ridgetop deposit models, however, pit designs or reserves have been slightly adjusted due to the new NSR parameters.

The Phase VI PFS Technical Report was compiled for MintoEx by Capstone Mining Corp. (“Capstone”) with contributions from SRK Consulting (Canada) Inc. (“SRK”) for block model validation as well as open-pit and underground geotechnical evaluations; Genivar Consulting (“Genivar”) for site infrastructure revisions and ventilation modelling; and Eggert Consulting (“Eggert”) for information related to metallurgy and mineral processing.

Exploration on the Minto property is ongoing and pursued on a seasonal basis; the 2012 diamond drilling program commenced and was completed subsequent to the effective date of

this report. This program is designed to more fully define and, potentially, expand the mineral resources, as well as to explore for additional new zones of mineralization.

Based on the results of the 2010 Phase V PFS, MintoEx applied to the Yukon government for an amendment to its Quartz Mining Licence (“QML”) in order to increase production from the Main deposit to an annual production rate of 1.368 million tonnes which was granted in May 2011. An application to amend the Water Use Licence (“WUL”) is currently being assessed by the Yukon Water Board. A further application to amend its Quartz Mining Licence is expected to be filed by MintoEx in 2013 that enables extraction of the incremental Phase V reserves. This application may also include the incremental increase in underground reserves that have been generated in this Phase VI Technical report.

Many sections of this report are referenced to the SRK “Minto Phase V Preliminary Feasibility Study Technical Report” dated March 2011. Material that hasn’t changed from the SRK report has been included in *italics* for clarity and new material has been added in regular fonts.

1.2. Geology and Exploration

The Minto Project is found in the north-northwest trending Carmacks Copper Belt along the eastern margin of the Yukon-Tanana Composite Terrain. The belt is host to several intrusion-related Cu-Au mineralized hydrothermal systems. The Minto Property and surrounding area are underlain by plutonic rocks of the Granite Mountain Batholith of Early Mesozoic Age. The component of the batholith represented on the Minto Property is the Minto pluton and is predominantly of granodiorite composition. Hypogene copper sulphide mineralization at Minto is hosted wholly within this pluton in sub-horizontal horizons of structurally prepared rock.

Five deposits of copper-gold-silver mineralization are reported in this document. Each of these deposits closely share a similar style of mineralization hosted by vertically stacked, shallow dipping deformation zones within the intrusion. The Main deposit is exposed in the dormant Main open pit mine and confirms the modelled deposit geometry; similarly, the sub-horizontal geometry of mineralized zones is currently partially exposed in the Minto South Deposit (Area 2 resource area) starter pit. Three other deposits have drill-delineated mineral resources and/or reserves interpreted on a similar sub-horizontal geological control but mineralization is not exposed.

For the purpose of this report the Area 2, Area 118, Copper Keel, and Wildfire resource sub-domains are now considered continuous to contiguous, and reported as one deposit, namely Minto South Deposit (MSD) located immediately south of Minto Main Deposit. As mentioned above the names are still retained for ease of discussion when discussing regions within MSD. The Ridgetop deposit is located just over 300 m south of the Area 2/118 resource sub-domain, the Minto North deposit is located about 700 m north of the Minto Main Deposit, the Minto East deposit is located about 200 m east of the south end of the Minto Main Deposit. Two very recent drill discoveries, named Fireweed A and Fireweed B are located approximately 200 m and 400 m respectively due east from Minto East. A working model suggests that the Fireweed zones are related to Minto East and are displaced on a series of normal faults. At the date of this report there was insufficient drilling to delineate any mineral resources at Fireweed thus

far. It remains a priority exploration target for 2012. All deposits and mineral prospects in the Minto area define a general north-northwest trend informally called the Priority Exploration Corridor or PEC. Deposit emplacement within this corridor is presumed to be structurally related, but at this point cannot be proven.

Copper sulphide mineralization is found in the rocks that have a structurally imposed fabric, ranging from a weak foliation through to a strongly developed gneissic banding in which the rock is partitioned into mafic and felsic bands. The contact relationship between the foliated deformation zones and the massive phases of granodiorite is generally very sharp. These contacts do not exhibit chilled margins and are considered by MintoEx geologists to be structural in nature, not intrusive, separating the variably strained equivalents of the same or similar rock type.

The more intense deformation zones form sub-horizontal horizons and can be traced laterally for more than 1,000 m in the drill core. They are often stacked in parallel to sub-parallel sequences and it is postulated that the foliated granodiorite horizons represent healed, shallowly dipping shear zones within the Granite Mountain Batholith; theorized to have formed when the rocks passed through the brittle/ductile transformation zone in the earth's crust in transition from a deep emplacement environment of the batholith to eventual exhumation. There is an on-going debate regarding the stratigraphic, intrusive, or structural nature of the zones hosting the foliation and mineralization. MintoEx have engaged the Mineral Deposits Research Unit ("MDRU") of the University of British Columbia to help understand the mineral paragenesis and deformation history. No other recognized deposit type compares directly with the Minto mineralization. While an Iron Oxide Copper Gold (IOCG) style for the Minto deposit cannot be unequivocally demonstrated, the authors are of the opinion that this style of deposit provides the most consistent model for the current level of understanding.

The primary hypogene sulphide mineralization consists predominantly of chalcopyrite plus bornite, $\pm$ chalcocite, and minor pyrite. Metallurgical testing also indicates the presence of covellite, although these sulphide species have never been positively logged macroscopically. It is not clear if this is a hypogene or secondary copper mineral.

Texturally, sulphide minerals predominantly occur as disseminations and foliaform stringers along foliation planes in the deformed granodiorite (i.e. sulphide stringers tend to follow the foliation planes). Occasionally, coarse free gold is observed associated with chloritic or epidote lined fractures that cross-cut the sulphide mineralization. The free gold may be due to presumed secondary enrichment during a later hydrothermal process overprinting the main copper sulphide-gold event. Sulphide mineralization is always accompanied by variable amounts of magnetite mineralization and strong biotite mineralization; the latter is interpreted as an alteration mineral. While these minerals occur in the non-deformed rocks they are present in the mineralized horizons in a much greater abundance in the range of an order of magnitude or greater than background.

Supergene mineralization occurs proximal to near-surface extension of the primary mineralization and beneath the Cretaceous Age conglomerate. Chalcocite is the prime mineral in these horizons along with secondary malachite, minor azurite and minor native copper.

Observations of foliated and even copper mineralized cobbles in drilling indicate that “Minto-type” mineralization was exposed, eroded and reincorporated in conglomerate sedimentary deposits by the Cretaceous Age. Other rock types, albeit volumetrically insignificant, include thin dykes (typically less than 1 m) of simple quartz-feldspar pegmatite, aplite, and an aphanitic textured intermediate composition rock.

Structural deformation includes the ore-bearing deformation zones, as well as folding present on the regional to micro-scale. Within the deformation zones the foliation exhibits highly variable orientations with the presence of small-scale (several centimetres in amplitude) folds. The ore-bearing zones are also occasionally folded on a scale of several hundred metres. The larger-scale folds appear to be gentle folds with north-south axial traces. Late brittle fracturing and faulting is noted throughout the property area, some of these faults have displacements significant enough to compartmentalize the deposits. For example, the Minto Creek fault bisects the Minto Main deposit, dividing it into north and south areas. The fault is modelled as dipping steeply north-northeast with an apparent left lateral reverse displacement. The DEF fault defines the northern end of the Main deposit. It strikes more or less east-west and dips north-northwest and cuts off the main zone mineralization. The boundary between the Area 2 and Area 118 ore zones is an intermediate NE dipping fault, and at least two parallel structures displace mineralized domains in Area 118. A similar NW striking fault zone appears to define the north-eastern boundary of the Ridgetop deposit, and defines the outcrop of Cretaceous conglomerate.

Pervasive, strong potassic alteration occurs within the flat lying zones of mineralization, and is the predominant alteration assemblage observed in all of the Minto Deposits. The potassic alteration assemblage is characterized by elevated biotite contents and minor secondary K-feldspar overgrowth on plagioclase relative to the more massive textured country rock. Additional alteration includes the replacement of mafic minerals by secondary chlorite, epidote, or sericite observed both in mineralized and waste rock interstitially or fracture/vein proximal, as well as variable degrees of hematization of feldspars. Minor carbonate overprint is occasionally observed associated with secondary biotite. Silicification is present but not pervasive in the Minto deposits.

Mineral exploration on the Minto property has been conducted intermittently since 1971. Subsequent to the discovery of the Minto Main deposit, which was the initial deposit in production, the adjacent southern half of the property has undergone systematic brownfields exploration. Exploration on the northern half is more sporadic.

There are currently more than 1,100 drill holes within a roughly 16 square kilometre (km²) area. As such, following up on open mineralized horizons in geological models, projecting mineralized horizons into areas of little or no drilling, and drilling near historical drill hole intercepts were the principal exploration tools employed by MintoEx and its geologists. Subsequent to Capstone’s predecessor, Sherwood Copper’s, acquisition of Minto Explorations Ltd. in June 2005, exploration from 2005 to 2011 has concentrated mostly on diamond drilling. However, an extensive historic soil sample survey and some ground based and airborne geophysics have been conducted and are very useful to guide drilling activity.

The current exploration approach by MintoEx is the systematic evaluation of modern electrical (chargeability); geophysical methods by commissioning various “proof-of-concept” surveys over known mineralization and then expanding survey coverage outward into untested areas using these methods that are calibrated to known deposits. An emphasis is placed on looking for signature analogs as opposed to being pedantic about precise measurements of response. The predominant electrical geophysical methods used are Gradient Array Induced Potential (GAIP), Dipole-Dipole Induced Potential, and Titan-24 DC Induced Potential. Drill targeting is predominantly based upon the coincidence of an anomaly in one of the electrical (chargeability) methods with an anomaly in the 1993 total field airborne magnetic survey (MAG).

Within the currently known extent of the Priority Exploration Corridor (“PEC”), future exploration programs will likely be more reliant solely on electrical / chargeability methods as the near-surface potential and discrete magnetic bull’s-eyes have largely been targeted. Magnetic data in areas located north of Minto North plus areas west and east respectively of the PEC may still be useful as these regions are still relatively under explored.

The current highest priority exploration targets are based on the evaluation of geophysics, soil geochemistry, geologic modelling, and diamond drilling. The targets identified as Ridgetop Southwest, MSD GAP, and the newly discovered Fireweed prospect are all located within a 2 km by 2 km area, south of the DEF fault. South of the DEF, all of the priority chargeability anomalies within 350 metres of the surface have been drill tested and it is believed that the various electrical geophysical methods employed to date are ineffective below -350 metres from surface. The recent discovery at Fireweed was geologically driven from modelling MSD.

MintoEx also sees good exploration potential in the area north of the DEF fault, as evidenced by the discovery of the high grade Minto North deposit early in 2009 and the Inferno prospect discovered in late 2010; however, past drilling in this area is sparse and new targeting is driven largely by geophysics.

In 2009, several other historic bedrock copper occurrences discovered in the 1970s north of the DEF fault were relocated and confirmed. In addition various copper-in-soil geochemical anomalies, often coincident with magnetic geophysical anomalies, occur throughout the property and many of them remain untested. However, further understanding of the bedrock geology north of the DEF fault is required before many of these targets can be properly assessed and placed in perspective.

1.3. Mineral Resources

A primary objective was to produce a revised mineral resource evaluation for the Minto South Deposit (MSD). Mineral extraction of the Minto Main resource was completed as of April 2011. The Minto North and Minto East deposits, other integral parts of the Minto system, were evaluated but unchanged from Phase V.

The mineral resource estimate reported in this report is based on earlier mineral resource estimates presented in the Phase V Technical Report, the May 30, 2011 mineral resource news release and newly acquired results.

The Minto North and East deposit resource estimates were completed by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd., an independent qualified person as this term is defined in National Instrument 43-101. These resource estimates were completed as part of the Phase V Technical Report and as stated previously have not changed.

The MSD was defined as part of the May 30, 2011 news release. The news released stated that MSD covered previously estimated areas Area 2/118 and Ridgetop deposit. MSD also includes new resource estimates for the Wildfire and Copper Keel areas. The mineral resource estimate of MSD was completed by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an employee of SRK and an independent qualified person as this term is defined in National Instrument 43-101. The effective date of this resource estimate is September 29, 2011. Marek Nowak, P.Eng, also with SRK, analyzed the data, reviewed, and validated the mineral resource estimates.

In the opinion of SRK and Capstone, the block model resource estimate and resource classification reported herein provide a reasonable representation of the mineral resources at the Minto South Deposit, Ridgetop, Minto North and Minto East deposits at the current level of sampling. The mineral resources presented herein were estimated in conformity with generally accepted CIM *“Estimation of Mineral Resource and Mineral Reserves Best Practices”* guidelines and are reported in accordance with the Canadian Securities Administrators’ National Instrument 43-101. **Mineral resources are not mineral reserves and do not have demonstrated economic viability. Only Measured and Indicated mineral resources have been used in the preliminary feasibility study described in this report.**

SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the mineralized domains and that the assaying data is sufficiently reliable to support estimating mineral resources.

The “reasonable prospects for economic extraction” requirement for a mineral resource generally implies that the quantity and grade estimates meet certain economic thresholds, and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. Capstone considers that the Ridgetop and Minto North resource areas are amenable to open pit extraction. The Area 2/118 and Wildfire resource areas are amendable to both open pit and underground extraction while the East and Copper Keel resource areas are suitable for underground mining.

In order to demonstrate the reasonable prospect of economic extraction, SRK compared the overall mineral resource with an economic shell created with Whittle™ pit optimization software using the NSR model and parameters as shown in Table 1-1. The entire estimated resource is located within the shell except for local minor extensions considered mineable by underground mining techniques.

Table 1-1: Whittle™ Optimization Parameters for MSD Resource Estimate Comparison in Order to Demonstrate Reasonable Prospects of Economic Extraction

Constraining Parameter	Unit	Value
Copper Price	US\$/lb	3.50
	C\$/lb	3.85
Gold Price	US\$/oz	1300
	C\$/oz	1430
Silver Price	US\$/oz	16.00
	C\$/oz	17.60
Exchange Rate	C\$: US\$	1.11
Mining Cost	C\$/t mined	1.50
Processing and G&A Cost	C\$/t milled	5.00
Royalty	%	0.50
Slope angles	degrees (overall)	50

The mineral resource statement for the MSD is presented in Table 1-2.

Table 1-2: Mineral Resource Statement at 0.5% Cu Cut-off for Minto South, SRK Consulting September 29, 2011

Classification	Tonnes (000's)	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)	Contained Gold (000's oz)	Contained Silver (000's oz)
Measured (M)	10,387	1.21	0.45	4.07	278,028	151	1,360
Indicated (I)	28,556	0.98	0.32	3.44	616,152	290	3,162
Sub-total (M+I)	38,944	1.04	0.38	3.72	894,180	441	4,521
Inferred	8,134	0.81	0.24	2.93	145,718	63	767

The mineral resource statement for Ridgetop, Minto North and Minto East are presented in Table 1-3, Table 1-4, and Table 1-5 respectively.

Table 1-3: Mineral Resource Statement at 0.5% Cu Cut-off for Ridgetop, SRK Consulting August 30, 2010

Classification	Tonnes (000's)	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)	Contained Gold (000's oz)	Contained Silver (000's oz)
Measured (M)	1,531	0.98	0.25	2.14	33,204	12.3	105
Indicated (I)	3,534	0.87	0.30	2.87	67,901	33.8	326
Sub-total (M+I)	5,064	0.91	0.28	2.65	101,104	46.2	431
Inferred	318	0.75	0.13	1.57	5,250	1.3	16

Table 1-4: Mineral Resource Statement at 0.5% Cu Cut-off for Minto North, Kirkham Geosystems December 1, 2009

Classification	Tonnes (000's)	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)	Contained Gold (000's oz)	Contained Silver (000's oz)
Measured (M)	1,844	2.15	1.11	7.7	87,530	66	456
Indicated (I)	264	1.04	0.6	5.76	6,055	5	49
Sub-total (M+I)	2,108	2.01	1.04	7.46	93,585	71	505
Additional Inferred	25	0.84	0.40	4.4	457	0	3

Table 1-5: Mineral Resource Statement at 0.5% Cu Cut-off for Minto East, Kirkham Geosystems December 15, 2010

Classification	Tonnes (000's)	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)	Contained Gold (000's oz)	Contained Silver (000's oz)
Measured (M)	688	2.30	1.07	6.3	34,842	24	139
Indicated (I)	489	1.74	0.70	4.6	18,805	11	72
Sub-total (M+I)	1,177	2.07	0.92	5.57	53,647	35	211
Additional Inferred	14	1.03	0.45	2.8	316	0	1

Table 1-6 presents combined mineral resources at a 0.5% Cu cut-off for Minto South, Ridgetop, Minto Main, Minto North and Minto East Deposits. The Minto Main deposit resource has been

appropriately reduced to account for all material removed by mining up until December 31, 2011.

Table 1-6: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Minto South, Ridgetop, Minto Main, Minto North and Minto East Deposits, January 1, 2012

Classification	Tonnes (Kt)	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs)	Contained Gold (000's oz)	Contained Silver (000's oz)
Measured (M)	14,828	1.35	0.53	4.43	441,744	254	2,114
Indicated (I)	33,103	0.98	0.32	3.42	714,010	341	3,643
Sub-total (M+I)	47,931	1.09	0.39	3.73	1,155,754	595	5,758
Additional Inferred	8,493	0.81	0.24	2.88	151,764	65	788

1.4. Mine Production and Mineral Reserve Estimation

The Minto South, Ridgetop, Minto East and North deposits are planned to be developed by open pit ("OP") and underground ("UG") methods. Open Pit mining in the Phase VI pre-feasibility study assumes contractor mining rather than the assumption of Owner Operated in the Phase V pre-feasibility study. In addition, the initial underground mining (portal and decline development as well as vertical ventilation raise development) is by contractor rather than owner operated as was indicated in the Phase V pre-feasibility study.

The planning for this pre-feasibility study assumes a start date of January 1, 2012. Mining in the Main Pit was completed April 2011 and will be incorporated into this pre-feasibility study as an in-pit tailings deposition storage location. Development of the Area 2 Open Pit within the MSD is currently underway.

Based on a start date of January 1, 2012, the Phase VI open pit and underground mines will produce a total of 13.5 million tonnes ("Mt") of ore and 44.48 Mt of waste over approximately a 7 year mine operating life ending in 2019. Approximately 4.36 Mt of ore is planned to be produced from UG mining at a rate of 2,000 tpd. Once active mining is completed, mill operations will continue for an additional 3 years, processing an accumulated 4.08 Mt of stockpiled ore. Therefore, the total mill operating life is 10 years with final reclamation commencing mid-2022.

The life-of-mine plan ("LOMP") accelerates mining and processing of high-grade ore and defers stockpiled low grade mill feed until the end of the mine life whenever possible.

The Phase V pit designs and subsequently the open pit reserves have been revised in the Phase VI Preliminary Feasibility study to reflect the LOMP operating costs, metal prices and the exploration results contained with the MSD ore deposit model. Table 1-7 reviews the Whittle™ Parameters utilized to generate the pit economic shells. These shells were used to develop optimized pit designs by Minto personnel.

Mine design for the Phase VI underground expansion relies on the evaluation of a Net Smelter Return (“NSR”) model that includes estimates of metal prices (US\$2.50/lb Cu), exchange rate, mining dilution, mill recovery, concentrate grade smelting and refining payables and costs, freight and marketing costs and royalties. The NSR model was based on a 10 m northing x 10 m easting x 3 m elevation block size for Phase VI (see Table 1-8).

Table 1-7: Summary of Whittle™ Parameters used for Phase VI Open Pit Design

Item	Unit	Value
<i>Metal Prices and Exchange Rate</i>		
Copper	US\$/lb	2.50
Gold*	US\$/oz	300.00
Silver*	US\$/oz	3.90
Exchange rate	US\$/C\$	0.90
<i>Processing</i>		
Copper recovery to concentrate	Max	91%
Sulphide Gold recovery to concentrate	Max	70%
Sulphide Silver recovery to concentrate	Max	78%
Copper grade in concentrate	%	40
Gold grade in concentrate	g/t	variable with Cu
Silver grade in concentrate	g/t	variable with Cu
Concentrate moisture content		8%
<i>Smelter Payables</i>		
Payable copper in concentrate	%	96.75
Payable gold in concentrate	%	per MRI contract
Silver deduction	g/t in conc	per MRI contract
Remaining payable silver in concentrate	%	per MRI contract
<i>Other Parameters</i>		
Pit slope angles	Overall	As per Phase V PFS
Dilution	%	6%

Item	Unit	Value
Mining recovery	%	100
Annual Plant Throughput	Mtpa	1,370,000
Costs		
Waste mining cost	C\$/waste tonne	3.12
Ore mining cost	C\$/ore tonne	3.12
Processing cost	C\$/milled tonne	16.50
G&A cost	C\$/milled tonne	11.90
Royalties	%	1.00%
Transport, marketing, ins, etc.	US\$/dmt conc	169.54

*Base on terms of royalty stream agreement with Silver Wheaton

In summary, the open pit reserves (excluding the completed main pit) changed from 8.18Mt in Phase V to 9.18Mt in Phase VI, an increase of 24Mlbs contained Cu metal.

The Phase V Underground Mineral Reserves in the areas of Minto East, Area 2 and Area 118 were checked with the revised NSR model (see Table 1-8) and remain economically viable. Therefore, these reserves were included in the Phase VI reserves without revision.

The additional Phase VI underground reserves (Copper Keel and Wildfire) discussed in this technical report relied on an NSR model that was based on the following parameters:

Table 1-8: Underground Net Smelter Return Parameters

Parameter	Unit	Value
Metal Prices and Exchange Rate		
Cu price	US\$/lb Cu	2.50
Au price	US\$/oz Au	300
Ag price	US\$/oz Ag	3.9
Exchange rate	US\$/C\$	0.90
Processing		
Cu	%	91
Au	%	70
Ag	%	78

Parameter	Unit	Value
<i>Cu Concentrate Grade</i>		
Cu	%	40
Au	g/t	Variable
Ag	g/t	Variable
Moisture content	%	8
<i>Smelter Payables</i>		
Cu in Cu conc	%	96.75
Au in all cons	%	per MRI contract
Au deduction in all cons	g/t in conc	per MRI contract
Ag in all cons	%	per MRI contract
Ag deduction in all cons	g/t in conc	per MRI contract
<i>Offsite costs</i>		
Cu conc treatment	US\$/dmt conc	per MRI contract
Cu refining charge	US\$/lb pay Cu	per MRI contract
Au refining charge	US\$/oz pay Au	per MRI contract
Ag refining charge	US\$/oz pay Ag	per MRI contract
Transport, marketing, ins, etc.	US\$/dmt conc	169.54
Royalties	%	1.00
<i>Operating Costs</i>		
UG mining cost	C\$/ore tonne	36.00
Processing cost	C\$/milled tonne	16.50
G&A cost	C\$/milled tonne	11.90
Total Site Costs	C\$/milled tonne	64.40
Internal Dilution	%	5

Various small underground deposits were not converted to a Reserve if they required excessive development. The total mineral reserve estimate is summarized in Table 1-9 below.

Table 1-9: Minto – Mineral Reserves by Class for Phase VI

	Mineral Reserve Class	Tonnes	Diluted Grade			Contained Metal		
		(Kt)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit Reserves								
Minto North	Proven	1,596	2.26	1.21	8.12	79	62	417
	Probable	9	1.68	0.58	6.92	0	0	2
	Sub-total	1,604	2.26	1.21	8.12	80	62	419
Ridgetop	Proven	1,073	1.02	0.25	2.12	24	9	73
	Probable	1,020	1.00	0.28	2.97	22	9	97
	Sub-total	2,093	1.01	0.26	2.54	46	18	171
MSD – 118	Proven	-	-	-	-	-	-	-
	Probable	483	1.28	0.10	1.81	14	2	28
	Sub-total	483	1.28	0.10	1.81	14	2	28
MSD - Area 2	Proven	3,306	1.44	0.54	4.99	105	57	530
	Probable	1,694	1.00	0.27	3.20	37	15	174
	Sub-total	5,000	1.29	0.45	4.38	142	72	704
Open Pit Subtotal	Proven	5,974	1.58	0.67	5.31	208	128	1,020
	Probable	3,204	1.04	0.25	2.92	74	25	301
	Sub-total	9,179	1.39	0.52	4.48	282	153	1,321
Underground Reserves								
Minto East	Proven	-	-	-	-	-	-	-
	Probable	709	2.28	1.04	6.15	36	24	140
	Sub-total	709	2.28	1.04	6.15	36	24	140
MSD - Area 2 / 118	Proven	-	-	-	-	-	-	-
	Probable	1,731	1.76	0.74	7.19	67	41	400
	Sub-total	1,731	1.76	0.74	7.19	67	41	400

	Mineral Reserve Class	Tonnes (Kt)	Diluted Grade			Contained Metal		
			Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlbs)	Au (koz)	Ag (koz)
MSD - Copper Keel	Proven	106	1.74	0.61	6.3	4	2	22
	Probable	1,455	1.81	0.65	6.7	58	30	313
	Sub-total	1,561	1.81	0.64	6.67	62	32	335
MSD - Wildfire	Proven	301	1.8	0.77	6.06	12	7	59
	Probable	59	1.59	1.00	7.85	2	2	15
	Sub-total	360	1.76	0.80	6.35	14	9	73
Underground Subtotal	Proven	407	1.78	0.73	6.12	16	9	81
	Probable	3,954	1.87	0.76	6.83	163	97	868
	Sub-total	4,361	1.86	0.76	6.77	179	106	948
Stockpiles	Proven	852	1.23	0.41	3.71	23	11	102
	Probable							
	Subtotal	852	1.23	0.41	3.71	23	11	102
Total								
Total Reserves	Proven	7,233	1.55	0.64	5.17	247	148	1,203
	Probable	7,158	1.50	0.53	5.08	237	122	1,169
	Total	14,392	1.53	0.58	5.12	484	270	2,371

Area 2 Stage 3 and 4 in the Phase V PFS is combined into one stage; Area 2 – Stage 3.

The Phase VI surface mining sequence commences with ore production from Area 2 – Stages 1 & 2 followed by mining of 118, then Minto North, Area 2 – Stage 3 and finally the two Ridgetop pits: North and South. Underground mining operations will begin with production from the 118 zone while a ramp continues towards the Minto East deposit. Minto East, 118 and Area 2 underground reserves will then provide stable production as a new ramp develops towards Copper Keel. In 2018, a new portal will be developed to extract ore from the Wildfire deposit, which is the last deposit to be mined whereupon all mill feed will be provided by stockpiles.

The average rate of production for the underground deposits continued to be established at 2,000 tpd. The underground reserves are located in numerous “pods” which would require a high “vertical drawdown rate” above industry standards if the mining rate were increased. Low grade ore production from the surface pits and stockpiles was deferred and utilized when required as supplemental feed to attain the overall processing rate of 3,750 tpd. The stages continue to be designed to provide the required ore per period and defer stripping waste as

long as possible. There is an opportunity that UG production could be increased if development were increased. This would be investigated during the preparation of the annual Life of Mine Plan. The current Life of Mine production schedule is shown in Table 1-10 with the processing schedule summarized in Table 1-11.

Table 1-10: Phase VI LOM Mine Production Schedule

Parameter	Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019
Total OP and UG										
Total Waste	Kt	44,484	9,948	10,302	10,011	7,310	4,917	1,870	125	-
ROM ore	Kt	13,539	2,056	2,456	2,430	2,394	1,948	1,200	696	360
Open Pit Stripping Ratio	Waste:Ore	4.75	4.80	4.9	5.8	4.3	4.0	3.7	-	-
Cu Grade	%Cu	1.54	1.29	1.51	2.07	1.30	1.35	1.55	1.83	1.76
Au Grade	g/t	0.60	0.45	0.50	1.03	0.43	0.46	0.61	0.63	0.80
Ag Grade	g/t	5.18	4.35	4.81	7.38	4.31	3.65	5.93	6.66	6.35
Total Mined Cu	Mlbs Cu	460	59	82	111	69	58	41	28	14
Total Mined Au	koz Au	259	30	39	81	33	29	24	14	9
Total Mined Ag	koz Ag	2,256	288	380	576	332	228	229	149	73
ROM ore	Kt/day	-	5.63	6.73	6.66	6.56	5.34	3.29	1.91	0.99
Total Material	Kt/day	-	32.89	34.95	34.08	26.59	18.81	8.41	2.25	0.99

Table 1-11: Phase VI LOM Process Production Schedule

			Y E A R										
Parameter	UNIT	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Mill Feed Rate	dmt/day	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750
Mill Feed Total	Mt	14,393	1,373	1,369	1,369	1,369	1,369	1,369	1,369	1,369	1,369	1,369	654
Feed Grade	Cu %	1.53	1.50	1.88	2.29	2.19	1.74	1.70	1.70	1.17	0.74	0.74	0.74
	Au g/t	0.58	0.56	0.69	1.15	1.01	0.65	0.66	0.60	0.41	0.16	0.16	0.16
	Ag g/t	5.10	5.18	6.42	8.23	7.89	5.16	6.30	5.90	3.68	1.89	1.89	1.89
Recovery to Conc.	Cu	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	88%
	Au	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
	Ag	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%
Conc. Grade	% Cu	-	37.0	38.4	41.8	41.2	37.6	37.4	37.4	35.5	32.9	32.9	32.9
Conc. Production	dmt	532,689	51,487	61,600	68,920	66,993	58,332	57,343	57,271	41,340	28,294	28,216	12,894
Conc. Metal	Mlb Cu	444	42	52	63	61	48	47	47	32	21	20	9
	oz Au	188,826	17,311	21,122	35,524	31,230	20,112	20,479	18,449	12,509	4,888	4,875	2,329
	oz Ag	1,835,613	182,790	220,340	282,351	270,763	177,640	216,405	202,366	126,422	65,127	64,949	31,029

1.5. Waste Management and In-pit Tailings Disposal

Mill tailings are currently deposited on the existing permitted dry-stack location. In August 2010, MintoEx submitted the permit applications (Phase IV) required to enable deposition of mill thickened tailings into the Main Pit. The permit is expected to be approved in Q3 2012. The Main Pit will not be filled completely with tailings, i.e, 700,000 cubic metres (m³) will be available for the storage of freshet flow waters that allows treatment of this water to take place, if required, before discharge from the site.

Tailings will then be deposited into the Area 2 pit and Ridgetop North pit. Additional tailings storage capacity is available in the underground workings if required. This deposition of tailings into Area 2 and Ridgetop North will form part of the Phase V/VI permit application that is to be submitted Q4 2012 and is assumed to be approved in 2013.

During 2011, construction of the Main Pit Buttress, designed to inhibit movement in the Main Pit south wall, commenced with material excavated from the Area 2 pit. In addition, haulage of Area 2 waste rock to the permitted Mill Valley dump located to the east of the existing mineral processing plant commenced. Waste rock associated with first two stages of Area 2 pit will be hauled to the South West Valley dump.

Waste rock associated with the Minto North open pit will be hauled to the Main dump.

Waste rock from the Ridgetop South Pit will be deposited on the perimeter of the pit, the Ridgetop North Pit will then be deposited in the excavated Ridgetop South pit.

A portion of waste rock from the Area 2 pit, Stage 3, will be deposited in the Stage 2 pit, enabling short-haul waste dumps.

Waste rock from the underground development program will be hauled to the excavated stopes with an underground haulage truck that is equipped with an ejector box that enables dumping into the stopes without damaging backs. A portion of the waste rock will also be hauled to surface.

All overburden is hauled to temporary dumps for reclamation purposes or to the existing overburden dump.

Therefore, the comment noted in the Phase V PFS by SRK remains valid in the Phase VI PFS, *“Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits, reduces the amount of disturbed land that would normally be required by mining of the Phase V deposits, and provides a significant cost savings over the current dry-stack method.”* (SRK 2011).

1.6. Metallurgical Test Work

Metallurgical testing, by Eggert Engineering (“Eggert”) and MintoEx during 2011, was performed on the Wildfire and Copper Keel zones. The metallurgical testing is a continuation of the work performed by G&T Metallurgical Services LTD (“G&T”) in 2010.

The main objective of the test program was to determine the plant performance of the Copper Keel and Wildfire zones. The test work results have been reported and are considered when evaluating the process plant design.

The Copper Keel and Wildfire zones have been included in the most recent mine plan.

1.7. Process Plant

The process plant at Minto has been operating since 2007 and has undergone several significant upgrades that have increased throughput and recoveries. Recent historical operating results and testing support the projected plant operating parameters of:

- 3,750 dry tonnes per day for 2012 and beyond;
- Design availability of 91.3%, being 7,997 operating hours per year, with standby equipment in critical areas; and
- Sufficient plant design flexibility for treatment of all ore types as per test work completed at design throughput.

1.8. Environmental Assessment and Licensing

In the Yukon, mining projects require an environmental assessment prior to the issuance of significant operating licenses for mining, including a Type A Water Use License and a Quartz Mining Licence. The Phase IV expansion (2010) has been assessed under the Yukon Environmental and Socioeconomic Assessment Board (YESAB) beginning in August 2010. The Phase V project application is being formulated and will be submitted to YESAB this year (2012). The licensing process is partly complete with the QML issued and the Type A Water Use Licensing process underway with a decision pending in Q3 2012.

“The major instruments or authorizations permitting and governing operations for the project include a Type A Water Use Licence, issued by the Yukon Water Board and a Quartz Mining Licence issued by Yukon Government, Energy Mines and Resources” (SRK 2011).

Selkirk First Nation

“MintoEx claims lie within Selkirk First Nation (SFN) Category A Settlement Lands (Parcel R-6A), where both surface and mineral rights are reserved for SFN. In the event that Minto Exploration claims are allowed to lapse, they cannot be re-staked, and the surface and mineral rights would revert to the SFN.

On September 16, 1997, the company and the SFN entered a Cooperation Agreement concerning the Minto Project with respect to the development of the Minto Mine. This agreement was amended (November 4, 2009). In addition to establishing terms and conditions related to cooperation on the project, this confidential document deals with other economic and social measures and communication between Selkirk First Nation and the company. This agreement will continue to guide SFN involvement in the project as mine expansion planning and development proceeds” (SRK 2011).

Environmental Conditions

The Phase V expansion requires an update of the baseline environmental conditions of the drainage to the north of the Minto Creek drainage. The Minto North deposit is located approximately 100 m into the drainage which to date has not been disturbed by MintoEx mining activities. There is potential for effects to the aquatic environment downstream of the Minto North open pit. These risks are being evaluated as part of a Phase V / Phase VI project application that will be submitted to YESAB for review. An updated Environmental Conditions report has been completed that will support the Phase V development which updated all environmental data for the project area and is being used for the assessment and permitting processes that are projected to be approved in 2013.

The Phase VI expansion consists entirely of underground mining with minimal impact to the existing Environmental Conditions.

Water Management and Effluent Discharge

MintoEx, in its original water licence application submitted in 1996, outlined a water management plan based on the limited baseline information and project projections available for the Minto Mine at the time. In the intervening period since the application, screening and issuance of the Type A water use licence, significant additional baseline and operational data have been collected. These data formed the basis of a new Water Management Plan that has been screened through the YESAB process and subsequently licensed in 2011. This plan includes the construction and use of storm water diversions, a water treatment plant and revised project effluent discharge standards. Some aspects of the current water management plan will be revisited as part of development of the project application to YESAB related to Phase V including effluent standards currently in place at a compliance point in the receiving environment.

"The critical consideration with respect to water management planning for Phase V will be contingency runoff storage requiring treatment prior to discharge and ensuring that effects to McGinty Creek from the mining of the Minto North deposit are minimized and fully mitigated. Water treatment will continue to be a critical component of the water management strategy into the Phase V expansion, as it is in the currently proposed water management plan" (SRK 2011).

Metal Leaching / Acid Rock Drainage

The waste rock associated with the development of the Copper Keel and Wildfire reserves has the same characteristics as that noted in the exploration results from the Phase V Underground development program.

As noted in the SRK Phase V PFS, *"Characterization of mine rock and tailings from the Area 2/118, Ridgetop, and Minto North deposits has shown that there is sufficient neutralization potential (NP) to offset the acid potential (AP) within the waste materials. Both bulk mine rock and tailings had NP/AP >3 and the majority of mineralized rock samples tested also had NP/AP > 3.*

A small proportion of the mineralized waste has lower NP/AP values (a single sample had NP/AP < 1) indicating that localized pockets of potentially acid generating rock do exist. Overall, however, the Phase V characterization results indicate that waste management planning does not need to take prevention of acid rock drainage (ARD) into consideration.

Bulk mine rock has elemental concentrations typical of granitic rocks, therefore metal leaching from bulk waste is not expected to be environmentally significant. Mineralized waste has elevated concentrations of copper and other trace elements. Segregation of mineralized waste with elevated copper and disposal in a way that limits copper leaching (e.g. co-disposal with in-pit tailings) will be required to minimize loadings to the receiving environment over the long term". (SRK 2011).

Closure Planning

Closure measures for the site following the completion of the Phase VI Life of Mine Plan that includes the additional underground reserves from the Copper Keel and Wildfire deposits are expected to follow those currently authorized. There is minimal additional reclamation required due to the additional underground reserves and minimal change to the overall footprint of the site, i.e., two additional surface ventilation fan for the Copper Keel and Wildfire zones and a portal to access the Wildfire deposits.

It is noted that the existing Quartz Mining Licence requires a closure plan to be submitted at regular intervals (typically two years) with the most recent version having been approved on September 9, 2011. Closure plans must be updated as part of applications to YESAB as well.

1.9. Operating Costs

Table 1-12 presents a summary of the operating costs by major area. General, camp and administration costs are lower than reserve cost estimate due to the cost reductions after open pit and underground mining ceases. Costs have been generated using a combination of historical performance, existing contracts and first principles.

Table 1-12: Average LOM Unit Operating Costs by Major Area

Area	Unit	Cost Estimate
Open Pit Mining	\$/t mined	3.54
	\$/t milled	20.37
Underground Mining	\$/t milled	35.95
Total Mining (weighted average)	\$/t milled	24.68
Processing	\$/t milled	17.20
General, Camp and Administration	\$/t milled	10.81
Total	\$/t milled	52.69

1.10. Capital Costs

Table 1-13 reviews capital expenditures without closure costs. Costs have been generated using a combination of vendor quotes, operational budgets and first principles. The major decrease in CAPEX versus the Phase V Pre-Feasibility study was due to the continued utilization of contractor for open pit mining rather than owner operated. Therefore, purchase of the Open Pit mobile equipment fleet (Cdn \$32 million) was avoided; however, additional capital costs for the underground has increased the total capital spent at the site. A closure cost allowance of \$25M continued to be used in the cash flow analysis. In addition, a sustaining capital allowance of \$1.1M annually between 2017 and 2021 was also used in the cash flow analysis.

Table 1-13: Capital Costs over LOM by Major Area

Area	Unit	Cost Estimate
Open Pit Mining	\$M	2.5
UG equipment (fixed and mobile)	\$M	33.4
UG development	\$M	34.8
Process plant	\$M	10.3
Site Infrastructure	\$M	18.7
Permitting	\$M	3.3
TOTAL CAPITAL COST	\$M	102.7

1.11. Economics

The estimated economic benefit of mining the Minto Phase VI underground deposits is the incremental capital associated with development of the Copper Keel and Wildfire zones of MSD with revenues generated by the ore extraction from those deposits.

Phase VI economic model parameters are based on a long-term \$3.25 / lb copper price and an exchange rate at 1:1 USD:CAD. Gold and silver prices were fixed at \$300/oz and \$3.90/oz respectively as per Capstone's forward sales agreement.

Table 1-14 summarizes the economic and production assumptions utilized for each case.

Table 1-14: Phase VI Economic Model

Item	Unit	Value
Waste mined	Mt	44,484
Ore mined	Mt	13,539
Total mined	Mt	58,023
Mill Feed*	Kt	14,393
Copper millhead grade	% Cu	1.53
Gold millhead grade	g/t Au	0.58

Item	Unit	Value
Silver millhead grade	g/t Ag	5.1
Copper in cons	Mlb	444
Gold in cons	oz	188,826
Silver in cons	oz	1,835,613
Concentrate Grade	% Cu	Variable with Mill Feed
Copper Price	US\$/lb	3.25
Gold price (inc. hedging)	US\$/oz	300
Silver price (inc. hedging)	US\$/oz	3.90
Exchange rate	US\$/C\$	1:1
Unit Total OPEX	C\$/t milled	52.69
Unit On-site OPEX	C\$/lb Cu payable	1.76
Unit Off-site OPEX	US\$/lb Cu payable	0.30
Unit By-product Credit	US\$/lb Cu payable	(0.14)
Unit OPEX net by-product credits	US\$/lb Cu payable	1.92
Total Capital	\$M	103
Allowance for closure cost	\$M	25

*Note Mill Feed includes Ore Stockpile

The following table summarizes the economic key performance indicators:

Table 1-15: Phase VI Economic Results

Before-Tax Cash Flow (\$M)	After-Tax Cash Flow (\$M)	Before-Tax NPV _{8.0%} (\$M)	After-Tax NPV _{8.0%} (\$M)	After-Tax IRR
429	290	278	192	184%

Figures 1-1 and 1.2 provide the non-discounted After-Tax Cumulative Cash Flows and After-Tax Annual Cash Flows respectively with a maximum exposure ratio of 16 and an annual cash flow ratio of 3.

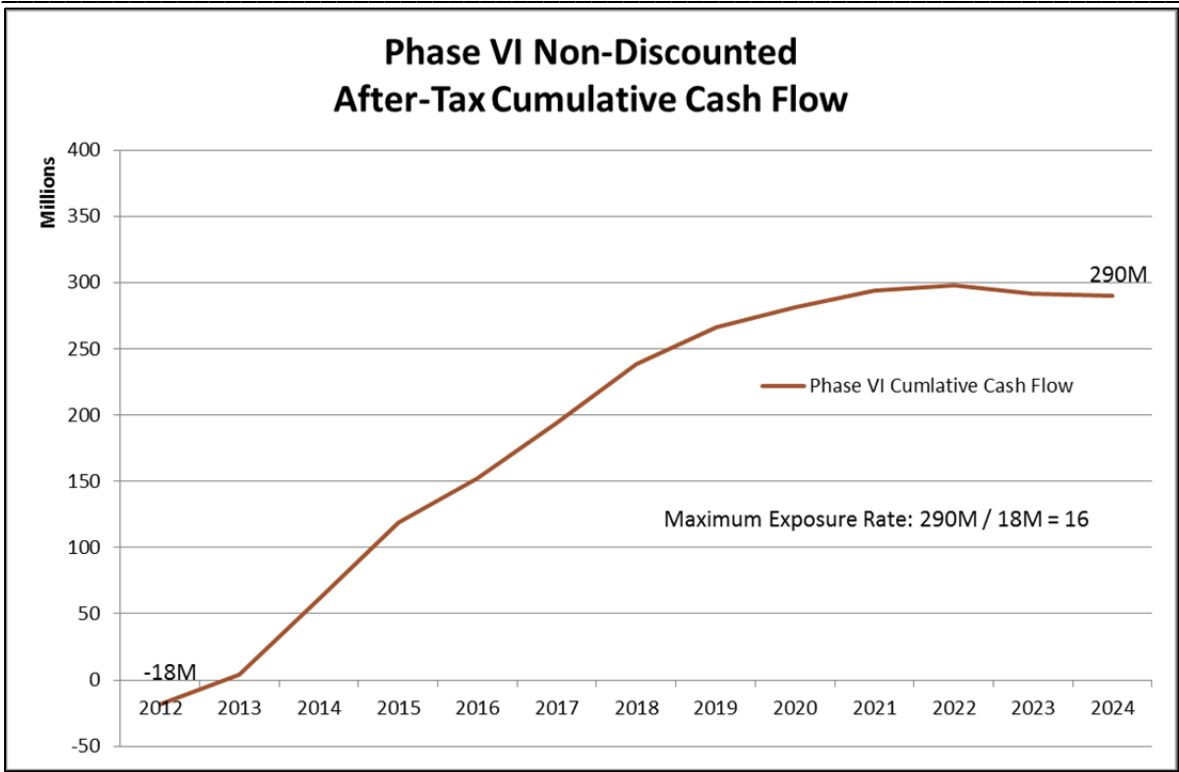


Figure 1—1: After Tax Cumulative Cash Flow

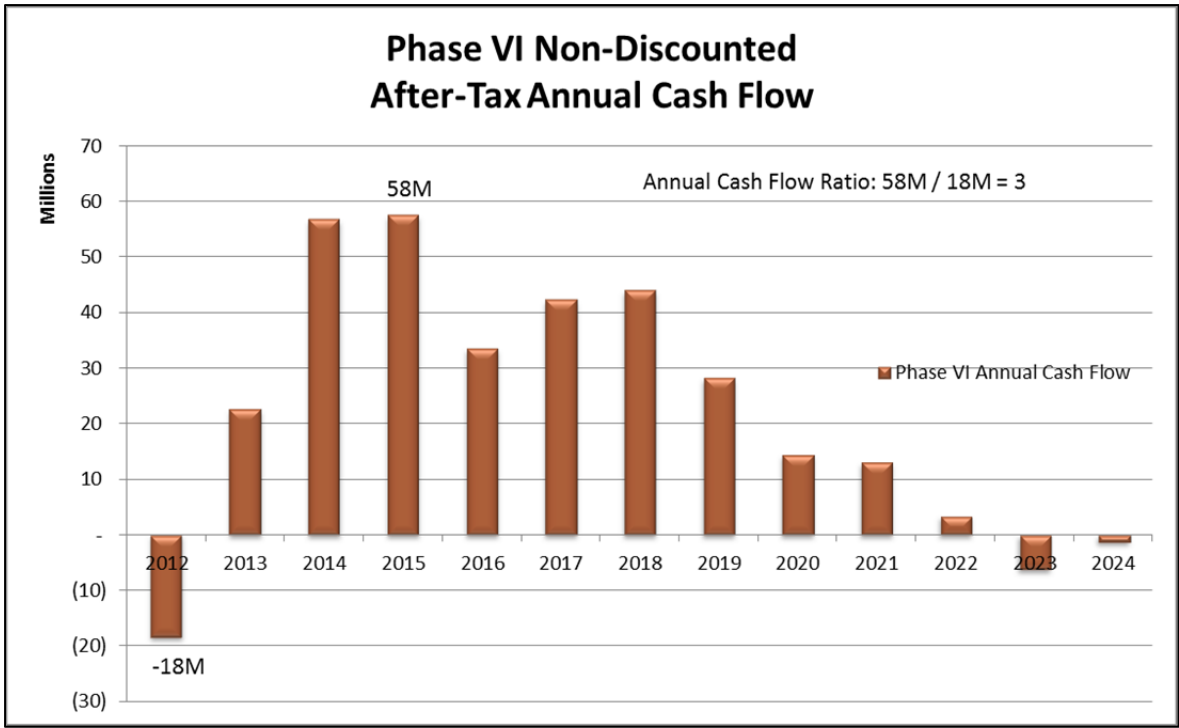


Figure 1—2: After-Tax Annual Cash Flow

The break-even copper price for the project (excluding capital) is US\$1.92/lb.

The Phase VI economic analysis has been compared to the Minto Mine January 2012 LOMP. Phase VI increases the undiscounted pre-tax value of Minto by \$101M with an NPV_{8.0%} of \$55M. The undiscounted after-tax incremental value increases by \$58M with an after-tax NPV_{8.0%} of \$35M. Incremental capital expenditure increases \$25M. Incremental total return vs. maximum exposure ratio is 3.94.

Sensitivity analyses were completed for Phase VI with variances applied to Copper Price, Capital Expenditures (“CAPEX”), Operating Expenditures (“OPEX”), and Copper Grade. Each variable was changed independently from $\pm 20\%$ of the base value and the after-tax, discounted (using a discount rate of 8%) after-tax values determined. The results are represented pictorially in Figure 1—3. As expected, the project shows high variability to copper price and grade, moderate sensitivity to operating costs and low sensitivity to capital costs.

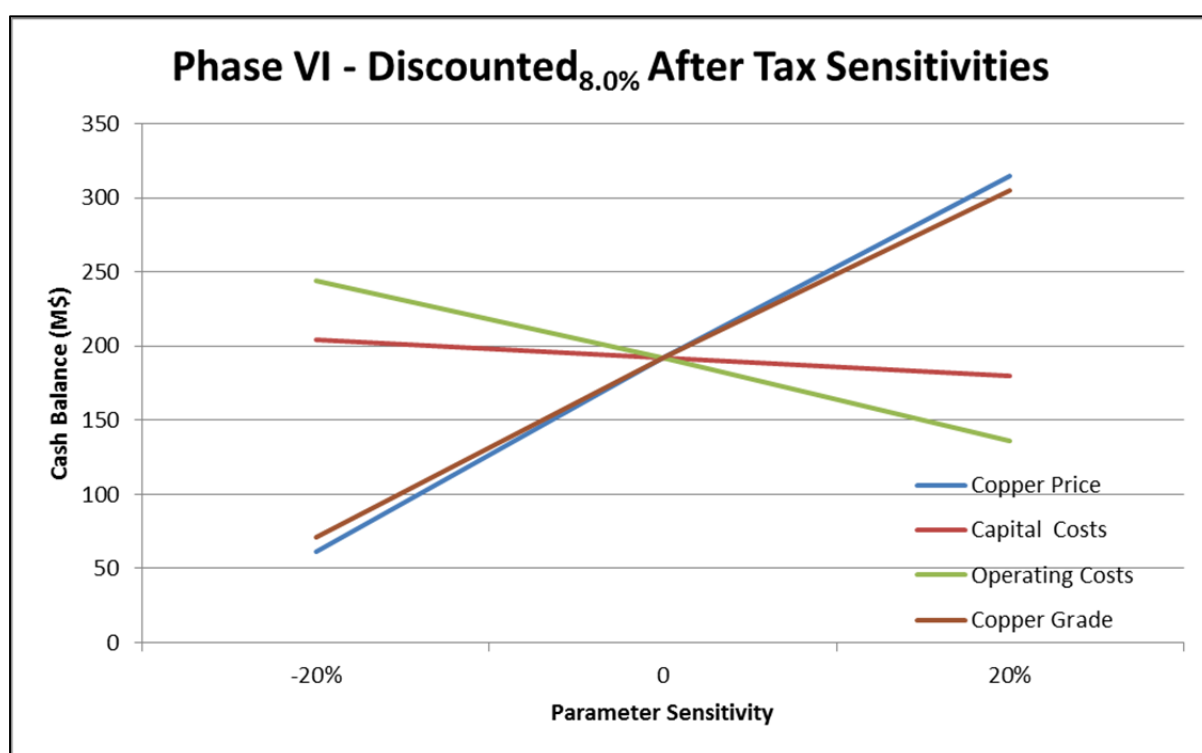


Figure 1—3: Phase VI Discounted_{8.0%} After-Tax Sensitivities

1.12. Conclusions

The conclusions of the Phase VI Preliminary Feasibility Study are:

- There continues to be strong exploration targets in the vicinity of the known deposits. This consists of new deposits and extensions to existing deposits;
- Based on the preliminary test work conducted to date, the Phase VI waste rock from the Copper Keel and Wildfire deposits does not indicate any ARD issues;
- The Phase VI Underground deposits (Copper Keel and Wildfire) are an increase in reserves that add value to the Minto Mine by increasing the NPV of the overall project with a minimal capital investment requirement;

-
- The Phase VI Preliminary Feasibility study has demonstrated economic justification for MintoEx to submit licence and permit applications to the various government agencies.

The major risk areas identified in this study are:

- Timing and approval of Phase V and Phase VI WUL permits;
- Exchange rates, metal prices and other external influences;
- The ability to execute the UG mine as per the Phase VI mining schedule;
- The ability to maintain assumed costs;
- Acquisition of experienced personnel for underground and open pit mining; and
- The ability to maintain minimum dilution through effective grade control practices.

The most important opportunities to improve the project are:

- Optimization of the mine plan;
- Reduce mineral processing and G&A unit operating expenditures;
- Continued strong demand for copper resulting in sustained high copper prices;
- Discovering new mineral resources and converting them to mineral reserves; and
- Expansion potential that would reduce operating expenditures resulting in significant improvements with respect to sustainability.

1.13. Recommendations

Detailed recommendations of this PFS are contained in Section 26 of this report. The main recommendations of note are:

- Optimization of the Life of Mine Plan with the objective of decreasing the mining rate to avoid excessive stockpiling that has a detrimental impact on NPV;
- Optimization of the Life of Mine Plan to take advantage of in-pit waste deposition that decreases the footprint and also lowers CAPEX stripping and OPEX waste expenditures.
- Continued assessment of geotechnical structures for both open pit and underground planning purposes;
- Continued exploration drilling from an underground drilling platform that can be developed in 2013; and
- Continued tailings settling testing to enhance deposition densities and accuracy of Life of Mine Planning.

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2. Introduction

2.1. Background

The Minto Exploration Ltd property is considered to be a “brownfield project” since it has been in operation since 2007. There have been numerous Technical Reports and Pre-feasibility Studies completed to date on the property.

This technical report, the Minto Phase VI Preliminary Feasibility Study Technical Report, was compiled by Minto Exploration Ltd. (“MintoEx”) to determine if a portion of the Minto South Deposit Resource (Copper Keel and Wildfire) can be converted to an NI 43-101 Mineral Reserves. The Pre-Feasibility Study describes:

- Mineral resource and reserve estimates;
- A proposed mine plan that now includes extraction of a portion of the Minto South Deposit Resource (Copper Keel and Wildfire);
- Updates to Plant capacity improvements; and
- Updated expenditures and economic performance.

This PFS was compiled under the direction of Capstone Mining Corp. and included significant contributions from a number of experienced professionals associated with various companies.

The exploration data with detailed assay and geology data from drilling and geophysical surveys associated with the MSD was compiled by SRK. SRK reviewed the Ridgetop deposits that have not been modified since the Phase V PFS was completed. In addition, Kirkham Geosystems reviewed the Minto North and East deposits that has also not been modified since the Phase V PFS was issued. Capstone reviewed and completed checks and validations on a portion of the Minto exploration database and verification checks were performed by Kirkham Geosystems.

Open Pit and Underground as well as other contract rates were provided by MintoEx. MintoEx also provided information regarding government agreements, advice on local labor rates and conditions as well as budget operating costs estimated for 2012 - 2016.

Previous technical reports or literature used in the compilation of this report are referenced throughout the text.

This report was prepared in compliance with National Instrument 43-101, Standards of Disclosure for Mineral Projects (“NI 43-101”).

All units in this report are based on the International System of Units (“SI”) and all currency values are Canadian dollars (“C\$” or “\$”) unless otherwise noted.

This report uses many common abbreviations and acronyms with explanations found in Section 29.

2.2. Qualified Persons

The Qualified Persons (“QPs”), as defined in NI 43-101, responsible for the preparation of this technical report include:

1. John Sagman, P.Eng., Capstone Mining Corp.;
2. Wayne Barnett, Pr.Sci.Nat., SRK Consulting (Canada), Inc.;
3. John Eggert, P.Eng, Eggert Engineering Ltd.;
4. Bill Hodgson, P.Eng., Genivar Inc.;
5. Garth Kirkham, P. Geo, Kirkham Geosystems Ltd.;
6. Michael Levy, PE, SRK Consulting (U.S.), Inc.;
7. Brad Mercer, P.Geo. Capstone Mining Corp.;
8. Pooya Mohseni, P.Eng. Minto Explorations Ltd.;
9. Bruce Murphy, P.Eng., SRK Consulting (Canada), Inc.; and
10. Colleen Roche, P.Eng. Capstone Mining Corp.

Personal visits to the Minto Mine were conducted by eight of the ten Qualified Persons (“QPs”) shown in Table 2-1.

Table 2-1: Qualified Persons

Name of QP	Area of Responsibility	Site Visit Date	Report Section Responsibility
John Sagman	Overview, Underground Mining, Economics	Numerous Occasions throughout 2010 & 2011	Sections: 1 to 1.5, 1.9 to 5.4, 15.1, 15.3, 16.1, 16.4.1 to 16.4.6, 16.4.6.2 to 16.4.8, 18.1.1, 18.1.3 to 18.2, 18.2.2 to 19.3.1, 21 to 29
Wayne Barnett	Geology and Resource Estimation	April 2011	Sections: 11, 12, 14.1 to 14.4
John Eggert	Metallurgy and Mineral Processing	Numerous Occasions throughout 2010 & 2011	Sections: 1.6 to 1.7, 13, 17, 25.3, 25.3.2, 25.3.4 and 25.3.5
Bill Hodgson	Ventilation and Camp Infrastructure	August 2011, September 2011	Sections: 16.4.6.1, 18.1.2, and 18.2.1
Garth Kirkham	Minto North and East Resource Estimation	No Site Visit	Sections: 14.5 and 14.6
Michael Levy	Open Pit Geotechnical	October 2011	Sections: 16.2
Brad Mercer	Mineral Resources	Numerous Occasions between 2005 & 2011	Sections: 1.1 to 1.3, 6 to 10 and 14.7
Pooya Mohseni	Open Pit Mining	Numerous Occasions in 2011 and 2012	Sections: 15.2, 16.5 to 16.7.3
Bruce Murphy	Underground Geotechnical	No Site Visit	Sections: 16.3
Colleen Roche	Environmental and Permitting	Numerous Occasions throughout 2011	Sections: 1.8 and 20

QP Murphy relied upon information obtained from site visits by QP Levy and Scott Carlisle, P.Eng.

3. Reliance on Other Experts

This report relies on information from the following non-qualified persons as defined in NI 43-101:

- Vivienne McLennan of MintoEx for exploration and land tenure databases and assisting in QA/QC; (Sections 4 to 13);
- Taras Nahnybida of MintoEx for assistance with geology, exploration and QA/QC; (Sections 4 to 13)
- Norm Matheson of Matheson Consulting for assistance in Underground Mine Design and Cost Estimation (Section 16);
- Metallurgical testing conducted by G&T Metallurgical Services Ltd;
- ARD-ML work completed by SGS Canada Inc;
- Richard Godfrey of MintoEx for assistance in tax calculations;
- Ted Kenny of MintoEx for assistance with Process Plant Design;
- B. Ross Design Inc. ("BRDI") for processing design and costing; and
- BESTECH for electrical system description and costing.

The QPs, authors of this Report, state that they are Qualified Persons for those areas as identified in the appropriate QP "Certificate of Author" in Appendix A. The authors have relied upon information derived from the following reports:

- SRK Consulting (Canada), Inc., "MintoEx Phase V Preliminary Feasibility Study, December 2010" dated March 2011;
- SRK Consulting (Canada), Inc., "Minto Phase IV Preliminary Feasibility Technical Report, December 2009. Planning & Design" Report dated December 15, 2010
- SRK Consulting (U.S.), Inc., "Pre-feasibility Geotechnical Evaluation, Phase IV, Minto Mine" dated December 2009.
- SRK Consulting (U.S.), Inc., "Pre-feasibility Pit Slope Evaluation for Area 2 Open Pit, Minto Mine" dated October 2007.
- SRK Consulting (Canada), Inc., "Area 2 Pre-feasibility Study Minto Mine, Yukon" dated November 2007.

The preparation of this report is also based upon public and private information provided by MintoEx and on information provided in additional Technical Reports that are listed in Section 28 of this report.

The authors believe that the information provided and relied upon for preparation of this report is accurate at the time of the report and that the interpretations and opinions expressed in them are reasonable and based on current understanding of mining and processing techniques, costs, economics, mineralization processes and the host geologic setting. The QPs have made reasonable efforts to verify the accuracy of the data relied on in this report.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made

herein the authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the authors subsequent to the date of this report.

4. Property Description and Location

The Minto Mine is located in the Whitehorse Mining District in central Yukon. The property is located approximately 240 km northwest of Whitehorse, the Yukon capital. (See Figure 4—1). The project consists of 164 Quartz Claims covering an area of approximately 2,760 ha.

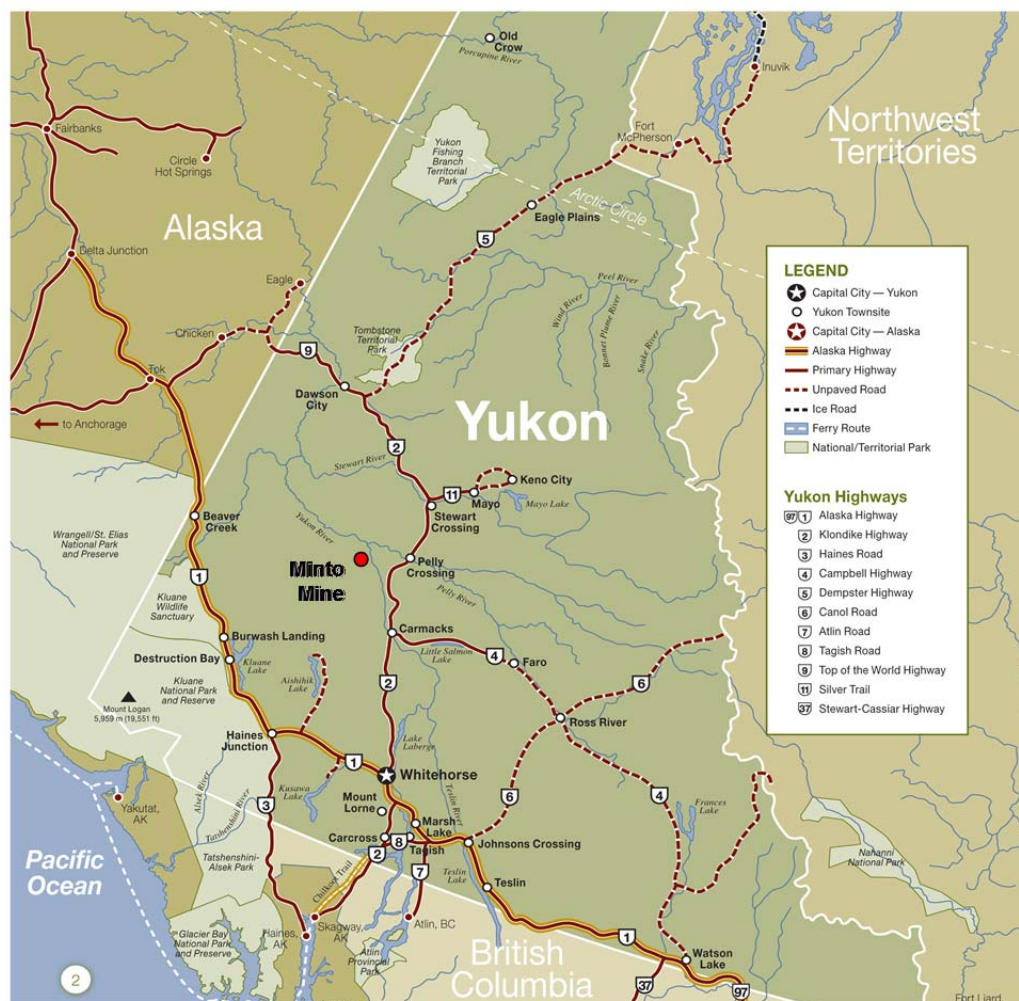


Figure 4—1: Location Map

The project is centered approximately on NAD 83; UTM Zone 8 coordinates 6,945,000 m N, 385,000 m E. The Minto Mine can be located on the Yukon Government Department of Energy, Mines and Resources 1:30,000 scale Mining Claims Map number 115I11, May 19, 2009. See for a portion of the map showing the boundaries of the Minto Explorations Ltd. claims.

The Mine is located on the west side of the Yukon River on Selkirk First Nation (SFN) Category A settlement land (SFN Parcel R-6A).

The 100% registered owner of the claims is Minto Explorations Ltd., a 100% owned subsidiary of Capstone Mining Corp. The current status of the claims is shown in Table 4-1 as per the Yukon

Government Energy, Mines and Resources Mining Claims Search website. The status of the claims has been recently confirmed with the Mining Recorder.

The lease, but not the claim boundaries, was surveyed by an authorized Canada Lands Surveyor in accordance with instructions from the Surveyor General.

There are no known back-in rights, payments, other agreements or encumbrances to which the property is subject other than a recently amended Cooperation Agreement with the Selkirk First Nations ("SFN") and a net smelter royalty payable to the SFN.

MintoEx is subject environmental regulations as outlined in the various permits required to operate a mine in the Yukon Territory. MintoEx is liable for reclamation of the site as outlined in the most up-to-date closure plan dated: September 9, 2011. MintoEx current QML licence requires the mine site to submit updated closure plans at regular intervals (typically every two years). More information on environmental liabilities to which the property is subject to can be found in Section 20.

MintoEx requires a variety of permits in order to conduct ongoing work on site. *The major instruments or authorizations permitting and governing operations for the project include a Type A Water Use Licence, issued by the Yukon Water Board and a Quartz Mining Licence issued by Yukon Government, Energy Mines and Resources (SRK 2011).* MintoEx has received a portion of the permits necessary to extract ore from additional mining areas, higher plant throughput, revised waste and tailings management facilities and other environmental aspects of the project. At this time there is good confidence that the additional permits to access the new deposits and waste management facilities will be granted after the government authorities have completed their reviews. More information on permits required to operate in the Yukon can be found in Section 20.

Table 4-1: Minto Explorations Ltd. Claim Status

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61620	Quartz	MINTO	1	8/9/1971	5/13/2018	Active	OW00001	500057691
Y 61621	Quartz	MINTO	2	8/9/1971	5/13/2018	Active	OW00002	500057692
Y 61622	Quartz	MINTO	3	8/9/1971	5/13/2018	Active	OW00003	500057693
Y 61623	Quartz	MINTO	4	8/9/1971	5/13/2018	Active	OW00004	500057694
Y 61624	Quartz	MINTO	5	8/9/1971	5/13/2018	Active	OW00005	500057695
Y 61625	Quartz	MINTO	6	8/9/1971	5/13/2018	Active	OW00006	500057696
Y 61626	Quartz	MINTO	7	8/9/1971	5/13/2018	Active	OW00007	500057697
Y 61627	Quartz	MINTO	8	8/9/1971	5/13/2018	Active	OW00008	500057698
Y 61628	Quartz	MINTO	9	8/9/1971	5/13/2018	Active	OW00009	500057699
Y 61629	Quartz	MINTO	10	8/9/1971	5/13/2018	Active	OW00010	500057700
Y 61630	Quartz	MINTO	11	8/9/1971	5/13/2018	Active	OW00011	500057701

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61631	Quartz	MINTO	12	8/9/1971	5/13/2018	Active	OW00012	500057702
Y 61632	Quartz	MINTO	13	8/9/1971	5/13/2018	Active	OW00013	500057703
Y 61633	Quartz	MINTO	14	8/9/1971	5/13/2018	Active	OW00014	500057704
Y 61634	Quartz	MINTO	15	8/9/1971	5/13/2018	Active	OW00015	500057705
Y 61635	Quartz	MINTO	16	8/9/1971	5/13/2018	Active	OW00016	500057706
Y 61693	Quartz	DEF	1	8/23/1971	10/7/2028	Active	OW00230	500057707
Y 61694	Quartz	DEF	2	8/23/1971	10/7/2028	Active	OW00231	500057708
Y 61695	Quartz	DEF	3	8/23/1971	10/7/2028	Active	OW00232	500057709
Y 61696	Quartz	DEF	4	8/23/1971	10/7/2028	Active	OW00233	500057710
Y 61697	Quartz	DEF	5	8/23/1971	10/7/2028	Active	OW00234	500057711
Y 61698	Quartz	DEF	6	8/23/1971	10/7/2028	Active	OW00235	500057712
Y 61699	Quartz	DEF	7	8/23/1971	10/7/2028	Active	OW00236	500057713
Y 61700	Quartz	DEF	8	8/23/1971	10/7/2028	Active	OW00237	500057714
Y 61701	Quartz	DEF	9	8/23/1971	10/7/2028	Active	OW00238	500057715
Y 61702	Quartz	DEF	10	8/23/1971	3/1/2017	Active		500057716
Y 61703	Quartz	DEF	11	8/23/1971	10/7/2028	Active	OW00239	500057717
Y 61704	Quartz	DEF	12	8/23/1971	3/1/2017	Active		500057718
Y 61705	Quartz	DEF	13	8/23/1971	10/7/2028	Active	OW00240	500057719
Y 61706	Quartz	DEF	14	8/23/1971	10/7/2028	Active	OW00241	500057720
Y 61707	Quartz	DEF	15	8/23/1971	10/7/2028	Active	OW00242	500057721
Y 61708	Quartz	DEF	16	8/23/1971	10/7/2028	Active	OW00243	500057722
Y 61709	Quartz	DEF	17	8/23/1971	10/7/2028	Active	OW00244	500057723
Y 61710	Quartz	DEF	18	8/23/1971	10/7/2028	Active	OW00245	500057724
Y 61711	Quartz	DEF	19	8/23/1971	3/1/2017	Active		500057725
Y 61712	Quartz	DEF	20	8/23/1971	3/1/2017	Active		500057726
Y 61713	Quartz	DEF	21	8/23/1971	3/1/2017	Active		500057727
Y 61714	Quartz	DEF	22	8/23/1971	3/1/2017	Active		500057728
Y 61715	Quartz	DEF	23	8/23/1971	3/1/2017	Active		500057729
Y 61716	Quartz	DEF	24	8/23/1971	3/1/2017	Active		500057730
Y 61717	Quartz	DEF	25	8/23/1971	3/1/2017	Active		500057731

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61718	Quartz	DEF	26	8/23/1971	3/1/2017	Active		500057732
Y 61719	Quartz	DEF	27	8/23/1971	3/1/2017	Active		500057733
Y 61720	Quartz	DEF	28	8/23/1971	3/1/2017	Active		500057734
Y 61721	Quartz	DEF	29	8/23/1971	3/1/2017	Active		500057735
Y 61722	Quartz	DEF	30	8/23/1971	3/1/2017	Active		500057736
Y 61723	Quartz	DEF	31	8/23/1971	10/7/2028	Active	OW00246	500057737
Y 61724	Quartz	DEF	32	8/23/1971	10/7/2028	Active	OW00247	500057738
Y 61904	Quartz	MINTO	17	8/31/1971	5/13/2018	Active	OW00017	500057910
Y 61905	Quartz	MINTO	18	8/31/1971	5/13/2018	Active	OW00018	500057911
Y 61906	Quartz	MINTO	19	8/31/1971	3/1/2017	Active		500057912
Y 61907	Quartz	MINTO	20	8/31/1971	3/1/2017	Active		500057913
Y 61908	Quartz	MINTO	35	8/31/1971	5/13/2018	Active	OW00021	500057914
Y 61909	Quartz	MINTO	36	8/31/1971	5/13/2018	Active	OW00022	500057915
Y 61910	Quartz	MINTO	37	8/31/1971	3/1/2017	Active		500057916
Y 61911	Quartz	MINTO	38	8/31/1971	3/1/2017	Active		500057917
Y 61914	Quartz	MINTO	23	8/31/1971	3/1/2017	Active		500057920
Y 61915	Quartz	MINTO	24	8/31/1971	3/1/2017	Active		500057921
Y 61916	Quartz	MINTO	25	8/31/1971	3/1/2017	Active		500057922
Y 61917	Quartz	MINTO	26	8/31/1971	3/1/2017	Active		500057923
Y 61918	Quartz	MINTO	27	8/31/1971	3/1/2017	Active		500057924
Y 61919	Quartz	MINTO	28	8/31/1971	3/1/2017	Active		500057925
Y 61920	Quartz	MINTO	31	8/31/1971	3/1/2017	Active		500057926
Y 61921	Quartz	MINTO	32	8/31/1971	5/13/2018	Active	OW00019	500057927
Y 61922	Quartz	MINTO	33	8/31/1971	3/1/2017	Active		500057928
Y 61923	Quartz	MINTO	34	8/31/1971	5/13/2018	Active	OW00020	500057929
Y 61926	Quartz	MINTO	41	8/31/1971	3/1/2017	Active		500057932
Y 61927	Quartz	MINTO	42	8/31/1971	3/1/2017	Active		500057933
Y 61928	Quartz	MINTO	43	8/31/1971	3/1/2017	Active		500057934
Y 61929	Quartz	MINTO	44	8/31/1971	3/1/2017	Active		500057935
Y 61930	Quartz	MINTO	45	8/31/1971	5/13/2018	Active	OW00023	500057936

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61931	Quartz	MINTO	46	8/31/1971	5/13/2018	Active	OW00024	500057937
Y 61932	Quartz	MINTO	29	8/31/1971	3/1/2017	Active		500057938
Y 61933	Quartz	MINTO	30	8/31/1971	3/1/2017	Active		500057939
Y 61934	Quartz	MINTO	47	8/31/1971	5/13/2018	Active	OW00025	500057940
Y 61935	Quartz	MINTO	48	8/31/1971	5/13/2018	Active	OW00026	500057941
Y 61936	Quartz	MINTO	49	8/31/1971	5/13/2018	Active	OW00027	500057942
Y 61937	Quartz	MINTO	50	8/31/1971	5/13/2018	Active	OW00028	500057943
Y 61938	Quartz	MINTO	51	8/31/1971	5/13/2018	Active	OW00029	500057944
Y 61939	Quartz	MINTO	52	8/31/1971	5/13/2018	Active	OW00030	500057945
Y 61978	Quartz	DEF	33	9/8/1971	10/7/2028	Active	OW00248	500057958
Y 61979	Quartz	DEF	34	9/8/1971	10/7/2028	Active	OW00249	500057959
Y 61980	Quartz	DEF	35	9/8/1971	3/1/2017	Active		500057960
Y 61981	Quartz	DEF	36	9/8/1971	3/1/2017	Active		500057961
Y 61982	Quartz	DEF	37	9/8/1971	10/7/2028	Active	OW00250	500057962
Y 61983	Quartz	DEF	38	9/8/1971	10/7/2028	Active	OW00251	500057963
Y 61984	Quartz	DEF	39	9/8/1971	3/1/2017	Active		500057964
Y 61985	Quartz	DEF	40	9/8/1971	3/1/2017	Active		500057965
Y 61986	Quartz	DEF	41	9/8/1971	3/1/2017	Active		500057966
Y 61987	Quartz	DEF	42	9/8/1971	3/1/2017	Active		500057967
Y 61988	Quartz	DEF	43	9/8/1971	3/1/2017	Active		500057968
Y 61989	Quartz	DEF	44	9/8/1971	3/1/2017	Active		500057969
Y 61990	Quartz	DEF	45	9/8/1971	3/1/2017	Active		500057970
Y 61991	Quartz	DEF	46	9/8/1971	3/1/2017	Active		500057971
Y 61992	Quartz	DEF	47	9/8/1971	3/1/2017	Active		500057972
Y 61993	Quartz	DEF	48	9/8/1971	3/1/2017	Active		500057973
Y 61994	Quartz	DEF	49	9/8/1971	3/1/2017	Active		500057974
Y 61995	Quartz	DEF	50	9/8/1971	3/1/2017	Active		500057975
Y 61996	Quartz	DEF	51	9/8/1971	3/1/2017	Active		500057976
Y 61997	Quartz	DEF	52	9/8/1971	3/1/2017	Active		500057977
Y 61998	Quartz	DEF	53	9/8/1971	3/1/2017	Active		500057978

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 61999	Quartz	DEF	54	9/8/1971	3/1/2017	Active		500057979
Y 62000	Quartz	DEF	55	9/8/1971	3/1/2017	Active		500057980
Y 62001	Quartz	DEF	56	9/8/1971	3/1/2017	Active		500057981
Y 62002	Quartz	DEF	57	9/8/1971	3/1/2017	Active		500057982
Y 62003	Quartz	DEF	58	9/8/1971	3/1/2017	Active		500057983
Y 62004	Quartz	DEF	59	9/8/1971	3/1/2017	Active		500057984
Y 62005	Quartz	DEF	60	9/8/1971	3/1/2017	Active		500057985
Y 62006	Quartz	DEF	61	9/8/1971	3/1/2017	Active		500057986
Y 62007	Quartz	DEF	62	9/8/1971	3/1/2017	Active		500057987
Y 62008	Quartz	DEF	63	9/8/1971	3/1/2017	Active		500057988
Y 62009	Quartz	DEF	64	9/8/1971	3/1/2017	Active		500057989
Y 62010	Quartz	DEF	65	9/8/1971	3/1/2017	Active		500057990
Y 62011	Quartz	DEF	66	9/8/1971	3/1/2017	Active		500057991
Y 62012	Quartz	DEF	67	9/8/1971	3/1/2017	Active		500057992
Y 62013	Quartz	DEF	68	9/8/1971	3/1/2017	Active		500057993
Y 62014	Quartz	DEF	69	9/8/1971	3/1/2013	Active		500057994
Y 62015	Quartz	DEF	70	9/8/1971	3/1/2017	Active		500057995
Y 62016	Quartz	DEF	71	9/8/1971	3/1/2017	Active		500057996
Y 62017	Quartz	DEF	72	9/8/1971	3/1/2017	Active		500057997
Y 62018	Quartz	DEF	73	9/8/1971	3/1/2017	Active		500057998
Y 62019	Quartz	DEF	74	9/8/1971	3/1/2017	Active		500057999
Y 62020	Quartz	DEF	75	9/8/1971	3/1/2017	Active		500058000
Y 62021	Quartz	DEF	76	9/8/1971	3/1/2017	Active		500058001
Y 62022	Quartz	DEF	77	9/8/1971	3/1/2017	Active		500058002
Y 62023	Quartz	DEF	78	9/8/1971	3/1/2017	Active		500058003
Y 62296	Quartz	MINTO	65	9/22/1971	5/13/2018	Active	OW00031	500058004
Y 62297	Quartz	MINTO	66	9/22/1971	5/13/2018	Active	OW00032	500058005
Y 62298	Quartz	MINTO	67	9/22/1971	5/13/2018	Active	OW00033	500058006
Y 62299	Quartz	MINTO	68	9/22/1971	5/13/2018	Active	OW00034	500058007
Y 62300	Quartz	MINTO	69	9/22/1971	3/1/2017	Active		500058008

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 62301	Quartz	MINTO	70	9/22/1971	5/13/2018	Active	OW00035	500058009
Y 62302	Quartz	MINTO	71	9/22/1971	5/13/2018	Active	OW00036	500058010
Y 62303	Quartz	MINTO	72	9/22/1971	3/1/2017	Active		500058011
Y 62304	Quartz	MINTO	73	9/22/1971	3/1/2017	Active		500058012
Y 62305	Quartz	MINTO	75	9/22/1971	3/1/2017	Active		500058013
Y 62306	Quartz	MINTO	76	9/22/1971	3/1/2017	Active		500058014
Y 62307	Quartz	MINTO	77	9/22/1971	3/1/2017	Active		500058015
Y 62308	Quartz	MINTO	78	9/22/1971	3/1/2017	Active		500058016
Y 62309	Quartz	MINTO	79	9/22/1971	3/1/2017	Active		500058017
Y 62310	Quartz	MINTO	80	9/22/1971	3/1/2017	Active		500058018
Y 62311	Quartz	MINTO	81	9/22/1971	3/1/2017	Active		500058019
Y 62312	Quartz	MINTO	82	9/22/1971	3/1/2017	Active		500058020
Y 62313	Quartz	MINTO	83	9/22/1971	3/1/2017	Active		500058021
Y 62314	Quartz	MINTO	84	9/22/1971	3/1/2017	Active		500058022
Y 62315	Quartz	MINTO	85	9/22/1971	3/1/2017	Active		500058023
Y 62316	Quartz	MINTO	86	9/22/1971	3/1/2017	Active		500058024
Y 62317	Quartz	MINTO	87	9/22/1971	3/1/2017	Active		500058025
Y 62318	Quartz	MINTO	88	9/22/1971	3/1/2017	Active		500058026
Y 62319	Quartz	MINTO	89	9/22/1971	3/1/2017	Active		500058027
Y 66779	Quartz	DEF	79	7/11/1972	10/7/2028	Active	OW00252	500058071
Y 66780	Quartz	DEF	80	7/11/1972	10/7/2028	Active	OW00253	500058072
Y 66781	Quartz	DEF	81	7/11/1972	10/7/2028	Active	OW00254	500058073
Y 66782	Quartz	DEF	82	7/11/1972	10/7/2028	Active	OW00255	500058074
Y 66783	Quartz	DEF	83	7/11/1972	10/7/2028	Active	OW00256	500058075
Y 66784	Quartz	DEF	84	7/11/1972	10/7/2028	Active	OW00257	500058076
Y 76953	Quartz	DEF	1379	8/31/1973	10/7/2028	Active	OW00258	500058311
Y 76954	Quartz	DEF	85	8/31/1973	3/1/2017	Active		500058312
Y 76955	Quartz	DEF	86	8/31/1973	3/1/2017	Active		500058313
Y 76956	Quartz	DEF	87	8/31/1973	3/1/2017	Active		500058314
Y 77310	Quartz	MINTO	94	10/1/1973	3/1/2017	Active		500058315

Grant Number	Reg Type	Claim Name	Claim No.	Operation Recording Date	Claim Expiry Date	Status	Quartz Lease	Ops Number
Y 77311	Quartz	MINTO	95	10/1/1973	3/1/2017	Active		500058316
Y 78024	Quartz	MINTO	96	11/13/1973	3/1/2017	Active		500058317
Y 78025	Quartz	MINTO	97	11/13/1973	3/1/2017	Active		500058318

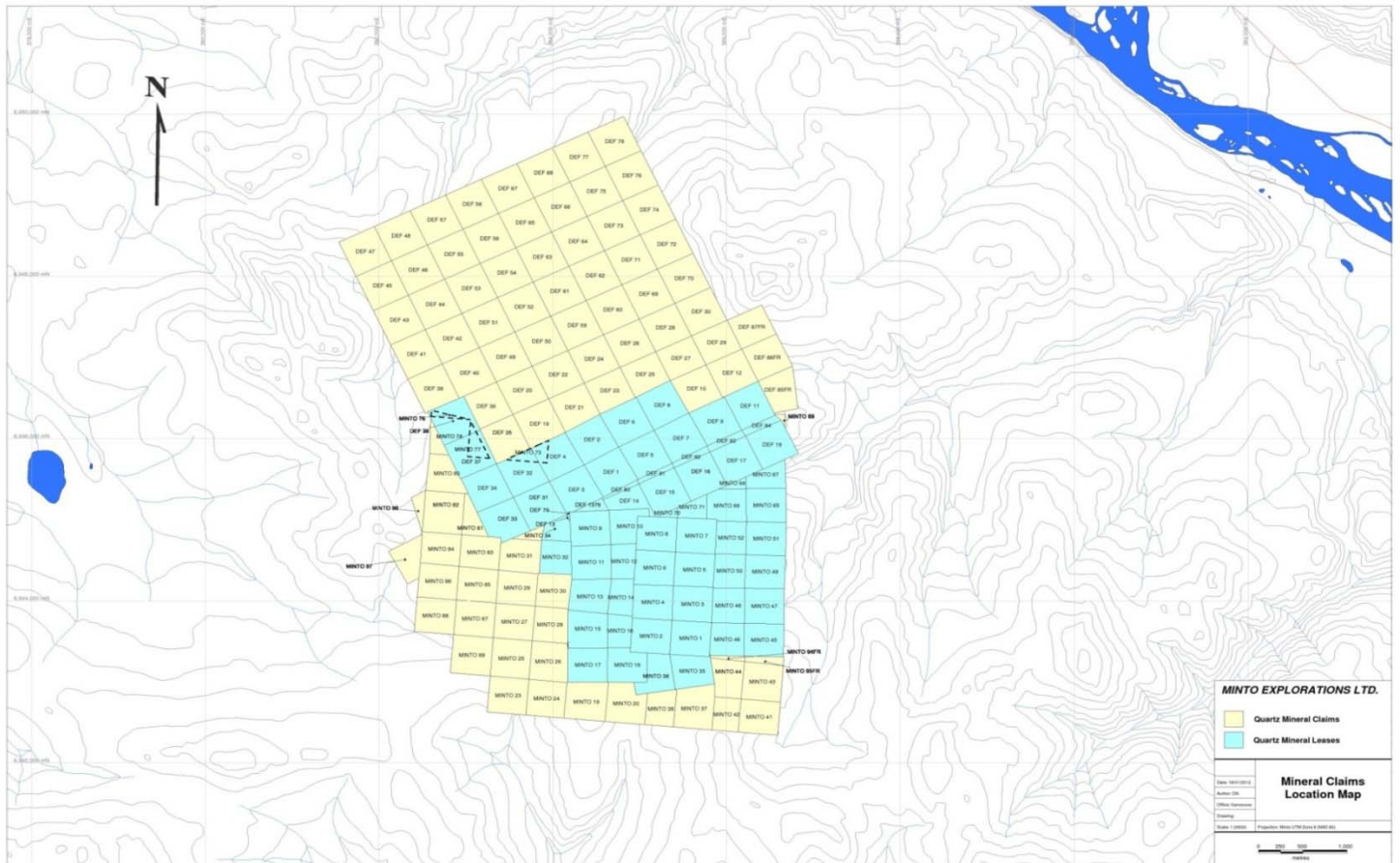


Figure 4—2: Mineral Claims Location Map

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1. Accessibility

The Minto Mine continues to be accessible via the Klondike Highway (No. 2) to Minto Landing on the east side of the Yukon River. At Minto Landing, the mine operates a barge across the river in the summer months and constructs an ice bridge in the winter. The barge has the capacity to carry one B-train transport trailer and truck (see Figure 5—1). There is typically a 6 to 8 week period during each break-up and freeze-up of the Yukon River when there is no access across the river. A 27 km long, all-weather gravel road provides access from the west side of the Yukon River to the project site. The mine access road crosses one major tributary of the Yukon River, Big Creek, via a single-lane steel span bridge made with reinforced concrete abutments and deck. The highway, river crossing and gravel mine access road are suitable for heavy transport traffic.

When access across the Yukon River is available, operations personnel are transported to the site in commercial buses based out of Whitehorse. During the river freeze and thaw periods, personnel are transported from Whitehorse via charter air services that land on the 1,300 m airstrip located at the mine.



Figure 5—1: Minto Barge Crossing the Yukon River (SRK, 2009)

5.2. Climate

The climate in the Minto area of the Yukon is considered sub-arctic with short cool summers and long cold winters. The average temperature in the summer is 10°C and the average temperature in the winter is -20°C. Average precipitation is approximately 25 cm of rain equivalent per annum in the form of rain and snow.

Like most northern Canadian mines the weather does not impede year round operation of the mine and the processing plant except in short periods of harsh cold temperatures which may drop to -50°C, that can cause open pit mining operations to be temporarily suspended.

5.3. Physiography

The property lies in the Dawson Range, part of the Klondike Plateau, an uplifted surface that has been dissected by erosion. Local topography consists of rounded rolling hills and ridges and broad valleys (Figure 5—2). The highest elevation on the property is approximately 1,000 m above sea level, compared to elevations of 460 m along the Yukon River. Slopes on the property are relatively gentle and do not present accessibility problems. Bedrock outcrops can often be found at the tops of hills and ridges. There are no risks of avalanche on the property.

Overburden is colluvium primarily comprised of granite-based sand from weathering of the granitic bedrock in the area and is generally thin but pervasive but can reach plus 50 m in depth. Seams of clay and ice lenses are also present sporadically. South-facing slopes generally provide well-drained, sound foundation for buildings and roads. North-facing slopes in the area typically contain permafrost.



Figure 5—2: Mine Access Road Showing General Relief and Vegetation in the Area (SRK, 2006)

Vegetation in the area is sub-Arctic boreal forest made up of largely spruce and poplar trees. The area has experienced several wildfires over the years, the latest in 2010, and has no old-growth trees remaining. The fire in 2010 led to the partial evacuation of the camp and a short stoppage in production.

5.4. Local Resources and Infrastructure

The nearest services, including fuel, groceries, hotel, restaurant and clinic, are at Carmacks, approximately 75 km south of Minto on Highway 2. Some services are available at Pelly Crossing, 35 km to the east of Minto.

The nearest large community is Whitehorse, the capital of Yukon. Whitehorse has a population of approximately 26,000, and is the transportation, governmental and commercial hub for the region. It is serviced with commercial flights daily from Vancouver, Edmonton and other northern communities. Whitehorse is also connected via paved highways to British Columbia to the south, Alaska to the west, and south to the port of Skagway, where Minto concentrate is trucked for loading onto ocean-going vessels.

The Minto mine has been a commercial operation for more than four years and has sufficient power and water, and additions approved to camp and personnel to continue operations through the life of mine plan.

6. History

Production results for 2007 to 2011 are shown in Table 6-1 (as provided by MintoEx). Commercial production was declared on October 1, 2007 after a 4-month commissioning period. Results for 2008 and 2009 showed consistent increase in production and recovery as the mill facility optimization plans were carried out and mill expansion plans were implemented. Operations in 2010 were constrained for an extended period as a result of constraints in the tailings filtration facility, which is planned to be eliminated going forward. The positive processing results at Minto have been largely driven by the amenability of the ore to flotation at a coarse primary grind size.

Table 6-1: 2007 to 2011 Operating Results

Parameter	Unit	2007	2008	2009	2010	2011
Waste mining	Mt	9.26	8.37	11.13	7.97	11.7
Ore mining	Mt	0.75	0.83	1.15	1.49	0.73
Total material mined	Mt	10.01	9.53	12.28	9.47	12.43
Tonnes processed	Tonnes	238,446	809,426	1,031,190	915,051	1,258,308
Mill head copper grade	%	2.16	2.91	2.59	2.22	1.61
Mill head gold grade	g/t	n/a	1.28	1.14	0.93	0.68
Mill head silver grade	g/t	7.7	11.8	11	8.7	5.93
Copper recovery	%	85.1	91.9	92.6	90.3	87.9
Gold recovery	%	n/a	77.7	75.3	81.1	75.8
Silver recovery	%	77.5	84.6	81.9	80.6	78.6
Concentrate produced	Dmt	12,630	53,148	59,863	47,065	49,159
Concentrate grade – Cu	%	34.7	40.7	41.4	39	36.32
Concentrate grade – Au	g/t	n/a	15.9	14.9	14.7	12.00
Concentrate grade – Ag	g/t	113	152	155.8	137	124.08
Copper in concentrate	M lb.	9.66	47.69	54.63	40.5	37.12
Gold in concentrate	Oz	n/a	27,202	18,828	22,284	18,439
Silver in concentrate	Oz	45,885	217,489	299,767	206,838	196,098

The following section was taken from Section 8 from the “Technical Report (43-101) for the Minto Project” by Hatch (August 2006) found on the [sedar.com](http://www.sedar.com) website and updated to describe events and information subsequent to the effective date of that report.

Mineral exploration on the Minto property has been conducted since 1971. Exploration efforts by MintoEx since July 2005 are explained in Section 6.2 MintoEx 2005-2011, and a description of drilling during this time is contained in Section 10: Drilling.

6.1. Chronology

A history of mineral exploration to production in the area is summarized below.

1970

- Regional stream sediment geochemical survey by the Dawson Syndicate, a joint venture between Silver Standard Mines Ltd. and Asarco Inc.

1971

- Follow-up of stream sediment anomalies and staking of the Minto claims in July;
- Soil sampling, IP geophysical surveys and manual excavated prospect pits on the Minto claims;
- 7 diamond drill holes completed (1,158 m);
- DEF claims staked by United Keno Explorations;
- A joint venture formed with United Keno Hill Mines, Falconbridge Nickel and Canadian Superior Explorations, to cover follow-up prospecting;
- IP and VLF-EM geophysical surveys, soil sampling and mapping on the DEF claims.

1972

- Mapping, airstrip construction and bulldozer trenching, 12 diamond drill holes (1,871 m) on 4;
- zones on the Minto claims;
- Grid soil sampling and bulldozer trenching on the DEF claims.

1973

- 62 diamond drill holes (7,887 m) on the Minto claims;
- Bulldozer trenching, EM and magnetic geophysical surveys and 41 diamond drill holes (7,753 m) on the DEF claims;
- Main mineralized body discovered in June.

1974

- Winter road built from Yukon Crossing and 58 diamond drill holes (11,228 m) on the Minto claims;
- Additional geophysics, rock mechanics, feasibility studies and 52 diamond drill holes (8,238 m) on the DEF claims.

1975-1976

- Joint feasibility studies.

1984

- Silver Standard changed its name to Consolidated Silver Standard and transferred its interest in the Minto claims to Western Copper Holdings, a subsidiary of Teck Corp;
- 5 percussion drill holes (518 m) on the DEF claims.

1989

- Western Copper Holdings transferred its interest in the Minto claims to Teck Corp;
- 84 percussion drill holes (4,897 m) on the DEF claims.

1993

- MintoEx was formed;
- Asarco and Teck sold their interest in the Minto claims (and leases) for shares in MintoEx and provided \$375,000 in working capital;
- Asarco and Teck also received a net smelter royalty of 1.5% to be divided evenly;
- Falconbridge, the parent of United Keno Hill, sold its interest in the DEF claims to MintoEx for \$1M, payment due in 1996;
- Falconbridge was granted an option to repurchase the DEF claims on January 1, 2005 if the deposit was not in production by then;
- MintoEx carried out an airborne geophysical survey and drilled 8 diamond drill holes (960 m).

1994

- Initial public offering of shares of MintoEx completed;
- 5,912,501 shares were issued and outstanding with Asarco the majority shareholder with 3,297,500 shares (55.8%);
- 19 diamond drill holes (2,185 m);
- Feasibility study began with engineering and geo-technical studies.

1995

- 6 diamond drill holes (572 m) on magnetic anomalies and 1 condemnation diamond drill hole north of the proposed mill site;
- Feasibility study completed, reserves are 8,818,000 t of 1.73% Cu, 0.014 oz/t Au and 0.22 oz/t Ag at 0.5% Cu cut-off grade;
- Recoveries are 95% for Cu and 85% for Au and Ag;
- Mine life was projected to be 12 years at production rate of 477,000 tonnes per year.

1996

- Funding arranged with Asarco to bring the deposit into production whereby Asarco would provide up to US\$25M. Under the funding arrangement, Asarco would acquire a 70% interest in the project, MintoEx would retain a 30% interest and remain as operator;
- MintoEx makes the \$1M payment to Falconbridge for the DEF claims completing the consolidation of the Minto and DEF claims;
- 16 km access road constructed including a barge landing site on the west side of the Yukon River and a bridge over Big Creek;
- 4 diamond drill holes (545 m).

1997

- A further 12.8 km of road construction to complete the new access road;
- Site for camp excavated;

-
- 72 m water well for domestic water supply;
 - Mill site excavated and 2 used grinding mills moved onto site using an ice bridge over the Yukon River;
 - Co-operation agreement signed with SFN.

1998

- Mill concrete foundations poured with cement trucks from Whitehorse barged across the Yukon River;
- Type A Water license granted by Yukon government;
- Concentrator design completed;
- Access road completed, camp constructed and the location of the proposed tailings dam was grouted;
- Phase 1 open pit mining plan completed.

1999

- Production license received;
- Five diamond drill holes (957 m) for engineering purposes.

2000

- Minor maintenance of on-site facilities;
- Hatch completes review of 1995 feasibility study.

2001

- Additional maintenance of camp facilities;
- 5 confirmation diamond drill holes (552 m) in the centre of the deposit;
- Most of the Asarco core and all of the Falconbridge core destroyed by time and forest fire;
- Regional airborne magnetic and radiometric surveys carried out by the Yukon government.

2002

- A limited amount of the old Asarco core that could be recovered was re-sampled;
- All the drill and geophysical data compiled in a data base to aid further exploration;
- 3 Landsat anomalies examined and prospected;
- Road maintenance scheduled to keep permits active;
- Asarco bought 100,000 shares of MintoEx to hold a total of 3,397,500 shares.

2004

- MintoEx announces all its shares are for sale.

2005

- Sherwood Copper Corp. acquires the Minto Mine property June 2005;
- 44 confirmation drill holes (5937 m) to confirm the Main Minto Deposit Resources and Reserves.

2006

- Confirmation drilling program executed in order to update the precious metal resource;

-
- Development of Minto Project and commencement of pre-stripping the Minto Deposit;
 - Drill discovery and definition of Area 2 deposit ;
 - Copper Keel prospect discovered;
 - Mill construction commences;
 - C\$85 M debt package arranged, forward sales complete, concentrate off-take agreement executed October 2006.

2007

- Power Purchase Agreement for Minto signed;
- Resource estimate for Area 2 deposit completed;
- First copper-gold concentrates at Minto Mine produced;
- 1 exploration and 4 metallurgical drill holes (754 m) at Minto Deposit;
- Area 118 and Ridgetop deposits discovered and partially drill defined;
- Airstrip prospect discovered;
- First concentrates from Minto mine delivered to Port of Skagway, Alaska July 2007;
- Minto Mine declares commercial production and first Minto concentrates shipped from Skagway October 2007;
- Pre-feasibility Study for expansion of Minto copper-gold mine December 2007;
- Phase 2 mill expansion at Minto Mine completed ahead of schedule.

2008

- Minto Mine achieves and exceeds design capacity;
- Reported copper-gold resources increased at Minto Mine June 2008;
- Capstone and Sherwood announce combination to create intermediate copper producer with Sherwood shareholders overwhelmingly approving business combination;
- Closing of precious metal transaction; Silverstone provides upfront payment of US\$37.5 M for payable gold and silver from Minto;
- Minto Mine connects to electrical grid;
- Capstone and Sherwood complete business combination November 2008;
- Definition of thick zones of near surface copper mineralization at Ridgetop and deeper mineralization at Area 118.

2009

- High grade Minto North Deposit discovered and defined;
- Increased copper-gold mineral resources at Minto announced in June 2009;
- Dipole-dipole geophysical survey over northern regional targets;
- Titan 24 survey over the Minto Priority Exploration corridor;
- Drill discovery of the Minto East prospect.

2010

- High grade Minto East Deposit defined;
- Expanded copper-gold mineral resources for Area 2/118 and Ridgetop plus preliminary resource for Minto East in August 2010;
- Extended Titan 24 survey over 85% of the Minto property;

-
- Drill discovery of the Wildfire and Inferno prospects; and
 - Updated Mineral Resource for Minto East.

2011

- Copper Keel and Wildfire resource sub-domains defined and incorporated into larger MSD model May 2011;
- Updated resource for Minto South Deposit May 2011;
- Fireweed Prospect discovered by diamond drilling down dip to north from MSD;
- Updated Mineral Resource for Minto South Deposit December, 2011

6.2. Drilling

The project has been actively explored since the early 1970s. Companies controlled by ASARCO and Falconbridge drilled on the property in 1973 and 1974. All drill cores collected prior to 1993 were destroyed by forest fires. MintoEx completed further drilling programs between 1993 and 2001 before it was acquired by Capstone Mining Corp. and a further seven drill programs from 2005 to 2011 since Capstone's acquisition and control of MintoEx.

6.2.1. ASARCO and Falconbridge 1972 to 1974

Most of the drilling on the property is recent and was performed by the current operators of MintoEx and has resulted in significant new discoveries and resource additions. However, the initial discovery phase of exploration drilling was performed in the early 1970s by companies controlled by Falconbridge (United Keno Hill Mines Ltd.) and ASARCO (Silver Standard). Subsequent definition drilling by these operators was conducted once the Main deposit was discovered and exploration in the area continued sporadically until 2005 when the project was purchased by the current operators. The early project reports fail to detail their drilling procedures, but basic drilling procedures have unlikely changed little over time.

Early drilling was conducted with BQ drill rods, which return a core diameter of 1.43 inches. Within the main zone of the deposit, the drill hole density is on 100 ft centers on the DEF (Falconbridge) part of the deposit (locally as close as 50 ft), and generally on 150 ft to 200 ft centers on the Minto (ASARCO) side as illustrated in Figure 6—1.

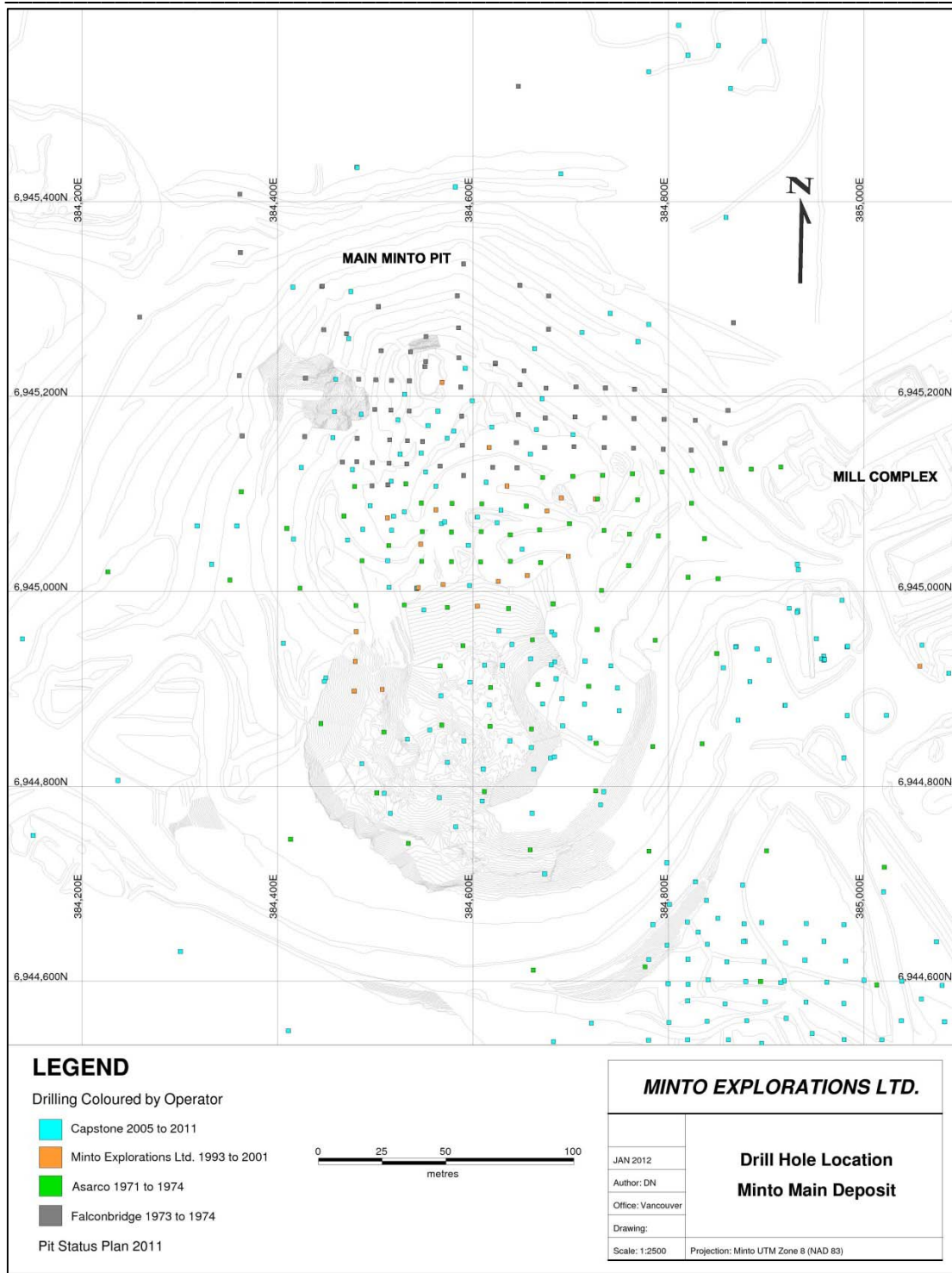


Figure 6—1: Asarco and Falconbridge Drilling Program

Falconbridge drilled 11 angled holes, while all other holes were drilled vertically. The average sample length for ASARCO is 2.4 m with the majority of samples being either 1.5 m or 3.0 m long. The average sample length for the Falconbridge drill holes is 1.5 m.

The locations of the holes were surveyed by Underhill Geomatics using a local grid controlled by local benchmarks. Prior to the commencement of pre-stripping of the Minto Deposit in 2006, the drill roads and pads for this drilling were still visible and the holes were often identifiable by casing and/or wooden posts protruding from the ground, although the labels were no longer attached or legible.

The core from this drilling was stored onsite in two core sheds. Over time the sheds have collapsed and/or have been burned out by wildfires, rendering most of the core unusable. In addition, the labels on the few remaining intact boxes are missing and/or are not legible.

In their compilation of the results, MintoEx has distinguished the ASARCO drill holes with an 'A' prefix and the Falconbridge hole with a 'K' prefix.

The results of this drilling have been instrumental in estimating the grade and tonnage of the deposit. The drilling was carried out using accepted practices of the time and is documented well enough to be reliable for the purposes of grade and tonnage estimations, particularly when compared to the results of subsequent infill drill completed by MintoEx in 1993-2001 and in 2005-2006.

6.2.2. MintoEx 1993 to 2001

MintoEx has carried out several diamond drilling programs for deposit definition drilling and exploration on the property in general, as follows:

1993

- 960 m drilled in eight holes (93 – A to H) within the deposit area to sample the two main mineralization types (foliated granodiorite and quartzofeldspathic gneiss) for metallurgical test work;
- Six of the holes were located to intersect the lower zone mineralization immediately below the main zone and one was designed to test deeper mineralization indicated in the 1970s drilling;
- The core was used for metallurgical testing and some of it was not split and assayed;
- Four of the holes were logged for magnetic susceptibility.

1994

- 2,185 m drilled in 19 exploration holes to test mineralization south of the main deposit;
- This drilling outlined a mineralized horizon roughly 6 m thick grading 2 – 3% Cu;
- One hole (94-17) filled in a large gap in the deposit area.

1995

- 572 m drilled in 6 holes: 425 m drilled in five exploration holes to test geophysical anomalies; and 160 m completed in one condemnation hole north of the proposed mill site;
- The exploration holes failed to intersect any anomalous mineralization.

1996

- 545 m completed in four condemnation holes in the area of the proposed west waste rock dump.

2001

- 552 m drilled in five confirmation holes within the proposed open pit area.

All the drilling on the project was contracted to E. Caron Diamond Drilling of Whitehorse.

The 1993, 1994, 1995 and 2001 programs utilized HQ core and the 1996 drilling was NQ core. This historical drilling was completed in the 1990s, prior to the legislation for NI 43-101. There was less regulatory scrutiny and results were the focus of reporting, rather than details of data collection. There is little in the way of documentation for the methods used in the pre-1990s drilling and sampling.

The 2001 drilling was subject to a rigorous report by both MintoEx (Minto Explorations Ltd., 2003) and ASARCO (Simpson, 2001), which loaned a geologist to the project to log and sample the core. The results of the 2001 drilling are discussed in the Data Verification section of this report. Some of the core from the 1993, 1996, and 2001 drilling programs is stored in the Ken Bostock Core Library in Whitehorse.

Some of the other core from the exploration on the property (away from the deposit) is stacked on site in behind the camp buildings. Older core was stored in sheds, which were burnt in a forest fire and is now unidentifiable.

6.3. Historic Resource Estimates

The Minto deposit has been subject to several historical tonnage and grade estimations, as summarized in Table 6-2.

These mineral resource estimates were based on up to 160 drill holes (totaling more than 25,000 m of drilling).

Table 6-2: Historical Tonnage & Grade Estimates of the Minto Deposit

Year	Source	Cut-off (%Cu)	Short Tons	Cu (%)	Au (oz/t)	Ag (oz/t)	Comments
1976	R.T Heard UKHM	unknown	8,219,370	2.04			
1976	L.A. Wigglesworth Falconbridge	unknown	8,210,219	2.03			
1975	R.J. Prevedi ASARCO	0.60	8,441,941	1.74			
1976	R.J. Prevedi ASARCO	unknown	7,220,900	1.86			

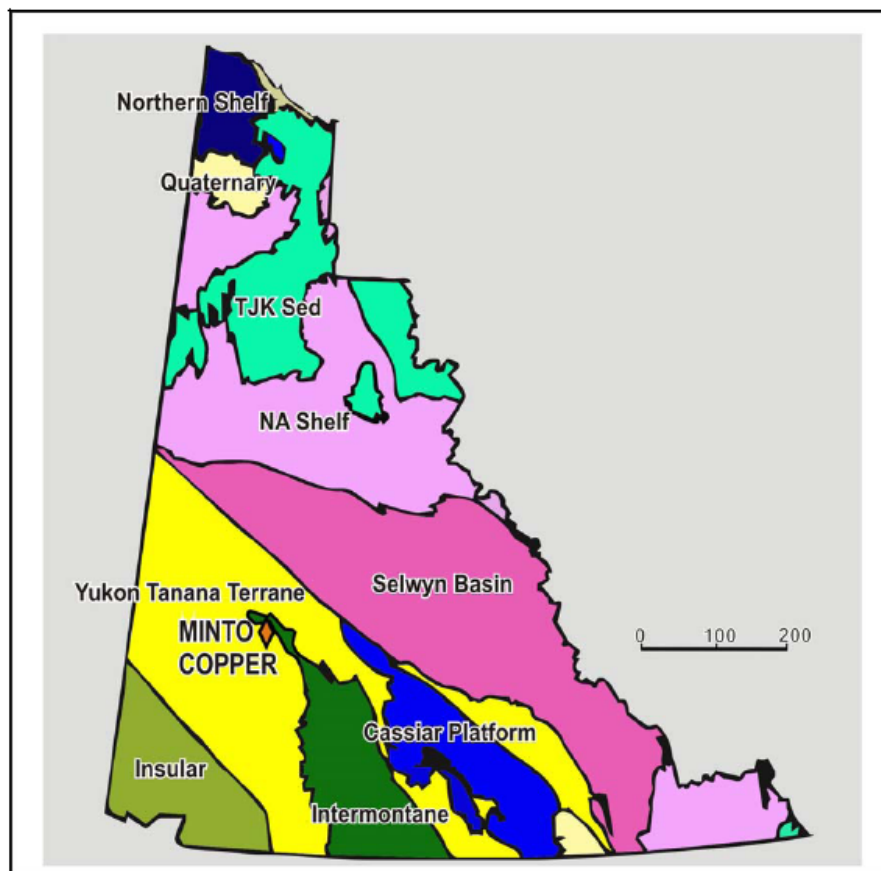
Year	Source	Cut-off (%Cu)	Short Tons	Cu (%)	Au (oz/t)	Ag (oz/t)	Comments
1980	D.M. Fletcher ASARCO	2.00	2,968,600	3.24	0.027	0.411	
1989	J. Proc & H.L Klingmann Minto Explorations	0.80	6,368,000	2.11	0.016	0.33	Open Pit and Underground Recovery at 75% and 5% dilution
1990	SRK/Falconbridge	unknown	7,592,318	1.88	0.016		Includes Lower Zone
1992	J. Proc & H.L Klingmann Minto Explorations	unknown	6,071,000	2.21	0.018	0.28	Open Pit and Underground UG = 1,600,000 ton @ 3.73% Cu, 0.038 oz/t Au, 0.49% oz/t Ag
1994	G. Giroux Montgomery Consultants	0.50	8,780,000	1.76	0.015	0.223	Pre 43-101 "proven" + "probable"

The estimates in Table 6-2 do not follow the required disclosure for mineral reserves and mineral resources (as outlined in National Instrument 43-101) because they were prepared prior to the inception of NI 43-101. The mineral resource estimates have been obtained by sources believed reliable and are relevant but cannot be verified. No effort has been made to refute or confirm these estimates and they can only be described as historical estimates. All historical estimates are superseded by more recent, compliant ones conducted by the Company. Mining operations on the Main Deposit have ceased.

7. Geological Setting and Mineralization

7.1. Regional Geology

The Minto Project is found in the north-northwest trending Carmacks Copper Belt along the eastern margin of the Yukon-Tanana Composite Terrain, which is comprised of several metamorphic assemblages and batholiths (Figure 7—1). The Belt is host to several intrusion-related Cu-Au mineralized hydrothermal systems. The Yukon-Tanana Composite Terrain is the easternmost and largest of the pericratonic terrains accreted to the Paleozoic northwestern margin of North America (e.g., Colpron *et al.*, 2005). It is regarded to be the product of a continental arc and back-arc system, preserving meta-igneous and metasedimentary rocks of Permian age on top of a pre-Late Devonian metasedimentary basement (e.g., Piercey *et al.* 2002).



From: Yukon Geologic Survey "Maps Yukon" website (www.geology.gov.yk.ca)

Figure 7—1: Yukon Geology (from Yukon Geologic Survey "Maps Yukon" website (www.geology.gov.yk.ca))

The Minto Property and surrounding area are underlain by plutonic rocks of the Granite Mountain Batholith (Early Mesozoic Age) (Figure 7—2: Regional Geology) that have intruded into the Yukon-Tanana Composite Terrain. They vary in composition from quartz diorite and granodiorite to quartz monzonite. The batholith is unconformably overlain by clastic sedimentary rocks thought to

be the Tantalus Formation and andesitic to basaltic volcanic rocks of the Carmacks Group, both of which are assigned a Late Cretaceous age. Immediately flanking the Granite Mountain Batholith, to the east, is a package of undated mafic volcanic rocks, outcropping on the shores of the Yukon River. The structural relationship between the batholith and the undated mafic volcanics is poorly understood because the contact zone is not exposed

Geobarometry and geothermometry data (Tafti and Mortensen, 2004) suggests that the Granite Mountain Batholith was emplaced at a depth of at least 9 km, while the presence of euhedral to subhedral epidote, interpreted by Tafti and Mortensen as magmatic in origin, suggests a deeper emplacement depth in the order of 18 to 20 km.

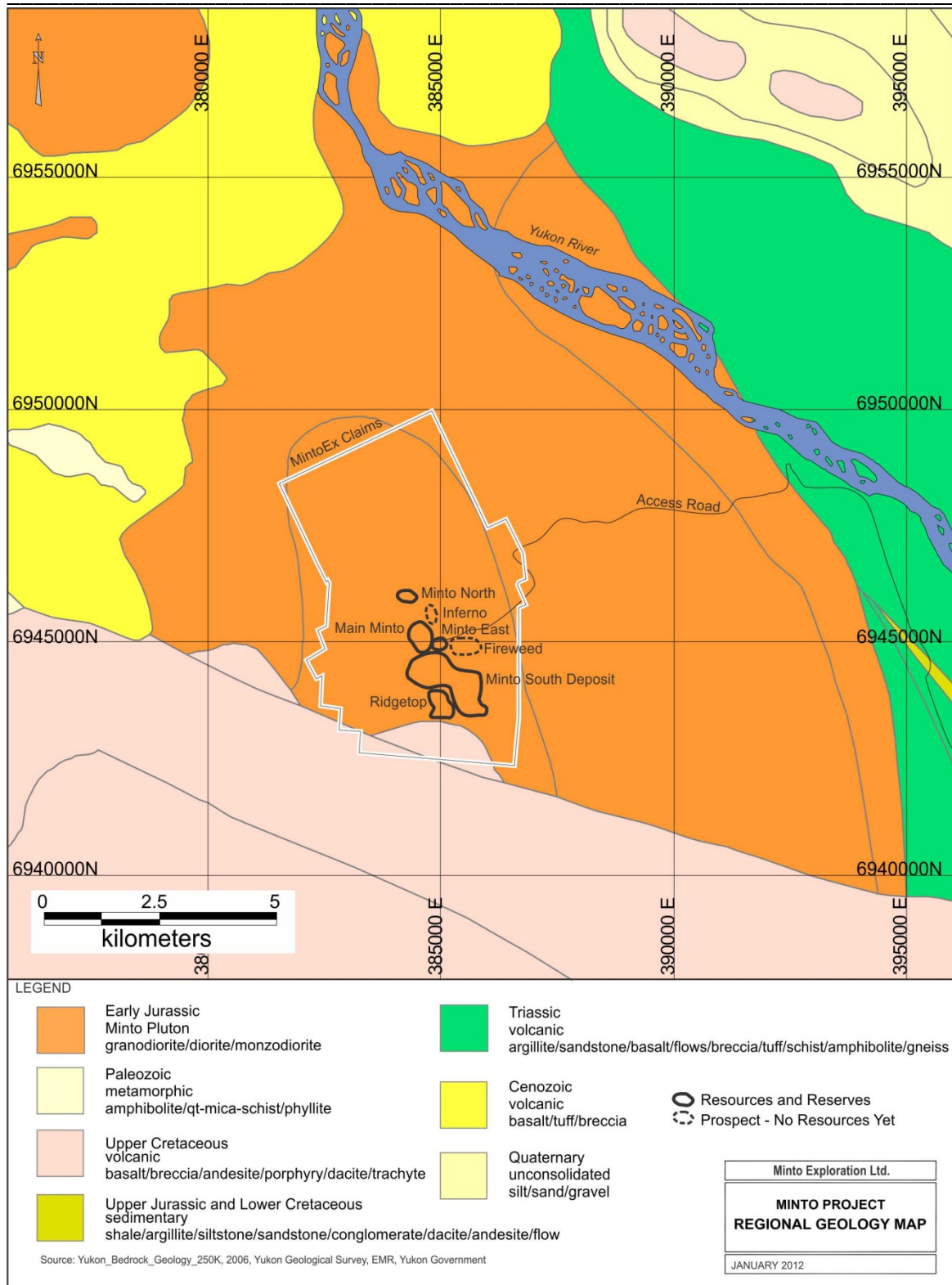


Figure 7—2: Regional Geology

7.2. Property Geology and Lithological Description

Much of the geological understanding of the rock around the Minto deposits is based on observations from diamond drill core and extrapolation from regional observations. The reason for this is poor outcrop exposure (less than 5% coverage), as well as the deep weathering and oxidation of any existing exposed outcrop. The terrain was not glaciated during the last ice age event.

Four additional deposits of mineralization are reported in this document outside of the Minto Main Deposit; Minto South Deposit, Ridgetop, Minto North, and Minto East (Figure 7—2). For the purpose of this report Area 2, Area 118, Copper Keel, and Wildfire resource sub-domains are now considered continuous and reported as one deposit, namely Minto South Deposit (MSD) located immediately south of Main Minto. The names of these resource sub-domains are still retained for ease of discussion when discussing regions within MSD. The Ridgetop deposit is located just over 300 m south of the Area 2/118 resource sub-domain, the Minto North deposit is located about 700 m north of the Main Deposit, the Minto East deposit is located about 200 m east of the south end of the Minto Main deposit, while the most recently discovered resource areas to be reported are Copper Keel and Wildfire forming a southeast extension from Area 2/118. Each of these deposits closely shares a similar style of mineralization of shallow dipping copper sulphide mineralized zones. The Main Minto deposit is exposed in the dormant open pit mine; similarly, this flat geometry is currently exposed in the Minto South Deposit (Area 2 resource sub-domain) starter pit. In addition to these mineral deposits which have NI 43-101 compliant mineral resources there are several significant mineral prospects. These deposits and prospects define a general north-northwest trend informally called the Priority Exploration Corridor or PEC. The most significant of these prospects are Fireweed, Airstrip, and Inferno.

The hypogene copper sulphide mineralization at Minto is hosted wholly within the Minto pluton, which intrudes near the boundary between the Stikinia and Yukon-Tanana terrains, however since the contact is not exposed it is unclear if the pluton stitches the two terrains. The Minto pluton is predominantly of granodiorite composition. Hood *et al.* (2008) distinguish three varieties of the intrusive rocks in the pluton. The first variety is a megacrystic K-feldspar granodiorite. It gradually ranges in mineralogy to quartz diorite and rarely to quartz monzonite or granite, typically maintaining a massive igneous texture. An exception occurs locally where weakly to strongly foliated granodiorite is seen in distinct sub-parallel zones several meters to tens of meters thick.

A second variety of igneous rock is quartzofeldspathic gneiss with centimeter-thick compositional layering and folded by centimetre to decimetre-scale disharmonic, gentle to isoclinal folds (Hood *et al.*, 2008). The third variety of intrusive is a biotite-rich gneiss. MintoEx geologists consider all units to be similar in origin and are variably deformed equivalents of the same intrusion.

Copper sulphide mineralization is found in the rocks that have a structurally imposed fabric, ranging from a weak foliation to strongly developed gneissic banding. For this reason all core logging by the past and present operators separates the foliated to gneissic textured granodiorite as a distinctly discernible unit. It is generally believed by MintoEx geologists that the foliated granodiorite is just variably strained equivalents of the two primary granodiorite textures and not a separate lithology.

While this interpretation, based upon detailed observations from logging of tens of kilometers of drill core is highly likely but it still needs to be conclusively proven. Tafti & Mortensen (2004) noted that the relatively massive plutonic rocks have similar mineral and chemical composition as the foliated rocks. Research in collaboration with the Mineral Deposits Research Unit (“MDRU”) of the University of British Columbia is on-going.

The contact relationship between the foliated deformation zones and the massive phases of granodiorite is generally very sharp. These contacts do not exhibit chilled margins and are considered by MintoEx geologists to be structural in nature, separating the variably strained equivalents of the same rock type. Tafti and Mortensen (2004) had interpreted the sharp contacts to be zones of deformed rock within the unfoliated rock (i.e. rafts or roof pendants). Supergene mineralization occurs proximal to near-surface extension of the primary mineralization and beneath the Cretaceous conglomerate.

Conglomerate and volcanic flows have been logged in drill core by past operators. New drilling has confirmed the presence of conglomerate, but not the volcanic flows. The latter cannot be confirmed by the authors as the drill core from historic campaigns was largely destroyed in forest fires and no new drilling has intersected such rocks. However, undated volcanic rocks are mapped by Hood, near the southwest margin of the property, south of a fault that is inferred from geophysics to separate them from the Jurassic Age intrusive rocks. The conglomerate has been dated (unpublished date pers. com. Dr. Maurice Colpron - Yukon Geological Survey) as Cretaceous Age. It is now recognized as an outcrop within a borrow pit exposure located west of the airstrip as well as in numerous recent drill holes. Observations of foliated and even copper mineralized cobbles in drilling indicate that “Minto-type” mineralization was exposed, eroded and reincorporated in sedimentary deposits by the Cretaceous Age.

Other rock types, albeit volumetrically insignificant, include dykes of simple quartz-feldspar pegmatite, aplite; and an aphanitic textured intermediate composition rock. Bodies of all of these units are relatively thin and rarely exceed the one meter core intersections. These dykes are relatively late, and observed contact relationships suggest they generally postdate the peak ductile deformation event; however some pegmatite and aplite bodies observed in a rock cut located north of the mill complex are openly folded.

It is unclear if this folding is contemporaneous with foliation development in the deformed rocks or post-dates the foliation development. Observations from drill core and open cut benches in the mine show examples where the foliation and the pegmatitic/aplitic intrusions are both folded, as well as examples where the intrusions are not folded, suggesting two populations of minor dykes.

7.3. Structure

There are both ductile and brittle phases of deformation around the Minto deposits. As noted above copper-sulphide mineralization is strongly associated with foliated granodiorite. This foliation is defined by the alignment of biotite in areas of weak to moderate strain and by the segregation of quartz and feldspar into bands in areas of higher strain, giving the rock a gneissic texture in very strongly deformed areas. The deformation zone forms sub-horizontal horizons within the more massive plutonic rocks of the region and can be traced laterally for more than 1,000 m in the drill core. They are often stacked in parallel to sub-parallel sequences. The regular,

sub-horizontal nature of the deformation zones allows for a high degree of predictability when planning diamond drilling campaigns.

Contrary to previous reports (Orequest, 2005), the foliated zones do not appear to inter-finger with the more massive rocks. Rather, it appears that blocks of unfoliated granodiorite are sometimes incorporated within the thicker deformation zones that surround them.

The similarity of chemistry and texture of both the deformed and massive granodiorite suggest the deformation zones are structural in origin and not stratigraphic. Several of these foliated units can be traced in drill holes over long distances at similar elevations.

While this could suggest either a structural or a stratigraphic origin for the foliated rocks it was noted that obvious plutonic textures were found in both the deformed and the massive rocks. However the absence of chill margins or absorption rims at contacts, combined with the great depth of emplacement (Tafti and Mortensen, 2004) likely preclude them from being remnant rafts or roof pendants of metasedimentary or metavolcanic strata, as some workers have postulated. No sedimentary or volcanic features have been observed in these foliated and mineralized rocks. A structural origin remains the best explanation.

It is therefore postulated that the foliated granodiorite horizons represent healed, shallowly dipping shear zones within the Granite Mountain Batholith, and may have formed when the rocks passed through the brittle/ductile transformation zone in the earth's crust in transition from a deep emplacement environment of the batholith to eventual exhumation. They may represent thrust faults related to regional crustal thickening of the Yukon-Tanana Terrain when the batholith was being exhumed.

Internally, the foliation exhibits highly variable orientations within individual deformation zones with the presence of small-scale folds. The foliation is often observed to be at a high angle to contacts with more massive textured rock units.

Observations by Hood *et al.* (2008) along a transect in the Area 2 resource sub-domain suggest that foliation orientations within deformed horizons have a geometry of tight to isoclinal folding with a wavelength on the order of about 30 m.

The observed trend of folds within this area is approximately northwest, parallel to regional structural trends (Tempelman-Kluit, 1984). The ore-bearing zones are also occasionally folded on a scale of several hundred meters. Based upon horizon modeling for resource estimation of Ridgetop the folds have wavelength of about 280 m. The folds appear to be gentle folds with north-south axial traces. Simple shear strain of the foliated zones is also noted adjacent to late cross-cutting fault zones.

Late brittle fracturing and faulting is noted throughout the property area. Some of these faults are significant from an economic standpoint. The Minto Creek fault (MC Fault) bisects the Minto Main deposit, dividing it into north and south areas and is modeled as dipping steeply north-northeast with an apparent left lateral reverse displacement. The northern block moved up and to the west relative to the southern block. Both the vertical and horizontal displacements are evident by offsets in the main zone mineralization and appear to be minimal. A lack of marker horizons in the plutonic rocks, however, makes it difficult to determine the absolute magnitude of the movement (Figure 7—3).

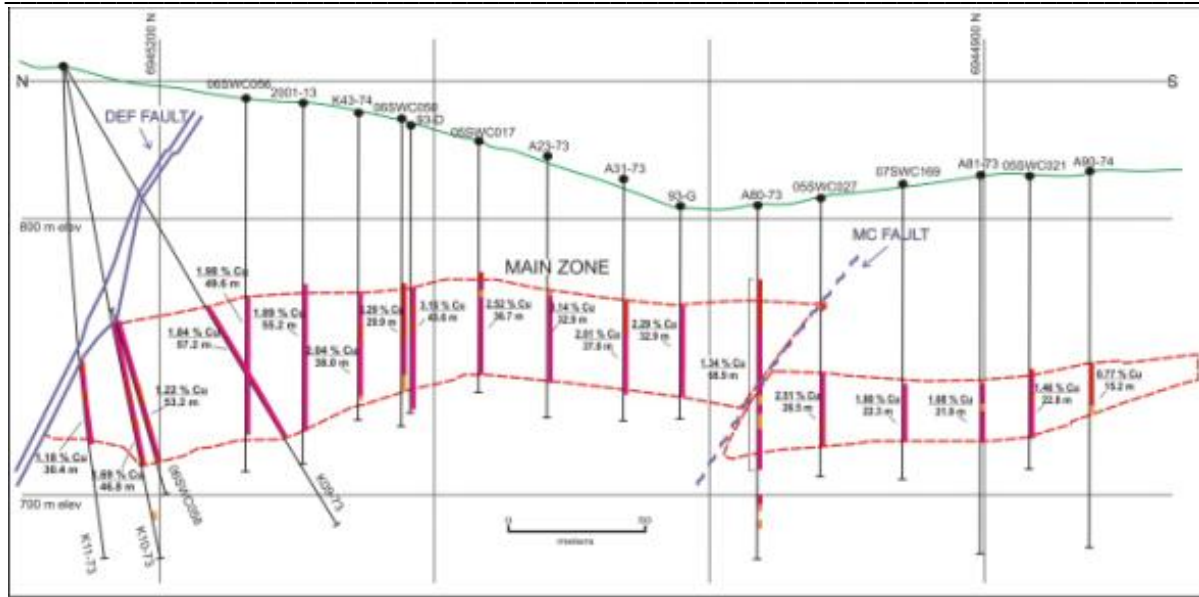


Figure 7—3: North-South Cross Section through Minto Main Deposit showing DEF Fault and MC Fault

The DEF fault defines the northern end of the Main deposit. It strikes more or less east-west and dips north-northwest and cuts off the main zone mineralization, as shown in Figure 7—3. The vertical orientation of most of the drilling is less than optimal to intersect steep to vertical faults. It may share a similar sense of movement to the MC fault, but a significant amount of displacement is inferred. Determining the magnitude of this displacement could lead to locating an extension of the main zone mineralization on the north side of the DEF fault. This late block faulting is noted throughout the Granite Mountain Batholith and in some instances a rotational component is noted as well. Tafti & Mortensen (2004) found the Cretaceous Age Tantalus Formation rotated up to 60° from horizontal in areas located south of the Minto deposit.

A zone of pervasive fracturing on the west side of the deposit limits ore grades in this direction. Limited historical drilling west of this structure did intersect some weak copper mineralization, although foliated horizons do not line up across this fracture zone. It is presumed to be one of the north-south faults that are part of the late brittle conjugate set.

While the limits to Minto Main mineralization on the north and west sides are structural in nature, the southern limit is an erosion channel cutting below the elevation of the mineralization and thereby removing it. This zone of deeper erosion is a paleo-channel that is interpreted to follow another roughly east-west striking fault. Only on the east side does mineralization appear to fade out and have no obvious structural limit.

The boundary between the Area 2 and Area 118 resource sub-domains is an intermediate NE dipping fault. The displacement of the mineralization is significant. At least two parallel structures displace mineralized domains in Area 118.

The shear sense on this structure has not been analyzed in detail, but attempts to correlate ore zones across the main boundary fault are complicated by the difficulty in finding a specific characteristic to unambiguously identify the zones. The easiest zone to identify (based on mineralization and texture) is the “N” zone, which has up to 66 m of vertical throw across the boundary fault. Other zones show changes in thickness and orientation, suggesting the presence

of pure strain and block rotation. A better structural model is required. A similar NW striking fault zone appears to be present that defines the northeastern boundary of the Ridgetop deposit and the southwest extent of the Wildfire resource area, and defines the outcrop of Cretaceous conglomerate in both regions. The dip of this structure is unknown.

All mineralized horizons exhibit locally pervasive fracturing (typically chloritic or hematitic), which are interpreted to postdate the main copper-sulphide mineralization event. This late structural/hydrothermal event may have potential economic significance, as coarse-grained visible gold has been logged on chloritic fractures.

7.4. Veining

Veins in the Minto Deposit appear to have been emplaced after the copper sulphide mineralization and are therefore not economically significant. The most common veins are very narrow (less than 30 cm) steeply dipping, simple quartz-feldspar pegmatite veins that often contain cavities that are indicative of shallow emplacement. The veins crosscut foliation in the deformation zones and the sulphide mineralization; evidence of their post sulphide mineral emplacement. Other types of late veins found in the deposit include thin (less than 2 mm) calcite, epidote, hematite and gypsum stringers, and fracture coatings. Quartz veining is extremely rare and economically insignificant.

7.5. Mineralization

The Minto deposits have essentially no surface exposure with the exception of minimal exposure in historical trenches of the shallow partially oxidized zones associated with the Ridgetop deposit. Observations for the deposits are therefore based almost entirely on hand-specimen and petrographic studies of drill core. The primary hypogene sulphide mineralization consists of chalcopyrite, bornite, euhedral chalcocite, and minor pyrite. Metallurgical testing also indicates the presence of covellite, although this sulphide species has never been positively logged macroscopically. Texturally, sulphide minerals predominantly occur as disseminations and foliaform stringers along foliation planes in the deformed granodiorite (i.e. sulphide stringers tend to follow the foliation planes). However, sulphide mineral content tends to increase where this foliation is disrupted by intense folding. In addition, semi-massive to massive mineralization is also observed; this style of mineralization tends to obliterate the foliation altogether. Silver telluride (hessite) is observed in polished samples but has not been logged macroscopically. Native gold and electrum have both been reported as inclusions within bornite and accounts for the high gold recoveries in test copper concentrates. Occasionally, coarse free gold is observed associated with chloritic or epidote lined fractures that cross-cut the sulphide mineralization. The free gold may be due to secondary enrichment during a later hydrothermal process overprinting the main copper sulphide-gold event. Sulphide mineralization is almost always accompanied by variable amounts of magnetite mineralization and biotite alteration. While these minerals occur in the non-deformed rocks they are present in the mineralized horizons in a much greater abundance in the range of an order of magnitude greater than background.

The Minto Main deposit exhibits crude zoning from west to east. The bornite zone is dominant in the west while a thicker, lower grade chalcopyrite zone is dominant on the east side of the deposit. The bornite zone is defined by the metallic mineral assemblage magnetite-chalcopyrite-

bornite. Bornite mineralization is conspicuous, but chalcopyrite is the dominant sulphide species. Stringers and massive lenses of chalcopyrite with various quantities of bornite are typical. Massive mineralization occurs locally over intervals exceeding 0.5 m in thickness and semi-massive mineralization over several metres in thickness may occur. In these sulphide rich areas, textures often resemble those seen in magmatic sulphide zones with sulphide mineralization interstitial to the rock forming silicate minerals. The higher grade portion of the Minto Main deposits roughly corresponds to the bornite zone. Local concentrations of bornite of up to 8% are seen. The precious metal grades are elevated in the bornite zone (very fine gold and electrum occur as inclusions in bornite) and occurrences of coarse grained native gold are noted almost exclusively in bornite-rich material. The chalcopyrite zone is characterized by the metallic mineral assemblage of chalcopyrite-pyrite $\pm$ very minor bornite and magnetite.

Empirical observations indicate the highest concentrations of bornite are associated with coarse grained, disseminated and stringer-style magnetite mineralization, up to 20% by volume locally. The stringers of magnetite are often folded or boudinaged, suggesting that at least some of the magnetite mineralization predates peak ductile deformation.

Sulphide mineralization on the other hand, shows both evidence and absence of ductile deformation locally and is interpreted to have formed contemporaneous with, or late in the ductile deformation history.

The Minto North and Minto East Deposits also exhibit a zoning from west to east. High-grade bornite-dominant mineralization is observed in the west with lower grade chalcopyrite-dominant mineralization in the east. The bornite zone is defined by the metallic mineral assemblage bornite-magnetite-chalcopyrite. Bornite mineralization occurs as strong disseminations and foliaform stringers locally >10% to occasional semi-massive to massive lenses up to 2 m in thickness. Chalcopyrite concentrations are typically within the 1 to 2% range, but locally can reach concentrations of 10%. Precious metal grades are elevated in the bornite zone, and visible gold has been observed on occasion.

Mineralization at the Area 2/118/Copper Keel resource sub-domains is distinct in that mineralization is predominantly disseminated (plus occasional foliaform stringers) and that semi-massive to massive sulphide mineralization is absent; as a whole, the mineralization is more homogenous and consistent as compared to Minto Main and Minto North. The primary mineral assemblage in the Area 2/118/Copper Keel resource sub-domains includes chalcopyrite-bornite-magnetite with minor amounts of pyrite; and a crude zoning is present in that the higher grade northern half of the Minto South Deposit shows increased bornite concentrations up to 8% locally.

Mineralization at both the Ridgetop deposit and the Wildfire resource sub-domain are subdivided into the near surface horizons that have been affected by supergene oxidation and the more typical primary sulphide mineralization of the deeper zones. The lower zones are defined by a mineral assemblage of chalcopyrite-magnetite with minor amounts of pyrite. Chalcopyrite is the dominant sulphide in the lower zones, and bornite is only observed in minor amounts. Texturally, chalcopyrite occurs as disseminations and foliaform stringers, and is rarely observed as semi-massive to massive bands. Magnetite is coarse grained, disseminated, stringer-style, and can occur in bands up to 0.3 m in thickness, up to 20% volume locally.

These empirical observations of bornite/chalcopyrite relative abundances are supported by a copper and gold grade trend in mineral resources discovered to date within the PEC where the Ridgetop deposit and the southern extent of the Minto South Deposit sit at the lower grade southern end and Minto North sits at the much higher grade northern end of the currently defined trend.

7.6. Alteration, Weathering and Oxidation

Pervasive, strong potassic alteration occurs within the flat lying zones of mineralization, and is the predominant alteration assemblage observed in all of the Minto deposits. The potassic alteration assemblage is characterized by elevated biotite content and minor secondary k-feldspar overgrowth on plagioclase relative to the more massive textured country rock. Biotite concentrations range up to 30 to 70% by volume locally, compared to about 5 to 8% in waste rock. Additional alteration includes the replacement of mafic minerals by secondary chlorite, epidote, or sericite observed both in mineralized and waste rock interstitially or fracture/vein proximal, as well as variable degrees of hematization of feldspars. Uncommon but locally pervasive sericite-muscovite alteration is observed associated with post-mineral brittle faults; this type of alteration is most common within the Area 2/118 resource sub-domains.

Hematization is the most pervasive at the Minto Main deposit proximal to the DEF fault, whereas in the other deposits it is predominantly fracture controlled within narrow alteration selvages. It is interpreted to be supergene in origin. Minor carbonate overprint is occasionally observed associated with secondary biotite. The contacts between the altered and unaltered rocks are sharp, as are the contacts between mineralized rocks and waste rocks.

Silicification is present but not pervasive nor uniform in distribution in the Minto deposits. At Minto Main, Minto East, and Minto North it is sporadic within the bornite zone (west) and lacking in the chalcopyrite zone (east). Within the Area 2/118/Copper Keel resource sub-domains, silicification intensity is variable in all ore zones. On rare occasions, silicification is pervasive enough to almost entirely overprint both primary and deformation textures (Area 2) while it is essentially absent at Ridgetop and Wildfire. The relationship between silicification and the mineralization is unclear due to inconsistent core logging over three decades, although in most cases higher grade sulphide mineralization is coincident with silicification.

Copper oxide mineralization, like the hematization seen at surface in float, trenches, and in the upper mineralized zones at Ridgetop and Wildfire is the result of supergene oxidation processes. This near surface mineralization represents either the erosion remnants of foliated horizons that are located above the deposits or is vertical remobilization of copper up late brittle faults and fracture zones that intersect primary sulphide mineralization at depth. Chalcocite is the prime mineral in these horizons along with secondary malachite, minor azurite, and rare native copper. The mineralization is found as fracture fill and joint coatings and more rarely interstitial to rock forming silicate minerals.

At the Ridgetop deposit and the Wildfire resource sub-domain, the upper near surface mineralized zones are unique in that the dominant oxide facies mineral is the sulphide chalcocite rather than chalcopyrite or bornite, and it is believed to be a secondary supergene enrichment associated with a paleo water table, or fault proximal oxidation via circulating groundwater. Minor malachite,

azurite, remnant chalcopyrite-bornite, and native copper are also present within these near surface mineralized zones.

Cobbles and pebbles of this supergene chalcocite mineralization in Cretaceous age (unpublished data) conglomerate that unconformably overlies the plutonic rocks of the Granite Mountain Batholith indicate that the upper parts of the Minto System were on surface and being partially oxidized and eroded in the Late Cretaceous.

In addition to the obvious copper oxide minerals, oxidation is also evident by pervasive iron staining (limonite), earthy hematite, clay alteration of feldspars, and a significant loss in bulk density. The degree and distribution of copper oxide minerals appears to be directly related to the depth of the water table. For the most part this is confined to about -30 m (but up to -90 m) beneath the surface and is generally sub parallel with the present topographic surface. The Minto Main zone has experienced relatively little oxidation since it is generally more than 60 m below the surface except at its southern end where it crops out directly beneath unconsolidated overburden in the Minto Creek Valley. Very locally this oxidation may be drawn deeper along late brittle faults cutting primary sulphide mineralization.

7.7. Additional Mineralization Targets

The most favorable exploration targets (based on the evaluation of geophysics, soil geochemistry, geologic modeling, and diamond drilling are summarized below. The targets identified as Ridgetop Southwest, , Airstrip, MSD GAP, and the newly discovered Fireweed prospect are all located within a 2 km by 2 km area, south of the DEF fault. MintoEx also sees good exploration potential in the area north of the DEF fault, as evidenced by the discovery of the high grade Minto North deposit early in 2009 and the recently discovered Inferno prospect in late 2010 as well as the presence of multiple Titan-24 DCIP anomalies.

Also in 2009, several other historic bedrock copper occurrences discovered in the 1970s north of the DEF fault were relocated and confirmed. In addition various copper-in-soil geochemical anomalies, often coincident with magnetic geophysical anomalies, occur throughout the property and many of them remain untested. However, further understanding of the bedrock geology north of the DEF fault is required before many of these targets can be properly assessed and placed in perspective. Various exploration targets that MintoEx geologists identify as having potential are identified in Figure 7—4 and select targets are described in more detail below.

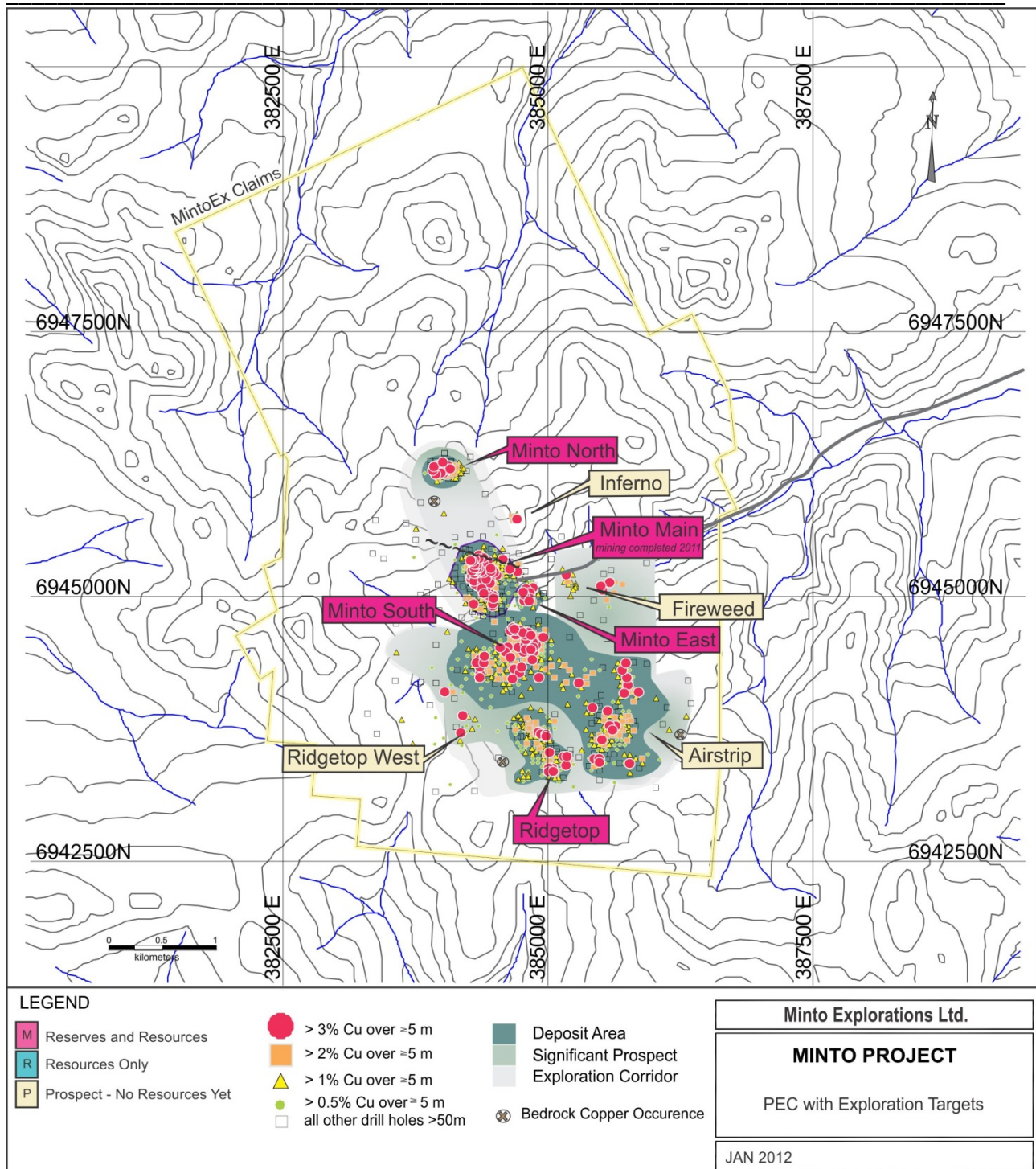


Figure 7—4: Exploration Targets

7.7.1. Fireweed

This target, which spans an area 200 m to 700 m east of Minto East, was identified by drill testing a Titan-24 anomaly in 2011 correlating with an interpreted down-dip northern extension of the Minto South Deposit geology model (Copper Keel resource sub-domain). Currently the target is divided into two areas denoted as Fireweed-A and Fireweed-B. The Fireweed-A chargeability anomaly measures 180 m north-south by 180 m east-west, and is centered below the drystack tailings conveyor infrastructure at 700 m elevation; however the majority of the Fireweed-B target is not coincident with any geophysical anomaly. Previous drilling in the area was restricted to historical Asarco drilling conducted in the 1970's, with the majority of drill holes not testing to the required depth. The most significant historical drill holes include A127-74 which intersected the shallow mineralization at the western flank of the target, and A125-74 which intersected moderate depth mineralization at the eastern flank of the target.

Discovery drill hole 11SWC859 targeted a down-dip extension of the Minto South Deposit coincident with the Titan-24 anomaly at 700 m elevation; intersecting both shallow, low-moderate grade mineralization and moderate-depth, moderate-high grade mineralization. Follow-up drill holes 11SWC860 and 861, stepping out 40m to the north and east respectively, successfully intersected the same horizons as the discovery hole solidifying Fireweed as a bona fide exploration target. Shortly thereafter, larger step-outs identified higher grade mineralization at depth, subsequently expanding the size of the prospect; the most important drill hole being 11SWC866 at Fireweed-B not associated with any geophysical anomaly.

Systematic drilling on east-west oriented sections across the Wildfire target commenced with the goal of establishing the spatial extent of the mineralization. Drilling to date indicates that in some areas the mineralized horizons lack continuity and coherence; this appears to be the result of offsets from the numerous fault zones logged to date. MintoEx geologists believe that the faulting is related to the Minto creek fault or splays of this fault. In some instances it appears that mineralization may be stepped down moving northerly or easterly associated with these faults; effectively dismembering the zones into wedges or slivers of mineralization and making it more difficult to determine the true potential of the prospect without further drill testing and modeling of faults. With the ultimate goal to identify zones that are still largely intact probably more so in an east-west direction; one possibility is that the mineralized zones may have been rotated from the typical "Minto north-northwest trend" to the apparent east-west trend as suggested by current drill results .

A total of 26 holes (10,826 m) were drilled at the Fireweed target in 2011; the results of the exploration drilling indicate a stacked system of mineralization similar to other Minto deposits with the best grades observed at moderate to deep depths. The shallow mineralization at Fireweed-A is principally comprised of chalcopyrite, along with lesser bornite, chalcocite, and trace malachite. Deeper mineralization at Fireweed-A and B is comprised of chalcopyrite and bornite, typical of Minto-style hypogene mineralization.

The additional potential of the Fireweed prospect will be evaluated when step-out drilling resumes in early 2012.

Significant assay results for Fireweed are presented in Table 7-1.

Table 7-1: Select Assay Interval Highlights from Fireweed Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
A125-74	185.3	195.4	10.1	2.14	0.88
11SWC859	216.7	224.9	8.2	1.16	0.28
11SWC859	318.0	327.2	9.2	1.50	1.03
11SWC863	136.1	142.9	6.8	1.11	0.22
11SWC873	357.5	378.4	20.9	1.07	0.47
11SWC882	397.4	411.8	14.4	3.35	1.42
11SWC884	393.4	406.7	13.3	3.04	1.87
11SWC886	395.5	412.5	17.0	1.34	0.61

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 6-1 are to be near true widths.*

7.7.2. Inferno

This target, which lies immediately north of the DEF fault and about 150 m northeast of the Minto Main deposit, was initially identified by a Titan-24 anomaly in 2009. Initial test drilling of the target in 2 holes did not intersect any significant copper-gold mineralization, and MintoEx geologists believed that the position of the anomaly was in question similar to what was observed at Minto East. Upon the completion of the expanded 2010 Titan-24 survey, the center of the anomaly was better constrained and shifted about 150 m to the northeast of the original location.

The first drill hole 10SWC718 targeted the center of the 500 m by 200 m anomaly at 600 m elevation and intersected high-grade copper-gold mineralization at depth. A mise-a-la-masse down hole IP survey was immediately carried out, and the results indicated a strong southern vector. Follow-up drilling in 3 separate holes confirmed the continuity of a shallow, moderate grade zone underlain by a narrower, high grade zone in a 40 m by 40 m area.

In 2011, step-out drilling in 4 holes (1,646 m) did not return any significant results. An additional mise-a-la-masse downhole IP was completed, with the results suggesting conflicting chargeability vectors than the original 2010 downhole vectors.

MintoEx favours this area as a drill target as it is positioned in the gap between the Minto Main and Minto North deposits. Furthermore, this appears to be the best candidate for the off-faulted northern extension of the Minto Main deposit. The Inferno prospect will be evaluated when drilling resumes in early 2012.

Significant assay results for Inferno are presented in Table 7-2.

Table 7-2: Select Assay Interval Highlights from Inferno Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
10SWC723	61.9	75.9	14.0	1.14	0.59
10SWC746	31.6	61.4	29.8	0.82	0.57
10SWC746	278.3	282.3	4.0	3.19	1.83
10SWC750	274.2	276.3	2.1	19.39	5.49
10SWC750	330.6	345.0	14.4	1.07	0.57

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 7-2 are to be near true widths.*

7.7.3. MSD Gap

This target, which lies in the gap between the Minto South and Ridgetop deposits, is currently considered a high priority target. In total, 13 holes exist in this region including both historical and more recent drilling. Results of drilling to date have identified prospective mineralization at shallow to moderate depths. MintoEx geologists believe that there may be continuity in the foliated granodiorite zones between these deposits, and will look to further test this target in early 2012.

A summary of historical drill results are presented in Table 7-3.

Table 7-3: Select Assay Interval Highlights from MSD GAP Drilling

DDH ID	From (m)	To (m)	INT (m)*	Cu (%)	Au (g/t)
08SWC378	257.1	263.3	6.2	1.00	0.26
10SWC705	135.0	141.2	6.2	1.97	0.60
10SWC708	125.2	129.5	4.3	1.27	0.04
11SWC783	111.1	117.8	6.7	1.27	0.54

**Geological modelling shows that the best continuity between drill holes indicates horizontal to sub-horizontal mineralized horizons. Therefore the intervals indicated in Table 7-3 are to be near true widths.*

8. Deposit Types

Each of the deposits reported in this technical report are considered to have the same style of mineralization as the Minto Main deposit. The copper sulphide mineralization is associated with sub-horizontal, sub-parallel foliated horizons within a granodioritic pluton. MintoEx have engaged the MDRU of the University of British Columbia to help understand the nature of mineral paragenesis and deformation history at Minto. This research is now complete, and final modifications to the associated MSc thesis are in progress.

At various times since its discovery the Minto deposit has been described as an example of Porphyry Copper, Volcanogenic Massive Sulphide (VMS), Redbed Copper, Magnetite Skarn (see discussion by Pearson and Clark, 1979) and Iron Oxide Copper Gold "IOCG" (Minto Explorations Ltd., 2003). Based on the preceding paragraph it is reasonable to say that the origin of the Minto deposit is enigmatic. Various workers (including the current authors) appear to have ascribed different interpretations for the most part based on their empirical observations, the background of the observer and the popular models of the day. The abundance of the high Cu/S mineral bornite in a moderately oxidized magmatic system along with the obvious magnetite association suggests that Minto belongs to one of two recognized deposit types: Magnetite Skarn or Iron Oxide Copper Gold ("IOCG"). The lack of a typical calc-silicate skarn mineral assemblage seems to preclude the skarn deposit type, this appears to leave the IOCG model or alternatively it belongs to a previously unrecognized deposit type.

The host rocks to the Minto deposit were emplaced in a deep batholithic setting (exceeding 9 km deep to perhaps as much as 18-20 km deep), which is not considered to be the typical porphyry environment. The host is a moderately oxidized magma (Tafti and Mortensen, 2004) with widespread iron oxide (magnetite and hematite) mineralization. At least some of the hematite is supergene in origin but it is unclear if some hematite is also primary. There are very strong structural controls on ore mineral emplacement and there is no apparent genetic link to a specific phase of intrusion. Typical porphyry-type alteration zoning such as widespread propylitization, argillization, barren silicic core, or large barren pyritic halo is not recognized. Stockwork style, fracture or vein mineralization is also not present.

MintoEx geologists have been advised that some examples of IOCG mineralization (in personal communications) exhibit some similar characteristics and setting to Minto including Copperstone in Arizona, Candelaria in Chile, and Ernest Henry in Australia (Williams et al., 2005). From a genetic and structural prospective, albeit not size wise, the Sossego Deposit in Brazil may be a reasonable analog. While an IOCG origin for the Minto Deposit cannot be unequivocally demonstrated, MintoEx geologists are of the opinion that this style of deposit provides the most consistent model for their current level of understanding. However, the unique nature of this mineralization style and apparent lack of close analogs elsewhere suggests the Minto Copper-Gold deposits may represent an unrecognized mineral deposit type.

9. Exploration

Mineral exploration on the Minto property has been conducted intermittently since 1971. Subsequent to the discovery of the Main deposit, now a mined out open pit, the adjacent southern half of the property has undergone systematic brownfields exploration. Exploration on the northern half is more sporadic. There are currently more than 1,100 drill holes within a roughly 16 square kilometre area. As such, following up on open mineralized horizons in geological models, projecting mineralized horizons into areas of little or no drilling, and drilling near historical drill hole intercepts were the principal exploration tools employed by MintoEx and its geologists. Subsequent to Capstone's predecessor, Sherwood Copper's acquisition of Minto Explorations Ltd. in June 2005, exploration from 2005 to 2011 has concentrated mostly on diamond drilling. However, an extensive historic soil sample survey and some ground based and airborne geophysics have been conducted and are very useful to guide drilling activity.

The current approach by MintoEx is the systematic evaluation of modern electrical (chargeability), geophysical methods by commissioning various "proof-of-concept" surveys over known mineralization and then expanding survey coverage outward into untested areas using these methods that are calibrated to known deposits. An emphasis is placed on looking for signature analogs as opposed to being pedantic about precise measurements of response. The predominant electrical geophysical methods used are Gradient Array Induced Potential (GAIP), Dipole-Dipole Induced Potential, and Titan-24 DC Induced Potential. Drill targeting is predominantly based upon the coincidence of an anomaly in one of the electrical (chargeability) methods with an anomaly in the 1993 total field airborne magnetic survey (MAG). Within the currently known extent of the PEC, future exploration programs will likely be more reliant on deep penetrating electrical / chargeability methods as the near-surface potential and discrete magnetic bull's-eyes have largely been targeted. Magnetic data in areas located north of Minto North plus areas west and east respectively of the PEC may still be useful as these regions are still relatively under explored. Local test surveys of Bouger gravity over the Main deposit and horizontal loop electromagnetics (HLEM) over the Area 2 deposit failed to detect the mineralization and proved to be of little use, they were not conducted over other areas.

In a cycle of discovery and definition, new deposits have now been identified by diamond drilling in four separate areas outside of the original or Minto Main deposit that was known when the project was acquired in 2005. The new deposits include Minto South (Area 2 resource sub-domain discovered in 2006, Area 118 resource sub-domain discovered in 2007, Copper Keel sub-domain discovered in 2006, Wildfire resource sub-domain discovered in 2010), Ridgetop drilled for the first time by MintoEx in 2007, Minto East discovered in 2007, and Minto North discovered in 2009.. Also, as described in the previous section there are multiple other prospects distributed throughout the property; Inferno was discovered in 2010 and Fireweed in 2011. The focus of exploration since 2005 involves systematic exploration of the property area both south and north of the current open pit mine in a south-southeast to north-northwest striking trend MintoEx calls the Priority Exploration Corridor described ("PEC") (Figure 9—1).

A brief chronological summary of work conducted on the property is contained in the history section of this report and is also described in the "Technical Report (43-101) for the Minto Project"

by Hatch (August 2006) and “Area 2 Pre-feasibility Study Minto Mine, Yukon” (November 2007) found on the [sedar.com](http://www.sedar.com) website.

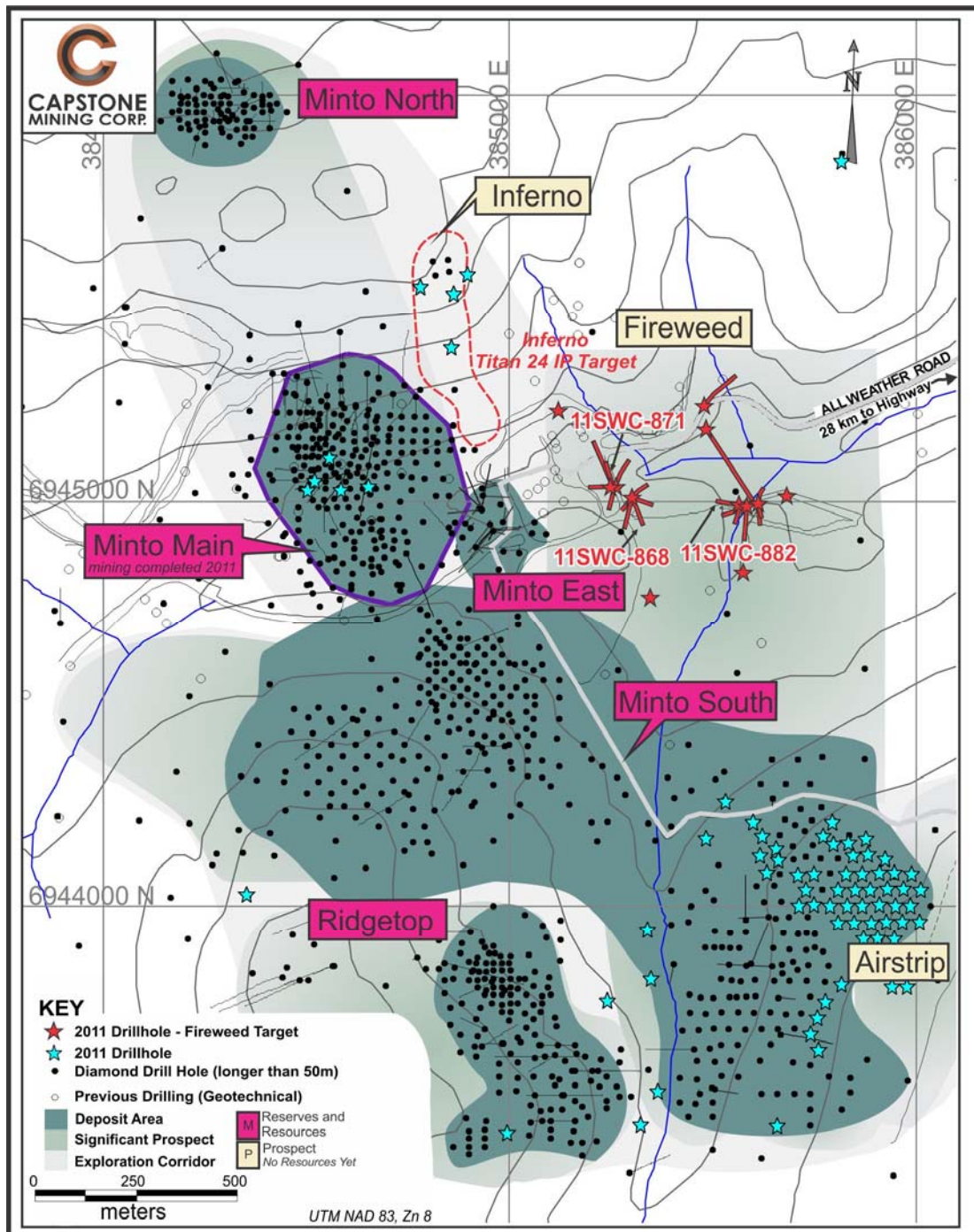


Figure 9—1: Priority Exploration Corridor (PEC) with Drill Collars Current to December 31, 2011

In 2008 and 2009, 61 additional infill and margin step-out drill holes into the Area 2/118 resource sub-domain allowed for the completion of a NI 43-101 resource estimation that was released June 9, 2009. In 2010, 22 additional infill and southern margin step-out drill holes into the Area 2/118

resource sub-domain lead to a more robust NI 43-101 resource estimation that was released August 30, 2010. The results of the 2010 drilling effectively linked the deeper mineralization in the southeast portion of Area 2/118 to the mineralized zones at Copper Keel North.

MintoEx geologists reassessed the Ridgetop area in 2007 (ASARCO's original Area 1 or Main discovery area) and drilled 25 new diamond drill holes, following up on 16 historical holes between the 1970's and early 1990's. The subsequent interpretation and drill density allowed for the completion of an NI 43-101 compliant resource estimate for Ridgetop East released December 12, 2007. In 2008 and 2009, 116 additional infill and step-out drill holes into the Ridgetop Deposit lead to a more robust NI 43-101 compliant resource estimation, which was released June 9, 2009. Subsequent follow-up drilling in the fall of 2009 totalling 40 drill holes initiated another NI 43-101 compliant resource estimate that was released August 20, 2010.

Early in 2008, a limited program of drilling in the overburden filled upper area of the Minto Creek valley identified several previously unknown areas of copper-gold mineralization now considered prospective. These discoveries are totally blind to surface, not discernible with GAIP surveys, have very muted magnetic high signatures and are essentially wildcat discoveries. Geological modelling at the western edge of the PEC at West Ridgetop and the western margins of Area 118, suggested the mineralized horizons may continue westward and dip beneath upper Minto Creek, expanding the Priority Exploration Corridor.

In 2009, MintoEx geologists followed up on two historic drill holes K88-74 and K91-74, that were originally collared to test a historic geophysical anomaly with a similar signature to the Minto Main deposit. Both drill holes failed to intersect any significant copper mineralization. The current 3D model now shows that one angled hole from 1974 drilled from the north passed beneath the main Minto North horizon, narrowly missing the discovery. A geology report dating from 1974 in the MintoEx archives indicates the two holes were designed to test an IP feature.

The 1974 report suggests that the geophysical anomaly must have been misallocated in error. A more modern (2007) GAIP survey places the chargeability anomaly approximately 90 m further south than the historic anomaly. Drill testing based upon this new data resulted in the discovery of Minto North in 2009.

The first drill hole at Minto North 09SWC390, collared in the center of both the GAIP and MAG anomalies, intersected high-grade, near surface, Minto-style mineralization. The discovery drill hole was followed up by two additional preliminary step-out holes 09SWC392 and 09SWC393, that also intersected significant mineralization. MintoEx geologists now understand that the 1974 vertical drill hole K88-74 completely missed the deposit, and that angled drill hole K91-74 drilled underneath the deposit.

Upon the confirmation of the high-grade mineralization by assays, the new northern target was denoted as Minto North, and plans were made for additional step-out and possible infill drilling. After the first phase of step-out and infill drilling was completed April 13, 2009 a preliminary resource estimated was released on June 9, 2009. Shortly after, another infill program was completed by August 6, 2009 leading to the NI 43-101 compliant resource estimate completed June 9, 2009 contained herein. A total of 87 drill holes are included in the resource estimate reported herein.

The drilling at Minto North in 2009 returned some of the best copper mineralization intersected to date on the property. Similar to the Minto Main deposit, Minto North displayed a zoning from high-grade bornite dominant mineralization in the west to lower grade bornite + chalcopyrite mineralization in the east. The high-grade bornite-rich core also returned excellent gold grades, and in some cases visible gold was observed along epidote lined fractures.

The Minto East target was initially identified during the 2007 drilling in the gap between the Minto Main deposit and Area 2/118 deposit. A drill program was designed drill hole 07SWC176 collared approximately 200 m east of the southeast corner of the Minto Main deposit intersected 11.7 m of high grade copper-gold mineralization that looked remarkably similar to the Minto Main deposit mineralization, including abundant stringers of massive chalcopyrite. At the time, MintoEx geologists suspected that this intersection was the extension of the deep mineralization seen at Area 2. In 2008, a second drill hole (08SWC286) was collared approximately 120 m south-southeast of 07SWC176. This hole intersected mineralization at the anticipated depth although it was narrow in width and only moderate grade. The target remained dormant until 2009 when a geophysical survey (Titan-24) identified a sizable DCIP chargeability anomaly in the area at the right elevation.

The deep penetrating Titan-24 survey returned a chargeability anomaly spanning a minimum of 180 m long by 180 m wide being strongest at 600 m elevation. However because the anomaly was located only on one line on the easternmost flank of the survey it was poorly constrained. The first drill hole in 2009 drilled nearly on the geophysical survey line 09SWC583 intersected only a narrow zone.

Because the Titan-24 survey was a localized test of the technology, it was suspected the source of the anomaly was due to mineralization located some distance off the survey line. Drill holes 09SWC584, 09SWC586, and 09SWC591 were collared further east and returned excellent copper grades and thickness' confirming Minto East as a bona fide exploration target. A down hole geophysical survey in 09SWC584 produced a southern vector that was the focus of initial follow-up drilling in 2010.

In 2010, an additional 27 holes at Minto East defined the deposit and lead to a robust NI 43-101 compliant resource estimate released August 30, 2010. This drilling essentially cut-off the mineralization in all directions, but is marginally still open in the south towards the east side of Area 2. Similar to the Minto Main deposit and Minto North, Minto East displayed a zoning from high-grade bornite dominant mineralization in the west to lower grade bornite + chalcopyrite mineralization in the east.

The high-grade bornite-rich core also returned excellent gold grades, and in some cases visible gold was observed along epidote lined fractures or directly associated with bornite.

Company geologists proposed, in 2006, that the separate prospects and deposits mentioned above comprise a single large continuous to contiguous mineralized system that has subsequently been deformed; openly folded and cut by late regional faults (some with vertical displacements and some with inferred lateral displacements). The sum of MintoEx's drilling and geological modelling since 2005 to date continues to support the single system thesis and upcoming exploration work in 2012 and beyond will focus on creating a unified geological model for the property south of the DEF fault, and possibly extending north of the DEF fault to Minto North.

Projecting 3-D geological models based on drill hole data into untested areas and then following up on promising targets remains the most important exploration tool at Minto. A significant portion of exploration work in 2008 and 2009 concentrated on infill drilling followed by stepping out from the Area 2/118 resource sub-domains and Ridgetop deposit. At Minto North, 2009 drilling evolved from exploration, to delineation, to infill. A similar pattern was followed for Minto East in 2009 and 2010. Infill drilling for all deposits yielded statistically more robust resource calculations, supporting the current PFS study, while step-out drilling continued to test for further extensions of the deposits. During 2009, two separate deep penetrating geophysical surveys were completed in order to fill in gaps not covered by the 2006-2007 GAIP survey, to test areas with deep overburden or permafrost, and to test deep ground under known deposits in the PEC. The first program of Dipole-Dipole Induced Polarization (DDIP) was completed by Aurora Geosciences of Whitehorse, Yukon over areas northwest, north, and northeast of the Minto deposit. The second program of Titan-24 DCIP and MT was completed by Quantec Geosciences of Toronto, Ontario over the PEC. In 2010, an expanded Titan-24 DCIP survey was completed covering ~85% of the property. The Titan-24 surveys are discussed in more detail in section 9.3.

In 2010, drill testing of Titan-24 anomalies successfully identified two new high priority targets including the Wildfire prospect and the Inferno Prospect. Subsequently, infill drilling at Wildfire and Copper Keel confirmed the continuity of mineralized zones to the Area 2/118 resource sub-domains supporting the completion of a new NI 43-101 resource estimate for the combined Minto South Deposit released May 30, 2011. Step-out drilling at the Inferno prospect and the Fireweed prospect discovered in 2011 is slated to continue in 2012. An initial resource estimate for the Fireweed prospect is anticipated for mid-2012 and a more robust estimate should be completed later in that same year after the seasonal cessation of drilling activities.

The discovery of nine new copper-gold deposits/resource sub-domains or significant prospects (Figure 9—2) in six years attests to the validity of the exploration methods being used at the Minto Mine by Capstone Mining Corporation and its subsidiary MintoEx.

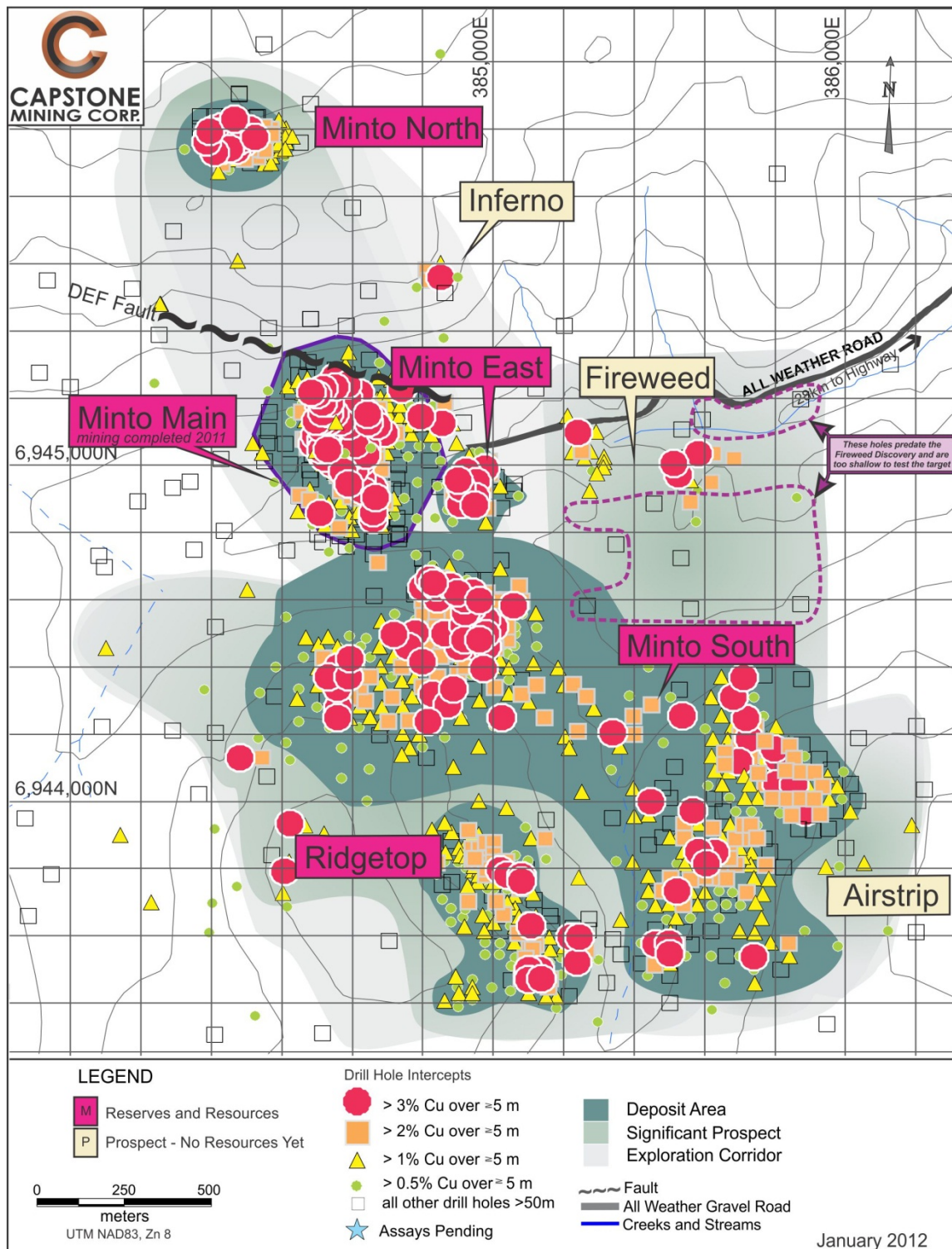


Figure 9—2: Priority Exploration Corridor (PEC) with Drill Results Showing the Highest Copper Grade over a Minimum Continuous 5 m Interval

9.1. Gradient IP Geophysical Surveying

An important component of the 2007 exploration program included increasing the coverage of the Gradient Array Induced Polarization (“GAIP”) survey at Minto. A total of 138 line kilometres of GAIP surveys were completed in 2007, a four-fold increase over the 33 km completed in the 2006 program, bringing the total GAIP kilometres surveyed by MintoEx for both years to 171 km. The GAIP surveying for 2006 and 2007 was conducted by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Gradient
- Dipole Spacing: 50 m
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points minimum every 300 m and at line-ends; (<10 m accuracy)

The 2007 survey was completed on ten separate blocks expanding upon the 2006 survey area to provide near seamless coverage over a total area of approximately 10 km². Areas with extensive mining activity or infrastructure could not be surveyed. The 2007 GAIP program was much more extensive than the 2006 pilot survey because drilling of the chargeability anomalies generated in the 2006 survey was positive. The GAIP survey showed a coincidence of significant copper sulphide mineralization with chargeability anomalies and suggested MintoEx had developed an additional exploration tool for prioritizing exploration drill targets.

The focus of the 2007 geophysical program was two-fold; firstly, to evaluate areas south of the main Minto deposit, expanding coverage into areas of known prospectivity that was not covered in the 2006 program and secondly, to begin evaluating areas north of the Minto mine, where there are multiple coincident copper-in-soil and magnetic anomalies, but very little core drilling. After positive drill results were obtained late in the drill program on a chargeability anomaly, located at the Airstrip SW and Copper Keel prospects on the southern limit of the 2006 survey area, a decision was made to expand the GAIP survey to an area south of the drill discovery.

The additional survey at Airstrip-Copper Keel defined a large chargeability anomaly in an under-explored region located to the south of the diamond drilling. This area was previously thought to be not prospective due to the presence of Cretaceous age cover rocks. These cover rocks are thought to represent a significant down throw and burial of the prospective host Jurassic age granodiorite. The new drilling had indicated the cover sequence was shallower than expected and granodiorite is locally exposed beneath overburden in small erosion windows through the conglomerate.

9.2. Modified Pole-Dipole Geophysical Surveying

A new exploration tool implemented in 2009 included the completion of a modified pole-dipole geophysical survey over areas west and north of the DEF fault from July 18 to August 10, 2009. The survey targeted areas of known historical geophysical anomalies, and well as overlapping GAIP coverage where permafrost or deep overburden ground conditions returned poor results (Figure

9—3). A total of 20.6 line kilometres were completed by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Modified Pole-Dipole Array
- Dipole Spacing: 50 m on all lines
- Dipole Read: N = 1 through 10 (10 Channels)
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points minimum every 250 m and at line-ends
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

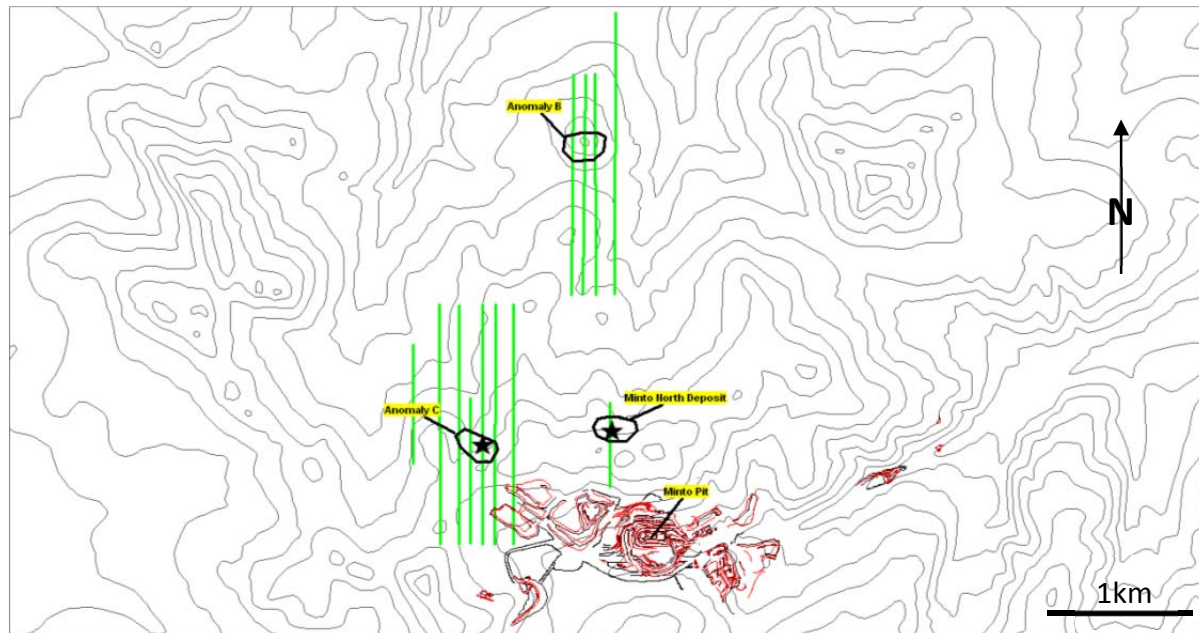


Figure 9—3: Modified Pole-Dipole 2009 Survey Grid Location Map (Green Lines) and Location of modified Mise-a-la-Masse Drill hole Surveys (Black Stars)

The results of the 2009 modified pole-dipole survey indicated two separate anomalies, one approximately 1,000 m due west of Minto North, and the second approximately 2,400 m due north of Minto North.

These 2009 anomalies were in good agreement with the historical pole-dipole survey anomalies denoted as Anomaly B (north) and Anomaly C (west) identified by ASARCO in 1974 (Figure 9—3). Similar to the historical Minto North anomaly (“Anomaly A”), ASARCO geologists believed that both of these anomalies were promising targets since the chargeability results were in similar magnitude to that of the Minto Main deposit. Due to the positive results of drilling at Minto North in 2009, MintoEx executed 1 drill hole into Anomaly B and 2 drill holes into Anomaly C. Drill results were enigmatic in that no significant copper-gold mineralization was encountered despite the intersection of multiple, thick sequences of foliated favourable host rock.

Minor pyrite and trace chalcopyrite or bornite was sporadically encountered in the 3 drill holes but it is believed that the low concentration of this mineralization does not satisfactorily explain the chargeability results.

Since the 2009 modified pole-dipole test line over Minto North with known high-grade copper mineralization confirmed a similar chargeability response to Anomalies B and C, MintoEx geologists felt that the results of the preliminary drilling were inconclusive. Thus, a single down hole mise-a-la-masse survey was completed at Anomaly C in hopes of further vectoring follow-up drilling (see below for details of the survey). Preliminary field results of this down hole survey were again in agreement with a calibration survey at Minto North suggesting that Anomaly C was still an intriguing exploration target. Both Anomalies B and C remain priority targets for future drill programs, and follow-up drilling will be focused using the results of the combined 3-D modelling of survey and incorporated down hole survey results.

As part of the 2009 modified pole-dipole geophysical survey, one calibration (Minto North) and one follow-up (Anomaly C) mise-a-la-masse drill hole IP survey were completed by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Radial Array
- Dipole Spacing: 25 m on all lines
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points at line-ends and the center of each line
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

In 2010, mise-a-la-masse drill hole IP surveys were completed in single drill holes at Copper Keel NE and the Inferno prospect by Aurora Geosciences of Whitehorse, Yukon Territory, using the following specifications:

- Array: Radial Array
- Dipole Spacing: 50 m on main lines, 25 m for infill lines
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points at line-ends and the center of each line
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

In 2011, mise-a-la-masse drill hole IP surveys were completed in two drill holes at Copper Keel NE and two holes at the Inferno prospect by Aurora Geosciences of Whitehorse, Yukon, using the following specifications:

- Array: Radial Array
- Dipole Spacing: 50 m and 25 m
- Tx: Time domain, 50% duty cycle, reversing polarity, 0.125 Hz
- Stacks: Minimum 15
- Rx Error: 5 mV/V or less, otherwise repeated several times
- Grid Registration: Handheld GPS points at line-ends and the center of each line

-
- <10 m accuracy; all coordinates in UTM NAD83 Zone 8V North

9.3. Titan-24 Geophysical Surveying

This section is summarized from the “Quantec Titan-24 Distributed Acquisition System DC Resistivity, Induced Polarization and MT Resistivity Survey over the Minto Mine Interpretation Report” by Quantec Geoscience (September 2009) and from “Titan-24 DC/IP Survey Geophysical Report, Minto Mine” by Quantec Geoscience (August 2010).

Another new exploration tool implemented in 2009 included the completion of the deep penetrating Titan-24 geophysical survey over the Minto priority exploration corridor from July 29 to August 8, 2009. The survey included three double spread direct current resistivity/induced polarization (DC/IP) and magnetotelluric (“MT”) lines totalling 21 line kilometres.

Each line was positioned on an azimuth of 341 degrees extending from south of Ridgetop to north of Minto North. Each line was surveyed with pole-dipole geometry with a dipole spacing of 100 m. The array length was 2.4 km and two arrays were used with 400 to 500 m overlap to measure the 4 km long line. The data were inverted using 2D inversion algorithms to produce plan and section maps of DC and MT resistivity and chargeability of the subsurface. Data quality was very good, especially for an active mine site; typical measurements errors for DC were well below 0.5% and approximately 5% for the IP data with MT data in the quality range of 10 kHz to 0.01 Hz for most of the sites. The Titan-24 surveying for 2009 was conducted by Quantec Geoscience of Toronto, Ontario.

The 2009 DC/IP surveys used the following specifications:

- Survey Array: Dipole-Pole-Dipole (combined PDR and PDL)
- Receiver Configuration: 24-25 Ex = Continuous in-line voltages
13 Ey = Alternating (2-station) cross-line voltages
- Array Length: 2400-2500 meters
- Number of Arrays/Line: 2
- Dipole Spacing: 100 meters
- Sampling Interval: Ex = 100 meters
Ey = 200 meters
- Rx-Tx Separation: N-spacing (Pn-Cn min) = 0.5 to 39.5
- Infinite Pole Location: UTM: 392344E, 6948844N (NAD 83, Zone 08V North)
- Spectral Domain: Tx = Frequency-domain square-wave current
- Spectral Domain: Rx = Full waveform time-series acquisition
- Transmitter Waveform: 30/245 Hz square wave (~ 4 s. pos/neg), 100% duty cycle

The 2009 MT surveys used the following specifications:

- Technique: Tensor soundings, remote-referenced
- Base Configuration: 24-25 Ex = Continuous in-line E-fields
13 Ey = Alternating (2-station) cross-line E-fields
1 pair LF coils
1 pair HF coils
- Remote Configuration: 1 Ex = in line E-fields

-
- Ey = cross-line E-fields
 - 1 pair LF coils
 - 1 pair HF coils
 - Array Length: 2400-2500 meters
 - Number of Arrays/Line: 2
 - Dipole Spacing: 100 meters
 - Sampling Interval: Ex = 100 meters
Ey = 200 meters
 - Ex/Ey Sampling Ratio: 2:1
 - E/H Sampling Ratio: Ex = 24:1 and 25:1
Ey = 13:1
 - Remote Measurements: 1Hx/Hy set (1 Ey/Ex for verification/monitoring)
 - Remote Position: 424855E, 7001518N (NAD 83, Zone 08V North)
 - Frequency bandwidth: 0.01 to 10000 Hz
 - Data Acquisition: Full-waveform time-series acquisition
Data processing/output in frequency-domain

An expanded Titan-24 DC/IP survey covering about 85% of the property was completed from May 19 to July 15, 2010; MT was not included in the expansion. In total, ten additional parallel lines expanding outwards to the east and west from the 2009 PEC coverage with 600 m current extensions on either end were surveyed as well as one northern extension to an existing 2009 line. Each line was positioned on an azimuth of 341 degrees, and line spacing varied between 200 m and 400 m. The survey included ten double spread and one single spread direct current resistivity/induced polarization (DC/IP) lines totalling 62.5 line kilometres.

Each line was surveyed with pole-dipole geometry with a dipole spacing of 100 m. The array length was between 1.7 and 2.7 km and multiple arrays were used with 400 to 600 m overlap to measure lines up to 6.6 km long (including current extensions).

The data were inverted using 2D inversion algorithms to produce section and plan maps of DC resistivity and chargeability of the subsurface. Data quality was very good, especially for an active mine site; typical measurement errors for DC were well below 0.5% and approximately 5% for the IP data. The Titan-24 surveying for 2010 was conducted by Quantec Geoscience of Toronto, Ontario.

The 2010 DC/IP surveys used the following specifications:

- Survey Array: Dipole-Pole-Dipole (combined PDR and PDL)
- Receiver Configuration: 24-25 Ex = Continuous in-line voltages
13 Ey = Alternating (2-station) cross-line voltages
- Array Length: 2400-2500 meters
- Number of Arrays/Line: 2
- Dipole Spacing: 100 meters
- Sampling Interval: Ex = 100 meters
Ey = 200 meters
- Rx-Tx Separation: N-spacing (Pn-Cn min) = 0.5 to 39.5

-
- Infinite Pole Location: UTM: 392344E, 6948844N (NAD 83, Zone 08V North)
 - Spectral Domain: Tx = Frequency-domain square-wave current
 - Spectral Domain: Rx = Full waveform time-series acquisition
 - Transmitter Waveform: 30/245 Hz square wave (~ 4 s. pos/neg), 100% duty cycle

The 2009 Titan-24 survey was completed over the Minto PEC in order to first test the geophysical response over Ridgetop, Area 2/118, Minto Main, and Minto North; and secondly to evaluate the possibility of deep mineralization lying beneath these areas to a depth of approximately 750 m. Thirdly, using the maps of the resultant resistivity to possibly identify and characterize large scale structures over the Minto Mine area. Where the survey grid was positioned over the Minto Pit, the west and east flanking lines were bent around the pit and the central line was executed by using rafts to position electrodes across the flooded pit bottom. The expanded 2010 Titan-24 DC/IP survey was completed in order to constrain the positioning of anomalies identified in 2009 as well as to evaluate areas of the property proximal to and well outside the PEC. The combined 2009-2010 Titan-24 survey grid is presented in Figure 9—4.

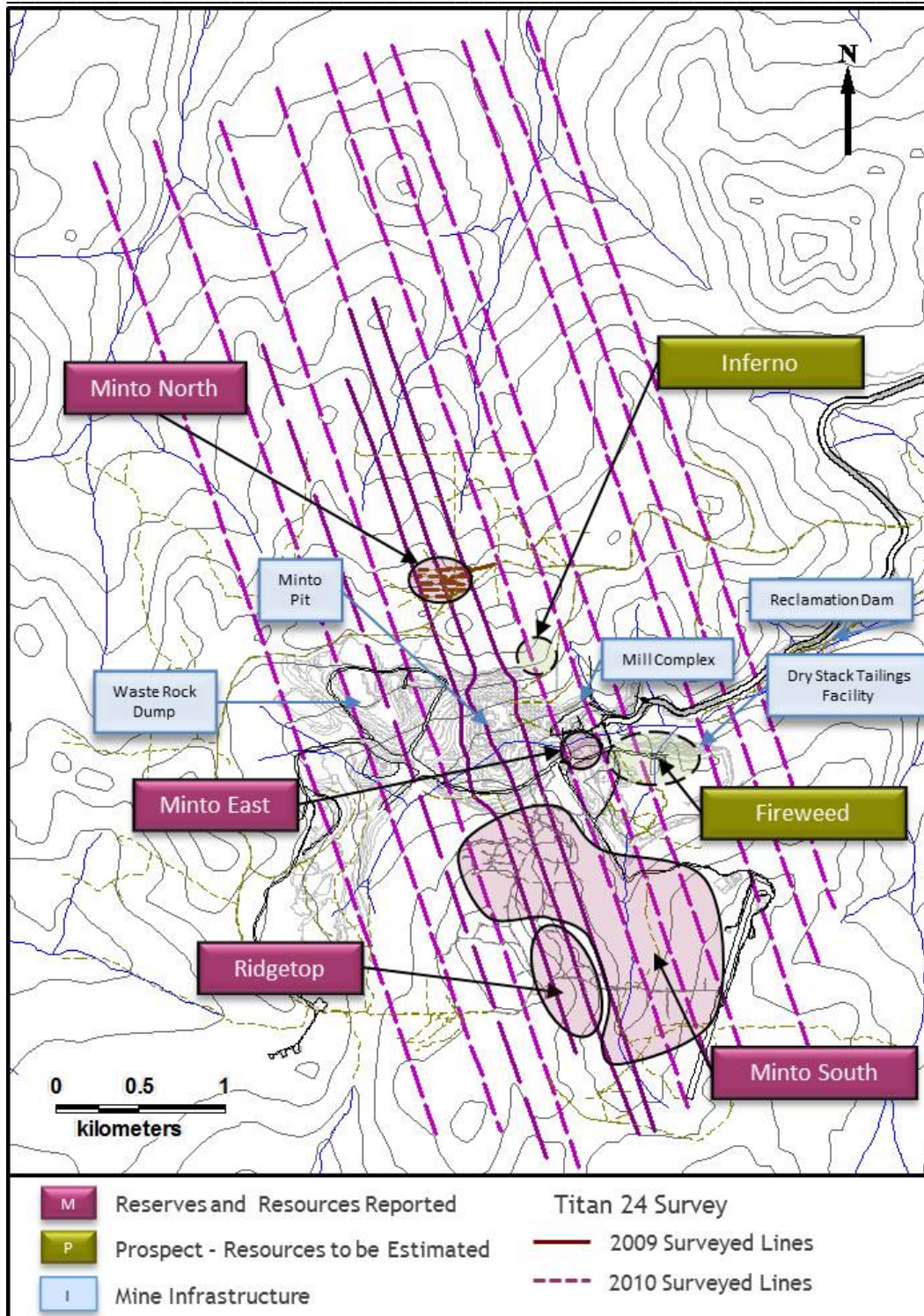


Figure 9—4: Titan-24 Combined 2009-2010 Survey Grid

The 2009 Titan-24 survey showed a coincidence of significant copper sulphide mineralization of known deposits with chargeability anomalies as well as several previously unknown deep anomalies, suggesting that MintoEx had developed an additional exploration tool for prioritizing exploration drill targets. The most attractive deep targets were located south of Ridgetop, flanking the Minto Main Pit (west, southeast, northwest, and northeast), and flanking the Minto North deposit (east, west, and north). The survey also identified a near surface target southwest of Ridgetop. MT results indicated steeply dipping fault-like structures with an estimated 70° dip to the north, the most prominent being the DEF fault.

Preliminary drill testing of the Titan-24 chargeability targets spanned from September 4 to October 17, 2009. Results of the drilling were variable returning promising copper mineralization intersections in 9 drill holes at Ridgetop Southwest and significant copper-gold mineralization in 2 holes southeast of Minto Pit (Minto East discovery); however, in 9 holes at 8 other separate targets no significant copper-gold mineralization was encountered despite the intersection of multiple, thick sequences of foliated favourable host rock. Based upon the discussions with representatives of Quantec Geosciences and the experience gained at Minto East where the first hole missed and a second hole drilled more than 130 m east of the actual survey line confirmed the discovery, the lack of success at some of these other anomalies was attributed to at least in part due to the limited coverage of the survey. In other words, the method appeared to be able to “see” anomalous features that actually sit well to the side of the survey area. Because the initial proof-of-concept survey was only three lines wide, and all significant and unexplained anomalies lay on either of the two flanking lines, these anomalies were considered to be poorly constrained.

Minor pyrite and trace chalcopyrite were sporadically encountered in the nine unsuccessful 2009 test holes, but it was believed that the low concentration of this mineralization did not satisfactorily explain the chargeability results. MintoEx geologists suspected that the poor intersections into the various targets may have reflected a positioning problem with these specific anomalies. As mentioned above these anomalies flanked either the eastern or western survey lines and the exact locations were thus poorly constrained. Follow-up down hole DC/IP surveys were completed in five holes where drilling results were in question in order to guide follow-up drilling. The decision was made to complete additional parallel survey lines positioned to the east and the west of the PEC coverage to further vector in on the precise locations of the anomalies using more constraining data to provide better resolution and precise locations of chargeability anomalies.

Similar to the 2009 Titan-24 survey, the expanded 2010 survey identified previously unknown moderate to deep anomalies (Figure 9—5). The most attractive new targets were located east of the Copper Keel trend (Wildfire), at Copper Keel NE, southwest of Ridgetop, at Airstrip SW, and northeast of the Minto airstrip. The positioning of the 2009 anomalies flanking the Minto Main Pit (west, southeast, northwest, and northeast) and flanking the Minto North deposit (east, west, and north) shifted as expected due to better constraining data. For example, the discovery hole for the Inferno prospect targeted the 2010 position of the anomaly northeast of the Minto Pit that shifted by 180 m to the northeast as compared to the 2009 anomaly location.

However, the positioning of the Minto East anomaly did not shift to the east as was expected, and the conflict between the anomaly location and the deposit location remains unexplained.

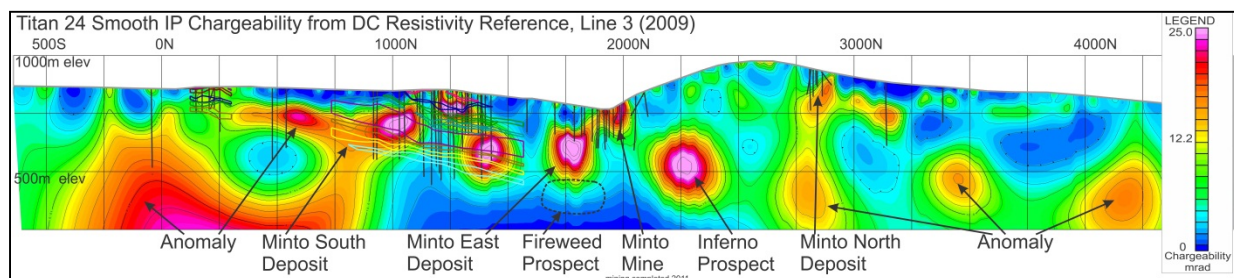


Figure 9—5: North-South Cross-Section (Line 1 from Figure 9.4) showing Titan-24 Anomalies)

Drill testing of the 2010 Titan-24 chargeability targets spanned from June 25 to November 5, 2010. Further testing of select Titan-24 chargeability targets continued throughout the 2011 drilling campaign.

Results of the 2010 drilling were variable returning significant copper mineralization intersections in more than 70 plus drill holes east of the Copper Keel trend (Wildfire discovery) and in 4 holes northeast of Minto Pit (Inferno discovery). Promising copper-gold mineralization was observed in 3 holes southwest of Area 118, 4 holes at Copper Keel NE, and in 1 hole at Ridgetop NE. No significant copper-gold mineralization was encountered in 5 holes at three other separate targets despite the intersection of multiple, thick sequences of foliated favourable host rock.

Results of the 2011 drilling were variable returning significant copper mineralization intersections in more than 70 plus drill holes at Copper Keel NE and 26 holes at the Fireweed Discovery. Similar to 2010 drilling, some targets tested in 2011 did not encounter significant copper-gold mineralization despite the intersection of multiple, thick sequences of foliated favourable host rock.

In some cases, the new 2010 anomalies flanked the survey lines and it is suspected as with the 2009 survey that the positioning of these anomalies is again questionable. However, those anomalies drill tested that were well constrained that did not intersect significant mineralization remain unexplained. Based upon discussions with representatives of Quantec Geosciences the lack of success at some of these other anomalies may be attributed to at least in part to the presence of magnetite or platy minerals (i.e. biotite) that are present in the foliated granodiorite horizons at Minto.

Testing of outstanding Titan-24 targets as well as revisiting 2009-2010 anomalies that have been better constrained is slated to continue into 2012 with the prime focus of follow-up drilling at Fireweed, Inferno, , and other targets north of the DEF. The authors recommend that all initial holes testing new anomalies should be followed up by DHIP to vector future drilling.

10. Drilling

MintoEx drilled a total of 44,079 m in 129 drill holes on the Minto Property in 2011. The 2011 drilling program was conducted between February 8 and November 11, 2011 and was contracted to Driftwood Diamond Drilling Ltd. of Smithers, British Columbia under the direct supervision of MintoEx and Capstone Mining Corporation staff geologists. Seventy-four 2011 drill holes (24,447 m) were used in the resource estimations discussed in this report; however 30 drill holes (12,472 m) completed in 2011 are associated with the Fireweed and Inferno prospects that are still being explored and as such are not incorporated into the mineral resource estimates used in this report.

A total of 74 holes or 24,447 m of the 2011 drilling were incorporated into the four resource models described in this report. 55 holes for 19,632 m were drilled specifically at exploration prospects outside of these resource models. The median length of 2011 MintoEx drill holes was 342 m (average 349 m), with the shallowest hole being 110 m in length and the deepest, 606 m. MintoEx diamond drill holes by year and deposit, from 2005 through 2011, are summarized in Table 10-1 below.

Table 10-1: Summary of MintoEx Drill holes by Deposit (2005 to 2011)

Company	Deposit	Year	No. DDH	Type	Core Size	Metres
MintoEx	Minto	2011	5	DDH	NQ	2,077
MintoEx	Minto	2010	14	DDH	NQ	1,090
MintoEx	Minto	2009	2	DDH	(1) HQ, (1) NQ	591
MintoEx	Minto	2008	-	-	-	-
MintoEx	Minto	2007	5	DDH	(3) HQ, (2) NQ	754
MintoEx	Minto	2006	25	DDH	NQ	4,119
MintoEx	Minto	2005	44	DDH	NQ	5,369
MintoEx	Minto South	2011	74	DDH	NQ	24,447
MintoEx	Minto South	2010	109	DDH	(7) NTW, (102) NQ	29,981
MintoEx	Minto South	2009	27	DDH	NQ	6,443
MintoEx	Minto South	2008	49	DDH	NQ	11,461
MintoEx	Minto South	2007	56	DDH	NQ	15,772
MintoEx	Minto South	2006	80	DDH	NQ	18,352
MintoEx	Ridgetop	2009	71	DDH	NQ	7,858
MintoEx	Ridgetop	2008	46	DDH	NQ	6,099
MintoEx	Ridgetop	2007	25	DDH	NQ	3,432
MintoEx	Minto North	2009	88	DDH	NQ	11,548

Company	Deposit	Year	No. DDH	Type	Core Size	Metres
MintoEx	Minto East	2010	27	DDH	(4) NTW, (23) NQ	8,892
MintoEx	Minto East	2009	4	DDH	NQ	1,410
MintoEx	Minto East	2008	1	DDH	NQ	385
MintoEx	Minto East	2007	1	DDH	NQ	380

The Minto South Deposit (Area 2/118/Wildfire/Copper Keel) resource estimation incorporates the majority of 2006-2011 drilling within Area 2, 118, Copper Keel, Wildfire and 28 drill holes (5,770 m) completed by ASARCO in the 1970s. The size of the historical ASARCO drill core was not recorded, but is believed to be BQ size, based on observations of core found in core storage sheds destroyed by forest fires.

MintoEx drilled a total of 106,456 m in 376 vertical and 19 angled diamond drill holes at the Minto South Deposit from February 28, 2006 to July 5, 2011. The size of the drill core is NQ and NTW. The median length of the drill holes is 276 m (average 269.5 m); the shallowest hole was 78 m long and deepest hole was 693 m. All 395 drill holes were used in the Minto South Deposit resource estimation. 3 angled and 25 vertical holes drilled by ASARCO in 1971 to 1974 were included in the Minto South Deposit resource estimate. Drill hole collars are spaced at approximately 28 m to 60 m at the Area 2 resource sub-domain, 40m at the Area 118 resource sub-domain, and 40 m to 60 m centers at the Wildfire and Copper Keel sub-domains (Figure 10-1).

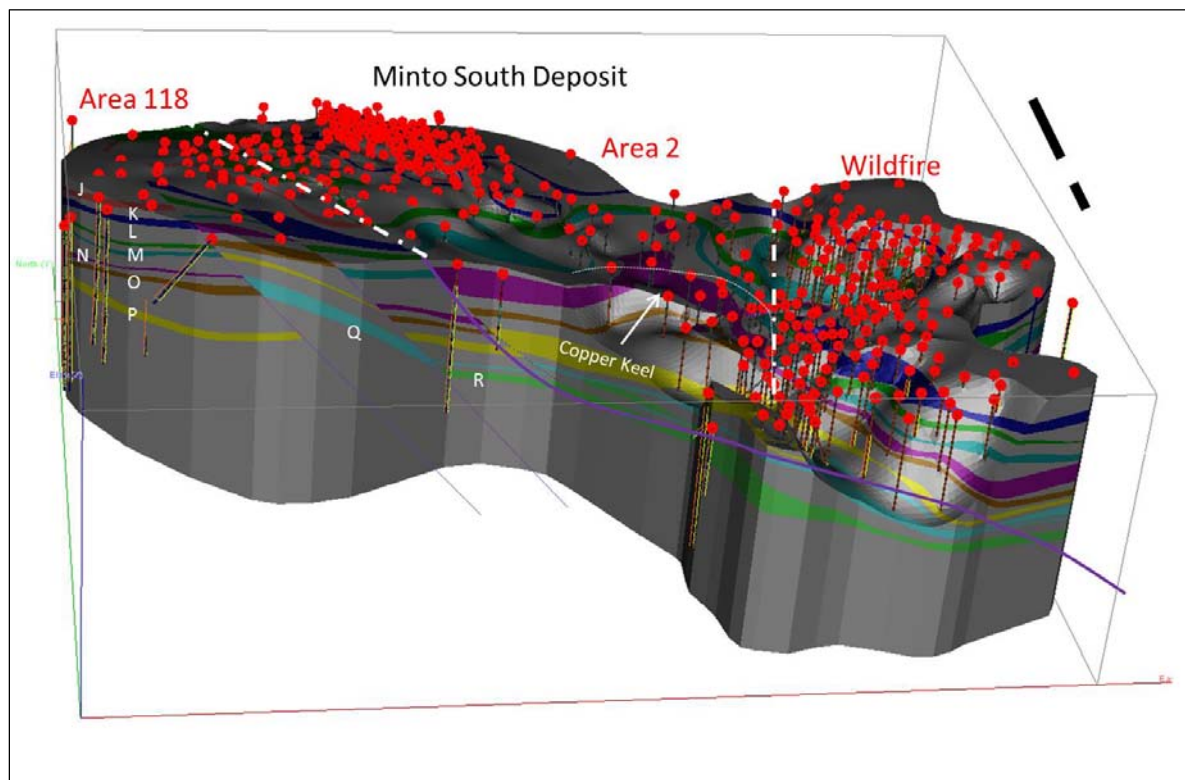


Figure 10-1: Minto South Deposit: Labelled Mineralized Domains with Drill Holes View Northeast

At Ridgetop, MintoEx drilled a total of 16,850 m in 139 vertical drill holes and 3 angled diamond drill holes from May 27, 2007 to September 20, 2009. The size of the MintoEx drill core is NQ. The median length of the Ridgetop drill holes is 120.5 m (average 122.5 m); the shallowest hole was 54 m long and the deepest hole was 322 m. One vertical hole (150 m) and three angled holes (492 m) drilled by ASARCO in 1971, and four vertical (635 m) holes and four angled holes (571 m) drilled in 1972 were included in the resource. Size of the ASARCO drill core is assumed to be BQ. In 1994, four vertical holes (520 m) and five angled holes (654 m) of HQ-sized core were drilled; these holes were used in the resource estimate. Drill hole collars are spaced at approximately 20 to 60 m centers. Mineralized zones are dipping moderately to the northeast (Figure 10—2).

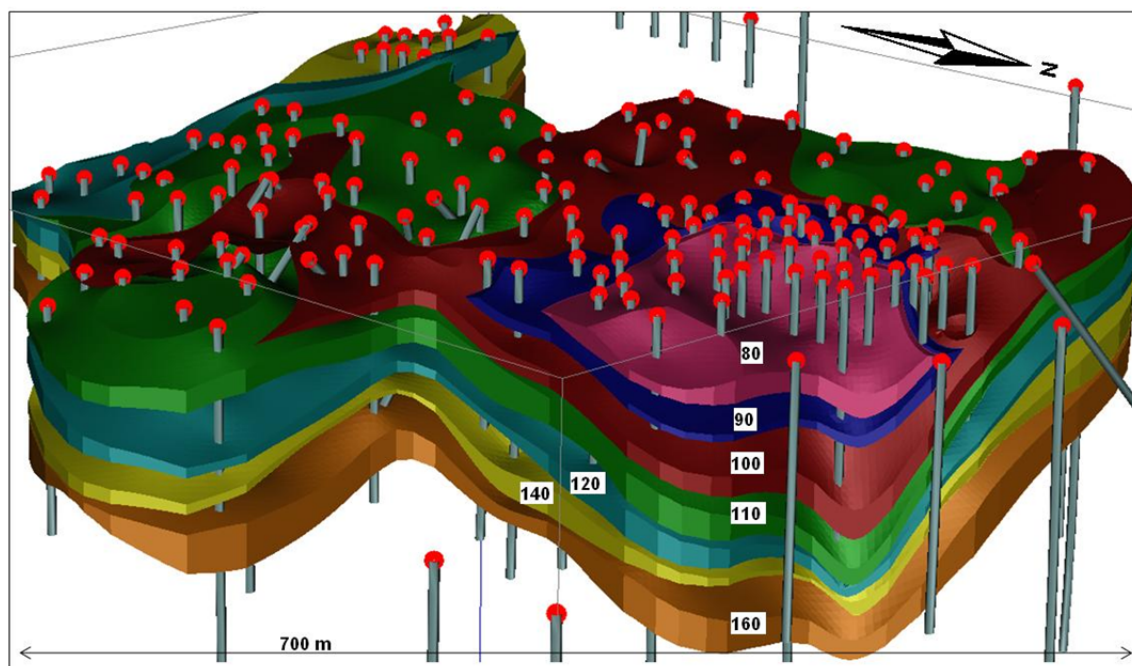


Figure 10—2: Wireframes of Labelled Mineralized Domains with Drill Holes, Ridgetop

At Minto North, MintoEx drilled a total of 11,548 m in 71 vertical and 17 angled diamond drill holes from January 27 to October 4, 2009. In total, 87 drill holes are included in the resource model; one drill hole is excluded because it is located well outside the currently defined deposit boundaries. No historical drill holes are included in the resource model. The size of the MintoEx drill core is NQ. The median length of the 2009 Minto North drill holes is 120 m (average 130 m); the shallowest hole was 57 m and the deepest hole was 342 m. Drill hole collars are spaced at approximately 15 to 20 m centers. Mineralized zones are shallowly dipping to the northwest (Figure 10—3).

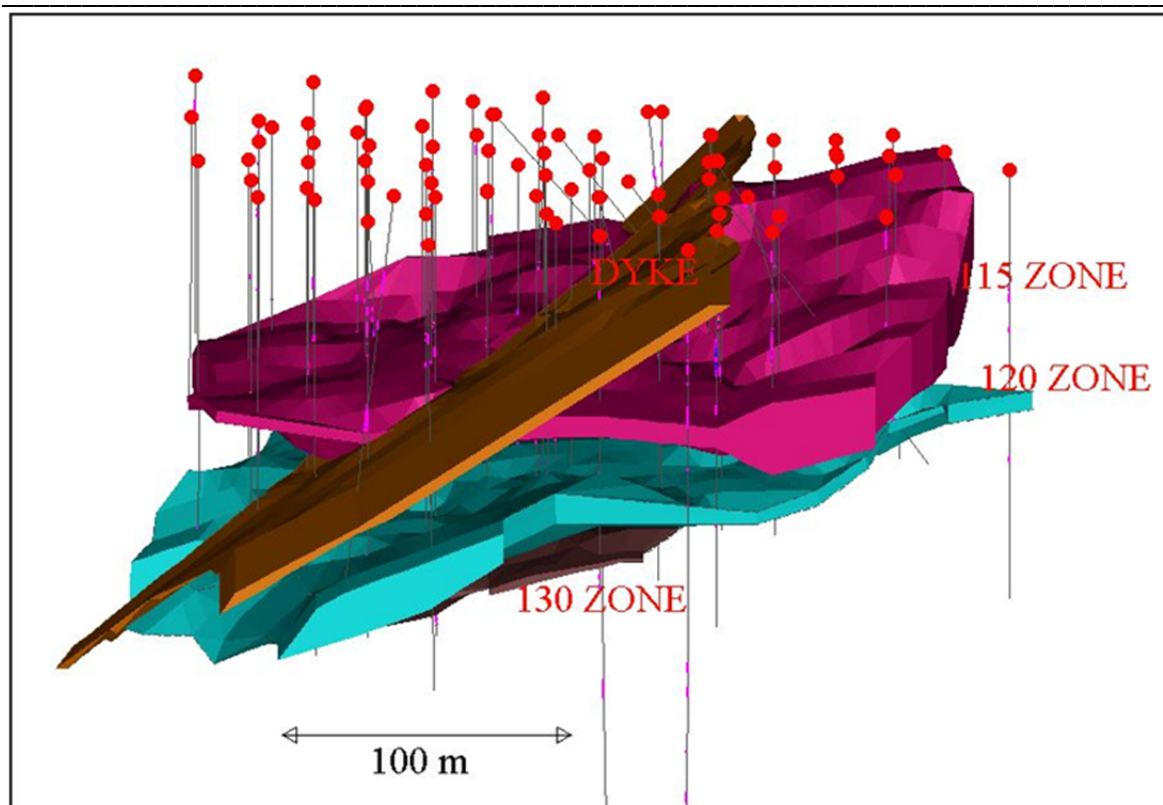


Figure 10—3: Wireframes of Mineralized Domains with Drill holes, Minto North

At Minto East, MintoEx drilled a total of 11,067 m in 13 vertical and 20 angled diamond drill holes from April 18, 2007 to August 21, 2010. In total, 33 drill holes are included in the resource model. No historical drill holes are included in the resource model. The size of the MintoEx drill core is NQ with the exception of four drill holes in NTW. The median length of the Minto East drill holes is 332 m (average 336 m); the shallowest hole was 179 m and the deepest hole was 408 m. Drill hole collars are spaced at approximately 40 m centers. Mineralized zones are shallowly dipping to the northwest (Figure 10—4).

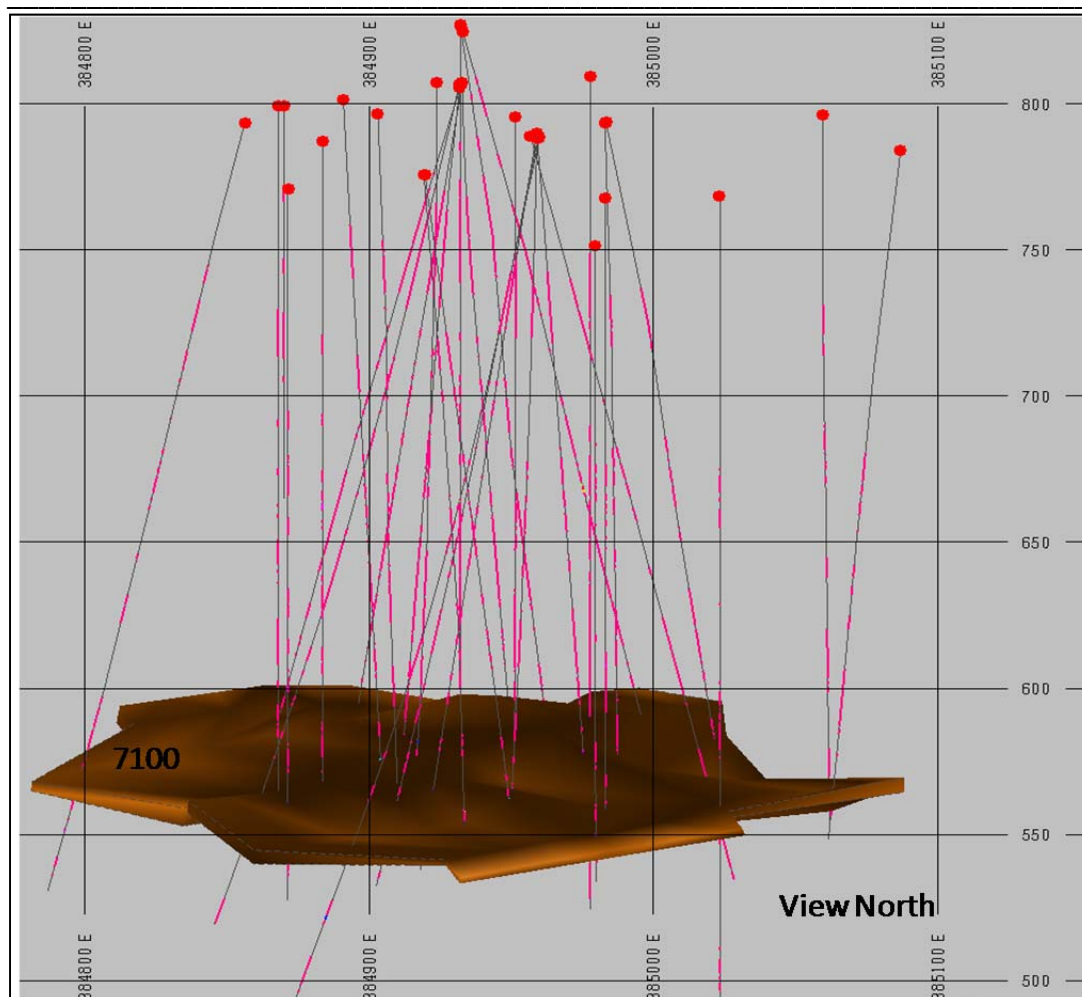


Figure 10—4: Wireframe of Mineralized Domain with Drill holes, Minto East

Prior to 2008, all drilling for MintoEx was completed using the imperial system, and footages were converted to metres by MintoEx personnel who logged and recorded all data in metres. Since 2008, drilling for MintoEx was completed using the metric system. Drill hole collar locations were initially located using a differential GPS unit, and more precise location coordinates were surveyed after completion of drilling by the Minto Mine survey team using a Trimble R8 GPS unit.

Acid tests were performed at the end of each hole or at various depths down the hole in the winter of 2008 and in some occasions during 2010 drilling. Minimal deviations were typical in all holes which were predominantly drilled at a vertical inclination.

Since the spring of 2008, down hole surveys were primarily performed using a FLEXIT down hole survey tool. Although local magnetite concentrations sometimes prevented measurement of azimuth deviations, the tool overall provided realistic readings showing minor deviation in azimuth and dip. In 2010, a Maxibor tool was utilized in 22 drill holes in areas known to be highly magnetic. Mineralized intervals measured in the vertical drill holes are believed to

represent very close to the true widths of mineralized layers within the deposit because of the sub-horizontal attitude of the mineralized zones.

The core was transported from the drill rig to the logging facility by the drilling contractor, where MintoEx personnel logged it for geological, sampling, and geotechnical purposes. Geological data including lithology, structure, alteration, and mineralization was recorded for all drill holes.

All drill core was photographed for easy reference when constructing geological models for resource estimation.

Geotechnical data was collected on all drill holes in 2008 to 2011, including RQD, core recovery, fracture density and orientation, hardness and joint data. Recovery was typically very good to excellent. Orientation data for individual joints and structures was not measured for most holes as they were drilled vertically, but the approximate alpha angle was recorded. Orientation data for individual joints and structures were recorded in 14 oriented geotechnical drills totalling 3,405 m, including one hole at Area 2 (261 m), three holes at Area 118 (981 m), three holes at Ridgetop (525 m), two holes in the DEF area of the Minto Main deposit (591 m), three holes at Wildfire (753 m), and two holes at Minto North (294 m).

Magnetic susceptibility data was also collected for each drill hole in 2008 to 2011. No direct correlation between the degree of magnetic susceptibility and grades of mineralization can be made, but a marked increase in the magnetic susceptibility is noted in mineralized intervals. This is not surprising since increased magnetite content is frequently logged in all mineralized horizons. However, magnetite is often more pervasive than sulphide mineralization and magnetite concentrations are not directly proportional to copper grade. Elevated levels of magnetite are found within the mineralized horizons, but where sulphide mineralization has a sharp transition from foliated to unfoliated domains, magnetite alteration can persist, although at much lower concentrations into unmineralized domains. In some instances, the presence of hematite or hematite/magnetite combinations in unmineralized domains corresponds to brittle structures, suggesting some remobilization of iron after mineralization and is thought to be due largely to supergene processes. In such case, the magnetic susceptibility readings are muted somewhat.

Sample intervals were marked on the core and a cut line was drawn with a china marker for the diamond saw cutter to follow. Half of the core was placed in a sample bag and the other half was returned to the core box. Sample intervals were nominally taken at 1.5 m in the mineralized zones, with a minimum of 2 shoulder samples taken into the waste contact. Waste material between successively stacked mineralized zones was sampled at 3 m intervals to avoid gaps in assay data.

Sample intervals from the vertical holes approximate the true width of the mineralized zones, whereas FLEXIT or Maxibor down hole survey data was used to determine the true width of mineralized zones in angled drill holes. Sampling results are described in detail in subsequent sections.

Bulk density measurements were taken from nearly all holes drilled from 2005 through 2011 in both mineralized and waste material. Measurements were taken at approximately every 1 to 3 m intervals in ore, corresponding to one to three measurements per run in strongly mineralized material, 1 every 5 m in poorly mineralized material, and at least one measurement every 20 to 30 m in waste.

Pieces of core were weighed both in air and in water using an Ohaus triple beam balance. Spot checks on the field data were undertaken internally by MintoEx, where 159 samples from 66 drill holes were analyzed. Measurements were recorded on a triple beam scale on the same piece of core that was originally measured.

Bulk density data obtained prior to 2005 were not used in the resource estimations because the data was constructed by correlating bulk density to copper grade based upon too few actual measurements and because the core upon which this method was constructed was destroyed in forest fires and the methodology could not be audited.

For additional information regarding drilling and bulk density measurements obtained prior to 2008 for the Minto, Area 2, Area 118, and Ridgetop Deposits, please refer to Section 7 in *“Technical Report (43-10 1) for the Minto Project”* by Hatch (August 2006) and to Section 11 in *“Area 2 Pre-feasibility Study Minto Mine, Yukon”* (November 2007) and to Sections 11 and 12 in *“Technical Report Minto Mine, Yukon”* prepared by SRK Consulting (Canada) Inc. (June 2008) found on the [sedar.com](http://www.sedar.com) website.

11. Sample Preparation, Analyses and Security

11.1. Historic Samples

11.1.1. ASARCO 1971 to 1974

No detailed descriptions of historical sampling methods, preparation and analyses by ASARCO were recorded, however, based on observation, 5 and 10 foot long samples were favoured. Very few ASARCO holes are used in the resource and all are near MintoEx holes, limiting the effect of the ASARCO data on the resource calculation. No usable core survives from that period. It is inevitable that company employees would be involved in sampling but the exact activities and names of these ASARCO employees are unknown. It is not known whether officers or directors of ASARCO were involved in the sample preparation, but this is considered unlikely given the minor nature of the project. Subsequent sample preparation such as crushing, pulverizing and sample splitting would have been the responsibility of the laboratory.

Chemex in Vancouver is believed to have been responsible for the 1970s analyses (Simpson, 2002). At the time, copper analyses were typically performed by digesting a 2 g sample pulverized to 100 mesh, in perchloric and nitric acid with an atomic absorption spectroscopy (AAS) finish. Modern practices use a 0.4 g 150 mesh samples and aqua regia digestion. Gold analyses in the 1970s probably used a 10 g pulp digested in aqua regia and an AAS finish. Electronic microbalances and improvements in AA analysis have combined to reduce detection limits in the past 25 years.

Some of the early samples were not analyzed for precious metals. Most samples were analyzed solely for total copper, resulting in an incomplete data set of gold and silver. Copper oxide mineralization is confined typically to the upper level of the deposit and, historically, non-sulphide copper was not universally quantified by analysis of soluble copper.

11.1.2. TECK 1993 to 2001

From 1993 to 2001, TECK (now part of Teck Cominco) drilled 48 diamond drill holes on the Minto property. Sample lengths vary from 0.55 m to 2.75 m, averaging 1.59 m with a median of 1.53 m. Sampling protocols and information regarding security of samples, as required in NI 43-101, was not well documented during the 1993 to 2001 drill programs. The historic samples would likely have been prepared on site from split core under the supervision of TECK and MintoEx geologists, bagged and shipped to the laboratory. As in 1974, it is assumed company employees would be involved in the sampling process but it is not known exactly who would have been involved other than the project manager, F.T. Graybeal. It is considered unlikely officers or directors of TECK or MintoEx were involved in sample preparation. Subsequent sample preparation such as crushing, pulverizing and sample splitting would have been the responsibility of the laboratory.

Northern Analytical Services of Whitehorse, Yukon conducted the analyses for copper, gold and silver. Analytical methods are not documented in the certificates of analysis for this work, but are believed to be equivalent to the methods listed on the certificates for check analysis

performed by Chemex, detailed below. Non-sulphide copper was not initially quantified by analysis of soluble copper at Northern Analytical Services.

Bondar-Clegg of North Vancouver carried out the analyses of the 2001 samples. Each 0.25 g sample was digested with HCL, HNO₃, HCLO₄ and HF acids with final copper determination by AAS. Gold and silver were determined by fire assay using a 30 g sample and AAS finish.

No useable mineralized intersections of the 1994 TECK Ridgetop East drill holes remain on-site. A few stacks of 1994 core were discovered at the old location of the Minto Exploration camp site and at the Yukon Geoscience core library but the bottom of the holes containing mineralized intervals were not present. No other useable drill core from the 1993 to 2001 period remains on-site.

11.2. MintoEx Samples

11.2.1. MintoEx 2005 and 2006 Samples

During 2005 and 2006, drill core samples, Standard Reference Materials ("SRM") and blanks were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, Canada. In addition, Chemex was also instructed to perform analysis on pulp duplicates injected into the sample stream at regular intervals. In 2005, all samples were processed in Vancouver. In 2006, some samples were processed at other Chemex locations. Chemex-Elko, NV, USA processed 9% of the total number of samples and Chemex-Thunder Bay, ON processed 11%. The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm). A 100 to 250 g subsample was then split and pulverized to better than 85% passing -75 µm.

Copper was determined through a four acid digestion method (HF, HNO₃, HCLO₄ digestion and HCL leach) with final copper determination by AAS. Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination.

Gold was determined by one assay-tonne fire assay analysis. During 2005, all sample analysis was completed by gravimetric finish. During 2006, the first 17% (1,955) of the sample analysis was completed by gravimetric finish. For the remaining samples (9,182), the gold analysis was determined using AAS method. Silver was analyzed using aqua regia digestion and AAS finish.

11.2.2. MintoEx 2007 Samples

The 2007 drill core samples, blanks, SRMs and duplicates were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, Canada. Some samples were processed at other locations. SGS Laboratories under agreement with MintoEx processed 485 samples (6% of the total number of samples); assays were all performed at the Vancouver Chemex Lab. Sample preparations were performed at Chemex at Elko, NV, USA, 4% of the total number of samples, Chemex at Reno, NV, USA 10%, and Chemex at Terrace, Canada 50%.

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm). A 100 to 250 g subsample was then split and pulverized to better than 85% passing -75 µm.

Copper was determined by the four acid digestion method (HF, HNO₃, HClO₄ digestion and HCL-leach) with final copper determination by AAS. Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was analyzed by one assay-tonne fire assay followed by AAS. Silver was analyzed using aqua regia digestion and AAS finish.

11.2.3. MintoEx 2008 Samples

Two laboratories were used in 2008. Drill core samples, blanks, SRMs and duplicates were submitted to SGS Laboratories under agreement with MintoEx, and to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver, BC after processing at the sample preparation facility in Terrace, BC. SGS Laboratories under agreement with MintoEx processed 61% of the total number of samples from areas outside of Ridgetop. The remaining 39% of the samples were analysed at the Vancouver Chemex Lab.

The samples submitted to SGS were first crushed in a jaw crusher to reduce the material to greater than 85% -10 mesh (2 mm). A 250 g subsample was then split and pulverized to better than 90% passing -75 µm. The pulp was split with one part analysed for copper and silver at the SGS facility at the Minto site and one part analysed for gold and non-sulphide copper at SGS Red Lake, ON operation. During mid-July, silver analyses were performed by SGS at Lakefield, ON and Don Mills, ON after a switch failure in SGS Minto ICP-AAS equipment. Copper reanalysis due to SRM failures were done by SGS at Lakefield and Don Mills in Ontario.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Samples were assayed for gold using a fire assay procedure on a thirty grams sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

The samples submitted to Chemex from July 27 to August 19 were first crushed in a jaw crusher to reduce the material to greater than 85% -10 mesh (2 mm). A 250 g subsample was then split and pulverized to better than 90% passing -75 µm. The sample turnaround time increased to nearly seven weeks after implementing the finer crush, so subsequent samples were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

At Chemex, copper was determined by the four acid digestion method (HF, HNO₃, HClO₄ digestion and HCL-leach) with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was determined by one assay-tonne fire assay analysis followed by AAS. Silver was analyzed using aqua regia digestion and AAS finish.

11.2.4. MintoEx 2009 Samples

The 2009 drill core samples, blanks and SRMs were submitted to the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver. In addition, Chemex was also instructed to perform analysis on pulp and coarse reject duplicates injected into the sample stream at regular intervals.

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Gold was determined using a fire assay procedure on a thirty grams sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

11.2.5. MintoEx 2010 Samples

The 2010 drill core samples, blanks and SRMs were analyzed at the Vancouver Chemex laboratory for copper and gold analysis in North Vancouver. In addition, Chemex was also instructed to perform analysis on pulp and coarse reject duplicates injected into the sample stream at regular intervals. After August 2010, the pulp and coarse reject duplicates were returned to the MintoEx office in Vancouver, where they are transferred to fresh Kraft paper bags, assigned new sample numbers and resubmitted to Chemex as "blind duplicates".

The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

11.2.6. MintoEx 2011 Samples

The 2011 drill core samples, blanks and SRMs were analyzed at the Chemex laboratory for copper and gold analysis in North Vancouver. The pulp and coarse reject duplicates were returned to the MintoEx office in Vancouver, where they were transferred to fresh Kraft paper bags, assigned new sample numbers and resubmitted to Chemex as "blind duplicates". The samples submitted to Chemex were first crushed in a jaw crusher to reduce the material to greater than 70% -10 mesh (2 mm) with a 250 g subsample split and pulverized to better than 85% passing -75 µm.

Copper was determined by aqua regia digestion method with final copper determination by atomic absorption spectroscopy ("AAS"). Non-sulphide copper was analyzed using sulphuric acid leach with AAS determination. Until May 3, 2011, a screen metallic copper method was used on any samples where native copper was logged. This practice was discontinued when comparison of the two methods showed no appreciable difference in results. Gold was determined using a fire assay procedure on a 30 g sub-sample with atomic absorption spectroscopy finish. Silver was analyzed using aqua regia digestion and AAS finish.

11.3. Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability of exploration data. Exploration work by MintoEx was conducted using a quality assurance and quality control program generally meeting industry best practices. All aspects of the exploration data acquisition and management including surveying, drilling, sampling, sample security, and

assaying and database management were conducted under the supervision of appropriately qualified geologists and include written field procedures and verifications.

Analytical control measures typically involve internal and external laboratory control measures to monitor the precision and accuracy of the sampling, preparation and assaying. Insertion of certified Standard Reference Material ("SRM") and blank material ("blanks") monitors the reliability of assaying results and is also important to prevent sample mix-up and monitor potential contamination of samples.

Assaying protocols typically involve regular duplicate and replicate assays to monitor the reliability of assaying results throughout the sampling and assaying process. Umpire assaying is typically performed as an additional reliability test of assaying results by re-assaying a set number of sample rejects and pulps at a secondary laboratory.

ALS-Chemex and SGS implemented internal laboratory measures consisting of inserting quality control samples (blanks and certified reference materials and duplicate pulp) within each batch of samples submitted for assaying.

Quality control procedures used during the 1971 to 2001 drill programs are not known, with the exception of 10 samples submitted for umpire analysis in 1994. The 2001 sample shipments were accompanied by four types of quality control samples, namely: a blank (granodiorite from the site), an ASARCO coarse standard, prepared pulp samples and duplicate splits (coarse ground rejects and the pulverized rejects).

MintoEx inserted an SRM and a blank with every 16 core samples. A coarse reject duplicate and pulp reject duplicate were processed with every 16-17 samples; these were returned to the MintoEx office in Vancouver, transferred to new envelopes and assigned new sample numbers. In 2011, the umpire assaying of pulps at a secondary laboratory was conducted periodically, typically involving analysis of 1% or more of the sawn core samples. The analytical quality control data produced by MintoEx from 2006 to the end of August 2011 are summarized in Table 11-1.

Quality control data analysis was done on all 2011 data and found to be acceptable, but is not included in this report. Refer to the March 2011 Phase V PFS Technical Report and 2009 Phase IV PFS Technical Report for older data.

Table 11-1: Quality Control Data Produced by MintoEx in 2006 through 2011

		2006		2007		2008		2009		2010		2011	
Total Samples Collected		13,121		13,552		15,119		13,056		18,762		14,167	
SRM Used	CGS-5	36	SRM-95	4	SRM-95	13	SRM-95	3	SRM-1	84	SRM-1	50	
	Cu-115	47											
	Cu-116	48							SRM-2	58	SRM-2	26	
	CGS-10	116	CGS-10	120	CGS-10	24	SRM-1	27					
	CGS-7	103	CGS-7	137	CM-3	27	SRM-2	27	SRM-3	33	SRM-3	13	
	CGS-12	54	CGS-12	139	CGS-12	31	SRM-3	24					
	GSP-5	19	CGS-8	17	CGS-17	56	CGS-17	12	CGS-22	152	CGS-22	54	
	GS-2A	17							CGS-23	372	CGS-23	532	
	CM-1	6	CM-2	8	CM-2	99	CM-2	117	CGS-24	126	CGS-24	133	
	CGS-9	109	CGS-9	51	CGS-18	120	CGS-18	190	CM-5	150	CGS-27	20	
	CGS-11	52	CGS-11	175	CGS-11	156	CGS-11	123	CM-8	100	CGS-28	2	
	Cu-132	50	CGS-13	17	CGS-15	177	CGS-15	191	CGS-15	100	CGS-29	8	
	Cu-128	40	Cu-128	15									
	Total SRM		697		682		703		714		1175		849
Blanks		595		674		685		698		1186		848	
Paired Data	Coarse Reject Duplicate	404		556		590		590		1137		436	
	Pulp Reject Replicate	597		702		568		568		1168		416	
Total QC samples		2293		2614		2582		2570		4666		2549	
Frequency (percent)		17		19		17		20		25		18	
Umpire checks (percent)		2		1		0.5		0.5		1		1.5	

11.3.1. Summary of Quality Assurance and Quality Control Programs in 2006 and 2007

Of the 1,269 blank samples analyzed in 2006 and 2007, 11 returned elevated gold and copper results. Internal review by MintoEx indicated six of these erroneous values may have been the result of sample switches. No systematic or long term contamination during sample preparation is evident.

Varying grades of copper and gold SRM were purchased from CDN Resource Laboratories of Delta, BC (“CDN”) and WCM Sales Ltd of Burnaby, BC (“WCM”). In 2007, a copper-only SRM with mean value of 2.59% Cu was purchased from Analytical Solutions Ltd. of Toronto, ON. The SRM were submitted in sequence with the sawn core samples. A total of 1,379 SRM were analyzed. The performance of copper and gold standards was acceptable overall. Performance of SRM for gold improved part way through 2006 when AAS finish was used instead of gravimetric finish after fire assay.

Analyses of the pulp reject and coarse reject laboratory duplicates indicate the 2006 and 2007 sample preparation protocols were excellent for copper analysis and acceptable for gold analysis. To optimize the reproducibility of gold analysis, MintoEx considered increasing the amount of material passing fine meshes during sample prep in 2007. However, no adjustments were made to the sample preparation protocol as any change to the standard preparation procedure was anticipated to increase turnaround time for results to lag times that would have been unacceptable.

In 2006, 5% of the Area 2 samples (approx. 2% of all 2006 sawn core samples) were submitted back to Chemex for blind analysis. Results in all grade ranges were reproducible. For copper, more than 95% had absolute differences of less than 10%; for gold 79% of the pairs were within 15% of each other. One gold outlier pair was removed from the data set.

Umpire pulp check samples representing 1% of the sawn core samples submitted in 2007 were sent to Inspectorate Laboratories in Richmond, Canada. Inspectorate analyzed the check samples using the same analytical procedure as Chemex. Overall, the gold and copper values exhibit unbiased scatter about the mean. No outliers were removed from the dataset. The target for pulp samples analyzed at different labs should have a relative difference not exceeding 15% at the 90th percentile. Copper results for the check samples in 2007 had a relative difference of 12% for the 90% of the population. Gold results for the check samples in 2007 had a relative difference of 15% for 65% of the population. The level of precision is excellent for copper. The level of precision for gold is acceptable but warrants improvement. However, results are shown to be reproducible.

For additional information regarding performance of quality control samples in 2006 and 2007 please refer to *“Technical Report (NI-43101) for the Minto Project”*, Hatch, August 2006 and to *“Area 2 Pre-feasibility Study Minto Mine, Yukon”*, SRK Consulting (Canada) Inc., November 2007 and to *“Minto Mine Technical Report”* SRK Consulting (Canada) Inc., June 2008.

11.3.2. Performance of Blanks in 2008, 2009, 2010 and 2011

MintoEx personnel inserted one field blank sample into the sample stream for every 16 drill core samples submitted for analysis. The blank sample was inserted to ensure sample preparation procedures did not introduce any contamination of gold or copper to the sawn drill core samples.

The field blanks consisted of pieces of local, barren granodiorite, void of any gold or copper values. A total of 685 blanks were submitted with the sawn core samples from the Minto, Area 2/118 and Ridgetop 2008 drilling campaign. In addition, 698 blanks were submitted with sawn core samples from the Minto North, Area 2/118, Copper Keel and Ridgetop 2009 drilling campaign. In 2010, 1186 blanks were submitted with sawn core samples from drilling campaigns covering Minto East, Minto Main, Copper Keel, Area 2/118, Wildfire and Inferno. In 2011 until the end of August, 848 blanks were submitted with sawn core samples from Minto South Deposit (Copper Keel, West Ridgetop, and Wildfire), Inferno, Main Pit, Minto North and Fireweed. Blanks performed very well, showing only very minor, sporadic contamination during sample preparation. The results indicate adequate control procedures during the laboratory's preparation stages in the assaying process.

11.3.3. Performance of SRM in 2008, 2009, 2010 and 2011

Standard reference material (SRM) control samples provide a means to monitor the precision and accuracy of the laboratory assay deliveries. SRMs of varying grades for copper and gold were purchased from CDN of Delta, BC and Analytical Solutions Ltd. of Toronto, ON in 2008 and 2009. Three custom SRMs of varying grades for copper and gold were created from Area 2 coarse reject materials in 2009. The custom SRMs were certified for copper, gold and silver by Dr. Barry Smee of Smee and Associates Consulting Ltd. of North Vancouver, BC. In 2011, two custom SRMs of varying grades for copper and gold were created from Minto East and Wildfire coarse reject material. Details of the results from the SRM assays 2011 are available but not provided in this report. Refer to the March 2011 Phase V PFS Technical Report and 2009 Phase IV PFS Technical Report for older data.

MintoEx personnel inserted one SRM sample within every group of 20 samples. MintoEx considered a copper or gold SRM sample to have failed if a single value exceeded three standard deviations or if more than two consecutive standards fell outside of the two standard deviation limit. When a sample failed, MintoEx reviewed the data and if a re-assay was warranted, the assay laboratory was contacted and instructed to re-assay the failed sample batch. The laboratory was instructed to review the samples for sufficient material for re-analysis. If an SRM had insufficient material left in the sample bag, then the laboratory was supplied with a new standard before re-assaying of the batch began. Some re-assayed samples and internal lab investigations requested by MintoEx are outstanding at the time of this report. For silver SRM, any values outside of the three standard deviations limit or periods of bias were reported to the lab. Re-assays were not ordered unless the SRM also failed for copper or gold.

In 2009, 2010 and 2011, the purchased SRM samples typically performed well for gold analysis. The results are distributed about the mean with periods of bias above and below the mean. The bias is within acceptable limits.

In summary, performance of the SRM samples is acceptable. For copper and gold, most of the charts for each of the SRM show good distribution about the mean with little or no bias. Periods of some bias are evident on some of the charts but all are within acceptable limits. For gold, all SRM assays generally quite closely follow the mean and, as with copper, there is little or no bias.

11.3.4. Performance of Pulp Reject and Coarse Reject Duplicates in 2008, 2009, 2010 and 2011

From 2008 to 2010, within every batch of 20 samples, a pulp reject and a coarse reject (for Chemex only) samples were selected for reanalysis by the geologist logging the borehole to test whether lab methods were sufficient to homogenize material for reproducible analysis.

During the second half of 2010, the pulp materials for pulp reject and coarse reject samples were not analyzed in sequence with the parent samples. Instead, the samples were placed in fresh envelopes, given new sample numbers and SRM samples were inserted every 20 samples.

During 2011, the geologist logging the borehole selected either a pulp reject or a coarse reject sample within every batch of 20 samples for reanalysis. The materials were not analyzed in sequence with the parent samples; the practice of placing the samples in fresh envelopes, assigning new sample numbers and inserting an SRM in every batch continued.

Copper and gold results were again shown to be reasonably reproducible compared to the parent materials. Values are acceptable for resource estimation purposes although the gold in the duplicates is elevated.

A graphical analysis of duplicate quality control data was undertaken but is not provided in this report.

11.3.5. Performance of Umpire Analyses in 2008, 2009, 2010, and 2011

Umpire assaying was done to further check reliability of assay results by re-assaying a set number of sample pulps at a secondary laboratory. The pulps were selected across all grade ranges and repackaged into newly numbered pulp bags with SRM inserted every 20 samples. The target for pulp samples analyzed at different labs was a relative difference not exceeding 20% at the 80th percentile. Generally, the copper and gold values exhibit unbiased scatter about the mean on Q-Q plots. In addition, the target relative differences were met for copper, gold and silver from 2008 to 2011 and to a lesser extent for gold and silver in 2008 and 2009 (refer to the March 2011 Phase V PFS Technical Report and 2009 Phase IV PFS Technical Report for older data.). The level of precision is excellent for copper during the period and improvement was seen for gold and silver in 2010 and 2011. In short, the results were shown to be sufficiently reproducible for resource estimates.

12. Data Verification

12.1. Verification by MintoEx

12.1.1. 1973 to 2001

Independent data verification consisted of drilling by MintoEx, 2005 through 2007, in the Minto Deposit. No confirmation drilling was undertaken in the Area 118 and Ridgetop East. At Ridgetop East, however, two 2007 drill holes were drilled within 30 m of a historic hole, five vertical 2008 drill holes were drilled along the trace of two historic holes and one 2009 hole was drilled within 30 m of a historic hole. At Area 118, three 2008 drill holes were drilled within 40 m of two historic holes. No additional data verification was carried out on historic work. The historic work on the property has been carried out by reputable companies and there does not appear to be any reason to question the validity of the information. Core from the early drilling programs is not useable since both the Falconbridge and ASARCO core sheds have either collapsed and/or burned during regional forest fires, i.e. much of the old core is now in piles on the ground. The core boxes appear to have been labelled by felt pen, rather than metal or plastic tags and the labels on core boxes that remain intact are not legible.

12.1.2. 2005 and 2006

Of the 79 drill holes in the 2006 Area 2 database, eleven collars (13%) were selected at random in the area of the resource estimation boundaries and were checked by a handheld Garmin GPS. Table 12-1 compares the results of the collar locations as documented by SRK and Sherwood Copper. MintoEx sighted the drill hole collars by differential GPS, which were later surveyed by the Minto Mine Survey team. The recorded values show good agreement and differences lie within the error of the handheld GPS.

Table 12-1: Comparison of Selected Drill Hole Collars by SRK and MintoEx

Hole ID	Collars – SRK Handheld GPS				Collars – Minto Mine Survey		
	Easting	Northing	Elevation	Accuracy	Easting	Northing	Elevation
06SW068	384948	6944463	860	7	384949	6944461	854
06SW095	384914	6944522	858	7	384917	6944523	851
06SW114	384975	6944503	851	5	384979	6944499	844
06SW115	384854	6944467	872	3	384855	6944465	865
06SW116	384878	6944521	864	6	384880	6944519	857
06SW122	384938	6944379	870	3	384940	6944378	861
06SW133	385037	6944601	829	3	385039	6944600	821
06SW151	384980	6944622	834	3	384981	6944621	825
06SW153	384918	6944603	845	5	384919	6944600	835
06SW168	385081	6944561	827	3	385083	6944558	818

12.1.3. 2008

In December 2008, MintoEx conducted a review of the drilling data from Area 2/118 and Ridgetop deposits. A total of 10% of the values in the database were checked against primary sources including the borehole collar surveys against survey records, lithology and mineralization data against core logs and assays for copper and gold against signed certificates of analysis. No significant errors were found.

12.2. Verification by Kirkham Geosystems

In November of 2009, Kirkham Geosystems manually compared the Minto North Deposit database assays against original assay certificates. A total of 15% of the values were checked and no errors or omissions were found. In addition, a spreadsheet check was run against the Area 2, Area 118 and Ridgetop database.

12.3. Verification by SRK

12.3.1. Site Visits

In 2007, In accordance with NI 43-101 guidelines, MintoEx commissioned SRK to provide an independent verification of exploration data for Area 2. Data verification consisted of a site visit, examination of drill hole collars, examination of selected drill core and a check of the assay database against original laboratory certificates. Andrew Ham of SRK visited the Minto property between the 24th and 26th of January, 2007. Dr. Ham personally inspected drill core storage facilities, drill collars and selected drill core from mineralized zones within the Area 2 resource. In addition, he personally checked collar coordinates in eleven drill holes with a handheld Garmin GPS (see Table 12-1).

Wayne Barnett visited the Minto property between the 4th and 6th of March, 2009, and again from the 6th to 10th April 2011. Dr. Barnett personally inspected the drill core logging and storage facilities and a drill site. Mineralized and non-mineralized drill core was reviewed and the geological logging procedure was discussed with the core loggers. Sample bags were inspected for tags and the sampling tagging process was reviewed. During the 2011 visit, Dr. Barnett spent time geologically logging drill holes and mapping in the Minto Main pit in order test and improve the current geological understanding of the controls on mineralization.

12.3.2. Verification from Electronic Lab Files

SRK compared electronic lab files from 2008 and 2009 drill campaigns in 2009 as described in the SRK Technical Report, December 2009. The electronic lab files for 2009, 2010 and 2011 were sent directly to SRK by Chemex and SGS Labs. Approximately, 90% of the Cu and Au assays were checked. The assays were found accurately compiled, i.e., current assay database is an accurate reflection of Cu and Au assay grades generated by the labs. Ag assays were spot checked and were not extensively verified. No problems were found.

12.3.3. Comparison of Assays from Historical and New Drill Holes

All assays older than 2006 in Area 2/118 and 2007 in the Ridgetop area have been designated as historical. The comparison was carried out on 3.0 m composite Cu assay grades within mineralized

domains. To compare the data, a nearest neighbour block model was created. Only the blocks estimated from both datasets within a maximum distance of 30 m from the nearest sample were compared. Figure 12—1 show Q-Q plots of the block estimates from the historical and the MintoEx data. Overall, the historical data compare well with the new data, indicating no bias between the two data sets. Based on the results, the historical data have been included in the resource estimates.

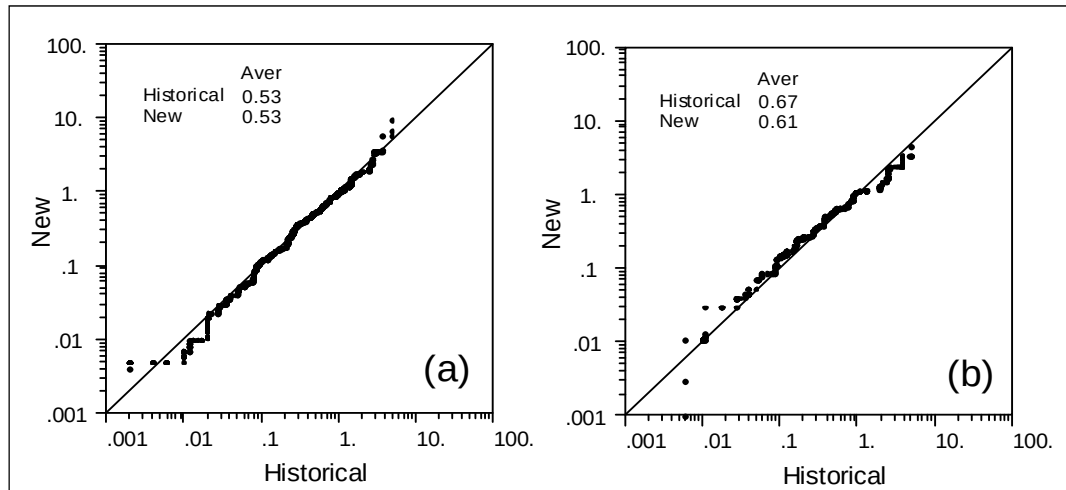


Figure 12—1: Comparison of historical and new data in: (a) Area 2/118 and (b) Ridgetop

13. Mineral Processing and Metallurgical Testing

13.1. Introduction

Metallurgical testing by Capstone Gold SA was performed with oversight by Qualified Person John Eggert P.Eng. on the four proposed new zones at the Minto mine site. The zones were Copper Keel North East, Copper Keel South, Wildfire Upper and Wildfire Lower.

The objective of the test program was to estimate the plant performance for the zones.

Findings from the test work program are summarised in the following reports from Capstone Gold SA:

- Estudio Metalurgico – Zona CKNE
- Estudio Metgalurgico – Zona WF-Lower

Samples received comprised composite samples of individual diamond drill cores. The cores were chosen by Minto Exploration Geology to represent the various ore zones. As such, there is a geological variation of core within the ore zones. Each individual composite was the total of the remainder of the core split that was originally assayed for geological purposes. Thus for each zone, there were a number of composite samples, that will be referred to as Core Composites. For each zone, the core composites were further combined, to generate a Zone Composite.

Core composites were tested in open circuit flotation to provide an expected range of recovery and concentrate grades. Composite test conditions were identical for each test; hence estimates of expected grind variability are also possible.

Zone composites were tested in locked cycle tests to provide a robust estimate of recovery and grade for the zone and to assess mass pulls and reagent consumptions.

13.2. Historical Testing

Over the years, many metallurgical testing programs on the Minto ores have been undertaken. In addition, many plant trials have been undertaken. These have been used to optimize circuit performance and provide guidelines for changes to installed equipment. In addition to the testing by outside organizations, in plant trials continue to be undertaken. The Phase IV and Phase V pre-feasibility studies provide detailed summaries of previous metallurgical testing.

Operation data from 2011 is summarized in Table 13-1. Tonnage increased from approximately 2,800 tonnes per day to approximately 3,500 tonnes per day. This is an increase in tonnage of about 25%. During this time, operations treated historically problematic stockpiles. These stockpiles contained fully oxidized minerals that are not indicative of the majority of the ore zones. The increase in tonnage and the treatment of problematic stockpiles resulted in recovery dropping from about 95% to approximately 87% or about 8%. Various mill design initiatives have been pursued and / or are being implemented that will enable recovery to be increased to 92% which has been reflected in the economic analysis. The 5% increase in recovery from the lowest values seen in 2011, can be ascribed to process improvements, such as the use of alternate reagents. The 3% drop in recovery from the highest values seen in 2011 can be ascribed to the increase in tonnage.

Table 13-1: Minto Mine Mill Performance Summary, 2011

	January	February	March	April	May	June	July	August	September	October	November	December
Mill Feed Dry Tonnes	87,770	77,348	93,039	96,738	115,913	107,493	105,880	119,969	117,209	114,301	103,920	118,728
Grind P80 (um)	195	220	211	211	241	221	203	205	227	215	222	243
Feed Grade % Cu	1.61%	1.42%	1.60%	1.79%	1.66%	1.75%	1.56%	1.52%	2.49%	1.39%	1.10%	1.32%
Feed Grade gpt Au	0.63	0.56	0.67	0.75	0.71	0.67	0.54	0.52	1.60	0.45	0.69	0.36
Feed Grade gpt Ag	5.47	5.52	6.39	6.56	5.44	8.39	4.55	5.29	9.06	4.54	5.19	4.71
Plant Availability	92.1%	91.5%	94.8%	93.6%	94.1%	88.3%	91.4%	93.2%	95.7%	90.6%	94.8%	96.1%
Crusher Availability	80.7%	75.3%	81.5%	82.4%	85.7%	83.1%	71.9%	87.4%	83.7%	82.6%	70.6%	87.2%
Copper Conc Wet Tonnes	3,538	3,215	3,866	4,600	5,365	4,890	4,403	4,925	8,176	4,063	3,323	3,980
Copper Conc Moisture	7.4%	7.6%	9.0%	9.9%	10.8%	4.9%	8.1%	8.6%	11.1%	10.4%	12.9%	12.8%
Copper Conc Dry Tonnes	3,274	2,970	3,518	4,143	4,787	4,650	4,047	4,499	7,266	3,642	2,893	3,470
Copper Conc Grade % Cu	40.0%	35.0%	37.2%	37.0%	35.1%	36.3%	36.3%	35.8%	36.5%	36.9%	32.3%	37.8%
Copper Conc Grade gpt Au	12.4	11.1	12.6	12.9	12.9	11.9	10.7	10.5	18.6	9.7	12.1	8.6
Copper Conc Grade gpt Ag	156	117	130	122	104	149	94	115	125	112	148	125
Copper Tonnes Produced	1,310	1,039	1,309	1,534	1,680	1,687	1,468	1,609	2,652	1,342	934	1,310
Gold Grams Produced	40,481	32,959	44,469	53,430	61,664	55,404	43,315	47,188	134,875	33,551	55,733	31,572
Silver Grams Produced	509,536	364,426	455,705	503,497	498,002	694,164	379,293	516,318	905,175	406,130	428,010	433,257
Tailings Grade % Cu	0.08%	0.08%	0.20%	0.21%	0.22%	0.19%	0.18%	0.19%	0.25%	0.22%	0.21%	0.22%
Tailings Grade gpt Au	0.16	0.14	0.20	0.20	0.19	0.17	0.14	0.14	0.48	0.16	0.16	0.09
Tailings Grade gpt Ag	1.06	1.06	1.55	1.41	1.20	2.02	1.00	1.02	1.43	1.02	1.11	1.09
Tailings Dry Tonnes	84,496	74,378	89,521	92,595	111,126	102,842	101,833	115,469	109,942	110,659	101,027	115,258
Overall Copper Recovery	95.4%	94.4%	88.1%	88.6%	87.4%	89.6%	89.1%	88.1%	90.7%	84.6%	81.6%	83.7%
Overall Gold Recovery	73.5%	76.0%	70.9%	73.9%	74.9%	76.4%	75.1%	75.1%	72.0%	65.4%	77.7%	74.8%
Overall Silver Recovery	81.1%	81.5%	76.7%	79.4%	78.9%	76.9%	78.8%	81.4%	85.2%	78.2%	79.3%	77.5%
Days	31	28	31	30	31	30	31	31	30	31	30	31
Feed Tonnes/Day	2,831	2,762	3,001	3,225	3,739	3,583	3,415	3,870	3,907	3,687	3,464	3,830

13.3. Test Work Program

The metallurgical test work program used as the basis for this report consisted of flotation test work on core from the following areas identified by Capstone geology:

- Copper Keel North East;
- Copper Keel South;
- Upper Wildfire; and
- Lower Wildfire.

The test program comprised open circuit testing on each core composite and a locked cycle test at the target grind of the plant using a flow sheet that duplicates the current operation. No attempt was made to further optimize the circuit. Optimization work is an ongoing mission of the metallurgical department at Minto. Changes that result from these programs will maintain or improve on the results reported here.

13.3.1. Test Work Comments

The core samples at Minto are not stored in nitrogen as a matter of course. This results in some potential surface oxidation of ore minerals. As such, the testing results from the Cozamin lab will be conservative with plant performance expected to exceed the lab results.

The locked cycle testing will reflect the recoveries at Minto at lower tonnages. Plant performance for these zones can expect to be impacted similarly to the impact on current operations, should elevated tonnage rates be continued without changes to the circuit.

13.3.2. Sample Origin

Samples for the new Minto deposits totalled 106 with a combined weight of 314 kg. The distribution of the samples for each new deposit was:

- Copper Keel North East, 89 samples from 5 holes, of which 54 were ore, the remainder low grade and waste for dilution
- Copper Keel South, 33 samples from 3 holes, of which 23 were ore, the remainder low grade and waste for dilution
- Wildfire Upper, 60 samples from 4 holes, of which 35 were ore, the remainder low grade and waste for dilution
- Wildfire Lower, 72 samples from 4 holes, of which 41 were ore, the remainder low grade and waste for dilution.

13.3.3. Flotation Test Work Results

Flotation test work for the Minto Phase VI study was completed by Laboratorio Metalurgico Cozamin, of Capstone Gold SA in Zacatecas Mexico. All of the test work focussed on bulk sulphide flotation in accordance with the existing Minto plant to produce a copper concentrate. Analysis of the test work was used to determine expected grades and recoveries for the ores under treatment using the current Minto flow sheet.

13.3.4. Copper Keel North East

Five core composites were prepared. These were floated first in a single open circuit to estimate maximum expected recovery and an open circuit test with cleaning to estimate maximum expected grade. Head assays for these ranged from 0.78 % Copper to 2.37 % Copper. Recovery ranged from 89% of copper to the concentrate to 95.3% to concentrate. Concentrate grades ranged from 33.4% Copper to 40.9% copper. The locked cycle test provides a single point recovery of 92.7% of copper to concentrate with a concentrate grade of 36.6% Copper on a head grade of 1.65% copper (diluted grade).

13.3.5. Wildfire Upper

Four core composites were prepared. These were floated first in a single open circuit to estimate maximum expected recovery and an open circuit test with cleaning to estimate maximum expected grade. Test results for these were highly variable. Plant operation will require heightened care to assure that performance is maximized. Head assays for these ranged from 1.25 % Copper to 2.05 % Copper. The acid soluble component of copper mineralization in this zone is significant. Recovery ranged from 41.2% of copper to the concentrate to 96.9% to concentrate. Lower recoveries occurred in those samples with a higher soluble copper value. As such, these would be expected to have a poorer performance in the standardized procedure used to assess the range of recovery. Concentrate grades ranged from 26.6% Copper to 50.4% copper. The best grades also occurred with lower recovery. This indicates that worst case plant recoveries can be expected to be higher than indicated by the laboratory tests. The locked cycle test provides a single point recovery of 67.8% of copper to concentrate with a concentrate grade of 23.9% Copper on a head grade of 1.70% copper (diluted grade). This result is not considered to be representative of the expected plant recovery. For the purposes of cash flow estimates, it is considered likely that the performance indicated by the Wildfire lower zones would be suitable.

13.3.6. Copper Keel South

Three core composites were prepared. These were floated first in a single open circuit to estimate maximum expected recovery and an open circuit test with cleaning to estimate maximum expected grade. Head assays for these ranged from 2.17% Copper to 2.39% Copper. Recovery ranged from 85.4% of copper to the concentrate to 94.5% to concentrate. Concentrate grades ranged from 43.7% Copper to 53.9% copper. The locked cycle test provides a single point recovery of 85.1% of copper to concentrate with a concentrate grade of 35.2% Copper on a head grade of 2.1% copper (diluted grade). The high grades of some of the concentrates indicate a higher proportion of chalcocite and similar minerals that would be subject to oxidation effects. As such, it is reasonable to assert that the copper recovery to concentrate will be higher than projected by the locked cycle tests. For budgeting purposes, a recovery of 87% would be conservative. Mill performance can be expected to be higher than this, but it is impossible to confirm by how much.

13.3.7. Wildfire Lower

Four core composites were prepared. These were floated first in a single open circuit to estimate maximum expected recovery and an open circuit test with cleaning to estimate maximum expected grade. Head assays for these ranged from 0.90 % Copper to 1.47% Copper. Recovery ranged from 65.1% of copper to the concentrate to 77.6% to concentrate. The lower recoveries

are attributable to the relatively low grades of the composite samples which resulted in the reduced metallurgical performance.

Concentrate grades ranged from 31.2% Copper to 42.0% copper. The locked cycle test provides a single point recovery of 94.5% of copper to concentrate with a concentrate grade of 32.9% Copper on a head grade of 1.17% copper (diluted grade).

These results have the greatest variances of all of those tested. Locked cycle values are appropriate for budgeting, but operations personnel must be informed of the sensitive nature of this zone to avoid wide swings in mill performance.

13.3.8. Summary

The laboratory testing indicated that all of the ore zones will respond similarly to the Minto flow sheet and further that this performance will be similar to current operations. The following results are considered conservative.

At design tonnages, the following results can be expected:

Copper Keel North East:	Recovery of Copper to Concentrate: 92.7% at 36.6% grade
Wildfire Upper:	Recovery of Copper to Concentrate: 94.5% at 32.9% grade
Wildfire Lower:	Recovery of Copper to Concentrate: 94.5% at 32.9% grade
Copper Keel South:	Recovery of Copper to Concentrate: 87% at 34% grade

At the elevated tonnages currently being achieved at Minto, concentrate grades are being maintained, but recovery has been derated from 97% to 92% that is based on recovery enhancement programs that commenced in 2011.

Recovery for the Copper Keel South has not been de-rated as much. For budgeting purposes, this is reasonable, but further testing may be necessary.

13.4. Mineral Processing

Prediction of the flotation performance was determined following analysis of the locked cycle tests. The design recoveries of the target metals are generally in line with, or slightly lower than those achieved in the locked cycle tests suggesting a degree of conservatism.

The average copper recovery to final concentrate is 91.7%. Average gold recovery is 70.6% and the average silver recovery to final concentrate for the locked cycle test work was 78.2%.

The overall project economics for the Study were based on:

- Copper recovery of 92%;
- Gold recovery of 70%; and
- Silver recovery of 78%.

In light of the test work to date, Eggert Engineering believes these recoveries are reasonable for a copper concentrate from the new Minto ore bodies based on the flow sheet selected for the upgraded plant. The overall design grade and recovery numbers predicted by Eggert and Minto are shown in the values selected are generally lower than the actual test work values shown. No

deleterious elements were found in the new deposits. Eggert Engineering believes the metallurgical samples can be considered representative for the deposits.

Multi element ICP scans of the head samples were performed. These indicate very low to no detection for arsenic, mercury, chromium and molybdenum. The metallurgical sampling of all samples is considered representative, with the exception of the Wildfire Upper samples. These exhibited a very wide range of metallurgical performance that has never been reflected by plant performance. It is possible that some of these samples were contaminated with fully oxidized materials that are not sent to the mill.

14. Mineral Resources Estimates

14.1. Introduction

To the best of SRK's knowledge and as described above, there are currently no title, legal, taxation, marketing, permitting, socio-economic or other relevant issues that may materially affect the mineral resources described in this Technical Report. Future changes to legislation (mining, taxation, environmental, human resources and related issues) and/or government or local attitudes to foreign investment cannot be, and have not been evaluated within the scope of this Technical Report.

SRK's findings are based on reviews of readily available data sources at the time of preparing this report. The mineral resources presented herein represent an update to the 2010 resource evaluation described in the Minto Phase V Pre-Feasibility Technical Report (effective date December 15, 2010). It also supersedes the mineral resource update publically disclosed on May 30, 2011. A primary objective of SRK's work was to produce a revised independent mineral resource evaluation for the Wildfire extension of the Minto South deposit (previously called Minto Area 2/118).

The mineral resource for the Ridgetop deposit has not been modified from the previous estimate with an effective date of August 30, 2010. The Minto North deposit was evaluated by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd (Kirkham Geosystems) and completed with an effective date of December 1, 2009. The Minto East deposit was also completed by Kirkham Geosystems, with an effective date in December 15, 2010. The Minto North and East zones of mineralization have also not been revised. The Minto Main deposit was updated by Lions Gate Geological Consulting Inc., and independently verified by SRK as reported in the Area 2 Pre-feasibility Study Minto Mine technical report (2007) and the Minto Phase IV Pre-Feasibility Technical Report (2010).

The resource estimate in the Minto South and Ridgetop deposits was completed by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., of SRK, an independent qualified person as defined in National Instrument 43-101. The effective date of the revised Minto South deposit is September 29, 2011. Marek Nowak, P.Eng., of SRK, analyzed the data, reviewed and validated the mineral resource estimates.

This section describes the work undertaken by SRK and Kirkham Geosystems, including key assumptions and parameters used to prepare the mineral resource models for Minto South, and Ridgetop deposits together with appropriate commentary regarding the merits and possible limitations of such assumptions. The following discussion concentrates on Cu grades, the most valuable commodity in the Minto deposits.

In the opinion of SRK and Kirkham Geosystems, the block model resource estimate and resource classification reported herein are a reasonable representation of the global mineral resources at Minto South, Ridgetop, Minto Main, Minto North and Minto East deposits at the current level of sampling. The mineral resources presented herein have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with Canadian Securities Administrators' National

Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The estimated mineral resources have been used in the preliminary feasibility study described in this report.

14.2. Resource Database

The database used to estimate the Minto South and Ridgetop deposits was prepared by MintoEx personnel and verified by SRK. The Minto North and Minto East database was also prepared by MintoEx, and verified by Kirkham Geosystems. The mineralized domains of the deposits were modelled using Gemcom software based on lithological and structural interpretations.

SRK is of the opinion that the current exploration and structural information is sufficiently reliable to confidently interpret the mineralized boundaries and that the assay data are sufficiently reliable to support the estimation of mineral resources.

Table 14-1 provides a summary of the samples included in the Minto South, Ridgetop, Minto East and Minto North deposits database. Note that the actual number of samples within the modelled geology domains was lower.

Table 14-1: Exploration Data within the Modelled Deposits

Project	Year	DD Drill holes	Number of Drill Samples	Sample Length Total (m)
Minto South	2011	74	9,543	14,817
	2010	109	11,670	16,452
	2009	27	2,854	3,938
	2008	49	5,586	6,895
	2007	56	7,265	9,788
	2006	80	9,391	14,799
	Historical	28	742	2,071
	TOTAL	423	47,051	68,760
Ridgetop	2009	71	5,067	7,330
	2008	46	4,177	5,604
	2007	25	1,993	2,704
	Historical	21	840	2,068
	TOTAL	163	12,077	17,706
Minto East	2007	1	168	380
	2008	1	103	385
	2009	4	575	1,410
	2010	27	3,772	8,892
	Total	33	4,618	11,067
Minto North	2009	87	4,651	11,263
	Total	87	4,651	11,263

14.3. Minto South Deposit

14.3.1. Geology Model

The Minto South deposit is composed of the Area 2 (including “Copper Keel” extension), Area 118 and Wildfire zones that form part of the same system of mineralization (Figure 14—1). Area 118 zone is recognized to be structurally more complex and the boundary between Area 118 and Area 2 zones is defined in this study to be a fault dipping at 50° towards the northeast. The 2010 update of the resource included the Copper Keel southeastern extension to Area 2, but for the purposes of the resource definition Copper Keel is inclusive within Area 2. The Wildfire zone is interpreted to be an extension of the Area 2 zone further towards the east, and the modification forms the primary component of the 2011 Minto South resource update. SRK have only modified the solids representing the Area 2 and Wildfire zones of mineralization for the 2011 update of the Minto South deposit. The Area 118 zone has not been modified and has not been re-estimated.

The copper, gold and silver mineralization in the Minto South deposit is associated with foliated granodiorite lithological units. The background non-mineralized rock is an unfoliated granodiorite. To constrain the interpolation during grade estimation, SRK built three dimensional solids of the foliated granodiorite units. They are modeled to be generally shallow dipping (19 to 30°) towards the northeast.

The geological nature of foliated zones is still under investigation. They appear in high strain zones with a strong closely spaced foliation developed from coaxial strain. Locally there are textural indications of rotational strain but the nature and timing of the deformation requires further investigation. The continuity of the foliated zones has been established by multiple drill hole intersections of the zones showing that the zones in a particular deposit to be traceable over the entire deposit.

The foliated zones have mineralogical, geochemical, grade and textural signatures that can be picked up in the logs and assays data, and can be used to identify zones and show continuity at least over several hundred meters. The style of mineralization also appears identical for all the other deposits in the area. In particular, the Main Minto deposit is currently being mined and the continuity of mineralization can be established without question.

There are number of aspects that complicate the resource continuity:

The zones bifurcate, which means that a mineralized zone can contain a significant amount of waste, or that thinner ore zones can merge with larger zones. A bifurcating geometry complicates geological modeling and may expect to increase internal dilution.

The width and dip of mineralized zones are locally variable. The zones therefore appear to pinch-and-swell. The change in thickness might be as much as an order of magnitude over less than 30 m in horizontal distance.

At least some of the irregularity in the geometry and thickness of the mineralized zones is due to small-scale and large-scale structural displacements. No detailed structural model has been completed for either deposit, but at least two faults appear to be present in Area 2, and three possible faults displace the modeled zones in Area 118. Similar structures may be present throughout the deposit, each with displacements of a few meters or less.

On the other hand, the Minto Main deposit pit exposures and the large number of drilling intersections define the range of possible geometries fairly well, and reduce the risk of incorrect geological interpolation away from known data. In addition, the understanding of the local geometries has been a successful factor in local exploration.

The updated Minto South resource model was created using a commercial three-dimensional block modelling and mine planning software, GEMS[®] version 6.3. The models were created in metric units using the mines local co-ordinate system (UTM NAD83 zone 8). The mineralized zone solids were considered hard boundaries where grades were not allowed into blocks outside of these solids.

The mineralized zone solids were built using top and bottom Laplacian grid surfaces that pass through the vertices representing the top and bottom drill hole intersecting contacts. The interpretation was initially done using vertical sectional interpretations provided by MintoEx geologists as references. These sections are spaced on 25 m intervals. SRK reviewed, adjusted and resolved the interpretations where necessary. The contacts for a specific contact surface are made active by snapping polylines to the drill hole vertices, such that the polyline vertices are then used by GEMS as controls on the surface gridding. The grid triangulation vertices are saved as points. The final contact surface is then created from the imported grid points and the original polyline vertices using a regular surface creation technique. This final surface has the surface triangulation vertices snapped precisely to both the grid points and the polyline vertices. The result is a contact surface that looks like a smoothed Laplacian grid but actually snaps to the drill hole intersections. The surfaces are then used to clip out or “carve-off” the mineralized zone domains and waste domains from an original solid wireframe representing the entire resource extents.

Up to 9 primary mineralized zones were assigned the following domain codes historically used by MintoEx geologists; J, K, L, M, N , O, P, Q, and R. Table 14-2 includes a list of the domain coding assigned to the drill data and the block model. Note that additional zones were modeled as bifurcations of the primary zones (noted in Table 14-2). These bifurcations are closely associated with the primary zones and for the purpose of the interpolation were considered part of the primary zone. Figure 14—1 is a 3-dimensional view of the zone solids, showing their block model codes (or Zone-ID).

Table 14-2: Primary Mineralized Zones in the Minto South Deposit

Domain Name	Block Model Code		Comments
	Area 2	Area 118	
J	20	21	Includes zone L2 Primary grade bearing domain. Includes zone M2
K	30	31	
L	40	41	
M	50	51	
N	60	61	Very thick domain in Area 2. Appears to become thinner with weaker foliated texture in Area 118.
O	70	71	
P	80	81	Includes zone Q2
Q	90	91	
R	100	101	
Overburden (OB)	500		
Air	0		
Waste	200		

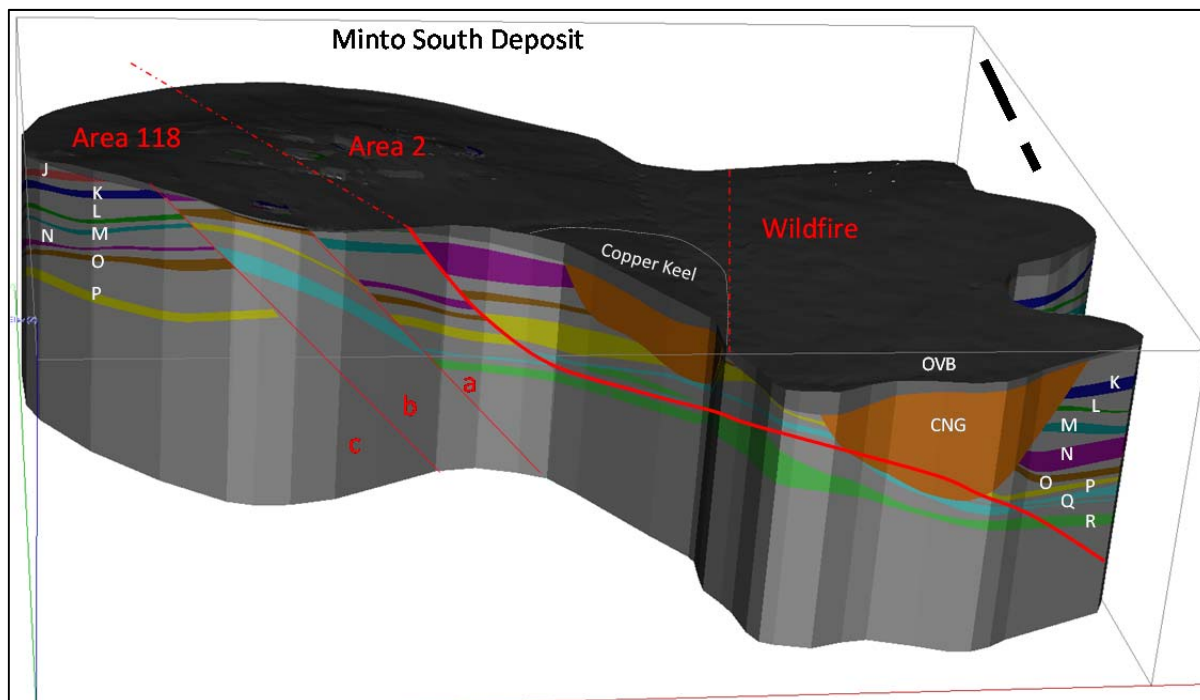


Figure 14—1: Axonometric View Northwards of Minto South showing the approximate zones defined as Area 2, Area 118 and Wildfire. Mineralization Domain Solids are labeled. The location of the Copper Keel extension referred to in previous sections is also marked.

The boundary between Area 2 and Area 118 zones has been modeled as a fault. The drill hole intersections are of sufficient density to show the position of the fault accurately. Two additional faults have been modeled in order to explain intersection positions in Area 118, and these faults divide the Area 118 resource into three domains (labeled a, b and c in Figure 14—1). No study has been done on the drill core in order to define the characteristics of the faults. There are indications that these faults have the characteristics high strain shear zones, rather than brittle structures.

The solids were then used to assign the domain and block model codes to the drill hole data (assays and composites) and the block model cells. Blocks above the topography surface were tagged as Air and the blocks outside of the zone solids were tagged as Waste.

14.3.2. Data

Nearly 40,000 grade measurements have been used in the update (Area 2 and Wildfire zones) of the Minto South mineralized domains from holes drilled roughly at 30 to 40 m spacing. Close to 30% of the samples within the modelled domains were collected from 1.5 m intervals (Figure 14—2). All assays were composited to 1.5 m lengths.

Choice of the shorter composite length was guided by a small proportion (approximately 17%) of relatively narrow, less than 4.5 m, mineralized zones. Shorter composite lengths ensured that most relevant, undiluted assays were included in the resource assessment.

Within the mineralized domains 33,832 composite assays were produced from 391 holes. Statistics of polygonally declustered 1.5 m copper and gold composites within each mineralized zone are presented in Figure 14—3 thru Figure 14—5.

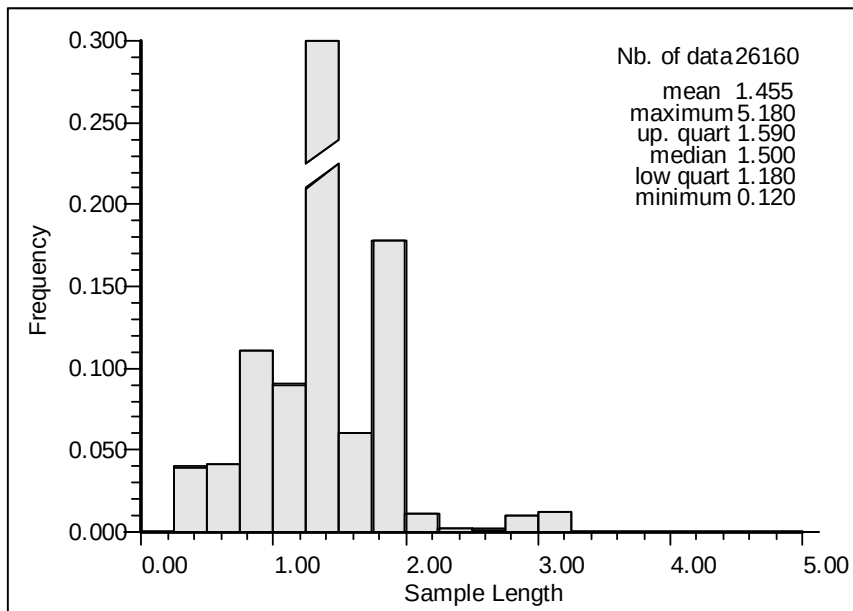


Figure 14—2: Area 2 and Wildfire regions of MSD Update - Histogram of Sample Lengths

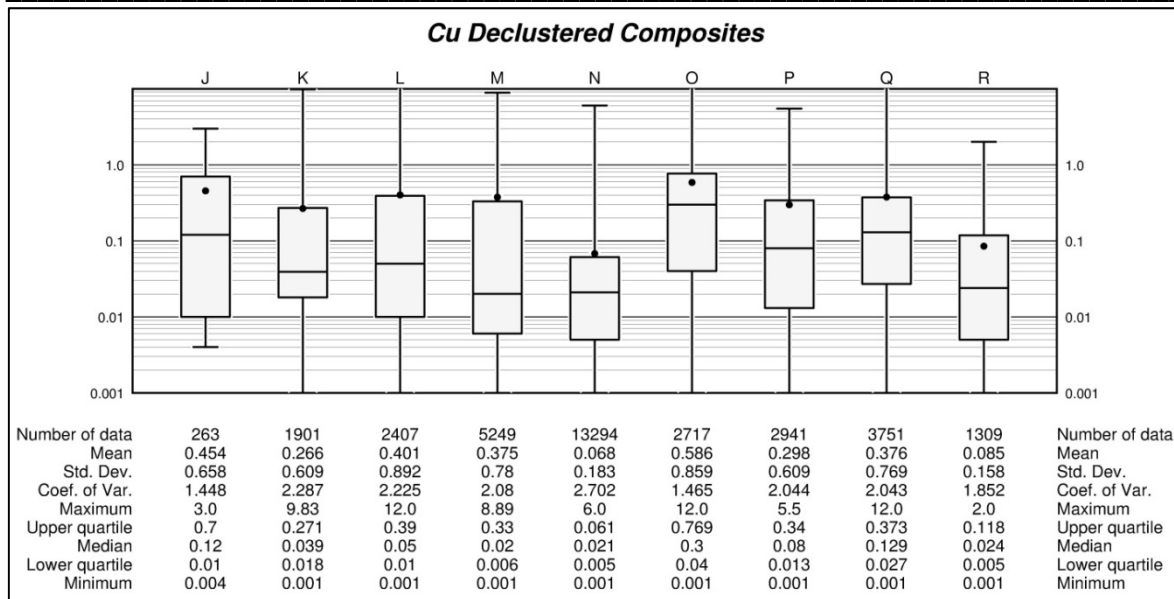


Figure 14—3: Area 2 and Wildfire regions of MSD Update - Basic Statistics of Declustered Copper Composite Grades

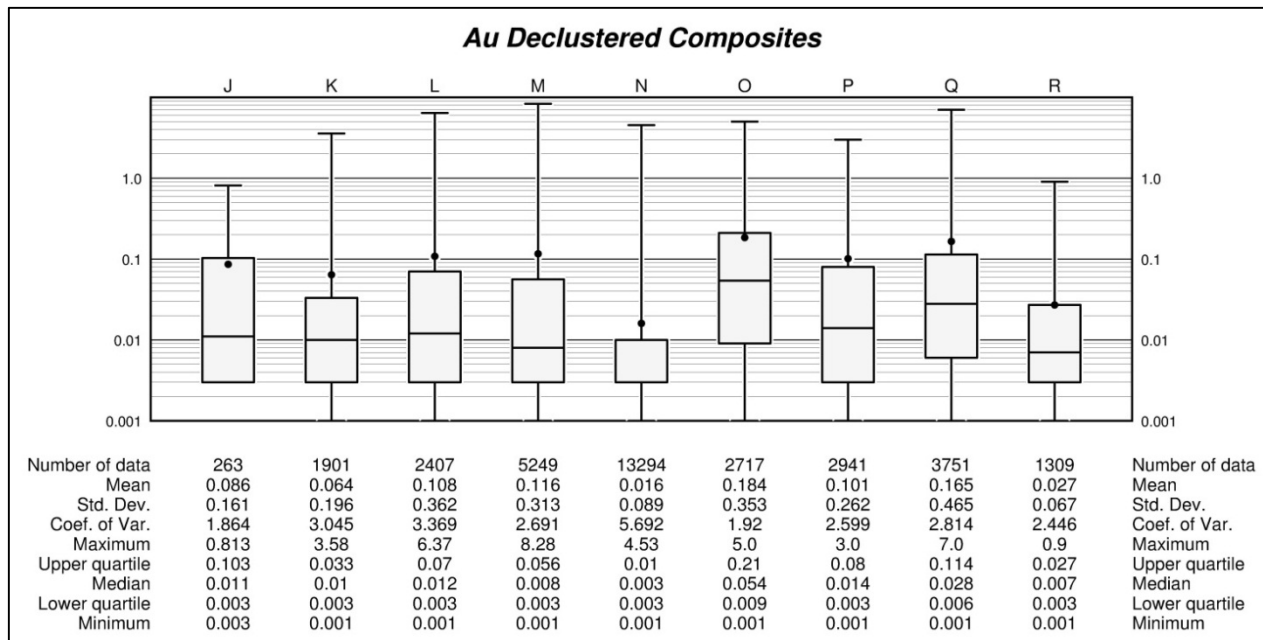


Figure 14—4: Area 2 and Wildfire regions of MSD Update - Basic Statistics of Declustered Gold Composite Grades

Figure 14—5 shows bivariate statistics of the copper and gold composite assays. Note: very good correlation, indicated by a regression curve (white thick line) showing a general tendency of increased copper assays for higher gold assays. This high correlation lead to a design of variogram models along identical major directions of continuity for both copper and gold grades.

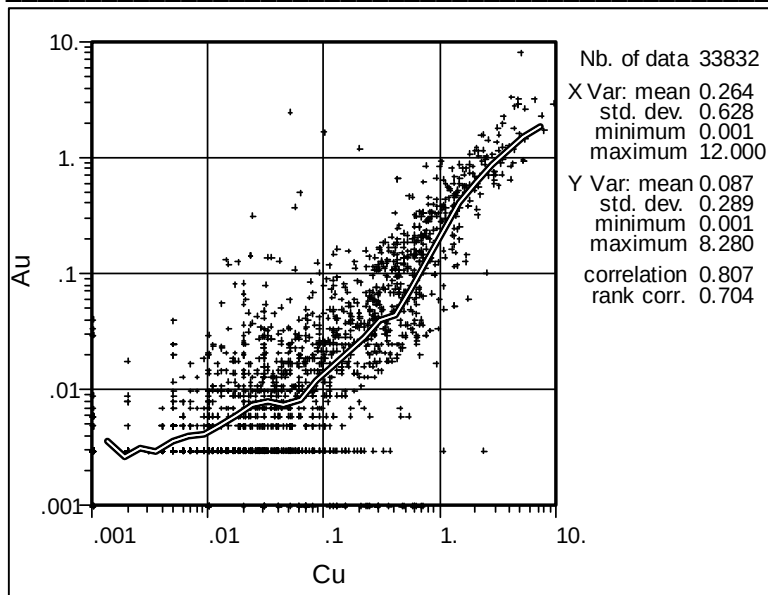


Figure 14—5: Area 2 and Wildfire regions of MSD Update - Bivariate Statistics of Copper and Gold Assays

14.3.3. Evaluation of Extreme Assay Values

Block grade estimates may be unduly affected by very high grade assays. Therefore, the assay data were evaluated for the high grades outliers. An analysis of the high grade assays indicates negative correlation between the assay data and the sample lengths (Figure 14—6).

This suggests that sampling was based on visual indications of mineralization. In view of the above, no capping was done before assay compositing to 1.5 m lengths.

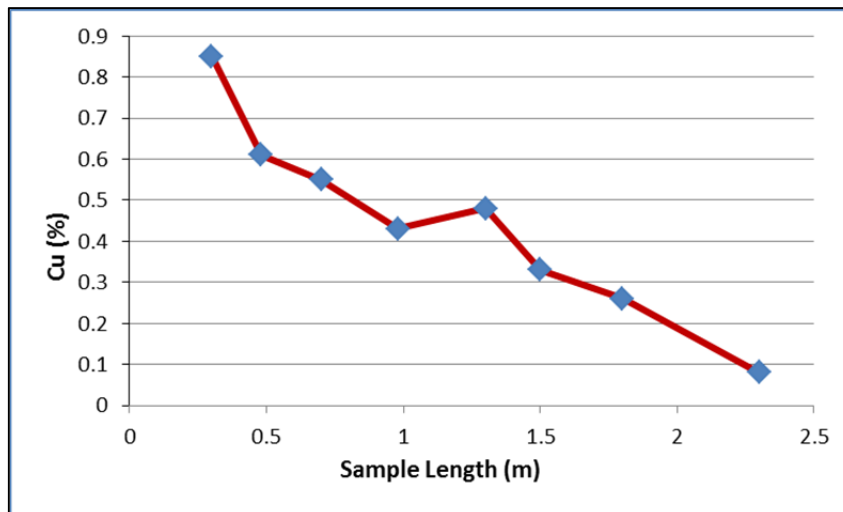


Figure 14—6: Area 2 and Wildfire region of MSD Update - Grade Variation with the Sample Length

14.3.4. Variogram Analysis

Experimental variograms and variogram models in the form of correlograms were generated for copper and gold grades in Area 2 and its Wildfire extension. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values

for copper range from 10 to 45 percent of the total sill. The nugget values for gold range from 5 to 35 percent of the total sill. Note that the sill represents the grade variability at a distance beyond which there is no correlation in grade. Variogram models used for copper and gold grade estimation are summarised in Table 14-3 and Table 14-4. The variogram models used for Cu in Area 118 zone in 2010 are shown in Table 14-5.

Note that no variogram models were designed for silver grades. Silver was estimated by the inverse distance squared method.

Table 14-3: Area 2 and Wildfire regions of MSD Update: Copper Exponential Variogram Models

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J	0.1	0.20	-60	-15	0	50	30	10
		0.70				90	40	18
K	0.10	0.40	-30	-15	0	60	60	8
		0.50				220	120	50
L	0.15	0.40	45	15	0	30	70	8
		0.45				400	150	50
M	0.25	0.30	-60	-15	0	200	200	12
		0.45				800	300	200
N	0.30	0.30	45	15	0	15	15	7
		0.40				100	130	40
O	0.25	0.30	-95	-15	50	50	75	5
		0.45				500	200	50
P	0.25	0.30	-75	-15	0	15	15	15
		0.45				400	400	70
Q	0.45	0.30	65	15	-95	50	30	10
		0.25				250	80	40
R	0.25	0.40	30	15	-90	15	15	8
		0.35				180	100	14

Table 14-4: Area 2 and Wildfire regions of MSD Update: Gold Exponential Variogram Models

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J	0.2	0.25	-60	-15	0	80	15	8
		0.55				100	40	15
K	0.05	0.60	-30	-15	0	20	20	10
		0.35				120	60	50
L	0.15	0.40	45	15	0	30	120	20
		0.45				400	150	40
M	0.20	0.30	-60	-15	0	60	30	12
		0.50				800	300	100
N	0.20	0.55	45	15	0	15	40	6
		0.25				80	150	50
O	0.35	0.20	-95	-15	50	100	150	10
		0.45				500	300	60
P	0.25	0.20	-75	-15	0	15	15	14
		0.55				450	450	60
Q	0.22	0.10	65	15	-95	70	70	20
		0.68				200	120	45
R	0.30	0.20	30	15	-90	30	30	8
		0.50				100	60	10

Table 14-5: Area 118 region of MSD Cu Exponential Variogram Models

Zone	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
J118	0.25	0.5	45	20	0	60	40	20
		0.25				100	70	60
K118	0.25	0.5	45	30	0	60	40	20
		0.25				100	70	60
L118	0.15	0.7	60	30	0	50	90	15
		0.15				250	450	40
M118	0.15	0.7	60	30	0	50	90	15
		0.15				250	450	40
N118	0.35	0.4	-30	75	-15	15	50	25
		0.25				30	200	80
O118	0.1	0.4	90	30	0	120	80	30
		0.5				300	150	60
P118	0.05	0.4	90	30	0	40	30	5
		0.55				250	120	10
Q118	0.05	0.4	0	-60	30	5	40	30
		0.55				10	250	120
R118	0.05	0.4	30	15	0	30	40	5
		0.55				120	250	10

14.3.5. Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 14-6.

Table 14-6: Specifications for the Minto South Block Model

Description	Easting (X)	Northing (Y)	Elevation (Z)
Block Model Origin (Lower left corner)	384,200	6,943,270	325
Parent Block Dimension	10	10	3
Number of Blocks	192	169	225
Rotation	0	0	0

All 1.5 m composite assays were coded by modelled mineralized domains. Blocks in a mineralized domain were estimated only from the assays within that domain. Ordinary Kriging was used to estimate copper and gold grades and inverse squared distance weighting to estimate silver grades.

14.3.6. Treatment of High Grade Composite Grades

Instead of capping the composites for high grade assays, SRK elected to limit the influence of the high grade intersections during the estimation process. Continuity of the high grade assays was studied with a technique called “p-gram”.

Figure 14—7 shows the continuity of high grade copper assays at different thresholds. High grade continuities can be indicated up to a distance where plotted curves roughly level off. For example, at 4% threshold maximum distance at which the continuity could be shown in the M Zone is roughly 40 to 80 m.

For grade estimation in all mineralized zones high grade assays were only used if they were found within search ellipsoid of 40 x 30 x 15 m size. Regardless, in addition, some very high grade composite assays were still capped at relatively high values. In both Area 2 and its Wildfire extension high grade thresholds were defined from statistical analysis, separately for each domain. The direction of the search ellipsoid was aligned with the overall direction of grade continuity in each zone.

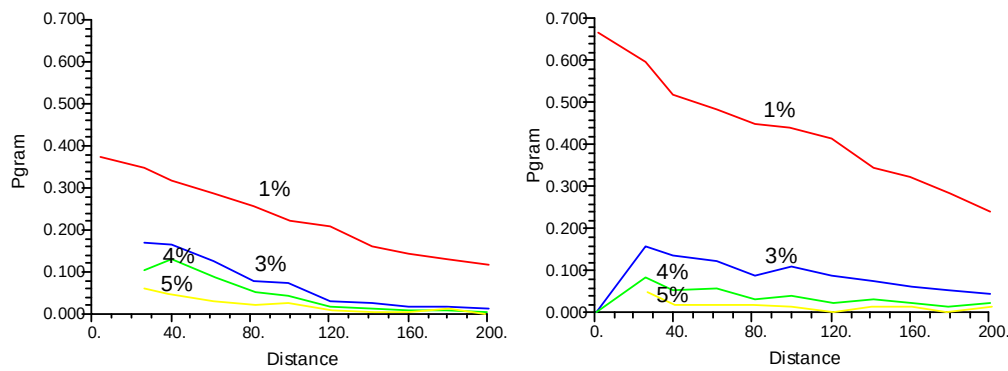


Figure 14—7: Area 2 and Wildfire regions of MSD Extension - Continuity of High Grade Copper Assays at Different Thresholds: (left) Zone L, (right) Zone M

14.3.7. Estimation Parameters

The selection of the search radii was guided by modelled ranges from variograms and was established to estimate a large portion of the blocks within the modelled area with limited extrapolation. The parameters were established by conducting repeated test resource estimates and reviewing the results as a series of plan views and sections (see Table 14-7 and Table 14-18). As mentioned in the previous section, high grade assays were only used during the estimation process if they were found within a much smaller high grade ellipsoid of 40 x 30 x 15 m size.

Table 14-7: Area 2 and Wildfire regions of MSD Update - Estimation Parameters

Parameters	J	K	L	M	N	O	P	Q
Rotated Search X (m)	90	90	100	100	75	70	90	100
Rotated Search Y (m)	50	65	70	75	100	100	90	70
Rotated Search Z (m)	30	30	30	30	30	30	30	30
Min data	4	4	4	4	4	4	4	4
Max data	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1

Table 14-8: Area 118 region of MSD - Estimation Parameters

Parameters	J118	K118	L118	M118	N118	O118	P118	Q118	R118
Rotated Search X (m)	100	90	70	70	30	100	100	30	70
Rotated Search Y (m)	70	60	100	90	100	70	70	100	100
Rotated Search Z (m)	30	30	30	30	70	30	30	70	30
Min data	4	4	4	4	4	4	4	4	4
Max data	16	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1	1

14.3.8. Specific Gravity Estimation

There is sufficient amount of specific gravity data (Figure 14—8) to warrant estimating specific gravity into the block model. For the estimation, all specific gravity (“SG”) values lower than 2.4 were adjusted to 2.4 and all very high values were capped at 3.2. Block specific gravity values were estimated by the inverse squared distance method. At least eight samples within a 200 x 200 x 50 m radius were needed to estimate a block.

All un-estimated blocks in mineralized domains were assigned average SG values within those domains.

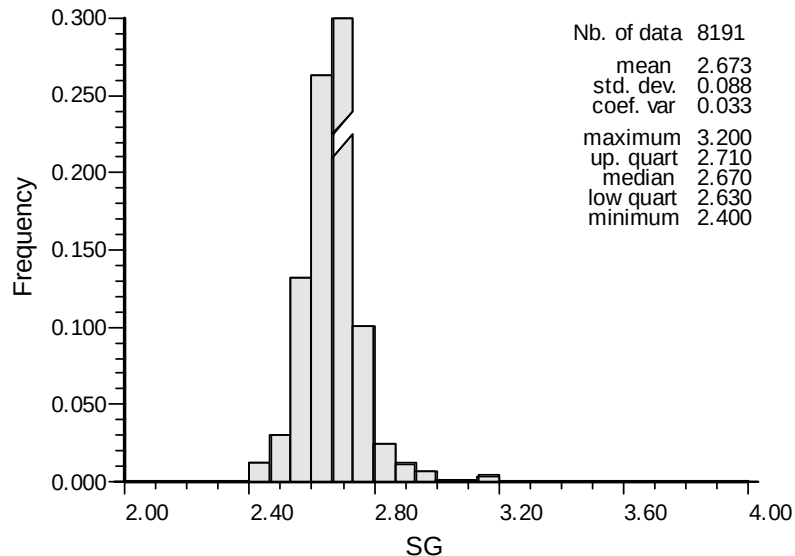


Figure 14—8: Area 2 and Wildfire regions of MSD Update – Histogram of Specific Gravity Data

14.3.9. Resource Validation

Most of the dollar value of the Minto South deposit is in copper (approx. 85%). Therefore, the validation was limited to the copper block estimates. The zones were validated by completing a series of visual inspections and by:

- Comparison of local “well-informed” block grades with composites contained within those blocks;
- Comparison of average assay grades with average block estimates along different directions – swath plots.

Figure 14—9 shows a comparison of estimated copper block grades with drill hole assay composite data contained within those blocks in the M and O zones. On average, the estimated blocks are very similar to the composite data, with high correlation between the estimates and the assays.

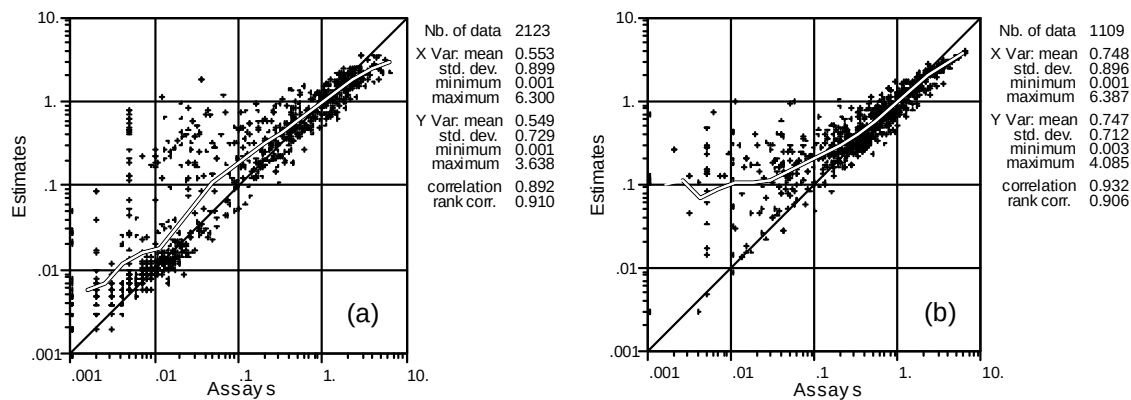


Figure 14—9: Area 2 and Wildfire regions of MSD Update - Comparison of Copper Block Estimates with Composite Assay Data Contained Within the Blocks in (a) M zone (b) O zone

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south and horizontal swath.

Figure 14—10 and Figure 14—11 show the copper swath plots from the M and the O zones. Here, and similarly in other zones, the average copper composite grades and the average copper estimated block grades are quite similar in all directions. Overall, the validation shows that current resource estimates are excellent reflection of drill hole assay data.

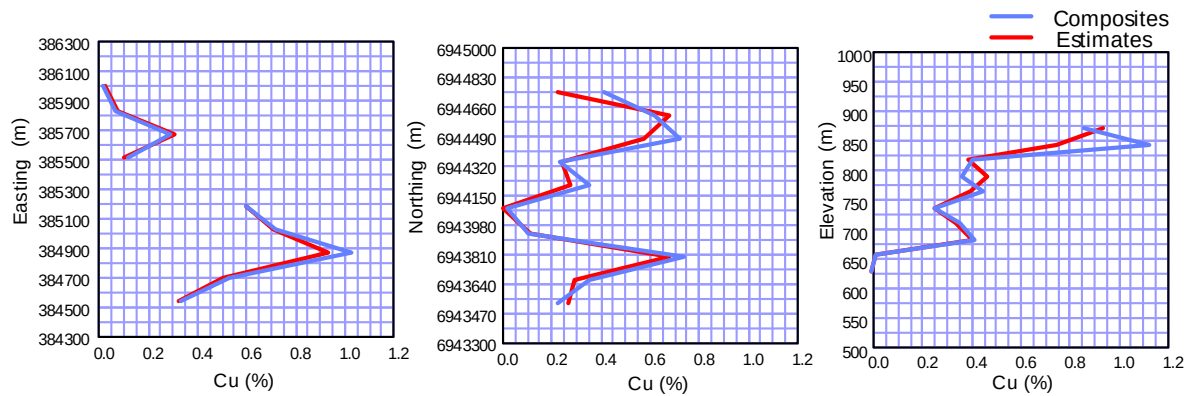


Figure 14—10: Area 2 and Wildfire regions of MSD Update - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the M zone

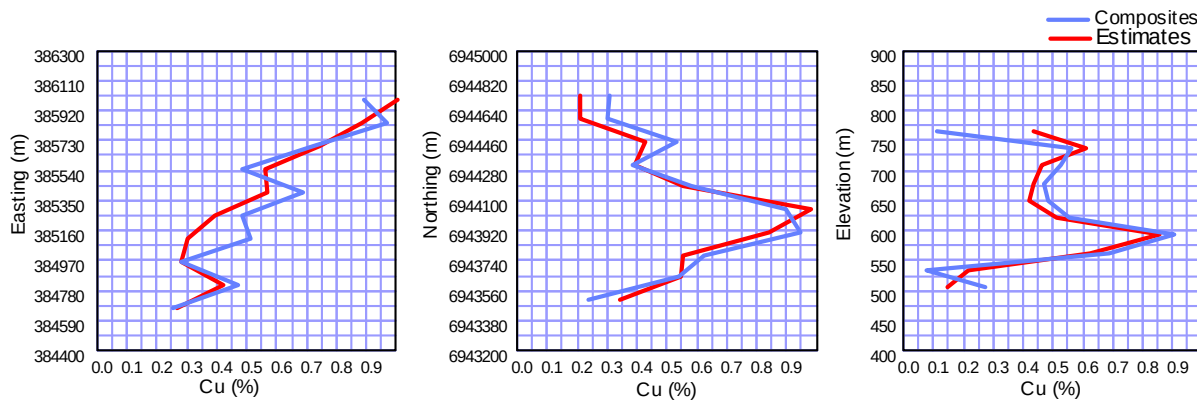


Figure 14—11: Area 2 and Wildfire regions of MSD Update - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the O zone

14.3.10. Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Minto South deposit was classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an “independent competent person” as defined by National Instrument 43-101.

Drill hole spacing in Area 2 and Wildfire Extension is sufficient for geostatistical analysis and evaluating spatial grade variability. SRK is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in both deposits.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block; and
- Average distance to the composites used to estimate a block.

In order to classify mineralization as Measured Mineral Resource, “quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters”. To satisfy this requirement, the following procedure was used to classify blocks as Measured:

- Blocks were flagged as measured if informed from more than 12 composites from more than three separate drill holes and an average distance to the data used to estimate the grade was less than 35 m

In order to classify mineralization as an Indicated Mineral Resource, “the nature, quality, quantity and distribution of data” must be “such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization.” (CIM Definition Standards on Mineral Resources and Mineral Reserves, December 2005) To satisfy this requirement, the following procedure was used to classify blocks as Indicated:

- Blocks were flagged as indicated if informed from more than 8 composites from three or more separate drill holes and if an average distance to the data used to estimate the grade was more than 35 m and less than 60 m
- Final broad areas of measured and indicated resources were designed from classification envelopes encompassing blocks flagged for the measured and indicated categories. This approach ensured consistent definition of the areas assigned to measured and indicated categories, thereby removing small, discontinuous clusters of blocks assigned to those categories. All estimated block grades not assigned to either measured or indicated category were given an inferred resource category.

14.3.11. Sensitivity of the Block Model to Selection Cut-off Grade

The complete revised Minto South block model comprises two components: (1) the 2010 mineral resource for Area 118; and (2) the updated 2011 Area 2 and Wildfire resource. The two components were merged to form a single Minto South resource block model. The mineral resources are sensitive to the selection of cut-off grade.

Table 14-9 shows global quantities and grade in the Minto South deposit at different Cu cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Grade tonnage curves for different resource categories are presented in Figure 14—12 and Figure 14—13.

Table 14-9: Minto South - Sensitivity Analysis of Global Tonnage and Grades Deposit at Various Copper Cut-off Grades

Classification	Cut-Off (Cu%)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs.)*	Contained Gold (000's oz)*	Contained Silver (000's oz)*
Measured (M)	>2.0	1,363	2.55	1.13	9.30	76,639	50	408
	>1.5	2,836	2.12	0.89	7.53	132,668	81	687
	>1.0	5,198	1.72	0.68	5.93	196,891	114	991
	>0.5	10,387	1.21	0.45	4.07	278,028	151	1360
	>0.4	12,517	1.08	0.40	3.60	299,024	159	1450
	>0.3	15,562	0.94	0.33	3.10	322,329	167	1550
	>0.2	19,860	0.79	0.28	2.58	345,700	176	1649
	>0.1	27,410	0.61	0.21	1.98	369,638	183	1749
Indicated (I)	>2.0	1,699	2.55	0.93	9.86	95,645	51	539
	>1.5	3,973	2.08	0.74	7.85	181,856	95	1003
	>1.0	9,215	1.58	0.55	5.81	321,833	163	1723
	>0.5	28,556	0.98	0.32	3.44	616,152	290	3162
	>0.4	38,223	0.84	0.27	2.95	711,454	326	3625
	>0.3	52,740	0.71	0.22	2.47	822,695	365	4182
	>0.2	72,825	0.58	0.17	2.03	931,449	400	4748
	>0.1	110,292	0.43	0.12	1.53	1,050,475	439	5420
Total (M+I)**	>2.0	3,062	2.55	1.02	9.61	172,284	100	947
	>1.5	6,809	2.10	0.80	7.72	314,524	176	1,690
	>1.0	14,413	1.63	0.60	5.86	518,724	277	2,714
	>0.5	38,943	1.04	0.35	3.61	894,180	441	4,522
	>0.4	50,740	0.90	0.30	3.11	1,010,478	485	5,075
	>0.3	68,302	0.76	0.25	2.61	1,145,024	532	5,732
	>0.2	92,685	0.62	0.19	2.15	1,277,149	576	6,397
	>0.1	137,702	0.47	0.14	1.62	1,420,113	622	7,169
Additional Inferred	>2.0	193	2.35	0.83	10.68	9,986	5	66
	>1.5	368	2.04	0.75	8.90	16,519	9	105
	>1.0	1,415	1.46	0.55	6.31	45,581	25	287
	>0.5	8,134	0.81	0.24	2.93	145,718	63	767

Classification	Cut-Off (Cu%)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs.)*	Contained Gold (000's oz)*	Contained Silver (000's oz)*
	>0.4	12,577	0.68	0.19	2.40	189,501	76	970
	>0.3	18,715	0.57	0.15	1.97	236,244	89	1186
	>0.2	27,810	0.47	0.12	1.61	285,449	103	1443
	>0.1	45,166	0.34	0.08	1.26	341,005	121	1831

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

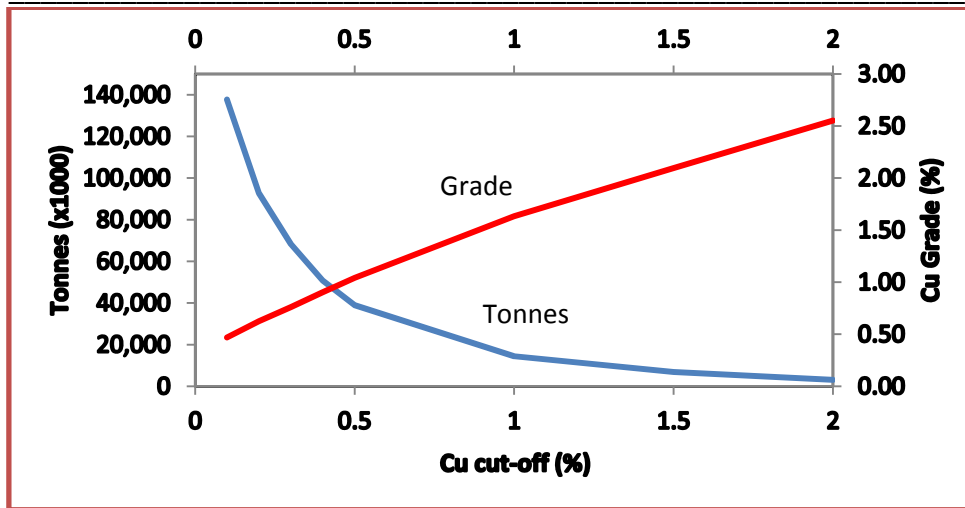


Figure 14—12: Minto South - Copper Grade Tonnage Curve for Measured and Indicated Blocks

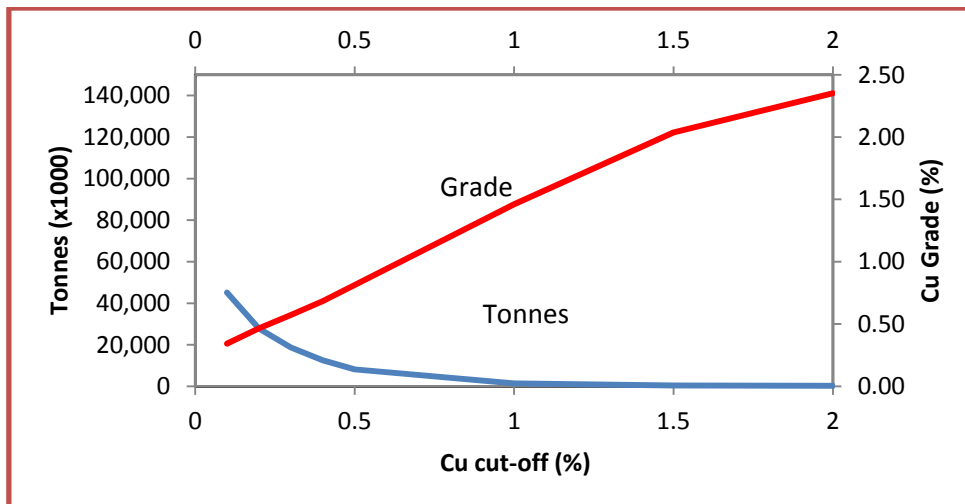


Figure 14—13: Minto South - Copper Grade Tonnage Curve for Inferred Blocks

14.3.12. Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

"[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries.

In order to meet this requirement, SRK considers that the Minto South resource is amenable for open pit extraction.

The open pit mineral resources are reported at a cut-off value of 0.5% Cu per tonne.

The Minto South resource extent was compared against an optimized Whittle™ shell based on the NSR model and parameters in Table 14-10. Nearly the entire defined resource fell within the Whittle™ shell, except for small, local extensions of individual zones at depth. SRK understand that Minto Mine is already an operating mine extracting the parts of the Minto South resource, and had positive economic evaluations of underground mining scenarios. The final Minto South resource is therefore not constrained by the Whittle™ shell but is considered to have reasonable prospects of economic extraction in its entirety.

Table 14-10: Whittle™ Optimization Parameters for MSD Resource Estimate Comparison in Order to Demonstrate Reasonable Prospects of Economic Extraction

Constraining Parameter	Unit	Value
Copper Price	US\$/lb	3.5
	C\$/lb	3.85
Gold Price	US\$/oz	1300
	C\$/oz	1430
Silver Price	US\$/oz	16
	C\$/oz	17.6
Exchange Rate	C\$: US\$	1.11
Mining Cost	C\$/t mined	1.50
Processing and G&A Cost	C\$/t milled	5.00
Royalty	%	0.5
Slope angles	degrees (overall)	50

Table 14-11 presents the mineral resource statement for the Minto South resource. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14-11: Mineral Resource Statement at 0.5% Cu Cut-off for the Minto South, SRK Consulting September 29, 2011

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	10,387	1.21	0.45	4.07	278,028	151	1,360
Indicated (I)	28,556	0.98	0.32	3.44	616,152	290	3,162

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Sub-total (M+I)**	38,944	1.04	0.35	3.61	894,180	441	4,522
Inferred	8,134	0.81	0.24	2.93	145,718	63	767

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

Note that mining commenced on the Minto South deposit during 2011. The mined material was moved to a stockpile and has not yet been processed. The resource shown in

Table 14-11 is therefore still considered part of the Minto South resource inventory.

14.4. Ridgetop Deposit

14.4.1. Geology Model

The Ridgetop deposit consists of seven mineralized foliated granodiorite zones. As in the case of Minto South deposit, these zones are generally shallow dipping, on average 24° towards the northeast. The same geometrical characteristics are evident for this deposit as for Minto South, and the same geological understanding applies. However, the zones have undergone gentle folding along N-S trending axes. At least one synformal and one antiformal axis can be identified from the wireframe interpolation. In addition, the zones in this deposit get progressively steeper towards the north, apparently reaching a dip of 70° within 15 m from the northeastern boundary limit of the modeled deposit. It is believed that the northeastern boundary is controlled by a northwest striking fault, and the ore zones are dragged downwards towards this fault zone. The exact position, orientation and properties of the fault zone have not been identified yet.

The Ridgetop resource model was created using a commercial three-dimensional block modelling and mine planning software, GEMS version 6.2 (Gemcom®). The model was created in metric units using the mine's local co-ordinate system (UTM NAD83 zone 8). The solids were considered hard boundaries where grades were not allowed into blocks outside of these solids.

The seven mineralized horizons were assigned the following domain codes based on those codes historically used by mine geologists; 80, 90, 100, 110, 120, 140 and 160. The process of identifying and naming the zones was done by importing and reviewing the sectional interpretation provided by MintoEx geologists. Minor modifications to contacts and zone orientations allowed simplification and enhanced continuity of the zones in places. There are therefore fewer interpreted and modeled zones than identified during the exploration program. Table 14-12 includes a list of the domain coding assigned to the drill data and the block model. Figure 14—14 is a three dimensional view of the zone solids, showing their domain codes.

Table 14-12: Ridgetop Modeled Domain Names and Block Model Codes

Domain Name	Block Model Code	Comments
R80	82	Chalcocite partial oxidation
R90	93	Chalcocite partial oxidation

Domain Name	Block Model Code	Comments
R100	103	Chalcocite dominant zone. Primary ore-bearing zone.
R110	110	Chalcopyrite dominant zone. Primary ore-bearing zone.
R120	120	
R140	140	
R160	160	
Waste	200	Non-mineralized granodiorite
Conglomerate (Cng)	300	Cretaceous aged erosion surface, removing ore zones
Overburden (OB)	500	Unconsolidated waste material
Air	0	

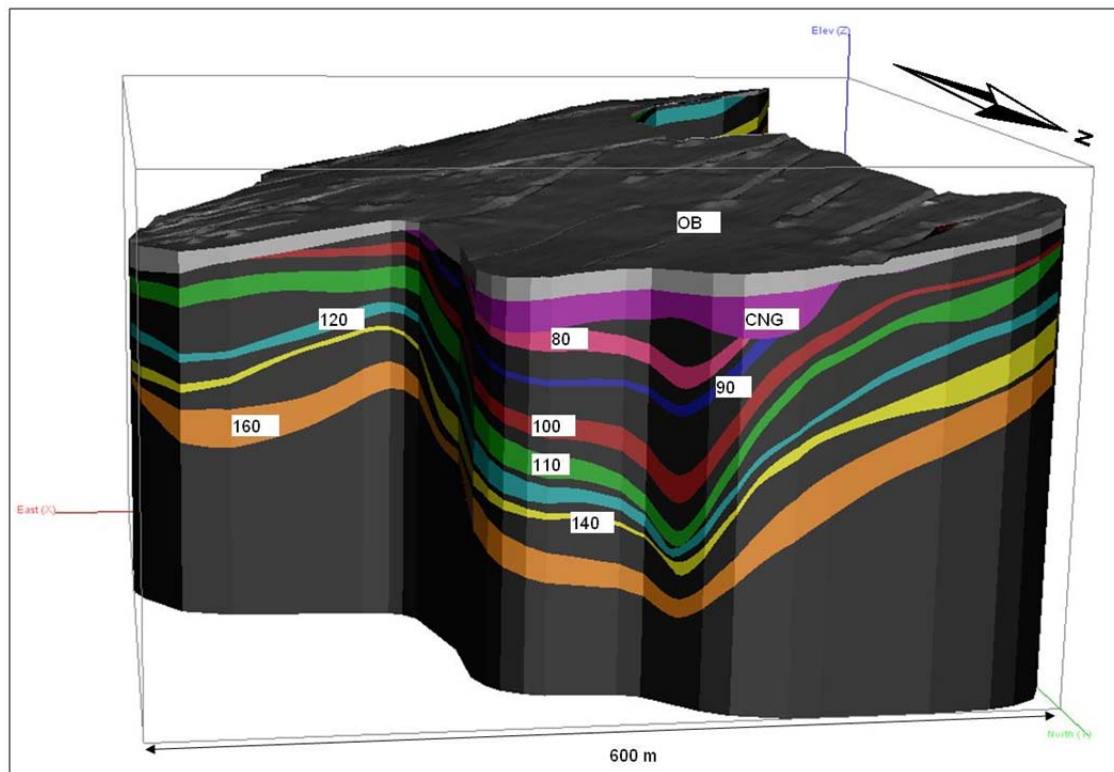


Figure 14—14: View South of the Modelled Ridgetop Mineralized and Waste Domains

The solids were then used to assign the domain and block model codes to the drill hole data (assays and composites) and the block model cells. Blocks above the topography surface were tagged as Air and the blocks outside of the zone solids were tagged as Waste. A Cretaceous conglomerate is developed on the northeastern side of the deposit. It gets rapidly thicker towards the northeast and is presumably strongly influenced by the bounding fault zone. A small amount of conglomerate was included in the model. There is also unconsolidated material near surface, which is included in the model as Overburden.

SRK reviewed the tagged assay, composite and block data on sections and in 3D, and in exported text files using external customized software to ensure the process had worked properly.

To assess how well the modeled solids differentiate between lower and higher grade mineralization, grades on either sides of the modeled contacts were queried and listed. Any anomalous assay values were checked in 3D to determine whether the problems are errors or not. There were far fewer of such anomalous assay values than for Area 2, primarily because the holes are more recent and logged to consistent standards.

14.4.2. Data

A total of 6,193 grade measurements have been used in the design of mineralized domains from holes drilled roughly at 20 m spacing in the North-West and 40 m spacing in the South-East portions of the deposit. Approximately 27% of the samples within the modelled domains were collected from 1.5 m intervals (Figure 14—15). Similarly to the Minto South deposit, the assays were composited to 1.5 m lengths. Shorter composite lengths ensured that most relevant, undiluted assays were included in the resource assessment. Within the mineralized domains 5,833 composite assays were produced from 163 holes. The average thickness of the high grade mineralized horizons is 12 m and in lower grade mineralized horizons is 10 m.

Statistics of polygonally declustered 1.5 m Cu composites within each mineralized zone are presented in Figure 14—16.

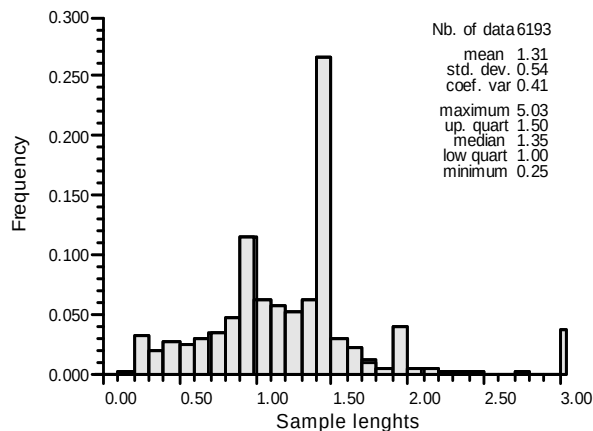


Figure 14—15: Ridgtop - Histogram of Sample Lengths

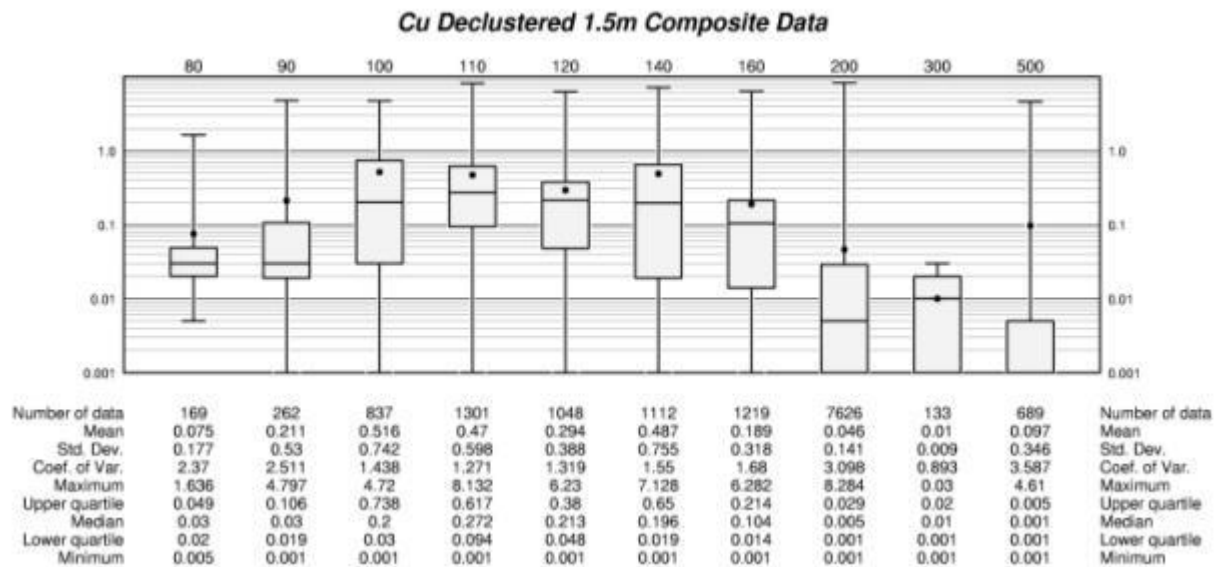


Figure 14—16: Ridgetop - Basic Statistics of Declustered Cu Composite Grades

Figure 14—17 shows bivariate statistics of the Cu and Au assays in two higher grade domains. Note moderate degree of correlation in the 100 domain and high degree of rank correlation in the 110 domain, indicated by a regression curve (white thick line) showing a general tendency of increased Cu assays for higher Au assays. This positive correlation lead to a design of variogram models along identical major directions of continuity for both Cu and Au grades.

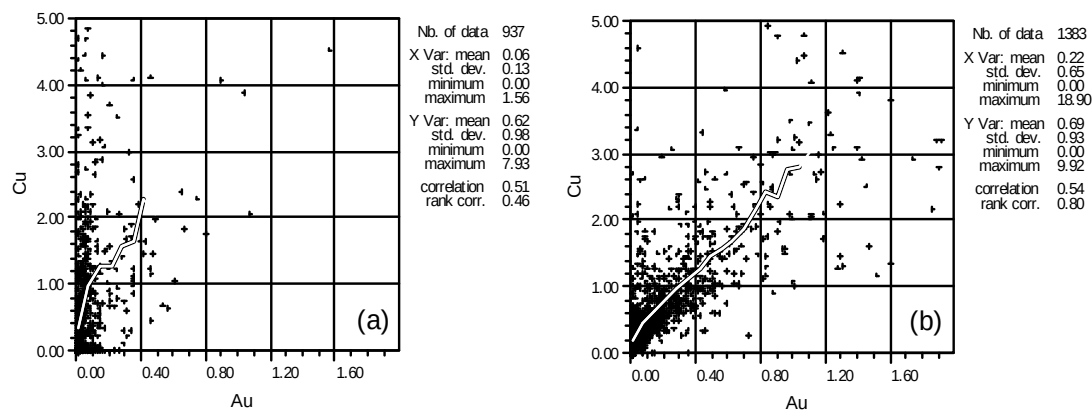


Figure 14—17: Ridgetop - Bivariate Statistics of Cu and Au Assays in (a) 100 domain, (b) 110 domain

14.4.3. Evaluation of Extreme Assay Values

Block grade estimates may be unduly affected by very high grade assays. Therefore, the assay data were evaluated for the high grades outliers. An analysis of the high grade assays indicates relatively strong negative correlation between the assay data and the sample lengths (see Figure 14—18). This suggests that sampling was based on visual indications of mineralization. In view of the above, as in Minto South, no capping was done before assay compositing to 1.5 m lengths.

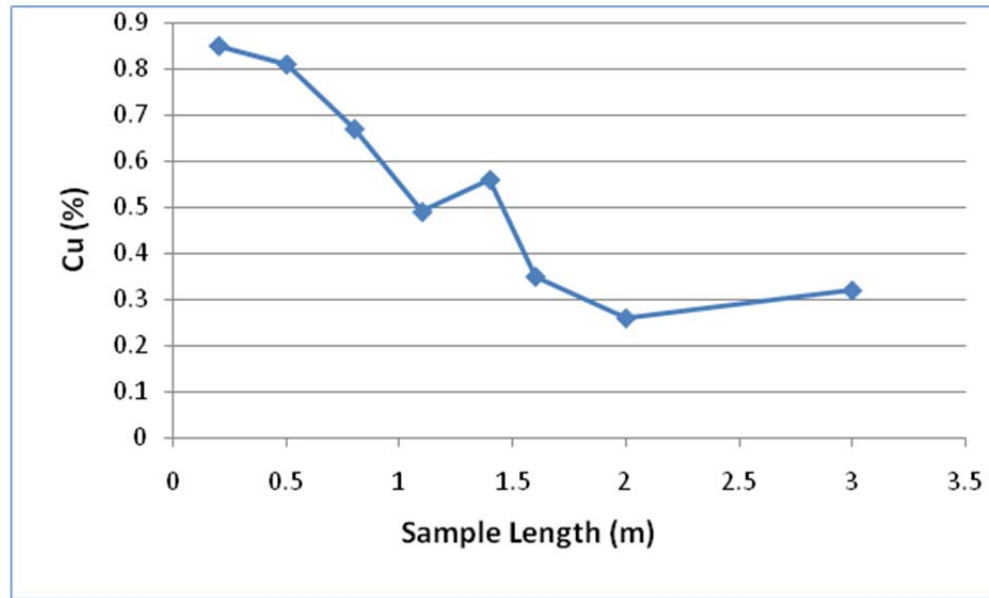


Figure 14—18: Ridgetop - Grade Variation with the Sample Length

14.4.4. Variogram Analysis

Experimental variograms and variogram models in the form of correlograms were generated for Cu and Au grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from 5 to 25 percent of the total sill. Cu variogram models used for grade estimation are summarised in Table 14-13. Note that no variogram models were designed for Ag grades. The Ag was estimated by the inversed distance squared method.

Table 14-13: Ridgetop Cu Exponential Variogram Models

Domain	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
80*	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
90*	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
100	0.05	0.75	50	24	-48	25	40	15
		0.20				300	80	20
110	0.05	0.60	50	24	-45	30	35	16
		0.35				180	500	35
120	0.20	0.50	50	24	-48	50	20	10
		0.30				200	80	12
140	0.05	0.75	50	24	-45	55	65	15

Domain	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
		0.20				85	550	20
160	0.10	0.55	50	24	-48	15	75	15
		0.35				220	150	40

* Variogram models assigned from Domain 100

Exponential variogram models have been used with practical ranges of continuity

14.4.5. Resource Estimation Methodology

The geometrical parameters of the block model are summarised in Table 14-14.

Table 14-14: Specifications for the Ridgetop Block Model

Description	Easting (X m)	Northing (Y m)	Elevation (Z m)
Block Model Origin (Lower left corner)	384,650	6,943,200	595
Parent Block Dimension	10	10	3
Number of Blocks	90	90	135
Rotation	0	0	0

All 1.5 m composite assays were coded by modelled mineralized domains. Blocks in a mineralized domain were estimated only from the assays within that domain. Ordinary kriging was used to estimate Cu and Au grades and Inverse Squared Distance weighting to estimate Ag grades.

14.4.6. Treatment of High Grade Composite Grades

As in Minto South, instead of capping the composites for high grade assays, SRK elected to limit the influence of the high grade intersections during the estimation process. Figure 14—19 shows the continuity of high grade assays at different thresholds. High grade continuities can be indicated up to a distance where plotted curves roughly level off. For example, at 4% threshold maximum distance at which the continuity could be shown is roughly 40 m.

For grade estimation in all mineralized zones high grade assays were only used if they were found within search ellipsoid of 40 x 30 x 15 m size. High grade thresholds were defined from statistical analysis, separately for each domain. The direction of the high grade search ellipsoid was aligned with the overall direction of grade continuity in each zone.

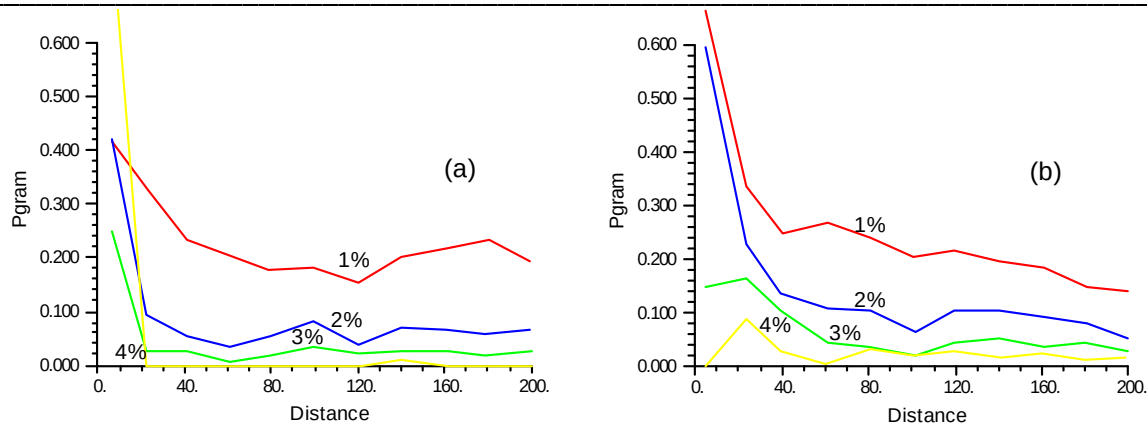


Figure 14—19: Ridgetop – Continuity of High Grade Assays at Different Thresholds: (left) in Zone R100, (right) Zone R140

14.4.7. Estimation Parameters

The selection of the search radii was guided by modelled ranges from variograms and was established to estimate a large portion of the blocks within the modelled area with limited extrapolation. The parameters were established by conducting repeated test resource estimates and reviewing the results as a series of plan views and sections (see Table 14-15).

Table 14-15: Ridgetop Estimation Parameters

Parameters	80	90	100	110	120	140	140	160
						Step 1	Step 2	
Rotated X (m)	60	60	70	35	70	35	50	60
Rotated Y (m)	35	35	40	60	40	60	80	60
Rotated Z (m)	20	20	25	20	25	20	30	20
Min data	4	4	3	4	3	3	3	4
Max data	16	16	16	16	16	16	16	16
Max number of samples per dh	4	4	4	4	4	4	4	4
Minimum number of octants	1	1	1	1	1	1	1	1
Minimum number of holes	1	1	1	1	1	1	1	1

14.4.8. Specific Gravity Estimation

There is sufficient variation in specific gravity data (Figure 14—20) to warrant estimating specific gravity into the block model. For the estimation, 11 very high SG values were capped. Block specific gravity values were estimated by the ID2 method. At least eight samples within a 120 x 120 x 40 m radius were needed to estimate a block.

All un-estimated blocks in mineralized domains were assigned average SG values within those domains.

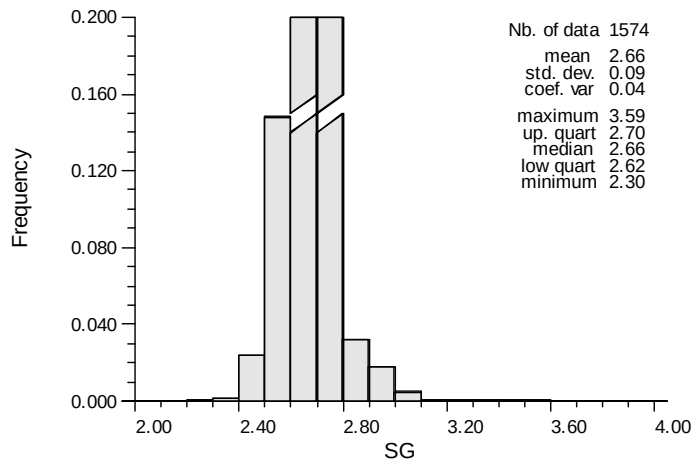


Figure 14—20: Ridgetop Update – Histogram of Specific Gravity Data

14.4.9. Resource Validation

Most of the dollar value in the Ridgetop deposit is in copper (approx. 90%). Therefore, the validation was limited to the Cu block estimates. The deposits were validated by completing a series of visual inspections and by:

- Comparison of local “well-informed” block grades with composites contained within those blocks;
- Comparison of average assay grades with average block estimates along different directions – swath plots.

Figure 14—21 shows a comparison of estimated Cu block grades with drill hole assay composite data contained within those blocks. On average, the estimated blocks are similar to the composite data, with good correlation between the estimates and the assays.

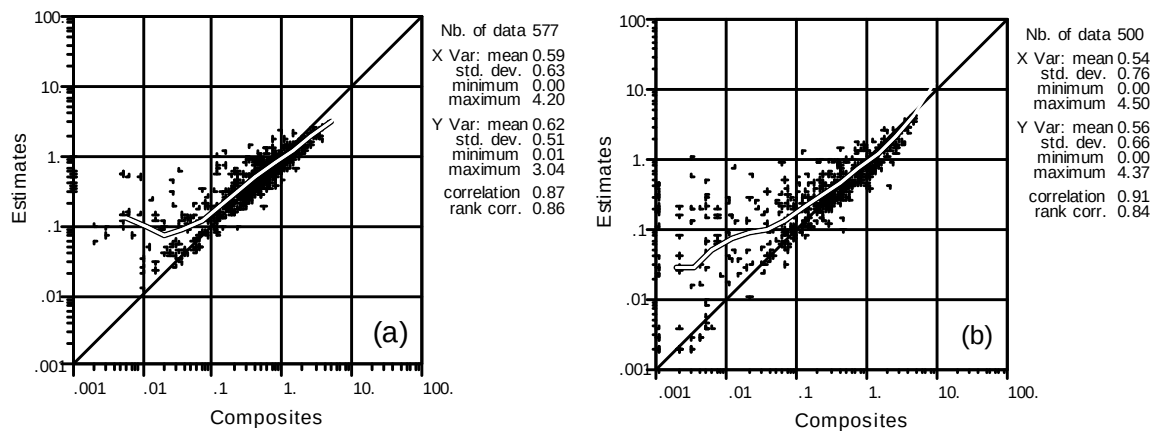


Figure 14—21: Ridgetop - Comparison of Cu Block Estimates with Composite Assay Data Contained within the Blocks: (a) 110 domain, (b) 140 domain

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south and horizontal swaths

Figure 14—22 shows the swath plots from the 140 zone. Here, and similarly in other zones, the average Cu composite grades and the average Cu estimated block grades are quite similar in all directions. Overall, the validation shows that current resource estimates are very good reflection of drill hole assay data.

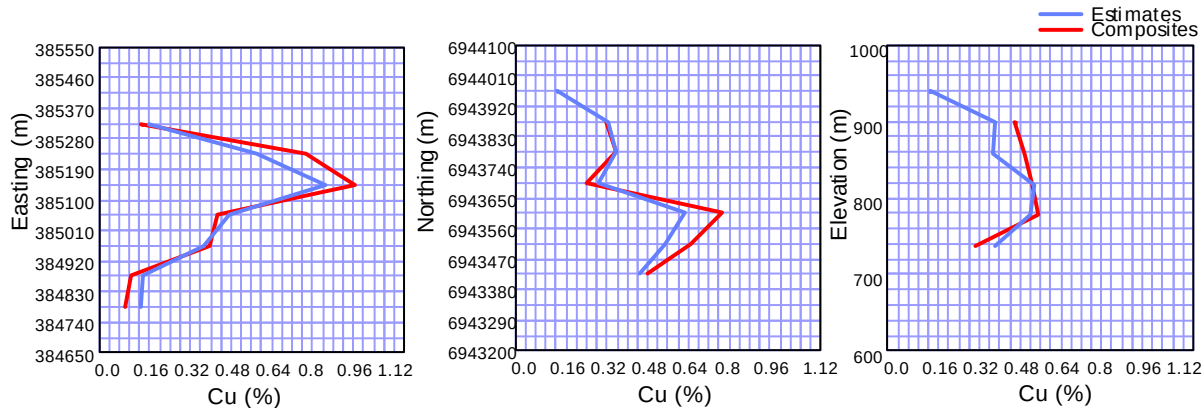


Figure 14—22: Ridgetop - Declustered Average Cu Composite Grades Compared to Cu Block Estimates in the 140 domain

14.4.10. Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Ridgetop deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Dr. Wayne Barnett, Ph.D., Pr.Sci.Nat., an “independent competent person” as defined by National Instrument 43-101.

Drill hole spacing at Ridgetop is sufficient for geostatistical analysis and evaluating spatial grade variability. SRK is therefore of the opinion that the amount of sample data is adequate to demonstrate good confidence of the grade estimates in the deposit.

- The estimated blocks were classified according to:
- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block; and

-
- Average distance to the composites used to estimate a block.

In order to classify mineralization as Measured Mineral Resource, “quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters”. To satisfy this requirement, the following procedure was used to classify blocks as Measured:

- Blocks were flagged as measured if informed from more than 12 composites from more than three separate drill holes and an average distance to the data used to estimate the grade was less than 35 m

In order to classify mineralization as an Indicated Mineral Resource, “the nature, quality, quantity and distribution of data” must be “such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization.” (CIM Definition Standards on Mineral Resources and Mineral Reserves, December 2005) To satisfy this requirement, the following procedure was used to classify blocks as Indicated:

- Blocks were flagged as indicated if informed from more than 8 composites from three or more separate drill holes and if an average distance to the data used to estimate the grade was more than 35 m and less than 60 m

Final broad areas of measured and indicated resources were designed from classification envelopes encompassing blocks flagged for the measured and indicated categories. This approach ensured consistent definition of the areas assigned to measured and indicated categories, thereby removing small, discontinuous clusters of blocks assigned to those categories. All estimated block grades not assigned to either measured or indicated category were given an inferred resource category.

14.4.11. Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 14-16 shows global quantities and grade in the Ridgetop deposit at different Cu cut-off grades. Resource tabulation is limited to a Whittle™ shell with slope angles of 50 degrees using 10 x 10 x 3 m block model. The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Grade tonnage curves for different resource categories are presented in Figure 14—23 and Figure 14—24.

Table 14-16: Ridgetop Sensitivity Analysis of Global Tonnage and Grades in the Ridgetop Deposit at Various Cu Cut-off Grades

Classification	Cut-Off (Cu %)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	>2.0	55	2.36	0.69	4.40	2,874	1.2	8
	>1.5	187	1.90	0.54	3.66	7,816	3.3	22
	>1.0	555	1.45	0.39	2.91	17,731	7.0	52
	>0.5	1,531	0.98	0.25	2.14	33,204	12.3	105
	>0.4	1,821	0.90	0.22	1.97	36,083	13.1	116
	>0.3	2,150	0.81	0.20	1.80	38,607	13.7	124
	>0.2	2,472	0.74	0.18	1.66	40,381	14.2	132
	>0.1	2,836	0.67	0.16	1.51	41,588	14.6	138
Indicated (I)	>2.0	114	2.53	1.66	12.35	6,380	6.1	45
	>1.5	324	1.99	1.07	8.41	14,235	11.2	88
	>1.0	841	1.51	0.68	5.70	28,055	18.5	154
	>0.5	3,534	0.87	0.30	2.87	67,901	33.8	326
	>0.4	4,965	0.75	0.24	2.41	82,039	38.4	384
	>0.3	6,905	0.64	0.19	2.00	96,892	42.7	445
	>0.2	9,896	0.52	0.15	1.62	113,142	46.9	514
	>0.1	13,015	0.43	0.12	1.34	123,595	49.9	562
Sub-total (M+I)**	>2.0	170	2.48	1.35	9.76	9,254	7.3	53
	>1.5	511	1.96	0.88	6.67	22,051	14.4	110
	>1.0	1,396	1.49	0.57	4.59	45,786	25.5	206
	>0.5	5,064	0.91	0.28	2.65	101,104	46.2	431
	>0.4	6,786	0.79	0.24	2.29	118,122	51.6	500
	>0.3	9,054	0.68	0.19	1.96	135,499	56.4	569
	>0.2	12,369	0.56	0.15	1.62	153,523	61.0	646
	>0.1	15,851	0.47	0.13	1.37	165,183	64.5	700
Inferred	>2.0	0	0.00	0.00	0.00	0	0.0	0
	>1.5	4	1.60	0.17	3.59	130	0.0	0
	>1.0	50	1.22	0.07	1.00	1,338	0.1	2
	>0.5	318	0.75	0.13	1.57	5,250	1.3	16

Classification	Cut-Off (Cu %)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
	>0.4	446	0.66	0.12	1.45	6,523	1.7	21
	>0.3	597	0.58	0.10	1.29	7,695	2.0	25
	>0.2	798	0.50	0.09	1.12	8,769	2.2	29
	>0.1	1,057	0.42	0.07	0.95	9,667	2.4	32

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

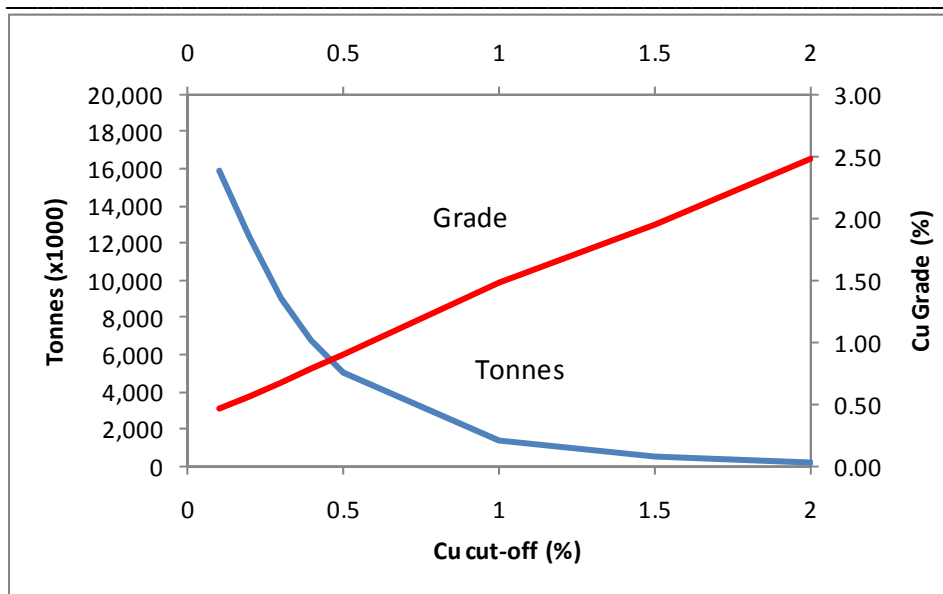


Figure 14—23: Ridgetop - Cu Grade Tonnage Curve for Measured and Indicated Blocks

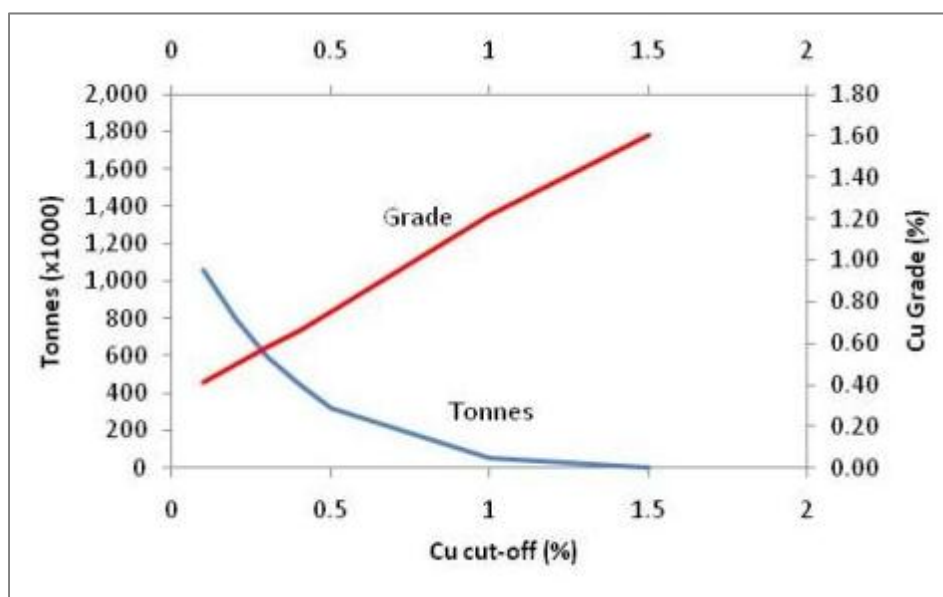


Figure 14—24: Ridgetop - Cu Grade Tonnage Curve for Inferred Blocks

14.4.12. Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

“[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”

The “reasonable prospects for economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries.

In order to meet this requirement, SRK considers that the Ridgetop deposit is amenable for open pit extraction.

The open pit mineral resources are reported at a cut-off value of 0.5% Cu per tonne, based on a combined processing and G&A cost of C\$18.00 per tonne of material processed, mining cost of C\$1.8 per tonne and metal prices of US\$3.00 per pound for copper. The open pit resource is constrained by an optimized Whittle™ shell based on the NSR model, overall slope angles of 45 degrees and the site operating costs listed above.

Table 14-17 presents the mineral resource statement for the Ridgetop deposits.

Table 14-17: Mineral Resource Statement at 0.5% Cu Cut-off for the Ridgetop Deposit, SRK Consulting August 30, 2010

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz)*	Contained Ag (000's oz)*
Measured (M)	1,531	0.98	0.25	2.14	33,204	12.3	105
Indicated (I)	3,534	0.87	0.30	2.87	67,901	33.8	326
Sub-total (M+I)**	5,064	0.91	0.28	2.65	101,104	46.2	431
Inferred	318	0.75	0.13	1.57	5,250	1.3	16

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

14.5. Minto North Deposit

The Minto North deposit is a new discovery made in early 2009 and comprises near surface, higher grade copper-gold mineralization. In June 2009, the first mineral resource estimate for the Minto North deposit, using a 0.5% copper cut-off, was estimated (Table 14-18) and presented in the Capstone Press Release dated June 9, 2009. The June resource was based on 31 drill holes. Solids were created based on mineralized intersections and used to constrain the interpolation of grades.

Subsequently, additional 56 drill holes were drilled from June through September 2009 as part of an in-fill and delineation program. The goal of this program was to better define the ore boundaries and constraining solids and upgrade indicated and inferred resources to measured and indicated. The resultant resource estimate is detailed and reported in the following sections.

Table 14-18: Tonnage & Grade Estimates of the Minto North Deposit Reported in June 2009

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Cu (000's lbs.)*	Contained Gold (000's oz.)*	Contained Silver (000's oz.)*
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	1,237	2.49	1.86	9.7	67,853	74	385
Sub-total (M+I)**	1,237	2.49	1.86	9.7	67,853	74	385
Additional Inferred	634	1.88	1.03	6.4	26,318	21	130

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

14.5.1. Geology Model

A solid model of the 115, 120 and 130 ore zones within the Minto North Deposit was created from sections and based on a combination of lithology, copper grades and site knowledge (see Figure 14—25). It is important to note that the 2009 drilling resulted in new insights into the mineralization and grade distribution which greatly assisted in the creation of the solids. The ore zone solids were used for constraining the interpolation procedure. In addition, a large cross-cutting dyke that transects the deposit and the zones was also modelled using sectional interpretations and subsequently utilized to mask out the estimated tonnage related to this barren unit.

Every intersection was inspected and the solids were then manually adjusted to match exactly the interval intercepts. Once the solids models were created, they were used to code the drill hole assays and composites for subsequent geostatistical analysis. For the purpose of the resource model, the solid zone was utilized to constrain the block model by matching assays to those within the zones in a process called geologic matching so that only composites that lie within a particular zone are used to only interpolate the blocks within that zone. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from strike and dip of the mineralized zone, site knowledge and on-site observations by MintoEx's geological staff.

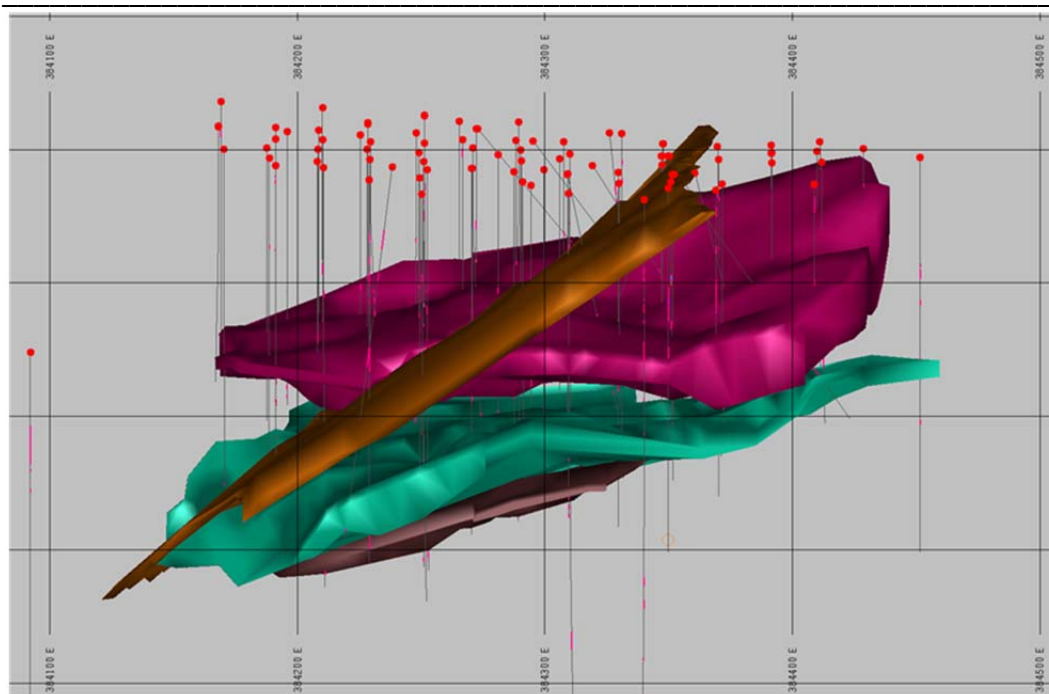


Figure 14—25: View from the North of the Modelled Minto North Mineralized Domains

14.5.2. Data

The drill hole database was supplied in electronic format by MintoEx. This included collars, down hole surveys, lithology data and assay data (i.e. Au g/t, Cu%, Ag g/t, SG with down hole from and to intervals in metric units. The database was numerically coded by mineralized zone solid; 115 Zone Ore = 115, 120 Zone Ore = 120, 130 Zone Ore = 130 and Waste = 8. The database was then manually adjusted drill hole by drill hole to insure accuracy of zonal intercepts.

Table 14-19 and Figure 14—26 show statistics of copper assays weighted by assay intervals. The highest by far average Cu, Au, and Ag grades are found in zone 115 (2.12%, 1.15 g/t, 7.62 g/t respectively). Note that the overall average grades from all three mineralized domains are higher than in Minto South and at Ridgetop deposits.

Table 14-19: Minto North – Statistics for Copper Assays Weighted by Assay Interval

CU	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
115	1,637.0	0.00	39.60	2.12	0.69	1.37	2.56	2.79	1.32
120	651.8	0.00	13.85	0.33	0.06	0.14	0.34	0.86	2.62
130	124.6	0.00	2.07	0.26	0.02	0.14	0.30	0.37	1.43
Total	2,413.4	0.00	39.60	1.54	0.22	0.89	1.92	2.49	1.62
All	4,943.5	0.00	39.60	0.77	0.02	0.02	0.81	1.90	2.48

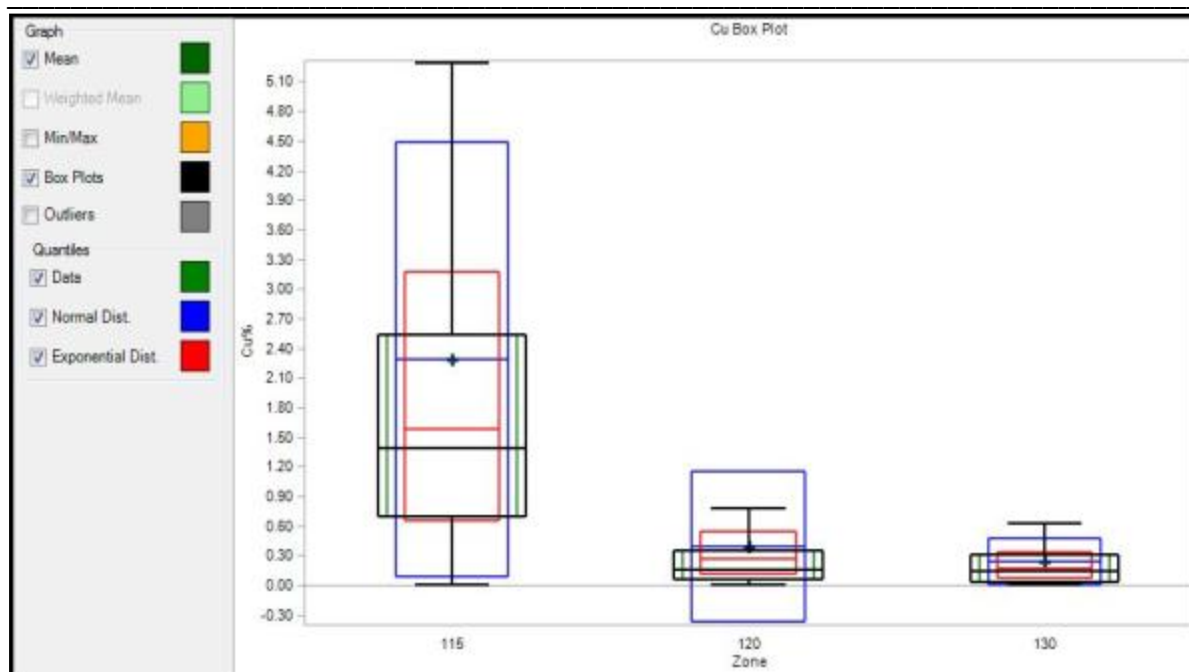


Figure 14—26: Minto North - Basic statistics of Cu assay grades in the mineralized zones

14.5.3. Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the undue influence of very high grades. Table 14-20 and Figure 14—27 show the basic statistics for the 1.5 m Cu composite grades within the mineralized domains.

Table 14-20: Minto North - Composite Statistics Weighted by Length

CU	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	COV
115	1,637.0	0.00	27.41	2.12	0.84	1.39	2.40	2.40	1.13
120	651.8	0.00	7.83	0.33	0.07	0.15	0.32	0.69	2.12
130	124.6	0.00	1.56	0.26	0.07	0.15	0.32	0.32	1.26
Total	2,413.4	0.00	27.41	1.54	0.26	0.92	1.85	2.18	1.41
All	4,943.5	0.00	27.41	0.77	0.01	0.04	0.84	1.70	2.22

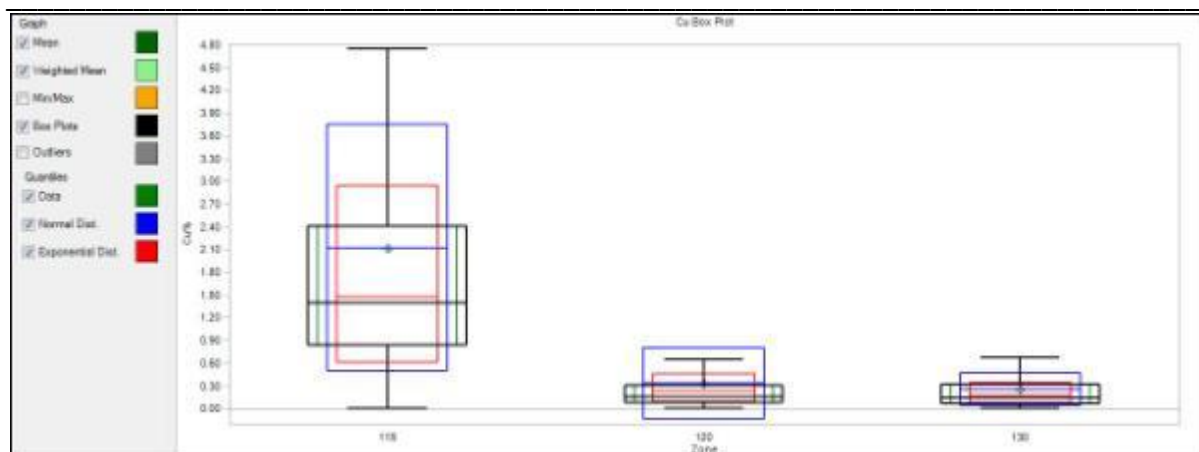


Figure 14—27: Minto North – Basic Statistics of Cu Composite grades in the mineralized zones

14.5.4. Evaluation of Extreme Assay Values

During the estimation process in Zone 115 influence of assays higher than 11% Cu, 50 g/t Ag, and 5 g/t Au has been quite limited. Similarly, in Zone 120 the same restriction was applied to assays higher than 1.2% Cu, 15 g/t Ag, and 2 g/t Au. There are no very high grades in the 130 Zone; therefore, during the estimation process there was no restriction on high grade influence in that zone. The range at which to limit grades greater than the high grade assay cut-off was chosen to be 40 x 30 x 7 m oriented at 165 degrees in the major axis and 0 degrees dip. In other words, composite grades greater than the threshold amounts would not be used in the estimation of blocks if those high grade composites are outside the respective distance from that block. It is important to emphasize that the method employed for this study was not to cut the high grade outliers but to limit their influence.

14.5.5. Specific Gravity Estimation

A total of 2,711 bulk specific gravity (SG) measurements were provided by MintoEx of which 1,422 are within the mineralized solids. The SG's in the mineralized solids ranged from a low of 2.07 to a high of 4.56 with a mean value of 2.71, standard deviation of 0.14 and CV of 0.05 illustrating a very tight distribution. The SG values were interpolated into the blocks using the inverse distance to the second power interpolator. At least 4 samples within a 100 x 100 x 25 m radius were needed to estimate a block. Values greater than 3.3 were limited to a 20 m radius in influence.

14.5.6. Variography

Experimental variograms and variogram models in the form of correlograms were generated for Cu, Au and Ag grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from 15 to 22 percent of the total sill. Cu, Au and Ag variogram models used for grade estimation are summarised in Table 14-21. Note that the rotations of the angles are given according to the GSLIB convention used by MineSight™ Compass.

Table 14-21: Minto North - 115 Zone Variogram Model

Parameter	Cu			Au			Ag		
Nugget (C0)	0.15			0.22			0.14		
C1	0.85			0.78			0.86		
	Range	Rotation	Angle	Range	Rotation	Angle	Range	Rotation	Angle
Major	70	R1	166	60	R1	37	80	R1	115
Minor	60	R2	-1	30	R2	-11	60	R2	20
Vertical	7	R3	-28	37	R3	12	10	R3	-16

Note; R1 is the rotation around the Z axis, R2 is the rotation around the X axis with counter-clock wise being positive and R3 is the rotation around the Y axis with clock-wise being positive.

14.5.7. Block Model Definition

The Block Model used for calculating the resources was defined according to the limits specified in Table 14-22. The block model is orthogonal and non-rotated reflecting the orientation of the deposit. The block size chosen was 10 x 10 x 3 m, roughly reflecting drill hole spacing (i.e. 1 – 2 blocks between drill holes) which is at approximately 15 to 20 m centers and a proposed 3 m bench height.

Table 14-22: Specifications for the Minto North Block Model

Description	Easting (X m)	Northing (Y m)	Elevation (Z m)
Block Model Origin	384,000	6,945,750	750
Block Dimension	10	10	3
Number of Blocks	60	50	80
Rotation	0	0	0

14.5.8. Resource Estimation Methodology

The estimation plan includes the following items:

- Mineralized zone code and percentage of modeled mineralization in each block;
- Estimated bulk specific gravity based on an inverse distance squared method;
- Estimated block Cu, Au, and Ag grades by ordinary kriging, using a two pass estimation strategy for all mineralized zones. The two estimation passes enabled better description of local metal grades.

For the 115 Zone, major direction of continuity of the Cu grades was the ellipsoid direction chosen for the estimation process was chosen to be 165 degrees azimuth and 0 degrees dip for the major axis, 285 degrees azimuth and 0 degrees dip for the minor axis and 0 degrees azimuth and 90 degrees dip for the vertical axis. This direction follows the general orientation

of the modelled 115 Zone. For the 120 and 130 Zones, the ellipsoid direction chosen for the estimation process was same as for the 115 Zone. Table 14-23 summarizes the search ellipse dimensions for the estimation passes.

Table 14-23: Minto North Search Ellipse Parameters for 115, 120 and 130 Zones

Pass	Major Axis	Semi-Major Axis	Minor Axis	1 st Rotation Angle Azimuth	2 nd Rotation Angle Dip	3 rd Rotation Angle	Min. No. Of Comps	Max. No. Of Comps	Max. Samples per Drill hole
1	70	60	10	165	0	0	4	16	4
2	40	30	7	165	0	0	4	16	4

14.5.9. Resource Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites;
- Checks that the general drift and the local grade trends compared to the drift and local grade trends of the composites;
- Ensures that all blocks in the core of the deposit have been estimated;
- Checks that topography has been properly accounted for;
- Checks against manual approximate estimates of tonnage to determine reasonableness; and
- Inspection and explanation for potentially high grade block estimates in the neighborhood of the extremely high assays.

A full set of cross sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites, the variogram model used, and the estimation plan applied.

These validation techniques included the following:

- Visual inspections on a section-by-section and plan-by-plan basis;
- The use of Grade Tonnage Curves;
- Swath Plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates;
- An inspection of histograms of distance of closest samples to the estimated blocks, average distance to blocks for all composites used in the estimation which gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources; and

-
- Analysis of Relative Variability Index, which quantifies variability and relative error on a block-by block basis within the deposit in addition to assisting with the classification of resources.

14.5.10. Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Minto North deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Garth Kirkham, P.Geo., an “independent competent person” as defined by National Instrument 43-101.

Drill hole spacing in Minto North deposit is sufficient for geostatistical analysis and evaluating spatial grade variability. Kirkham Geosystems is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block;
- Number of composites allowed per drill hole;
- Distance to nearest composite used to estimate a block;
- Average distance to the composites used to estimate a block; and
- An evaluation of relative error on a block by block basis.

The classification of resources was based primarily upon distance to nearest composite however all of the quantitative measures, as listed above were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

For the 115 Zone, measured blocks were determined to have a block to nearest composite of 30 meters. In addition, the blocks were inspected for average distance to composite which was less than 40 meters, minimum number of drill holes which was 3 however in cases where the minimum number of drill holes was less than 3 then the distance to composite, average distance to composite, number of composites and error were evaluated to insure that confidence in the categorization of resources was warranted. Indicated blocks were determined

to have a distance to composite greater than 30 meter however there were no blocks that exceeded 50 meters.

In addition, the number of drill holes, average distance to block from composite and the number of composites used along with relative error, were evaluated to ensure confidence.

For the 120 zone, the same criteria was employed however resources categorized for the indicated category were determined to have a block to nearest composite of 30 meters. In addition, the blocks were inspected for average distance to composite which was less than 40 meters, minimum number of drill holes was in most cases 2 however in cases where the minimum number of drill holes was less than 2 then the distance to composite, average distance to composite, number of composites and error were evaluated to insure that confidence in the categorization of resources was upheld. Inferred blocks were determined to be have a distance to composite greater than 30 meter however there were no block that exceeded 50 meters. In addition, the number of drill holes, average distance to block from composite, number of composites used along with relative error was evaluated.

For the 130 Zone, although the zone has demonstrated geological continuity, it does not have demonstrated geostatistical continuity by virtue of the relatively low number of data points available and the relatively small footprint of the zone. Therefore, the 130 zone is categorized as inferred at this time.

14.5.11. Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade.

The reader is cautioned that these values should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Cu grade tonnage curves for different resource categories are presented in Figure 14-28 and Figure 14-29.

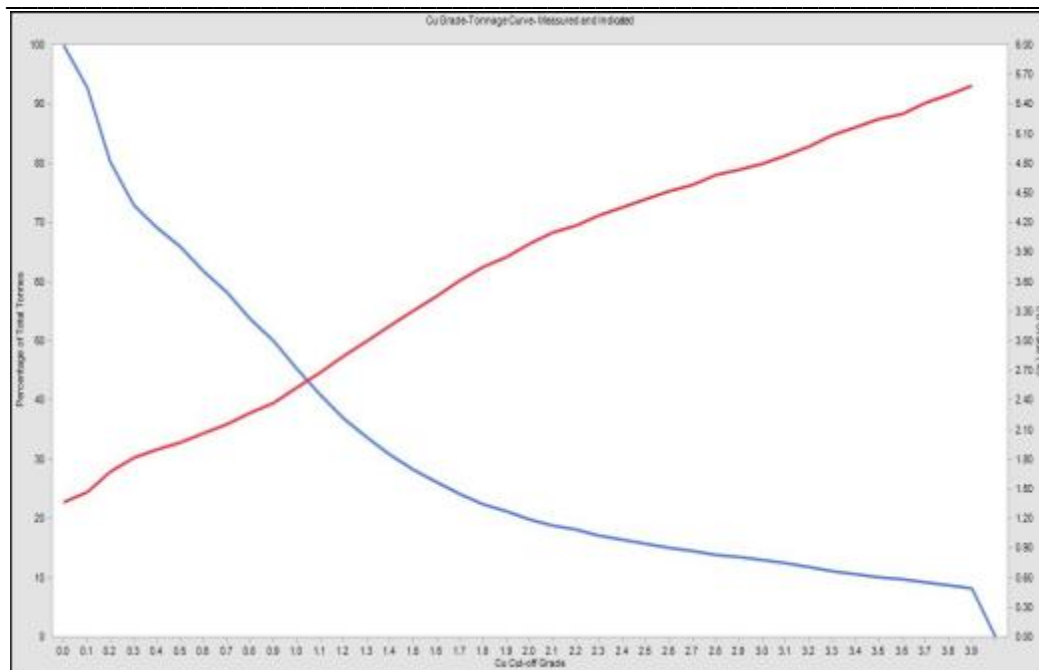


Figure 14-28: Minto North - Cu Grade Tonnage Curve for Measured and Indicated Blocks

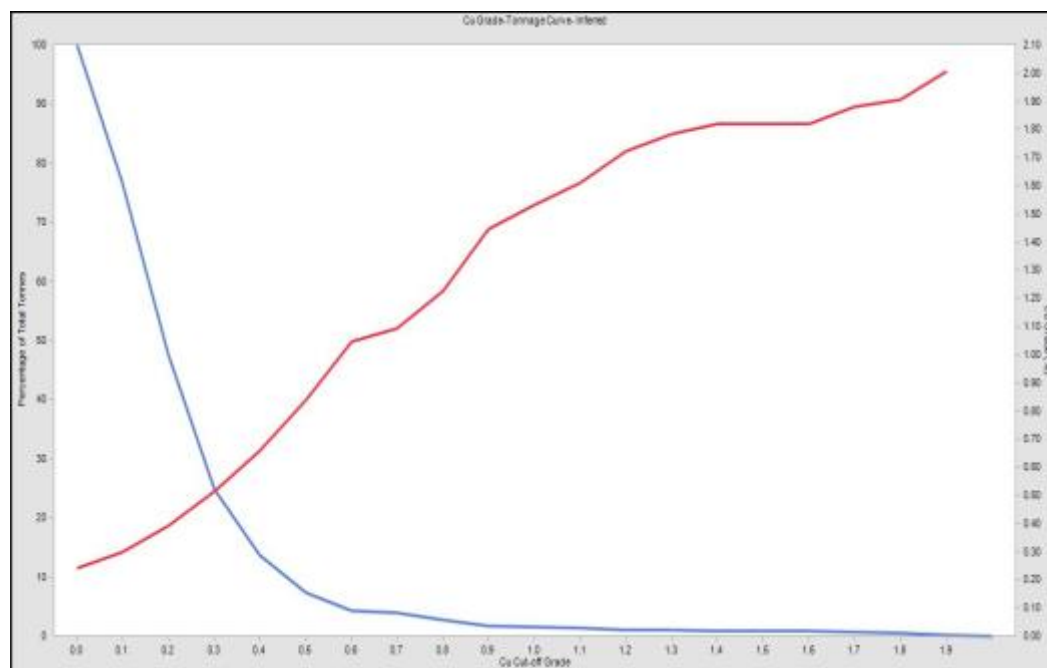


Figure 14-29: Minto North - Cu Grade Tonnage Curve for Inferred Blocks

14.5.12. Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

“[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth’s crust in such form and quantity and of such a grade or

quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”

The “reasonable prospects for economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries. It is the opinion of the Qualified Person that the Minto North Deposit, as classified, has a reasonable expectation of economic extraction.

Table 14-24 presents the mineral resource statement for the Minto North deposit.

Table 14-24: Mineral Resource Statement at 0.5% Cu Cut-off for the Minto North Deposit, Kirkham Geosystems December 1, 2009

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs.)*	Contained Gold (K oz.)*	Contained Silver (K oz.)*
Measured (M)	1,844	2.15	1.11	7.7	87,530	66	456
Indicated (I)	264	1.04	0.6	5.76	6,055	5	49
Sub-total (M+I)**	2,108	2.01	1.04	7.46	93,585	71	505
Additional Inferred	25	0.84	0.40	4.4	457	0	3

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

14.6. Minto East Deposit

The Minto East deposit is also a new discovery made in late 2009 and comprises relatively near surface (i.e. approximately 280 metres below surface), higher grade copper-gold mineralization. This is the first mineral resource estimate for the Minto East deposit and it is reported using a 0.5% and a 1.5% copper cut-off, as tabulated below (Table 14-25). This resource estimate was based on a total of 33 holes within a very tightly constrained area. A solid was created based on mineralized intersections and used to constrain the interpolation of grades.

Table 14-25: Tonnage & Grade Estimates of the Minto East Deposit Reported in June 2010

Classification	Tonnes	Copper	Gold	Silver	Contained Cu	Contained Gold	Contained Silver
	(000's)*	(%)	(g/t)	(g/t)	(000's lbs.)*	(000's oz)*	(000's oz)*
For a 0.5% Cu Cut-off							
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	736	2.35	0.99	6.2	38,132	23	147
Additional Inferred	588	1.73	0.81	5.6	22,426	15	106
For a 1.5% Cu Cut-off							
Measured (M)	-	-	-	-	-	-	-
Indicated (I)	541	2.88	1.16	7.1	34,349	20	123
Additional Inferred	284	2.55	0.95	6.4	15,966	9	58

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

This resource estimate was updated based on the addition of 17 drill holes from the 2010 exploration campaign that were focussed at upgrading the resources for the Minto East Deposit. These new resources are the subject of this section.

14.6.1. Geology Model

A solid model of the 700 ore zone within the Minto East Deposit (Figure 14—30) was created from sections and based on a combination of lithology, copper grades and site knowledge. Although there are a number of mineralized layers intersected within the drill holes, the grades are not sufficient at this time to warrant estimation of the upper zones and the estimation was limited to the 700 zone only. The ore zone solid was used to constrain grade interpolation.

Every intersection was inspected and the solid was then manually adjusted to match the drill intercepts. Once the solid model was created, it was used to code the drill hole assays and composites for subsequent geostatistical analysis. The solid zone was utilized to constrain the block model by matching assays to those within the zones. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from strike and dip of the mineralized zone and site knowledge and on-site observations by MintoEx's geological staff.

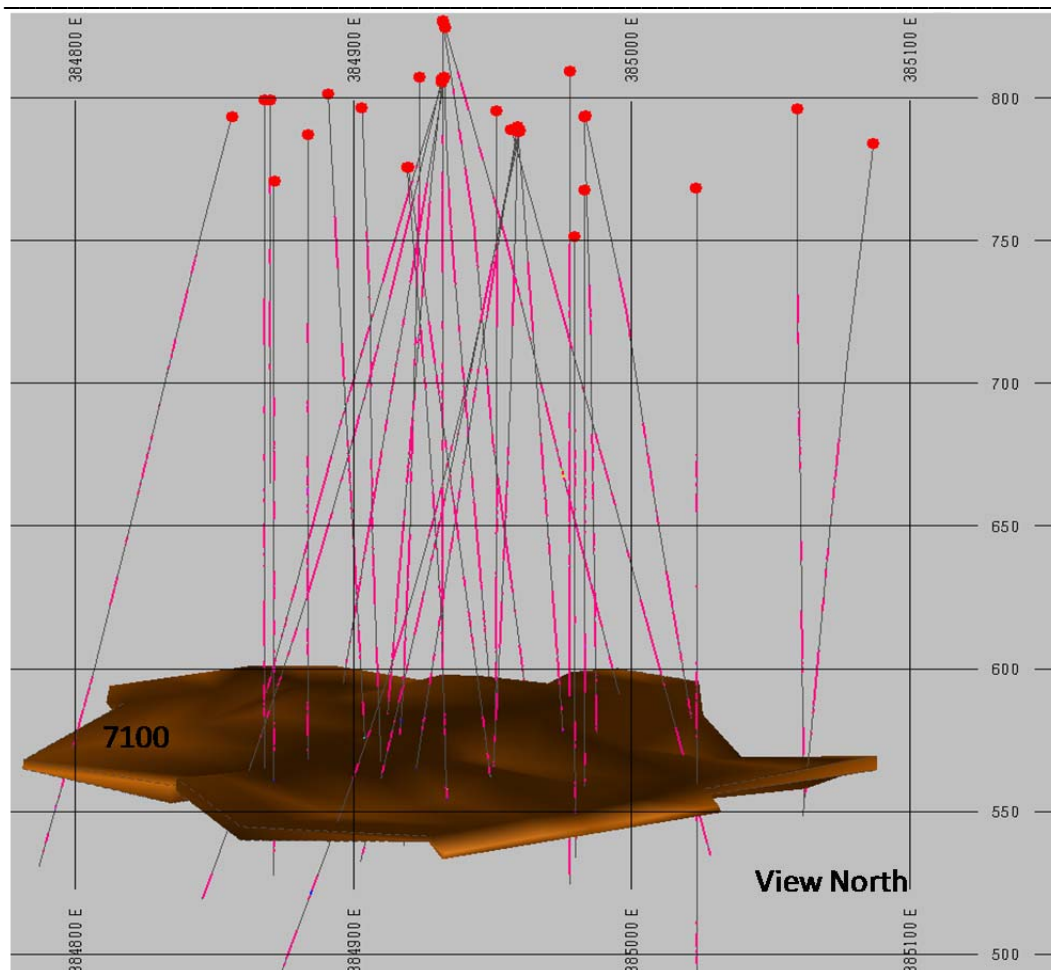


Figure 14—30: View North of the Modelled Minto East Mineralized Domain

14.6.2. Data

The drill hole database was supplied in electronic format by MintoEx. This included collars, down hole surveys, lithology data and assay data (i.e. Au g/t, Cu %, Ag g/t, SG with down hole from and to intervals in metric units). The database was numerically coded by mineralized zone solid; 700 Zone Ore. The database was then manually adjusted drill hole by drill hole to insure accuracy of zonal intercepts.

Table 14-26 shows statistics of copper, gold and silver assays. The average Cu, Au, and Ag grades for the 700 zone are 1.88%, 1.05 g/t, 5.58 g/t, respectively. Note that the overall average grades from the 700 zone is relatively similar to the Minto North (120 zone) deposit which are both higher than in Minto South and the Ridgetop deposits.

Table 14-26: Statistics for Un-weighted Copper, Gold and Silver Assays for the Minto East Deposit

700 Zone	#	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
Cu	232	0	7.07	1.88	0.09	1.77	2.91	1.78	0.95
Au	232	0	31	1.05	0.02	0.7	1.13	2.96	2.8
Ag	232	0.1	48.8	5.59	0.51	4.6	8.99	5.85	1.05

All Minto East Assays	#	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
Cu	4,448	0	7.07	0.18	0	0.03	0.11	0.59	3.34
Au	4,448	0	31	0.07	0.02	0.02	0.02	0.72	9.95
Ag	4,448	0.1	48.8	0.64	0.12	0.22	0.51	1.86	2.9

Note that the table above shows those intervals that intersect the high grade 700 zone only in addition to all of the intervals for those holes. This illustrates the significance of the 700 zone in addition to the high variability outside the zone (in the waste).

14.6.3. Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the undue influence of very high grades. Distribution of sample lengths within the mineralized domains is presented in Figure 14—31. Table 14-27 shows the basic statistics for the 1.5 m Cu composite grades within the mineralized domains.

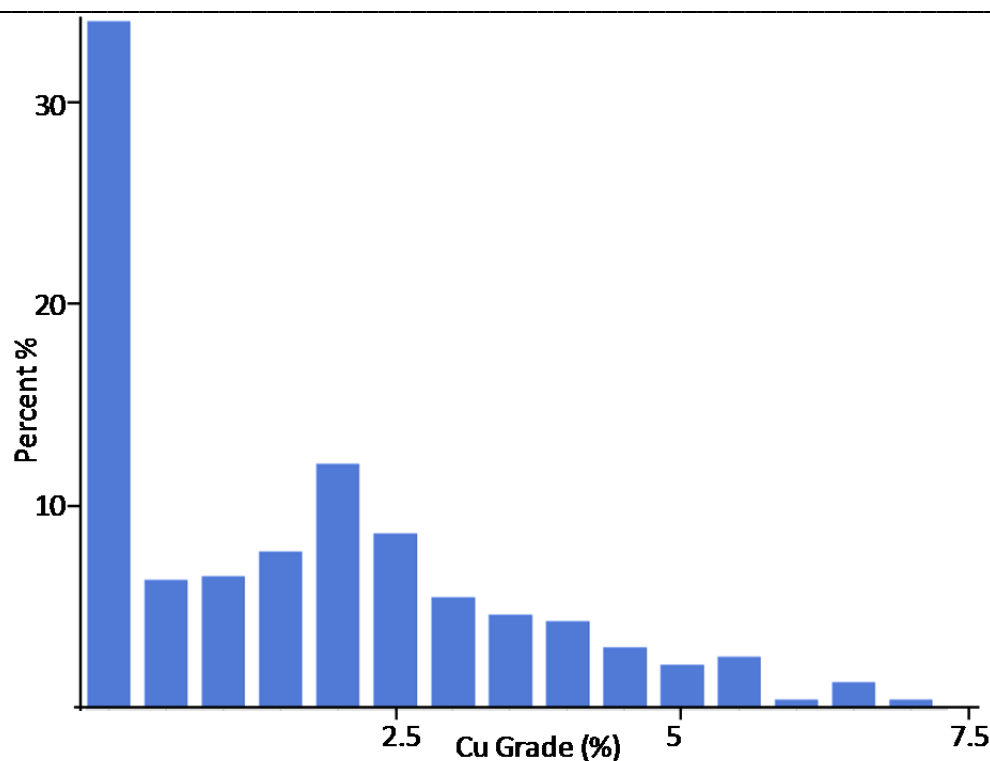


Figure 14—31: Minto East – Histogram of Cu Composite grades in the mineralized zones

Table 14-27: Composite Statistics Weighted by Length

700 ZONE	Length	Min	Max	Mean	1st Quartile	Median	3rd Quartile	SD	CV
700 Zone									
Cu	347.4	0.00	7.07	1.88	0.09	1.77	2.91	1.78	0.94
Au	347.4	0.00	31.00	1.05	0.02	0.70	1.13	2.94	2.81
Ag	347.4	0.10	48.80	5.58	0.51	4.60	8.99	5.83	1.04
All Minto East Assays									
Cu	6,441.9	0.00	7.07	0.18	0.00	0.03	0.11	0.60	3.28
Au	6,441.9	0.00	31.00	0.07	0.02	0.02	0.02	0.72	9.81
Ag	6,441.9	0.10	48.80	0.66	0.12	0.22	0.51	1.89	2.86

Note that the table above shows those intervals that intersect the high grade 700 zone only in addition to all of the intervals for those holes. This illustrates the significance of the 700 zone in addition to the high variability outside the zone (in the waste).

14.6.4. Evaluation of Extreme Assay Values

During the estimation process the influence of composites greater than 5.2% Cu, 13 g/t Ag, and 3 g/t Au have been limited. In the case of the Cu composites, values higher than 5.2% were cut whilst in the case of Ag and Au, grades greater than 13 g/t and 3 g/t, were restricted to a 15 m

range. In other words, composite grades greater than the threshold amounts were not used in the estimation of blocks if those high grade composites are outside the respective distance from that block.

14.6.5. Specific Gravity Estimation

A total of 615 in-situ dry bulk density (ISBD) measurements were provided by MintoEx of which 133 are within the mineralized solids. The SG's in the mineralized solids ranged from a low of 2.47 to a high of 3.34 with a mean value of 2.78, standard deviation of 0.15 and CV of 0.05 illustrating a very tight distribution. The SG values were interpolated into the blocks using the inverse distance to the second power. At least 2 samples within a 150 x 150 x 50 m radius were needed to estimate a block.

14.6.6. Variography

Experimental variograms and variogram models in the form of correlograms were generated for Cu, Au and Ag grades. The nugget effect values (i.e., metal variability at very close distance) were established from down hole variograms. The nugget values range from less than 1% to 32% of the total sill. The Cu, Au and Ag variogram models are summarised in Table 14-28. The block model derived from ordinary kriging was useful for comparison purposes but appeared overly smoothed so it was judged by the author that the inverse distance model was a better representation of the grade distribution within the Minto East deposit.

Table 14-28: 700 Zone Variogram Model

	CU			AU			AG		
Nugget (C0)	0.05			0.002			0.321		
C1	0.718			0.91			0.41		
C2	0.232			0.088			0.289		
First Structure									
	Range	Azim	Dip	Range	Azim	Dip	Range	Azim	Dip
Maximum	26	357	-7	23	281	-1	17	335	-11
Intermediate	16	131	-80	25	11	20	32	49	55
Minimum	10	86	7	3	194	70	3	252	33
Second Structure									
	Range	Azim	Dip	Range	Azim	Dip	Range	Azim	Dip
Maximum	274	291	54	132	153	72	197	340	-15
Intermediate	78	172	19	27	23	12	20	60	31
Minimum	36	72	29	130	110	-14	50	271	54

14.6.7. Block Model Definition

The Block Model used for estimating the resources was defined according to the limits specified in Table 14-29. The block model is orthogonal and non-rotated reflecting the orientation of the deposit. The block size chosen was 10 m by 10 m by 3 m, roughly reflecting drill hole spacing (i.e. 3 – 4 blocks between drill holes) which is at approximately 30 to 40 m centers and a proposed 3 m bench height.

Table 14-29: Specifications for the Minto East Block Model

Description	Easting (X)	Northing (Y)	Elevation (Z)
Block Model Origin	384,700	6,944,750	3
Block Dimension	10	10	3
Number of Blocks	40	40	171
Rotation	0	0	0

14.6.8. Resource Estimation Methodology

The estimation plan includes the following items:

- Mineralized zone code and percentage of modeled mineralization in each block;
- Estimated bulk specific gravity based on an inverse distance squared method;
- Estimated block Cu, Au, and Ag grades by inverse distance to the second power, using a three estimation pass strategy for the mineralized zone. The three estimation passes enabled better description of local metal grades.

For the 700 Zone, the search ellipsoid was oriented along the major direction of continuity of the Cu grades, 340° azimuth and -15° dip for the major axis, 50° azimuth and 0° dip for the minor axis and 340° azimuth and 75° dip for the vertical axis. Table 14-30 summarizes the search ellipse dimensions for the estimation passes.

Table 14-30: Search Ellipse Parameters for the 700 Zone

Pass	Major Axis	Semi-Major Axis	Minor Axis	1 st Rotation Angle Azimuth	2 nd Rotation Angle Dip	3 rd Rotation Angle	Min. No. Of Comps	Max. No. Of Comps	Max. Samples per Drill hole
1	30	30	10	340	-15	0	6	20	5
2	60	60	20	340	-15	0	6	20	5
3	150	150	50	340	-15	0	2	20	5

14.6.9. Resource Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

-
- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites;
 - Checks that the general drift and the local grade trends compared to the drift and local grade trends of the composites;
 - Ensures that all blocks in the core of the deposit have been estimated;
 - Checks that topography has been properly accounted for;
 - Checks against manual approximate estimates of tonnage to determine reasonableness; and
 - Inspection and explanation for potentially high grade block estimates in the neighbourhood of the extremely high assays.

A full set of cross sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites, the variogram model, and the estimation plan applied.

- These validation techniques included the following:
- Visual inspections on a section-by-section and plan-by-plan basis;
- The use of Grade Tonnage Curves;
- Swath Plots comparing Kriged estimated block grades with Inverse Distance and Nearest Neighbour estimates;
- An inspection of histograms of distance of first composite to nearest block, average distance to blocks for all composites used which gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources; and
- Analysis of Relative Variability Index, which quantifies variability and relative error on a block-by block basis within the deposit in addition to assisting with the classification of resources.

14.6.10. Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the mineral resources will be affected by these factors that are more suitably assessed in a conceptual study.

Mineral Resources for the Minto East deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) by Garth Kirkham, P.Geo., an “independent qualified person” as defined by National Instrument 43-101.

Drill hole spacing in Minto East deposit is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. Kirkham Geosystems is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Continuity of Cu grades defined from variogram models;
- Number of data used to estimate a block;
- Number of composites allowed per drill hole;
- Distance to nearest composite used to estimate a block;
- Average distance to the composites used to estimate a block; and
- An evaluation of relative error on a block by block basis.

The classification of resources was based primarily upon distance to nearest composite however all of the quantitative measures, as listed above were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

For the 700 Zone, blocks were classified as measured if they were within 20 m of a composite, had an average distance of all composite used less than 40 m and were interpolated with a minimum of two drill holes. Blocks were classified as indicated if the nearest composite greater than 20 metre but less than 40 metres away. In addition, the number of drill holes used for the estimate, the average distance of all composites used and the number of composites and the relative error were evaluated to ensure confidence. The remaining blocks were classified as inferred; however, all of these blocks were within a maximum of 60 m of the nearest composite and had at least 2 drill holes contributing to the estimate of that block. There were a small percentage of inferred block that were interpolated with one drill hole however these blocks are below cut-off and not reported.

14.6.11. Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 14-31 shows global quantities and grade in the Minto East deposit at different Cu cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to cut-off grade.

Table 14-31: Minto East – Sensitivity analyses of Global Tonnage and Grades Deposit at Various Cu Cut-off Grades

Classification	Cut-Off (Cu %)	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (000's lbs.)*	Contained Gold (000's oz)*	Contained Silver (000's oz)*
Measured (M)	2	425	3.03	1.42	7.8	28,382	19	106
	1.5	487	2.87	1.34	7.5	30,766	21	118
	1	562	2.65	1.24	7.1	32,803	22	128
	0.5	688	2.30	1.07	6.3	34,842	24	139
	0.4	713	2.23	1.04	6.1	35,092	24	140
	0.3	725	2.20	1.03	6.0	35,188	24	141
	0.2	732	2.18	1.02	6.0	35,224	24	141
	0.1	745	2.15	1.00	5.9	35,266	24	141
Indicated (I)	2	197	2.69	1.01	6.5	11,663	6	41
	1.5	268	2.44	0.95	6.1	14,401	8	52
	1	344	2.17	0.86	5.5	16,437	10	61
	0.5	490	1.74	0.70	4.6	18,805	11	72
	0.4	526	1.65	0.67	4.4	19,167	11	74
	0.3	568	1.56	0.62	4.1	19,483	11	75
	0.2	627	1.43	0.57	3.8	19,804	12	77
	0.1	671	1.35	0.54	3.6	19,946	12	78
Sub-total (M+I)	2	622	2.92	1.29	7.37	40,045	26	147
	1.5	755	2.71	1.20	7.00	45,167	29	170
	1	906	2.47	1.09	6.47	49,240	32	189
	0.5	1,177	2.07	0.92	5.57	53,647	35	211
	0.4	1,239	1.99	0.88	5.37	54,259	35	214
	0.3	1,293	1.92	0.85	5.19	54,671	35	216
	0.2	1,359	1.84	0.81	4.99	55,028	36	218
	0.1	1,416	1.77	0.78	4.81	55,212	36	219
Additional Inferred	2	-	0	0	0	-	-	-
	1.5	4	1.76	0.78	4.9	151	0	1
	1	5	1.66	0.72	4.6	191	0	1
	0.5	14	1.03	0.45	2.8	316	0	1
	0.4	16	0.96	0.41	2.6	336	0	1
	0.3	21	0.81	0.34	2.2	373	0	1
	0.2	23	0.75	0.31	2.0	385	0	2
	0.1	24	0.74	0.31	2.0	387	0	2

*Rounded to nearest thousand

Cu grade tonnage curves for different resource categories are presented in Figure 14—32 and Figure 14—33.

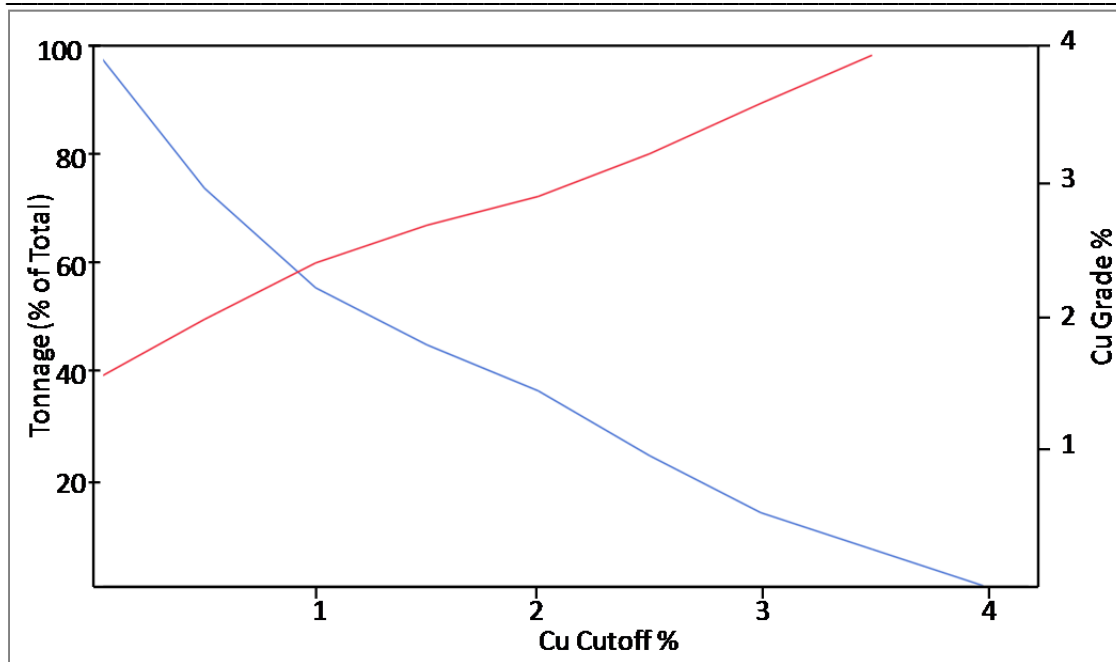


Figure 14—32: Minto East - Cu Grade Tonnage Curve for Measured and Indicated Blocks (red- Grade, blue- Tonnes)

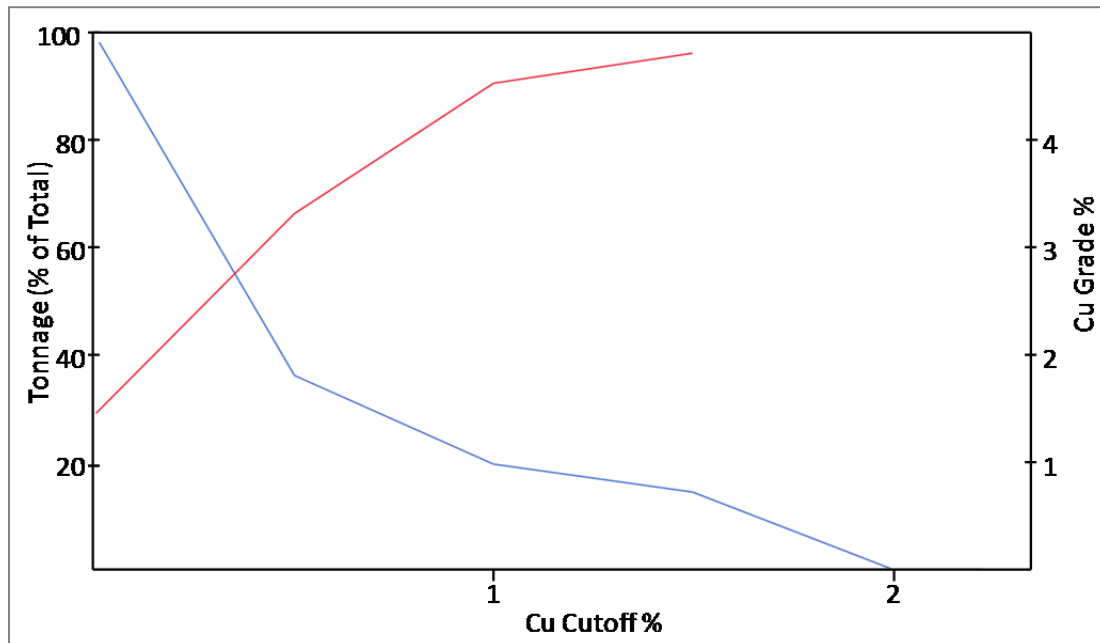


Figure 14—33: Minto East - Cu Grade Tonnage Curve for Inferred Blocks (red- Grade, blue- Tonnes)

14.6.12. Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

“[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”

The “reasonable prospects for economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account the likely extraction scenarios and process metal recoveries. It is the opinion of the Qualified Person that the Minto East Deposit, as classified, has a reasonable expectation of economic extraction.

Table 14-32 presents the mineral resource statement for the Minto East deposit.

Table 14-32: Tonnage & Grade Estimates of the Minto East Deposit Reported by Class in October 2010 (at a 0.5% Cu cut-off)

Classification	Tonnes (000's)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs.)*	Contained Gold (K oz.)*	Contained Silver (K oz.)*
Measured (M)	688	2.30	1.07	6.3	34,842	24	139
Indicated (I)	489	1.74	0.70	4.6	18,805	11	72
Sub-total (M+I)**	1,177	2.07	0.92	5.57	53,647	35	211
Additional Inferred	14	1.03	0.45	2.8	316	0	1

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

14.7. Resource Estimation Interpretations and Conclusions

SRK reviewed and audited the exploration data available for Minto South and Ridgetop deposits. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

SRK, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization on three deposits. A total of 39 (Minto South) and 7 (Ridgetop) separate wireframes were constructed in GEMS to represent ore zones alone. SRK considers that the geological model is a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, SRK constructed new mineral resource block models for Minto South and Ridgetop deposits constraining grade interpolation to within the modelled mineralization domains. After validation and classification, SRK considers that the mineral resources for the all three deposits are appropriately reported at a 0.5% Cu cut-off considering the open pit mining scenario discussed in the report.

Mineral resources for Minto South and Ridgetop deposits have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” Guidelines. In the opinion of SRK, the block model resource estimate and resource

classification reported herein are very representative of the copper, gold, and silver mineral resources found in the three deposits. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

Kirkham Geosystems reviewed and audited the exploration data available for the Minto North deposit. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

Kirkham Geosystems, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization the deposit. A total of three wireframes (i.e. 115, 120 and 130 zones along with a cross-cutting dyke) were constructed in MineSight™. Kirkham Geosystems considers that the geological model is a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, Kirkham Geosystems constructed new mineral resource block models for Minto North constraining grade interpolation to within the modelled mineralization domains. After validation and classification, Kirkham Geosystems considers that the mineral resources for the Minto North deposit is appropriately reported at a 0.5% Cu cut-off considering open pit mining scenario as discussed in the report.

Mineral resources for the Minto North deposit has been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” Guidelines. In the opinion of Kirkham Geosystems, the block model resource estimate and resource classification reported herein are very representative of the copper, gold, and silver mineral resources found in Minto North. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

A number of factors may affect the quality and quantity of the current resource estimates, and thereby highlight opportunities for improvement:

- There are gaps in the understanding of the mineralization paragenesis. Improved understanding could benefit exploration models as well as the constraint on high-grade continuity and orientation. MintoEx are proactively making an effort in fundamental research to answer these questions.
- There are still some details that need to be constrained with respect to the structural geometries that are influencing the resource. Ductile and brittle fault structures and folding on various scales deform the ore horizons. The deformation history needs to be better constrained, and again research is currently on-going in order to answer these questions.
- There is poor control on the brittle structures that could impact the geotechnical assessment. It would be beneficial to undertake a mapping exercise of the current pits to determine the brittle fault and joint pattern. This information should be combined with drill hole logs, modeled structural information, mineralization offsets, exploration data and geophysical data (e.g. Titan 24 MT) to determine the structural patterns and position of major faults and folds.

Table 14-33 presents combined mineral resource at a 0.5% Cu cut-off for Minto South, Ridgetop, Minto North and Minto East Deposits.

Table 14-33: Combined Mineral Resource Statement at 0.5% Cu Cut-off for Minto South, Ridgetop, Minto Main, Minto North and Minto East Deposits, January 1, 2012

Classification	Tonnes (Kt)*	Copper (%)	Gold (g/t)	Silver (g/t)	Contained Copper (K lbs.)*	Contained Gold (K oz.)*	Contained Silver (K oz.)*
Measured (M)	14,828	1.35	0.53	4.43	441,744	254	2,114
Indicated (I)	33,103	0.98	0.32	3.42	714,010	341	3,643
Sub-total (M+I)**	47,931	1.09	0.39	3.73	1,155,754	595	5,758
Additional Inferred	8,493	0.81	0.24	2.88	151,764	65	788

*Rounded to nearest thousand

**Totals may not add exactly due to rounding

15. Mineral Reserve Estimates

15.1. Mineral Reserve – Summary

Table 15-1 shows a summary of the Minto reserve estimation as of January 1, 2012.

Table 15-1: Reserve Estimation Estimate

	Mineral Reserve Class	Tonnes (Kt)	Diluted Grade			Contained Metal		
			Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit Reserves								
Minto North	Proven	1,596	2.26	1.21	8.12	79	62	417
	Probable	9	1.68	0.58	6.92	0	0	2
	Sub-total	1,604	2.26	1.21	8.12	80	62	419
Ridgetop	Proven	1,073	1.02	0.25	2.12	24	9	73
	Probable	1,020	1.00	0.28	2.97	22	9	97
	Sub-total	2,093	1.01	0.26	2.54	46	18	171
MSD - 118	Proven							
	Probable	483	1.28	0.10	1.81	14	2	28
	Sub-total	483	1.28	0.10	1.81	14	2	28
MSD - Area 2	Proven	3,306	1.44	0.54	4.99	105	57	530
	Probable	1,694	1.00	0.27	3.20	37	15	174
	Sub-total	5,000	1.29	0.45	4.38	142	72	704
Open Pit Subtotal	Proven	5,974	1.58	0.67	5.31	208	128	1,020
	Probable	3,204	1.04	0.25	2.92	74	25	301
	Sub-total	9,179	1.39	0.52	4.48	282	153	1,321
Underground Reserves								
Minto East	Proven							
	Probable	709	2.28	1.04	6.15	36	24	140
	Sub-total	709	2.28	1.04	6.15	36	24	140
MSD - Area 2 / 118	Proven							
	Probable	1,731	1.76	0.74	7.19	67	41	400
	Sub-total	1,731	1.76	0.74	7.19	67	41	400

	Mineral Reserve Class	Tonnes (Kt)	Diluted Grade			Contained Metal		
			Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlbs)	Au (koz)	Ag (koz)
MSD - Copper Keel	Proven	106	1.74	0.61	6.30	4	2	22
	Probable	1,455	1.81	0.65	6.70	58	30	313
	Sub-total	1,561	1.81	0.64	6.67	62	32	335
MSD - Wildfire	Proven	301	1.80	0.77	6.06	12	7	59
	Probable	59	1.59	1.00	7.85	2	2	15
	Sub-total	360	1.76	0.80	6.35	14	9	73
Underground Subtotal	Proven	407	1.78	0.73	6.12	16	9	81
	Probable	3,954	1.87	0.76	6.83	163	97	868
	Sub-total	4,361	1.86	0.76	6.77	179	106	948
Stockpiles	Proven	852	1.23	0.41	3.71	23	11	102
	Probable	0	0	0	0			
	Sub-total	852	1.23	0.41	3.71	23	11	102
Total Reserves	Proven	7,233	1.55	0.64	5.17	247	148	1,203
	Probable	7,158	1.50	0.53	5.08	237	122	1,169
	Sub-total	14,392	1.53	0.58	5.12	484	270	2,371

15.2. Mineral Reserves Open Pit

There are slight modifications conducted as part of the Phase VI open pit design for Area 2, Area 118, Minto North, Ridgetop North and Ridgetop South. The Main zone reserves were depleted due to the completion of the Main Pit in 2011 with no subsequent stages being planned.

Stockpiles indicated in the Phase V Pre-feasibility Study were adjusted in the Phase VI PFS to reflect mining activities that occurred in 2011. Since there has been no extraction of underground ore the stockpiles are considered to be a Proven Open Pit Reserve.

There were no new open pits generated by the Phase VI Pre-feasibility Study.

15.2.1. Open Pit Reserve Model

The 3-D Minto North resource model (provided by Kirkham Geosystems) was not modified since it was utilized in the Phase V PFS pit design. In addition, the 3-D Ridgetop model was not modified since it was issued by SRK in 2010 and utilized in the Phase V Pit Design. The Area 2 and 118 resource models were replaced with the MSD model that was issued by SRK in Q3 2011.

The Phase VI parameters utilized by Capstone are summarized in Table 15-2 below.

Table 15-2: Open Pit Parameters for Whittle™ Pit Shell

Metal Prices				Comments
Metal prices (US\$)	US	\$2.50	/lb. Cu	C\$2.78/lb.
	US	\$300	/oz. Au	as per Silver Wheaton agreement
	US	\$3.90	/oz. Ag	as per Silver Wheaton agreement
Exchange Rate				Comments
US Dollars/Canadian Dollars		0.90		Estimate
Grade Factor				Comments
Dilution		6.00%	% waste rock in mill feed	All open pits
Grade of waste rock		0.00%	% Cu	
		0.00%	Au g/t	
		0.00%	Ag g/t	
Mill Recovery				Comments
Mill Recovery	Cu	91%		
	Au	70%		
	Ag	78%		
Concentrate Produced				Comments
Moisture Content in Concentrates		8.00%		
Contained Metal in Concentrate	Cu	40.00%		
	Au	variable	varies with Au and Cu head grade	
	Ag	variable	varies with Ag and Cu head grade	
Payable Metal in Concentrate				Comments
Payable metal terms were used as per the MRI Trading contract (confidential)				
Treatment and Refining				Comments
Cu conc. treatment	US	Per MRI contract	/dmt	
Cu refining	US		/lb. Cu	
Au refining	US		/oz. Au	
Ag refining	US		/oz. Ag	

Freight and Marketing				Comments
Freight & Marketing (all inclusive)	US	\$155.98	/wmt	includes trucking; shipping; port charges; insurance
Freight & Marketing (all inclusive)	US	\$169.54	/dmt	
Royalty				Comments
Royalty charge		1.00%	of Net Value	Payable to Selkirk First Nations

15.2.2. Economic Pit Limit

The Whittle™ pit optimization evaluations for the Area 2/118 zones of MSD, Ridgetop and Minto North deposits were then utilized to confirm that the optimized approach to extract the reserves using open pit mining methods. The Wildfire zone of MSD was also evaluated using Whittle™, however, it was confirmed that the optimized approach was to extract the reserves with underground mining methods.

15.2.3. Optimization Parameters and Results

Table 15-3 summarizes the various parameters utilized in the analysis. In addition, the mill throughput continued to be based on a maximum annual throughput rate of 1.369 Mt.

During 2011, stripping of the Area 2 pit commenced. Therefore, the estimated “as-built” pit shell for December 31, 2011 was used as the starting point for the Phase VI PFS.

Table 15-3: Pit Optimization Parameters for Phase VI Open Pit Designs

Item	Unit	Value		
		North	Ridgetop	Area 2/118
Operating Costs				
Waste mining Cost	C\$/waste tonne	3.12		
Ore Mining Cost	C\$/ore tonne	3.12		
Processing and G&A Cost	C\$/milled tonne	28.40		
UG Mining Cost	C\$/ore tonne	36.00		
Pit Slope Angles				
Overburden	inter-ramp °	30°	30°	30°
Rock	inter-ramp °	52°	47°west, 53°east	47°west, 53°east
Dilution	%	6.0%	6.0%	6.0%
Mining recovery	%	100%	100%	100%
Strip ratio (est.)	t:t	7.0	6.0	6.0
Processing rate	t/day milled	3,750		
Processing rate	t/yr milled	1,370,000		

The results of the Whittle™ pit optimization evaluation for varying revenue factors are summarized in Table 15-4 through Table 15-6, as well as Figure 15—1 through to Figure 15—3,

for measured and indicated resources only. The selected Whittle shell (based on an evaluation of the results) was used as the basis for the detailed pit designs is highlighted in each of the tables.

Table 15-4: North Pit Optimization Results

Final	Revenue	Mine Life	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	NPV Best	NPV Worst
Pit	Factor	Years	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C\$x1000)	(C\$x1000)
10	0.68	0.82	1,093	2.86	1.66	10.66	\$ 139.93	69	58	375	9,428	8.63	10,521	99,146	99,146
11	0.7	0.84	1,125	2.82	1.63	10.51	\$ 137.92	70	59	380	9,623	8.55	10,748	99,761	99,761
12	0.72	0.94	1,258	2.64	1.49	9.71	\$ 128.87	73	60	393	9,864	7.84	11,122	101,338	101,338
13	0.74	1.02	1,352	2.53	1.41	9.25	\$ 123.38	75	61	402	10,104	7.47	11,456	102,489	102,486
14	0.76	1.02	1,361	2.52	1.4	9.2	\$ 122.86	76	61	403	10,101	7.42	11,462	102,517	102,513
15	0.78	1.03	1,374	2.5	1.39	9.14	\$ 121.90	76	61	404	10,092	7.34	11,466	102,535	102,531
16	0.8	1.07	1,430	2.45	1.34	8.91	\$ 119.31	77	62	410	10,312	7.21	11,742	103,287	103,231
17	0.82	1.09	1,461	2.42	1.32	8.79	\$ 117.82	78	62	413	10,439	7.15	11,900	103,608	103,510
18	0.84	1.10	1,473	2.41	1.31	8.74	\$ 117.30	78	62	414	10,446	7.09	11,919	103,646	103,540
19	0.86	1.13	1,513	2.38	1.29	8.62	\$ 115.82	79	63	419	10,726	7.09	12,239	104,024	103,819
20	0.88	1.14	1,528	2.36	1.28	8.57	\$ 114.85	80	63	421	10,799	7.07	12,327	104,101	103,876
21	0.9	1.15	1,554	2.34	1.26	8.47	\$ 113.81	80	63	423	10,875	7.00	12,429	104,222	103,917
22	0.92	1.17	1,587	2.31	1.24	8.36	\$ 112.33	81	63	427	11,044	6.96	12,631	104,377	103,966
23	0.94	1.19	1,612	2.29	1.22	8.27	\$ 111.29	81	64	429	11,108	6.89	12,720	104,452	103,940
24	0.96-0.98	1.21	1,641	2.26	1.21	8.16	\$ 109.88	82	64	431	11,160	6.80	12,801	104,493	103,893
25	1	1.23	1,685	2.22	1.18	8.01	\$ 107.87	83	64	434	11,280	6.69	12,965	104,511	103,768
26	1.02	1.24	1,710	2.2	1.16	7.93	\$ 106.84	83	64	436	11,356	6.64	13,066	104,498	103,659
27	1.04	1.26	1,739	2.18	1.15	7.84	\$ 105.87	84	64	439	11,510	6.62	13,249	104,446	103,474
28	1.06	1.27	1,762	2.16	1.13	7.78	\$ 104.84	84	64	441	11,605	6.59	13,367	104,405	103,329

Final	Revenue	Mine Life	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	NPV Best	NPV Worst
Pit	Factor	Years	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C\$×1000)	(C\$×1000)
29	1.08	1.28	1,770	2.16	1.13	7.76	\$ 104.84	84	64	442	11,698	6.61	13,468	104,375	103,282
30	1.1	1.31	1,828	2.12	1.1	7.59	\$ 102.83	85	65	446	11,972	6.55	13,800	104,167	102,774
31	1.12	1.32	1,840	2.11	1.1	7.58	\$ 102.38	86	65	449	12,075	6.56	13,915	104,117	102,693
32	1.14	1.32	1,842	2.11	1.1	7.58	\$ 102.38	86	65	449	12,075	6.56	13,917	104,115	102,689
33	1.16	1.33	1,863	2.09	1.08	7.51	\$ 101.35	86	65	450	12,166	6.53	14,029	104,004	102,473
34	1.18	1.33	1,869	2.09	1.08	7.5	\$ 101.35	86	65	451	12,177	6.52	14,046	103,991	102,451
35	1.2	1.33	1,877	2.08	1.08	7.47	\$ 100.90	86	65	451	12,176	6.49	14,053	103,980	102,429
36	1.22	1.34	1,887	2.07	1.07	7.45	\$ 100.38	86	65	452	12,224	6.48	14,111	103,913	102,317
37	1.24	1.34	1,892	2.07	1.07	7.43	\$ 100.38	86	65	452	12,228	6.46	14,120	103,903	102,303
38	1.26	1.34	1,906	2.06	1.06	7.39	\$ 99.86	86	65	453	12,287	6.45	14,193	103,808	102,150

Table 15-5: Minto South (Area 2/118) Pit Optimization Results

Final	Revenue	Mine	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	NPV Best (C\$x1000)	NPV Worst (C\$x1000)
Pit	Factor	Years	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)	(ktonnes)	Ratio	(ktonnes)	(C\$x1000)	(C\$x1000)
10	0.68	1.50	2,059	1.47	0.53	4.97	\$ 69.64	67	35	329	10,567	5.13	12,626	55,140	55,125
11	0.7	1.63	2,239	1.45	0.52	4.90	\$ 69.00	72	38	353	11,214	5.01	13,453	58,945	58,767
12	0.72	1.67	2,282	1.45	0.52	4.88	\$ 68.86	73	38	358	11,374	4.98	13,656	59,821	59,589
13	0.74	1.93	2,644	1.44	0.50	4.70	\$ 68.28	84	42	399	13,120	4.96	15,764	67,024	66,123
14	0.76	1.97	2,702	1.43	0.49	4.66	\$ 67.87	85	43	405	13,272	4.91	15,974	67,663	66,605
15	0.78	2.44	3,338	1.38	0.46	4.35	\$ 65.10	101	49	467	15,520	4.65	18,858	76,260	73,501
16	0.8	2.47	3,388	1.37	0.46	4.33	\$ 64.89	103	50	472	15,699	4.63	19,087	76,809	73,880
17	0.82	2.58	3,537	1.36	0.45	4.29	\$ 64.42	106	51	488	16,240	4.59	19,777	78,731	75,292
18	0.84	2.71	3,707	1.36	0.44	4.22	\$ 63.93	111	52	503	16,879	4.55	20,587	80,913	76,825
19	0.86	3.17	4,348	1.34	0.43	4.26	\$ 63.02	128	61	595	20,805	4.79	25,153	87,252	81,398
20	0.88	3.28	4,493	1.32	0.43	4.20	\$ 62.47	131	62	607	21,294	4.74	25,787	88,198	81,947
21	0.9	3.39	4,640	1.32	0.42	4.18	\$ 62.13	135	63	624	21,921	4.72	26,561	89,557	82,861
22	0.92	3.45	4,733	1.31	0.42	4.18	\$ 61.96	137	64	636	22,345	4.72	27,078	90,498	83,479
23	0.94	3.52	4,820	1.31	0.42	4.15	\$ 61.81	139	65	644	22,760	4.72	27,581	91,306	83,969
24	0.96	3.60	4,926	1.31	0.42	4.16	\$ 61.74	142	67	658	23,311	4.73	28,237	92,587	84,841
25	0.98	4.19	5,743	1.27	0.40	4.05	\$ 59.92	161	74	747	28,323	4.93	34,066	94,218	82,685
26	1	4.25	5,818	1.27	0.40	4.02	\$ 59.68	162	74	753	28,603	4.92	34,421	94,313	82,657
27	1.02	4.33	5,928	1.26	0.39	4.00	\$ 59.33	165	75	762	28,984	4.89	34,912	94,524	82,661

Final	Revenue	Mine	Ore Diluted	Diluted Grades				Contained Metal			Waste	Strip	Total	NPV Best	NPV Worst
Pit	Factor	Life	(kt)								(ktonnes)	Ratio	(ktonnes)	(C\$x1000)	(C\$x1000)
		Years	(ktonnes)	Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)					
28	1.04	4.49	6,152	1.25	0.39	3.95	\$ 58.83	169	78	782	30,098	4.89	36,250	95,000	82,583
29	1.06	4.56	6,243	1.24	0.39	3.93	\$ 58.55	171	78	789	30,506	4.89	36,749	94,836	82,160
30	1.08	4.59	6,287	1.24	0.39	3.92	\$ 58.42	172	78	792	30,734	4.89	37,021	94,732	81,917
31	1.1	4.65	6,376	1.24	0.39	3.92	\$ 58.44	174	80	803	31,282	4.91	37,658	95,684	82,563
32	1.12	4.69	6,428	1.24	0.39	3.90	\$ 58.30	175	80	806	31,526	4.9	37,954	95,596	82,278
33	1.14	4.75	6,502	1.23	0.39	3.88	\$ 58.04	177	81	811	31,859	4.9	38,361	95,283	81,641
34	1.16	4.81	6,583	1.23	0.39	3.89	\$ 58.11	179	82	822	32,422	4.93	39,005	96,209	82,225
35	1.18	4.83	6,616	1.23	0.39	3.88	\$ 58.02	180	83	826	32,660	4.94	39,276	95,994	81,837
36	1.2	4.88	6,686	1.23	0.39	3.87	\$ 58.00	181	83	833	33,115	4.95	39,800	96,438	81,966
37	1.22	5.12	7,014	1.23	0.39	3.88	\$ 57.94	190	87	876	35,536	5.07	42,549	98,167	81,845
38	1.24	5.13	7,032	1.23	0.39	3.88	\$ 57.88	190	87	877	35,633	5.07	42,665	98,026	81,624
39	1.26	5.17	7,088	1.22	0.38	3.87	\$ 57.68	191	88	881	35,892	5.06	42,981	97,631	80,974

Table 15-6: Ridgetop Pit Optimization Results

Final Pit	Revenue Factor	Mine Life Years	Ore Diluted (ktonnes)	Diluted Grades				Contained Metal			Waste (ktonnes)	Strip Ratio	Total (ktonnes)	NPV Best (C\$x1000)	NPV Worst (C\$x1000)
				Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)					
10	0.68	0.16	174	1.48	0.67	6.09	\$ 71.35	7	5	44	939	4.16	1,164	7,620	7,620
11	0.7	0.19	205	1.45	0.66	5.99	\$ 69.94	8	5	50	1,064	4.12	1,322	8,344	8,344

Final Pit	Revenue Factor	Mine Life Years	Ore Diluted (ktonnes)	Diluted Grades				Contained Metal			Waste (ktonnes)	Strip Ratio	Total (ktonnes)	NPV Best (C\$x1000)	NPV Worst (C\$x1000)
				Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)					
12	0.72	0.20	222	1.44	0.66	5.94	\$ 69.31	9	6	53	1,164	4.19	1,442	8,737	8,737
13	0.74	0.22	247	1.42	0.64	5.79	\$ 68.19	9	6	56	1,247	4.13	1,549	9,159	9,159
14	0.76	0.23	261	1.40	0.63	5.66	\$ 67.30	10	6	57	1,258	4.01	1,571	9,304	9,304
15	0.78	0.25	290	1.36	0.61	5.49	\$ 65.57	10	7	61	1,346	3.9	1,691	9,713	9,713
16	0.8	0.28	322	1.34	0.58	5.27	\$ 64.19	11	7	64	1,462	3.86	1,841	10,104	10,104
17	0.82	0.28	336	1.33	0.58	5.24	\$ 63.74	11	7	65	1,468	3.81	1,854	10,166	10,166
18	0.84	0.32	387	1.29	0.51	4.72	\$ 61.58	13	7	68	1,693	3.8	2,138	10,666	10,666
19	0.86	0.35	428	1.26	0.49	4.59	\$ 60.15	13	8	71	1,785	3.7	2,267	10,926	10,926
20	0.88	1.19	1,451	1.13	0.34	2.92	\$ 52.90	41	18	153	7,370	4.51	9,003	19,218	19,218
21	0.9	1.26	1,561	1.11	0.33	2.87	\$ 52.17	42	18	159	7,565	4.39	9,290	19,638	19,601
22	0.92	1.29	1,638	1.10	0.33	2.84	\$ 51.79	43	19	162	7,653	4.32	9,425	19,800	19,739
23	0.94	1.34	1,735	1.10	0.33	2.83	\$ 51.54	44	19	167	7,934	4.32	9,769	20,000	19,897
24	0.96	1.37	1,812	1.09	0.32	2.80	\$ 51.10	45	19	170	7,996	4.25	9,879	20,071	19,937
25	0.98	1.41	1,893	1.08	0.32	2.78	\$ 50.70	46	20	173	8,115	4.19	10,049	20,103	19,935
26	1	1.47	2,021	1.07	0.31	2.75	\$ 50.19	48	20	179	8,408	4.16	10,429	20,103	19,875
27	1.02	1.53	2,137	1.06	0.31	2.72	\$ 49.72	49	21	183	8,610	4.12	10,702	20,048	19,761
28	1.04	1.56	2,222	1.05	0.31	2.71	\$ 49.39	50	21	186	8,709	4.08	10,845	19,981	19,648
29	1.06	1.60	2,321	1.04	0.30	2.69	\$ 49.01	51	21	190	8,920	4.06	11,117	19,836	19,439
30	1.08	1.66	2,437	1.03	0.30	2.66	\$ 48.55	52	22	194	9,162	4.03	11,435	19,603	19,119
31	1.1	1.68	2,507	1.03	0.30	2.65	\$ 48.37	52	22	196	9,244	4.02	11,545	19,493	18,981

Final Pit	Revenue Factor	Mine Life Years	Ore Diluted (ktonnes)	Diluted Grades				Contained Metal			Waste (ktonnes)	Strip Ratio	Total (ktonnes)	NPV Best (C\$x1000)	NPV Worst (C\$x1000)
				Cu (%)	Au (g/t)	Ag (g/t)	NSR1(C\$/t)	Cu (Mlbs)	Au (koz)	Ag (koz)					
32	1.12	1.70	2,592	1.03	0.30	2.63	\$ 48.16	53	22	198	9,341	4.01	11,672	19,334	18,789
33	1.14	1.75	2,712	1.02	0.29	2.63	\$ 47.75	54	23	204	9,624	4.01	12,027	18,935	18,294
34	1.16	2.33	3,727	1.02	0.34	3.00	\$ 48.07	72	35	308	17,917	5.61	21,108	11,631	9,689
35	1.18	2.37	3,857	1.01	0.34	2.99	\$ 47.85	72	36	311	18,131	5.59	21,372	11,262	9,258
36	1.2	2.39	3,953	1.01	0.34	2.98	\$ 47.76	73	36	314	18,345	5.61	21,617	10,975	8,933
37	1.22	2.40	4,046	1.01	0.34	2.98	\$ 47.69	73	36	315	18,452	5.61	21,743	10,779	8,710
38	1.24	0.16	4,237	1.00	0.34	2.95	\$ 47.43	74	36	319	18,915	5.62	22,277	10,001	7,830

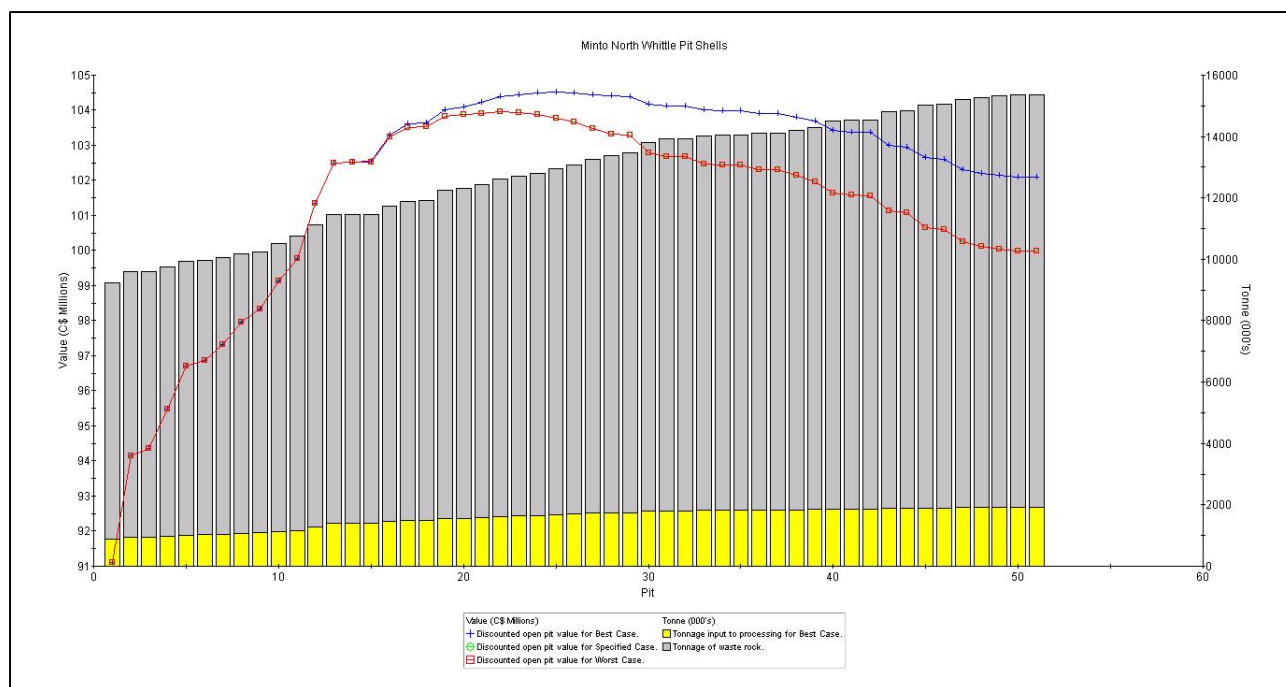


Figure 15—1 North Pit Optimization Results

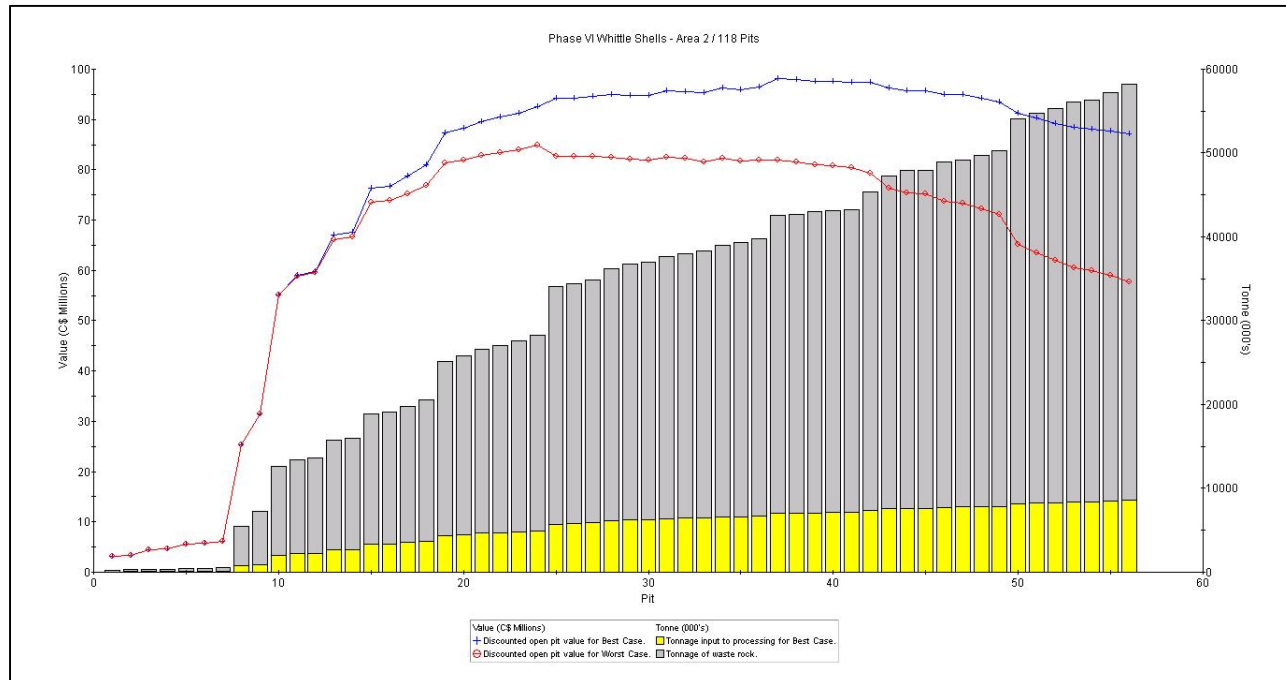


Figure 15—2 Minto South (Area 2/118) Optimization Results

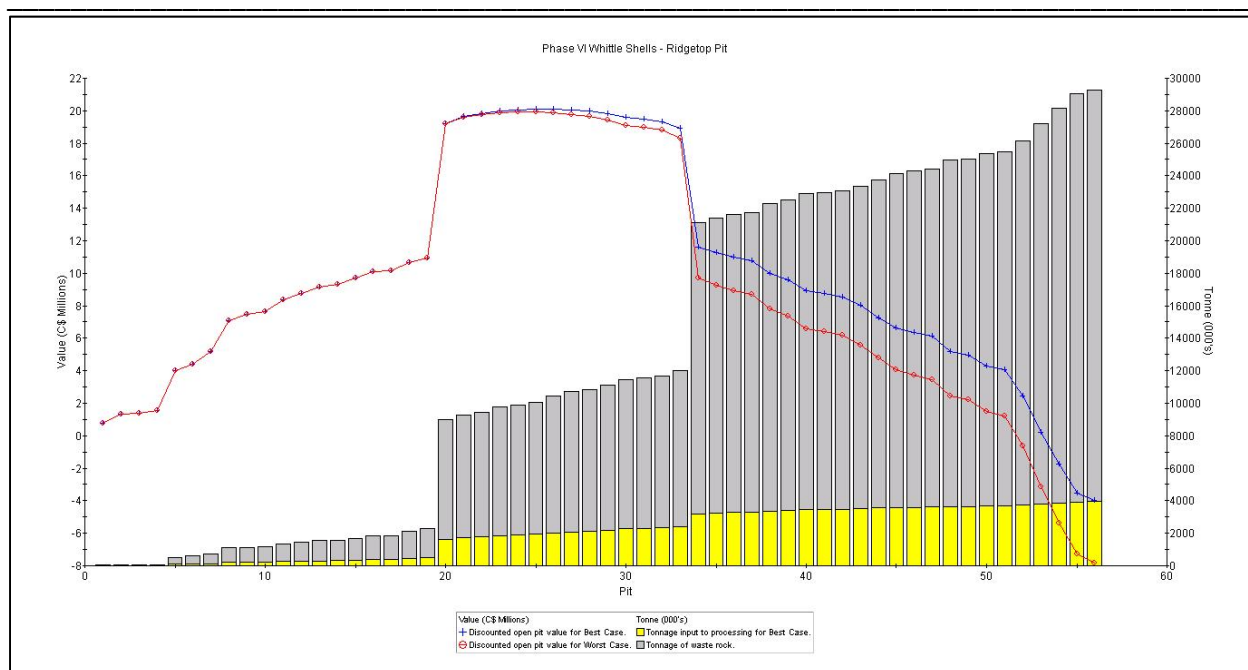


Figure 15—3 Ridgetop Optimization Results

The Whittle™ pit shells were used to develop pit optimized pit designs. Some parameters involved in pit design include inter-ramp angles, access ramp widths, and detailed bench configurations as summarized in Table 15-7 below.

Open pit mining of the Wildfire deposit was investigated by Capstone technical personnel who determined that the optimal method of ore extraction should be based on underground mining methods.

Table 15-7: Detailed Pit Design Parameters

Design Parameter	Unit	Wildfire		North	Area2/118	Ridgetop
Overburden angle	°	30		30	30	30
Material		Conglomerate	Granodiorite			
Inter-ramp angle	°	47	55	52	47 west, 53 east	47 west, 53 east
Ramp width (2-way traffic)	m	24	24	25	25	25
Ramp width (single lane traffic)	m	15	15	15	15	15
Ramp grade	%	10	10	10	10	10
Bench height	m	12	24	9	9	9
Bench face angle	°	63	73	72	74	64
Bench configuration	single/double	Double	Single	double	Double	Double
Berm Width	m	5.1	9.5	8	8	8/4.8

Sub-out maximum depth 6.0 m

15.2.4. Open Pit Mineral Reserves

The mineral reserves estimate for the detailed open pit designs are summarized in Table 15-8 below for both proven and probable reserve classification.

The mineral reserve includes the ore stockpile balance predicted for the beginning of 2012. Minto management has decided to use an overall 0.50% copper cut-off grade for open pit mining.

Table 15-8: Phase VI PFS Open Pit Mineral Reserve Estimates

Deposit	Reserve	Tonnes (Kt)	Diluted grade			Contained Metal		
	Class		(%Cu)	(g/t Au)	(g/t Ag)	Cu (Mlb)	Au (koz)	Ag (koz)
Stockpiles	Proven	852	1.23	0.41	3.71	23	11	102
	Probable							
	Sub-total	852	1.23	0.41	3.71	23	11	102
North	Proven	1523	2.36	1.28	8.55	79	63	419
	Probable	5	2.25	0.81	9.38	0	0	2
	Sub-total	1529	2.36	1.27	8.56	79	63	421
Minto South Deposit (Area 2 & 118)	Proven	3,306	1.44	0.54	4.99	105	57	530
	Probable	2,178	1.06	0.23	2.89	51	16	202
	Sub-total	5,483	1.29	0.41	4.15	156	73	732
Ridgetop	Proven	1,073	1.02	0.25	2.12	24	9	73
	Probable	1,018	1.00	0.28	2.96	22	9	97
	Sub-total	2,091	1.01	0.27	2.53	46	18	170
OP Total	Proven	6,826	1.54	0.63	5.11	231	139	1,122
	Probable	3,204	1.04	0.25	2.92	74	25	301
	Sub-total	10,031	1.38	0.51	4.41	305	164	1,423

Within these detailed pit designs there is a total of 21 Kt of inferred mineral resources at a copper grade of 1.32%. These inferred tonnes were not included in the LOM production plan. There is no certainty that these inferred mineral resources will be converted to the measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

15.3. Underground

There were no revisions to the Phase V Underground Mine Plan. The Phase V reserve underwent a review based on the new NSR parameters and remain economically viable.

Therefore, these mineral reserve were included in the Phase VI mineral reserve estimate without revision.

Table 15-9 highlights the results from the Phase V Underground Mineral Reserve.

Table 15-9: Minto Phase V Underground Mineral Reserve Estimate

Mineralized Zone	Mineral Reserve Class	Tonnes (kt)	Diluted Grade			Contained Metal		
			Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlbs)	Au (koz)	Ag (koz)
Area 118	Probable	764	1.78	0.69	7.17	30.0	17.0	176.1
Area 2	Probable	967	1.73	0.75	6.77	36.8	23.5	210.6
Minto East	Probable	709	2.28	1.04	6.15	35.6	23.8	140.2
Total - Phase V	Probable	2,441	1.90	0.82	6.71	102.4	64.3	526.8

Section 15.3.1 is the evaluation of the MSD block model for the Copper Keel and Wildfire mineralized zones. The following sections provided the methodology of the design.

15.3.1. Net Smelter Model

Calculations were performed on the model in order to determine the net smelter return (“NSR”) of each individual block. The parameters used in the calculations are summarized in Table 15-10.

Table 15-10: NSR Parameters for Phase VI Underground Reserves

Parameter	Unit	UG COG
Revenue, smelting & refining		
Cu price	US\$/lb. Cu	2.50
Au price	US\$/oz Au	300.00
Ag price	US\$/oz Ag	3.90
Exchange rate	US\$/CAD\$	0.90
Recovery to Cu Concentrate		
Cu	%	91
Au	%	70
Ag	%	78
Cu Concentrate Grade		
Cu	%	40
Au	g/t	Variable
Ag	g/t	Variable
Moisture content	%	8

Parameter	Unit	UG COG
Smelter Payables		
Payable metal terms were used as per MRI Trading contract (confidential)		
Off-site costs		
Cu conc treatment	US\$/dmt conc	per MRI contract
Cu refining charge	US\$/lb. pay Cu	per MRI contract
Au refining charge	US\$/oz pay Au	per MRI contract
Ag refining charge	US\$/oz pay Ag	per MRI contract
Transport, marketing, ins., etc.		
Transport, marketing, ins., etc.	US\$/wmt conc	155.98
Transport, marketing, ins., etc.	US\$/dmt conc	169.54
Royalties	%	1.00
Operating Costs		
UG mining cost	C\$/ore tonne	36.00
Processing cost	C\$/milled tonne	16.50
G&A cost	C\$/milled tonne	11.90
Total Site Costs	C\$/milled tonne	64.40
Internal Dilution	%	5

All of the above parameters were used to calculate factors for input to the block model for each contributing metal, as follows:

$$\text{NSR (CAD\$/t)} = \$6.73 \times \text{Au (g/t)} + \$0.07 \times \text{Ag (g/t)} + \$45.29 \times \text{Cu (\%)}$$

Those factors were incorporated into geological block model to estimate the value of each ore block. Capstone Mining Corp. then set the NSR cut-off to \$64.40 for design of stope shapes in the Wildfire and Copper Keel zones. Therefore, the copper cut-off grade utilized for mine planning purposes was 1.3% Cu which assumed a 10% by-product credit.

Isoshells were created using GEMS were used to produce “mineable solids” and estimate tonnes and grade within those envelopes. Only Measured and Indicated Resources were extracted and were evaluated by mine planners.

Isoshells were separated into individual stopes. The stopes were then reviewed to ensure development expenditures were considered when the economic benefit of each stope was determined.

15.3.2. Dilution and Recovery

The Wildfire and Copper Keel underground ore is planned to be extracted with Room and Pillar and post-pillar cut and fill methods.

Internal waste dilution is defined as mineralized material that is below cut-off grade but must be excavated in order to attain ore production. Quite often, a “development cut-off grade” is established which considers the economic benefit of processing this mineralized material since the drilling, blasting and appropriate haulage operating costs must be expended regardless of whether or not it is considered to be “ore”.

External dilution is the mineralized material below cut-off grade that is mixed with the ore and processed. Quite often, it is material along the hanging wall and footwall that has fallen into the ore which was then excavated for processing. This is often referred to as “overbreak”. It also consists of excavating fill that has been placed in a stope for geotechnical support purposes. Dilution material is given zero copper, gold and silver grade to continue a conservative approach.

In addition to dilution, recovery of the ore was also considered in the Phase VI Technical Report, i.e., there is a portion of the ore that is not recovered during underground mining. Failures of the surrounding rock (back, footwall, and hanging wall) can result in excessive dilution that had a detrimental impact on the economic viability of extracting that ore. On occasion, it is necessary to leave a portion of the ore intact as a “pillar” which provides ground support. Finally, if the ore is not drilled and blasted efficiently the lack of fragmentation will have a detrimental impact on recovery since some of the ore will remain on the stope walls.

Table 15-11 summarizes the dilution and recovery associated with the Copper Keel and Wildfire deposits.

Table 15-11: Measured and Indicated Resource at \$64.40 NSR

Area	Stope	Resource (Kt)	Copper (%)	Gold (g/t)	Silver (g/t)	Average Thickness (m)	Recovery (%)	External Dilution (%)
Copper Keel	510	138	2.04	1.26	14.76	11.44	69%	16%
	558	148	1.76	0.86	8.81	10.23	72%	16%
	600	1,338	2.13	0.68	6.71	15.21	75%	10%
	610	28	1.85	0.75	8.58	6.48	90%	8%
	617	80	1.89	0.69	7.30	10.72	80%	10%
	635	82	2.04	0.76	9.49	7.79	83%	10%
Wildfire	800	49	2.08	0.96	4.86	8.78	83%	10%
	760	265	2.04	0.86	7.26	12.24	86%	12%
	750	63	1.94	1.26	9.83	7.80	69%	16%
Total		2,193*	2.06	0.77	7.62	13.19	76%	11%

*Rounded to nearest thousand

External dilution and recovery factors were then applied to estimate the underground mineral reserve for Copper Keel and Wildfire.

The addition of the Copper Keel and Wildfire with the existing Phase V Underground mineral reserve estimate is tabulated in Table 15-12.

Table 15-12: Minto Mineral Reserve Estimates for Underground Mine

Deposit	Reserve Class	Diluted Tonnes (Kt)	Diluted grade			Contained Metal			Source
			Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)	
Area 2 / 118	Proven								Provided in Phase V
	Probable	1,731	1.76	0.74	7.19	67	41	400	
	Total	1,731	1.76	0.74	7.19	67	41	400	
Minto East	Proven								Provided in Phase V
	Probable	709	2.28	1.04	6.15	36	24	140	
	Total	709	2.28	1	6	36	24	140	
Copper Keel	Proven	106	1.74	0.61	6.30	4	2	22	Phase VI Addition
	Probable	1,455	1.81	0.65	6.70	58	30	313	
	Total	1,561	1.81	0.64	6.67	62	32	335	
Wildfire	Proven	301	1.80	0.77	6.06	12	7	59	Phase VI Addition
	Probable	59	1.59	1.00	7.85	2	2	15	
	Total	360	1.76	0.80	6.35	14	9	73	
UG Subtotal	Proven	407	1.78	0.73	6.12	16	9	81	Subtotal
	Probable	3,954	1.87	0.76	6.83	163	97	868	
	Total	4,361	1.86	0.75	6.74	179	106	949	

16. Mining Methods

16.1. Introduction

Minto Mine currently utilizes open pit mining methods to extract the ore for mineral processing. Development of the underground mine is to commence in Q3 2012. The open pit mining operations use a standard truck and shovel methods which are contractor operated.

Underground mining has not yet taken place at Minto but is planned to be developed using ramp system access. Standard underground haulage trucks are planned to be used for ore and waste movement. Underground mining operations will be standard room and pillar (“RAP”) mining methods and Post-Pillar Cut-and-Fill (“PPCF”) near the upper cuts. There is potential to eventually utilize longhole and other bulk mining methods in specific areas once detailed definition exploration drilling is completed.

16.2. Geotechnical Pit Slope Design

SRK (2009) has carried out a pre-feasibility level geotechnical rock mechanics evaluation for pit slopes at the Area 2, Area 118, Ridgetop and North deposit areas. For the purposes of this technical report, SRK (2012) also conducted a pre-feasibility level open pit slope geotechnical evaluation for the Wildfire deposit. These geotechnical evaluations are discussed in the following sections and are based on the following principles:

- Discontinuity orientation and geotechnical logging of core;
- Geomechanical laboratory strength testing and geologic materials characterization;
- Development of geotechnical models to provide bases for excavation stability analyses; and
- Recommendation of optimal pit slope angles and pit architecture for mine design purposes.

As commissioned, the work reported was performed at a pre-feasibility design level.

16.2.1. Geotechnical Data Collection

Geotechnical core logging programs were developed to yield information pertinent to modeling of pit slope stability, such as geologic contacts, profiles of rock strength, and characteristics and frequency of discontinuities.

Geotechnical logging, field point load testing and discontinuity orientation of core recovered from a total of 15 drill holes were conducted between the five deposits; three at Wildfire, seven at Area 2 and 118, three at Ridge Top and two at Minto North. Collar locations and average drill hole plunges and azimuths of the geotechnical drill holes are summarized in

Table 16-1.

Table 16-1: Summary of Drill holes Oriented and Logged for Geotechnical Data

Hole ID	Collar Coordinates			Azimuth (deg)	Inclination (deg)	Length (m)
	Northing (m)	Easting (m)	Elev. (m)			
11SWC875	6943561.1	385500.3	852.8	200	-60	201.2
11SWC876	6943691.0	385668.4	863.5	95	-60	291.1
11SWC878	6943974.5	385583.5	845.5	335	-60	260.6
11SWC880	6944877.4	384749.7	769.0	158	49	260.6
09SWC424	6944462.5	384615.2	876.8	236	-57	325.0
09SWC422	6944276.4	384751.3	893.9	239	-58	280.5
09SWC420	6944390.8	384933.1	861.4	213	-61	376.5
09SWC427	6943813.0	384955.7	890.1	245	-60	175.5
09SWC429	6943654.8	384933.1	916.9	58	-59	199.5
09SWC431	6943632.3	385112.7	889.2	238	-60	150.0
09SWC495	6945925.0	384238.0	951.4	196	-60	153.0
09SWC497	6945953.0	384320.0	940.7	47	-55	141.0
07SWC206	6944784.8	384609.5	822.6	223	-61	155.1
07SWC201	6944506.4	384808.9	861.0	211	-57	243.5
07SWC196	6944640.7	384876.9	832.9	070	-60	249.6

16.2.2. Laboratory Strength Testing

Geomechanical testing was conducted at The University of Arizona Rock Mechanics Laboratory in Tucson, Arizona, and Advanced Terra Testing, Inc. (ATT) in Lakewood, Colorado to determine strength characteristics of the in-situ materials. The overall laboratory program consisted of direct shear, uniaxial and triaxial compressive strength, and direct tensile strength testing and measurement of unit weight and elastic properties. Soil testing of overburden material at Wildfire was also conducted at ATT.

16.2.2.1. Wildfire Investigation

For the Wildfire investigation, a total of 28 laboratory tests were conducted on samples selected to represent the range of the rock conditions observed in the three geotechnical borings. Results of the program are summarized below. The SRK (2012) report should be referenced for additional detail. Laboratory uniaxial axial compressive strength (UCS) test results are summarized in Table 16-2:

Table 16-2: Summary of Laboratory UCS Testing for the Wildfire zone

Hole ID	Sample Depth (m)	UCS (MPa)	Young's Modulus (GPa)	Poisson's Ratio	Unit Wt. (kN/m ³)	Rock Type
11SWC875	68.07	103.45			25.84	Granodiorite
11SWC875	84.70	80.08			25.76	Granodiorite
11SWC875	115.94	71.55			25.67	Granodiorite
11SWC875	145.36	105.07	63.4	0.280	26.42	Granodiorite
11SWC875	186.87	114.63			26.53	Granodiorite
11SWC876	45.66	68.08			25.40	Conglomerate
11SWC876	74.96	109.90			26.69	Granodiorite
11SWC876	44.71	59.40			24.89	Conglomerate
11SWC876	121.29	93.02	57.8	0.261	26.40	Granodiorite
11SWC876	182.61	53.79			27.04	Granodiorite
11SWC876	196.25	59.01			26.89	Granodiorite
11SWC876	247.00	131.36			26.52	Granodiorite
11SWC878	62.35	28.61			24.00	Conglomerate
11SWC878	80.54	38.75			25.17	Conglomerate
11SWC878	111.96	33.27			24.80	Conglomerate
11SWC878	169.90	81.23			26.36	Granodiorite
11SWC878	218.25	45.47			26.33	Granodiorite
11SWC878	125.43	97.28	44.1	0.256	26.48	Granodiorite

Triaxial compressive strength (TCS) testing was conducted on five samples of core, under confining pressures (σ_3) ranging between 6.9 and 20.7 MPa.

Five samples of natural joints encountered in the core were tested using four-point, small-scale direct shear tests to obtain discontinuity shear strength data, resulting in:

- Calculated friction angles (Φ) ranged from 30° to 40°, with a mean value of 36°; and,
- Apparent cohesion values ranging from 7 to 124 kPa, with a mean value of 42 kPa.

Laboratory soil properties for the Wildfire area are summarized in Table 16-3:

Table 16-3: Summary of Laboratory Soil Testing for the Wildfire zone

Hole ID	Depth (m)		Atterberg Limits		Grain Size		Moisture Content (%)	Dry Density (pcf)	Direct Shear		USCS Class.
	From	To	L.L.	P.I.	+4	-200			Phi (deg)	Coh. (kPa)	
11SWC878	20	21	42.3	25.2	7.1	79.9	26.1	-	-	-	CL
11SWC878	24	25	20.9	8.3	33.8	29.4	11.7	-	-	-	SM
11SWC878	33	34	16.4	2.2	-	35.0	16.7	-	-	-	ML
11SWC878	35	36	24.2	10.2	11.1	52.0	9.9	-	-	-	CL
11SWC878	44	45	52.6	33.3	-	92.9	28.2	-	-	-	CH
11SWC878	49	50	61.8	40.0	1.5	94.9	34.3	13.7	14.1	59.9	CH
11SWC878	52	53	66.5	43.5	-	96.2	28.2	-	-	-	CH

16.2.2.2. Phase IV Investigation

A total of 51 laboratory tests were conducted on samples selected to represent the range of the rock conditions observed in the eight 2009 geotechnical borings.

Laboratory uniaxial axial compressive strength (UCS) test results are summarized by geotechnical domain in Table 16-4.

Table 16-4: Summary of Laboratory Testing by Geotechnical Domain

Domain	Hole ID	Depth (m)	UCS (MPa)	E (GPa)	v	BTS (MPa)
A118 Weathered	09SWC424	32.10	88.21	50.5	0.22	
A118 Fresh	09SWC424	89.50	119.56			
A118 Fresh	09SWC424	187.00	150.39			
A118 Fresh	09SWC424	220.30	164.68	66.5	0.30	
A118 Fresh	09SWC424	293.16	156.10			
A118 Fresh	09SWC422	122.67	71.69	49.2	0.21	
A118 Fresh	09SWC422	150.10				10.8
A118 Fresh	09SWC422	179.54	128.30			
A118 Fresh	09SWC422	271.90	149.87			9.36
Area 2 Weathered	09SWC420	38.00	48.94	14.9	0.08	
Area 2 Weathered	09SWC420	77.33	72.30			
Area 2 Fresh	09SWC420	130.84	66.03			
Area 2 Fresh	09SWC420	161.03	104.39	47.3	0.23	7.63
Area 2 Fresh	09SWC420	282.10	102.63			

Domain	Hole ID	Depth (m)	UCS (MPa)	E (GPa)	ν	BTS (MPa)
Area 2 Fresh	09SWC420	361.70	149.58			
Ridgetop Weathered	09SWC427	30.40	63.15			
Ridgetop Weathered	09SWC429	33.00	70.92			
Ridgetop Weathered	09SWC431	37.20	121.20			8.9
Ridgetop Weathered	09SWC431	71.22	131.32	52.5	0.29	
Ridgetop Fresh	09SWC427	91.10	140.72			
Ridgetop Fresh	09SWC427	150.25	153.42			
Ridgetop Fresh	09SWC429	92.70	74.34			
Ridgetop Fresh	09SWC429	150.11	86.71	53.9	0.26	7.2
Ridgetop Fresh	09SWC431	108.35	122.78			
Ridgetop Fresh	09SWC431	138.00	100.70			
Minto North	09SWC495	29.32	172.29			
Minto North	09SWC495	86.34	139.69			
Minto North	09SWC495	124.57	124.68			
Minto North	09SWC497	47.53	157.71			
Minto North	09SWC497	89.15	94.31			
Minto North	09SWC497	129.40	153.60			

Triaxial compressive strength (TCS) testing was conducted on six samples of core, under confining pressures (σ_3) ranging between 6.9 and 20.7 MPa.

Ten samples of naturally-occurring discontinuities encountered in the core were tested using four-point, small-scale direct shear tests to obtain discontinuity shear strength data, resulting in:

- Calculated friction angles (Φ) ranged from 33° to 46°, with a mean of 36°; and,
- Apparent cohesion values ranging from 1 to 22 kPa, with a mean of 10 kPa.

Brazilian tensile strength (BTS) testing was conducted on five samples producing intact tensile strengths ranging from 7.2 to 10.8 MPa, with a mean value of 8.8 MPa.

Prior to actual testing of UCS and TCS core samples, sample dimensions and weights were measured and used to calculate total unit weights for each sample. The combined data set included 36 unit weight measurements ranging from 24.9 to 26.7 kN/m³ with a 26.2 kN/m³ mean value.

16.2.3. Open Pit Mining Geotechnical Domains

16.2.3.1. Wildfire Investigation

Profiles of the rock mass rating, RMR (1989) with depth were created for each of the three geotechnical holes drilled at Wildfire in 2011. Based upon the RMR as well as upon its individual components, available site geology information and laboratory test results, drill cores were divided into geotechnical intervals or domains that are expected to behave uniformly when exposed to open pit excavation-induced stresses. Given the relatively consistent nature of geologic materials at Wildfire, the materials were divided into three individual domains; Soil Overburden, Conglomerate and Granodiorite. Distributions of RMR values are summarized by domain in Table 16-5.

Table 16-5: Summary of Wildfire RMR (1989) Distributions

Lithology	No.	Mean	Std. Dev.	Min	Max
Conglomerate	73	69	9	45	82
Granodiorite	333	74	7	45	85

Soil Overburden

The relatively deep soil overburden cover consists primarily of transported silt and fine sand with occasional coarse sand to gravel, and high plasticity clay above bedrock. The soil is high in organic content and is known to contain permafrost lenses. Previous geotechnical work done by SRK and others have indicated that the material contains discontinuous permafrost down to near the bedrock contact at its deepest portions and is most likely frozen down to the bedrock contact in shallower portions. Ubiquitously, the upper 1 m is “active”, i.e., seasonally freezing and thawing. The maximum anticipated wall height to be exposed in the soil overburden is approximately 70m at the north side of the proposed pit.

Conglomerate

A Cretaceous aged conglomerate of unknown origin exists below the deep soil overburden in some portions of Wildfire. The shape of the conglomerate deposit suggests that it may infill erosional channels in the bedrock below. The conglomerate unit is generally characterized by poorly sorted, angular to well-rounded clasts with a hematite cemented matrix. The clasts range in size from small gravel to large boulders. Laboratory testing and geotechnical logging show that the conglomerate is of relatively lower geomechanical quality than the fresh rock which underlies it. The maximum thickness of conglomerate exposed in pit walls is expected to be approximately 60 m. The conglomerate showed RMR (1989) values ranging between 45 and 83 with an average value of 69.

Granodiorite

In general, the granodiorite at Wildfire is consistently of good geomechanical quality, possessing relatively low fracture frequencies and high intact rock strengths based on geotechnical logging and laboratory test results of the three holes drilled for this investigation.

The granodiorite is generally fresh and exhibits few signs of alteration and weathering. Granodiorite RMR (1989) values ranged between 45 and 85 with an average value of 74.

Although several zones of foliated granodiorite were encountered, those zones exhibited similar intact rock strengths and rock mass properties as did samples of non-foliated granodiorite collected from the same coreholes. The foliated zones are judged to be discontinuous and are not expected to impact overall pit slope stability differently than will the non-foliated zones. Therefore, the foliated and non-foliated granodiorite were grouped together as one engineering lithology, i.e. the Granodiorite domain.

16.2.3.2. Phase IV Investigation

During the Phase IV (SRK 2009) geotechnical evaluation, the basic geotechnical parameters recorded for each core run were applied to the Laubscher (1990) In-situ Rock Mass Rating (IRMR) system, thereby creating a profile of IRMR with depth for each of the eight geotechnical holes drilled for the investigation. Based upon the IRMR as well as upon its individual components, available site geology information and laboratory test results, drill cores were divided into geotechnical intervals or domains that are expected to behave uniformly when exposed to open pit excavation-induced stresses, for each of the deposit areas. Given the relatively consistent nature of geologic materials, the rock mass materials were divided into two basic domains at Area 2, Area 118 and Ridgetop, i.e., weathered and fresh rock. The Minto North rock mass was classified into a single domain. Distributions of RMR values are summarized by domain in Table 16-6.

Table 16-6: Summary of RMR (1990) Distributions

Deposit	Domain	Distribution	Sample No.	Mean	Std. Dev.	Min	Max
Area 2	Weathered	Normal	162	46.4	8.6	18	68
Area 2	Fresh	Normal	409	59.8	9.7	29	82
Ridgetop	Weathered	Normal	225	51.8	12.3	18	84
Ridgetop	Fresh	Logistic	99	51.0	10.1	18	76
North	-	Logistic	172	50.5	10.0	19	82
Area 118	Weathered	Logistic	59	50.8	9.2	21	72
Area 118	Fresh	Logistic	334	58.3	10.8	22	81

The weathered rock domain is typically characterized by relatively higher fracture frequencies, consistently lower intact rock strengths and zones of heavy alteration and oxidation as a result of moderate to heavy surface weathering. Drill core in the weathered zone also consistently shows lower RQD and IRMR values. Consequentially, the weathered bedrock is of significantly lower geomechanical quality than is the fresh rock which underlies it.

In general, the fresh rock is consistently a much more competent rock mass than is the weathered bedrock, possessing relatively lower fracture frequencies and higher intact rock strengths. The fresh rock encountered is relatively massive and exhibits fewer signs of

alteration and weathering when compared to the weathered rock and, consequently, possesses higher overall RQD and IRMR values.

The fresh rock domains contain intermittent zones of weaker material which typically correspond to intervals of increased fracturing, weathering and/or alteration, including minor fault zones and surface weathering. However, such intermittent weaker rock zones represent a relatively small portion of the overall fresh rock domain and are not anticipated to adversely impact the performance of the fresh rock mass. Continuous fault zones that are oriented such that displacement into the pit are kinematically feasible are considered independently from rock mass domains.

Several zones of foliated granodiorite were encountered in the fresh rock, but those zones exhibited similar intact rock strengths and rock mass properties as did samples of non-foliated granodiorite collected from the same coreholes. The foliated zones are judged to be relatively discontinuous and are not expected to impact overall pit slope stability differently than will the non-foliated.

Therefore, the foliated and non-foliated rock was grouped together into their respective weathered or fresh domains.

Area 2

A relatively deep soil overburden deposit exists under the northeast portion of the proposed Area 2 Pit, consisting primarily of transported silt and fine sand with occasional lenses of clay and coarse sand to gravel. The soil is high in organic content and is known to contain permafrost. It appears that the soil has filled a relatively deep erosional feature on the order of 60 to 90 m deep with an invert located between Area 2 and the Main Pit to the north. Previous geotechnical work done by SRK and others have indicated that the material contains permafrost down to near the bedrock contact at its deepest portions and is most likely frozen down to the bedrock contact in shallower portions. Ubiquitously, the upper 1 to 3 m zone is “active”, i.e., seasonally freezing and thawing.

Based on available information from resource and geotechnical drilling, Area 2 is covered with soil overburden ranging from about 5 to 15 m in depth in the southwest portion, with up to 20 to 45 m along much of the north and east walls, and reaching a maximum depth of 70 m at the far north.

While it is possible that the frozen overburden may extend farther south, available information suggests that the overburden at the south and west ends of the proposed Area 2 Pit consists of a thin veneer of organic soil underlain by approximately 5 m to 15 m of completely weathered, in-situ bedrock (granular soil) or residuum.

Based on geotechnical drill hole data, the Area 2 weathered domain is adjudged to extend to depths of approximately 50 to 100 m below the current ground surface.

Area 118

The majority of the proposed Area 118 open pit footprint is covered with up to approximately 5 m of overburden, except in its southwestern portion, where the soil locally deepens to

approximately 16 m. The depth of bedrock weathering at Area 118 is generally to about 30 to 60 m below the current ground surface.

Ridgetop

The western regions of the proposed Ridgetop pits are anticipated to contain 1 to 5 m of soil overburden, deepening to the east to from 5 to 15 m on the east side and with a maximum depth of 21 m at the northeast portion of Ridgetop North and the east portion of Ridgetop South.

The bedrock at Ridgetop is generally weathered to a depth of approximately 45 to 70 m below current ground surface.

Minto North

Due to the relatively shallow depth of the Minto North pit and the presence of multiple structures and weaker zones, there was a less significant distinction between the weathered and fresh rock materials and, consequentially, materials at Minto North were combined together into a single domain for modeling.

16.2.4. Slope Stability Analysis

16.2.4.1. Wildfire Investigation

The following section refers to the geotechnical analysis associated with open pit extraction of the wildfire deposit. Once the geotechnical review was completed, the associated economic analysis determined that the optimal extraction would be with underground mining methods. The underground geotechnical review is discussed in Section 16.3, Underground Geotechnical Investigation.

Overall and interramp slope stability for the Wildfire deposit was modeled with limit equilibrium methods using *Slide* v6.015 and *Phase2*, v8.008 software packages (Rocscience, 2012). Two critical sections were selected for overall slope stability modeling. At the time of the study, mine plans were very preliminary and consequently, the maximum sections of conglomerate and soil overburden were targeted for the two overall slope models and the pit bottom was estimated based on the Whittle™ shell provided. For the overall slope stability modeling, a rock mass shear strength/normal stress relationship was developed for the Conglomerate and Granodiorite domains using the Generalized Hoek-Brown strength model (Hoek, et. al. 2002). For the Hoek-Brown criteria, a Geological Strength Index (GSI), material constant (m_i) and a disturbance factor (D) were selected based on geotechnical logging data, laboratory testing and observations made in the Main and Area 2 open pit excavations Table 16-7.

Table 16-7: Hoek-Brown Parameters for Bedrock

Domain	UCS	GSI	m_i	D	Density
Conglomerate	25	50	18	1.0	26.0
Granodiorite	80	67	29	0.9	26.0

Strengths of the soil overburden were selected based on previous investigations by SRK (2009) and EBA (2011). The overburden soils were subdivided into two material types; the upper, mostly silty and sandy soils and the bottom 5m of highly plastic clay similar to what has been encountered beneath the dry stack tailings facility and in the Main pit south wall. The Mohr-Coulomb parameters used to describe the shear strengths of the overburden soil materials are summarized in Table 16-8.

Table 16-8: Mohr-Coulomb Parameters for Overburden Soils

Domain	Phi	C (kPa)	Density
Upper Soils	33	20	20.0
Bottom 5m	0	60	18.5

No information is currently available for hydrogeologic conditions at Wildfire and consequently, conservatively high groundwater levels were assumed for the slope stability analyses. A target safety factor of 1.3 was used for the overall slope stability modeling.

Discontinuity orientation data from each of the geotechnical coreholes was contoured on equal area percent stereoplots for analysis of structural stability. Upon visual inspection, taking into consideration the directional bias against encountering structures with orientations sub-parallel to the bearing of each corehole, it was determined that the structure orientations observed in drill hole 11SWC875 (east side) were dissimilar to those encountered in holes 11SWC876 and 11SWC878 (west side). The Wildfire area was subsequently divided into two structural sectors under the assumption that a major north-south trending structure likely exists beneath the drainage on the western side of Wildfire that delineates the two sectors. A summary of discontinuity sets derived from each sector is presented in Table 16-9.

Table 16-9: Wildfire Discontinuity Sets

Sector	Set ID	No.	Dip		Dip Direction	
			Mean	Std. dev.	Mean	Std. dev.
East	J1	46	52	10	296	8
	J2	41	73	11	176	8
	J3	19	82	9	133	6
	J4	12	89	5	76	6

Sector	Set ID	No.	Dip		Dip Direction	
West	J1	188	39	14	109	24
	J2	128	44	12	021	22

Backbreak analyses were carried out using stochastic simulations of discontinuity properties (such as orientation, spacing, persistence, and shear strength) to analyze the likelihood for plane shear and wedge type failures to occur in a given bench configuration and orientation. The analyses yielded distributions of achievable bench face angles and catch bench widths. Results indicated that, based on the existing data, achievable bench face angles of approximately 75 degrees should be achievable for both the East and West sectors.

Major geologic structures are those features, such as faults, dikes, shear zones, and contacts that have dimensions on the same order of magnitude as the area being characterized. These structures are treated as individual elements for design purposes, as opposed to joints, which are handled statistically. Very few major structures were encountered in the three drill holes drilled for this investigation. The structure encountered are not anticipated to significantly impact pit slope stability due to their apparent lack of persistence and associated limited degree of rock degradation; however, it is suspected that a major, north-south trending structure may exist beneath the drainage located at the west edge of Wildfire. This is supported by the apparent change in structural domains between the east and west Wildfire geotechnical drill holes; although no recent major structures were identified in core photos evaluated for several resource drill holes drilled in the area of the drainage.

16.2.4.2. Phase IV PFS Investigation

For the (2009) Phase V geotechnical evaluation, the mathematical geotechnical model was input into the commercially available slope stability modeling software package Slide 5.039, developed by Rocscience, Inc. (2003).

Results of slope stability modeling generally indicated probabilities of failure (PoF) ranging from near zero to approximately 5%. It should be noted that while a near zero percent probability of failure does demonstrate a very low likelihood of slope instability; it does not imply that slope instability is impossible; rather, a reported zero probability simply indicates that, for the potential failure surfaces characterized by one of 300 samples drawn from the strength distributions defined, no surfaces had a Factor of Safety (FoS) less than 1.0. Results of the inter-ramp / overall slope stability modeling are summarized in Table 16-10.

Table 16-10: Inter-ramp/Overall Slope Stability Modeling Results

Deposit	Sector	Height (m)	Mean FS	PoF (%)
Area 2	Northeast	130	2.5	1
Area 2	Southwest	214	2.1	3
North	-	130	2.3	0
Ridgetop		130	2.2	2

Given the small size of the proposed Area 118 pit as well as its close proximity and geotechnical similarities to Area 2, additional inter-ramp slope stability modelling was not deemed necessary for Area 118 at the current pre-feasibility level.

Major geologic structures are those features, such as faults, dikes, shear zones, and contacts that have dimensions on the same order of magnitude as the area being characterized. These structures are treated as individual elements for design purposes, as opposed to joints, which are handled statistically.

Several faults or shear zones have been identified in resource and geotechnical drilling at all subject Minto sites. Most of these structures are not, however, anticipated to significantly impact pit slope stability due to their apparent lack of persistence and to the generally limited degree of rock degradation, e.g., highly plastic gouge development, associated with them. However, the potential for one or more major structures to adversely impact stability of the Area 2 west wall has been identified and, as discussed in the SRK recommendations, should be further investigated as the project advances to the feasibility and design levels.

Specifically, both resource and geotechnical drilling in south-western Area 2 suggest the presence of a major fault or faults, potentially striking sub-parallel to the Area 2 Pit west wall, with a moderate to steep northeast dip similar to faults suggested by resource geology in adjacent Area 118. In particular, exploration holes 06SWC082 and 06SWC106 encountered deep brittle structure(s) approximately 279 m and 243 m, respectively, down hole. Similar indications of fault intercepts were not observed in adjacent holes, thereby suggesting a high dip angle for the structure or structures.

Geotechnical drill holes 09SWC420 and 07SWC201 also encountered zones of major rock disturbance at shallower depths that would be consistent with the potential structure(s) and would coincide with the western Area 2 ultimate pit wall.

Minor discontinuities such as joints, foliation and bedding planes, represent an infinite population for practical purposes and, due to sampling limitations, are best modeled with stochastic (probabilistic) techniques.

Both the weathered and fresh rock domains at Minto are characterized by relatively strong intact rock strengths and by very similar discontinuity orientations. As such, pit slope design sectors were delineated based primarily on variations in structural (discontinuity) systems relative to mean pit wall orientations.

Field discontinuity measurements were converted into in-situ orientations and the combined data set of discontinuities was divided into categories of which, given significant persistency, had the potential to create structurally controlled failures. Plane shear and wedge type failures were evaluated for pit sectors assuming an average orientation of the pit walls in each sector.

Preliminary kinematic analyses indicated that the south and west sectors of Area 2, Area 118 and Ridgetop had potential for bench scale instabilities; consequentially, additional, back-break analyses were carried out for those sectors. SRK's back-break analyses use stochastic simulations of discontinuity properties (such as orientation, spacing, persistence, and shear strength) to analyze the likelihood for plane shear and wedge type failures to occur in a given

bench configuration and orientation. The analyses yield a distribution of predicted achievable bench face angles and catch bench widths. The inter-ramp/overall and bench stability analyses together yield an optimized pit slope angle, providing of sufficient rock fall containment.

Results indicated that, based on the existing data, achievable mean bench face angles of approximately 64 degrees should be expected for the south and west sectors of Area 2 and Area 118. Due to the flatter discontinuity dips at Ridgetop relative to the anticipated shear strength of the discontinuities, steeper achievable bench face angles, on the order of 73 degrees, are expected for both Ridgetop pits.

While discontinuity analyses indicate that there is a slight potential for bench scale instability in the southwest section of the Minto North pit, the relatively low probability and the relatively small size of the pit, recommendations for Minto North are based on inter-ramp slope angles alone.

16.2.5. Pit Slope Design Recommendations

For certain geologic environments, the combination of the average anticipated bench face angle and the preferred inter-ramp angle, based on global stability considerations, alone, do not provide a sufficiently wide average catch bench width to efficaciously control rock fall and/or overbank slough accumulation. In such instances, recommended inter-ramp angles are flattened sufficiently to provide adequately wide average catch benches.

16.2.5.1. Wildfire Investigation

Based on the criteria described above, pit slope design recommendations for Wildfire are summarized in Table 16-11.

Table 16-11: Wildfire Pit Slope Design Recommendations

Domain	Max. Slope Height (m)	Max. Inter-ramp Angle (°)	Average Bench Face Angle (°)	Bench Height (m)	Catch Berm Width (m)	Step-out Width* (m)
Soil Overburden	60	10	-	-	-	12
Conglomerate	70	49	64	24	9.3	-
Granodiorite	170	55	73	24	9.3	-

* Where soil overburden depths are anticipated to exceed 7m, a 15 m offset or step-out should be incorporated at, or near, the overburden soil – bedrock contact.

Due to a highly plastic clay layer located at the base of the overburden soils, very similar to the dry stack tailings facility and the Main pit south wall, there is high risk of slope instability in the overburden materials, particularly on the north side and south sides of Wildfire where the overburden soils are deepest. Based on the strengths back analyzed by EBA (2011), a maximum slope of approximately 10 degrees will be required to provide a safety factor of 1.3 or greater.

16.2.5.2. Phase IV PFS Investigation

Based on SRK's experience, inter-ramp/overall slope angles that yield probabilities of failure of up to 30% for slopes with low failure consequences and approximately 5% to 10 % for high failure consequences are appropriate for most open pit mines. Slopes of high failure consequence are generally those slopes that are critical to mine operations, such as those on which major haul roads are established, those providing ingress or egress points to the pit, or those underlying infrastructure such as processing facilities or structures.

Based on the criteria described above, pre-feasibility pit slope design recommendations for each of the Minto Phase V areas are summarized in Table 16-12.

Table 16-12: Summary of Pit Slope Design Recommendations

Deposit Area	Sector	Max. Slope Height (m)	Max Inter-ramp Angle (°)	Bench Face Angle (°)	Bench Height (m)	Berm Width (m)	Step-out Width* (m)
Area 2	Soil Overburden	50	30	30	-	-	15
Area 2	Rock – Northwest and Northeast	170	53	73	18	8	-
Area 2	Rock – South and West	210	47	64	18	8	-
Area 118	Soil Overburden	18	30	30	-	-	15
Area 118	Rock - Northeast	35	53	73	18	8	-
Area 118	Rock - Southwest	36	47	64	18	8	-
Minto North	Soil Overburden	14	30	30	-	-	15
Minto North	Rock	125	52	72	18	8	-
Ridgetop - North	Soil Overburden	13	30	30	-	-	15
Ridgetop - North	Rock	132	53	73	18	8	-
Ridgetop - South	Soil Overburden	19	30	30	-	-	15
Ridgetop - South	Rock	78	53	73	18	8	-

* Where soil overburden depths are anticipated to exceed 7 m, a 15 m offset or step out should be incorporated at, or vertically near, the contact between the overburden and the bedrock.

16.3. Underground Mining Geotechnical Assessment

Underground mining was proposed for deeper zones in Areas 2, 118 and Minto East in the Phase V PFS. Copper Keel and Wildfire will be adopting similar underground mining practices.

SRK completed a rock mass assessment for the Copper Keel and Wildfire underground mining areas. Data sources for the review included geotechnical drill hole data collected in 2007, 2010 and 2011. No underground specific geotechnical drilling was completed in the immediate Copper Keel area for the Phase VI study.

16.3.1. Structural Assessment

A detailed structural assessment has not been completed for any of the Minto South underground mining area. This is a risk that should be considered during mine planning. Based on the review of available core photos, damage zones have been identified within the mining areas. Major structure will need to be assessed and characterized during mine development and delineation drilling to understand the impact these will have on back and pillar stability.

16.3.2. Hydrogeological Assessment

It is understood that a hydrogeological assessment has not been completed to define the potential inflows to the underground workings from large and small-scale structures. Experience in the Minto open pits shows that there are not significant groundwater concerns related to the use of dry explosives or slope stability. The potential for structure to carry water should be investigated during early mine development to confirm that water inflows can be managed in a proactive manner.

16.3.3. Rock Fabric Assessment

Orientation of joints and foliation has been extracted from geotechnical drill holes completed for the Area 2 open pit and underground slope stability studies, and a single 'data bias check' drill hole completed as part of the 2011 geotechnical studies. These orientations could vary across geological structure, however, based on the available information and interpretation, one structural domain has been assumed. Joint sets used for the evaluation are presented in Table 16-13. Joint set data for the Wildfire open pit assessment has been utilized for the underground assessment. A wedge analysis has been completed to understand the potential geometries and sizes of blocks formed during mining.

Table 16-13: Copper Keel Joint Set Summary

Joint Set	Dip	Dip Direction	Description
1	36	91	Strong, shallow dipping joint set
2	50	44	NE dipping set identified in most drill holes. Sub parallel to deposit scale fault
3	51	301	NW dipping joint set identified in pit mapping and verified through core drilling. Sub parallel to deposit scale faulting
4	87	79	Low frequency sub-vertical joint set. Wide scatter of data

16.3.4. Rock Mass Assessment

Rock mass parameters based on Barton's Q system have been adopted for the underground evaluation for use in support stability graphs for underground excavation design. Table 16-14 describes these values.

Table 16-14: Copper Keel and Wildfire Rock Mass Parameters (excludes major structures)

Area	RQD Range	IRS Range	RMR Range	RMR Description	Q Range	Q Description
Copper Keel	60 – 90	75 - 150	58 - 69	Fair-Good	4.6 - 16	Fair-Good
Wildfire	60 – 90	70 – 100	50 – 61	Fair	1.95 – 6.61	Poor – Fair

The Wildfire rock mass is considered to be of lower rock mass quality due to the shallow depth relative to Copper Keel. These rock mass parameters are expected to represent the majority of the ground encountered during mining at each area.

16.3.5. Underground Excavation Design

Excavation design has been completed for man entry spans and pillars on based empirical guidelines adjusted to the anticipated rock mass characteristics at Copper Keel and Wildfire. The rock mass parameters provided in Table 18.12 were used for the evaluations. The following is a summary of the findings:

- Development headings: 5mWx5mH arch back
- Rooms: 10m span limit with pattern support
- Pillars: 5mWx5mH
5mWx >5mH support or fill required

A design span limit of 10m is recommended for mining in areas of good rock mass quality. Without underground exposures an initial conservative extraction ratio of around 75% should be anticipated to account for major structures, adverse small scale structure, or zones of lower rock mass quality. This would meet the 1:1 pillar height criteria. The required ground support systems to increase spans through zones of reduced rock quality should be tested during early mine development.

For most mining areas, superimposed 5m by 5m pillars are planned to limit spans to 10m. These spans would be mined using a cut height of 5m. Pillars over 5m height should be assessed on an individual case basis, and rock support or fill should be utilized.

In thicker parts of the Copper Keel and Wildfire mineralization, waste rock backfill will be used to provide support to tall slender pillars (c.f. bolt and shotcrete to floor). Tall slender pillars will be developed following benching of the floor where orebody thicknesses are greater than 5.0m. As the tops of the pillars will no longer be accessible by mining equipment, fill will be required both for support and access reasons through mined areas.

16.3.6. Ground Support Recommendations

Ground support recommendations are provided in Table 16-15. These support recommendations do not consider weak ground conditions generated by fault structures and highly foliated areas. Additional support will be required in these areas.

Table 16-15: Minimum Support Requirements

Area		Maximum Dimensions (m)	Support Requirements	Comments
Main Infrastructure	Main Decline	5.0Wx5.0H	2.4m threaded rebar (19mm) on a 1.2m diamond spacing (8 or 9 anchors in a ring) Welded wire mesh across back and down walls to 1.8m from floor If required: 25-50mm shotcrete	Shotcrete and spiling may be required where fault zones are intersected. Friction anchors may be substituted when headings are considered to be open for the short term only.
	Intersections	>7.0m dia. Circle	In addition to standard support: 4.0m cable bolts on a 2m spacing throughout intersection	Final support to be installed prior to taking breakaway.
	Alimak Raise	3.0x5.0	1.8m threaded rebar (19mm) on a 1.2m diamond spacing (14 anchors in a ring) Welded wire mesh on all permanent surfaces If required: 25-50mm shotcrete	Raisebore mining is an option. Stability and support should be reviewed once location is selected.
Production Development	Rooms	10.0x5.0	3.0m Swellex on a 1.2m diamond spacing (8 or 9 anchors in a ring) Welded wire mesh across back and down walls to 1.8m from floor Required for fault/broken zones: 25-50mm shotcrete 5m cable bolts on 1.5x1.5m pattern.	It is estimated that 20% of all production areas will require additional support. Spans will likely need to be reduced in localized areas; shotcrete pillars could be considered. Additional longer support may be required to control adverse structure.
	Pillars	5.0x5.0	If required: 2.4m threaded rebar (19mm) installed to floor level Welded wire mesh/straps to floor level 25-50mm shotcrete	It is estimated that 20% of all pillars will require additional support due to adverse structure/broken ground. This can be assessed on an individual basis with good geotechnical mapping.
		5.0x >5.0	Fill room. Or 2.4m threaded rebar (19mm) installed to floor level Welded wire mesh/straps to floor level 25-50mm shotcrete	

16.3.7. Crown Pillar Design – Area 2

A Map3D numerical model has been constructed to understand the interaction between the Area 2 open pit and underground mining areas. The current mine plan has the open pit and underground areas mined concurrently. It is expected that while the underground will not have an impact on the open pit due to limited extraction, the pit may impact underground workings through ground relaxation, rebound, and blast vibrations.

The model input parameters were based on laboratory testing completed for the Phase V PFS study (Table 16-16). The 3D geometries have been simplified based on pillar and span recommendations provided by SRK.

Table 16-16: Map3D material parameters

Material	IRS (MPa)	Hoek-Brown Criterion		Young's Modulus (GPa)	Poisson's Ratio
		mb	s		
Granite	125	6.4	0.0117	30	0.3

Figure 16—1 shows the initial phase of underground mining (10m spans, 5m pillars), followed by extraction of the open pit. In-situ vertical stresses of approximately 7 MPa are increased to greater than 30 MPa through the pillars following mining of panel 201 closest to the pit. Once the open pit is mined out, a significant reduction in vertical stress is observed through the pillars. Loss of confinement could result in damage to ground support, development of loose, and back- break.

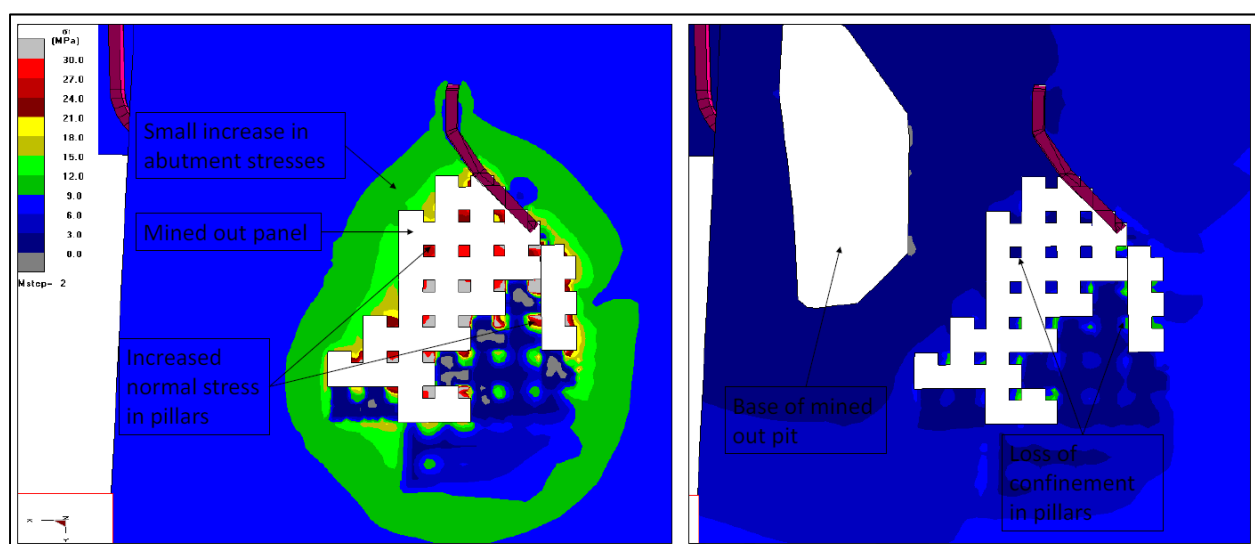


Figure 16—1: Map3D plan stress plan view showing pre (left) and post (right) Area 2 open pit mining. Significant loss of confinement is observed in pillars

It is considered that the main interaction between the pit and underground will be in the stoping closest to the base of the pit. SRK understands that the Area 2 pit and underground mining areas may change. Any changes to the depth of pit and underground mining areas should be reviewed to assess the impacts of stress change as discussed here.

16.4. Underground Mine

16.4.1. Introduction

The known Phase V and Phase VI deposits that have the grade, continuity, and volume that potentially support an underground mining program are Area 118, Area 2, Minto East, Copper Keel and Wildfire.

“An OP/UG cross-over study was done to determine the pit limit for the Area 2 mineralized zone. A 20 m crown pillar was assumed as a barrier between the pit walls, pit bottom and the proposed underground mine” (SRK 2011).

16.4.2. Underground Mining Context

There are well over 30 individual mineral deposits in the Area 2, 118, East, Copper Keel and Wildfire zones. The deposits range from a few hundred tonnes to several hundred thousand tonnes. Each deposit is reviewed in terms of geotechnical constraints, recovery and economic factors to determine the optimal mining method. The general characteristics of the UG Minto deposits are shown in Table 16-17.

Table 16-17: Deposit Characteristics

Parameters	Unit	Value	Comment
Depth below surface	m	100-320	
Dip	deg.	10-30	
Thickness	m	3-25	10 m average
Size (aerial)	m	100x150	Average size
Production Capacity	T / vm	10,000	Approximate tonnes per vertical metre
Mineral Value	\$/t NSR	98	Average value
Mineralization	Mineralized zones are visually and geochemically obvious due to density of visible sulphides and the degree of foliation.		
Continuity	The zones appear to be continuous over tens of metres.		
Regularity	The deposits appear to be well defined zones that are thick in the middle and thin toward the edges with sharp hanging wall and footwall contacts.		
Geotechnical	Generally very favourable rock conditions with strong granitic rock in deposit and in FW and HW. Some faulting but generally not seen to be a significant issue. No anticipated concerns with seismic activity created by mine excavations		
Hydrogeology	Not well defined, but tightness of the rock infers that there will likely not be hydrogeological issues.		
Constraints	There are no known constraints such as heat, radiation, groundwater or rock stress.		

16.4.3. Underground Mining Methods

Underground bulk mining methods continue to be avoided at Minto Mine since the ore in the Copper Keel and Wildfire deposits exhibit a low dip of 20° average that would result in poor extraction due to accumulation of ore on the footwall. In addition, the irregular geometry would result in high dilution if bulk mining methods were pursued. Room and Pillar mining was deemed appropriate for the Copper Keel and Wildfire deposits as it mitigates the risk of dilution and optimizes extraction. Efficient production levels can be attained with the Room and Pillar method provided multiple faces can be established.

Backfill would be provided by depositing waste rock in the stope with a haulage truck equipped with an ejector box and scoop trams. This placement method decreases operating haulage expenditures since the material remains underground rather than hauled to surface.

The SRK geotechnical review of the Wildfire and Copper Keel deposits indicated that the SRK assumptions in the Phase V PFS regarding the Area 2, 118 and Minto East deposits remained

valid with respect to the Copper Keel and Wildfire deposits: *“The strong, massive nature of the Minto rock and shallow depth of the deposits mean that fairly high extraction ratios (plus 75%) would reasonably be expected.”* (SRK 2011)

In areas where the ore thickness is greater than 5m, backfill will have to be placed in mined-out rooms to support pillars prior to taking the next lift. This modification turns the R&P method into a Post Pillar Cut and Fill (“PPCF”) method that would be used in the final lifts of the Wildfire and Copper Keel stopes. Fill for the PPCF method would come from the placement of development waste rock with the scoops and haulage trucks.

16.4.4. Description of Room and Pillar (RAP) Mining

“RAP mining is an open stoping method that utilizes un-mined rock as pillars to support a series of rooms or small stopes around the pillars. The method normally is designed with pillars in a checkerboard pattern. The pillars can be under survey control or done in a more random manner depending on the geotechnical needs. It is usually advantageous to leave lower grade rock in pillars so higher grade material can be mined. Pillars can sometimes be mined on retreat to help improve the overall mining extraction.

The RAP method is normally quite productive, flexible, and requires minimal access development before production starts.

At Minto, many of the mineralized zones are thicker than can be mined in a single pass. In these areas, a hanging wall (HW) cut will be made first, the back supported and then the bottom cut or bench taken out. This sequence enables the back to only be supported (rock bolted) once and would help the overall productivity. A two-boom development jumbo drill would be used for drilling both the initial HW drift and the bench. Based on the thickness of the mineralised zones, an estimate of the percent volume of each deposit that could be potentially benched, as opposed to drift mined, was calculated. Benching is more efficient than drifting and thus has a lower mining cost per tonne.” (SRK 2011).

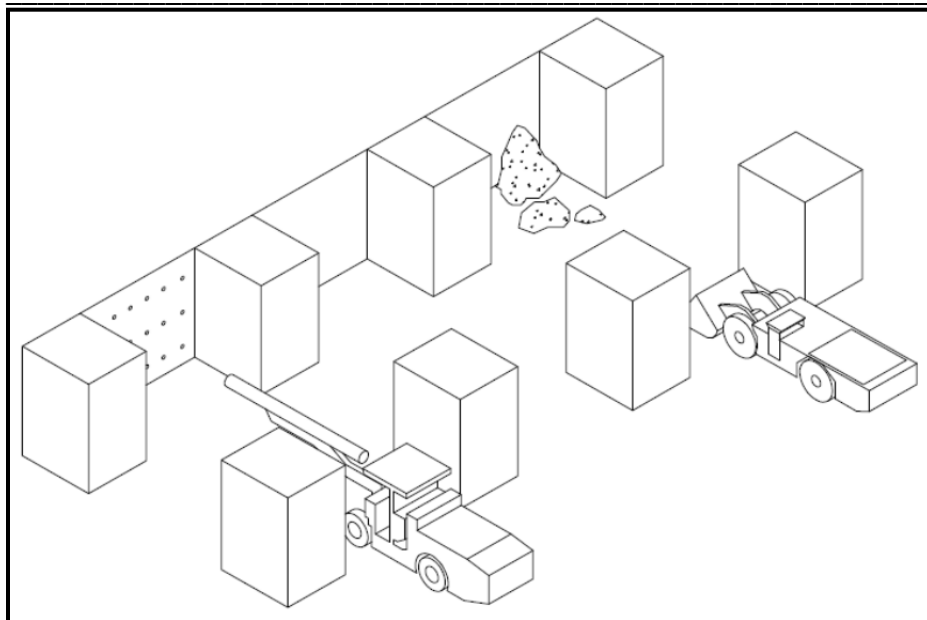


Figure 16—2: Simplified RAP Mining Method Illustration

16.4.5. Description of Post-Pillar Cut and Fill (C&F) Mining

Typically, mechanized cut and fill mining is accessed from sublevels with a crosscut in the footwall. Each crosscut provides access for 6 “cuts” of ore, i.e, a vertical extent of 20 to 30 meters. The access development to the stope “cut” is at a maximum grade of 17 percent. Each “cut” represents a 5 to 6 metre stope lift.

C&F mining can proceed in either an overhand (bottom up) or undercut (top down) manner, often depending on ground conditions. Minto Mine is currently planning on an overhand bottom up approach where the initial access ramp is from the footwall side to the bottom of the mining block. As a stope lift or cut has been completely mined out, the area will be filled with waste rock. There is an opportunity to utilize a hydraulic or pastefill system at a later date pending economic assessments and future exploration.

Once the bottom two cuts have been extracted, subsequent “cuts” will be accessed by back-slashing (breasting) the access ramp to the designed gradient. The majority of waste rock from the back-slash remains as “fill” in the heading. Room and Pillar mining then commences once the modified access ramp has been completed.

The pillars would be designed to be on top of the previous lift pillar which has been supported by confinement from the waste rock fill, rock bolts and on occasion shotcrete. In addition, further support can be provided by concrete posts if required. Instrumentation is often utilized to confirm overall ground stability during the excavation process.

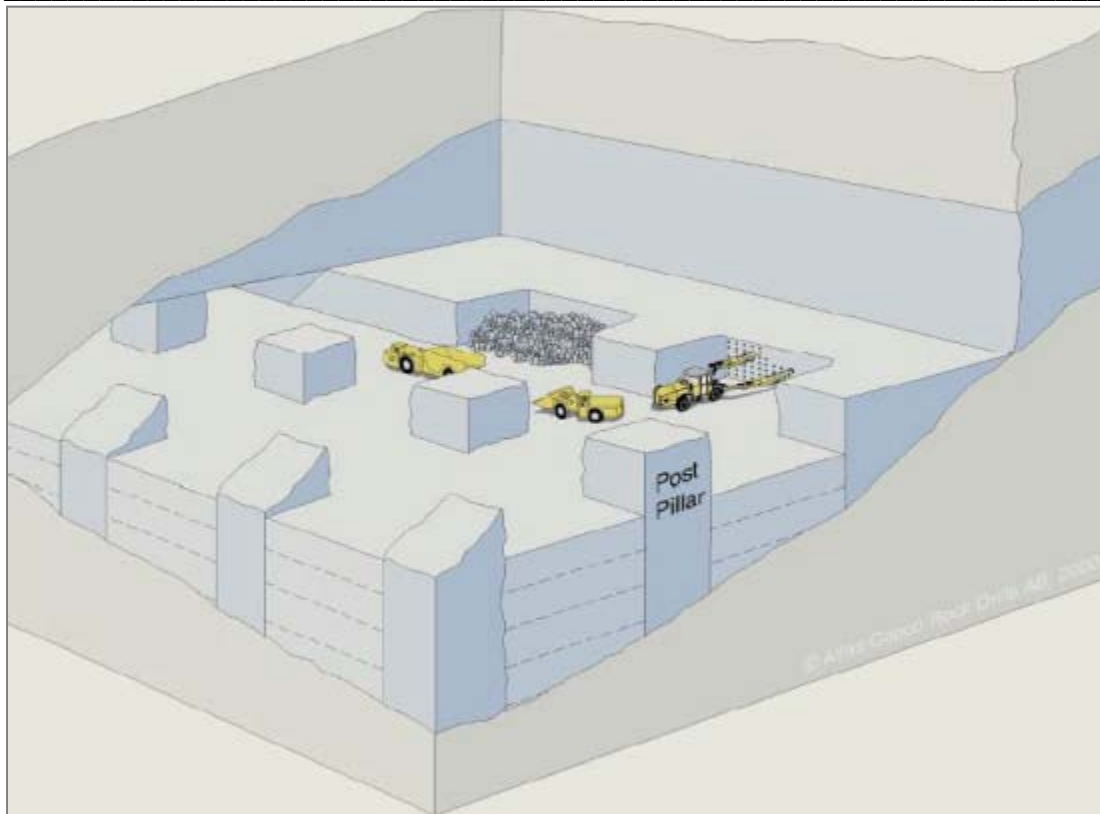


Figure 16—3: Simplified PPCF Mining Method Illustration (from Atlas Copco)

Stoping

All mining is planned to be done with electric-hydraulic two-boom jumbos. Ground support would be done using mechanized rockbolters, and in some cases manually from the blasted rock muck pile or from a scissor lift platform, using jacklegs and stopers. Mucking was planned to be done using diesel LHDs loading underground trucks. The trucks would be loaded near the mining face. Mined-out rooms and cross-cuts were planned to be used as re-muck bays to store ore as needed to keep face advance moving without delay. The loaded haul trucks would transport the ore up to the 15% main access ramp out of the mine to the main mine stockpile area. (SRK 2011)

Ore and Waste Haulage

There have been no modifications with respect to ore and waste haulage since the Phase V PFS.

“A combination of 5.4 m³ (10 t) LHD and 40 t truck with 22 m³ box was selected as the most economical option for ore and waste haulage at Minto underground mine. The equipment cycle times, productivities, mucking and trucking requirements, equipment operating and capital costs were considered to select an optimum equipment combination.

The waste rock from the development headings would be mucked by LHDs directly to the trucks or to remuck bays located up to 150 m from the face. The waste rock would then be hauled by the trucks to the waste dump on surface during the pre-production period.

When underground mine production commences, it would be possible to use mine waste rock from development as stope backfill along with the waste rock from the surface.

The broken ore from the stopes was planned to be mucked by stope LHDs to remuck bays, or loaded directly onto 40 t underground trucks. The trucks would be used to carry ore from the mine directly to the plant, or to a stockpile.” (SRK 2011)

Table 16-18 describes the parameters were used for mucking and haulage productivities and fleet requirements that have been modified to facilitate development and mining of the Copper Keel and Wildfire ore deposits.

Table 16-18: Input Parameters for Mucking and Haulage Estimate

Description	Unit	LHD	Truck
Rock Swell Factor	%	50	50
Capacity	t	10	40
Actual Capacity	t	8.9	35.8
Bucket / Box Size	m ³	5.4	22
Fill Factor	%	90	90
Load / Dump / Manoeuvre	min	1.25	10.2
Speed Loaded (0%)	km/hr	5	15
Speed Empty (0%)	km/hr	8	15
Speed Loaded (+15%)	km/hr	5	6
Speed Empty (+15%)	km/hr	10	10
Speed Loaded (-15%)	km/hr	5	7.5
Speed Empty (-15%)	km/hr	15	15
Availability	%	80	80

The average truck haulage distances from the stope to the ramp portal were estimated using the mine design model to estimate trucking productivities and cost. It was assumed that the trucks would travel approximately 700 m on surface from the portal to the ore stockpile.

The same trucks are scheduled to bring backfill material from the surface to mined-out stopes. Coordination of the trucks would be important to ensure safe and efficient haulage. It is important to note that a 30 tonne haulage truck with an ejector box will be utilized to haul waste fill to the stopes. The ejector box negates the necessity of taking a back slash to raise the box and also eliminates much of the rehandle required by the LHD's.

16.4.5.1. Primary Portal

There have been no modifications to the location or the design of the Primary Portal that was proposed by SRK in the Phase V PFS,

"The main access to the ore body is proposed to be via a single ramp developed at a -15% gradient. A secondary portal is proposed to access the Wildfire deposit in 2018. It will be used for ore and waste haulage, access for personnel, equipment, materials, and services. It would also be utilized as an exhaust airway.

A trackless ramp is considered as the most appropriate mine access method for Minto ore body geometry, location of the production stopes, selected mining method, and mine life. It would provide early access to ore and the ability to start production at earlier stage, reduce initial capital cost, and provide access to the Copper Keel.

The ramp is proposed to be driven on the footwall side of the deposit and would provide multiple accesses to the ore body through the cross-cuts.

The size of the ramp was selected according to the mobile equipment size, required clearances, and ventilation requirements during development and production. It was estimated that a 5.0 m wide by 5.0 m high ramp would be satisfactory for a 40 t truck (and 50-t trucks in the future, if desired) and ventilation requirements for 2,000 t/d production rate. A 25 m ramp curve radius was assumed for convenience to drive a mobile drill jumbo.

Re-muck bays were planned to be developed every 150 m along the ramp to allow efficient use of the drilling equipment and would hold two rounds of development muck. The re-muck bays would be of a similar size as the ramp and would be typically 15 m long. After they are no longer used for development, the bays would be used for equipment storage, pump stations, drill bays, refuge chambers, etc.

Installation of 1.8 m fully grouted resin rebar bolts on the back and the walls of the ramp on 1.2 m x 1.2 m pattern, 100% mesh coverage and an allowance of 50 mm of shotcrete for 5% of the total length of the ramp was assumed for ground support." (SRK 2011)

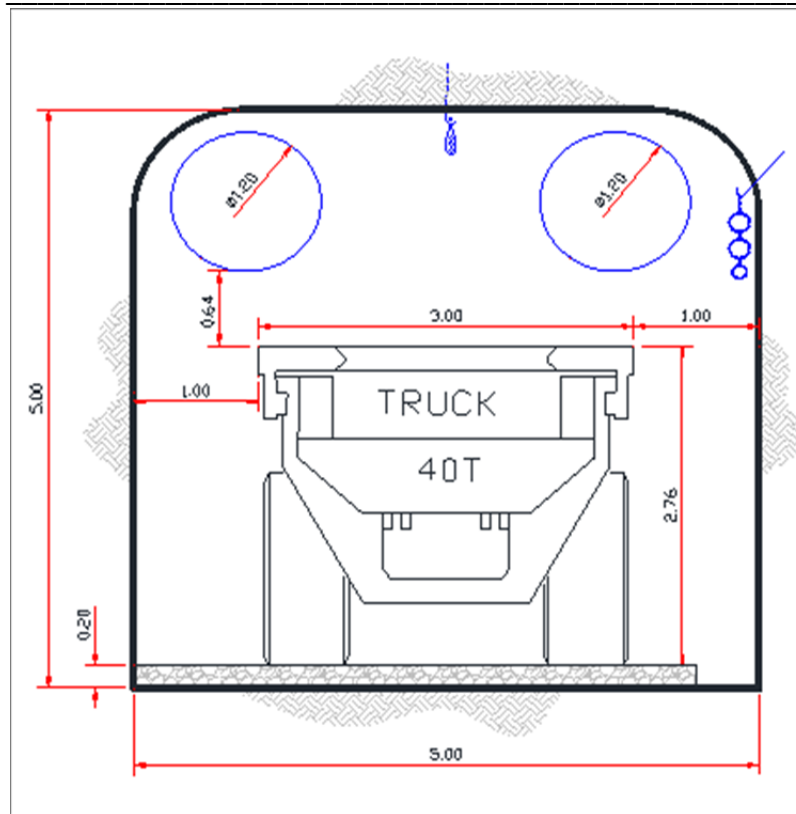


Figure 16—4: Proposed Ramp Cross Section (SRK 2011)

Three ventilation raises with exits to surface are planned at strategic locations off of the main ramps. The ramps and raises will provide a ventilation circuit that enables fresh air to be provided to the work places and an efficient exhaust route back to surface.

A portion of the ventilation raises will include a 1.0 x 3.0 m man-way equipped with ladders and platforms to provide an auxiliary exit from the mine in case of emergency. The raises are planned to be square of a size of 3.0m x 5.0m that includes the man-way.

An intake raise would be developed on the west side of the Area 2 pit with the raise collar in an area of minimal overburden. The top 145 m of the ventilation raise from the ramp elevation of 776 m to surface is planned to be developed using an Alimak. It is planned to extend the intake raise down with the main ramp to the bottom of the 101 stope, which will be in production in year 2012. Below the 776 m ramp elevation, the raise will be mined in four stages between sub-levels. The lower stages are short (from 15 m to 30 m long) and would be developed using drop raises technique or conventional method of raise development.

Ventilation access drifts would be developed to connect the level development and ramp to the ventilation raises. Those drifts would be 15 m to 40 m long and could be developed at 15% gradients to reduce length of the raise.

Two internal ventilation raises will be developed between the stopes in Area 2 to provide flow-through ventilation. All raises will be developed at 70° inclination.

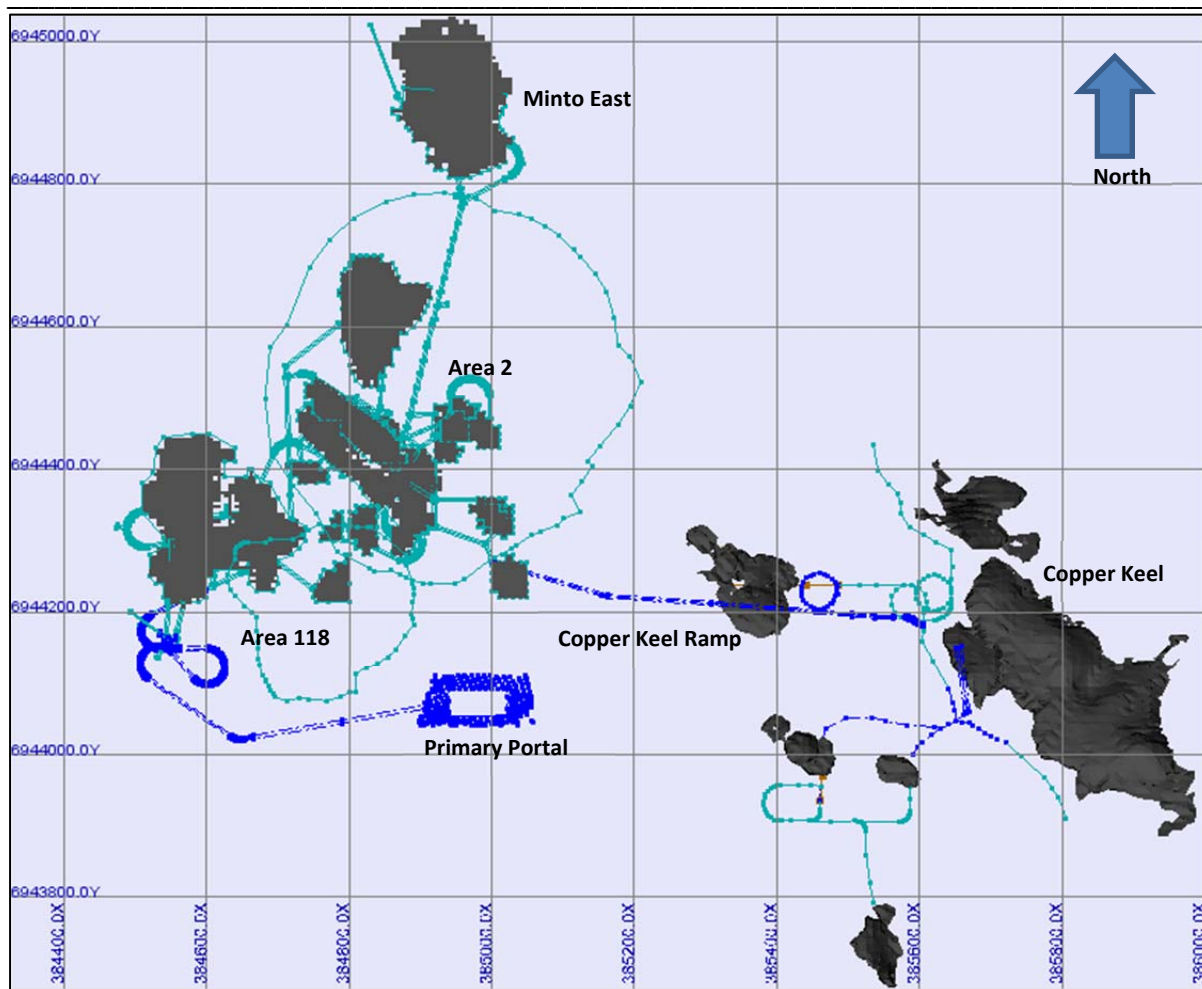


Figure 16—5: Primary Portal Plan View

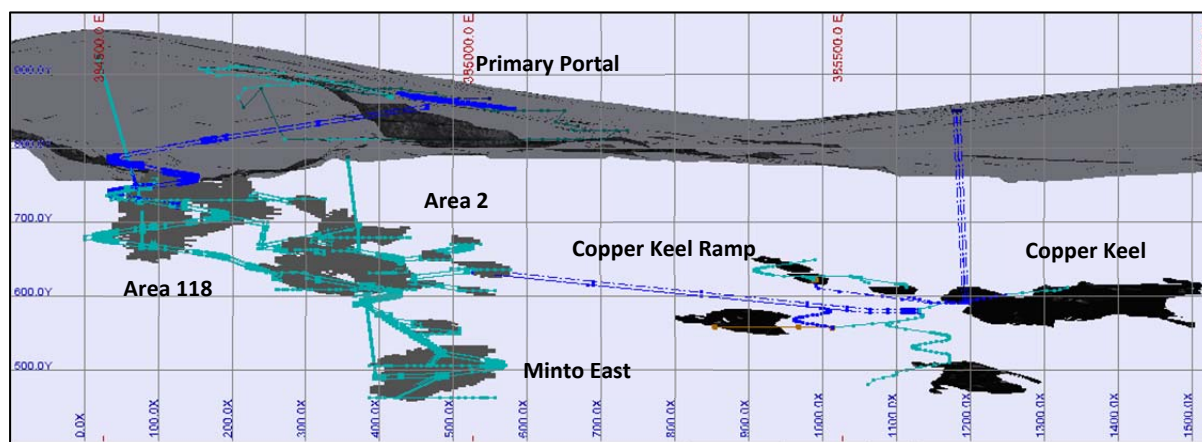


Figure 16—6: Primary Portal Cross Section Looking North

16.4.5.2. Wildfire Portal

A secondary portal is proposed to access the Wildfire deposit in 2018. It will be used for ore and waste haulage, access for personnel, equipment, materials, and services. It would also be utilized as an exhaust airway. The Wildfire portal will serve as access to the Wildfire resource and will remain in production for approximately one year.

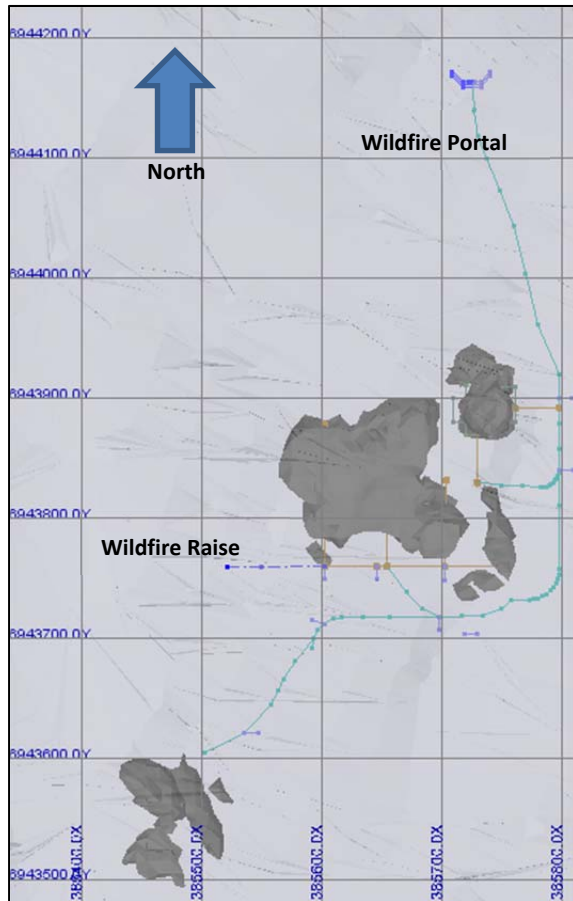


Figure 16—7: Wildfire Portal Plan View

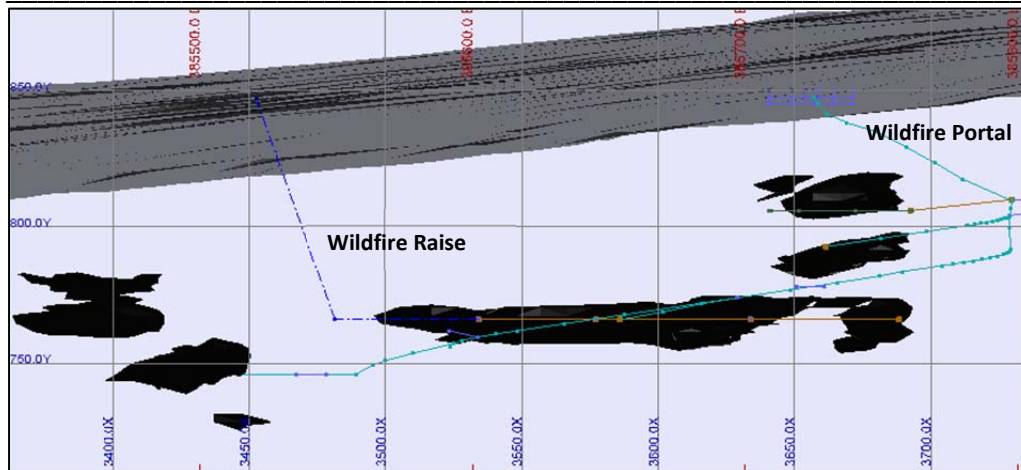


Figure 16—8: Wildfire Access. Cross Section Looking North

16.4.6. Mine Services

16.4.6.1. Ventilation

The design basis of the ventilation system at Minto underground operation is to adequately dilute exhaust gases produced by underground diesel equipment. Air volume was calculated based on a factor of 0.064 m³/s per installed kW of diesel engine power (100 cfm per installed hp). The kW rating of each piece of underground equipment was determined and then utilization factors, representing the diesel equipment in use at any time, applied to estimate the amount of air required.

Air velocity in the main ramp is restricted to a range of 0.25 m/s to 6 m/s. The main intake fan would be installed on surface at the collar of the Area 118 intake ventilation raise and the main exhaust fan at the collar of the Minto East exhaust ventilation raise. Another exhaust fan at the collar of Copper Keel will be installed once mining operations transition to that mine area.

In the first year of production, fresh air is designed to be downcast through the 118 intake ventilation raise, and exhaust up-cast through the ramp. This improves the quality of the ventilation since viceated air from the active stopes would then precede past the haulage trucks on the main ramp and exhausted to surface. Once the Minto East exhaust ventilation raise is developed and equipped with an exhaust fan, about 75% of the total air would be exhausted through that raise and the remaining 25% will be viceated air that proceeds up the ramp. No ventilation doors or regulators would be installed in the main ramp as the exhaust fan will provide an appropriate air distribution between the mining areas. Once Copper Keel exhaust ventilation raise is completed and mine production begins, ventilation doors will be installed at the Minto East and other mined out areas. Air will travel down the Copper Keel ramp to provide appropriate air distribution. Air movement to the stopes would be controlled by directing air flow with ventilation curtains and using the auxiliary ventilation fans. Ventilation regulators, doors, and bulkheads would also be used to control airflow in the mine.

Fresh air ventilation for Wildfire will be provided using ventilation tubing down the ramp initially and through the ventilation raise / escape-way once it is completed

Stage 1 Ventilation

Ventilation of the underground workings will start with the collaring of the portal. At this initial stage ventilation will consist of a fan and vent tube to ventilate the ramp. The 5 m x 5.5 m arched back ramp will be advanced from the surface portal 500 m down ramp to access the planned location of the 118 ventilation raise.

The estimated ventilation requirement for Stage 1 is 26.11 m³/s (55,400 cfm):

Table 16-19: Stage 1 Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
LHD – 7.2 m ³	1	305	100	305	19.52
Rock-Bolter	1	115	20	23	1.47
Jumbo	1	120	10	12	0.77
Sub-Total:				340	21.76
Losses:	20%				4.35
Total Air Requirement:					26.11

A single fan connected to 1.52 m diameter vent tubing is planned. Fan power is estimated to be 44.74 kW at a flow of 26.11 m³/s and pressure of 1.2 kPa.

Stage 2 Ventilation

Once the location of the 118 ventilation raise is reached the ramp will continue to be advanced for a two to three month period while the 118 raise is being driven. Ventilation will be required for advancing the ramp as well as for driving the raise. It is estimated that the ramp will be driven to a total of 800 m from portal by the time the raise is completed.

Table 16-20: Stage 2 Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
LHD – 7.2 m ³	1	305	100	305	19.52
Jumbo	1	120	10	12	0.77
Rock-bolter	1	115	20	23	1.47
Scissor Lift	1	112	25	28	1.79
Explosives Truck	1	95	20	19	1.22
Raise Driving	Allowance				9.44
Sub-Total:				387	34.21
Losses:	30%				10.26

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Total Air Requirement:					44.47

An assumption has been made that muck removal from the mine will continue to be completed by LHD only at this stage. Additional ventilation of 9.44 m³/s has been allotted for the alimak raise driving operation and estimated losses have been increased to account for the longer duct length.

Total ventilation requirement during this stage is estimated to be 44.47 m³/s (94,300 cfm):

Ventilation will continue to be provided with a fan and vent tube. A second 1.52 m diameter vent tube is recommended at the higher flow and length to maintain fan pressure at a reasonable level. Fan power at this stage is estimated to be 75 kW at flow of 44.47 m³/s and pressure of 1.0 kPa.

Stage 3 Ventilation

Stage 3 ventilation will start once the 118 ventilation raise is completed with the main intake fan installed on surface at the top of the raise and will continue until the Minto East ventilation raise is complete. Full mine production starts in the 118 and Area 2 UG areas as the ramp is extended to Minto East.

The 118 ventilation raise will provide fresh air to the mine and will serve as an emergency exit from the mine via the use of a man way. Capstone Mining has indicated a preference to heated air during the winter.

With the 118 ventilation raise supplying air to the mine, exhaust from the mine will exit up-cast through the portal. Total mine ventilation requirement during this stage is estimated to be 176.18 m³/s (373,400 cfm). The ramp and stopes will be ventilated using auxiliary ventilation fans, regulators, doors and bulkheads.

Table 16-21: Stage 3 Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Jumbo	2	120	10	24	1.54
Rockbolter	2	115	20	46	2.94
LHD – 7.2 m ³	2	305	80	488	31.23
LHD - 3.2 m ³	1	150	80	120	7.68
40 t Truck	4	375	80	1200	76.80
Grader	1	149	30	45	2.88
Explosives Truck	1	95	20	19	1.22
ANFO Loader	2	95	30	57	3.65

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Cassette Carrier	2	112	50	112	7.17
Mechanics Truck	1	112	25	28	1.79
Scissor Lift	2	112	25	56	3.58
Supervisor/Engineering Vehicle	3	95	20	57	3.65
Electrician Vehicle	1	95	30	29	1.86
Forklift/Tractor	1	63	20	13	0.83
Sub-Total:				2294	146.82
Losses:	20%				29.36
Total Air Requirement:					176.18

The 118 ventilation raise dimensions are proposed to be 3.0 m x 5.0 m. The 118 supply fan is estimated to require 350 kW of power to provide 176.18 m³/s at a pressure of 1.4 kPa.

It is recommended that the 118 supply fan be selected to provide the flow and pressure required for Stage 4 ventilation and be operated on variable speed control to provide the lower requirements of Stage 3 ventilation.

Stage 4 Ventilation

Stage 4 ventilation will start once the Minto East raise is completed with exhaust fan installed on the surface at the top of the raise. Mine production will come from Minto East, 118 and Area 2 as the ramp to Copper Keel progresses. Stage 4 ventilation will continue until the Copper Keel ventilation raise is completed.

The fresh air supply for the mine will continue to be provided by the 118 Raise with exhaust through both the Minto East raise and through the portal.

Total ventilation requirement during Stage 4 is estimated to be 212.58 m³/s (450,500 cfm). Mine ventilation requirements are estimated to be highest during Stage 4.

Table 16-22: Stage 4 Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Jumbo	3	120	10	36	2.30
Rockbolter	3	115	20	69	4.42
LHD – 3.2 m ³	2	150	80	240	15.36
LHD – 7.2 m ³	2	305	80	488	31.23
40 t Truck	5	375	80	1500	96.00
Grader	1	149	30	45	2.88
Explosives Truck	1	95	20	19	1.22
ANFO Loader	2	95	30	57	3.65
Cassette Carrier	2	112	50	112	7.17
Mechanics Truck	1	112	25	28	1.79
Scissor Lift	2	112	25	56	3.58
Supervisor/Engineering Vehicle	4	95	20	76	4.86
Electrician Vehicle	1	95	30	29	1.86
Forklift/Tractor	1	63	20	13	0.83
Sub-Total:				2768	177.15
Losses:	20%				35.43
Total Air Requirement:					212.58

The intake fan at the 118 raise will not change from that installed during Stage 3 ventilation but will operate at a higher operating point to produce 212.58 m³/s. It is estimated to require 450 kW of power to provide 212.58 m³/s of flow at a pressure of 1.4 kPa.

Upcast exhaust airflow through the portal will be approximately 54 m³/s (114,500 cfm). The other 158.58 m³/s (336,000 cfm) exhaust will be through the Minto East raise.

Minto East raise dimensions are proposed to be 3.0 m x 5.0 m. The Minto East exhaust fan is estimated to require 560 kW of power to provide 158.58 m³/s flow at a pressure of 2.24 kPa. The Minto East exhaust fan is expected to see its highest flow and pressure requirements during Stage 4 ventilation.

Stage 5 Ventilation

Stage 5 ventilation will start once the Copper Keel ventilation raise is completed with exhaust fan installed on surface at the top of the raise. At the beginning of this phase the majority of the

mine production will be in the Minto East area and over time the majority of the mine production will shift to Copper Keel.

The total mine ventilation requirement is expected to decrease to a total of 179.40 m³/s (380,200 cfm).

Table 16-23: Stage 5 Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Jumbo	2	120	10	24	1.54
Rockbolter	3	115	20	69	4.42
LHD – 3.2 m ³	1	150	80	120	7.68
LHD – 7.2 m ³	2	305	80	488	31.23
40 t Truck	4	375	80	1200	76.80
Grader	1	149	30	45	2.88
Explosives Truck	1	95	20	19	1.22
ANFO Loader	2	95	30	57	3.65
Cassette Carrier	2	112	50	112	7.17
Mechanics Truck	1	112	25	28	1.79
Scissor Lift	2	112	25	56	3.58
Supervisor/Engineering Vehicle	4	95	20	76	4.86
Electrician Vehicle	1	95	30	29	1.86
Forklift/Tractor	1	63	20	13	0.83
Sub-Total:				2336	149.50
Losses:	20%				29.90
Total Air Requirement:					179.40

Air supply to the mine will continue to be provided by the 118 ventilation raise. Air will exhaust from the portal, Minto East ventilation raise, and Copper Keel ventilation raise.

The exhaust rate from the Portal is expected to stay consistently near 54 m³/s (114,500 cfm). Exhaust flows from the Minto East and Copper Keel ventilation raises are expected to vary as production in the mine shifts. Once production in Minto East is completed it is expected that the Minto East area will be closed and blocked and all ventilation will exhaust up the portal and from Copper Keel ventilation raise.

The Copper Keel ventilation raise is therefore designed to handle up to 125.4 m³/s (265,800 cfm) and is proposed to be 3.0 m x 5.0 m. The Copper Keel exhaust fan is estimated to require 245 kW of power at a flow of 125.4 m³/s and pressure of 1.4 kPa.

Wildfire Ventilation

Wildfire ventilation will start once the Wildfire portal is completed. Similar in nature to Stage 1, it will utilize a small crew and ventilate down the ramp. It will reach the first stope 270 meters down the ramp where production will begin. A crew will continue to develop the ramp to the 750 level, approximately 640 meters in ramp length. Mining will continue at the 760 and 750 levels in 2019. A fresh air ventilation raise will be built on the 760 level to provide fresh air and an escape way for mine personnel.

The total mine ventilation requirement is expected to be 107.7 m³/s (228,300 cfm).

Table 16-24: Wildfire Ventilation Requirements

Equipment	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
Jumbo	2	120	10	24	1.54
Rockbolter	1	115	20	23	1.47
LHD – 7.2 m ³	2	305	80	488	31.2
40 t Truck	2	375	80	600	38.4
Grader	1	149	30	45	2.88
Explosives Truck	1	95	20	19	1.22
ANFO Loader	1	95	30	29	1.86
Cassette Carrier	1	112	50	56	3.58
Mechanics Truck	1	112	25	28	1.79
Scissor Lift	1	112	25	28	1.79
Supervisor/Engineering Vehicle	1	95	20	19	1.22
Electrician Vehicle	1	95	30	29	1.86
Forklift/Tractor	1	63	20	13	0.83
Sub-Total:				1,401	89.7
Losses:	20%				18
Total Air Requirement:					107.7

Fresh air supply to Wildfire will be provided by one, 150 kW fan located at the Wildfire raise. Air will exhaust up the Wildfire portal.

Ventilation of Headings during Development

“35 m³/s (75,000 cfm) of air is required to dilute and remove exhaust from a 40 t truck, a 5.4 m³ LHD, and a double-boom jumbo working in development heading as outlined in Table 16-25

Table 16-25: Ventilation Requirements for Development Heading

Description	Quantity	Diesel (kW)	Utilization (%)	Utilized (kW)	Air Volume (m ³ /s)
LHD, 5.4 m ³	1	220	100	220	13
Truck, 40 t	1	354	100	354	21
Jumbo, two-boom	1	111	10	11	1
Total				585	35

The requirements for auxiliary ventilation were estimated for the 1,300 m long development heading (toward Minto East), the longest ramp development distance. The auxiliary ventilation fans and ventilation ducts would be used to provide required amount of air at the development face. Only duct resistance was considered to calculate the pressure loss and power requirements as the resistance of the heading is negligible by comparison. Using Atkinson’s equation for air flow in ducts, two 75 kW auxiliary fans with two separate 1.2 m diameter ventilation ducts were selected for the longest distance of ramp development Table 16-26.

Table 16-26: Atkinson Equation for Air Flow in Ventilation Ducts

Duct Diameter (m)	Duct Area (m ²)	Duct Perimeter (m)	Air Volume (m ³ /s)	Duct Air Velocity (m/s)	Friction Factor (kg/m ³)	Duct Length (m)	Pressure Loss (kPa)	Power Required (kW)	Fan Power (kW)
Two Fans and Ducts									
1.0	0.79	3.14	18	22.4	0.003	1,300	7.7	135	181
1.2	1.13	3.77	18	15.5	0.003	1,300	3.1	54	73
1.3	1.33	4.08	18	13.2	0.003	1,300	2.1	36	49
1.4	1.54	4.40	18	11.4	0.003	1,300	1.4	25	34
1.5	1.77	4.71	18	9.9	0.003	1,300	1.0	18	24
Single Fan and Duct									
1.0	0.79	3.14	35	44.7	0.003	1,300	30.9	1,084	1,445
1.2	1.13	3.77	35	31.1	0.003	1,300	12.4	435	581
1.3	1.33	4.08	35	26.5	0.003	1,300	8.3	292	389
1.4	1.54	4.40	35	22.8	0.003	1,300	5.7	201	269
1.5	1.77	4.71	35	19.9	0.003	1,300	4.1	143	190

The 50 kW and 40 kW fans and 1.2 m diameter ducts would be used to provide auxiliary ventilation in other development headings and production stopes.”

16.4.6.2. Mine Air Heating

During the winter months, the mine will utilize a heated ventilation system to prevent water from freezing rather than a brine system as proposed in the Phase V evaluation. The mine air heating system will use propane heaters to heat the intake air. A propane farm will be required will be located on surface.

16.4.6.3. Underground Electrical Power Distribution System

“The major electrical power consumption in the mine would be from the following:

- *Main and auxiliary ventilation fans;*
- *Drilling equipment;*
- *Mine dewatering pumps;*
- *Air compressors; and*
- *Maintenance shop*

High voltage cable would enter the mine via the ramp and be distributed to electrical sub-stations located near production stopes. The power cables would be suspended from the back of development headings. All equipment and cables would be fully protected to prevent electrical hazards to personnel.

High voltage power would be delivered at 4.16 kV and reduced to 600 V at electrical sub-stations. All power would be three-phase. Lighting and convenience receptacles would be single phase 120 V power”. (SRK 2011)

Table 16-27 lists equipment power usage for underground mine at peak production.

Wildfire Electrical Power System

Wildfire portal require its own power distribution system separate from the primary portal. Capital expenditures have been allocated for the set-up, however, electrical units salvageable from different mining area will use if possible. The electrical distribution system will use standard practice as described in the previous subsection. The Wildfire power usage for underground is outlined in .Table 16-28.

Table 16-27: Power System Requirements for Underground Mine at Peak Production

Description	Quantity	Unit (kW)	Load Factor (%)	Utilization (%)	Power Consumption (kW/yr)
Surface					
Shop Equipment	1	50	80	20	70,080
Air Compressor	1	100	80	10	70,080
Lighting	1	10	80	60	42,048

Description	Quantity	Unit (kW)	Load Factor (%)	Utilization (%)	Power Consumption (kW/yr)
Office	1	10	80	80	56,064
Parking Lot	1	10	80	30	21,024
Main Ventilation Fan	1	350	80	100	2,452,800
Pumps	1	10	85	67	49,888
Heat Trace	1	25	80	60	105,120
Underground					
Jumbo, two-boom	3	135	95	60	2,022,246
Rockbolter	2	70	95	60	699,048
Exploration Drill	1	75	95	50	312,075
Portable Compressor	2	100	80	30	420,480
Portable Welder	2	34	80	10	47,654
Exhaust Fan (Minto East or Copper Keel)	1	475	80	100	3,328,800
Auxiliary Fan, 75 kW	2	75	80	90	946,080
Auxiliary Fan, 50 kW	2	50	80	90	630,720
Auxiliary Fan, 40 kW	2	40	80	90	504,576
Refuge Chamber	2	5	80	100	70,080
Main Dewatering Pump	1	100	85	67	498,882
Portable Pump	4	15	85	50	223,380
Sub-total					12,571,126
Miscellaneous Power Allowance	10%				833,129
Total Power					13,828,238

Table 16-28: Power System Requirements for Wildfire Underground

Equipment	Quantity	Unit (kW)	Load Factor	Utilization Factor	Power Consumption kWhr/year
Surface					
Wildfire Ventilation Fan	1	150	80%	100%	1,051,200
Pumps	1	10	85%	67%	49,888
Heat Trace	1	25	80%	60%	105,120
Underground					

Equipment	Quantity	Unit (kW)	Load Factor	Utilization Factor	Power Consumption kWhr/year
2-boom Jumbo	2	135	95%	60%	1,348,164
Rockbolter	1	70	95%	60%	349,524
Portable Compressor	1	100	80%	30%	210,240
Portable Welder	1	34	80%	10%	23,827
Auxiliary Vent Fan, 40 kW	2	40	80%	90%	504,576
Refuge chamber	1	5	80%	100%	35,040
Portable Pump	3	15	85%	50%	167,535
Sub-total					3,845,114
Miscellaneous Power Allowance	0.1				384,511
Total Power					4,229,626

16.4.6.4. Underground Communication System

“A leaky feeder communication system would continue to be utilized in the Wildfire and Copper Keel zones that would be utilized as the communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the electrical sub-stations, refuge stations, and main sump. During 2012, wireless systems on surface to the portals will be reviewed.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will continue to be supplied with an underground radio for contact with the leaky feeder network.” SRK 2011

16.4.6.5. Explosives Storage and Handling

“Explosives required for the development of the Wildfire and Copper Keel zones would continue to be stored on surface in the permanent magazines that are permitted as per the Yukon and Federal regulations. Therefore, detonation supplies (NONEL, electrical caps, detonating cords, etc.) would be stored in a separate magazine. Day boxes would be used as temporary storage for daily explosive consumption.”

Utilization of packaged explosives is to be minimized as much as possible due to the associated increase in expenditures. The Phase VI Pre-feasibility Study includes provisions to enable utilization of bulk emulsions for the mine development and production. This will improve “break”, ground control and efficiency. The mine will also have the capability to utilize Ammonium nitrate (AN) and fuel oil (FO) should this alternative be preferred at a later date. Packaged emulsion may be required as a primer and for loading lifter holes in the development headings.

16.4.6.6. Fuel Storage and Distribution

An average fuel consumption rate of approximately 5,482 l/d is estimated for the period of full production as shown in Table 16-29.

Table 16-29: Underground Mining Fuel Consumption

Description	Quantity	Consumption (l/hr)	Load Factor (%)	Utilization (%)	Total Fuel (l/day)
LHD, 5.4 m ³	3	57.5	75	80	1,768
Truck, 40 t	4	68.9	70	80	2,637
Jumbo, two-boom	3	22	50	10	56
Rockbolter	2	18	50	20	62
Grader	1	36	75	30	138
Explosives Truck	1	27	50	20	46
ANFO Loader	2	22	50	30	113
Cassette Carrier	2	27	70	50	323
Mechanics truck	1	22	50	25	47
Scissor Lift	2	27	50	25	115
Supervisor Vehicle	3	22	50	20	113
Electrician Vehicle	1	22	50	30	56
Forklift	1	16	60	20	33
Total					5,482

Haulage trucks, LHDs, and all auxiliary vehicles would be fuelled at fuel stations on surface. The fuel/lube cassette will be used for the fuelling/lubing of drills and rock bolters.

16.4.6.7. Compressed Air

The underground mobile drilling equipment such as jumbos, rockbolters, and ANFO loaders would be equipped with their own compressors. No reticulated compressed air system was envisioned to be required underground.

Two portable compressors would be required to satisfy compressed air consumption for miscellaneous underground operations, such as: jackleg and stoper drilling, Alimak raise development and pumping with pneumatic pumps. (SRK 2011)

16.4.6.8. Process Water Supply

The majority of water is recycled from the underground workings to the drills, jumbos, and headings where it is utilized for dust control during mucking operations.

16.4.6.9. Mine Dewatering

Water volumes from the Wildfire and Copper Keel underground zones are expected to be low due to the tight nature of the rock. As noted above, it is critical that re-circulation of the water is efficient and filtration quality objectives attained. The preference is to avoid treatment of water at the Water Treatment Plant.

The Copper Keel zone would utilize the main sump at the bottom of the mine near the Minto East ventilation raise. It would be a typical two-bay sump to allow suspended solids to settle out of the water before pumping. Coarse material settled out in the main sump will be removed periodically by LHD and disposed. Old remuck bays would be utilized as temporary sumps during main access ramp development.

“Water was planned to be pumped from the main sump by a high-pressure pump through a 150 mm diameter steel pipe located in the ventilation raise to the final tailing pump box on surface. The sump would be equipped with two high-head submersible pumps – one for operation and one on standby. (SRK 2011)”.

16.4.6.10. Transportation of Personnel and Materials Underground

“All mine supplies and personnel would access the underground via the main access ramp.

Two personnel vehicles would be used to shuttle employees from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors would use diesel-powered trucks as transportation underground. Mechanics and electricians would use the mechanics’ truck and maintenance service vehicles.

A boom truck with a 10-t crane would be used to move supplies, drill parts, and other consumables from surface to active underground workings”. (SRK 2011)

16.4.6.11. Equipment Maintenance

“Mobile underground equipment was envisioned to be maintained in a mechanical shop located on the surface. Some small maintenance and emergency repairs would be performed in a service bay underground. A mechanics truck would be used to perform emergency repairs underground.

A maintenance supervisor would provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. The supervisor would also provide training for the maintenance workforce.

A maintenance planner would schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle. A computerized maintenance system is recommended to facilitate planning.

The equipment operators would provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.” (SRK 2011)

16.4.6.12. Mine Safety

“The portable refuge stations would be provided in the main underground work areas. The refuge chambers are designed to be equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers would be capable of being sealed to prevent the entry of gases. The portable refuge chambers would be move to the new locations as the working areas advance, eliminating the need to construct permanent refuge stations.

Fire extinguishers would be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and other strategic areas. Every vehicle would carry at least one fire extinguisher of adequate size and proper type. It is recommended that underground heavy equipment would be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system would be installed at the main intake raise to alert underground workers in the event of an emergency.

The main access ramp would provide primary access and the ventilation raises with dedicated man way would be equipped with ladders and platforms providing the secondary exit in case of emergency”. (SRK 2011)

16.4.7. Mine Equipment

Criteria used in the selection of underground mining equipment include:

- Mining method;
- Ore body geometry and dimensions;
- Mine production rate;
- Ventilation requirements;
- Operating and capital cost.

Table 16-30 lists underground mobile equipment selected for the 2,000 t/d mine production rate.

Table 16-30: Underground Mobile Equipment List

Equipment	Quantity
Drilling Equipment	
Development / Production Jumbo (2 boom)	3
Rockbolter	2
Loading & Hauling Equipment	
Production / Development LHD, 9.4yd (7.2 tonne)	3
Production / Development LHD, (6.5tonne)	1
Haulage Truck, 40 t	4

Equipment	Quantity
Service Vehicles	
Grader	1
Explosive Truck	1
ANFO Loader	2
Cassette Carrier	2
Personnel Cassette	2
Boom Cassette	1
Fuel / Lube Cassette	1
Mechanics Truck	1
Scissor Lift	1
Supervisor/Engineering Vehicle	4
Electrician Vehicle - Scissor Lift	1
Shotcrete Sprayer	1
Transmixer	1
Forklift / Tractor	2

The equipment list was developed based on the scheduled quantities of work and estimated from first principle cycle times and productivities (83% operational efficiency was used accounting for 50 min of usable time in one operating hour). Some other efficiency factors such as: 80% efficiency for the second boom on the drill jumbo, fill factors for LHD and trucks, additional time for travel, setup and teardown were used in cycle time estimations. The number of operating units was calculated based on 85% shift efficiency (shift change, lunch break, and equipment inspection time were excluded from the shift hours) and then converted to a fleet size by accounting for 80% equipment mechanical availability.

Stationary equipment was selected and would be installed and used for the following:

- Primary and auxiliary ventilation;
- Compressed air;
- Mine water management;
- Underground electrical;
- Communication;
- Mine safety;
- Explosives storage;
- Engineering equipment;
- Miscellaneous.

16.4.8. Personnel

The mining employees at the Minto underground operation were divided into two categories: salaried personnel, and hourly labour.

The personnel requirement estimates were based on the following:

- A 2,000 t/d production rate; and
- A crew rotation of two 11-h shifts per day with two crews working on site and two crews off.

A mining contractor would be used for the raise development and main portal development. Contractor labour and supervisory staff is not included in this section. The labour and personnel requirements described in this section were estimated for the production stage of the mine life. Salaried personnel requirements, including engineering, technical, and supervisory staff, are listed in Table 16-31. Technical staff will be providing services to both open pit and underground services.

Table 16-31: Technical and Supervisory Staff

Quantity	
Staff Mine Operation	
Mine Superintendent	1
Senior Mining Engineer	1
Mine Ventilation/Project Engineer	1
Geotechnical Engineer	1
Geologist	1
Geological Technician	1
Mine Rescue / Safety / Training Officer	2
Surveyor	2
Mine Technical	1
Mine Captain	1
Mine Supervisor / Shift Boss	4
Total Operating Staff	16
Staff Mine Maintenance	
Maintenance Superintendent	1
Maintenance Planner	2
Mechanical / Electrical Foreman	1
Maintenance Supervisor/Shift Boss	2

Quantity	
Total Mine Maintenance Staff	6
Total Mining Staff	22

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation.

Hourly labour requirements at full production are listed in Table 16-32.

Table 16-32: Hourly Labour

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
HOURLY MINE LABOUR			
Production / Development			
Jumbo Operator	3	6	12
Ground Support	4	8	16
Blaster	1	2	4
Haulage			
Scoop-Loader Operator	4	8	16
Truck Drivers	4	8	16
Mine Services & Safety			
General Labourer / Service Crew	1	2	4
Grader Operator	1	1	2
Utility Vehicle Operator/Nipper	1	2	4
General Helper	1	2	4
Sub-total Mine Operating	20	39	78
MINE MAINTENANCE			
Lead Mechanic / Electric	1	1	2
HD Mechanic, mobile	1	2	4
Mechanic, stationary	1	1	2
Electrician	1	2	4
Welder	1	1	2
Tireman / Instrument Man	1	1	2
Mechanic Apprentice	1	1	2

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
Dry / Lampman / Bitman	1	1	2
Sub-total Mine Maintenance	8	10	20
Total Mine Operating	28	49	98

16.5. Open Pit Mine Plan

The open pit mine has changed slightly to the Phase V mine plan. Review of the Phase V mine plan was conducted using a combination of Gemcom GEMS™ and Whittle™ software by Capstone. Minto personnel used Whittle™ economic pit shells to design optimized pit designs using MineSight™. The 3-D mineral block model for Minto North was produced by Kirkham Geosystems Ltd., while the MSD comprising of Area 2, 118, Ridgetop, Copper Keel and Wildfire models were created by SRK. The detailed pit designs and production scheduling was conducted by Minto personnel with the use of GijimaAst™'s Enhanced Production Scheduler ("EPS").

16.5.1. Mine Planning

The 2012 budget was used to determine the starting point of this pre-feasibility study. Based on the review of the Phase V pit designs, revised Phase VI Whittle™ pit shells and preliminary schedules (discussed in Mineral Reserve section of the report). It was determined that the Phase VI Whittle™ pit shells would provide the basis for the detailed ultimate pit designs for Area 2, 118, North and Ridgetop along with the associated pit staging. Waste dumps were taken from the Phase V as no new waste dump designs were required. Pre-stripping in Area 2 since April 2011 has occurred and therefore the ore and waste tonnages already mined are no longer included in these tables.

Table 16-33 below summarizes the detailed pit design tonnages and grades for each of the deposits (using the internal cut-off grade and dilution calculated above). Table 16-34 further summarizes the Minto pits by material types.

Table 16-33: Open Pit Design

Deposit	Diluted Ore (Kt)	Waste (Kt)	Total material (Kt)	Strip ratio (t:t)	Ore grade			Contained Metal		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Stockpile	852		852	NA	1.24	0.41	3.70	23	11	102
Minto North	1,604	9,901	10,983	6.17	2.26	1.21	8.12	80	62	419
Ridgetop	2,091	8,893	11,506	4.25	1.01	0.27	2.53	46	18	170
118	483	1,970	2,462	4.09	1.28	0.10	1.81	14	2	28
Area2	5,000	22,932	27,944	4.59	1.29	0.45	4.38	142	72	704
Subtotal O/P	10,030	43,696	53,727	4.36	1.38	0.51	4.41	305	164	1,423

Table 16-34: Open Pit Material by Type

Pit	Rock (Kt)	Overburden (Kt)	Ore (Kt)	Total Material (Kt)
Main Ore Stockpile	NA	NA	852	852
Minto North	8,981	921	1,604	11,506
Ridgetop	7,596	1,297	2,091	10,984
118	1,617	352	483	2,453
Area 2	16,921	6,011	5,000	27,932
Total	35,115	8,581	10,030	53,727

The open pit mining activities for the Minto pits will continue with current contract mining for this pre-feasibility study rather than an owner operated scenario. This resulted in a lower CAPEX as compared to Phase V and a higher unit operating cost. Operating costs will be reduced since Minto Mine will purchase a waste rock drill in late-2012 and owner-operate them. The owner-operated waste drill will begin operation in early 2013. Capital expenditures have been allocated to complete this task. Technical services and senior management costs were incorporated into the G&A costs. Mining costs were developed from historical expenditures, recent budget forecasts, contract quotes and operational experience.

16.5.2. Grade Control

The Phase VI PFS did not modify the grade control process that was noted by SRK in the Phase V PFS, “In order to minimize ore dilution, maximize ore recovery, and thereby improve the operation’s overall economics, grade control will play an important role throughout the mining process.

Grade control begins with the proper identification of the ore/waste zones and contacts in the field through;

- Information obtained from up-to-date 3-D resource model;
- Blast hole sampling;
- Driller reports;
- Face sampling (includes mapping, visual inspections, sampling); and
- Trenching (as required, to provide better definition of ore/waste contacts, sampling).

Once the above information has been gathered and compiled it will be communicated to operational personnel through;

- Daily/weekly production meetings;
- Detailed “dig” maps – outlining ore zones, waste contacts, faults; and
- Field surveying and layout of dig limits, ore contacts, trenching required.

In order to maintain the effectiveness of the grade control process; regular field inspections will be undertaken by engineering/geology personnel; and clear lines of communication will be

maintained with operational personnel, including equipment operators and front line supervisors.

As part of the grade control process, variable bench heights may be necessary in order to maximize the ore recovery. These include: variable bench heights in waste in order to target the top of the ore zone; and a varying bench height within the ore zones (reduce height at the periphery of the zone). Drill and blast control will also play an important role in order to minimize dilution of the ore zones during the blasting process (e.g. minimize heave in the ore zone)". SRK 2011

16.5.3. Open Pit Mine Equipment

The major mining equipment requirements are indicated in Table 16-35 and are based current practices at Minto. The proposed plant processing rate of 1.4 Mtpa was used to estimate the mining equipment fleet required. The fleet has an estimated maximum capacity of 25,000 tpd total material, which will be sufficient for the proposed LOM plan.

Table 16-35: Open Pit Mine Contractor Equipment

No. of units	Equipment Type
6	Cat 777D Haul Truck
7	CAT 769C Haul Truck
1	Cat 992G Loader
1	Cat 388B Loader
2	CAT 385 Excavator
2	CAT 330 Excavator
1	Cat D9 Dozer
1	Cat 16 Grader
1	Ingersoll-Rand EMC 690
2	Ingersoll-Rand DML 45
1	Cat 777C Water Truck
1	Cat 777B w/trailer

16.5.4. Unit Operations

The I.R. DML45 drill performs the majority of the production drilling in the mine, utilizing a down-the-hole hammer and a 10" diameter bit. Contract waste drills will be replaced with a newly purchased drill that will be operated by mine personnel. The I.R. ECM690 hydraulic drill with a 4" diameter bit is being used for secondary blasting requirements and has potential to drill the tighter spaced patterns required for ore blasts. The main loading and haulage fleet consists of Cat 777D haul trucks, which are loaded with the Cat 992G loader or the Cat 385 excavator, depending on pit conditions. The fleet of Cat 769C haul trucks supplement the main

haulage fleet, if and when necessary. As pit conditions dictate, Cat D11R and Cat D10N dozers are used to rip and push material to the excavators, as well as maintaining the waste dumps.

The additional equipment listed in Table 16-35 are used to maintain and build access roads, along with various site facility requirements, including ore stockpile maintenance, tailings storage construction, maintenance and further exploration development. The work schedule is based on two twelve hour shifts, seven days a week.

Equipment maintenance is being maintained by the current contract mining company at Minto. Open pit mine services such as construction, road maintenance and wall stability is contracted out to the current contract mining company.

16.6. Production Schedule

16.6.1. Mine Sequence and Phasing – Open Pit and Underground

The Phase VI pit designs for the deposits were planned to maximize grade early and at the same time improve pit operations efficiency versus mining of smaller multiple stage shells. As noted by SRK in the Phase V 2011 PFS, this also defers stripping which in turn enhances economic key performance indicators.

The mining sequence is as follows: First two stages of Area 2, 118, Minto North, Area 2 – Stage 3 and end with Ridgetop. Underground production supplements the open pit mine ore feed. The construction of the Main Pit buttress (required for slope stability) is required prior to the placing of tailings in Main Pit.

During the initial pre-stripping of the open pits, the mill feed will be supplemented with stockpiled ore from the Main Pit in order to attain the scheduled mill throughput, while maintaining highest possible copper head grades. Ridgetop has been split into two pits, North and South. Area 2 Pit has been divided into four stages. The underground production will come from Area 2, Area 118, Minto East, Copper Keel and Wildfire. The stage tonnages and associated grades are summarized in Table 16-36 for both OP and UG. Pre-stripping in Area 2 since April 2011 has occurred and therefore the ore and waste tonnage already mined are no longer included in these tables.

Table 16-36: OP, UG and Stockpile Stage Tonnages and Grades

Stage	Stage Quantities									
	Diluted Ore (Kt)	Waste (Kt)	Total material (Kt)	Strip ratio (t:t)	Ore grade			Contained Metal		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
OPEN PIT										
Main Ore Stockpile	852	NA	852	NA	1.24	0.41	3.73	23	11	102
Subtotal Main	852	NA	852	NA	1.24	0.41	3.73	23	11	102
Subtotal Minto North	1,604	9,901	11,506	6.17	2.26	1.21	8.12	79.78	62.2	418.7
Ridgetop South	280	1,563	1,843	5.58	1.29	0.70	6.32	8	6	57

Stage	Stage Quantities									
	Diluted Ore (Kt)	Waste (Kt)	Total material (Kt)	Strip ratio (t:t)	Ore grade			Contained Metal		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Au (koz)	Ag (koz)
Ridgetop North	1,811	7,330	9,141	4.05	0.97	0.20	1.94	38.52	11.6	113.1
Subtotal Ridgetop	2,091	8,893	10,984	4.25	1.01	0.27	2.53	46	18	170
Subtotal 118	483	1,970	2,453	4.09	1.28	0.10	1.81	14	2	28
Area2 - Stage 1	1,577	5,448	7,024	3.46	1.45	0.56	5.20	50.51	28.16	263.50
Area2 - Stage 2	2,069	9,497	11,566	4.59	1.32	0.46	4.32	60.39	30.63	287.05
Area2 - Stage 3	1,355	7,987	9,342	5.90	1.05	0.29	3.52	31.27	12.77	153.41
Subtotal Area 2	5,000	22,932	27,932	4.59	1.29	0.45	4.38	142	72	704
SUBTOTAL O/P	10,030	43,696	53,727	4.36	1.38	0.51	4.41	305	164	1,423
UNDERGROUND										
Area 118	764	326	1,102	NA	1.78	0.69	7.17	30	17	176
Area 2	967	4	967	NA	1.73	0.75	6.77	37	23	211
Minto East	709	30	709	NA	2.28	1.04	6.15	36	24	140
Copper Keel	1,561	283	1,561	NA	1.81	0.64	6.67	62	32	335
Wildfire	360	105	360	NA	1.76	0.8	6.35	14	9	73
SUBTOTAL U/G	4,361	748	5,109	NA	1.86	0.75	6.74	179	106	949
GRANDTOTAL O/P and U/G	14,353	44,484	58,836	NA	1.54	0.60	5.18	484	270	2,372

Proven and Probable reserves only (Inferred included in Waste)

Figure 16—9 further summarizes the stage designs for each of the OP deposits illustrating mineralized rock and waste tonnages, and copper grade. The staging of the figure reflects the open pit mine schedule.

The pit stages were based on the detailed pit designs created. Phase VI OP and UG mining will see the expansion of the existing Main Waste Dump, the Overburden dump, as well as the Valley Fill Dump. Backfilling of Ridgetop and Area 2 pits will also be required as well as the creation of a Mill Valley Dump. All process plant feed rock will be hauled to the appropriate ROM ore stockpiles.

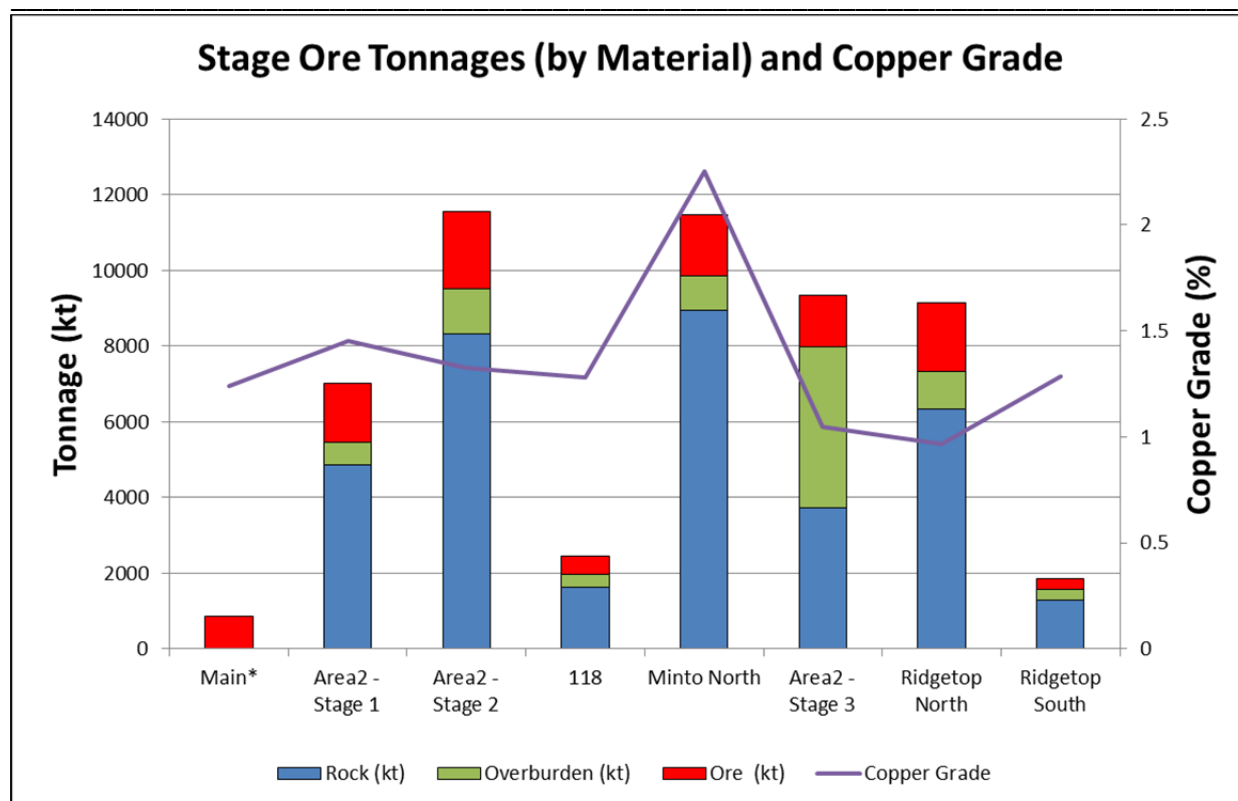


Figure 16—9: Stage Summary

16.6.2. Mine Production Schedule – Open Pit and Underground

The production schedule for the Minto deposits was developed with the aid of GijimaAst™ EPS software, and incorporated the open pit deposits at Main, North, Ridgetop, 118 and Area 2, as well as the underground components of Area 2, 118, Minto East Copper Keel and Wildfire, mentioned above. The process plant throughput will continue at an average rate of 3,750 tpd.

Area 2 pre-striping began in mid-2011. As Area 2 moves into mine production, underground development will begin in mid-2012. Upon approval of the Phase V permit, assumed to be mid-2013, pre-striping will commence in Minto North while mine operation of Area 2 Stage 1 and 2 maintains mill feed until the end of 2013. The mining fleet will concentrate on Minto North in 2014 where it will move towards Area 2 Stage 3 and Ridgetop North in 2015. Production at Ridgetop North will continue until 2017 and Ridgetop South will be extracted in 2017.

The production rate from the Phase VI open pits is approximately 32,141 tpd between 2012 and 2015 and reduces to an average rate of 13,620 tpd between 2016 and 2017. The maximum underground production rate is 2,000 tpd of ore. Only measured and indicated resources were used in the LOM plan.

Table 16-37 summarizes the total material movement by year for the mine production schedule while Table 16-38 summarizes the process schedule.

Table 16-37: Mine Production Schedule

Parameter	Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019
Total OP/UG										
Total Waste	Kt	44,484	9,948	10,302	10,011	7,310	4,917	1,870	125	-
ROM ore	Kt	13,539	2,056	2,456	2,430	2,394	1,948	1,200	696	360
Cu Grade	%Cu	1.59	1.29	1.51	2.07	1.30	1.35	1.55	1.83	1.76
Au Grade	g/t	0.68	0.45	0.50	1.03	0.43	0.46	0.61	0.63	0.80
Ag Grade	g/t	5.50	4.35	4.81	7.38	4.31	3.65	5.93	6.66	6.35
Total Mined Cu	Mlbs Cu	460	59	82	111	69	58	41	28	14
Total Mined Au	koz Au	259	30	39	81	33	29	24	14	9
Total Mined Ag	koz Ag	2256	288	380	576	332	228	229	149	73
ROM ore	Kt/day		5.63	6.73	6.66	6.56	5.34	3.29	1.91	0.99
Total Material	Kt/day		32.89	34.95	34.08	26.59	18.81	8.41	2.25	0.99
Total OP PELLY FLEET only										
Total Waste	Kt	43,677	9,854	10,129	9,890	7,166	4,831	1,806		
ROM ore	Kt	9,178	2,056	2,073	1,697	1,656	1,211	484		
Cu Grade	%Cu	1.39	1.29	1.44	2.18	1.03	0.97	1.18		
Au Grade	g/t	0.61	0.45	0.86	1.15	0.25	0.21	0.55		
Ag Grade	g/t	4.39	4.35	3.97	7.76	3.17	1.96	4.73		
Total Mined Cu	Mlbs Cu	282	59	66	81	38	26	13		
Total Mined Au	koz Au	180	30	57	63	13	8	9		
Total Mined Ag	koz Ag	1,294	288	264	424	169	76	74		
Strip ratio	t:t	4.76	4.79	4.89	5.83	4.33	3.99	3.73		
ROM ore	t/day		5,633	5,679	4,650	4,537	3,318	1,327		
Total Material	t/day		26,998	27,751	27,097	19,634	13,235	4,949		
Total UG only										
Total Waste	Kt	807	94	173	120	144	86	64	125	-

Parameter	Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019
ROM ore	Kt	4,361		383	733	738	736	716	696	360
Cu Grade	%Cu	1.86		1.84	1.83	1.91	1.97	1.80	1.83	1.76
Au Grade	g/t	0.76		0.69	0.76	0.86	0.87	0.66	0.63	0.80
Ag Grade	g/t	6.67		7.23	6.49	6.88	6.42	6.74	6.66	6.35
Total Mined Cu	Mlbs Cu	179		16	30	31	32	28	28	14
Total Mined Au	koz Au	106		9	18	20	21	15	14	9
Total Mined Ag	koz Ag	935		89	153	163	152	155	149	73
ROM ore	t/day			2,100	2,008	2,022	2,018	1,960	1,906	1,970
Total Material	t/day		785	1,524	2,337	2,415	2,254	2,135	2,249	1,970

Table 16-38: Processing Production Schedule – Minto Deposits

Parameter	UNIT	Total	Y E A R										
			2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Mill Feed Rate	dmt/day	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750	3,750
Mill Feed Total	Mt	14,393	1,373	1,369	1,369	1,369	1,369	1,369	1,369	1,369	1,369	1,369	654
Feed Grade	Cu %	1.53	1.50	1.88	2.29	2.19	1.74	1.70	1.70	1.17	0.74	0.74	0.74
	Au g/t	0.58	0.56	0.69	1.15	1.01	0.65	0.66	0.60	0.41	0.16	0.16	0.16
	Ag g/t	5.10	5.18	6.42	8.23	7.89	5.16	6.30	5.90	3.68	1.89	1.89	1.89
Recovery to Conc.	Cu	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	88%
	Au	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
	Ag	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%	78%
Conc. Grade	% Cu		37.0	38.4	41.8	41.2	37.6	37.4	37.4	35.5	32.9	32.9	32.9
Conc. Production	dmt	532,689	51,487	61,600	68,920	66,993	58,332	57,343	57,271	41,340	28,294	28,216	12,894
Conc. Metal	Mlb Cu	444	42	52	63	61	48	47	47	32	21	20	9
	oz Au	188,826	17,311	21,122	35,524	31,230	20,112	20,479	18,449	12,509	4,888	4,875	2,329
	oz Ag	1,840,703	182,790	220,340	282,351	270,763	177,640	216,405	202,366	126,422	65,127	64,949	31,029

With an assumed schedule start date of January 2012, the Minto open pit and underground mine is planned to produce 14.3 Mt of mill feed (includes Main Pit stockpile balance at start of schedule) and 44.5 Mt of waste rock over an 7.5-year mine operating life (yielding an open pit strip ratio of 4.75:1 (t:t)). The mine schedule focuses on achieving the required plant feed production rate, mining of higher grade material early in schedule, incorporating underground production, while balancing grade and strip ratios.

Mill operations continue for an additional 2.5 years, processing the accumulated 2.9 Mt of ore stockpiled when mining ceases, for a total mill operating life of 10 years. The ROM stockpiles are used in the schedule in order to smooth out mill head grades and provide required mill feed Figure 16—10 summarizes mined ore production tonnages, grade by period and area and mill feed grade by period.

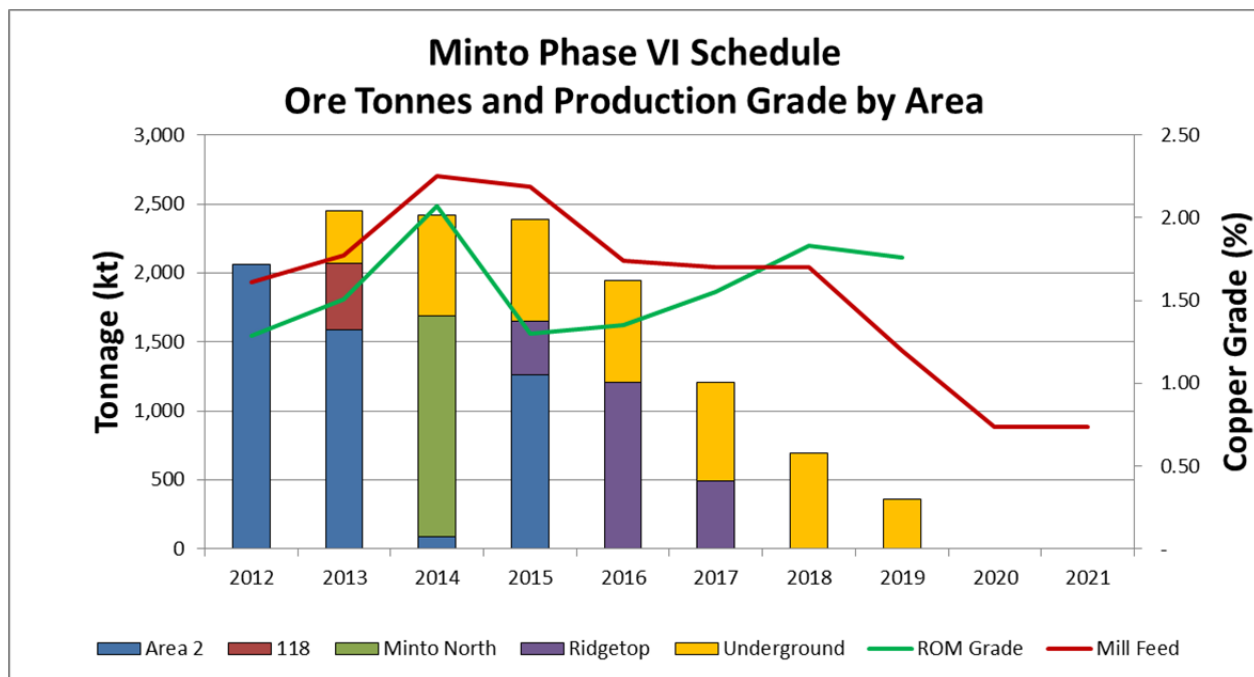


Figure 16—10: Period Ore Tonnage and Copper Grade

To further illustrate the progression of mining of the Minto deposit, Table 16-39 provides the OP and UG mined ore tonnages for each period by stage.

Table 16-39: Mined Ore Tonnage by Stage and Period

Area	Unit	2012	2013	2014	2015	2016	2017	2018	2019
UG	Kt	-	383	733	738	736	716	696	360
Minto North	Kt	-	-	1,600	-	-	-	-	-
Ridgetop North	Kt	-	-	-	390	1,210	210	-	-
Ridgetop South	Kt	-	-	-	-	-	280	-	-
Area 2 - Stage 1	Kt	1,580	-	-	-	-	-	-	-
Area 2 - Stage 2	Kt	480	1,590	-	-	-	-	-	-
Area 2 - Stage 3	Kt	-	-	90	1,260	-	-	-	-
118	Kt	-	480	-	-	-	-	-	-

Pit/Underground Development

2012: Mining of Stage 1 of Area 2 open pit completed and Stage 2 is started. Underground development begins in the summer. Open pit ore production is planned to be 2.06Mt at a strip ratio of 4.79.

2013: Mining of Stage 2 of Area 2 continues. 118 pit is mined. It is assumed that Phase V permit approval will be granted by mid-2013. Minto North mining begins at this time but is mainly waste removal. Underground production reaches steady state of 2,000 ore tpd in the middle of the year. Average total mined grade is 1.51% Cu. Processing rate maintained at 3,750 tpd with a mill head grade of 1.77% Cu.

2014: Minto North begins mining ore production and Area 2 Stage 3 once Minto North is mined out. Underground production ramps up to 2,000 ore tpd. Processing mill head copper grade is 2.25% Cu.

2015: Stage 3 of Area 2 continues production until all resources are mined out. Ridgetop North begins pre-stripping and production in the fall of the year. A total of 2.4 Mt of ore is mined and the mill feed at is 2.19% Cu.

2016: Ridgetop North continues production. Underground mine operations shift to Copper Keel at a steady rate of 2,000 ore tpd. Mill feed is 1.74% Cu.

2017: Ridgetop North production ends at the start of the year. Ridgetop South begins production and is mined out by the end of the year. Open pit mining is concluded. Underground operations continue in Copper Keel at a steady rate of 2,000 ore tpd. Open pit stockpiles and underground feed the mill with a head grade of 1.70% Cu

2018: Copper Keel completes development while development on the Wildfire deposit begins. All remaining tailings produced deposited in Area 2 ultimate pit.

2019: Wildfire is completed and thus completes all mining operations.

2020 - 2022: Stockpiles are fed through the mill until stockpiles are depleted

16.6.3. Ore Stockpiles

Several ore stockpiles exist on the property that will remain active throughout the LOM plan. The stockpiles are defined in terms of estimated copper grade mined as shown in Table 16-40 and their locations are noted on the site plan in the report.

Table 16-40: Ore Stockpiles

Stockpile	Copper grade (%)
Partially Oxidized Ore – Purple	varies
Low grade-Blue	cut-off to 1.0
Regular grade-Green	1.0-2.0
Medium grade-Yellow	2.0-4.0
High grade-Red	greater than 4.0

Table 16-41 and Figure 16—11 detail the various predicted ending stockpile balances on an annual basis. Stockpiles will be depleted on a required basis to supplement open pit and underground ore production in order to ensure maximum mill production. The annual mill feed is summarized in

Table 16-42. Figure 16-11 illustrates the priority placed on providing high grade feed to the mill, i.e, the distinct lack of year end inventory associated with the higher grade ores.

Of note is the large low grade stockpile which reaches a predicted maximum balance of 4.1 Mt. In order to accommodate this large volume a new stockpile location will be required.

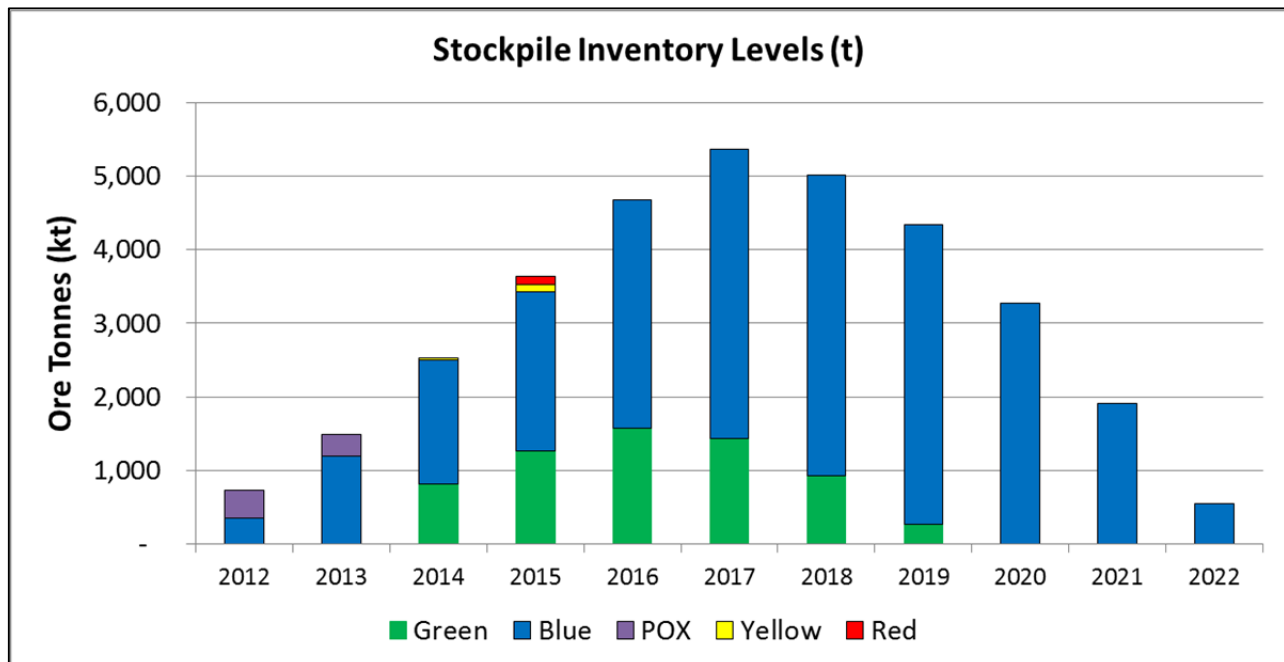


Figure 16—11: Stockpile Inventories

Table 16-41: Ore Stockpile Level – Year End

Stockpile	Unit	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Red (>4%) Stockpile	T				104,469							
	Cu (%)				5.75							
	Au (g/t)				3.26							
	Ag (g/t)				21.95							
Yellow (2-4%) Stockpile	T			35,393	101,568							
	Cu (%)			2.50	2.95							
	Au (g/t)			1.11	2.05							
	Ag (g/t)			9.07	12.32							
Green (1-2%) Stockpile	T			816,627	1,265,908	1,575,587	1,429,665	929,151	257,944			
	Cu (%)			1.55	1.55	1.56	1.57	1.57	1.57			
	Au (g/t)			0.48	0.54	0.55	0.56	0.56	0.56			
	Ag (g/t)			4.72	4.85	5.14	5.05	5.08	5.08			
Blue (COG-1%) Stockpile	T	352,820	1,196,178	1,677,700	2,160,336	3,105,085	3,933,938	4,090,060	4,090,060	3,278,911	1,906,411	537,661
	Cu (%)	0.82	0.75	0.75	0.75	0.74	0.74	0.74	0.74	0.74	0.74	0.74
	Au (g/t)	0.22	0.17	0.16	0.16	0.15	0.15	0.16	0.16	0.16	0.16	0.16
	Ag (g/t)	2.30	1.96	1.90	1.94	1.94	1.86	1.89	1.89	1.89	1.89	1.89
POX Stockpile	T	368,612	297,501									
	Cu (%)	1.72	1.72									
	Au (g/t)	0.62	0.62									
	Ag (g/t)	5.37	5.37									
Total All Stockpiles	T	721,432	1,493,678	2,529,720	3,632,280	4,680,672	5,363,603	5,019,211	4,348,004	3,278,911	1,906,411	537,661
	Cu (%)	1.28	0.94	1.03	1.23	1.02	0.96	0.89	0.79	0.74	0.74	0.74
	Au (g/t)	0.42	0.26	0.27	0.43	0.29	0.26	0.23	0.18	0.16	0.16	0.16
	Ag (g/t)	3.87	2.64	2.91	3.82	3.02	2.71	2.48	2.08	1.89	1.89	1.89

Table 16-42: Ore Stockpile Feed to Mill

Stockpile	Unit	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Red (>4%) Stockpile	T			69,000	127,719							
	Cu (%)			5.75	5.75							
	Au (g/t)			3.26	3.26							
	Ag (g/t)			21.95	21.95							
Yellow (2-4%) Stockpile	t	146,661	102,559	172,235	136,443							
	Cu (%)	2.56	2.50	2.74	2.95							
	Au (g/t)	1.02	1.11	1.59	2.05							
	Ag (g/t)	10.78	9.07	10.69	12.32							
Green (1-2%) Stockpile	t	226,079	32,367	121,717	155,798	346,099	468,980	673,173	314,263			
	Cu (%)	1.46	1.55	1.55	1.56	1.56	1.57	1.57	1.57			
	Au (g/t)	0.53	0.48	0.48	0.55	0.55	0.56	0.56	0.56			
	Ag (g/t)	4.98	4.72	4.72	5.14	5.14	5.08	5.08	5.08			
Blue (COG-1%) Stockpile	t	122,544	6,516						694,899	1,372,500	1,368,750	653,911
	Cu (%)	0.82	0.75						0.74	0.74	0.74	0.74
	Au (g/t)	0.22	0.17						0.16	0.16	0.16	0.16
	Ag (g/t)	2.30	1.96						1.89	1.89	1.89	1.89
POX Stockpile	t	47,699	332,376									
	Cu (%)	1.72	1.72									
	Au (g/t)	0.62	0.62									
	Ag (g/t)	5.37	5.37									
Total All Stockpiles	t	542,983	473,817	362,952	419,960	346,099	468,980	673,173	1,009,162	1,372,500	1,368,750	653,911
	Cu (%)	1.64	1.86	2.91	3.29	1.56	1.57	1.57	1.00	0.74	0.74	0.74
	Au (g/t)	0.60	0.71	1.54	1.86	0.55	0.56	0.56	0.28	0.16	0.16	0.16
	Ag (g/t)	5.98	6.08	10.83	12.59	5.14	5.08	5.08	2.88	1.89	1.89	1.89

16.6.4. Underground Development and Production Schedule

16.6.4.1. Mine Access Development Schedule

The objective of the mine schedule is to maintain high grade feed to the mill that allows for deferral of processing lower grade open pit ores and low grade stockpile ore. Mine development is categorized into two types: Capital and Operating. Capital development occurs before the production rate of 2,000 tpd is sustained for one month. Capital development also includes that development which provides longterm access to a number of deposits and exploration platforms. Operating development is generally that development that provides access to specific ore deposits and occurs after the one month of sustained production at 2,000 tpd has been attained.

Capital development was scheduled to:

- Provide access for trackless equipment;
- Provide ventilation and emergency egress;
- Establish ore and waste handling systems;
- Install mining services.

Initial underground mining (portal and ramp development as well as vertical ventilation raise development), a portion of the lateral development and extraction of 200,000 tonnes of development ore will be carried out by the contractor. The remaining lateral development and mine production will be performed by owner operator crews. The development schedule was planned based on estimated cycle times for jumbo and Alimak development and best practises of North American contractors (Table 16-43).

Table 16-43: Development Cycle Times

Unit	Ramp	Raise	Raise Slash
Design Criteria			
Width (m)	5.0	3.0	3.0
Height (m)	5.0	3.0	5.0
Gradient (% / deg.)	15%	70°	70°
Summary Cycle Times			
Drilling (Hrs)	4.2	4.7	2.7
Blasting (Hrs)	1.8	2.0	1.7
Re-Entry (Hrs)	0.5	0.5	0.5
Mucking (Hrs)	2.0	1.3	0.7
Support (Hrs)	5.3	5.2	3.6
Services (Hrs)	1.0	1.2	0.0
Secondary Mucking (Hrs)	5.3	0.0	0.0
Trucking (Hrs)	5.9	1.9	1.4

Unit	Ramp	Raise	Raise Slash
Single Heading			
Critical Path Cycle Time (Hrs)	14.8	13.5	8.4
Advance Per Shift (m)	2.7	2.0	3.3
Advance Per Day (m)	5.3	4.0	6.5

The raise would be developed in two stages with initial development size of 3.0 m x 3.0 m and then slashed to full raise size of 3.0 m x 5.0 m. The raise would then be equipped with ladders and platforms to provide a man way for secondary egress from the mine in case of emergency.

It is assumed that approximately one month would be required for mobilization of mining equipment and crews to the site and establish the required services. Another month would be required to develop a portal box-cut; the jumbo crew would then start developing ramp from the portal. It is assumed that in first month the ramp advance rate would be at 50% of average or 75 m.

It is assumed that the advance rate of the main ramp development would be approximately 150 m/month per single heading. When multiple headings are available, the advance rate per jumbo crew would increase to 220 m to 250 m per month.

It is planned that the main ramp would be developed all the way to the bottom of the mine to the Minto East area at the maximum advance rate to provide an opportunity to start production of the high grade ore from Minto East as soon as possible. In 2015, the underground operations will begin development towards the Copper Keel zone. Once the ramp is established, mining will begin to shift to Copper Keel until it reaches full production in 2017.

A new portal will be established for the Wildfire ramp in 2018 and will begin production that same year. Further development of Wildfire will continue in 2019 and will be completed by fall 2019

The pre-production and main access development schedule is shown in the Table 16-44.

Table 16-44: Pre-production and Capital Development Schedule

Development	Length (m)	Advance (m/d)	Duration (days)	2012	2013	2014	2015	2016	2017	2018
Crews Mobilization			15	1						
Setup Services			15	1						
Portal Boxcut Excavation			60	1						1
Decline Portal	50	2	25	25						25
Main Access Ramp	2,460	5.3	462	640	1510	310				
Remuck Bays and Sumps	585	4.3	137	90	165	60	50	50	50	120
Ventilation Drift	340	4.3	80	15	265	15	15			30
Crosscut	160	4.3	38	85	75					
Ore Development	180	5.3	34	180						
Alimak Nest	60	4.0	11	15	15		15			15
Copper Keel Ramp	2345	5.3	440				1199	661	485	
Wildfire Ramp	634	5.3	119							634
Total Jumbo Development	6,814			1,050	1,990	425	1,279	711	535	824
Vent Raise Collar			15							
Vent Raise 3 m x 3 m	830	4.0	206	60	85	308	277			100
Raise Slash to 3 m x 5 m	730	6.5	119	60	85	308	277			
Internal Raise 3 m x 5 m	180	2.5	72	27	153					
Total Raise Development	1,010			87	238	308	277			100

16.6.4.2. Underground Production Schedule

The criteria used for scheduling of underground mine production at the Minto mine were as follows:

- Target mining blocks with higher grade ore in the early stages of mine life to improve project economics;
- Production sequence of the mining blocks would be from the top down;
- An average annual ore production rate of 730,000 t is scheduled, including ore from development and stopes;
- The mine will operate two 12 hr shifts per day, 365 d/a;
- Provide enough production faces to support a daily mine production rate of 2,000 t/d.

As per the Phase V PFS, ore mining is planned to commence at stope 101 in Area 118, which provides access to higher ore grades at the early stage of the mine life. The smaller mining blocks with low grades would be mined at the end of the mine life.

The stope cycle times and productivities were estimated from first principles for each type of mining method per average production stope size of 8 m wide and 5 m high.

"It was estimated that the average production from the single heading at RAP mining would be approximately 450 t/day. It would require 4.5 production faces working at any time to meet daily production requirements of 2,000 t/day. The number of mining blocks and flexibility of RAP and PPCF mining methods would allow as many stopes in production as is required to support the production rate." (SRK 2011)

This criteria continued to be utilized for the Copper Keel and Wildfire zones.

The UG mine production schedule by stope is detailed in Table 16-45. Ore production by year is shown in Table 16-46.

16.6.4.3. Underground Mine Production Rate

The following factors were considered in the estimation of the underground mine production rate:

- Reserve tonnage and grade;
- Geometry of the ore bodies;
- Amount of the required development;
- Stopes productivities; and
- Sequence of the mining and stopes availability.

The underground mine production rate of 2,000 t/d was considered appropriate due to the high degree of mechanization, potential high productivities of selected stoping methods. Development scheduling was planned to have at least 2 production stopes, with multiple faces, available at all times.

The tonnage in an average production stope size of 8 mW x 5 mH with advance of 3.85 m per cycle would yield about 450 tonnes, therefore, 2,000 tpd of production would require approximately 4.5 full cycles per day. Based on the presence of several deposits and ability to have production from different sub-levels, SRK considers the UG production rate to be achievable.

Table 16-45: Underground Mine Production Schedule by Stope

Area	Stope	Reserve (Kt)	RAP Drifting (%)	RAP Benching (%)	PPCF (%)	2013 (Kt)	2014 (Kt)	2015 (Kt)	2016 (Kt)	2017 (Kt)	2018 (Kt)	2019 (Kt)
118	101	650	37	20	43	383	267					
	103	50	22	10	68			50				
	104	15	100						15			
	105	8	94	6				8				
	106	30	65	35				30				
	107	12	83	17				12				
Area 2	201	376	17	6	77		376					
	203	268	68	19	12			268				
	204	136	26	19	12		90	45				
	205	39	74	26					39			
	206	11	93	7					11			
	207	22	71	29					22			
	208	20	76	24					20			
	209	23	95	5					23			
	210	19	81	19					19			
	211	52	44	23	33				52			
East	301	709	16	7	77			325	384			
Copper Keel	510	117	35	33	32					80	37	
	558	131	16	7	77				50	80		
	600	1,137	16	7	77				100	550	482	
	610	28	73	13	14						27	
	617	72	35	52	13						71	
	635	76	16	7	77						76	
Wildfire	800	46	73	13	14							46
	760	260	16	7	77							260
	750	54	73	27	0							53
Total		4,361				383	733	738	736	716	696	360

Table 16-46: Underground Mine Production Schedule

Parameter	Units	Total	2012	2013	2014	2015	2016	2017	2018	2019
Ore tonnes	Kt	4,361		383	733	738	736	716	696	360
Waste Rock	Kt	805	94	173	120	144	86	64	125	
Total Material	Kt	5,166	94	556	853	882	823	779	821	360
Cu	(%)	1.86		1.84	1.83	1.91	1.97	1.80	1.83	1.76
Au	(g/t)	0.76		0.69	0.76	0.86	0.87	0.66	0.63	0.80
Ag	(g/t)	6.67		7.23	6.49	6.88	6.42	6.74	6.66	6.35
NSR	(\$/t)	89.72		\$88.67	\$88.42	\$92.86	\$95.37	\$86.52	\$87.69	\$85.77
Mined Contained Copper	Mlb	179		16	30	31	32	28	28	14
Mined Contained Gold	koz	106		9	18	20	21	15	14	9
Mined Contained Silver	koz	935		89	153	163	152	155	149	73

16.7. Waste Management

Tailings from the mill will be sent to the permitted existing dry-stack location until November 2012. Thereafter, thickened tailings generated from processing ores will then be deposited into the Main Pit. The permit application for the deposition of tailings into the Main Pit was part of the Phase IV permit that was filed in August 2010 and is assumed that it will be approved in Q3 2012. Additional capacity required annually to store approximately 700,000 cubic metres of water associated with freshet flows, plus incremental storage to meet minimum and maximum operational requirements has been taken into consideration.

Once the Main pit tailings capacity is reached, in-pit tailings deposition begins in Area 2 pit.

The Area 2 pit will hold a significant portion of the tailings that is produced in the Phase VI LOM plan. Ridgetop North is also used as an in-pit storage facility once the Area 2 pit reaches tailings storage capacity. Additional tailings storage capacity is available in the underground workings, however, it is unused in the tailings plan.

This deposition of tailings into the Area 2 and Ridgetop North Pits and underground workings will form part of the Phase V / VI permit application to be submitted early in 2013 and is assumed that it will be approved before Q1 2015.

Therefore, the comment noted in the Phase V PFS by SRK remains valid in the Phase VI PFS, *“Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits, reduces the amount disturbed land that would normally be required by mining of the Phase VI deposits, and provides a significant cost savings over the current dry-stack method.*

Waste rock from Area 2 pit is being deposited to the Mill Valley Fill dump to the west of the existing mill facilities. Thereafter, Area 2 waste rock will be sent to the Main Pit as part of the Main Pit buttress. Upon completion, Area 2 waste rock will be sent to the permitted Southwest Valley waste dump located in the lower valley southwest of the Main Pit.” (SRK 2011)

Area 2 Stage 1 & 2 and 118 mining will be completed in 2013 which enables short haul dumps for Area 2-Stages 3. In addition, waste rock from Minto North is proposed to be stacked onto the existing Main pit dump.

Backfilling of Ridgetop South pit will also provide waste storage capacity and benefit the final reclamation plan. Overburden material will be placed in temporary dumps adjacent to the various deposits and used for final reclamation. Any excess overburden will be added to existing Overburden dump.

16.7.1. Waste Rock Dump Designs

16.7.1.1. Southwest Valley Fill Dump

“The medium and high grade waste rock material, and low grade oxide, generated from open pits will continue to be placed in the currently permitted West Valley Fill dump in the lower valley. The dump has an overall face slope angle of 24 degrees (toe-crest) with safety berms spaced at regular 10 m (vertical) intervals. The berms are designed to have a width of 15 m.

Overburden material will be placed in temporary dumps adjacent to the various deposits and used for final reclamation. Any excess overburden will be added to existing Overburden dump located to the north of the West Valley Dump.” (SRK 2011)

16.7.1.2. Mill Valley Dump

“The Mill Valley Dump is a smaller dump located east of the existing process facilities and north of the dry stack tailings facility. Waste rock material from Area 2 pit will be placed in this dump.” (SRK 2011)

16.7.1.3. Main Pit Dump

“Minto North pit waste material is to be placed on top of the existing Main dump located on a south-facing slope west of the Main Pit and immediately south of the Minto North pit. The dump has an overall face slope angle of 25 degrees (toe-crest) with safety berms spaced at regular 12 m (vertical) intervals. The berms are designed to have a width of 12 m”. (SRK 2011)

16.7.2. Backfill Dumps

“Ridgetop South pit and the 118 pit will be backfilled with waste generated from subsequent mining. Overburden material will then be placed as a cap on the backfilled pits”. (SRK 2011)

16.7.2.1. Capacities and Sequence

Table 16-47 below summarizes the waste quantities produced by each stage of this pre-feasibility report. Material is reported in terms of type as well as tonnage and cubic metres.

Table 16-47: Waste Quantities by Stage

Zone	Overburden	Rock	Total Waste	Overburden	Rock	Total Waste
	(Kt)	(Kt)	(Kt)	(Km ³)	(Km ³)	(Km ³)
Minto North	921	8,981	9,902	599	4,309	4,907
Ridgetop	1,297	7,596	8,893	842	4,300	5,142
118	352	1,617	1,969	229	792	1,021
Area 2	6,011	16,921	22,932	3,907	8,278	12,185
Grand Total Waste	8,581	35,115	43,696	5,576	17,679	23,255

**Note 1.3 swell factor used (m³/bcm)*

A summary of the metal content of the waste rock material to be generated from the open pits is shown in

Table 16-48 below as well as in Figure 16—12 and Figure 16—13. The waste rock material has been classified into four grade bins based on Cu%:

- Grade Bin 1: 0% Cu;
- Grade Bin 2: 0.01 to 0.10% Cu;
- Grade Bin 3: 0.10 to 0.36% Cu; and
- Grade Bin 4 0.36% Cu to 0.5% Cu (cut-off grade).

Table 16-48: Mineralized Waste Rock Classification

Area	Grade Bin	Waste (Kt)	Cu (%)	Cu (000's lb)	Au (g/t)	Ag (g/t)
118	0.36 - 0.50	0	0.00	0.0	0.00	0.00
	0.10 - 0.36	166	0.3	965.0	0.03	0.72
	0.01 - 0.10	0	0.00	0.0	0.00	0.00
	0	1,452	0.00	0.0	0.00	0.00
Area 2 Stage 1	0.36 - 0.50	0	0.00	0.0	0.00	0.00
	0.10 - 0.36	784	0.23	4,002	0.04	0.61
	0.01 - 0.10	0	0.00	0.0	0.00	0.00
	0	4,088	0.00	0.0	0.00	0.00
Area 2 Stage 2	0.36 - 0.50	0	0.00	0.0	0.00	0.00
	0.10 - 0.36	1,572	0.28	9,549	0.05	0.82
	0.01 - 0.10	0	0.00	0.0	0.00	0.00
	0	6,759	0.00	0.0	0.00	0.00
Area 2 Stage 3	0.36 - 0.50	0	0.00	0.0	0.00	0.00
	0.10 - 0.36	487	0.25	2,713	0.05	0.76
	0.01 - 0.10	0.00	0.00	0.00	0.00	0.00
	0	3,232.00	0.00	0.00	0.00	0.00
Minto North	0.36 - 0.50	39	0.45	385.0	0.10	1.54
	0.10 - 0.36	9	0.28	59.0	0.10	1.39
	0.01 - 0.10	0	0.00	0.0	0.00	0.00
	0	8,932	0.00	0.0	0.00	0.00
Ridgetop North	0.36 - 0.50	485	0.43	4,583.0	0.06	1.03
	0.10 - 0.36	870	0.23	4,418.0	0.04	0.69
	0.01 - 0.10	719	0.04	710.0	0.03	0.24
	0	4,235	0.00	1.0	0.00	0.00
Ridgetop South	0.36 - 0.50	76	0.42	708.0	0.11	1.41
	0.10 - 0.36	270	0.22	1,303.0	0.05	0.65
	0.01 - 0.10	62	0.06	82.0	0.01	0.23
	0	861	0.00	0.0	0.00	0.00
Total	0.36 - 0.50	600	0.43	5,676	0.07	1.11
	0.10 - 0.36	4,158	0.25	23,009	0.05	0.73
	0.01 - 0.10	781	0.04	792	0.03	0.24
	0	29,559	-	0	-	-

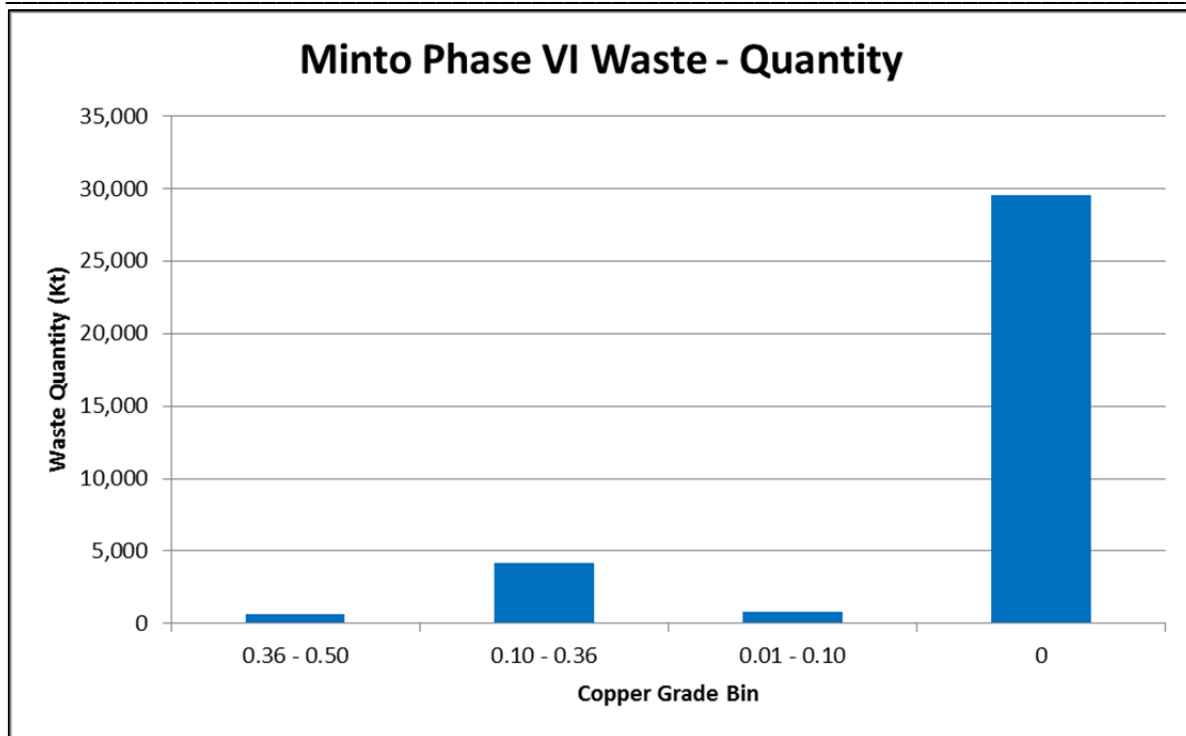


Figure 16—12: Mineralized Waste Quantity

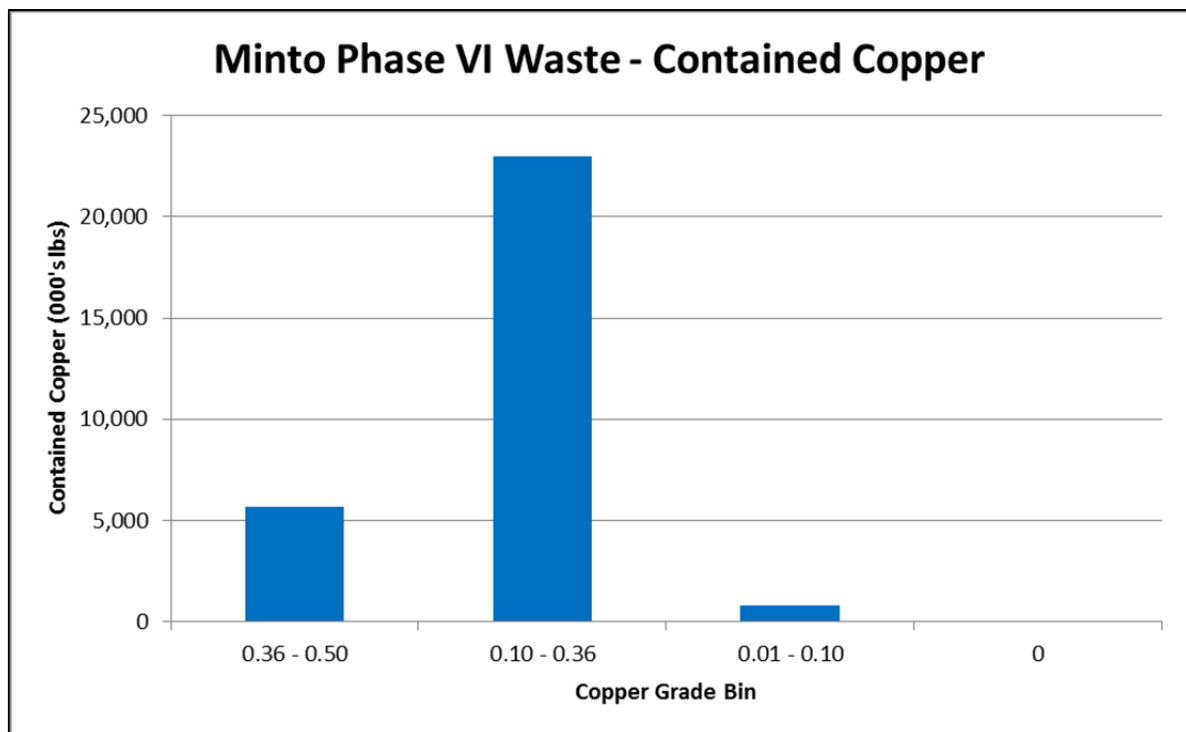


Figure 16—13: Mineralized Waste Contained Cu

16.7.3. In-pit Tailings Disposal

In-pit tailings management will continue to be an important part of the Life of Mine Plan. As noted in the Phase V PFS, based on preliminary laboratory testing on tailings from the thickener underflow, the average dry density of the deposited tailings continues to be estimated to be 1.30 t/m³ for the anticipated sub-aqueous deposition. Further laboratory tests have indicated that there is an opportunity that higher densities can be attained. Therefore, Minto is considering a number of initiatives that would improve consolidation such as flocculent addition to the tailings, wick drains and compaction.

For this Phase VI PFS it was assumed that tailings from the mill will be sent to the currently permitted existing dry-stack location until November 2012. Thickened tailings generated from processing ores from other Phase VI deposits will then be deposited into the Main Pit, Area 2, Ridgetop North pits and if required, underground workings. A total of 9.9 Mm³ of tailings storage capacity will be required for the Phase VI LOM plan.

The construction of the Main Pit buttress (required for slope stability) is required. Additional capacity required annually to store approximately 700,000 cubic metres of water associated with freshet flows, plus incremental storage to meet minimum and maximum operational requirements has been taken into consideration.

Upon completion of the Main pit tailings deposition, in-pit tailings deposition will begin in the the mined-out Area 2 pit. The entire Area 2 pit will increase in-pit tailings capacity by 5.5Mm³; providing the total capacity of 7.5Mm³. Ridgetop North is also used as an in-pit storage facility as well as the underground workings.

Although these tailings deposition plans are not yet permitted, they offer a potentially viable solution to tailings disposal that provides backfill material for the Main, Area 2 and Ridgetop North pits that reduces the amount of disturbed land that would normally be required by mining of the Phase VI deposits, and provides a significant cost savings over the current dry-stack method.

Table 16-49 shows the potential LOM in-pit storage capacities of the Main, Area 2 and Ridgetop North open pits and if required, underground workings. Minto North, Ridgetop South and 118 pits are not considered for in-pit tailings backfill.

Table 16-49: Tailings Storage Capacity by Pit

Pit	Max. elev (m)	Capacity (Mm ³)
Main Pit*	788	3.0
Area 2	800	5.8
Ridgetop North	860	1.1
OP Total		9.9
Underground Workings		1.5
Total		11.4

*incorporating increased buttress design

Main Pit had a design capacity of 5.1Mm³ in the Phase V PFS. However, due to slope stability issues, a larger buttress is required which has reduced the total capacity to 3.0 Mm³. The Main Pit maintains an additional 0.7Mm³ storage capacity for freshet. Area 2 tailings capacity has decreased from 7.5 Mm³ to 5.8 Mm³ to enable 1.0Mm³ in-pit waste dumping and 0.7Mm³ storage capacity for freshet.

It was determined that the maximum in-pit capacity available is 9.9 Mm³ and underground workings is 1.5Mm³.

Table 16-50 summarizes this information in terms of tailings produced and where it is planned to be deposited on a yearly basis. Tailings quantities along with tailings elevation levels are included.

In-pit tailings deposition commences in late-2012 upon completion of Main pit buttress and permit approval. The tailings deposition into Main pit continues until mid-2015 where the maximum estimated capacity is attained. In-pit tailings deposition then begins in the Area 2 pit which is scheduled to be completely mined by that time. All of the remaining tailings will be deposited in the Ridgetop North. An alternative location if required are the underground workings where the tailings would be deposited as hydraulic fill.

Table 16-50: Tailings Production and Storage Schedule

Item	Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
OP Mining schedule	(pit)		Area 2 Stage 1 Area 2 Stage 2	Area 2 Stage 2 118	Minto North Area 2 Stage 3	Area 2 Stage 3 Ridgetop North	Ridgetop North	Ridgetop North Ridgetop South					
Waste mined (Open Pit Only)	(Kt)	43,677	9,854	10,129	9,890	7,166	4,831	1,806					
Ore mined (Open Pit Only)	(Kt)	9,178	2,056	2,073	1,697	1,656	1,211	484					
Ore Processed	(Kt)	14,353	1,373	1,369	1,369	1,369	1,373	1,369	1,369	1,369	1,373	1,369	654
Total inpit tails only	m ³	9,917,566	263,161	1,011,385	998,709	1,000,166	1,015,014	1,013,075	1,013,180	1,026,495	1,041,507	1,038,661	496,213
Main pit tails w buttress	(m ³)	3,043,255	263,161	1,011,385	998,709	770,000							
Main pit tails cumulative	(m ³)		263,161	1,274,546	2,273,255	3,043,255							
Freshet	(m ³)	700,000	700,000	700,000	700,000	700,000							
Total Main pit storage req'd cumul. (max 3.75M)	(m ³)	3,750,000	963,161	1,974,546	2,973,255	3,743,255							
Area2 Stg1&2 inpit tails	(m ³)	230,166				230,166							
Area2 Stage1&2 Waste Rock	(m ³)	1,000,000				1,000,000							
Area2 Stg1&2 inpit tails cumulative	(m ³)					1,230,166	1,230,166						
Freshet	(m ³)	700,000				700,000	700,000						
Total Area2 Stg1&2 storage req'd cumul. (max. 2.0M)	(m ³)	1,930,166				1,930,166	1,930,166						
Area2 Final inpit tails	(m ³)	5,569,834					1,015,014	1,013,075	1,013,180	1,026,495	1,041,507	460,563	
Area2 Final inpit tails cumulative	(m ³)						2,245,180	3,258,255	4,271,435	5,297,930	6,339,437	6,800,000	
Freshet	(m ³)						700,000	700,000	700,000	700,000	700,000	700,000	
Total Area2 Final storage req'd cumul. (max. 7.5M)	(m ³)	7,500,000					2,945,180	3,958,255	4,971,435	5,997,930	7,039,437	7,500,000	
RGN inpit tails	(m ³)	1,074,311										578,098	496,213
RGN inpit tails cumulative (max 1.1M)	(m ³)	1,100,000										578,098	1,074,311
Total Storage	(m³)	9,917,566	263,161	1,011,385	998,709	1,000,166	1,015,014	1,013,075	1,013,180	1,026,495	1,041,507	1,038,661	496,213

17. Recovery Methods

17.1. Introduction

The Minto process plant design incorporates standard industry communiton, flotation and dewatering circuits to produce a final copper concentrate product.

Ore from the new deposits will be processed through a modified (see Section 17.7 below for details) Minto process plant.

17.2. Plant Design Basis

The key criteria selected for plant operations planning are:

- Treatment of an average 3,750 dry metric tonnes per day for 2012 and beyond;
- Material from Minto Main, Minto North, Ridgetop, Minto East, Area 2, Area 118, Copper Keel and Wildfire will be processed through the Minto plant;
- Increase recoveries of POX material through continued testing;
- Design availability of 92%, 7,997 operating hours per year, with standby equipment in critical areas; and
- Sufficient plant design flexibility for treatment of all ore types as per test work completed at design throughput.

The selection of these parameters is discussed in detail below.

17.3. Throughput and Availability

An overall plant availability of 92% or 7,997 hr/yr was chosen. Benchmarking indicates that similar well operated plants with moderately abrasive ore have consistently achieved 91 to 92% overall plant availability.

The existing Minto process plant availability has averaged 93.2%. Through monitoring of equipment and record keeping operations personnel have identified the cause of the lower availability and have commenced a program of preventative maintenance and equipment duplication (installing stand-by equipment). It is expected once the program is complete an availability of 92% will be achievable.

Major causes for reduced availability include:

- Failure of the installed flotation mechanisms. These have been replaced with a new supplier and replacement frequency and costs are expected to reduce;
- Poor availability of the tailings treatment facility, particularly the filter circuit;
- Original pipe work around the milling area was not rubber lined. Pipe work was replaced with rubber lined pipes which will reduce the frequency of change-outs;
- Various pumps have been upgraded and standby tailings pumps installed under operating cost budgets.

The throughput selected is mainly a function of modifications planed to the crushing circuit and the existing Minto plant grinding circuit. From the review of Minto Phase IV study and recent test work data a plant throughput of 171 dry metric tonnes per hour based on 80% of the SAG feed

material being finer than 25 mm is achievable. With a 91.3% availability and 25 mm top feed size an average of 3,750 tonnes per day can be processed.

17.4. Processing Strategy

The process design is based on treating ore with similar hardness to the current Minto Main ore being processed or similar to that tested by DJB Consultants in October 2007, and Eggert Engineering in 2011. Minto Mill Department are continuing to increase capacity, recoveries and reliability of the process plant through upgrades to its systems.

17.4.1. Partially Oxidized Ores

Recoveries in 2011 were adversely affected by the treatment of stockpiled partially oxidized (POX) material. A laboratory testing program (detailed below) was undertaken to quantify what recoveries could be expected from these ores. In addition, operations personnel undertook a program to assess the impact of slow additions of the POX to the mill. The laboratory testing indicated that addition of POX should not adversely affect recovery. This result was not supported by the metallurgical performance in 2011 and a short investigation by John Eggert was undertaken. The copper minerals in the ore that is referred to as POX are sulphides. These sulphides have been “oxidized” from chalcopyrite to other sulphides, such as bornite or chalcocite. Generally, these minerals float well. In addition to bornite and chalcocite the POX treated in 2011 also had a significant quantity of oxide minerals, such as azurite. Some of the feldspars had oxidized to clays. The net result of treating this unusual material was reduced recovery, filtration issues for the concentrate, and occasional issues with filtration of the tailings. In early 2012, operations continued to blend POX material into the plant feed. Recoveries have improved and are again at target levels, even at increased throughput.

17.5. Head Grade

The plant is designed to treat various tonnages of primary ore with a sustained maximum head grade of 2.5% Cu and 1.5 g/t Au.

17.6. Flow sheet and Equipment Sizing

The process plant flow sheet design for Minto is based on the strategies mentioned in the Phase VI PFS. Figure 17—1 shows a process schematic for the current Minto process plant with dry-stack tailings. Upon completion of Phase V permitting, the tailings will be thickened and sent to the in-pit deposition locations via pipeline. A number of changes to the schematic will be completed further enhance processing capabilities.

Details of the flow sheet design and major equipment is discussed in the sections below.

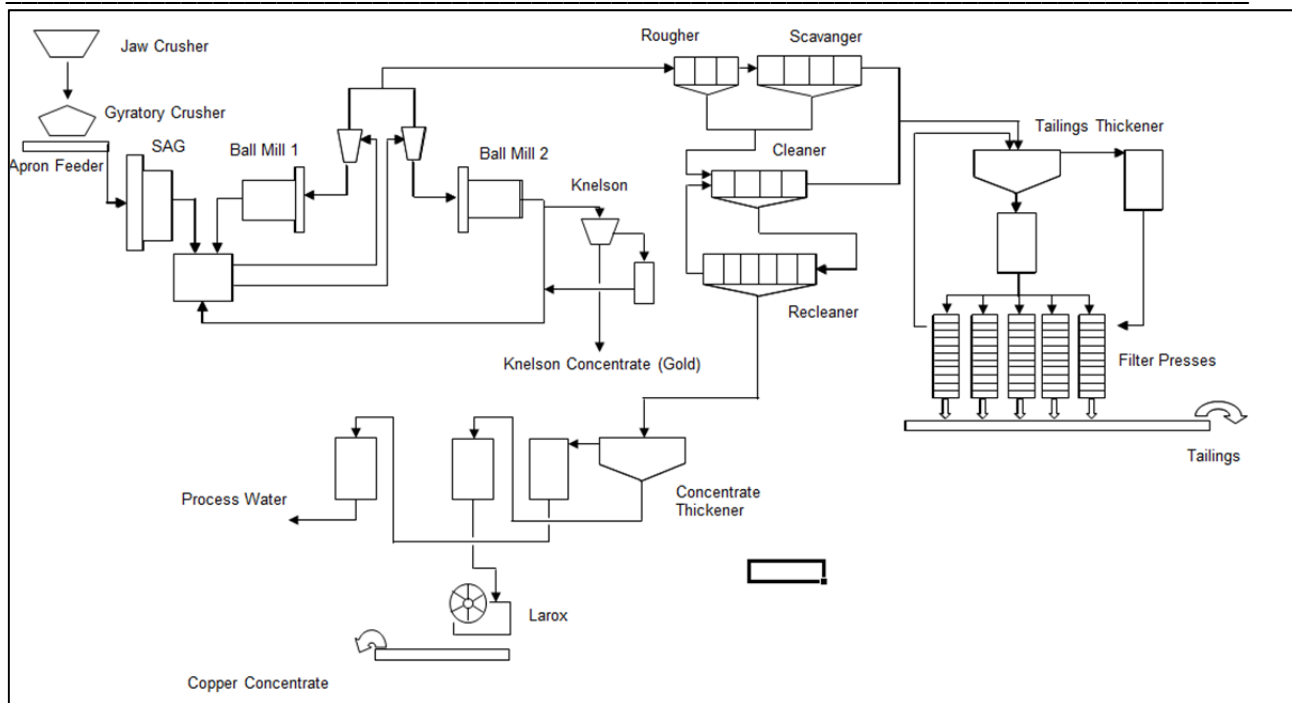


Figure 17—1: Minto General Flow Sheet with Dry-Stack Tailings

17.7. Plant Design

The plant design used to model the plant throughput and metallurgical performance are well proven from historical operational data. The flow sheet incorporates both new and existing unit process operations:

- Ore from the open pit is crushed using the existing primary jaw crusher to a crushed product size of nominally 80% passing (P80) 115 mm. Currently the jaw crusher product is fed to a gyratory crusher that reduces the ore down to P100 60 mm, P80 32 mm. A contract crusher is in operation. In 2012, work will be undertaken to screen the jaw crusher discharge and send the oversize to the gyratory crusher. The crusher product will be returned to the screen, putting the gyratory crusher in closed circuit with the screen. This flow sheet will generate a P100 of 45mm that should support the SAG mill feed rate of 171 dmt/hr. Operations will strive to improve the productivity of the new plant with the goal of eliminating the contract crushing;
- Conical stockpile with the existing single reclaim apron feeder;
- Existing 670 kW SAG mill, 5.03 m diameter with 1.52 m EGL;
- Existing twin 670 kW ball mills each 3.20 m diameter with 3.66 m EGL, in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P80) 250 micron;
- Ball mill 2 feeds a Knelson concentrator to concentrate any free gold;
- Bulk rougher/scavenger flotation consisting of the existing three 40 m³ forced air tank flotation cells and the existing four 15 m³ cells retrofitted of new tank cell 20 mechanisms;
- A Jamieson cell will be installed as a rougher-by pass system which will divert a portion of the concentrate from the cleaner/scavenger cells to the concentrate thickener thus increasing capacity of the cleaner/scavenger cells and improving recovery;

-
- Rougher/scavenger concentrate regrinding in a new 220 kW vertical stirred mill, grinding to a product size of nominally 80% passing (P80) 80 micron. Regrind circuit will be added in from the third quarter of 2012;
 - Cleaner 1 flotation consisting of the existing four 10 m³ forced air tank flotation cells;
 - Cleaner 2 flotation consisting of three new 10 m³ tank cells;
 - Cleaner 3 flotation consisting of the existing six 3 m³ trough shaped flotation cells to be replaced by two, 10m³ cells to increase capacity. Cleaner 3 flotation change will come on-line from the third quarter of 2012;
 - Final cleaner concentrate thickening in the existing 6 m diameter high rate thickener;
 - Concentrate thickened slurry filtration in the existing Ceramic disk filter;
 - Flotation tailings thickening in the existing 9.1 m diameter high rate thickener to an underflow density of 50% solids;
 - After completion of ore extraction, utilization Minto Main pit for tailings deposition directly from the flotation tailings thickener underflow pumps;
 - Plant reagents preparation and distribution systems as per the current Minto unit operations;
 - Raw process plant water supply from the existing site water storage facility reticulated throughout the plant as required;
 - Process water dam and distribution system for reticulation of process water throughout the plant as required per the existing facilities. Process water is supplied from water reclaimed from tailings deposition in the Minto Main pit, from process operations and site run-off with raw water used as make-up water as required;
 - Potable water as per the existing supply is distributed to the plant, and for miscellaneous purposes around the site; and
 - Plant, instrument and flotation air services and associated infrastructure as per the existing facilities.

17.7.1. Primary Crushing

The primary crusher is a 37' x 49' jaw crusher. 100% of the jaw crush product is then fed directly to the secondary gyratory crusher (S4800) and crushed to a nominal 80% passing 32 mm. Undersize from the secondary crusher is fed onto the existing stockpile stacking conveyor.

Crusher performance has a marked effect on the SAG throughput and a fines content of 30-35% is needed to maximize tonnage. It is particularly noticeable that, when the crusher is down for maintenance, that the feed becomes noticeably coarser within 30 minutes. Compounding the problem is the fact that there is only one slot feeder under the coarse ore stock pile eliminating any chance of mixing the feed. As a result the crusher must run 24 hours per day, while the design criterion was for one 12 hour shift per day.

17.7.2. Grinding Circuit

The grinding circuit comprises the SAG mill (16.5' D x 5.0' L), and two grate discharge ball mills (10.5' D x 12' L) in parallel. All three mills have 900 HP motors.

The limiting factor in increasing tonnage is the SAG mill. The SAG has a reverse spiral and a trommel screen with 25mm openings.

Minto has been able to increase tonnage thru the SAG mill by pre-crushing ore completed by a contractor on-site.

To compensate for the coarser material feeding the ball mills, the steel charge will be 50% 3" balls and 50% 2" balls instead of the current 2" balls. The power draw on the mills will also be increased to closer to maximum available power draw of 900HP.

Future plans include installing a vertical stirred mill within the Rougher/scavenger circuit. The vertimill will grind the concentrate to a product size of nominally 80% passing (P80) 80 micron. Regrind circuit will be added in from the third quarter of 2012.

17.7.3. Flotation

The flotation circuit consists of three 40 cubic meter rougher flotation cells, four 14.4 cubic meter scavenger flotation cells, four 10 cubic meter first cleaner cells, and six 2.8 cubic meter final cleaner mechanical cells. Future opportunities include installing a Jamieson cell as a rougher-by pass system which will divert a portion of the concentrate from the cleaner/scavenger cells to the concentrate thickener thus increasing capacity of the cleaner/scavenger cells and improving recovery. Other plans include a second stage of cleaner flotation cells as well as replacing the final cleaners with two tanks cells.

17.7.4. Concentrate Dewatering

The concentrate thickener is an Outotec 9.4 meter diameter Supaflo high rate thickener. The concentrate is dewatered using a Larox ceramic filter. Depending on slurry density, dewatering rate varies from 25-30 tonnes per hour with a moisture content of 7 - 8 %.

Concentrate storage shed on site is capable of holding 18,000 tonnes of concentrate. Shipping from site stops in the fourth quarter, while the Yukon River freezes up and the ice bridge is built across the Yukon River and in the second quarter for spring break-up. Fall freeze-up and spring break-up periods last 6 to 8 weeks.

Concentrate is trucked to Skagway for shipping overseas to customers.

17.7.5. Tailings Disposal

The tailings are thickened to 58% solids in a 13.5 meter diameter thickener manufactured by Westpro and pumped to a stock tank for temporary storage. Five Lasta 1500x1500 pressure filters are used to dewater the tailings. The filters are fully automatic and produce an average of 6.5 tonnes tailings per cycle at 16% moisture. Cycle times are between 8.5 and 9 minutes, giving each filter a capacity near 1,000 tonnes per day.

Filtered tails are conveyed to a stock pile and trucked to the dry stacked tailings area.

The plant operates exceptionally well, a credit to the operating personnel. Once permits have been granted, Minto will be moving towards an in-pit tailings deposition plan.

The in-pit tailings plan will pump thickened tailings into the Main Pit and other designated in-pit tailings locations using a 2,400 meter long high density poly-ethylene tailings pipeline.

17.8. Plant Consumables

17.8.1. Water Balance

Water used for mineral processing is provided from the mill pond and the water storage dam. After processing is complete, water exits the mill through concentrate, tailings and the mill pond. Water extracted from the dry-stack tailings facility is sent to the water treatment plant where the water is treated and released into the environment. Small amounts of water come from the dry stack tailings and make their way to Minto Creek. Detention structures move the water back up to the Main Pit where they are discharged into the water stored in the Main Pit. The current plan has this water in the Main Pit being processed through the Water Treatment Plant for several months each spring, summer and fall.

Figure 17—2 provides a general water balance for a typical month.

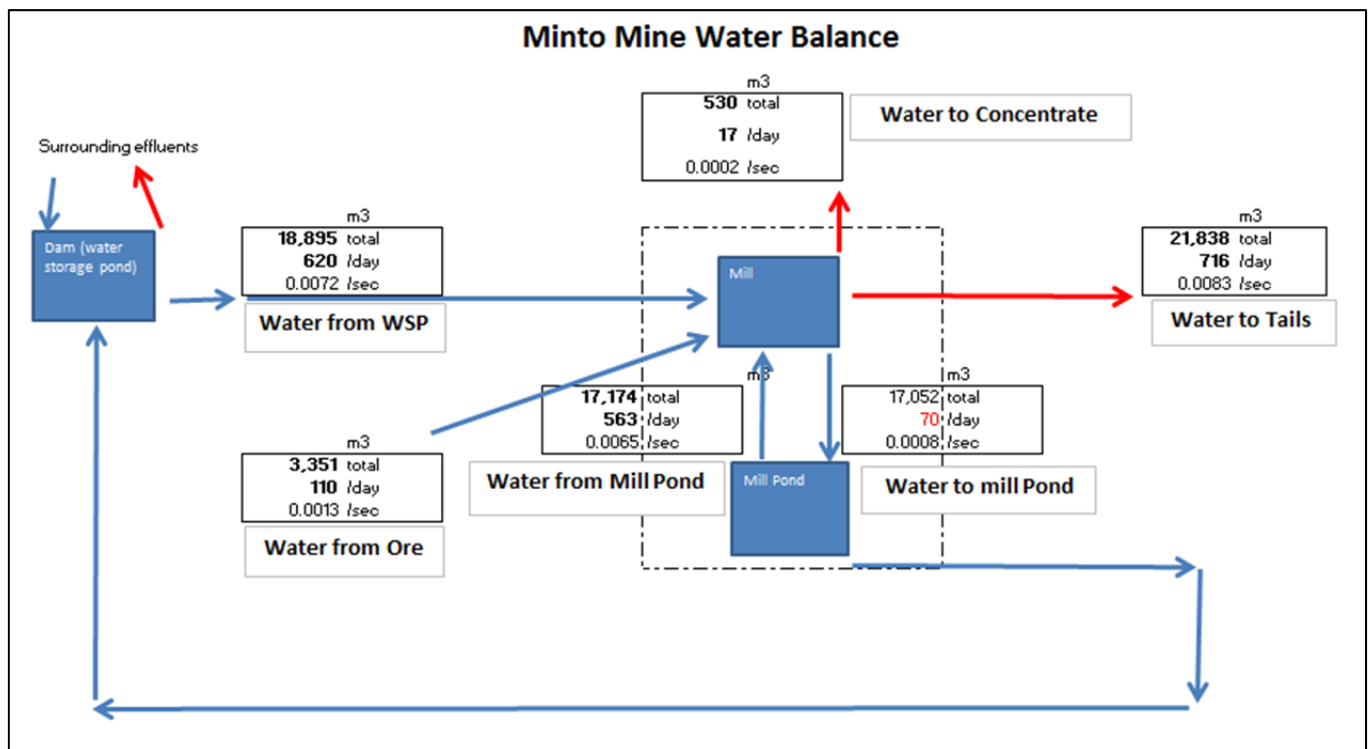


Figure 17—2: Minto Water Balance

There is an overflow pipe that comes off the Mill Pond. This overflow pipe is a CSP (corrugated steel pipe) that travels approximately 200 meters underground to a point on the Main Access Road where it discharges into small pit. The small pit then has a second CSP that travels under the Access Road to the Minto Creek Valley.

In the future, the overflow system will be shut down. The mill should be able to operate such that only a balanced amount of water will be brought from the WSP to the mill water systems. This would then keep the Mill Pond from getting too high or too low.

17.8.2. Power

Power will continue to be supplied to the mine site from the local power grid. The cost of power was based on the MintoEx 2012 budget unit power rate value of \$0.11 /kWh.

The power requirements for the current plant were developed based on a specific power load of 30kWh/t for the first 6 months of 2011, while the tailings filtration circuit is in operation. From July 2012 to 2021 the power load is reduced to 26kWh/t. The reduction in power load considers the standby operation of the filtration circuit which is offset by the plant modifications to accommodate the increase in throughput.

The plant power consumption is expected to vary slightly over the life of the mine primarily due to the variable comminution characteristics of the ore and resulting change in comminution energy requirement.

17.8.3. Reagents and other Consumables

17.8.3.1. Reagents

Reagent consumptions will vary according to metallurgical and production parameters. Generally the consumption rates used are in line with the average consumptions from the current plant performance.

Reagent unit costs were based on the actual unit rates used in the 2012 budget. The rates are freight in store (FIS) at the Minto mine. The site is reviewing other reagents than AM28 and NaSH that are required to enhance recovery associated with partially oxide ores. Therefore, there is good potential to decrease reagent expenditures. For the purposes of this study, a conservative approach was taken.

Table 17-1: Processing Reagents and Consumption Rate

Reagent	Consumption kg/t milled
Flotation Collector (PAX)	0.02
Flotation Frother (MIBC)	0.05
Nitric Acid (kg)	0.05
AM28	0.42
NaSH	0.25
Tailings Flocculant (AE4270)	0.03
Filtration Flocculant (AE4270)	0.03
Concentrate Flocculant (AE4330)	0.001

17.8.3.2. Crusher and Mill Liner Consumption

Plant consumables include major items, such as crusher and mill liners and grinding media. Consumption rates were estimated from the 2011 budget consumption rates as well as benchmarking. Consumption rates and unit costs are summarised in Table 17-2.

Table 17-2: Liner Consumption

Item	2012 - 2022 Consumption
SAG Mill Liners	SAG liners - 3 shell, 2 grates, half feed end and discharge liners
Ball Mill Liners	2.0 sets per year
Jaw Crusher Liners	2.7 sets per year
Secondary Crusher Liners	4.0 sets per year
Regrind Mill Liners	1.0 set per year

17.8.3.3. Grinding Media Consumption

The SAG and ball mill media consumption rates were estimated using the consumption rates for the 2011 budget. The regrind mill media consumption rate was estimated by benchmarking.

Table 17-3: Grinding Media Consumption

Mill	Diameter	Type	Plant Consumption Rate (kg/t)
SAG Mill	125 mm	Forged	0.48
Ball Mill	75 mm	Forged	0.35
Ball Mill	50 mm	Forged	0.30
Regrind Mill	12 mm	Forged	0.08

18. Project Infrastructure

18.1. Existing Infrastructure

18.1.1. Site Power Supply

Power is currently being supplied to the mine site from a 32km, 34 kV power line from Minto Landing. The power supplier is the Yukon Energy Corporation.

The current Yukon Energy Corporation's (YEC) Power Purchase Agreement with Minto Mine allows for a total capacity of 4400kVA with a total annual consumption limitation of 32.5GWh. Minto Mine currently utilizes approximately 3800kVA of its allotted capacity of 4400kVA.

The current mine's power is distributed from a YEC owned substation at 5kV. The current system configuration utilizes a main switch room in the mill with two smaller switch rooms in the tailings building.

Previously, Minto was powered by diesel-power generators and continues to utilize these generators during electrical service disruptions.

18.1.2. Camp and Camp Services

The existing workforce camp at the Minto Mine consists of eight single-storey modular housing units capable of housing a total of 192 people. Each housing unit consists of individual sleeping rooms with common showers, washrooms, and recreation areas. The building houses the kitchen and dining hall originally designed for a 60-person camp. A separate building has been provided to serve as a gym and other recreation facilities is provided as extension of the housing buildings. Most of the buildings are connected by interior corridors except some of the outlying buildings.

Domestic and fire water are provided from a well and storage tank system located in the camp area. Sanitary sewage is directed to septic tanks connected to two septic fields that are currently operating at maximum capacity. Power for the camp is provided from the main sub-station via a transformer located in the camp area.

The existing camp buildings and septic fields are using the majority of available space on the fairly narrow ledge overlooking the Minto valley where the camp is located. The existing camp arrangement can be seen on Figure 18—1.

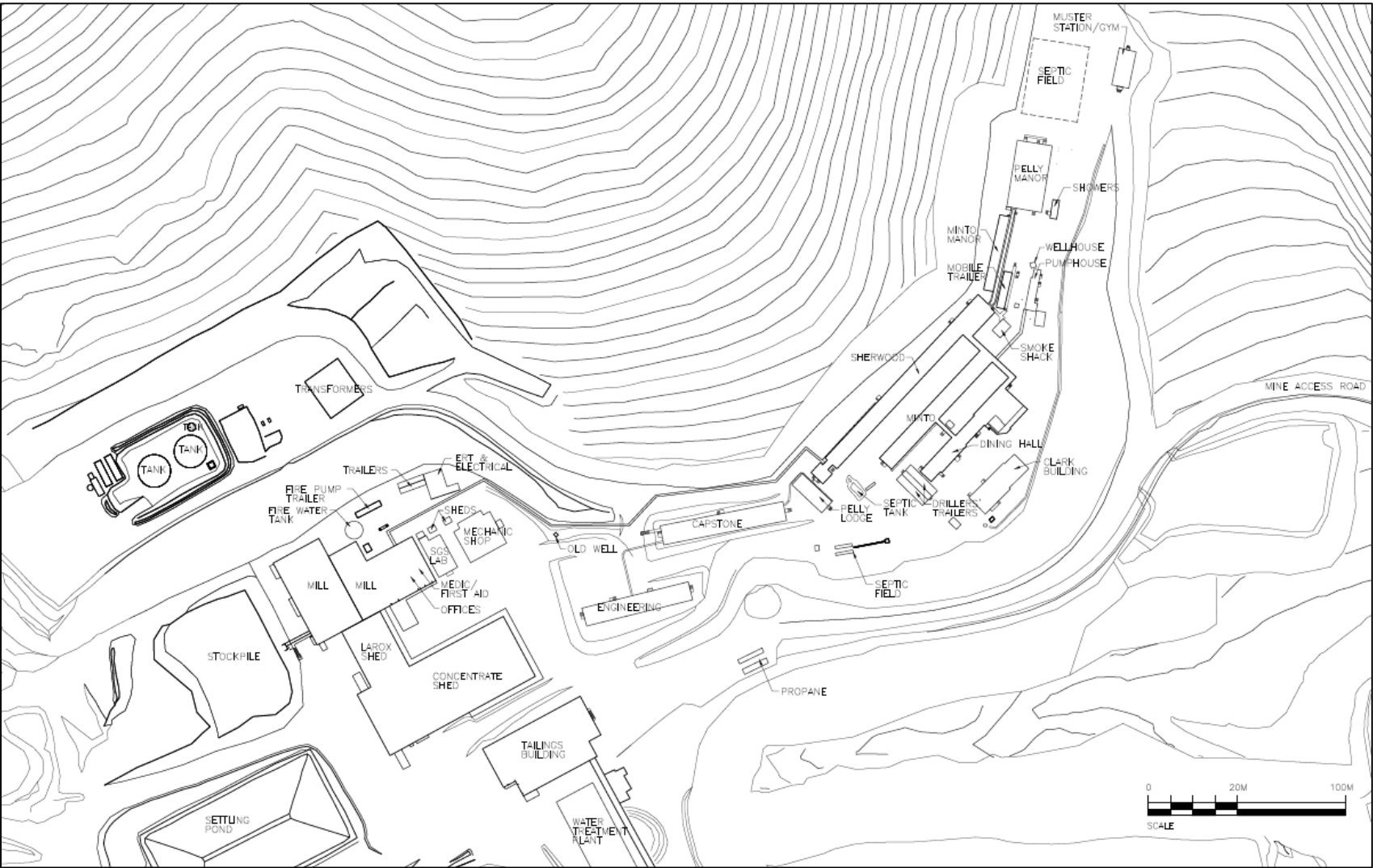


Figure 18—1: Minto Mine Site Plan

18.1.3. Warehouse and Mechanics Shop

The warehouse and mechanics shop is location beside the process plant. In addition, the contract mining company have set up an onsite workshop to maintain its mining fleet.

18.1.4. Fuel Storage

Diesel fuel is stored in diesel storage facility located just north of the process plant. Two, large tanks have a combined storage capacity of 3.2 million litres; sufficient for supplying open pit and underground mine equipment for several weeks as is required during freeze-up and break-up of the Yukon River.

18.1.5. Concentrate Building

Concentrate storage shed on site is capable of holding 18,000 tonnes of concentrate. Shipping from site stops in the fourth quarter, while the Yukon River freezes up and the ice bridge is built across the Yukon River and in the second quarter for spring break-up. Fall freeze-up and spring break-up periods last 6 to 8 weeks.

18.1.6. Water Treatment Plant

The water treatment plant has an inflow capacity of 250 m³/hr. The RO (Reverse Osmosis) system will then create 210 m³/hr of clean water that can be discharged to the environment. The balance of 40 m³/hr will be discharged back to the source as this is the concentrated brine carrying the target elements identified in the Permit.

18.1.7. Mill Pond and Water Storage Dam

The mill pond is located beside the process plant and has a capacity of 6,500 cubic meters of water storage capacity. The water storage dam is located approximately 1.5km up the mine access road and has a storage capacity of 270,000 cubic meters of water storage capacity. Both mill pond and water storage dam provide water to the process plant.

18.1.8. Dry-Stack Tailings Building

The dry-stack tailings building currently hold all the equipment required for 13.5 meter diameter tailings thickener and five Lasta pressure filters used for dry-stack tailings. Once dry-stack tailings operations are completed in mid-2012, the pressure filters will be decommissioned. The tailings thickener will still be operational used until the end of the mine life.

18.1.9. Explosives Storage

The explosives storage facility is run by the explosives contractor, Dyno Nobel. The explosive magazine is located south west region of the mine site and is isolated from any main infrastructure.

18.2. Upgrade and Expansion of Infrastructure

18.2.1. Minto Camp

The proposed camp upgrade will see the number of camp spaces increased to 248 through the use of three-storey modular housing units that will be installed in phases as the old buildings are decommissioned. The new modular units will be constructed on concrete or pressure treated foundations using jack-stands for leveling.

All of the new camp buildings will be connected with an interior corridor that will also contain the building electrical, communication and water distribution services. A common mudroom will be provided at the south end of the camp so that personnel can remove muddy boots as they enter the camp complex.

A modular sewage treatment plant will be provided to replace the existing septic fields and it will be installed first so that the septic field at the north end of the camp can be decommissioned to provide space for the first of the new housing units. New temporary and permanent sanitary piping, manholes and a grinder lift station will be required to direct sanitary sewage to the new sewage plant.

The proposed new camp arrangement is shown on drawing Figure 18—2.

A new dining hall, kitchen and recreation facility complex will be provided to accommodate the full camp loading of 248 people. An area of the new building will be fire-separated and assigned for the water treatment equipment, well and storage tanks. An electrical room and an IT communications room will also be provided in fire-separated rooms. This complex will be the service hub of the new camp.

The proposed floor plan arrangement of the new dining hall/kitchen/recreation facility can be seen on drawing Figure 18—3.

The modular buildings will be constructed at a factory off-site and shipped to site in sections for assembly. The majority of the three-storey housing units will have 10 bedrooms per floor for a total of 30 people per unit. A washroom will be provided for every two bedrooms and a common lounge area and laundry room will be provided on each floor. The proposed floor plan arrangement for one floor of a typical unit can be seen on drawing Figure 18—4.

New water treatment plant equipment is being provided to accommodate the increased camp loading as part of an existing project and it is anticipated that this work will be re-scheduled to coincide with the construction of the buildings. Propane distribution piping will be revised to service the new camp buildings. Electrical distribution equipment will be relocated and upgraded to service the new camp buildings. IT cabling will be re-routed to connect to the new camp buildings and to maintain service as the existing buildings are removed.

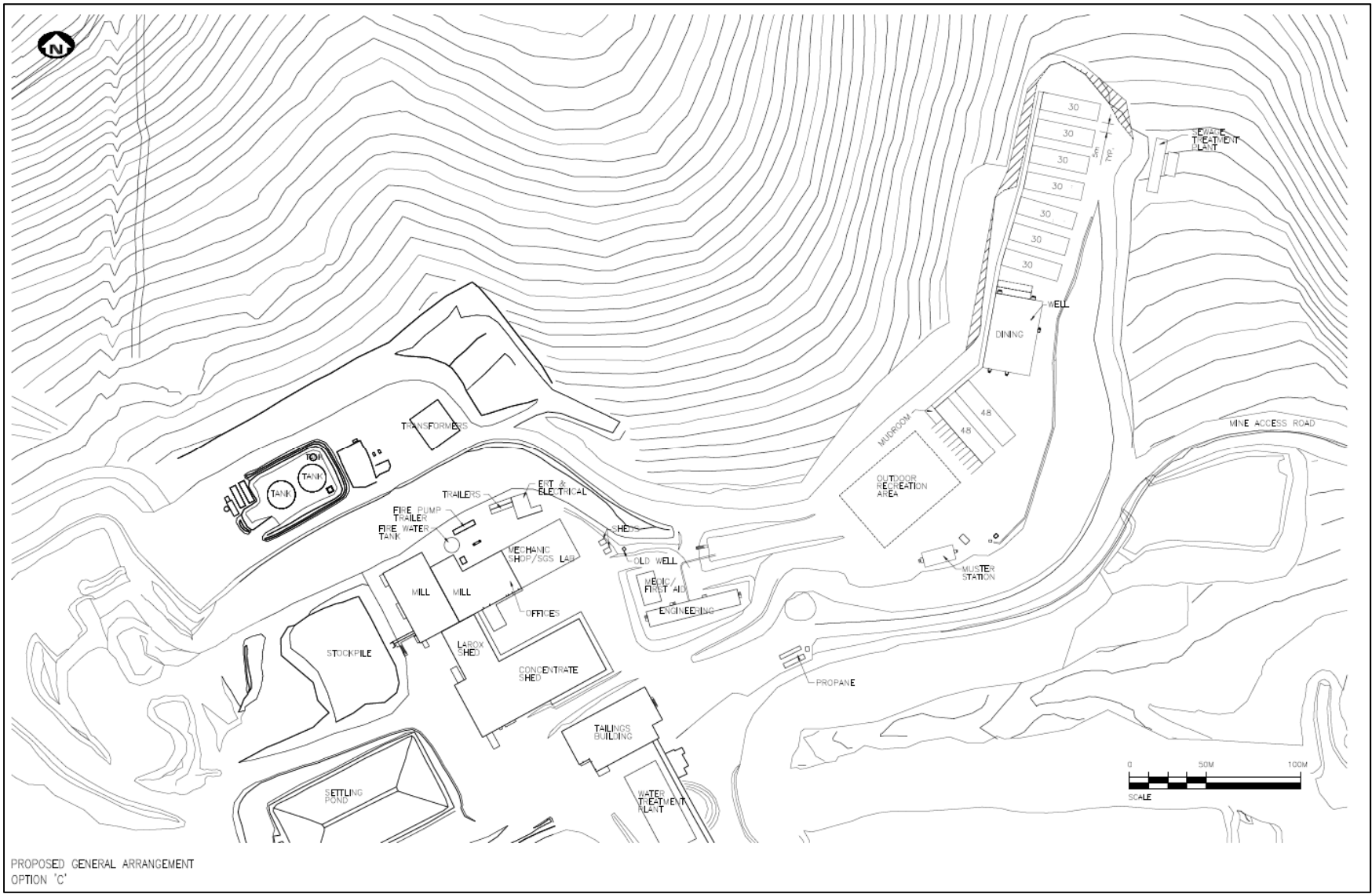


Figure 18—2: Proposed General Arrangement

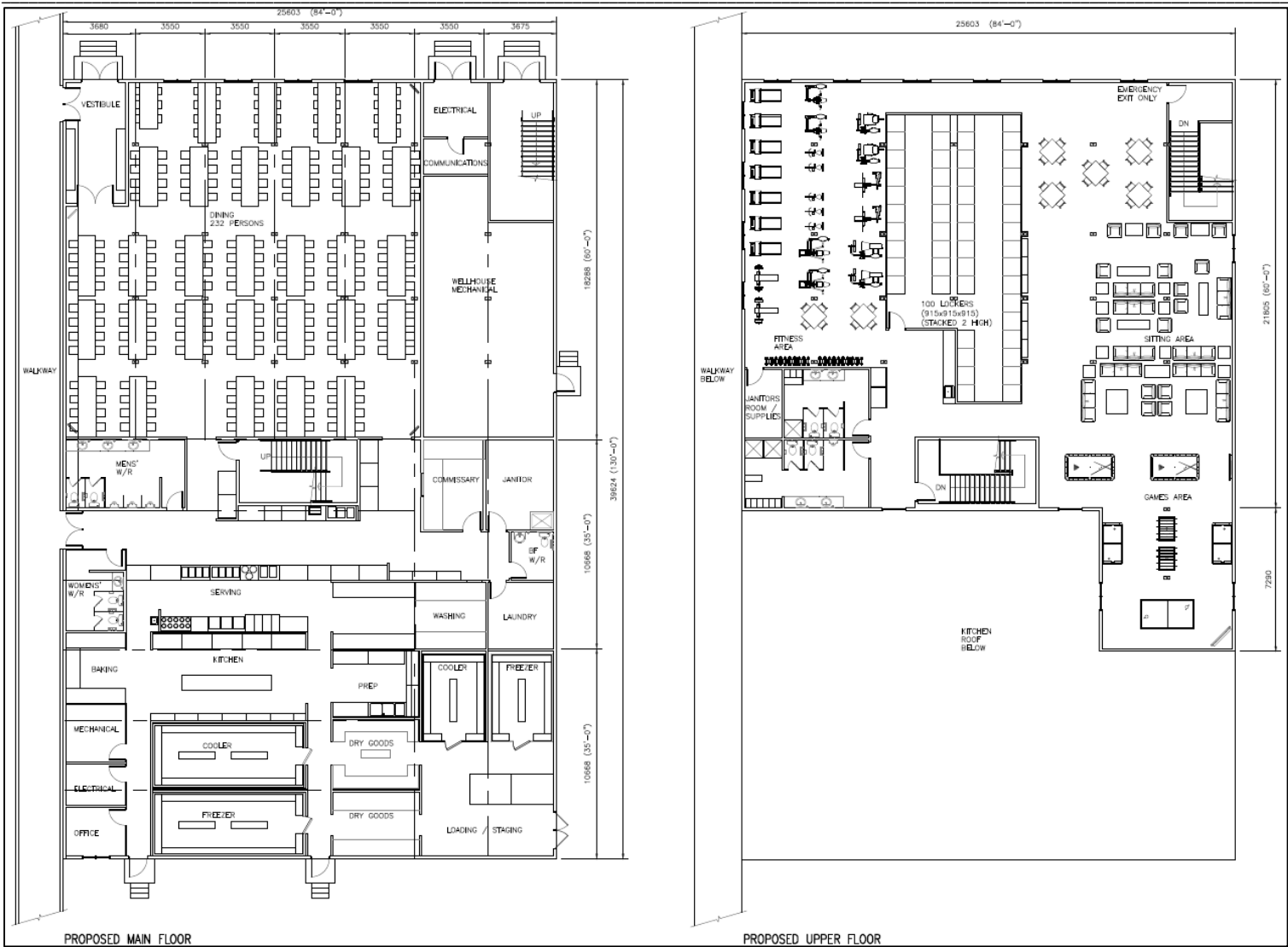


Figure 18—3: Proposed Dining Hall / Recreation Main Floor and Upper Floor Layout

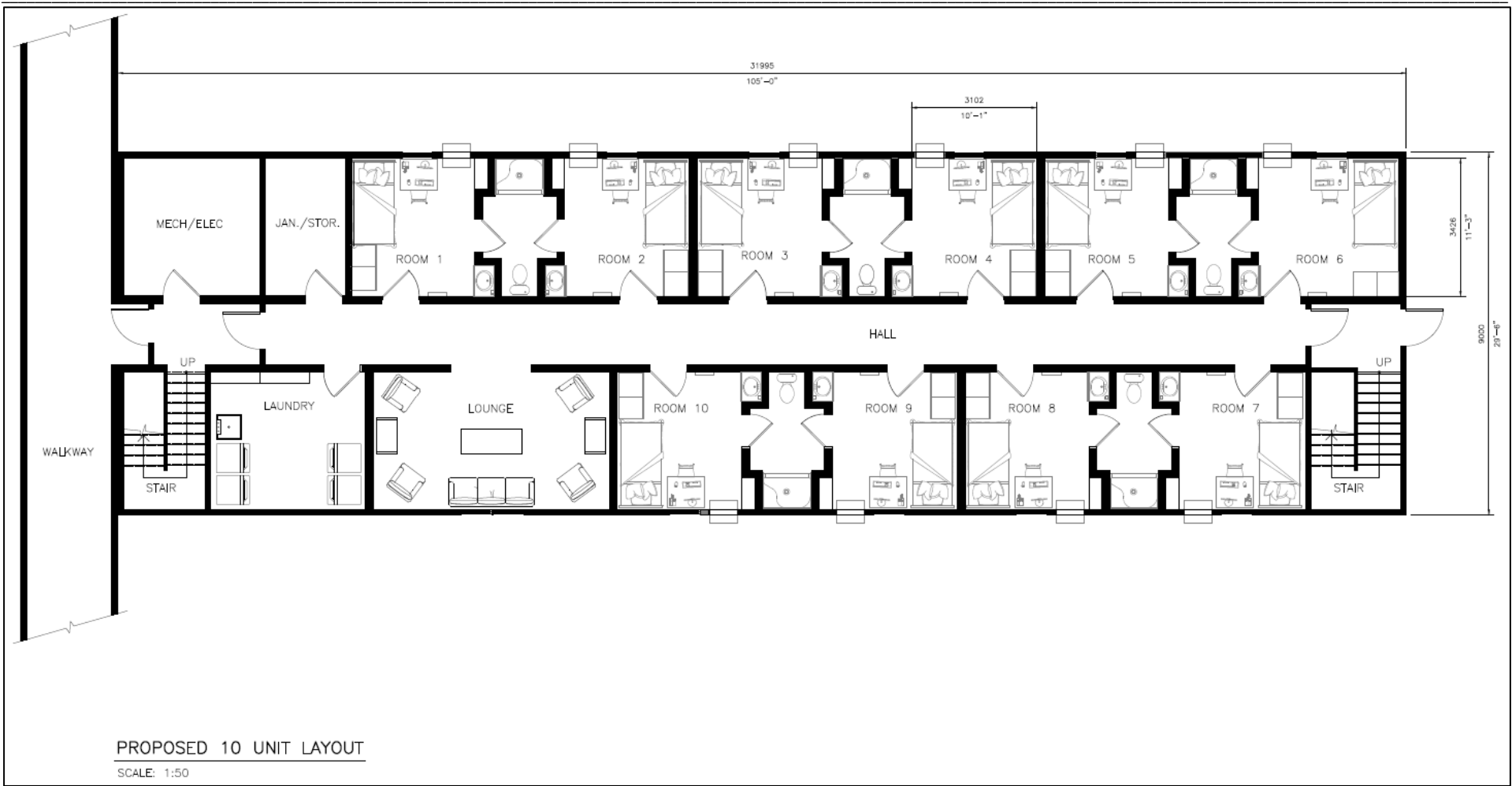


Figure 18—4: Proposed 10-Person Camp Layout

18.2.2. Site Power Distribution Upgrade

In order to accommodate the new loads required for the underground mining operations a 300 A, 5 kV feed will be routed from the spare contactor in the main mill switch room and will be routed to the portal site via a combination of direct burial and overhead power line that will be dependant of surface conditions. The new 5 kV feed will be terminated in a switch room near the portal site. The switch room will contain a line-up of 5 kV disconnect switches that will feed transformation for local surface loads and will feed the underground portion of the mine with a single 5 kV feed.

Vent fan sites will be fed from an additional switch in the tailings switch room and will be fed either by direct buried cable or via overhead power line. Each vent fan building will contain the required 5 kV switchgear, transformation, and distribution required for the fans.

19. Market Studies and Contracts

The Minto copper concentrate is considered highly desirable by smelters due to its high copper grade (plus 38% Cu) and its low contaminant levels. These attributes enable the Minto concentrate to be sold at a favourable smelter terms.

19.1. Concentrate Sales

MintoEx has a fixed copper concentrate sales contract with a metal trading company, MRI Trading AG ("MRI"). The terms of the contract are confidential; however, Capstone confirms that the appropriate terms were used in the economic model. Under the terms of the contract, MRI has the obligation to buy all of MintoEx's copper concentrate production and MintoEx has the obligation to sell all of its concentrate production to MRI. The contract is in effect to the end 2013. This study assumes that long-term treatment charges will be US\$31.00/dmt of concentrate and refining charges will be US\$ 0.031/lb of payable copper after 2013. These assumptions are based on the continuation of a general supply shortage of copper concentrate and in particular, high-quality concentrates from Minto.

19.2. Precious Metal Price Contract

MintoEx sold most of its gold and all of its silver production to Silverstone Resources in November 2008. Silverstone was subsequently bought by Silver Wheaton Corp. ("SLW") who now owns the Minto mine precious metals. SLW pays Minto US\$300/oz Au and US\$3.90/oz Ag through the mine life.

The terms of the agreement state that if Minto produces in excess of 30,000 ounces of payable gold in any one twelve-month period commencing December 1, 2010 (Excess Amount), Silver Wheaton shall be entitled to purchase only 50% of that Excess Amount.

19.3. Contracts

MintoEx has several contracts for the supply of goods and services to the mine, concentrate sales and metal price guarantees. The contracts terms and rates are within industry norms. A summary of some of the main contracts is shown in Table 19-1.

Table 19-1: Significant Minto Contracts

Company	Contract Coverage	Comments
Metal Trading Company	Concentrate purchase	Agreement to purchase all Minto concentrates up to the end of 2013. See Marketing Section
Silver Wheaton Corp. ("SLW")	Precious Metal Stream	Silver Wheaton has an agreement with MintoEx to purchase the LOM gold and silver production at Minto for US\$300/oz Au and US\$3.90/oz Ag. All of Minto's silver production is committed to SLW. Up to 30,000 Au oz/annum is committed to SLW and thereafter, 50% of the gold production is committed to SLW.
Yukon Energy Corporation	Grid power	Minto agrees to purchase power from YEC, pay for a portion of the new transmission line in exchange for YEC building the main new transmission line.
Dyno Nobel Canada Inc.	Explosive and accessory supply	Supply, storage, transportation and placement of explosives and accessories
Pelly Construction	Mining and mobile equipment supply	Pelly Construction currently performs open pit mining functions at Minto and uses its ancillary equipment for various jobs on site.
Canadian Lynden Transport Co.	Concentrate transport	Provides terms and conditions for the road transportation of concentrates to Skagway, AK. Valid until the 2 nd Qtr, of 2014.
Great Northern Oil	Diesel supply	Transport and supply of diesel fuel.
Sodexo	Camp Services	Lodging and catering

19.3.1. Contract Mining

Minto and Pelly Construction have entered an agreement to the end of 2013 for open pit mining and support activities. The unit rate per bank cubic metre ("bcm") for loading, hauling and dumping ("LHD") is based on two standard haul criteria; haul distance and road gradient. Variations to the haul criteria greater than 10% lead to change in contract costs. LHD rates are exclusive of fuel, explosives and force account charges. Drilling and blasting costs for waste vary based on powder factor ("PF") requirements for various types of material but is generally categorized as ore, waste and overburden (PF = kg of explosive per m³ of material blasted), as well as the drilling equipment used.

The blasting is conducted by another contractor, Dyno Nobel Canada, who also ships, stores, blends, delivers explosives on site, and performs blast hole loading services.

The work performed by the contractors appeared to be of good quality and they have been an integral part of the mining operation since the mine opened. Pelly and Dyno combined, maintain a workforce at Minto of between 30 and 40 people depending on the amount of work being done.

20. Environment Studies, Permitting and Social or Community Impacts

20.1. Environmental Assessment and Licensing

In the Yukon, mining projects require an environmental assessment prior to the issuance of significant operating permits for mining, including a Type A Water Use Licence and a Quartz Mining Licence.

As the Minto Project was originally submitted to DIAND for environmental assessment in December 1994, the project was assessed and a positive determination made under the Environmental Assessment Review Process Guidelines Order (EARPGO). In January 1995, the Canadian Environmental Assessment Act (CEAA) was enacted and project assessments related to the Type B Water Use Licence for the Big Creek bridge construction and Land Use Permit for the access road construction were conducted under this assessment regime by DIAND.

In April 2003, the Yukon Territory Government (YTG) assumed responsibility for management of minerals, water, lands and forestry resources in the Yukon, including the environmental assessment of development projects as part of the devolution transfer agreement with the Federal Government. Mirror environmental assessment legislation was created and subsequent assessments were then carried out by the YTG under the Yukon Environmental Assessment Act (YEAA). In November 2005, the Yukon Environmental and Socio-economic Assessment Act (YESAA) legislation created under the Umbrella Final Land Claims Agreement was formally enacted and this legislation now guides developmental assessments in Yukon. Any activities that trigger environmental assessment in the Yukon are now conducted in accordance with this legislation (see <http://www.yesab.ca/> for more information.)

Once an environmental assessment process is completed, the project moves through the regulatory permitting phase to obtain a Type A Water Use Licence, Quartz Mining Licence and other minor approvals. Water Use Licences (i.e. Type A Water Use Licence) are issued by the Yukon Water Board under the Yukon Waters Act (YWA) and *Waters Regulations*, with the approval of the YTG Minister of Executive Council Office. The Quartz Mining Licence is issued by YTG Minister of Energy Mines and Resources under the Yukon Quartz Mining Act (YQMA).

Elements of the Minto Project have undergone environmental assessment under EARPGO, CEAA and YEAA. A previous milling and mining rate increase (2008) and water management amendments were assessed under YESAA. These previous environmental assessment activities undertaken for the Minto Project are summarized in the following Table 20-1.

Table 20-1: Previous Environmental Assessments of the Minto Project

Activity	Period	Sources
Minto Explorations Ltd. Minto Mine Development, Operation and Closure	1996 to 1998	Government and company reports, 1996. DIAND EARP screening and Decision Report, Water Licence QZ96-006.
Minto Explorations Ltd. Minto Mine Development, Operation and	1999	Company report on Cumulative Effects, 1999. Quartz Mining Licence QLM-9902.

Activity	Period	Sources
Closure Cumulative Effects Assessment		
Minto Explorations Ltd. Minto Mine Development, Operation and Closure Licence Amendments – Expiry Extensions and Temporary Closure Modifications	2004 to 2005	Government and company reports, 2004. YG AO Development Assessment Branch YEAA Screening Water Licence Amendment and Quartz Mining Licence QML-0001
Minto Explorations Ltd. Mining and Milling Rate Increase, Minto Project	2008	Company Reports, 2008. YESAA DO Assessment Quartz Mining Licence QML-0001 Amendment
Minto Explorations Ltd. Water Management and Milling Rate Amendments, Minto Project	2009 to 2010	Company Reports, 2009 YESAA DO Assessment
Minto Explorations Ltd. Minto Mine Expansion- Phase IV	2010 to 2012	Company Reports, 2010, YESAB Project Proposal, August 2010

20.1.1. Current Licences

Several government agencies, both federal and territorial, are involved in reviewing, assessing, authorizing and monitoring Minto Mine in the form of regulatory and guideline based environmental instruments. The major instruments or authorizations and their attendant assessment and amendment histories are summarized below.

20.1.1.1. Type A Water Use Licence

In February 1997, MintoEx submitted a Type A Water Use Licence application (QZ96-006). The Yukon Water Board (YWB) convened a public hearing into the application in May 1997, and after deliberations by the YWB, the Type A Water Use Licence was subsequently issued in April 1998 pursuant to the Yukon Waters Act (YWA) and Regulations for the mine and milling operations. The Type A Licence and contained typical licence terms and conditions to ensure that mitigation measures identified during the environmental assessment were implemented. The expiry date for the Type A Water Use Licence was June 30, 2006. This licence was subsequently extended up to 2016, as discussed below.

20.1.1.2. Type B Water Use Licence

In August 1995, the company submitted a Type B Water Use Licence application, which was filed with the YWB for construction of the Yukon River barge landing sites, the Big Creek Bridge, and Minto Creek road culvert installations. In October 1995, a land use and quarry permit application for the access road construction was filed with DIAND Land Resources.

An integrated Canadian Environmental Assessment Act (CEAA) screening of the Type B and land use applications was completed and a positive determination was made in August 1996. Type B Water Use Licence MS95-013 and Land and Quarry Permit YA5F045 were issued in August 1996 and the initial 16 km of the Minto project access road, barge landings and Big Creek Bridge were installed in September and October 1996.

20.1.1.3. Yukon Quartz Mining Licence

In 1999, the Yukon Quartz Mining Act (YQMA) was amended and Section 139 of the Act required that all development and production activities related to quartz mining in the Yukon be carried out in accordance with a licence issued by the Minister. In June 1999, the company filed an application with DIAND Minerals for a Yukon Quartz Mining Licence, which included a cumulative effects assessment for the project to ensure that the provisions of CEAA were met. DIAND issued Yukon Quartz Mining Licence QLM-9902 in October 1999 with a licence expiry date of June 30, 2006. This licence was subsequently extended until June 30, 2016 (see chronology of amendments in the following section).

20.1.1.4. Amendments and Current Licensing

Water Use Licence QZ96-006 was amended (Amendment #1) to revise the decommissioning requirements for the project, and to request the submission of an interim plan as the project was not yet constructed. The project is still subject to Water Use Licence QZ96-006.

In addition, the Federal *Metal Mining Effluent Regulations* (MMER) under the Fisheries Act currently applies to the Minto mine. These Regulations are a law of general application and the requirements of this legislation are the responsibility of MintoEx. Generally, the Type A Water Use Licence is considered more restrictive than the MMER; however, separate reporting for effluent discharge and receiving water monitoring is required by the Federal Department of Environment Canada.

As the Type A Water Use Licence (QZ96-006), Type B Water Use Licence (MS95-013), and Yukon Quartz Mining Licence (QML-9902) were set to expire in June 2006, and in recognition of the project development delays, licence amendment applications to extend the licences to June 30, 2016 were filed with the YWB and Yukon Government (YG), Department of Energy, and Mines & Resources (EMR) in October 2004. In response to the amendment applications, YG Development Assessment Branch completed a Yukon Environmental Assessment Act (YEAA) screening of the Type A Water Use Licence using the previous EARPGO screening and issued their screening report in March 2005.

The YWB completed a YEAA screening of the Type B application and subsequently issued the amended Type B Water Use Licence (MS04-227) in February 2005. YG Development Assessment Branch completed a YEAA screening of the Type A Water Use Licence and Yukon Quartz Mining Licence using the previous EARPGO screening and issued their screening report in March 2005.

The YWB issued the amended Type A Water Use Licence (QZ04-064) in September 2005 (Amendment #2) and YG EMR issued amendments to the Yukon Quartz Mining Licence QLM-0001, Amendment No. 05-001 in December 2005 and Amendment No. 05-002 to change the mill rate to 2,500 today in October 2006. The Type A Water Use Licence (WUL) was further amended on April 6, 2006 (Amendment #3) following an application by MintoEx to address an apparent inconsistency in the original licence regarding the milling of sulphide ore.

In July 2008, the MintoEx submitted a Project Proposal to Yukon Environmental and Socio-Economic Assessment Board (YESAB) that outlined a proposed increase in the project mining and milling rate. The Mayo Designated Office (DO) issued a recommendation that the project proceed, and YG EMR as decision body released a decision document that concurred with the assessment

recommendations. Subsequently, Quartz Mining Licence QML-0001 was amended to increase the milling rate (and associated mining rate) to 3,200 tpd on July 24, 2008.

In response to exceptional precipitation received in the site area in late August 2008 and an imminent release of water from the Water Storage Pond (WSP) that did not meet water licence discharge standards, MintoEx applied on August 25, 2008 to the YWB for an emergency amendment to the Water Use License QZ96-006 under section 21 (4), c.19 of the Yukon Waters Act. The application to release 350,000 m³ of water from the WSP using the Metal Mining Effluent Regulations (MMER) effluent discharge criteria was approved and Amendment #4 to the WUL was issued on August 26, 2008.

The melting of significant snowpack accumulations in the winter of 2008/ 2009 required the retention of freshet runoff in the open pit and prompted concern about stability of the south pit wall should additional summer precipitation events need to be directed there as well. As a result, MintoEx applied again for an amendment to the Water Use Licence QZ96-006 under the same provision in June of 2009, to allow the release of water that would provide additional capacity for such an event. On June 26, 2009, the Yukon Water Board approved Amendment #5 which authorized the release of 300,000 m³ of water from the site, subject to the same MMER criteria and additional monitoring requirements.

On August 3, 2009, MintoEx received an Inspector's Direction from YG EMR to empty the pit of accumulated runoff water prior to October 15, 2009. Subsequently, MintoEx, in order to remain in compliance with both the Inspector's Direction and with its water use licence, applied for another amendment to WUL QZ96-006, again under the emergency provision of the Yukon Waters Act. The Yukon Water Board approved this amendment (Amendment #6) and MintoEx was permitted to release an additional 705,000 m³ of water from the Minto Mine site provided it met amended discharge standards.

MintoEx applied for an additional amendment (Water Use License Application #QZ09-094) to WUL QZ96-006 in June of 2010 to authorize changes to the water management plan and effluent discharge standards. A minor milling rate increase and other housekeeping items were also included. The Yukon Water Board is currently reviewing the application and information presented in a formal Public Hearing conducted December 6-8, 2010.

All of the above noted licences have an expiry date of June 30, 2016.

20.1.1.5. Assessment and Licensing for Phase VI

The expansion of the Minto Mine in the Phase VI development will require environmental assessment and major licence amendments. Environmental and socio-economic assessments under YESAA are conducted at different levels of review, depending on the project scope and thresholds of project elements. Most projects are assessed at the Designated Office (DO) level, while more complex projects are assessed at the Executive Committee (ExComm) level. Information requirements for project proposals at the ExComm level are more comprehensive than those required for DO assessments, and ExComm review and assessment timelines are longer.

Although the Phase VI development plans may not trigger an Executive Committee review (to be determined based on project details), a Designated Office reviewer may forward a project to the

Executive Committee review level if it is determined that the Project Proposal is too complex to be fairly assessed at the DO level or if significant public concern is demonstrated by the public or local First Nation during the review period.

20.2. Environmental Conditions

20.2.1. Conditions and Sensitivities

Table 20-2 below summarizes existing environmental conditions in the Minto Mine area. The information was compiled from various published and unpublished reports. This table is not intended to provide a thorough reflection of the environmental setting, but rather a succinct overview of the key environmental parameters.

A more detailed description of the environmental conditions in the Minto Mine area was presented as part of the environmental assessment and licensing process associated with the 1996 WUL application. Updates to these conditions (presented in the Project Proposal for the 2009 Water Management, Mining and Milling Rate Amendments, and the Project Proposal for the 2010 Minto Mine Expansion- Phase IV) have been based on further information collected at the Minto Mine site during the Interim Closure monitoring and recently, from monitoring associated with license conditions and operational management during mine construction and operations.

Table 20-2: Minto Mine Setting Summary

Project Area Attribute	Description
Region:	Yukon
Topographic Map Sheet:	NTS 115 I/10, 115 I/11
Geographic Location Name Code:	Minto Project
Latitude:	62° 36' N
Longitude:	137° 15' W
Drainage Region:	Yukon River
Watersheds:	Yukon River, Big Creek, Wolverine Creek, Dark Creek, Unnamed Creek B and Minto Creek.
Nearest Community:	Pelly Crossing, Yukon, approx. 33 km north on Klondike Highway.
Access:	Klondike Highway, Barge crossing on Yukon River at Minto Landing, Minto mine access road. Airstrip on site.
Traditional Territory:	Northern Tutchone, Selkirk First Nation peoples. Traditional use for hunting, trapping and fishing.
Surrounding Land Status:	Selkirk First Nation Settlement Lands and Federal Crown Land.
Special Designations:	Lhutsaw Wetland Habitat Protection Area located approx. 17 km NE of Minto Landing (outside the project area).

Project Area Attribute	Description
Ecoregion:	Yukon Plateau (Central) - Pelly River Ecoregion.
Study Area Elevation:	Rolling hills above mine site at 1131 m to 600 m at the Yukon River Valley bottom.
Site Climate:	Recorded site air temperature ranges from -43.2°C (Nov. 2006) to 25.9°C (Jun. 2006). Mean annual temp. of -3.0°C. Mean annual rainfall is 131 mm.
Vegetation Communities:	Riparian, black spruce, white spruce, paper birch, lodgepole pine, buck brush/willow and ericaceous shrubs, feather moss, sedge, sagewort grassland, mixed, aspen, balsam, and sub-alpine. Discontinuous permafrost is present on site. Site has been subject to recent forest fires.
Wildlife Species:	Moose, caribou, Dall sheep, mule deer, grizzly and black bear, varying hare, beaver, lynx, marten, ermine, deer mouse, fox, mink, wolverine, least weasel, wolf, squirrel, porcupine coyote, muskrat, otter and wood frog. Bird species include: spruce, blue, ruffed, and sharp-tail grouse, waterfowl, raptors, and a variety of smaller birds.
Fish Species:	In the Yukon River, chinook, coho, and chum salmon, rainbow trout, lake trout, least cisco, bering cisco, round whitefish, lake whitefish, inconnu, arctic grayling, northern pike, burbot, longnose sucker and slimy sculpin; In Big Creek, Chinook and chum salmon, arctic grayling and whitefish species; In Wolverine Creek, chinook salmon, arctic grayling, and slimy sculpins; In Minto Creek and project area watershed (lower reaches only), chinook salmon, slimy sculpin, round whitefish, arctic grayling.
Known Heritage Resources:	East side of Yukon River in the vicinity of Minto Landing four historic sites designated KdVc-2 (Minto landing), KdVc-3 (Minto Resort), KdVc-4 (Old Tom's Cabin), and KdVD-1 (Minto Creek).

(Table adapted primarily from Hallam Knight Piesold Ltd. 1994. Minto Project, Initial Environmental Evaluation, Supporting Volume II, Environmental Setting.)

Environmental conditions pre-mine development have been compiled, assessed and referenced in previous environmental assessments, but the environmental assessment and permitting process for the Phase V expansion will require that these conditions be further updated based on recent site monitoring program results. Specifically, baseline environmental conditions of the drainage to the north of the Minto Creek drainage will be of interest to assessors, as the Minto North deposit is located approximately 100 m into the drainage.

Although physically there will likely be minimal disturbance in this drainage from the mining activities, there is potential for there to be effects to the aquatic receiving environment. An updated Environmental Conditions report will be prepared to support the Phase V development and will update all environmental data for the project area and will be used for the assessment and permitting processes.

This watercourse and its drainage area north of Minto Creek (McGinty Creek) have been the subject of intermittent biophysical study since the 1970s, often as a reference (undisturbed) area for aquatic and geochemical investigations of Minto Creek. Additional water quality, hydrology, benthic invertebrate and fisheries investigations were all conducted in the McGinty Creek drainage in 2009 and 2010 by Access Consulting Group and a summary report on the baseline conditions will be prepared for inclusion with the Environmental Conditions report for the environmental assessment project proposal. The McGinty Creek and Minto Creek Drainages are presented in Figure 20—1 below, with the approximate location of the Minto North Deposit.

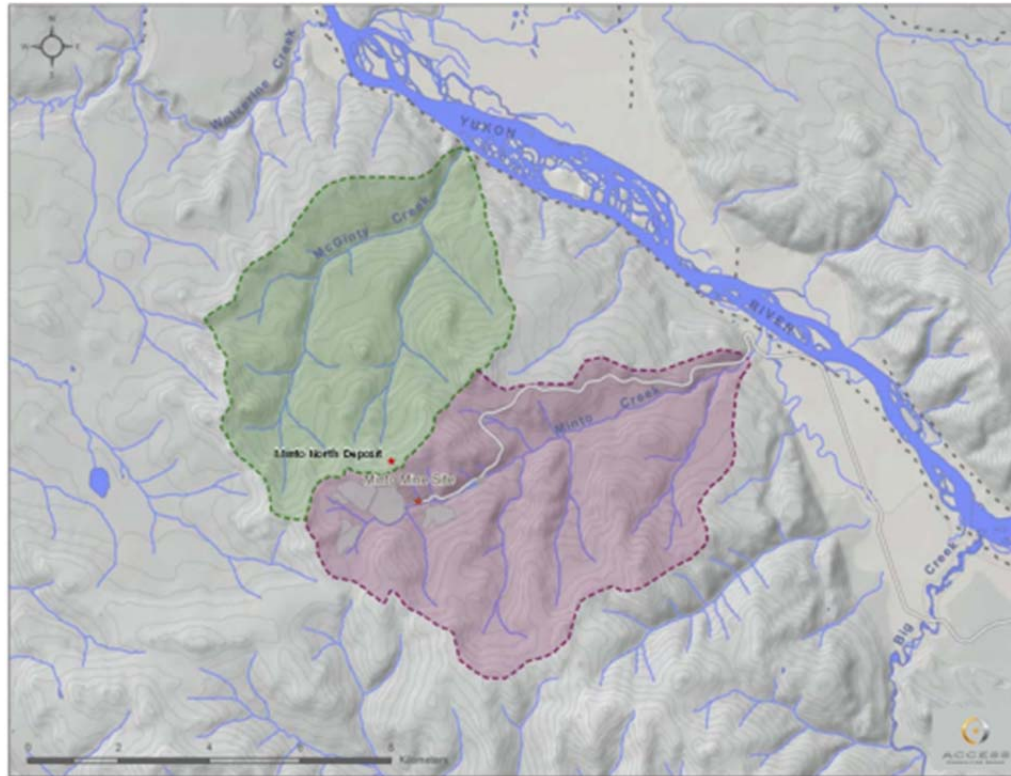


Figure 20—1: Minto Creek and McGinty Creek Drainages relative to Minto Mine Site, Minto North Deposit and Yukon River

Groundwater monitoring information is limited to two multi-level wells at the Minto site. Installation of additional wells to augment the groundwater monitoring network at the site is currently being planned, with the intention of characterizing the groundwater conditions in all areas of current and proposed mining activities.

20.3. Water Management and Effluent Discharge

MintoEx, in its original WUL application submitted in 1996, outlined a water management plan based on the limited baseline information and project projections available for the Minto Mine at the time. This information included hydrology and water balance information, operational water requirements, water storage, treatability studies and a diversion strategy for discharge to lower Minto Creek.

This 1996 WMP and supporting information formed the basis for the existing WUL QZ96-006 conditions that govern water use, treatment and effluent discharge at the Minto Mine, which include stringent effluent discharge standards relative to other major mining projects in the Yukon licensed around the same time (late 1990s). These WUL discharge standards are presented in below.

Table 20-3: Water Use Licence QZ96-006 Effluent Quality Standards for Minto Mine Project

Parameter	Units	WUL QZ96-006 Effluent Quality Standards	
		Frequency	Daily Limit
pH	pH units	weekly	6.5 - 9.0
Suspended Solids	mg/L	weekly	15
Aluminium	mg/L	weekly	0.5
Iron	mg/L	weekly	1
Copper	mg/L	weekly	0.01
Lead	mg/L	weekly	0.002
Manganese	mg/L	weekly	0.2
Nickel	mg/L	weekly	0.065
Zinc	mg/L	weekly	0.03
Total Ammonia	mg/L	weekly	1
Oil and Grease	visibility	weekly	no visible oil or grease
Rainbow Trout Acute Lethality Test	<50% mortality in 100% effluent	monthly	Pass

In the intervening period since the application, screening and issuance of the Type A water use licence, significant additional baseline and operational data have been collected. These data show that the conditions upon which the initial water management and treatment assumptions were predicated were not representative of actual conditions observed.

Since commencement of commercial production at the Minto Mine in 2007, MintoEx has responded to this discrepancy between modelled water quality and observed conditions with a number of progressively intensive and expensive measures aimed at maintaining compliance with the WUL QZ96-006 discharge criteria. These measures have only been partially and temporarily successful, and are not sustainable in the long term.

As a result, in the summers of 2008 and 2009, MintoEx sought and received authorizations to release significant volumes of stored runoff subject to adjusted discharge standards, as discussed previously.

In 2010, MintoEx was able to consistently meet the WUL QZ96-006 effluent discharge standards at low discharge rates through the adjusted use of the new site water treatment plant. Although this allowed the release of excess site water, it was first stored in the pit and required the temporary cessation of mining activities again. MintoEx revised the site Water Management Plan to include the construction and use of storm water diversions, a water treatment plant and updated project effluent discharge standards.

This water management plan provides for improved flexibility in the management of site runoff water and effluent discharge, while still protecting aquatic resources in the receiving environment. It has been noted that receiving environment conditions can change related to naturally occurring events, including a forest fire in the summer of 2010. The changing conditions in the receiving

environment have caused challenges in adhering to the obligations of the water licence effluent standards at a downstream compliance point. These conditions will be investigated as part of the preparation of the Phase V/Phase VI environmental assessment application. New water management strategies and/or new effluent standards may be proposed based on the findings of site-wide water quality predictions that will be conducted. Although the major elements of the water management revisions put forward in 2011 were designed to be functional beyond the mining of the Main Pit and into mine expansion plan currently being assessed for the Phase IV developments, the fundamentals of the approved amended plan will still apply for the Phase V and Phase VI development activities. However, the plan will still require further reassessment and adjustment (new drainage area, revisiting effluent standards) during the development of the Phase V/Phase VI environmental assessment application.

The critical considerations with respect to operational water management for Phase V/Phase VI planning will be contingency runoff storage of water requiring treatment of settling prior to discharge, and ensuring that effects to the unnamed drainage for the Minto North deposit are minimized and fully mitigated. Waste management plans for waste rock, tailings and wastewater will be scrutinized and these management plans must be fully integrated for the Phase V/Phase VI development. The currently proposed water management plan identifies the Main Pit as a contingency storage location. As the mine planning progresses for Phase V/Phase VI, this contingency storage requirement should be reassessed, as it may have implications on pit sequencing and/or waste deposition in the Main Pit.

20.4. Closure Planning

20.4.1. History

A Decommissioning and Reclamation Plan for the Minto Project was filed with the Yukon Water Board in April 2001 in accordance with WUL QZ96-006. This plan included cost estimates for closure activities. A review of the 2001 plan by YG Water Resources guided the preparation, as required in Part G – Decommissioning and Reclamation of WUL QZ96-006, of an Interim Care and Maintenance & Interim Closure Plan which was filed with the Yukon Water Board in November 2003. The Interim Plan addressed two scenarios:

- Continued care and maintenance of project infrastructure; and
- Closure issues related to the decommissioning of existing site developments at the Minto mine and reclamation of the site, including reclamation and security costs associated with the then dormant property.

The 2003 plan presented closure scenarios based on existing conditions in the construction phase at the time.

The submission of a detailed closure plan was also required under QML-0001, Section 14.1. Both the 2001 and 2003 plans were drawn upon in the preparation of the Detailed Decommissioning and Reclamation Plan (DDRP), which was submitted in November 2006 and approved in June 2007. A program was presented for site management and monitoring both during implementation of closure and after decommissioning and reclamation measures are completed.

Decommissioning and reclamation cost estimates were provided and financial security requirements were reviewed, leading to the provision of security to YG based on approved closure cost estimates.

The first update to this DDRP was submitted in September 2009. The updated DDRP addresses the long-term physical and chemical stability of the site, including reclamation of surface disturbances, and the unanticipated water quality and water quantity issues at the Minto Mine. Revised closure cost estimates for interim and final closure scenarios were also submitted and reviewed, and the security held with YG was adjusted accordingly.

Section 14.3 of QML-0001 required a subsequent DDRP update in 2011. This update reflected the Phase IV Expansion proposal and related closure strategies that were presented as part of the environmental assessment review under YESAA in 2010 and 2011. A review process under the QML occurred resulting in the approval of the current DDRP on September 9, 2011.

20.4.2. Closure Philosophy

A principle tenet of the philosophy followed during the development of the DDRP was to work towards an eventual passive closure, with eventual walk-away closure after long term chemical and physical stability has been demonstrated. It is anticipated that final determination of the effectiveness of closure measures for passive and eventual walk-away status will be the subject of review and concurrence with regulatory agencies, First Nations and the public. Under the Quartz Mining Act (QMA), the company would then apply for a certificate of closure from the YTG.

MintoEx has indicated its continued intent to implement progressive reclamation measures where possible during mine construction and operations. This approach should provide valuable reclamation success feedback for use in advanced/final closure and would reduce final reclamation liability and costs and shorten the overall reclamation implementation schedule. Progressive efforts will also help reduce slope erosion through physical slope stabilization of revegetation efforts, enhancing ultimate reclamation success.

Although the closure of the Dry Stack Tailings Storage Facility is scheduled to proceed in concert with the Phase IV development activities, no other substantial progressive reclamation was conducted on the site in the first three years of operations according to the proposed schedule in the 2007 DDRP. YTG, as a result, in 2009 increased the security requirements. In 2011, the security amount was increased further related to the increased footprint of the mine workings and an acknowledgement that active water treatment would be required during the initial phase of closure activities.

20.4.3. Current Closure Plan

Under the current plan, decommissioning of the site infrastructure will see some key diversions left in place and drainage of upper Minto Creek and minor tributaries re-established in channels where required. Proposed reclamation measures are primarily traditional in nature, i.e. re-contour, cover, and re-vegetate. This will apply to waste rock dumps, tailings, stockpile pads, lay-down areas and the mill complex and camp areas. Water treatment facilities will remain on site as long as required to maintain project water quality control, as will the main water dam. Re-vegetation prescriptions are being tested at various trial plot locations around the site to optimize revegetation success of progressive and final reclamation seeding.

20.4.4. Phase VI Closure Plan

Closure philosophies and conceptual closure measures for the Phase VI mine plan mirrored those presented in the previously submitted and approved DDRPs, but were expanded based on results of a Water Quality Prediction for Closure Conditions. This prediction indicated that additional source load mitigation measures would be required to reduce loading from the key sources – waste dumps and dry stack tailings in particular. Detailed plans to support the concepts of infiltration-reducing dump covers and passive treatment systems are currently being developed, and the cost estimate will also be adjusted accordingly.

20.4.5. Closure Planning for Phase VI Expansion

Closure philosophies and measures for the Phase VI mine plan will mirror those presented in the previously submitted and approved DDRPs. Although closure and reclamation concepts will be required for the Phase VI environmental assessment and attendant authorization amendments, it is expected that actual details (including closure cost estimates) will be presented in a subsequent revision to the DDRP to support the QML amendment. Closure measures for the site following the completion of the Phase VI mine plan are expected to generally follow those currently authorized for the Phase IV development phase.

20.5. Selkirk First Nations

On May 29, 1993, the Government of Canada, the YTG, and Yukon First Nations as represented by the Council of Yukon Indians (now the Council of Yukon First Nations) signed the Umbrella Final Agreement (UFA) after approximately 20 years of negotiation. The UFA provided a comprehensive land claim agreement for all Yukon First Nations and an outline for community based social well-being, political autonomy, and economic independence.

On July 21, 1997, Selkirk First Nation (SFN) became the fifth First Nation to sign a comprehensive land claim agreement. The Selkirk First Nation Final Agreement and the Selkirk First Nation Self Government Agreement (LCA) was negotiated by SFN, YTG and the Government of Canada. Through the LCA, the SFN was allocated 1,830 sq. miles of land over which the SFN has ownership and control. Of this land total, 930 sq. miles are Category A Settlement Lands, of which the SFN has the ownership of the surface and subsurface, including minerals, oil, and gas, and exclusive fish and wildlife harvesting rights. The balance of the land allocation is 900 sq. miles of Category B, on which SFN has ownership of surface only, and a small amount of land, (2.62 sq. miles) in the form of site-specific parcels.

Three years before the start of the land claims negotiations, the Minto and DEF mineral claims were staked by two competing exploration syndicates. These claims were extensively explored between 1971 and 1974 and feasibility studies were completed in 1975-76, but thereafter, activities ceased. Ownership was somewhat restructured in 1984 and 1989, which resulted in limited exploration in 1989, after which the property became dormant again. In 1993, MintoEx purchased the claims for the purposes of initiating mining in the area, and was active until 1999. During this time, SFN signed the LCA, which placed the MintoEx claims within Category A Settlement Lands. Recognizing that, pursuant to land claims agreement, the SFN were afforded the rights to exercise certain powers over land use and environmental protection.

MintoEx claims continue to lie within SFN Category A Settlement Lands (Parcel R-6A), where both surface and mineral rights are reserved for SFN. In addition, the mine access road lies within parcels Parcel R-6A and Parcel R-44A, and the east barge landing access point lies on Parcel R-43B. However, under the LCA, certain rights are reserved, including:

- All rights to mines (opened and unopened) and minerals (including precious and base metals) within settlement land are ceded to the Crown except on Category A lands, where mines and minerals are owned fee simple by SFN excepting pre-existing rights such as those that form the Minto property (SFN Final Agreement, Chapter 5.4.2);
- Where pre-existing rights lie within Category A land, such as the Minto mineral claims, the government will continue to administer those rights as though they were still Crown Land (SFN Final Agreement, Chapter 5.6.2) except that any royalties collected from those mineral rights will be paid to SFN (SFN Final Agreement, Chapter 5.6.3);
- A 30 m right of way within land parcels R-6A, R-40B and R-44A covering the existing access road from Minto Landing to the project, with the right to construct, maintain, upgrade and use the right of way and road for as long as the company holds its mineral rights (SFN Final Agreement); and
- The right of YTG to grant a surface lease over the mineral rights, subject to the consent of SFN, not to be unreasonably withheld (SFN Final Agreement).

If any of the claims are allowed to lapse, they could not be re-staked, and the surface and mineral rights would revert to the SFN. In September 16, 1997, MintoEx and the SFN entered a Cooperation Agreement concerning the Minto Project with respect to the development of the Minto Mine. This agreement was amended November 4, 2009. In addition to establishing cooperation with respect to permitting and environmental monitoring, this confidential document deals with other economic and social measures and communication between Selkirk First Nation and the company. This agreement will continue to guide SFN involvement in the project as mine expansion planning and development proceeds.

20.6. Metal Leaching/ Acid Rock Drainage Characterization

20.6.1. Introduction

The Phase VI mine plan will introduce the following components to the presently-permitted facilities at the Minto Mine:

- Development rock from the access drifts to the Copper Keel and Wildfire underground; and
- Tailings from processing ore from Copper Keel and Wildfire.

The geology of Copper Keel and Wildfire has been identified as mineral extensions of Area 2 / 118. Geochemical characterization of metal leaching/ acid rock drainage (ML/ARD) potential has been carried out for Area 2 / 118 as part of the Phase V PFS.

The results of the Phase V waste rock and tailings characterization for Area 2 /118 apply to Copper Keel and Wildfire extension.

The results of the Phase V ML/ARD program are shown in the following sections.

20.6.2. Phase V Waste Rock Characterization

20.6.2.1. Sample Selection

- Area 2: Two rounds of Area 2 waste rock testing were carried out.
- For the first round of testing, 36 samples were selected by SRK to include: host rock surrounding the ore horizons, unmineralized rock between the ore horizons, weakly mineralized rock, and ore grade material. Details of the sample selection process for the first round of Area 2 rock characterization, including the origins of the samples selected can be found in SRK (2007).
- Drilling in the vicinity of the southwest portion of the Area 2 Pit had not been completed at the time of the first round of Area 2 waste rock characterization. The second round of Area 2 testing was carried out in 2008 utilizing newly-available drill core from this southwest pit region. Samples were selected from drill core intervals by SRK based on metal and sulphur contents from exploration assays. Intervals were selected to target bulk waste (11 samples), mineralized waste (7 samples), and ore (2 samples) (based on metal and sulphur contents from MintoEx' exploration assays) and to ensure vertical and lateral coverage within the southwest region of the Area 2 Pit. A total of 20 samples were selected for the second round of Area 2 testing.
- Area 118: No Area 118 waste rock has been tested to determine ML/ARD characteristics. Waste rock from the Area 118 open pit is expected to have similar characteristics to Area 2 waste rock. There will be little to no waste rock brought to surface during underground mining of the Area 118 deposit, as there will be minimal quantities produced and that which is produced will be used as backfill in mined-out areas.
- Underground Development Rock: ABA testing was carried out in 2010 on drill core samples from rock along or adjacent to the alignment of the proposed ramp. Using the exploration drill hole database provided by MintoEx and the preliminary ramp alignment, SRK selected appropriate diamond drill hole intervals for testing. Intervals were selected from 2006, 2007, 2008 and 2009 drill holes, where the interval was within 20 m lateral distance of the ramp centreline and within 10m elevation of the ramp floor. Sample frequency was based on an approximate target of one sample for each 100 m length of ramp. Twenty-seven samples were tested.
- Ridgetop: The current understanding of the Ridgetop deposit geology is summarized in Chapter 8, and is similar to the geology of the Area 2 deposit. In general, there are several shallow-dipping mineralized horizons separated by barren granodiorite. Contacts between ore and bulk waste are sharp, and mineralized waste consists of portions of the mineralized zones with sub-ore concentrations of the metals of economic interest. A distinction between the geology of the Ridgetop and Area 2 deposits is the presence of a conglomerate unit within the Ridgetop pit shell. This conglomerate unit is made up primarily of detrital clasts of local granodiorite, and is likely to be geochemically similar from a ML/ARD perspective.
- Two rounds of Ridgetop waste rock testing were carried out.
- For the first round of testing, 20 drill core intervals from the 2007 Ridgetop drilling were selected from available core for ML/ARD testing by Dylan MacGregor of SRK. Sixteen

intervals of bulk granodiorite waste were selected, along with two intervals of mineralized waste and two ore-grade intervals.

- For the second round of testing, 12 drill core intervals from the 2007, 2008 and 2009 Ridgetop drilling were selected from intersections of the conglomerate unit by Dylan MacGregor and Andrew Hoskin of SRK.
- Minto North: The current understanding of the Minto North deposit geology is summarized in Chapter 8. The ore consists of shallow-dipping mineralized horizons separated by barren granodiorite, similar to the other Minto-area deposits. Contacts between ore and bulk waste are sharp. A late basaltic to andesitic dyke crosscuts the mineralized horizons; this material is barren and post-dates the mineralization. The late dyke will make up a small proportion of the Minto North waste, and it has not been characterized for ML/ARD potential.

Twenty-three drill core intervals were selected for ML/ARD testing by Dylan MacGregor of SRK. Sample intervals (18 in total) were chosen from 5 vertical diamond drill holes to provide lateral and vertical coverage of the porphyritic granodiorite that makes up the Minto North hanging wall rock (most of the Minto North waste rock will originate from excavation of the hanging wall). In addition, five drill core intervals were selected to characterize waste rock in the deposit footwall.

20.6.2.2. Testing Methods

Two rounds of ML/ARD testing were carried out on Area 2 waste rock.

- The ML/ARD testing on Area 2 samples in 2007 was carried out at ALS Chemex in North Vancouver BC. ABA analyses were carried out using the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses were performed according to ALS Chemex method ME-MS41 (aqua regia digestion followed by elemental determination by a combination of ICP-MS and ICP-AES).
- The ML/ARD testing on samples from the southwest region of the Area 2 Pit in 2008 were carried out at SGS CEMI in Burnaby BC. ABA analyses were carried out at CEMI according to the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses consisted of aqua regia digestion followed by elemental determination by ICP-MS.

Ridgetop and Minto North waste rock samples were tested for ML/ARD characteristics at SGS CEMI according to the procedures noted above for the Area 2 samples tested in 2008. Underground development rock samples were tested for ABA characteristics only by those same methods.

20.6.2.3. ABA Characteristics

Potential for development of acid weathering conditions is evaluated by categorizing waste materials based on the ratio of neutralization potential (NP) and acid potential (AP). A common categorization approach is: materials with $NP: AP < 1$ are designated as potentially acid generating (PAG); materials with $1 < NP: AP < 2$ are designated as having uncertain acid generating potential; and materials with $NP: AP > 2$ are designated as not potentially acid generating (NPAG).

The following sections summarize the ML/ARD characterization results for each Phase V pit. A plot of NP and AP values for all Phase V samples tested is shown in Figure 20—2. A line showing NP/AP

= 3 is included for reference purposes only, due to this value being referenced in the existing water licence for waste rock from the Minto Pit.

Area 2

- 34 samples of Area 2 bulk waste were tested. NP/AP values ranged from 7.6 to 180. All bulk waste samples were therefore classified as NPAG.
- 17 samples of Area 2 mineralized waste were tested. NP/AP values ranged from 0.6 to 61, with one sample classified as PAG (NP/AP of 0.6), one sample classified as uncertain (NP/AP of 1.96) and the remaining 15 samples classified as NPAG.
- Five samples of Area 2 ore were tested. NP/AP values ranged from 1.5 to 42, with two samples classified as uncertain (NP/AP of 1.5 and 1.8) and the remaining 3 samples classified as NPAG.

Ridgetop

- 16 samples of Ridgetop bulk waste were tested. NP/AP values ranged from 24 to 185. All bulk waste samples were therefore classified as NPAG.
- Two samples of Ridgetop mineralized waste were tested. NP/AP values were 4.2 and 9.9. Both samples were classified as NPAG.
- Two samples of Ridgetop ore were tested. NP/AP values were 2.1 and 10.9. Both samples were classified as NPAG.
- Twelve samples of Ridgetop conglomerate were tested. Sulphur content of all samples was at or below the detection level of 0.02% total sulphur. NP/AP values were calculated by adopting the detection level total sulphur value as a conservative indicator of AP, and those NP/AP values ranged from 41 to 125. All Ridgetop conglomerate samples were therefore classified as NPAG.

Underground Development Rock

- 27 samples of underground development rock were tested.
- Sulphide sulphur content of most of the samples is low, with corresponding AP values less than 5 kg CaCO₃ equiv. / tonne for 23 of 27 samples tested. NP/AP values ranged from 5.7 to 130. All samples with AP < 5 kg CaCO₃ equiv. / tonne were be classified as NPAG.
- Four of 27 samples tested had NP/AP values ranging from 0.5 to 1.4 would be classified as either Uncertain or PAG. Three of these four are some of the deepest samples tested (from drill holes 07SWC211, 06SWC080, and 06SWC116), and represent a location where the ramp passes through mineralization.
- All PAG and uncertain samples have copper content near 1%. This rock would either be sent directly to the mill or stockpiled for later processing.
- Overall, underground development rock results were similar to those from Area 2 rock testing, which showed that bulk rock will not generate ARD and that mineralized rock has some potential to generate ARD.

Minto North

- 18 samples of Minto North hanging wall waste were tested. NP/AP values ranged from 20 to 55. All hanging wall waste samples were therefore classified as NPAG.

- Five samples of Minto North footwall waste were tested. NP/AP values ranged from 39 to 62. All footwall waste samples were classified as NPAG.

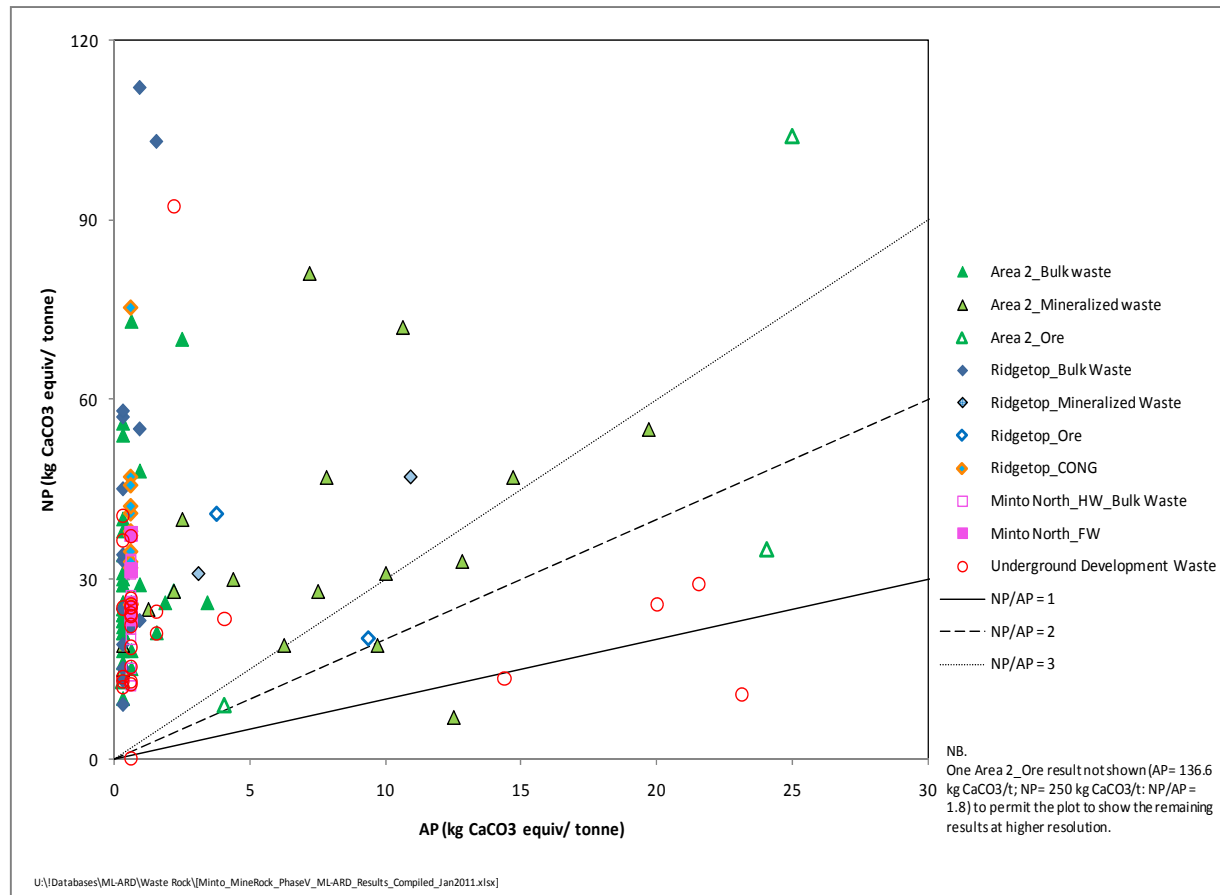


Figure 20—2: NP/AP Results for Phase V Mine Rock Samples

20.6.2.4. Elemental Content

Elemental content of mine rock tested in Phase V was compared with crustal average concentrations of granitic rocks (Price, 1997). A value of three times (3x) the crustal average concentration was used as a screen to determine whether Phase V mine rock contained anomalous elemental concentrations (based on median test results) that might indicate the potential for leaching at environmentally-significant rates. For bulk waste, median antimony concentrations in Ridgetop (median 1.1 ppm) and Area 2 waste rock (median 1.0 ppm) were reported to exceed 3x the crustal average concentration of 0.2 ppm. No other elements (copper included) had median concentrations exceeding 3x crustal average concentrations in bulk waste.

For Ridgetop conglomerate, the 12 samples tested had a median copper concentration of 153 ppm which was greater than 3x the crustal average concentration of 30 ppm. The remaining elements that were determined had median concentrations less than the 3x crustal average screening criteria.

For mineralized waste and ore samples tested, median concentrations of copper, molybdenum, and antimony exceeded 3x crustal average concentrations. Antimony concentrations in ore and mineralized waste were similar to bulk waste concentrations described above. Copper and

molybdenum concentrations were elevated in mineralized waste and ore relative to bulk waste, with median molybdenum concentrations ranging from 10 to 15x the crustal average range of 0.6 to 1.3 ppm and median copper concentrations ranging from 50 to 140x the crustal average range of 5 to 30 ppm.

The elevated copper content of mineralized waste suggests that there is a risk copper may leach from these materials at environmentally-significant concentrations. Implications for waste management are discussed below.

20.6.3. Phase V Tailings Characterization

20.6.3.1. Sample Selection

Tailings samples were selected from residues from metallurgical testing of ores from the Area 2, Area 118, Ridgetop, Minto North and Minto East deposits. The follow points summarize the samples tested.

- Area 2: Residues from locked cycle testing on ores from each of the seven discrete ore horizons (G&T, 2007) were tested for ML/ARD potential, along with a composite sample composed of 37% 272 horizon tails and 63% 280 horizon tails to evaluate the characteristics of a mixed tailings product (SRK, 2007). Ore samples were selected from drill core by MintoEx personnel, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.
- Area 118: Residues from locked cycle testing on two master composite ore samples from each of the upper and lower Area 118 ore zones (G&T, 2009a) were tested for ML/ARD potential. Area 118 ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.
- Ridgetop: Residues from locked cycle testing on master composite ore samples from the upper and lower portions of the Ridgetop East deposit were tested for ML/ARD potential. Three samples in total were tested, one from the Ridgetop East lower zone, and two from the Ridgetop East upper zone (one at a primary grind sizing of 100 μm K₈₀ and the second at a primary grind sizing of 200 μm K₈₀) (G&T, 2009a). Ridgetop East ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK, and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.
- Minto North: Residues from locked cycle testing on a single master composite ore sample from the Minto North ore zone (G&T, 2009b) were tested for ML/ARD potential. Minto North ore samples were selected from drill core by Gordon Doerksen, P.Eng., of SRK and metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.
- Minto East: Residues from locked cycle testing on a single master composite ore sample from the Minto East ore zone (G&T, 2010) were tested for ML/ARD potential. Metallurgical testing was carried out by G&T Metallurgical Services of Kamloops, BC.
- Area 2: Aliquots of rougher and cleaner tails from each ore horizon were combined, according to the 'as-produced' mass ratio, and submitted to ALS Chemex for ABA and elemental analysis. ABA analyses were carried out using an in-house version of the Sobek *et al.* (1978) procedure with sulphur speciation and additional determination of inorganic carbon content. Elemental analyses were performed according ALS Chemex method ME-

MS41 (aqua regia digestion followed by determination of 51 elements by a combination of ICP-MS and ICP-AES).

- Area 118, Ridgetop, Minto North and Minto East: aliquots of rougher and cleaner tails from each sample were combined, according to the 'as-produced' mass ratio, and submitted to SGS CEMI for ABA and elemental analysis. ABA analyses were carried out according to the Sobek et al. (1978) procedure with sulphur speciation and additional determination of inorganic carbon content; for the Minto East sample, NP was also determined by the Modified NP (MEND 1991). Elemental analyses consisted of aqua regia digestion followed by determination of 36 elements by ICP-MS.

20.6.3.2. Results

Phase V tailings samples were assigned ARD classifications based on the categories described for waste rock in the section above (PAG, uncertain, or NPAG).

All Phase V tailings tested were classified as NPAG, with NP/AP values ranging from 3.8 to 62. A plot of NP and AP values for all Phase V samples tested is shown in Figure 20—3. NP/AP = 3 and NP/AP = 4 lines are shown for reference purposes only, these values are referenced in the existing water licence for tailings from Minto Pit ore.

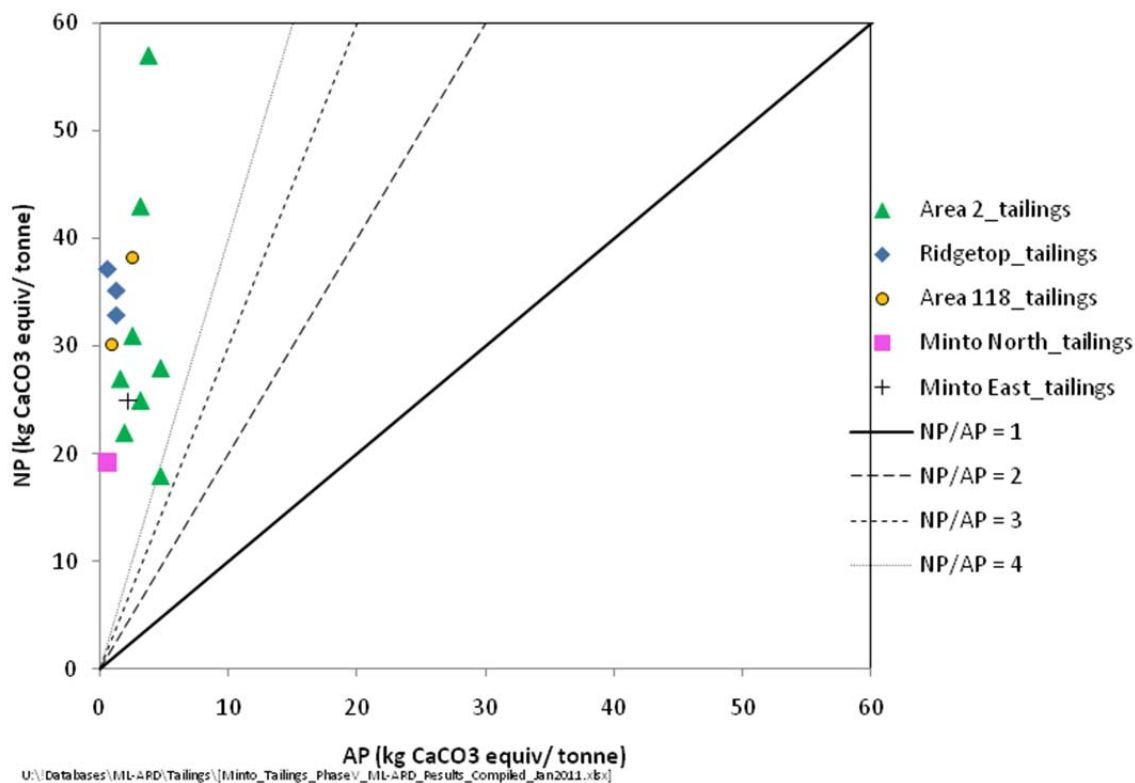


Figure 20—3: NP/AP Results for Phase V Tailings Samples

20.6.4. Implications of Phase VI ML/ARD Characterization for Waste Management

Waste rock produced during mining of the Phase V pits and Phase VI underground is expected to have a substantial excess of NP overall. As such, it is unlikely that the prevention of ARD will affect the development of management plans for Phase VI waste rock.

Detailed characterization of mineralized waste and ore from within the Area 2 Pit shell has shown that a small proportion of mineralized waste may either be PAG or have an uncertain potential to generate ARD, and it is likely that small quantities of similar material will be encountered during mining of the other Phase V pits as well. The abundant neutralizing potential present in the bulk waste is expected to consume any acidity produced locally within the waste rock dumps by small volumes of mineralized waste. The waste management plan submitted as part of the 2010 Mine Expansion- Phase IV project proposal included plans for segregation and segregated disposal of mineralized waste to reduce the magnitude of long term neutral pH metal leaching. This approach will be revisited in the development of the Phase VI waste management plan. The final disposal location for mineralized waste will either be below the long term water table elevation or in a discrete location suitable for capping with a very low permeability cover.

Due to the nature of underground excavations, it is difficult to appropriately characterize development waste in advance. Although most of the development rock samples tested was found to be NPAG, there will be a need to carry out verification testing on development rock during temporary storage at the portal laydown stockpile if the rock is to be disposed with open pit waste in surface dumps. The use of development waste as backfill for mined-out voids will be maximized, and any remaining development waste will either be co-disposed with thickened tailings in a mined-out pit or appropriately characterized and, where geochemically suitable, disposed in the upland dumps with open pit waste rock. The overall quantity of development waste will be minor in comparison with the quantity of both tailings and open pit waste rock.

Tailings produced during processing of Phase VI ore are expected to have a substantial excess of NP, and are therefore classified as NPAG. As such, it is unlikely that the prevention of ARD will affect the development of management plans for Phase VI tailings.

Neutral pH leaching of copper and other trace elements from mine rock and tailings is expected to continue to require management to meet authorized effluent quality limits for the mine during operations. The mine commissioned a contractor to design, build and operate a water treatment plant to ensure mine effluent met licensed discharge criteria; this plant operated at or above design thresholds during 2010 and 2011. It will remain in service for the duration of mining and beyond as necessary to meet effluent standards.

SRK prepared a post-closure site-wide water quality prediction as part of the Phase IV YESAB application which highlighted the need to manage neutral pH trace element leaching. This prediction has been revisited during the water licensing phase of the Phase V/Phase IV permitting process and will continue to be updated and used in the development of the Phase V/Phase VI environmental assessment application.

21. Cost Estimation

21.1. Operating Cost Estimate

Table 21-1: Average LOM Unit Operating Cost Estimates

Area	Unit	Cost Estimate
Open Pit Mining	\$/t mined	3.54
	\$/t milled	20.37
Underground Mining	\$/t milled	35.95
Total Mining (weighted average)	\$/t milled	24.68
Processing	\$/t milled	17.20
General, administration, camp, royalties	\$/t milled	10.81
Total	\$/t milled	52.69

21.1.1. Open Pit Mining

The open pit mining activities for the Minto mine were assumed to continue with the current contract mining scenario as the basis for this pre-feasibility study.

The mining unit rate was calculated based on current contract mining costs and experience at Minto. The open pit mining costs encompass pit and dump operations, road maintenance, and mine supervision. Technical services cost have been included in the G&A costs noted elsewhere in the report.

The open pit operating costs for a 1.37 Mtpa operation are presented in Table 21-2 by mining category.

Table 21-2: Average LOM Open Pit Operating Cost Estimate (Contract-Operated Fleet)

Cost Category	Estimated OPEX (\$/t mined)
Mining Operations	1.63
Drilling and Blasting	0.71
Pit Maintenance, Dewatering, Ore Stockpile	0.32
Fuel	0.87
Total	3.54

Mining operations, non-energy consumables, fuel, and miscellaneous operating costs were based on the mining fleet requirements described in the report.

21.1.2. Underground Mining

The underground operating costs were developed on an annual throughput basis. The estimate was prepared to a pre-feasibility study level of accuracy of $\pm 25\%$.

The Phase V underground mining operating expenditures were calculated for each cost category which included development, production, haulage, maintenance, mine services, and labour required for the 2,000 t/d operating mine. These expenditures were based on first principle calculations, experience, and factored costs. The Phase VI PFS reviewed these estimates and completed minor adjustments where required.

Table 21-3 shows the input data for cost estimation that were assumed in this study.

Table 21-3: Underground Operating Cost Input Data

Operating Factors	Unit	Quantity
Underground Production		
Mine Days	d/a	365
Nominal Mining Rate	t/d	2,000
Average Mining Rate	t/a	730,000
Rock Characteristics		
In Situ Density Ore	t/m ³	2.76
In Situ Density Waste	t/m ³	2.65
Swell Factor	%	50
Loose Density Ore	t/m ³	1.84
Loose Density Waste	t/m ³	1.77
Shift Data		
Working Days per Week	ea.	7
Shifts per Day	ea.	2
Shift Length	Hr	12
Shift Change	Hr	0.5
Lunch Break	Hr	1.0
Equip Inspection	Hr	0.25
Subtotal Non-productive	Hr	1.75
Usable Time per Shift	Hr	10.25
Shift Efficiency	%	85
Usable Minutes per Hour	Min	50
Hour Efficiency (50 min/h)	%	83
Effective Work Time per Shift	Hr	8.5

Productivities, equipment operating hours, labour, and supply requirements were estimated for each type of underground operation. Supply costs were based on estimated supply consumption and recent Canadian supplier's prices for drill and steel supplies, explosives, ground support, and services supplies, and were included in development and stoping costs. Maintenance consumables, such as parts, tyres, etc., as well as fuel and power were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs.

The stope production cost was estimated based on estimated cycle times for each operation, supplies and equipment requirements for an average stope size. As RAP and PPCF are development type stoping, the cycle times and costs were estimated per each type of development as shown in Table 21-4.

Table 21-4: Average Unit Development Cost

Cost Distribution	Unit	Cross-cut 5 m x 5 m	RAP Drifting 8 m x 5 m	RAP Break Through 8 m x 5 m	RAP Benching 8 m x 5 m
Equipment Cost	\$/m	\$733.54	\$454.30	\$435.09	\$275.65
Drill Steel and Bits	\$/m	\$172.52	\$224.98	\$196.97	\$131.37
Explosives	\$/m	\$329.83	\$364.20	\$264.08	\$292.76
Ground Support	\$/m	\$311.01	\$393.03	\$423.40	\$109.05
Piping	\$/m	\$69.46	\$24.37	\$0.00	\$0.00
Electrical	\$/m	\$64.91	\$64.91	\$0.00	\$0.00
Ventilation	\$/m	\$13.79	\$13.79	\$0.00	\$0.00
Miscellaneous	\$/m	\$6.89	\$6.89	\$6.89	\$6.89
Total without Labour	\$/m	\$1,702	\$1,546	\$1,326	\$816

The costs per metre of development were converted to cost per tonne of ore production. The underground stoping operating cost was estimated based on an average stope size of 8.0 m W x 5.0 m H is shown in Table 21-5.

Table 21-5: Average Underground Stopping Operating Cost Summary

Cost Distribution	RAP Drifting (\$/t)	RAP Benching (\$/t)	PPCF (\$/t)
Stope Production	\$12.94	\$7.11	\$12.94
Air Heating	\$0.63	\$0.63	\$0.63
Stope Backfill	\$0.00	\$0.00	\$1.51
Sub-total Stopping Cost	\$13.57	\$7.74	\$15.08

The cost of production development per tonne of ore was estimated based on the amount of secondary development required for stope production (Table 21-6).

The haulage cost was estimated for each production stope based on an average haulage distance, stope tonnage and stope backfill requirements (Table 21-7).

Table 21-6: Slope Development Cost

Area	Slope	Crosscut (m)	Lift / Bench (m)	Unit Cost (\$/m)	2012 (\$000's)	2013 (\$000's)	2014 (\$000's)	2015 (\$000's)	2016 (\$000's)	2017 (\$000's)	2018 (\$000's)	2019 (\$000's)	Total (\$000's)
118	101	212	238	\$1,702	765								765
	103	55	83	\$1,702				234					234
	104	38	19	\$1,702				97					97
	105	48	15	\$1,702		51	56						107
	106	55	40	\$1,702		94	68						162
	107	82	30	\$1,702		140	51						191
Area 2	201	417		\$1,702		378	477						854
	203	67		\$1,702			284						284
	204	57		\$1,702			293						293
	205	35		\$1,702				60					60
	206	40		\$1,702				68					68
	207	65		\$1,702			51	59					110
	208	140		\$1,702			238						238
	209	60		\$1,702					102				102
	210	40		\$1,702					68				68
	211	105	14	\$1,702					202				202
East	301	275	159	\$1,702			366	373					739
Copper Keel	510	252	84	\$1,702					429	143			572
	558	185	100	\$1,702					400	85			485
	600	200	100	\$1,702				170	170	170			510
	610	61	61	\$1,702							207		207
	617	88	42	\$1,702							221		221
	635	72	24	\$1,702							163		163
Wildfire	800	74	20	\$1,702							160		160
	760	167	70	\$1,702								403	403
	750	140	60	\$1,702								364	364
Total Cost		3030	1159		765	662	1,901	1,061	943	684	895	768	7,679

Table 21-7: Underground Truck Haulage Operating Cost

Area	Stope	Ramp Distance	Stope Backfill (%)	Haulage Cost (\$/t)	2013 (\$000's)	2014 (\$000's)	2015 (\$000's)	2016 (\$000's)	2017 (\$000's)	2018 (\$000's)	2019 (\$000's)	Total (\$000's)
118	101	1,020	16	\$2.70	1,054	733	-	-	-	-	-	1,787
	103	1,020	29	\$2.80	-	-	142	-	-	-	-	142
	104	1,580	0	\$3.04	-	-	-	47	-	-	-	47
	105	1,900	0	\$3.46	-	-	28	-	-	-	-	28
	106	1,880	0	\$3.46	-	-	105	-	-	-	-	105
	107	1,800	0	\$3.32	-	-	41	-	-	-	-	41
2	201	1,410	33	\$4.16	-	1,595	-	-	-	-	-	1,595
	203	1,680	5	\$3.23	-	-	882	-	-	-	-	882
	204	1,400	24	\$3.81	-	350	177	-	-	-	-	526
	205	1,980	0	\$3.59	-	-	-	144	-	-	-	144
	206	1,760	0	\$3.32	-	-	-	37	-	-	-	37
	207	1,620	0	\$3.18	-	-	-	72	-	-	-	72
	208	1,760	0	\$3.32	-	-	-	69	-	-	-	69
	209	1,790	0	\$3.32	-	-	-	78	-	-	-	78
	210	1,890	0	\$3.46	-	-	-	68	-	-	-	68
	211	1,88,	12	\$3.60	-	-	-	191	-	-	-	191
East	301	2,380	33	\$4.46	-	-	1,493	1,764	-	-	-	3,257
Copper Keel	510	2,858	13%	\$4.14	-	-	-	-	348	162	-	510
	558	2,412	32%	\$3.87	-	-	-	197	317	-	-	514
	600	2,565	32%	\$4.14	-	-	-	435	2,414	2,097	-	4,946

Area	Stope	Ramp Distance	Stope Backfill (%)	Haulage Cost (\$/t)	2013 (\$000's)	2014 (\$000's)	2015 (\$000's)	2016 (\$000's)	2017 (\$000's)	2018 (\$000's)	2019 (\$000's)	Total (\$000's)
	610	2,494	6%	\$4.14	-	-	-	-	-	121	-	121
	617	2,731	5%	\$4.14	-	-	-	-	-	313	-	313
	635	2,607	32%	\$4.14	-	-	-	-	-	332	-	332
Wildfire	800	400	6%	\$2.38	-	-	-	-	-	-	111	111
	760	600	32%	\$2.38	-	-	-	-	-	-	631	631
	750	900	0%	\$2.38	-	-	-	-	-	-	130	130
Total Cost				\$3.59	1,054	2,678	2,867	3,102	3,079	3,025	873	16,679

As per the Phase V PFS estimate by SRK,

“The mine services cost was estimated based on equipment working time and materials supply required for ventilation, compressed air, transportation of personnel and materials, mine and road maintenance, mine dewatering, and underground construction.

The mine maintenance cost was estimated based on required maintenance equipment, tools and supplies for maintenance shop. Maintenance consumables, such as parts, tyres, etc., were included in equipment operating costs and are part of mine development, stoping, haulage, or services costs. The maintenance labour costs were included in the overall underground mine labour costs.

Mine safety and training costs were estimated based on the number of underground mine personnel and required personal protective equipment, first-aid and safety supplies, mine rescue, and safety training.

The underground labour requirements and rates used for determining the overall mining cost were based on experience for similar operations of this size, and were divided into salaried and hourly personnel. The labour costs include overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage and educational assistance.” SRK 2011

Table 21-8: Underground Salaried Personnel List

Staff Description	Quantity
Mine Superintendent	1
Senior Mining Engineer	1
Mine Ventilation/Project Engineer	1
Geotechnical Engineer	1
Mine Technician	1
Surveyor	2
Geologist	1
Geological Technician	1
Mine Rescue/Safety/Training Officer	2
Mine Captain	1
Mine Supervisor/Shift Boss	4
Sub-total Mine Operating Staff	16
Maintenance Superintendent	1
Mechanical/Electrical G. Foreman	1
Maintenance Planner	2
Maintenance Supervisor/Shift Boss	2
Sub-total Mine Maintenance Staff	6
Total Mining Staff	22

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 11 hrs/day with 4 crews on rotation.

Table 21-9: Underground Hourly Personnel List

Staff Description	Quantity
Jumbo Operator	12
Ground Support/Services	16
Blaster	4
LHD Operator	16
Truck Driver	16
Utility Vehicle Operator/Nipper	4
General Labourer	4
Grader Operator/Road Maintenance	2
General Helper	4
Sub-Total Mine Operating Labour	78
Lead, Mechanic/Electric	2
HD Mechanic, mobile	4
Mechanic, stationary	2
Electrician	4
Tireman/Instrument Man	2
Welder	2
HD Mechanic Apprentice	2
Dry/Lampman/Bitman	2
Sub-Total Mine Maintenance Labour	20
Total Mine Labour	98

Summary of total underground mining operating cost by year is shown in Table 21-10.

Table 21-10: Underground Mining Operating Cost Summary

Item	Total Cost (M\$)	2012 (M\$)	2013 (M\$)	2014 (M\$)	2015 (M\$)	2016 (M\$)	2017 (M\$)	2018 (M\$)	2019 (M\$)	Average per year (\$/t)
Development	7.68	0.77	0.66	1.90	1.06	0.94	0.68	0.90	0.77	1.76
Stoping	59.86		5.01	10.11	9.94	10.19	10.08	9.56	4.97	13.73
Truck Haulage	16.68		1.05	2.68	2.87	3.10	3.08	3.03	0.87	3.82
Mine Services	12.42		1.19	2.27	2.29	2.28	2.22	1.43	0.74	2.85
Operating Labour	36.23		3.91	5.64	5.93	6.30	6.38	6.20	1.87	8.31

Item	Total Cost (M\$)	2012 (M\$)	2013 (M\$)	2014 (M\$)	2015 (M\$)	2016 (M\$)	2017 (M\$)	2018 (M\$)	2019 (M\$)	Average per year (\$/t)
Maintenance Labour	10.06		1.31	1.69	1.58	1.68	1.70	1.65	0.45	2.31
Supervisory Labour	13.85		1.73	2.23	2.08	2.21	2.24	2.17	1.19	3.18
Total Operating Cost (M\$)	156.78		14.87	26.52	25.75	26.71	26.38	24.93	10.85	35.95
Total Operating Cost (\$/t)	35.95		38.80	36.19	34.89	36.26	36.87	35.84	30.18	35.95

21.1.3. Processing Costs

The total process operating costs were developed in Canadian dollars (C\$) on an annual throughput basis. An operating cost estimate was generated for the current plant and formed a baseline for projecting the operating cost for the plant upgrade scenario. This baseline was verified against the Minto plant operating costs budget for 2012.

A summary of the average operating costs per tonne of ore treated for the Project is outlined in Table 21-11. The costs were divided into the key cost centres and all figures are as of the last quarter 2011 (calendar year).

Table 21-11: Estimated Average LOM Operating Costs (\$/t)

Summary	Year 2012 (3,750 tpd)	Year 2013 – 2021 (3,750 tpd)
Labour	7.83	7.41
Power	2.52	2.02
Reagents and Consumables	3.22	3.22
Secondary Contract Crushing	1.07	1.07
Other Maintenance Materials	1.52	1.52
Assay and Met Lab	1.01	1.01
Concentrate Handling	0.54	0.54
Tails filtration and dry stacking	0.64	0.00
TOTAL \$/t	18.34	16.79

The operating costs are considered to have an overall accuracy in the order of $\pm 25\%$. The assumptions listed in this section require validation during a subsequent detailed engineering phase of the project.

The calculated operating cost for the Minto process plant in 2012 is based on an annualised throughput of 1,368,750 tonnes was \$18.34 /t. The 2012 operating cost accommodates the tailings filter and dry stacking operation placed on stand-by at the end of October.

The calculated operating cost for the plant upgrade based on an annualised throughput of 1,368,750 tonnes was \$16.79/t. The reduction in operating cost (\$/t) is primarily due to the cease operation of dry stacking circuits. Some costs have remained for the tailings dewatering tank.

Secondary contract crushing costs have remained for this LOM analysis though only a small portion of the mill feed will undergo contract crushing. Minto is committed to eliminating secondary contract crushing by continuing its “Mine to Mill” program to increase fragmentation and installation of a screen plant. It is anticipated that contract crushing will be eliminated; however, for the purposes of this study, a conservative approach was taken.

21.1.3.1. Basis of Plant Operating Cost Estimate

The operating cost estimate was developed from a number of sources. Cost determinations were based on fixed and variable components relating to ore throughput and plant flow sheet. The source of data used for the operating cost estimation is summarised in Table 21-12.

Table 21-12: Derivation of Plant Operating Costs

Cost Category	Source Of Cost Data
Labour	Manning schedules and rates provided by MintoEx (2012 budget).
Power	Consumption from load estimate and power unit rate from MintoEx.
Reagents	Consumptions and unit prices from MintoEx 2012 budget and test work results on new deposits.
Consumables	Consumptions based on actuals as reported by MintoEx; unit prices from actuals as reported in the 2012 budget by MintoEx.
Maintenance Materials	Based on actuals as reported in the 2012 budget by MintoEx
Contract Secondary Crushing	Based on actuals as reported in the 2012 budget by MintoEx.
Tailings Filtration	Based on actuals as reported in the 2012 budget by MintoEx.
Assay and Metallurgical Laboratory	Based on actuals as reported in the 2012 budget by MintoEx.

Operating costs not considered in this section are listed as follows:

- General and administration costs;
- Community and environment costs;
- TSF construction;
- General site environmental management costs;
- Concentrate handling (including sea freight and insurance);
- Concentrate smelting and refining; and
- Government fees and charges.

Other miscellaneous items not considered in this section include:

- Commissioning support (included in capital estimate) and plant start-up labour costs;

- Sustaining capital;
- Ongoing exploration;
- Insurances;
- Inflation;
- Import duty and applicable taxes;
- Royalties;
- Interest and finance charges; and
- Contingency.

21.1.3.2. Plant Operating Cost Estimate Inclusions

Included in the operating cost estimate are:

- Labour for supervision, management and reporting of onsite organisational and technical activities directly associated with the processing plant;
- Labour for operating and maintaining plant mobile equipment and light vehicles, process plant and supporting infrastructure;
- Costs associated with direct operation of the processing plant, including all fuels, reagents, consumables and maintenance materials;
- Fuels, lubricants, tyres and maintenance materials used in operating and maintaining the plant mobile equipment and light vehicles;
- Operation of the TSF, including tailings discharge and management and return water, excluding construction and wall lifts;
- Cost of power that is supplied from the local hydro-power grid;
- Operation of raw water supply facility from site rivers;
- Labour and operational costs for the metallurgical and assay laboratories; and
- Labour and reagents for the future contract water treatment plant.

21.1.3.3. Labour

Labour costs for the plant were provided by MintoEx as part of the 2012 budget. The labour costs include all cost of travel, overtime and shift premiums, leave pay, bonuses, pension and superannuation benefits, insurance coverage, educational assistance and supply of uniforms and personal protective equipment.

A mill labour operating costs are summarised in Table 21-13.

Table 21-13: Site Labour Cost Summary

Labour (\$M/year)	Year 2012 (3,750 tpd)	Year 2013 – 2022 (3,750 tpd)
Mill Operations	5.4	4.8
Mill Maintenance	5.3	5.3
TOTAL \$M/y	10.7	10.1
TOTAL \$/t	7.83	7.41

21.1.3.4. Power

Power will continue to be supplied to the mine site from the local hydro power grid. The cost of power was based on the MintoEx 2012 budget unit power rate value of \$0.11 /kWh.

The power requirements for the current plant were developed based on a specific power load of 30kWh/t for the first ten months of 2012, while the tailings filtration circuit is in operation. From November 2012 to 2022 the power load is reduced to 26kWh/t. The reduction in power load considers the standby operation of the filtration circuit which will be slightly offset by the plant modifications to accommodate the increase in throughput.

The plant power consumption is expected to vary slightly over the life of the mine primarily due to the variable comminution characteristics of the ore and resulting change in comminution energy requirement. A summary of power costs by area for the plant site is given in Table 21-14.

Table 21-14: Process Plant Power Cost Summary

Power Costs (\$/t)	Year 2012 (3,750 tpd)	Year 2013 – 2022 (3,750 tpd)
TOTAL \$/t	\$2.58	\$2.02
TOTAL \$M/y	\$3.53	\$2.77

21.1.3.5. Reagents and Consumables

Reagent consumptions rates used for the estimate were based on the Minto 2012 budget.

Reagent consumptions will vary according to metallurgical and production parameters. Generally the consumption rates used are in line with the average consumptions from the current plant performance.

Reagent unit costs were based on the actual unit rates used in the 2012 budget. The rates are freight in store (FIS) at the Minto mine. The site is reviewing other reagents than AM28 and NaSH that are required to enhance recovery associated with oxide and partially oxide ores. Therefore, there is good potential to decrease reagent expenditures. For the purposes of this study, a conservative approach was taken.

Plant consumables include major items, such as crusher and mill liners and grinding media. Consumption rates were estimated from the 2012 budget consumption rates. The cost of secondary crusher liners for the current contract secondary crushing plant are included in the total cost (\$/t) estimated for the reagent and consumables.

The SAG and ball mill liner consumption rate is a function of the power drawn by the respective mills and the ore hardness properties. The mills on site are currently operated at or near maximum power draws and therefore it is not expected that the liner wear rates will increase in the future when treating moderately abrasive ores.

Table 21-15 shows the annual reagents and consumables operating costs for the LOM.

Table 21-15: Reagent and Consumables Operating Costs

Reagents and Consumables	Plant Cost (\$M)	\$/t
Reagents	1.51	1.10
Grinding Media	1.43	1.05
Liners	0.96	0.70
Miscellaneous Supplies	0.50	0.37
Total	4.4	3.22

21.1.3.6. Maintenance Consumables

Maintenance consumables include materials, tools/miscellaneous maintenance and specialized contractor costs for the purposes of estimating the operating cost. The maintenance labour costs were included in the overall plant labour costs as previously reported.

The total fixed cost estimated for maintenance consumables from 2012 to 2022 was \$2.1 million per year.

The maintenance material costs include:

- Mechanical equipment replacement parts;
- Pipes and fittings;
- Electrical equipment and replacement parts; and
- Instrumentation equipment and replacement parts.

21.1.3.7. Assay and Metallurgical Laboratory

The operating cost for the Assay and Metallurgical Laboratory was estimated based on the 2012 budget cost as supplied by MintoEx. A fixed cost of \$1.38 million per year was used for the operating cost calculations for both plant throughput scenarios.

21.1.3.8. Tails Filtration and Dry Stacking

The tails filtration and stacking plant is due to operate until July 2012 at which point it will be placed on standby. From July 2012 onwards tailings will be thickened and pumped direct to the open cut pit.

The following costs were excluded from the tails filtration and dry stacking operation but included elsewhere:

- Reagents for the tails filtration area and diesel for dry stacking of the tails, The cost for these items are included in the reagent costs Section;
- Power for the tailings filtration plant. This cost is included under power costs; and
- MintoEx supplied labour. This cost is included under labour costs.

The following costs were included in this estimation:

- Contractor labour and equipment;

- Filter cloths, scrapers and operating consumables;
- Mechanical replacement parts; and
- Lubes and glycol.

Tails filtration and dry stacking were removed from the plant flow sheet for the upgrade scenario based on backfilling the Minto main pit with tailings directly from the tailings thickener underflow pumps.

21.1.4. General and Administration Costs

The operating cost for general and administration is based on the 2012 budget as supplied by MintoEx. Camp services are contracted out to Sodexco. Site Services include light vehicle maintenance, access road maintenance, barging operations, and waste management services. Technical services include mine engineering, geology, surveying, geotechnical and environmental services. Administration services include site management, health and safety, human resources and information technology services. General, camp and administration costs are higher early in the mine life to reflect the ongoing costs of the open pit and underground operations. However, once mining ceases, G&A costs are reduced. Table 21-16 highlights the average LOM costs of general and administration.

Table 21-16: Average LOM General and Administration Costs

Item	Estimated OPEX (\$/t milled)
Camp	1.84
Site Services	0.48
Technical Services	2.44
Administration	6.05
Total General and Administration	10.81

21.1.5. Smelting and Refining Costs

Smelting and refining costs are fixed in the copper concentrate sales contract with MRI Trading AG (“MRI”). The terms of the contract are considered to be confidential but are applied correctly in the economic model.

21.1.6. Royalties

MintoEx pays royalties to the Selkirk First Nation (“SFN”). The SFN agreement is based on a percentage of net smelter return (“NSR”) and is incremented based on copper price. The specifics of the royalty agreement are not shown due to confidentiality but are applied correctly in the economic model.

21.1.7. Taxes

Federal and Provincial tax calculations start with the before tax cash flow amounts from the cash flow portion of the model and essentially deducts the cost of building the mine and mill (Class 41 UCC, CEE and CDE) as would be expected over the life of the mine as allowed by the Canadian tax

rules. Generally Class 41 UCC and CEE can be deducted 100% against profit from the mine while CDE can only be deducted on a declining balance basis at 30% per year. The losses that are generated in the first few years of mine operation are deducted against income in later years.

The Yukon Quartz Mining Royalty (“Yukon mining tax”) is a much different tax calculation than would normally be expected. It also starts with before tax cash flow from the cash flow portion of the model and deducts depreciation at 15% per year on a straight- line basis for the mine capital assets and mill capital assets. It deducts deferred pre-operating costs that are not capital assets on a unit of production method. The Yukon mining tax does not have a loss carryover or carry back provision. Taxes are paid at rates that increase as income increases to a maximum of 12%.

The opening balances for the tax pools for both taxes are included in the cash flow model.

Since the model is based on operating cash flow the actual tax results may differ between periods from the model as concentrate shipment dates vary from the model.

21.2. Capital Cost Estimation

Table 21-17: Summary of Capital Costs for over the LOM

Area	Unit	Cost Estimate
Open Pit Mining	M\$	2.5
UG equipment (fixed and mobile)	M\$	33.4
UG development	M\$	34.8
Process plant	M\$	10.3
Site Infrastructure	M\$	18.7
Permitting	M\$	3.3
TOTAL CAPITAL COST	M\$	102.7

Mine closure costs were estimated to be \$25M. In addition, a sustaining cost allowance of \$5.6M has been allocated between 2017 and 2021 within the cash flow analysis. Equipment already purchased has not been included in this estimate.

21.2.1. Open Pit Mining

All costs are expressed in Canadian dollars, with no allowance for escalation. The estimate was prepared at a pre-feasibility study level of accuracy of $\pm 25\%$.

The open pit mining capital cost estimate was based on the purchase of drilling equipment for owner-operated waste drilling.

The cost of the owner operated waste drill is approximately \$2.5 million dollars.

21.2.2. Underground Mining

All costs are expressed in Canadian dollars, with no allowance for escalation or interest during construction. The estimate was prepared at a pre-feasibility study level of accuracy of $\pm 25\%$.

The underground mining capital cost estimate was based on the following:

- Underground mining equipment list;
- Budget quotes for the major equipment obtained from equipment manufacturers;
- Cost for drilling and hauling equipment;
- Budget quotes for raise development obtained from a contractor;
- Western Mining estimation references; and
- Preliminary project development plan.

Mining capital was divided into equipment capital cost and mine development cost categories.

21.2.2.1. Equipment Capital Cost

The purchase of a permanent mining equipment fleet would be required as the underground mining operation will be performed by the owner.

It was assumed that for a mine life of 7 year there is no requirement to replace the mobile equipment due to a qualitative regular maintenance program.

Underground mining equipment capital cost by year is summarized in Table 21-18.

Table 21-18: Underground Mining Equipment Capital Cost Summary

Item	Year 2012	Year 2013	Year 2014	Year 2015	Year 2016	Year 2017	Year 2018	Year 2019	Total *
Mobile Equipment	3.6	6.9	0.8	2.6					13.9
Ventilation	0.2	0.3	7.7	1.3	1.2		0.4		11.1
Grading / Ground Prep	0.2								0.2
Dewatering	0.2	0.3							0.5
Electrical	0.9	0.5					0.4		1.8
Miscellaneous**	3.1	0.8	0.1						4
Indirects	0.5	1	0.3						1.8
Total*	8.7	10	8.8	3.9	1.2		0.8		33.4

**Totals may not add exactly due to rounding **Includes 2011 Capital Carry-Over*

21.2.2.2. Development Capital Cost

All development in year 2012, including ore development, is included in capital costs as that period is considered as pre-production. During mine production, the decline development, raise development, and ventilation drifts were considered as capital costs, but stope crosscuts development was included in the mining operating cost.

It was assumed that portal and initial ramp development will be completed by contractor. Once, underground production commences, jumbo development will be transferred to owner-operation and raise development by contractor. The estimated owner costs per metre of development excluding labour were applied for all jumbo development headings and the contractor rates were used for raise development. In year 2012, hourly and salaried labour, mine services and ore haulage costs were included in the capital cost.

Underground mine development capital costs is shown in Table 21-19.

Table 21-19: Underground Development Capital Cost

Item	Year 2012	Year 2013	Year 2014	Year 2015	Year 2016	Year 2017	Year 2018	Year 2019	Total*
Declines and Lateral Development	1.4	3.6	0.5	6.3	3.4	2.5	3.9		21.6
Vertical, Inclined Development		2.3	0.4	2.7			1		6.4
Labour	1	2.1	1						4.1
Development Contingency	1.1	1.1	1	0.1					3.3
Total*	3.6	9.1	2.1	9.1	3.4	2.5	4.9		34.6

*Totals may not add exactly due to rounding

21.2.3. Processing Plant

The plant capital cost estimate is presented in Canadian dollars (C\$) and has an overall accuracy of $\pm 25\%$ as of the fourth quarter 2011. Following is a breakdown of capital works planned for the Minto Mine.

The scope of work planned for 2012 includes:

- Installation of a new crushed ore scalping screen with belt feeder;
- Upgrade scavenger cell mechanisms and launders, and replacement of the existing 2nd cleaner cells with new tankcell-10 cells;
- Rerouting of the tailings thickener underflow pipework to bypass the tailings filter circuit;
- A new 2,600 meters long high density poly-ethylene tailings (HDPE) pipeline from the tailings thickener to the Minto Main pit; and
- Installation of a new regrind circuit including regrind tower mill, new cleaner flotation cells and associated secondary equipment (e.g. pumps etc.)

The scope of work planned for 2013 includes:

- Upgrade 3rd Cleaner cells

The estimate for upgrading the primary crushing area, flotation circuit, rerouting the tailings thickener pipework, and installation of the regrind mill, 1st cleaners and modification to the existing coverall building were supplied by B. Ross Design Inc. ("BRDI").

The estimate for the 2,600 meter tailings line was supplied by Ausenco and sourced from the Phase IV PFS estimate. Table 21-20 shows a summary of the costs which exclude any escalation or foreign currency fluctuations and are current day costs only.

Table 21-20: Cost Breakdown Summary – Process Plant

Facility	2012	2013	2014	2015	2016	Total*
Primary Crushing Area	1.6		0.2	0.8	1.9	4.5
Flotation Circuit Area	1.7	1.8				3.5

Facility	2012	2013	2014	2015	2016	Total*
Tailings and Dewatering Area	0.5					0.5
Miscellaneous	0.4		1.0			1.4
EPCM	0.5					0.8
Total*	4.6	1.8	1.2	0.8	1.9	10.3

**Totals may not add exactly due to rounding*

21.2.4. Site Infrastructure

The capital cost estimated includes the following:

- New camp upgrades;
- Safety supplies;
- Administration upgrades;
- Site Services upgrades;
- Technical Services projects;
- EPCM costs; and
- Permitting costs

The capital costs were taken from the 2012 MintoEx budget. Excluded from the capital cost estimate:

- Mining, mining vehicles/equipment, mine infrastructure;
- Owner's costs for the project;
- Owner's contingency, owner's costs, escalation and foreign currency fluctuation; and
- On-going, future or deferred capital costs.

Table 21-21 shows a summary of the costs which exclude any escalation or foreign currency fluctuations and are current day costs only.

Table 21-21: Site Capital Cost Estimation – Minto Mine

Facility	2012 (M\$)	2013 (M\$)	2014 (M\$)	2015 (M\$)	2016 (M\$)	Total*
Camp Upgrade	6.9					6.9
Safety	0.3					0.3
Site Services	1.0	0.3	0.2			1.5
Administration	0.7					0.7
Water Treatment Plant	2.9		2.8		0.3	5.6
Technical Services	1.6	0.3	0.3	1.00	0.4	3.6
Permitting	1.2	1.2	0.4	0.4	0.3	3.3
Total*	14.1	1.8	3.7	1.4	0.9	21.8

**Totals may not add exactly due to rounding*

22. Economic Analysis

22.1. Assumptions

An engineering economic model was compiled by Capstone based on the Phase VI LOM plan.

The Phase VI economic model assumes a constant copper price of US\$3.25/lb and a constant exchange rate of C\$1.00:US\$1.00 US. This case is slightly lower than current market prices.

It should be noted that payable gold, up to 30,000 oz/year, and all payable silver in the LOM plan are under contract to Silver Wheaton ("SLW") at a price of US\$300/oz and US\$3.90/oz respectively.

Other main economic factors include:

- A discount rate of 8.0%;
- Closure allowance of \$25M;
- Nominal 2012 dollars;
- Payment of capital from operating revenue;
- Sustaining cost allocation of \$1.1M annually between 2017 and 2021;
- No inflation; and
- Costs, revenues and taxes calculated for each period in which they occurred rather than at the actual date of payment.

It must be noted that spot copper prices on the effective date of this report was US\$3.43/lb.

22.2. Economic Results

The LOM economic model results are shown in Table 22-1.

It must be noted that the net present value ("NPV") calculations in the economic models were done using 2012 as the starting year and do not take into account approximately \$150M in capital spent for initial plant and mine construction (sunk costs) nor the revenue derived from operations from 2007-2011. As a result, the economic analyses show very high returns on the planned, future capital investments.

Results show the NPV at a 8.0% discount rate ("NPV_{8.0%}") to be \$278M before tax. Table 22-1 shows the NPV results at various discount rates for each case.

Table 22-1: Discount Factors and Related Net Present Values for All Cases

Case	Discount Rate	Pre-tax NPV (M\$)	After-Tax NPV (M\$)
Phase VI	0%	430	290
	5%	325	223
	8.0%	278	192
	10%	251	174

An incremental analysis was done to compare the economic results of Phase VI PFS and Minto's January 2012 LOMP. The January 2012 LOMP results were subtracted from the Phase VI results

and are summarized in Table 22-2. The comparison shows that the Phase VI results are higher than the January LOMP results by \$101M for the undiscounted pre-tax case and \$55M for the discounted pre-tax case.

The two studies used many similar assumptions and, as a result, the incremental analysis provides a thorough comparison. Some of the main assumption differences from the January 2012 LOMP and the Phase VI PFS are:

- \$25M in capital that would be spent over the next four years to account for capital expenditures relating to the underground development of Copper Keel and Wildfire; and
- Scheduling of re-design pits

Table 22-2: Pre-tax Incremental Economic Results

Net Present Value	Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024
Undiscounted	M\$	101	(2)	44	32	(82)	3	(26)	17	42	50	23	8	(6)	(1)
Discounted at 8%	M\$	55	(2)	38	26	(60)	2	(17)	10	23	25	11	3	(2)	0

22.3. Sensitivities

The project was evaluated for sensitivity to the operating costs, capital costs, copper price and copper grade. All sensitivities were assessed for a range of -20% to +20% with the resulting discounted_{8.0%} after-tax value shown. Figure 22—1 graphically depicts results of the sensitivity analyses.

All sensitivities were done as mutually exclusive variations. A combination of variable changes was not conducted nor was an analysis of the probability of any variations.

The economic models show the project is most sensitive to variances in metal prices. In Minto's case, a 20% drop in copper price yields a \$131 M (-68%) decrease in after-tax value. Gold and silver prices are fixed as per the Silver Wheaton contract. An increase of 20% in copper price yields a \$123M increase. The second most sensitive parameter is copper grade. A 20% drop in copper grade yield a \$121M (-63%) decrease in after-value value. Conversely, an increase in metal grade yields a \$113M (59%) increase in after-tax value.

A 20% reduction in OPEX yields an \$52M (27%) increase. Conversely, a 20% increase in OPEX yields a \$56M (30%) decrease.

A 20% increase in CAPEX in Case A represents a \$12M (6%) reduction.

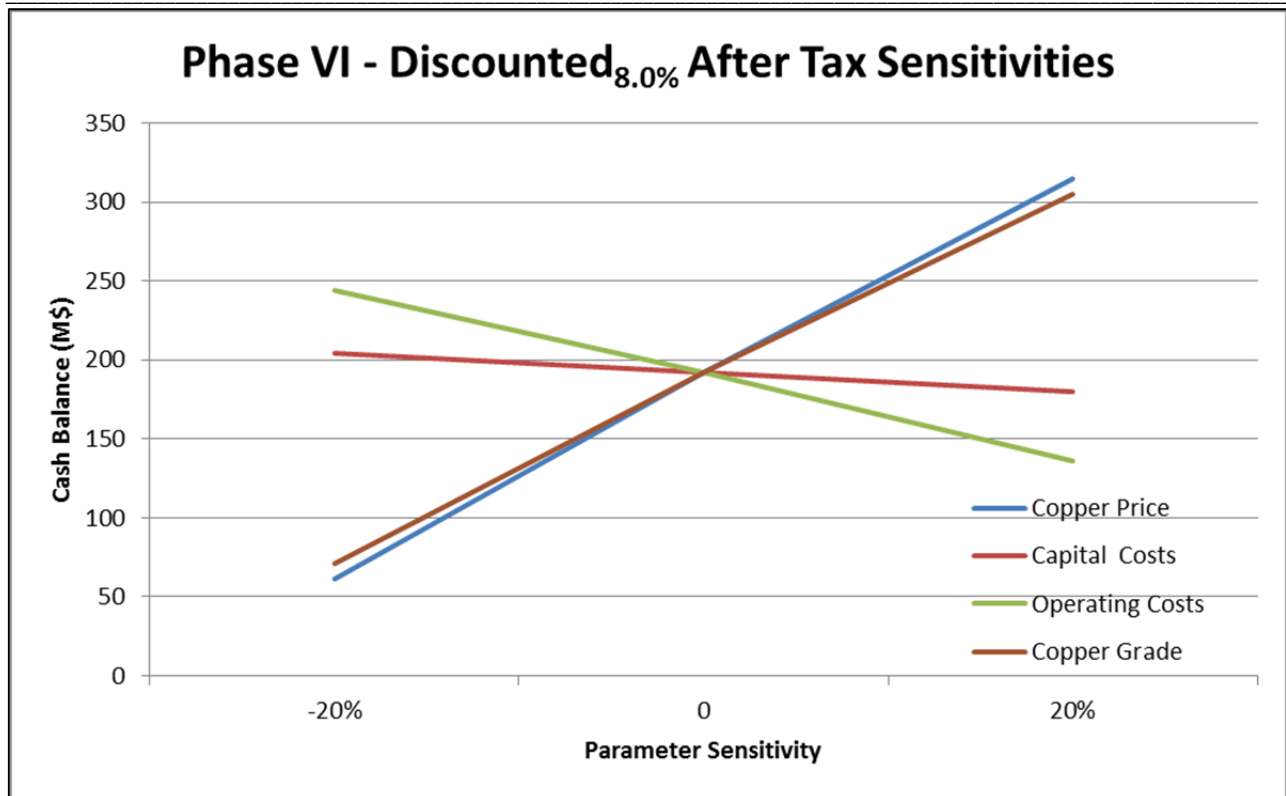


Figure 22—1: Case A Sensitivities

22.3.1. Payback

The payback period for initial capital investment in 2012 is recovered by the end of 2013. Thereafter, the revenue the mine generates more than covers the capital expenditure.

22.4. Mine Life

The Phase VI LOM plan has mine operations finishing in the second half of 2019. The mill will continue to run on stockpiled ore until mid-2022 at which time all operations will cease.

It is Capstone's opinion that there is potential for the mine to extend its life if additional resources can be turned into reserves. While there is no guarantee that this will happen, MintoEx has experienced a high level of success with its exploration drilling and there are still several exploration targets that show promise and are worth pursuing.

In addition to finding more mineralized material, the improvement in long-term copper price or the reduction of operating costs could increase the mine life by lowering cut-off grades and allowing more resources to be converted to reserves. The reserves generated for this study were based on a copper price of \$US2.50/lb, which is about 73% of the effective date copper price.

23. Adjacent Properties

No references to any adjacent properties other than general regional geology comments are used in this report. The mineral resource estimation, mineral reserve estimation, and exploration targets described in this report are based solely on work done on the Minto Property and are not influenced in any way by any potential mineralization on adjacent properties.

24. Other Relevant Data and Information

No other relevant data and information is provided.

25. Interpretations and Conclusions

Capstone has assessed the mining, process design and economic evaluation practices in this assessment of Minto Mine. Sufficient work has been carried out by the SRK, Kirkham Geosystems, and Minto Exploration to conclude that the mineral resource estimation is sufficient and conform to the CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices.” Historical and continuing work on permitting and environmental aspects of Minto Mine project provide understanding of the regulatory requirements to continue to operate and close the mine. After the currently proposed project is mined, Minto will still contain resources. If a higher copper price were used for mineral reserve estimation, Minto pits and underground reserves could be expected to expand, and the total mill feed and mine life would increase.

The following sections highlight the resource estimation, mining and process plant conclusions as well as review risks and opportunities.

25.1. Resource Estimation Interpretations and Conclusions

SRK reviewed and audited the exploration data available for Minto South and Ridgetop deposits. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

SRK, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization on two deposits. SRK considers that the geological model is a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, SRK constructed new mineral resource block models for Minto South and Ridgetop deposits constraining grade interpolation to within the modelled mineralization domains. After validation and classification, SRK considers that the mineral resources for the all three deposits are appropriately reported at a 0.5% Cu cut-off considering the open pit mining scenario discussed in the report.

Mineral resources for Minto South and Ridgetop deposits have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” Guidelines. In the opinion of SRK, the block model resource estimate and resource classification reported herein are representative of the copper, gold, and silver mineral resources found in the two deposits. Mineral resources are not mineral reserves and do not have demonstrated

economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

Kirkham Geosystems reviewed and audited the exploration data available for the Minto North and East deposits. This review suggests that the exploration data accumulated by MintoEx personnel is reliable for the purpose of resource estimation.

Kirkham Geosystems, guided by MintoEx geologists, modelled mineralized domains based on up-to-date interpretation of mineralization the deposits. A total of three wireframes (i.e. 115, 120 and 130 zones along with a cross-cutting dyke) were constructed for the Minto North Deposit and one wireframe (i.e. 700 zone) was constructed for the Minto East Deposit in MineSight™. Kirkham Geosystems considers that the geological models are a very good interpretation of the mineralized domains and is more than adequate for the resource estimation.

Following geostatistical analysis, Kirkham Geosystems constructed new mineral resource block models for both the Minto North and Minto East Deposit constraining grade interpolation to within the modelled mineralization domains.

After validation and classification, Kirkham Geosystems considers that the mineral resources for the Minto North deposit is appropriately reported at a 0.5% Cu cut-off considering open pit mining scenario as discussed in the report.

Mineral resources for the Minto North and the Minto East deposits have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” Guidelines. In the opinion of Kirkham Geosystems, the block model resource estimate and resource classification reported herein are very representative of the copper, gold, and silver mineral resources found in Minto North and Minto East. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

A number of factors may affect the quality and quantity of the current resource estimates, and thereby highlight opportunities for improvement:

- There are gaps in the understanding of the mineralization paragenesis. Improved understanding could benefit exploration models as well as the constraint on high-grade continuity and orientation. MintoEx are proactively making an effort in fundamental research to answer these questions.
- There are still some details that need to be constrained with respect to the structural geometries that are influencing the resource. Ductile and brittle fault structures and folding on various scales deform the ore horizons. The deformation history needs to be better constrained, and again research is currently on-going in order to answer these questions.
- There is poor control on the brittle structures that could impact the geotechnical assessment. It would be beneficial to undertake a mapping exercise of the current pits to determine the brittle fault and joint pattern. This information should be combined with drill hole logs, modelled structural information, mineralization offsets, exploration data and geophysical data (e.g. Titan 24 MT) to determine the structural patterns and position of major faults and folds.

25.2. Mining Conclusions, Risks and Opportunities

25.2.1. Conclusions

- The Minto Mine, encompassing Phase VI OP and UG deposits represents a long term sustainable operation.
- There continues to be strong exploration targets with respect to the underground workings
- Based on Phase V test work and exploration results, the Phase VI waste rock does not appear to have ARD issues.
- Room and pillar mining has been recommended for mineral extraction in the Copper Keel and Wildfire areas based on the interpreted low-continuity and variability of mineralization. This method allows mining areas to be defined based on “delineation by development”, whereupon a standard room and pillar layout can then be implemented.

25.2.2. Risks

The following risks have been identified for the Minto Phase VI project:

- Significant reliance on underground contractor development and mining that will increase CAPEX expenditures
- OPEX control
- Challenges associated with sourcing of underground personnel. The site has committed to a First Nations high quality training program that may gradually provide personnel to offset turnover.
- Mine Permit revisions: The mine is not currently permitted by regulators to carry out the mine plan as presented in the report. Changes to the permit involve a production increase, an increase in tailings deposition volume, extraction of various ore deposits with open pit and underground mining methods; and a change in tailings deposition modality.
- Exchange rates, metal prices and external influences: MintoEx has no control over exchange rates and their impact on the economics of the operation is significant. Metal prices are also not controllable, other than by forward sales contracts, and can have an appreciable effect on project return.
- Grade control: The open pits (in particular Area 2) are comprised of several zones of ore that are not as continuous and thick as the Main zone. As a result, a very thorough and proactive grade control program will be necessary in order to minimize dilution since excessive dilution will have a negative impact on the project economics.

25.2.3. Opportunities

The most significant opportunities that should be investigated are listed below:

- Minto has a consistent history of exploration success in both the exploration of new deposits and in the conversion of those deposits into mineral resources and reserves. There is no guarantee that this success will continue, however, the property has a number of high-quality exploration targets that are adjacent to existing reserves and worth pursuing.
- Trade-off studies to determine if owner-operator functions can be more cost-effective than contractor-operator functions. Owner-operated waste drilling will reduce mining costs

with low capital investment. Other trade-off studies should be initiated in areas such as re-handling and stockpile management.

- Increased underground reserves extraction from 76% to 90% through the utilization of concrete posts and a proactive geotechnical instrumentation program.
- Increase the density of in-pit deposition of tailings and an underground hydraulic fill program. This would allow for an increased volume of open pit waste material to be placed in the pits rather than the long haul up to the dumps.

25.3. Processing Plant Conclusions, Risks and Opportunities

25.3.1. Conclusions

- Mill recoveries and maintaining high throughput will be the primary objectives of the processing plant
- Blending partially oxidized ore with sulphide ore have stabilized mill recoveries
- Maintaining appropriate feed size for the grinding circuit is critical to stabilize recoveries and throughput

25.3.2. Crushing Circuit Risks and Opportunities

As noted in the Phase V PFS, the sizing of the existing jaw crusher is not seen as a risk for the plant upgrade. The published capacity of 37' x 49' jaw crusher with un-scalped feed and a closed side setting of 115 mm is approximately 290 tph and therefore the crusher is expected to achieve the design 228 tph.

The secondary crusher (S4800) has risk associated with the flow sheet design. There is no facility to screen the feed material prior to the cone crusher to remove fines. Therefore, the published de-rated (no fines scalping prior to crushing) crusher performance for a Sandvik S4800 with a closed side setting (CSS) of 25 mm is expected to be below the design throughput requirement of 228 tph. The risk with the secondary crusher flow sheet is that the CSS will be opened to achieve the required throughput which will increase the product size with a resultant decrease in SAG mill throughput. An opportunity exists to incorporate screening prior to the secondary crusher to reduce the load on the secondary crusher and provide the required final crushed product size for the milling circuit.

25.3.3. Crushed Ore Stockpile and Reclaim Risks and Opportunities

Previously approximately 50% of the feed to the existing SAG mill was secondary crushed. The plant upgrade design is based on 100% of the SAG mill feed being secondary crushed and, hence, substantially finer. The current stockpile consists of a single apron feeder. The risk with this design is that the finer crushed product on the stockpile will have a different drawdown angle resulting in a variation in live stockpile capacity. An opportunity exists to review the crushed ore properties through further test work and/or experience in operating the recently installed secondary crusher. A second reclaim feeder will improve the amount of recoverable material on the stockpile. A second feeder will have the added benefit of providing improved blending to the SAG mill and operating redundancy.

25.3.4. Comminution Circuit Risks and Opportunities

The risks associated with modelling of the Minto comminution circuit are:

- The limited ore samples tested for ore competency may not be “representative” of the range of ore competency characteristics;
- The limited ore samples tested for ore hardness may not be “representative” of the range of ore competency characteristics;
- The actual plant observations by Starkey do not align well with the power based mill performance modelling by Ausenco and current plant performance, indicating that the SAG is performing more like a ball mill than a SAG mill.
- The ore competency data is limited for deposits outside of the Main deposit and further work is recommended to confirm the assumptions made in this report with regard to SAG mill throughput. Further test work is recommended to mitigate risk associated with the new ore bodies being either, on average more competent than main ore, or, containing areas of ore that may be localised but may limit plant production in the future.

The modelling of the comminution circuit indicates that the existing mills will need to operate at maximum power draw to achieve the design throughput. Whilst this is normal practice for a ball milling circuit, operational control of a SAG mill at sustained maximum power draw increases the risk of mill overloads and potential downtime.

25.3.5. Optimum Grind Size Risks and Opportunities

A point of discussion from the test work reports is the optimum primary grind size target. Table 25-1 below summarises the effect of primary grind size as studied in the test work.

Table 25-1: Summary of Effect of Grind Size on Recovery

Ore body	Impact of P ₈₀ on Cu and Au Recovery
Minto North	Primary grind size had no impact on rougher tailings grade but flotation kinetics slower with coarser grind.
Ridgetop East	The partially oxidized upper zone is sensitive to the primary grind size, and a grind P ₈₀ below 200 µm is required.
Area 118	Primary grind size had no significant impact on rougher tailings grade.
Area 2	Primary grind size had no significant impact on copper recovery but gold recoveries were 10% worse at P ₈₀ 270 compared with P ₈₀ 150 µm.
Main	A primary grind size of 200 µm appears optimum. Grind sizes coarser than 200 µm have poorer gold (5 - 10%) recoveries.
Main (South)	Copper and gold recoveries appeared to decrease at primary grind sizes higher than 150 µm.

The test work indicated some potential benefits of a finer primary grind size for certain deposits.

25.3.6. Regrind Mill Capital Cost Risks and Opportunities

Ausenco searched for a used VTM300 regrind mill however there were none located at the time of this study. The cost for a new VTM300 mill is around \$1.2 million. A second hand VTM200 was

sourced at the time of the pre-feasibility study at a cost of around \$0.3 million. There is an opportunity to use this second hand regrind mill to reduce the overall capital expenditure however the risks are:

- The mill was not inspected by Ausenco during the pre-feasibility study and therefore refurbishment costs would need to be included; and
- The VTM200 would not provide the required regrind size of 80% passing 60 micron for the plant upgrade scenario. The P_{80} that would be achieved is around 72 micron for the nominal regrind circuit throughput of 21 tph (based on 171 tph fresh feed to the plant and 12.3% rougher/scavenger mass recovery).

25.3.7. Tailings Treatment Risks and Opportunities

Risks

- As tailings and water are stored behind the buttress in the Main Pit, the North wall of the Area 2 pit will have to be monitored to ensure slope design assumptions for the overburden layer are confirmed. It would be prudent to undertake “high level” mitigation planning in conjunction with the final design of the buttress. Therefore, in the event the slope shows movement which exceeds that which conforms to the design assumption, the development of a mitigation plan can be conducted expeditiously.
- Reclaim water pumped from the barge may not be sufficiently clarified for immediate use in the plant, which leads, for example, to an incremental water treatment requirement at the plant.
- Operational difficulties with the reclaim barge lead to the need for adjustments to the operation plan and/or the implementation of a different reclaim system.
- Operational difficulties with tailings deposition lead to a highly uneven tailings surface that, in turn, leads to significant incremental closure costs associated with creating an appropriately covered and graded tailings surface at closure.
- Due to possible additional inflows linked to seepage management and Area 2 Pit dewatering requirements, the anticipated annual surplus water treatment/disposal requirements could increase.

Opportunities

- The water collected in the wells which are likely to be installed in the buttress turns out to be appropriate in volume and quality that it can be pumped directly to the plant for re-use in the mill circuit.
- Cyclones could be used to deposit sand tailings on the benches around the pit in order to increase the storage capacity of the various pits.
- Once the use of the various pits for active tailings deposition has stopped, the concept of storing additional waste rock or overburden on the tailings surface (beyond what would be needed within the current closure concepts), could be considered.

26. Recommendations

26.1. Mining and Exploration

- Optimization of the Life of Mine Plan with the objective of decreasing the mining rate to avoid excessive stockpiling that has a detrimental impact on NPV;
- Optimization of the Life of Mine Plan to take advantage of in-pit waste haulage that decreases the footprint and also lowers CAPEX stripping and OPEX waste expenditures;
- Conduct trade-off studies to further reduce mining costs;
- A detailed structural assessment should be completed to better understand the large and small-scale structures in the Area 2 and Copper Keel areas. These structures would serve as a basis for the delineation of geotechnical domains, assessing exclusion areas for underground mining, and refining ground support
- Develop an underground exploration drift that can be utilized as a drilling platform to define resources at depth.
- Comprehensive geotechnical mapping should be completed as development is advanced for the exploration/production decline.
- Ground support components should be tested to assess their suitability to the rock mass encountered including threaded rebar, Swellex, and Split-set anchors.
- Volumes of water entering the decline should be monitored and tested for discharge purposes
- A comprehensive study of brittle structural geology would be beneficial to better quantify mineralization zone displacements and to increase confidence in geotechnical rock mass characterization

26.1.1. Geotechnical Work

- Continue geotechnical investigations with respect to the Main Pit buttress and Mill Valley Fill buttress. In-the-field instrumentation data can be compared to the FLAC numerical prediction to optimize designs and potentially increase the volume of deposition tailings the main pit.
- Additional geotechnical characterization and analyses should be conducted at the feasibility and design levels for each of the areas. Analyses and recommendations presented herein are based on ultimate pit designs as described in this report, and, as such, any significant changes to mine plans or pit architecture should be reviewed by SRK to verify that recommendations will remain valid for the new mine plans.
- The geological structure is being further evaluated to more accurately characterize the rock mass which, according to the current mine plans, will comprise the toe of the Area 2 western slope walls and which will better ascertain the likelihood of the existence and orientation of major structures that may adversely impact stability of that western wall.
- Additional geotechnical characterization and analysis will also be necessary at Minto North, to better define rock mass conditions and structural impacts on bench stability as the project advances. To accomplish this, one additional geotechnical corehole is recommended at Minto North drilled into the northwest wall for evaluation of rock mass conditions and structure.

- The Area 118 and Ridgetop open pits most likely will not require additional geotechnical drilling unless major changes are made to the current plans.

For a feasibility level study, the following components are considered primary requirements.

- A detailed structural assessment should be completed to better understand the large and small-scale structures in the Area 2 and Copper Keel areas. These structures would serve as a basis for the delineation of geotechnical domains, assessing exclusion areas for underground mining, and refining ground support.
- Additional geotechnical drilling maybe required to enhance the structural and rock mass interpretations.
- A hydrogeological assessment is required to characterize mine water inflows and pumping requirements.
- Sequencing of underground and open pit mining should be reviewed when a detailed mine plan is made available. This will include Map3D modelling to assess the impacts of the final open pit interaction with underground stoping.

26.2. Further Metallurgical Test Work

- A detailed feasibility study of flotation and comminution variability test work across the ore body is required to develop detailed models of plant throughput and grade/recovery that take into account variation in competency, mineralogy and head grade

26.2.1. Further Comminution Test Work

- Continue comminution testing to mitigate risk associated with the new orebodies being either on average harder than the current Minto Main / Area 2 ore, or containing localized zones of harder ore

26.2.2. Further Flotation and General Plant Design Test Work

- Installation of the new Jameson Cell, cleaner cell replacement, verti mill recommended comminution, flotation and general plant design test work is in the order of \$300K.

26.3. Tailings Solids

- Grain size distribution with hydrometer (-#200 fraction) and Atterberg limits (to evaluate cycloning potential, settlement characteristics, in situ permeability and potential for use of underflow as a drainage layer);
- Modified Proctor testing (cyclone underflow - to evaluate constructability and define parameters for direct shear and permeability testing);
- Specific gravity (to facilitate evaluation of slurry rheology);
- Shear strength (cyclone underflow and overflow fractions for embankment stability evaluation);
- Flexible wall permeability (total tailings permeability, or cyclone underflow and overflow drainage characteristics);
- Settled density and consolidation testing to evaluate the initial settled dry density and the dry density of the tailings under self-weight.

27. Illustrations

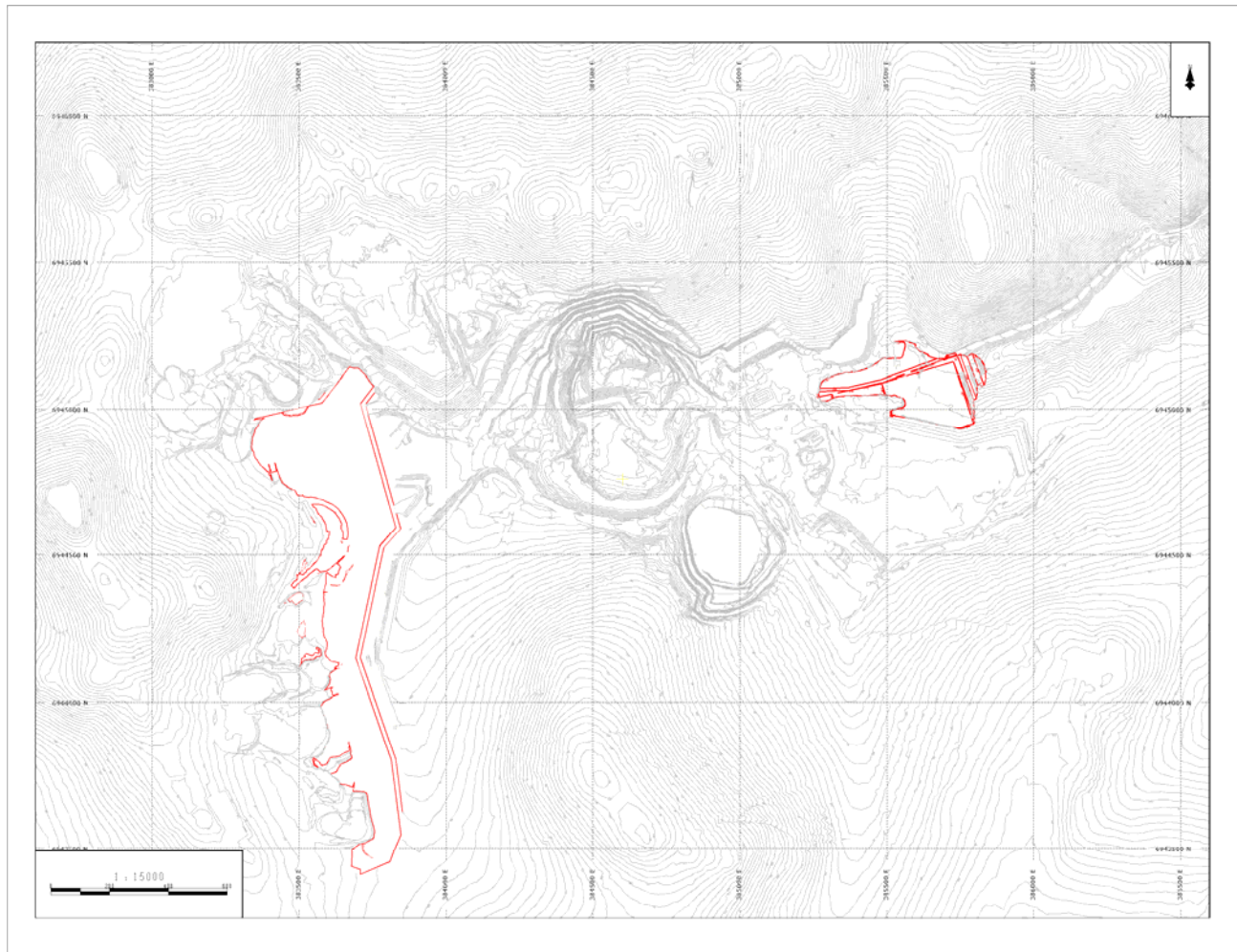


Figure 27-1: End of Period Map: January 1, 2013

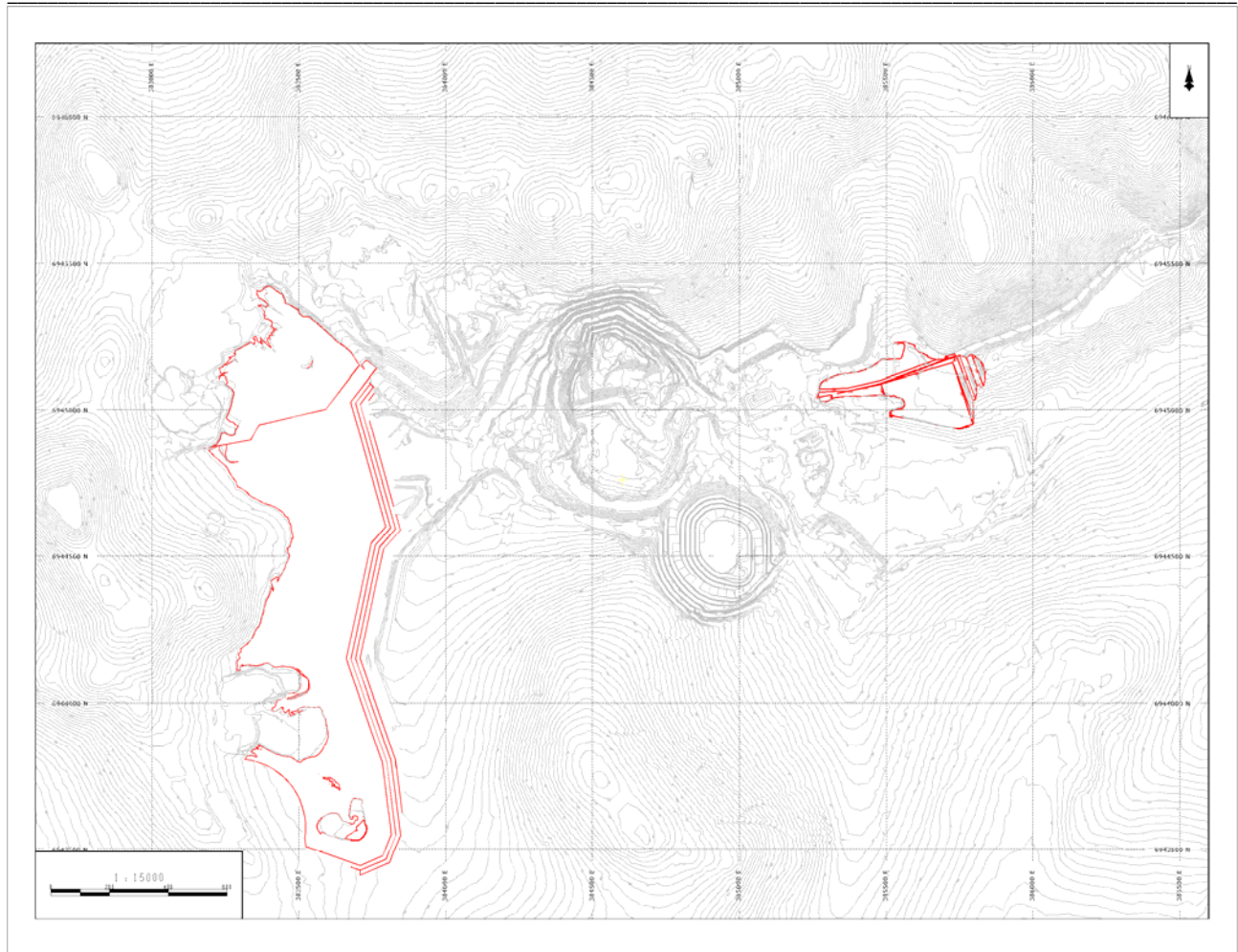


Figure 27-2: End of Period Map: July 1, 2013

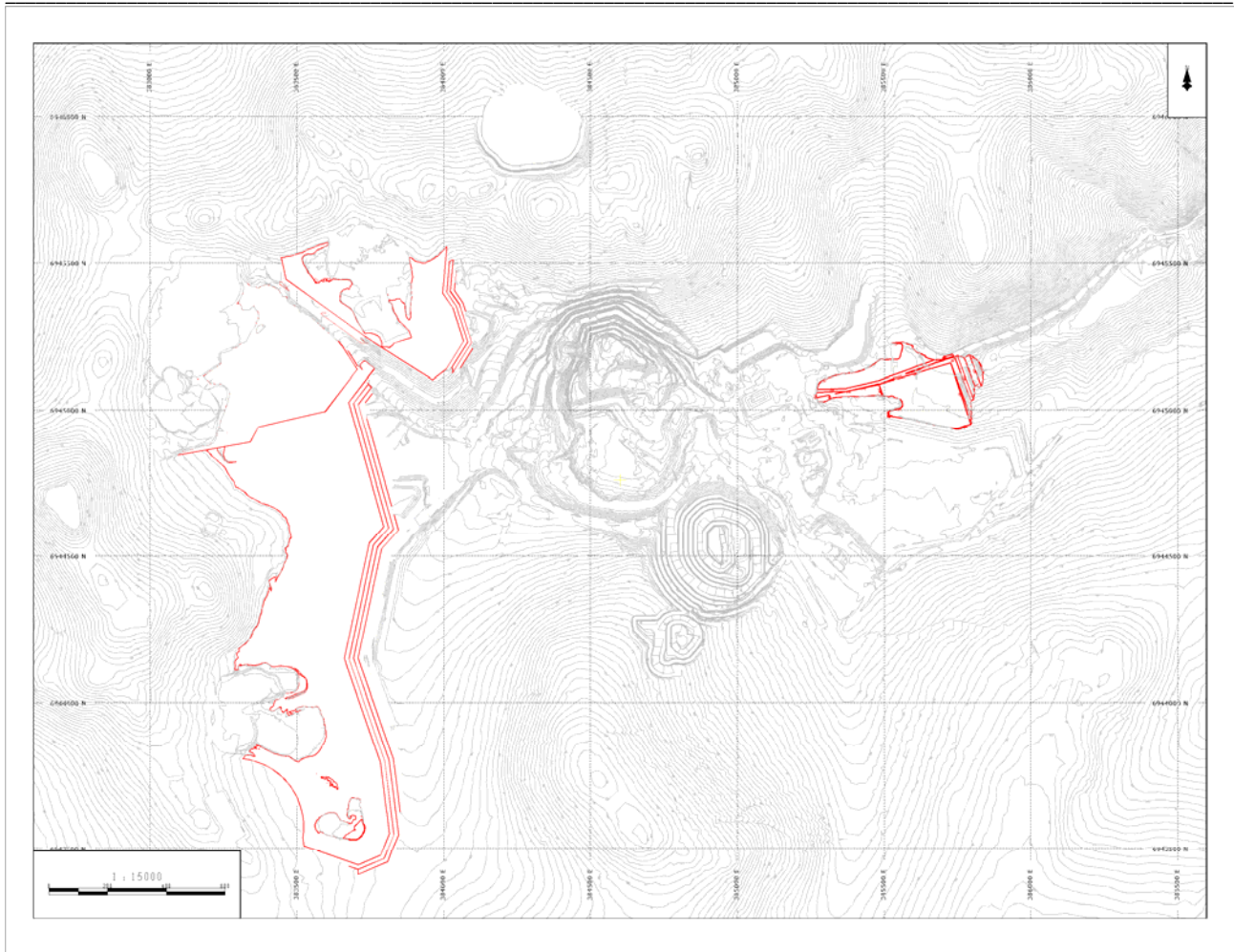


Figure 27-3: End of Period Map: January 1, 2014

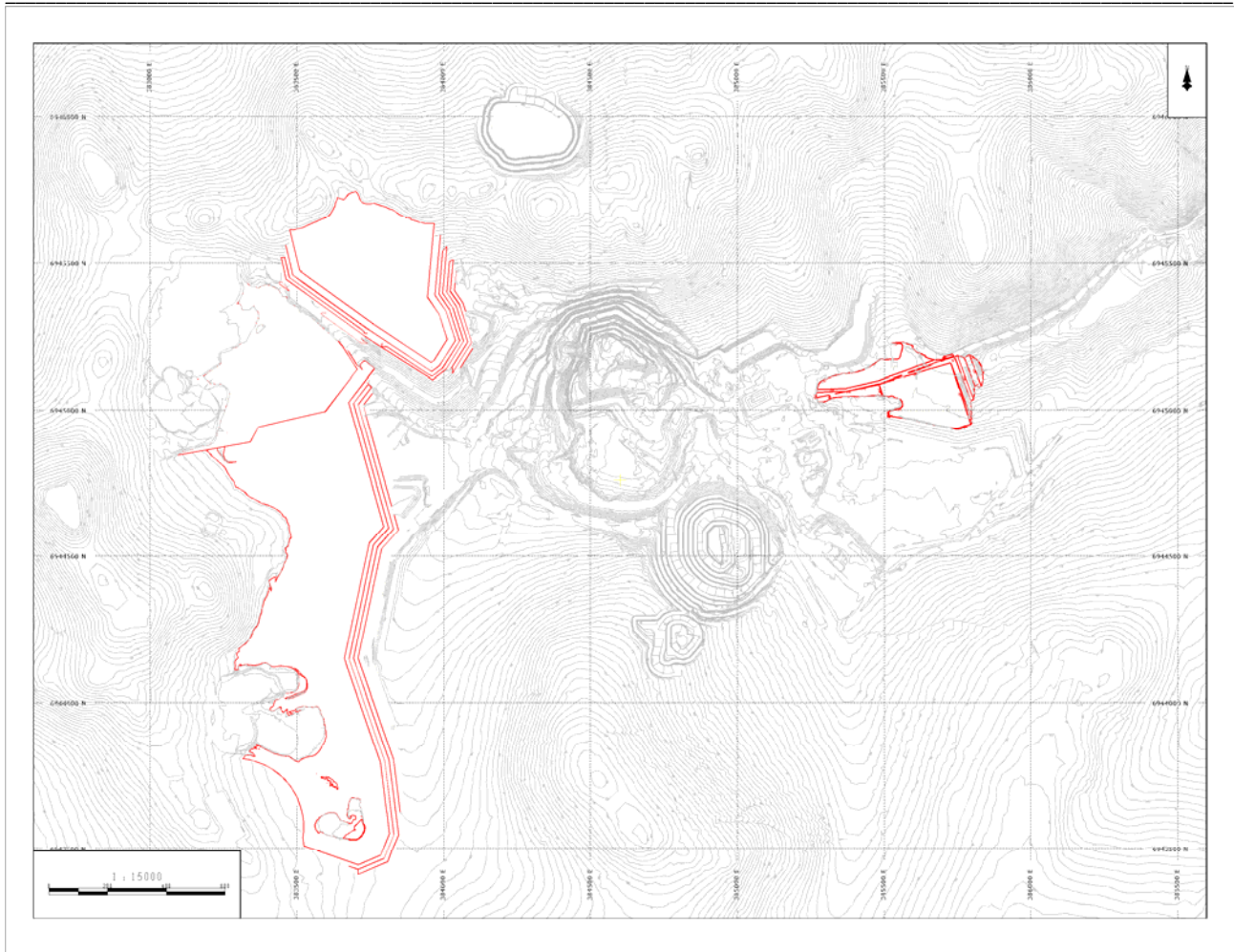


Figure 27-4: End of Period Map: July 1, 2014

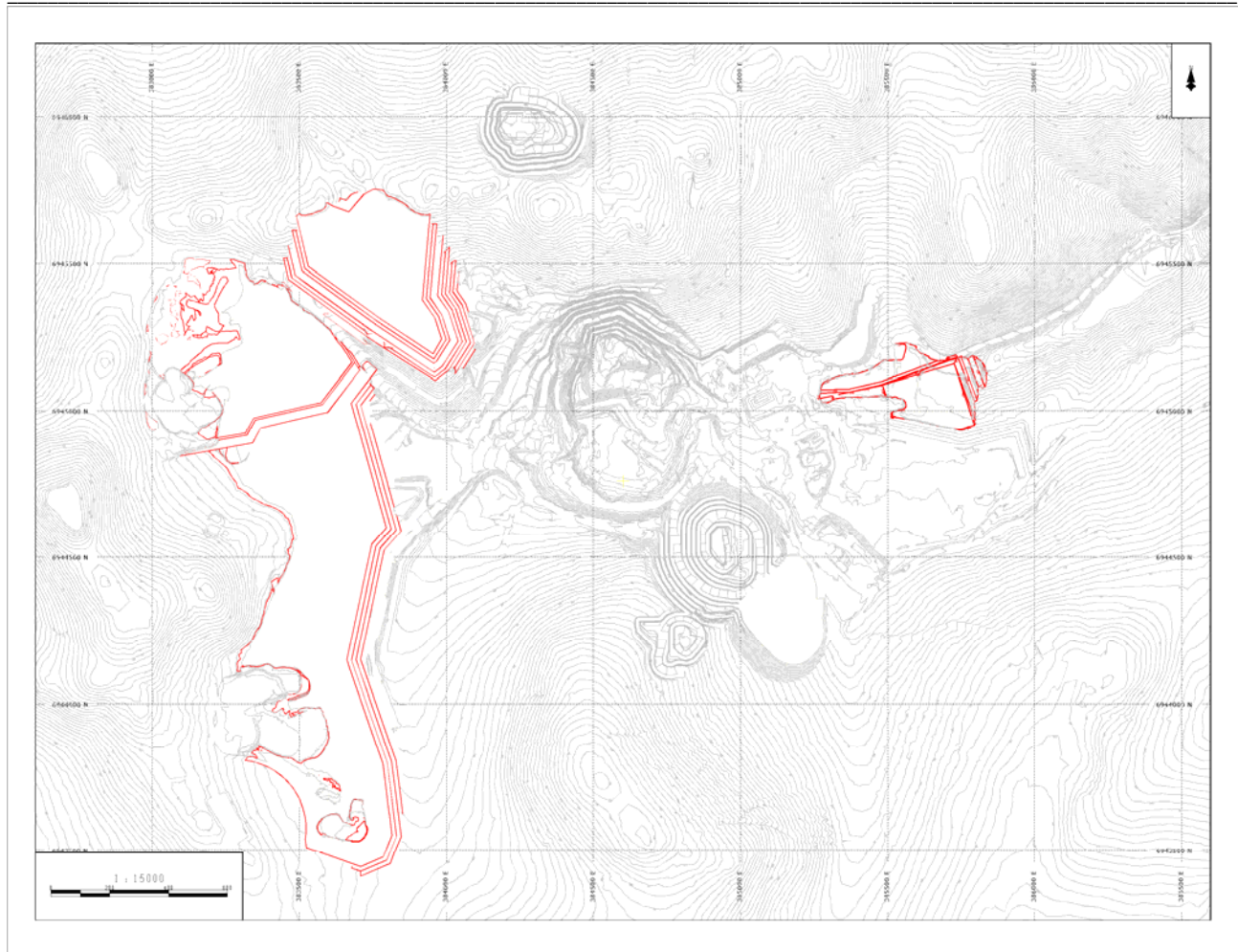


Figure 27-5: End of Period Map: January 1, 2015

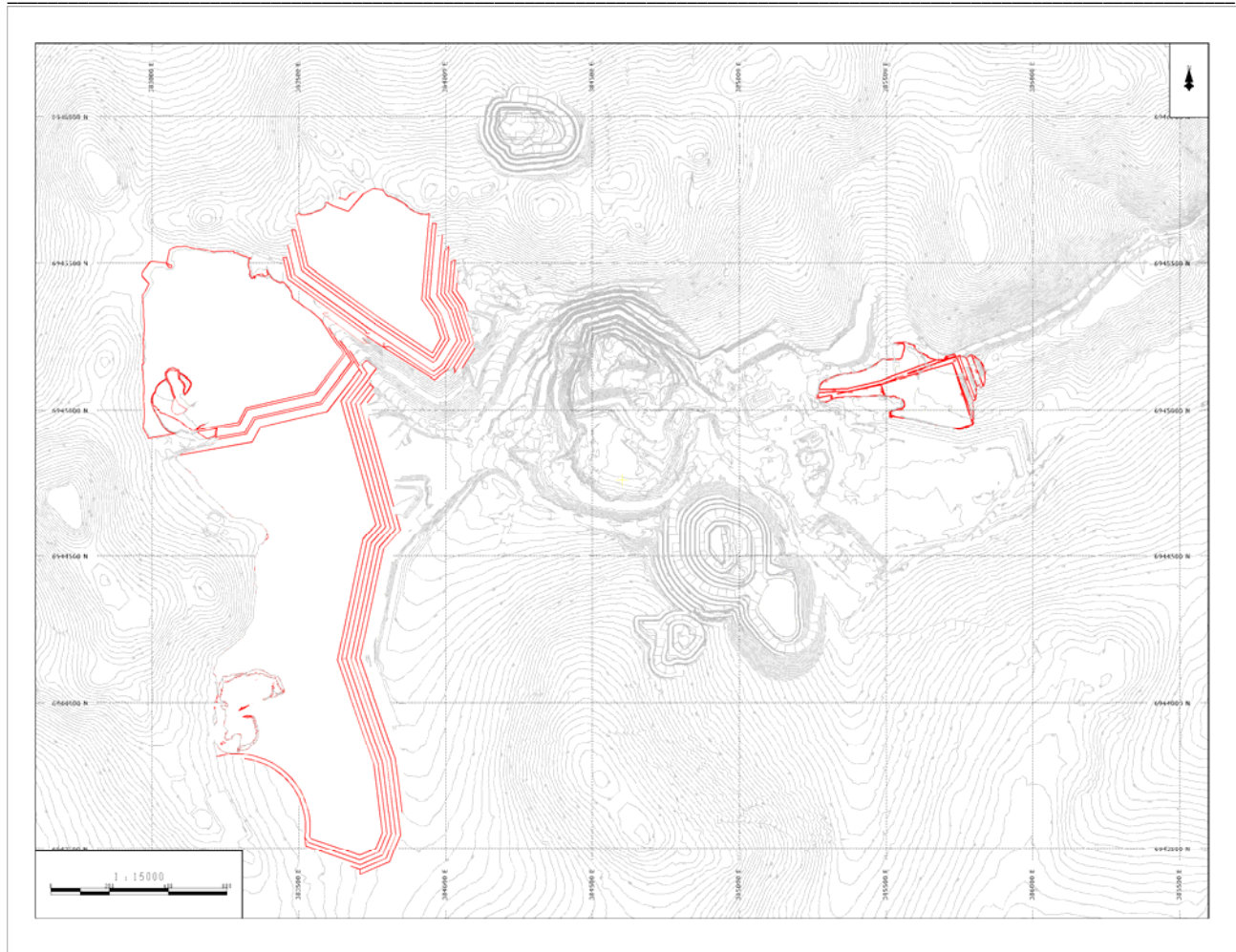


Figure 27-6: End of Period Map: July 1, 2015

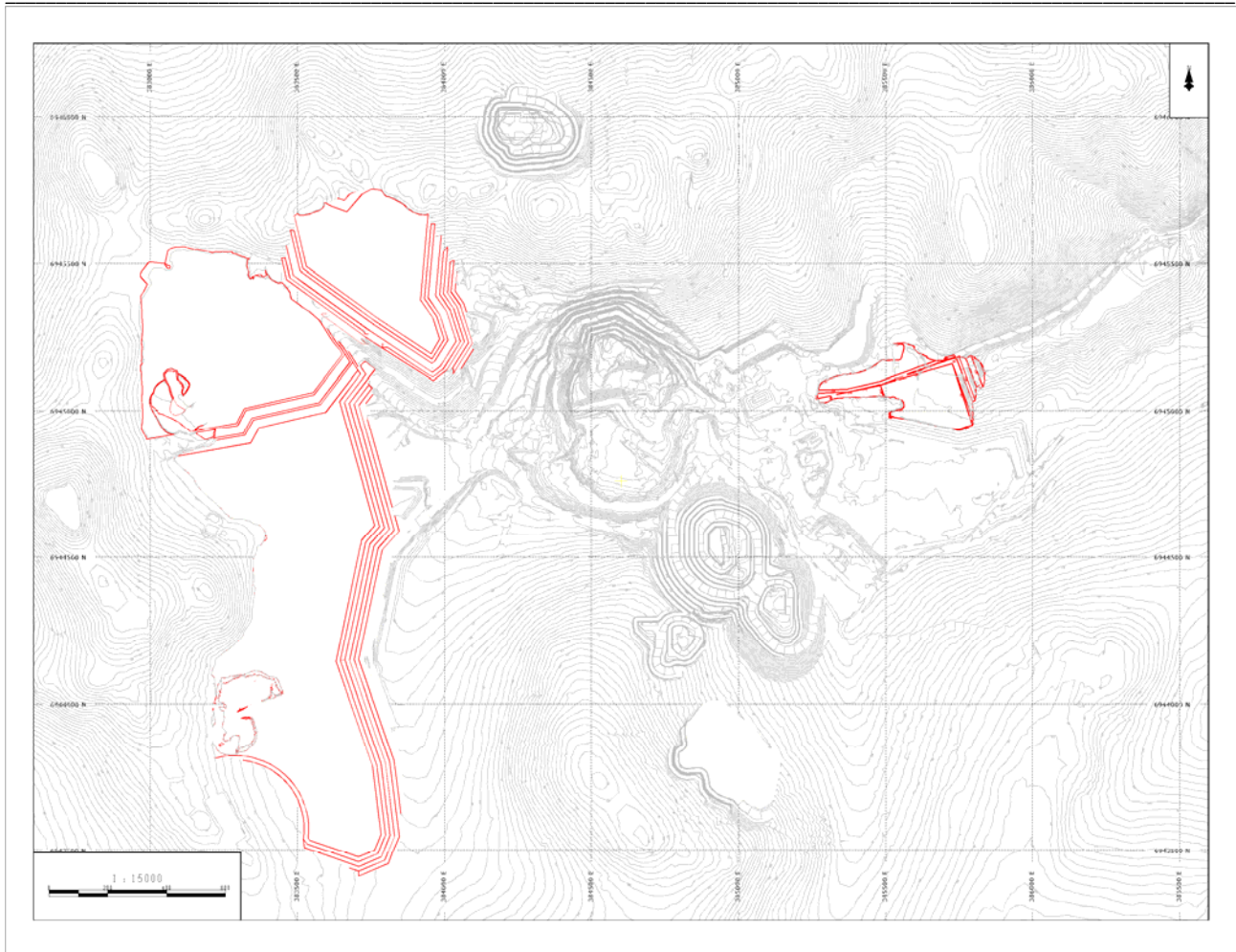


Figure 27-7: End of Period Map: January 1, 2016

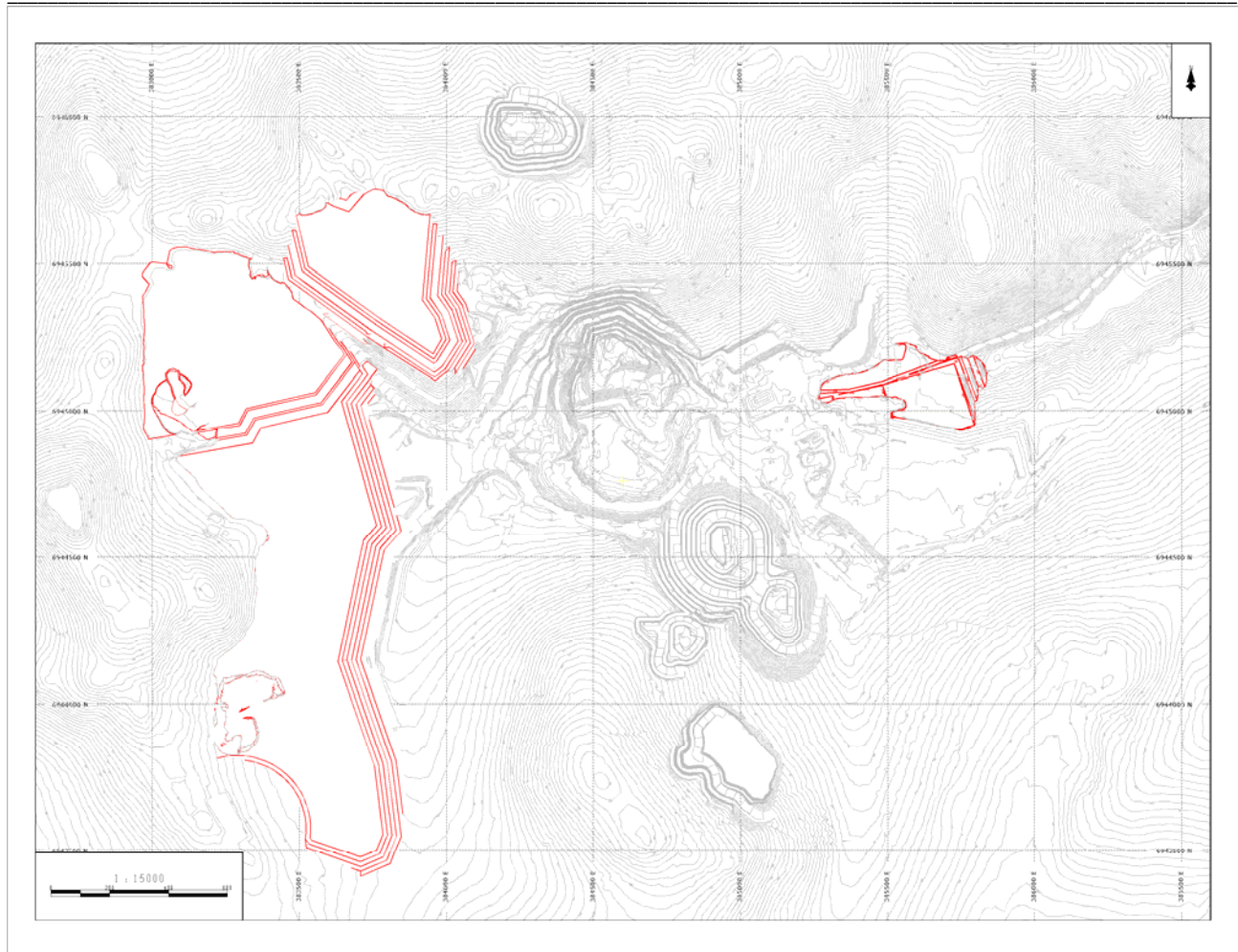


Figure 27-8: End of Period Map: July 1, 2016

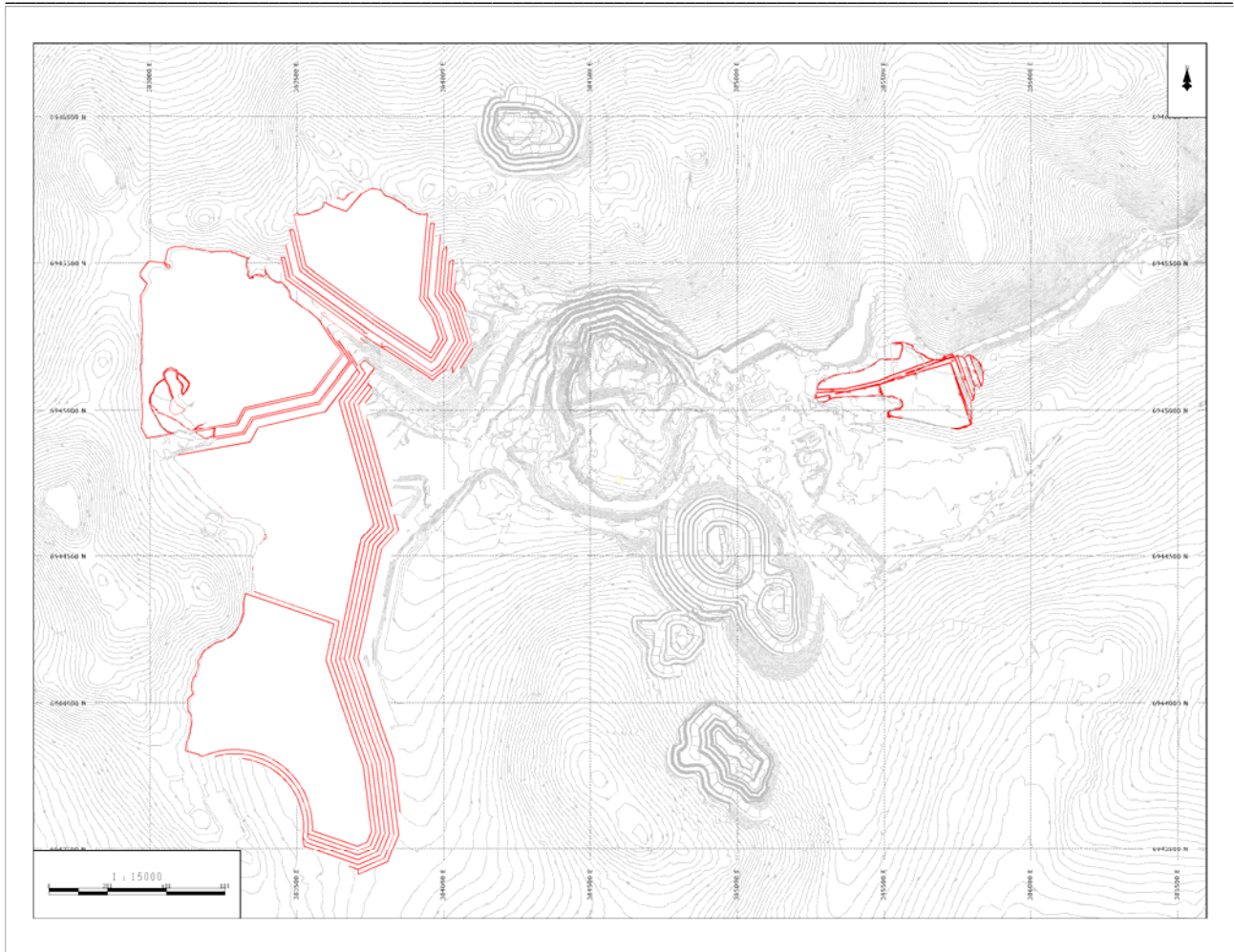


Figure 27-9: End of Period Map: January 1, 2017

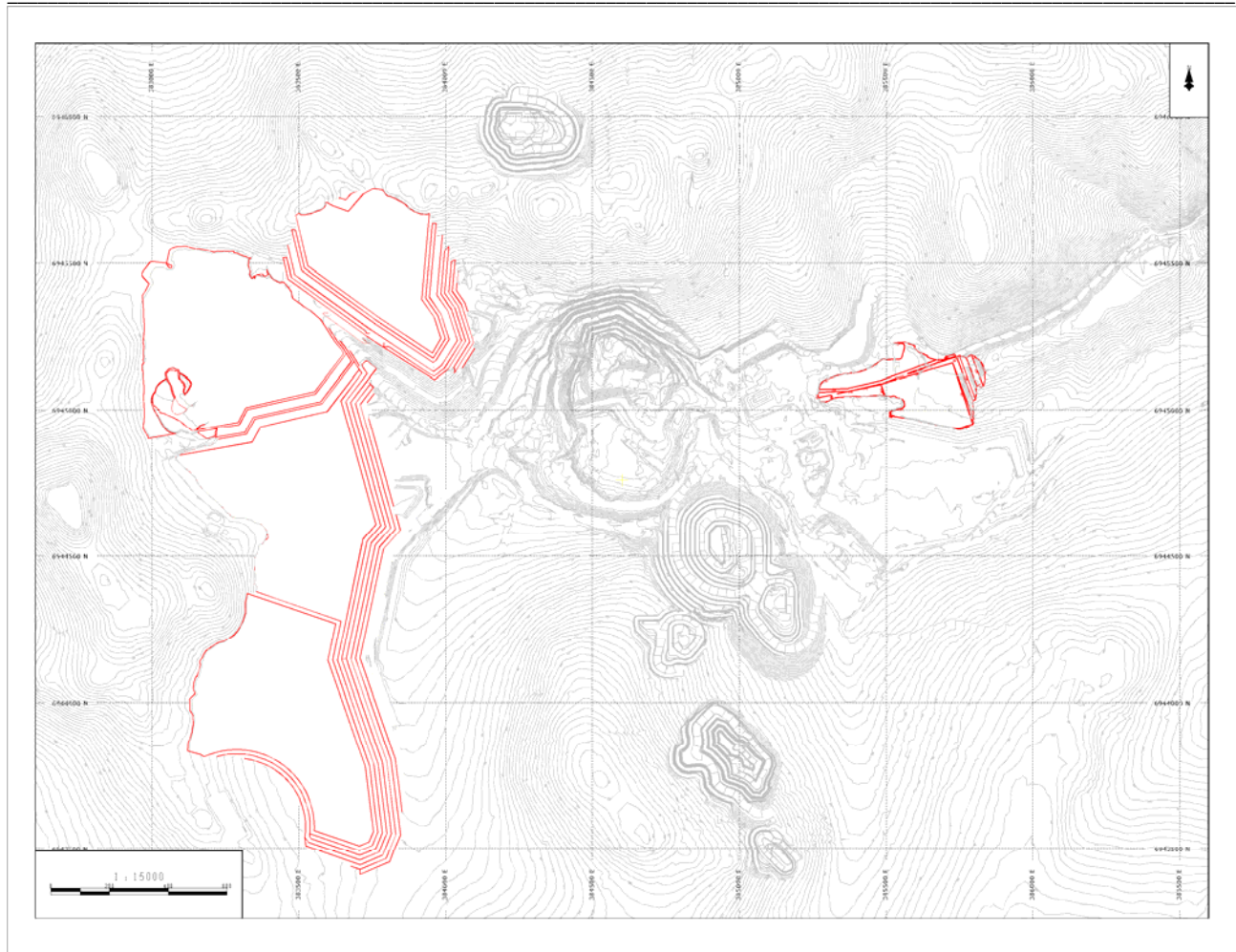


Figure 27-10: End of Period Map: July 1, 2017

28. References

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29. Standard Acronyms and Abbreviations

Distance		Other	
µm	micron (micrometre)	°C	degree Celsius
mm	millimetre	°F	degree Fahrenheit
cm	centimetre	Btu	British Thermal Unit
m	metre	cfm	cubic feet per minute
km	kilometre	elev	elevation above sea level
"	inch	amsl	above mean sea level
in	inch	hp	horsepower
'	foot	hr	hour
ft	foot	kW	kilowatt
Area		kWh	kilowatt hour
m ²	square metre	M	Million
km ²	square kilometre	mph	miles per hour
ac	acre	ppb	parts per billion
Ha	hectare	ppm	parts per million
Volume		s	second
l	litre	s.g.	specific gravity
m ³	cubic metre	usgpm	US gallon per minute
ft ³	cubic foot	V	volt
usg	US gallon	W	watt
lcm	loose cubic metre	Ω	ohm
Mlcm	Million lcm	A	ampere
bcm	bank cubic metre	tph	tonnes per hour
Mbcm	Million bcm	tpd	tonnes per day
Mass		∅	diameter

kg	kilogram
g	gram
t	metric tonne
Kt	Kilotonne
lb	pound
Mt	Megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	Pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
Cu	copper
Zn	zinc
CaCO ₃	Calcium carbonate
ANFO	Ammonium

Cu eq.	Copper equivalent
Acronyms	
ABA	Acid- base accounting
AP	Acid potential
CAPEX	Capital costs
CIM	Canadian Institute of Mining
CV	Coefficient of Variation
DDH	Diamond drill hole
HDPE	High-density polyethylene
IRR	Internal Rate of Return
MintoEx	Minto Explorations Ltd.
LOM	Life of mine
LOMP	Life of mine plan
ML/ARD	Metal leaching/ acid rock drainage
N, S, E, W	North, South, East, West
NI 43-101	National Instrument 43-101
non-PAG	Non-potentially acid generating
NP	Neutralization potential
NPTIC	Carbonate neutralization potential
NSR	Net Smelter Return
OPEX	Operating costs
PAG	Potentially acid

	Nitrate/Fuel Oil		generating
Conversion Factors		PFS	Preliminary Feasibility Study
1 tonne	2204.6 lb	% or pct	percent
1 oz	31.1 g	QA/QC	Quality Assurance/Quality Control
		JDS	JDS Energy & Mining Inc.
		X, Y, Z	Cartesian Coordinates, also “Easting”, “Northing”, and “Elevation”

Date and Signature Page

This technical report was written by the Qualified Persons listed below. The effective date of this technical report is January 1, 2012.

Name of QP	Signature	Report Section Responsibility
John Sagman, P.Eng.	Original Signed	Sections: 1 to 1.5, 1.9 to 5.4, 15.1, 15.3, 16.1, 16.4.1 to 16.4.6, 16.4.6.2 to 16.4.8, 18.1.1, 18.1.3 to 18.2, 18.2.2 to 19.3.1 , 21 to 29
Wayne Barnett, Pr.Sci.Nat.	Original Signed	Sections: 11, 12, 14.1 to 14.4
John Eggert, P.Eng.	Original Signed	Sections: 1.6 to 1.7, 13, 17, 25.3, 25.3.2, 25.3.4 and 25.3.5
Bill Hodgson, P.Eng.	Original Signed	Sections: 16.4.6.1, 18.1.2, 18.2.1
Garth Kirkham, P.Geo	Original Signed	Sections: 14.5 and 14.6
Michael Levy, PE	Original Signed	Sections: 16.2
Brad Mercer, P.Geo.	Original Signed	Sections: 1.1 to 1.3, 6 to 10 and 14.7
Pooya Mohseni, P.Eng.	Original Signed	Sections: 15.2, 16.5 to 16.7.3
Bruce Murphy, P.Eng.	Original Signed	Sections: 16.3
Colleen Roche, P.Eng.	Original Signed	Sections: 1.8 and 20



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CERTIFICATE OF AUTHOR

I, John Sagman, do hereby certify that:

I am the V.P. Technical Services with the firm Capstone Mining Corporation the parent corporation of Minto Explorations Ltd. with an office at Suite 900-999 West Hastings Street, Vancouver, Canada;

I am a graduate of the University of British Columbia with a Bachelor of Applied Science degree in Mining and Mineral Process Engineering, 1981. I have practiced my profession continuously since 1981;

This certificate applies to the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;

I am a Registered Professional Engineering in good standing with the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) and a Project Management Professional in good standing with the Project Management Institute;

I am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

I am an employee of Capstone Mining Corporation and I do expect to benefit directly in the securities of Capstone Mining;

That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

I have read National Instrument 43-101 and Form 43-101F1 and I am a Qualified Person for the purpose of NI 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;

I visited the Minto Mine site many times from 2010 to 2012;

I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Feasibility Study ("PFS") that is not reflected in the PFS, the omission to disclose which makes the PFS misleading;

I consent to the filing of the PFS with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the PFs;

I am responsible for and/or shared responsibility for Sections 1.1 to 1.5, 1.9 to 5.4, 15.1, 15.3, 16.1, 16.4.1 to 16.4.6, 16.4.6.2 to 16.4.8, 18.1.1, 18.1.3 to 18.2, 18.2.2 to 19.3.1, 21 to 29 of the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;

Signed, Sealed and dated this 16th day of July, 2012

"Original Signed and Stamped"

Stamp

John Sagman, P.Eng., PMP

Vice-President of Technical Services, Capstone Mining Corp.



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CERTIFICATE OF QUALIFIED PERSON

Wayne Barnett, Pr.Sci.Nat.

I, Wayne Barnett, am a Professional Natural Scientist, employed as a Principal Geologist with SRK Consulting (Canada) Inc.

This certificate applies to the report entitled “Minto Phase VI Pre-Feasibility Study Technical Report” with the effective date of January 1, 2012;

I am a member of the South African Council for Natural Scientific Professions, South Africa. I graduated with a geology honours degree from the University of Cape Town in 1996, and a doctorate degree from the University of Kwa-Zulu Natal in 2006.

I have been involved in mining and have practised my profession continuously since 1997. I have been involved in mining geology, exploration geology, geological modelling and estimation covering a wide range of mineral commodities in Africa, Australia, South America, North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Mine site between the 4th and 6th of March, 2009, and again from the 6th to 10th April 2011.

I am responsible for Sections: 11, 12, 14.1 to 14.4 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Minto Mine since 2009 undertaking geological modelling and estimation.

I have read National Instrument 43-101 and this report has been prepared in compliance with that 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.

Original Copy Signed

Wayne Barnett, Pr.Sci.Nat.

Dated: July 24, 2012

QP Certificate Barnett_20120724.doc



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CERTIFICATE OF QUALIFIED PERSON

John G. Eggert P.Eng.
158 David Street
Sudbury, Ontario
P3E 1T4

July 18, 2012

I, John G. Eggert am a Professional Engineer, employed as the President and Principle Minerals Processing Engineer with Eggert Engineering Inc.

This certificate applies to the technical report titled "*Minto Phase VI Preliminary Feasibility Study Technical Report*" with the effective date of January 1, 2012.

I am a registered Professional Engineer in the Province of Ontario, (#90397597). I graduated with a B. Sc. in Mining Engineering, Minerals Processing Option, from Queen's University at Kingston in 1990. I have practiced my profession continuously since 1990 and have been involved in operations, plant construction and design of a wide variety of milling operations, including gold processing, industrial minerals and base metal processing.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto site frequently over the past two years. I am responsible for the sections titled "Summary – Metallurgical Testwork, Summary – Process Plant, Mineral Processing and Metallurgical Testing (all) Recovery Methods (all), Interpretations and Conclusions – Processing Plant Conclusions and Risks – Crushing Circuit Risks and Opportunities – Comminution Circuit Risks and Opportunities and Optimum Grind Size Risks and Opportunities".

I am independent of Minto Explorations Ltd. as independence is described in Section 1.5 of NI 43-101.

I have provided contract engineering services to this site related to the efficient operation of the mill and to report on Quality Assurance and Quality Control.

I have read NI 43-101 and this report has been prepared in compliance with that 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.

"Original Signed and Stamped"

John Eggert P.Eng.



Certificate of Author

I, William G. Hodgson, am a Professional Engineer employed as a Project Manager/Senior Mechanical Engineer with GENIVAR Inc. with an office at 740 Syndicate Avenue South, Thunder Bay, Ontario.

This certificate applies to the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012.

I am a member of Professional Engineers Ontario, Association of Professional Engineers & Geoscientists of Saskatchewan, Association of Professional Engineers and Geoscientists of New Brunswick, and Association of Professional Engineers of Yukon.

I graduated with a B.Eng Degree in Mechanical Engineering granted by Lakehead University in 1999. I have practiced my profession continuously since 1999 and have been involved with studies, detailed design, and construction reviews for site services, surface facilities, material handling, dewatering and underground ventilation systems for mining projects.

I am a "qualified person" for the purposes of National Instrument 43-101 due to my education, experience and current affiliation with a professional organization as defined in National Instrument 43-101.

I have visited the Minto Mine site twice, most recently on September 26-29 of 2011.

I am responsible for the preparation of sections 16.4.6.1, 18.1.2, and 18.2.1 of the Technical Report;

I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

My only prior involvement with the Minto Mine was a site trip in August 2011 to review the design parameters for a new ventilation raise, a project that was later deferred.

I have read National Instrument 43-101 and Form 43-101F. The part of the Technical Report that I am responsible for has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

As of the effective date of the Technical Report, 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, sealed and dated this 18th day of July, 2012

"ORIGINAL SIGNED AND STAMPED"

W.G. Hodgson, P.Eng
Project Manager/Senior Mechanical Engineer
GENIVAR Inc.

CERTIFICATE OF AUTHOR

I, Garth David Kirkham, do hereby certify that:

- 1) I am a Principal Geoscientist with Kirkham Geosystems Ltd. with an office at 6331 Palace Place, Burnaby, BC, V5E 1Z6.
- 2) This certificate applies to the "Minto Phase VI Preliminary Feasibility Study Technical Report" with an effective date of January 1, 2012.
- 3) I am a graduate of the University of Alberta in 1983 a B. Sc. I am a member in good standing of the Association of Professional Geoscientists of Ontario, Association of Professional Engineers and Geoscientists of the Province of Alberta, the Association of Professional Engineers and Geoscientists of BC, and the Northwest Territories and Nunavut Association of Engineers and Geoscientists. I have continuously practiced my profession performing computer modelling since 1988, both as an employee of a geostatistical modelling and mine planning software and consulting company and as an independent consultant.
- 4) I have not visited the property.
- 5) In the independent report titled "Minto Phase VI Preliminary Feasibility Study Technical Report" with an effective date of January 1, 2012, I am responsible for the Sections 14.5 and 14.6.
- 6) I have been involved with the Minto Mine as an author of the previous technical Report titled "Minto Phase IV Preliminary Feasibility Study Technical Report" and "Minto Phase V Preliminary Feasibility Study Technical Report".
- 7) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.
- 8) I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101
- 9) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- 10) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Signed, sealed and dated this 27th day of July, 2012.

"ORIGINAL SIGNED AND STAMPED"

Garth Kirkham, P.Geo.
Principal Geoscientist, Kirkham Geosystems Ltd.



SRK Consulting (U.S.) Inc.
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Lakewood, Colorado
80235

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CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

I, Michael E Levy, am a Professional Engineer, employed as a Senior Geotechnical Engineer with SRK Consulting Inc.

This certificate applies to the technical report titled "Minto Phase VI Preliminary Feasibility Study Technical Report" with an effective date of January 1, 2012 ("Technical Report").

I am a registered Professional Engineer in the states of Colorado (#40268) and California (#70578) and a registered Professional Geologist in the state of Wyoming (#3550). I graduated with a B.Sc. in Geology from the University of Iowa in 1998 and a M.Sc. in Civil-Geotechnical Engineering from the University of Colorado in 2004.

I have practiced my profession continuously since March 1999 and have been involved in a variety of surface and underground geotechnical projects specializing in advanced analyses and design of soil and rock slopes for mining projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Minto Mine site several times in the past five years, most recently on May 9-12, 2012.

I am responsible for the Executive Summary and Sections 16.2 of the Technical Report.

I am independent of Minto Explorations Ltd. and Capstone Mining Corp. as independence is described by Section 1.5 of NI 43-101.

I have been involved with the Minto Mine since 2006 participating and managing various independent studies and reviews.

I have read National Instrument 43-101 and this report has been prepared in compliance with that 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.

--Original Signed--

"Michael E Levy"

Michael E. Levy, P.E., P.G.

Dated: July 5, 2012

QP Certificate Levy_PHV1.docx



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www.capstonemining.com

CERTIFICATE of QUALIFIED PERSON

I, Brad Mercer, do hereby certify that:

I am the V.P. Exploration with the firm Capstone Mining Corporation the parent corporation of Minto Explorations Ltd. with an office at Suite 900-999 West Hastings Street, Vancouver, Canada;

I am a graduate of the Memorial University of Newfoundland with a Bachelor of Science degree in Earth Science, 1984. I have practiced my profession continuously since 1984;

This certificate applies to the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;

I am a Registered Professional Geologist in good standing in the NWT; NAPEGG #1581;

I, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

I am an employee of Capstone Mining Corporation and I do expect to benefit directly in the securities of Capstone Mining;

That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

I have read National Instrument 43-101 and Form 43-101F1 and I am a Qualified Person for the purpose of NI 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;

I visited the Minto Mine site many times from 2005 to 2012 and I am currently supervising ongoing technical work on the property;

I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Feasibility Study ("PFS") that is not reflected in the PFS, the omission to disclose which makes the PFS misleading;

I consent to the filing of the PFS with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the PFs;

I am responsible for and/or shared responsibility for Sections 1.1-1.3, 6-10 and 14.7 of the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;

Signed, Sealed and dated this 18th day of July, 2012 in Vancouver, British Columbia Canada

"ORIGINAL SIGNED AND STAMPED"

Stamp

Brad Mercer, P.Geol

Vice-President of Exploration, Capstone Mining Corp.



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Certificate of Author

I, Pooya Mohseni, P.Eng., do hereby certify that,

I am the Chief Engineer with the firm Minto Explorations Ltd. with an office at P.O. Box 33174, Whitehorse, Yukon, Canada, Y1A 6S1.

This certificate applies to the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;

I am a graduate of the University of British Columbia with a Master of Applied Science degree in Mining and Mineral Process Engineering.

I am a Registered Professional Engineering in good standing with the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC).

I am a "qualified person" for the purposes of National Instrument 43-101 due to my experience and current affiliation with a professional organization as defined in National Instrument 43-101;

I, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

I am an employee of Minto Explorations Ltd., a subsidiary of Capstone Mining Corporation and I do expect to benefit directly in the securities of Capstone Mining;

I am responsible for the preparation of Section 15.2, 16.5 to 16.7.3 of the Technical Report;

I visited the Minto Mine site many times in 2012 and I am currently supervising ongoing technical work on the property;

I have read National Instrument 43-101, Form 43-101F1 and part of the report that I am responsible for has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;

As of the effective date of the Technical Report, 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;

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed, sealed and dated this 27th day of July, 2012

"Original Signed and Stamped"

Signature of Qualified Person
Pooya Mohseni, P.Eng
Chief Engineer, Minto Mine



Certificate of Author

I, Bruce Andrew Murphy, FSAIMM, do hereby certify that,

1. I am a Principal Consultant (Mining and Rock Mechanics), with the firm SRK Consulting (Canada) Ltd., having an office at 2200-1066 West Hastings St., Vancouver, B.C., V6E 3X2;
2. This certificate applies to the report entitled "Minto Phase VI Preliminary Feasibility Study Technical Report" with the effective date of January 1, 2012;
3. I graduated with a MSc. Eng (Mining) degree from the University Witwatersrand, in May 1996.
4. I am a Fellow of the South African Institute of Mining and Metallurgy and as a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101).
5. I have been involved in mining since 1990 and have practised my profession continuously since then. I have been involved in mining operations, mining related rock mechanics and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia.
6. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of Capstone Mining Corp. ("Capstone"), or any associated or affiliated entities.
7. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Capstone, or any associated or affiliated companies.
8. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Capstone, or any associated or affiliated companies.
9. I am independent of Capstone as independence is described by Section 1.5 of NI 43-101.
10. I am a "qualified person" for the purposes of National Instrument 43-101 due to my experience and current affiliation with a professional organization as defined in National Instrument 43-101;
11. I am responsible for the preparation of Section 16.3 of the Technical Report;
12. I have read National Instrument 43-101, Form 43-101F1 and Section 16.3 of the Minto Phase VI Pre-Feasibility Study Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
13. As of the effective date of the Technical Report, 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;
14. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed, sealed and dated this 27th day of July, 2012

"ORIGINAL SIGNED"

Signature of Qualified Person

Bruce Andrew Murphy, FSAIMM

Principal Consultant, Mining Rock Mechanics at SRK Consulting (Canada) Ltd.



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Certificate of Author

I, Colleen Roche, P.Eng., do hereby certify that,

I am the Manager of Sustainability and Environmental Affairs with the firm Capstone Mining Corp., the parent company of Minto Explorations Ltd. with an office at Suite 900 – 999 West Hastings, Vancouver Canada;

This certificate applies to the report entitled “Minto Phase VI Preliminary Feasibility Study Technical Report” with the effective date of January 1, 2012;

I am a graduate of McGill University with a Bachelor of Engineering (Mining) degree, 2000. I am also a graduate of the Masters of Engineering in Mining program from the University of British Columbia, 2008. I have practiced my profession continuously since 2000;

I am a Registered Professional Engineer in good standing with the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC # 36041). I am a “qualified person” for the purposes of National Instrument 43-101 due to my experience and current affiliation with a professional organization as defined in National Instrument 43-101;

I, am not independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

I am an employee of Capstone Mining Corporation and I do expect to benefit directly in the securities of Capstone Mining;

I am responsible for the preparation of Section 1.8 and 20 of the Technical Report;

I visited the Minto Mine site many times from 2007 to 2012;

I have read National Instrument 43-101, Form 43-101F1 and part of the report that I am responsible for has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;

As of the effective date of the Technical Report, 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;

I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed, sealed and dated this 27th day of July, 2012

“Original Signed and Stamped”

Signature of Qualified Person
Colleen Roche, P.Eng
Manager, Sustainability and Environmental Affairs