May 31, 2012

REPORT ON

Pre-feasibility Block Cave Mine Design - Iron Cap Deposit

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REPORT

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Executive Summary

This report presents the results of the pre-feasibility assessment of the proposed block caving mine for the Seabridge Gold Inc. Iron Cap deposit, part of the KSM property located in the Coast Mountains of northwestern British Columbia. The property is situated in challenging topography with potential for the development of three open pit and two underground mines. The deposit extends approximately 1,200 m SW-NE (along strike), 700 m NW-SE, and 700 m vertically. It is understood that the deposit remains open at depth. Open pit mining methods were used to evaluate the mining potential of this deposit as part of an update to a PFS published in 2011. However, due to various environmental concerns such as pre-stripping the overlying ice cap and creating additional potentially acid generating waste, Seabridge decided to assess other mining options. Golder was engaged to evaluate the potential to mine the Iron Cap deposit using block caving methods to the pre-feasibility level of engineering study. The location, dimensions, and dip of the mineralized material at Iron Cap indicated that it was a suitable candidate for block caving.

The mineral resource block model used for the study contained Gold (Au), Silver (Ag), Copper (Cu), and Molybdenum (Mo) grades as well as a Net Smelter Return (NSR) value based on the NSR formula in the pre-feasibility update (PFU) that was published on June 15, 2011. The model also contained measured, indicated, and inferred grades but the inferred grades were set to zero and are not included in this pre-feasibility study. The geological resource contains 362M tonnes of mineralized material grading 5.4 g/t Ag, 0.44 g/t Au, 0.21% Cu, and 37 ppm Mo. This resource was evaluated using Gemcom's Footprint Finder software to evaluate the economic potential for a block cave mine. A footprint at elevation 1210 m produced the most value and resulted in approximately 193M tonnes of block cave resources, including 5% unplanned waste dilution at zero grade as shown in Table A.

Table A: Summary of the Geological and Block Cave Resources

Category	Tonnes (millions)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological Resources[1]	362	5.4	0.44	0.21	37
Mineral Inventory	321	6.3	0.52	0.23	23
Block Cave Resources[2,3]	**193**	**5.3**	**0.45**	**0.20**	**21**
Dilution	16	0	0	0	0
Recovery	60%				
Total Dilution	5%				

[1] Geological Resources presented in Table 1.1 of the Pre-feasibility Update report (Seabridge 2011).

2 PCBC includes column mixing with dilution and shutting of columns (drawpoints) when NSR < $15.41 so a portion of the diluted mineral inventory is not recovered.

3 Block Cave Resources can be considered as Probable Mineral Reserves within the complete pre-feasibility study report.

The Iron Cap deposit appears to be composed of strong, moderately fractured rock. Rock quality variations are most commonly attributed to variations in fracture frequency as the strength of the rock mass does not vary significantly within the deposit. The fracture frequency is higher for Iron Cap than the nearby Mitchell deposit, resulting in a corresponding lower predicted median in situ block size of 2.5 m^3 compared to approximately 6 m^3 for the Mitchell deposit. No major structural features have been identified that might influence the caving mechanism and the progression of the cave in any significant manner. There are several gaps in data that have been identified in the geotechnical and hydrogeological studies. These gaps will need to be addressed as part of future studies if the project is advanced to the next level of study. The cavability assessments made using Laubscher's and Mathews' methods indicate that the size (diameter) of the footprint required to initiate and propagate caving is between approximately 100 m and 220 m. This footprint size is significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized, continuous nature of the deposit, suggests that the Iron Cap deposit is amenable to cave mining. There have been no fracture propagation assessments applicable to preconditioning designs or in situ stress interpretations developed at the Iron Cap deposit. Measurements carried out in the Mitchell deposit may not accurately reflect the fracture propagation and stress environment at Iron Cap because of the effects of surface topography. Future drilling programs should include hydraulic fracturing tests.

A significant proportion of the rock at Iron Cap is predicted to have block sizes greater than 2 m^3. Without some remediation measure being adopted, such large sized blocks will require significant secondary blasting, and there will likely be a significant adverse impact on production and significant damage to the drawpoints that will require ongoing rehabilitation. As a result of this, it is proposed to precondition the rock by hydrofracturing. The cost and schedule to do this have been incorporated into this study. There are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. It is also difficult to obtain definitive field data that demonstrates the degree of improvement obtained. The results from these mines are encouraging, however, and there is sufficient experience to indicate that such fragmentation concerns do not represent a fatal flaw at Iron Cap. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning to enhance fragmentation needs to be addressed via production and cost risks. It is also very difficult to quantify the effect of attrition as the rock is brought down within the cave except that experience has indicated that in caving mines operating under similar rock conditions to those at Iron Cap, fragmentation of rock, drawn down more than approximately 100 m is generally good. For this study, it was assumed that fragmentation of the initial 100 m of draw height is approximately equal to the estimated in situ block size and, above this, only limited secondary blasting would be required.

The expected coarse fragmentation at Iron Cap will result in relatively large isolated drawcone diameters of 13 m or more for a loading width of 5 m. The present experience in other operating mines is that a 15 m by 15 m drawpoint spacing performs well under these coarse fragmentation conditions. Some caving mines operating in good quality rock have successfully expanded the layout to approximately 17 m by 17 m, but it was considered prudent for this study to adopt the slightly more conservative 15 m by 15 m spacing.

The Iron Cap block cave design was based on modelling from FF and PCBC software. FF modelling indicated that the optimum footprint for the Iron Cap deposit is at an elevation of 1,210 m. It is approximately 545 m wide in the north-south direction, 570 m wide in the east-west direction, and has an average depth of 400 m. PCBC modelling indicated that the block cave could produce 15 million tonnes per year, requiring development of 120 new drawpoints per year. The mine design requires approximately 64 km of drifts and raises, including a 5% contingency to account for the excavations of detailed design items such as service bays, sumps and electrical substations. Four main levels are required to cave the Iron Cap deposit and include the preconditioning level, undercut level, extraction level, and conveying level. The design also includes a return air drift located below the conveying level. The floors of the extraction drifts and drawpoints are designed to be concreted, which will increase the speed and productivity of the Load-Haul-Dump (LHD) vehicles as well as reduce equipment maintenance. Personnel, material, and supplies will access the Iron Cap mine through a drift driven from the Mitchell Tiegen Tunnel (MTT). A conveyor drift will be driven parallel to the access ramp, and the two will be connected every 300 m to provide emergency egress and a ventilation loop during construction. The total length of the access ramp is 3.4 km. Two fresh air portals and one exhaust portal are planned on the north slope of the Mitchell valley. These tunnels may act as an alternative access to the underground from the surface in case of emergency. The fresh air tunnels will connect to surface and a perimeter drift will be constructed around the entire mine footprint to provide fresh air to the mine workings.

Production material will be hauled directly from the drawpoints to one of four gyratory crushers installed on the extraction level perimeter drift. The crushed material will be transported by one of two conveyor belts which both feed a conveyor that will transport the production material to a 2,000 tonne surge bin located above the MTT conveyor.

The proposed mobile equipment is typical of that used in underground mines and is comprised of those pieces directly related to moving ore to the crushers (8.6 m^3 LHDs and secondary rockbreakers), the development equipment (4.6 m^3 LHDs and 18 m^3 trucks) as well as the AnFo loaders and ground support machines. In addition, service equipment is included for construction and mine maintenance activities. At peak operation, Iron Cap will require a fleet of approximately 67 pieces of mobile underground equipment. The Iron Cap mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life with a peak quantity of 548 personnel in Year 7.

The required airflow for the Iron Cap mine is 526 m^3/s based upon the total diesel equipment used on each mining level including a minimum 20% contingency. The Iron Cap ventilation model is designed to operate as a positive pressure or forced air system to facilitate mine air heating during the winter months and to prevent any air being drawn into the mine through the caved material. Heating of mine air in the winter months is included in the design and cost estimates for Iron Cap. It is estimated that the Iron Cap mine will require approximately 9,200 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans.

The underground water management system at Iron Cap is currently designed to handle 7,640 m^3/d. This caters for the groundwater inflow and the ice melt. At the time of completing this pre-feasibility assessment, estimates by others of the surface inflows into the crater at Iron Cap were not available. These surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. In future studies, the water management system will need to be enhanced to cater for this additional inflow. To provide for good drainage, the underground drifts have been graded so that water will run towards the MTT or towards the Mitchell Valley. Water exiting the mine will be collected and processed in existing “dirty water” facilities.

The mine development schedule was separated into three phases; an initial pre-production phase which involves developing the primary access ramp and conveyor drifts; a second, ore production phase, that involves creating enough openings to start and ramp-up production from the cave; and, the final phase, once the mine has reached steady-state production and the development fleet is only required to create enough openings to maintain production. The average annual development quantity during the peak development period is about 10,000 m per year.

The mine production schedule was developed using Gemcom's PCBC software. It was assumed that sloughing of peripheral waste rock would occur into the crater and cover the upper surface of the material being drawn down. This was modeled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution. Due to the large fragmentation that is estimated to report to the drawpoints at Iron Cap, particularly during the early stages of mining, a draw rate of 200 mm/day was chosen as a maximum cap in the PCBC analysis but an average draw rate of 108 mm/day is required to reach production targets (the maximum draw rate modeled never exceeds 165 mm/day, and averages about 110 mm/d, so there are roughly twice as many drawpoints available as are required to meet production targets). Initially, it is assumed that a drawpoint can produce at 60 mm/day and that this will steadily increase until 50% of a column is mined. Then, the drawpoint will produce up to the set maximum of 200 mm/day. Iron Cap is estimated to have a production ramp-up period of 4 years, steady state production at 15 million tonnes per year for 9 years, and then ramp-down production for another 6 years.

The average mine operating cost is estimated at $6.15/tonne and consists of the equipment and labour that is required to move material from the drawpoint to the MTT conveyor tunnel and the fixed costs to run the mine (Table B). This includes the use of the LHDs, crushers, conveyors, mine services and the labour required to plan and execute the mining plan. Mine labour comprises approximately 56% of the total Iron Cap underground mining cost while crushing and conveying is 17%, production mucking is 13%, 9% accounts for fixed costs and secondary breaking is 5%.

Table B: Underground Mine Operating Cost Breakdown

Activity	OPEX ($/tonne mined)	(%)
Labour	$ 3.45	56%
Crushing and Conveying	$ 1.04	17%
Production Mucking (LHD)	$ 0.82	13%
Fixed	$ 0.53	9%
Mobile Rockbreaking	$ 0.28	5%
Rehabilitation	$ 0.04	0%
Total	**$ 6.15**	

The mine capital cost estimate includes the purchase and installation of all equipment and the excavation of all the underground workings. The pre-production capital expenses, over the first 5 years of the mine life, are estimated at $509 million with an average sustaining capital cost of $46 million over the remaining 21 years. The life-of-mine capital costs are estimated to be $1.5 billion.

Study Limitations

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Table 1: Units Used in the Text

Unit	Definition
m	Meter
km	Kilometres
mm	Millimetres
g/t	Grams per tonne
% Grade	Grade item in % (such as Copper)
US$/t	US dollars per tonne
$	Dollars - assumed Canadian unless specified
M	Million
%	Percent
ppm	Parts per million
m^2	Square meters
m^3	Cubic meters
m/s	Meters per second
MPa	Mega Pascal's
FF/m	Fracture frequency per meter
°	Degrees in an angle
Q'	Modified Q (Barton's rock mass classification system)
N	Stability Number
"	Inch
m^3/s	Cubic meters per second
kW	Kilo Watt
kWh	Kilowatt hour
HP	Horsepower
Pa	Pascal
BTU	British thermal unit
MMBTUH	Million British thermal units per hour
°C	Temperature - Degrees Celsius
cfm	Cubic foot per minute
cfm/bhp	Cubic foot per minute per boiler horsepower
Ns^2/m^8	Gaul - Resistance of an airway when one cubic meter per second air causes a pressure drop of one Pascal
m^3/d	Cubic meters per day
m^3/hr	Cubic meters per hour
mm/day	Millimetres per day
$/m	Dollars per meter

Unit	Definition
$M	Million Dollars
$/tonne	Dollars per tonne
Mtonnes	Million tonnes
tpd	tonnes per day

1.0 INTRODUCTION

Seabridge Gold Inc.'s (Seabridge) KSM project is a major gold-copper deposit located in northwest British Columbia (BC), approximately 40 km southwest of the Bell II lodge on Highway 37, and 21 km south-southeast of the Eskay Creek Mine (Figure 1). An aerial view looking to the east is shown in Figure 2. The site characteristics are described in detail in the KSM pre-feasibility study (PFS) report (Seabridge 2011).

The KSM property contains the Kerr, Sulphurets, Mitchell, and Iron Cap deposits. Golder Associates Ltd. (Golder) completed the pre-feasibility level assessment of block cave mining for the Mitchell and Iron Cap deposits. This report presents the results of the pre-feasibility assessment of the proposed block caving mine for the Iron Cap deposit. A similar evaluation for the Mitchell deposit will be presented under separate cover.

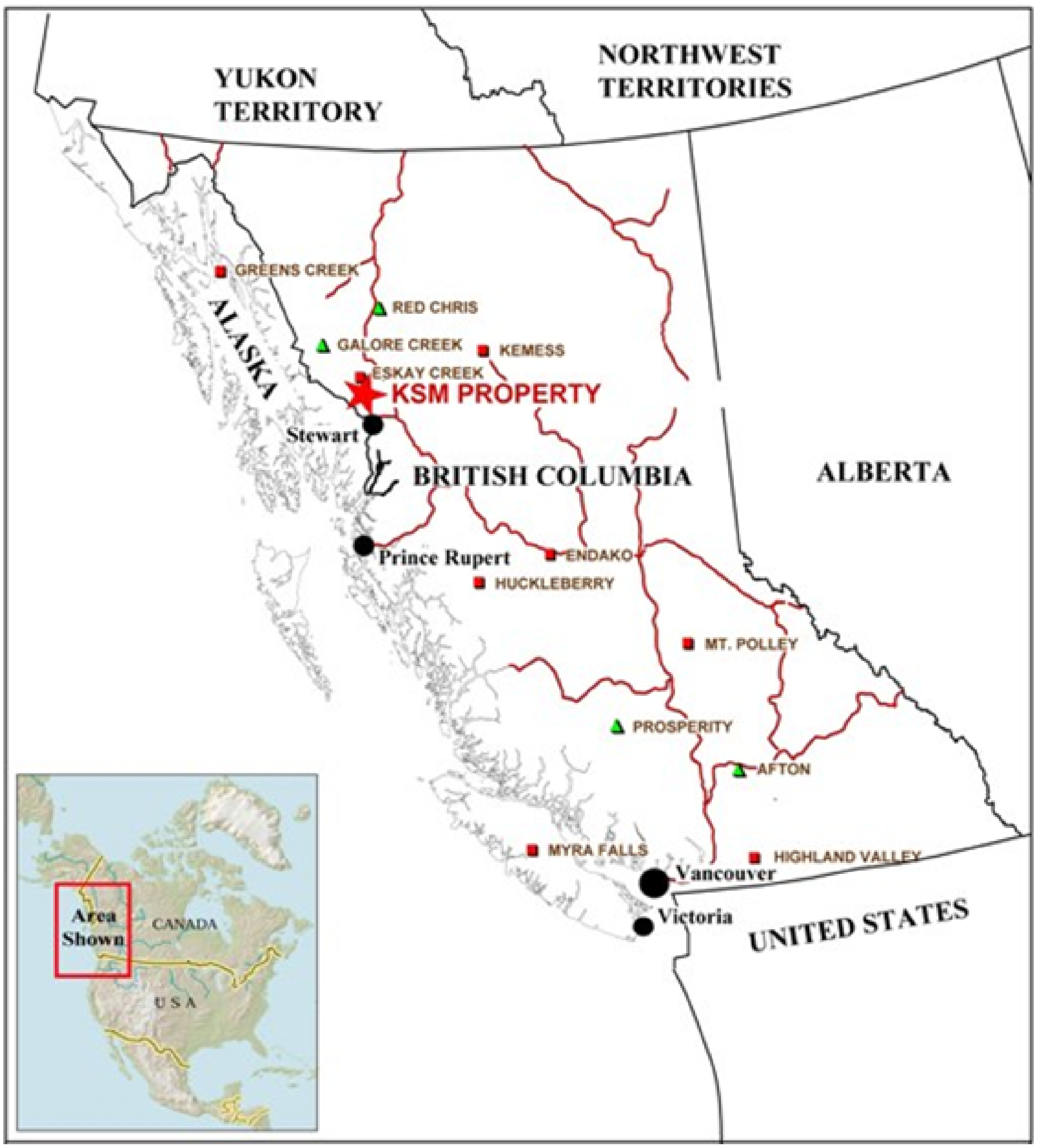


Figure 1: Location of the KSM property

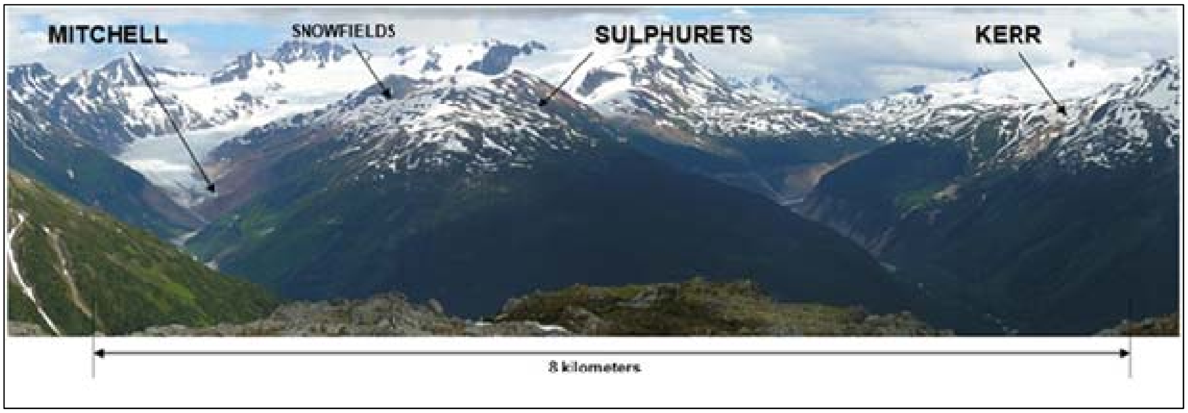


Figure 2: Aerial view of the general area of the KSM project with the Iron Cap deposit located to the north (left) of Mitchell in this photo (looking East)

1.1 Mining Concept

The Iron Cap deposit is a porphyry type intrusion that has been deformed by subsequent tectonic processes. The deposit outcrops in the north slope of the Mitchell valley, east of the Mitchell deposit and above the current Mitchell glacier. A small portion to the north-east is covered with an ice cap. Figure 3 shows a section through the Iron Cap deposit with a 0.25 g/t gold (Au) grade shell and the proposed block cave design. The site terrain is shown in Figure 4.

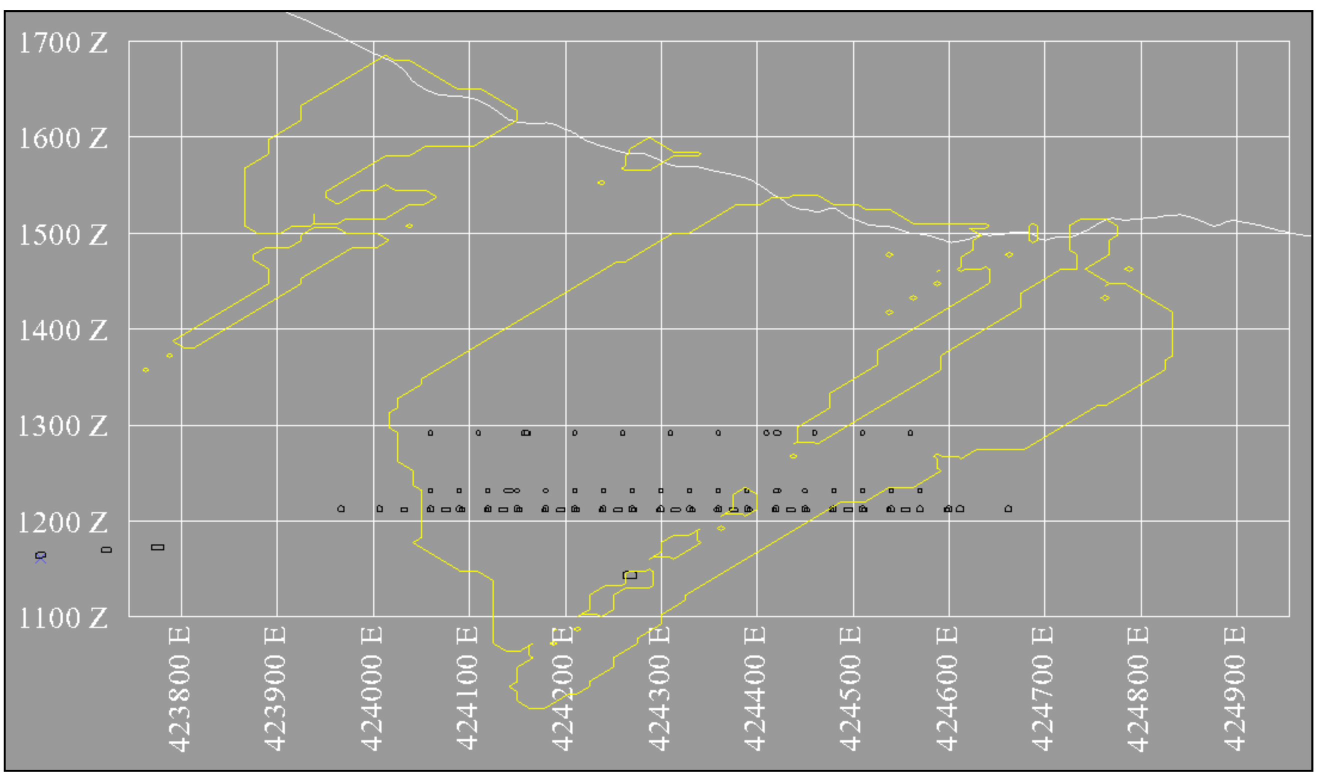


Figure 3: Cross Section at Northing 6,267,300 Showing Topography (White), Iron Cap 0.25 g/t Au Grade Shell (Yellow) and Proposed Block Cave Mine Design (black)

Figure 4: Site Terrain

1.2 Scope of Work

Iron Cap is part of the KSM property located in the Coast Mountains in northwestern British Columbia amid challenging topography with the potential for the development of three open pit and two underground mines. Several engineering consultants were engaged by Seabridge to evaluate the technical issues and economic potential of the property as part of an update to a PFS published in 2011. Golder was engaged to evaluate the potential to mine the Iron Cap deposit using block caving methods to the pre-feasibility level of engineering study and to include the following scope:

- underground mine access;
- fragmentation of the caved rock as it reports to the drawpoints;
- drawpoint spacing to maximize recovery and minimize dilution;

- stability assessments and support requirements for all underground excavations;
- drawpoint layout and extraction level design;
- mine ventilation and services (de-watering, shops, etc);
- mine development and production schedules;
- mine equipment selection; and
- capital and operating cost estimates of the block caving operations.

The design and cost estimation of the material handling system (e.g., all conveyors and crusher installations) to deliver material from the underground drawpoints to the Mitchell Teagan Tunnel (MTT) was completed by Boche Ventures and Wardrop. Also, the design and cost estimation for the underground electrical system required for underground mining was completed by WN Brazier Associates Inc. Golder was not involved in the design of the surface infrastructure, except where it relates directly to the underground operations (e.g., ventilation raises).

2.0 GEOLOGICAL SETTING

The Iron Cap deposit is a porphyry-type intrusion. A general view of the outcrop of the Iron Cap deposit and the surface expressions of relevant geological features are shown in Figure 5 The country rock is comprised mostly of deformed sediments (e.g., sandstones, siltstones), volcaniclastics (e.g., tuffs, pyroclastic breccias), and volcanics (e.g., basalts, andesite flows). The ore zone is located in the Hazelton Group rocks in the footwall of the Sulphurets Thrust Fault (STF).

Figure 5: Overview of the Iron Cap project area looking north from the south side of the Mitchell Valley (BGC 2011)

The geological information for the Iron Cap deposit provided by Seabridge includes the following:

- lithology;
- alteration;
- major faulting; and
- Au and Cu grade shells of 0.25 g/t Au and 0.1% Cu.

Less is understood about the Iron Cap deposit than the Mitchell deposit, other than that quartz-sericite-pyrite alteration is more intense at Iron Cap than at Mitchell and that there appears to be more base metal mineralization, particularly in narrow veins.

Major geological structures and fabrics of the study area include the following:

- north-south striking steeply dipping faults;
- gently dipping thrust faults, and striking east-west; and
- moderately to steeply dipping foliation/schistosity.

The geometrical shapes of the 0.25 g/t Au and 0.1% Cu grade shells are very similar and superimpose one another, as shown in Figure 6. The deposit extends approximately 1,200 m SW-NE (along strike), 700 m NW-SE (in plan in the down dip direction), and 700 m vertically. The deposit is massive and reasonably continuous and in general geometrically suitable to mine by block caving. It is understood that the deposit remains open at depth.

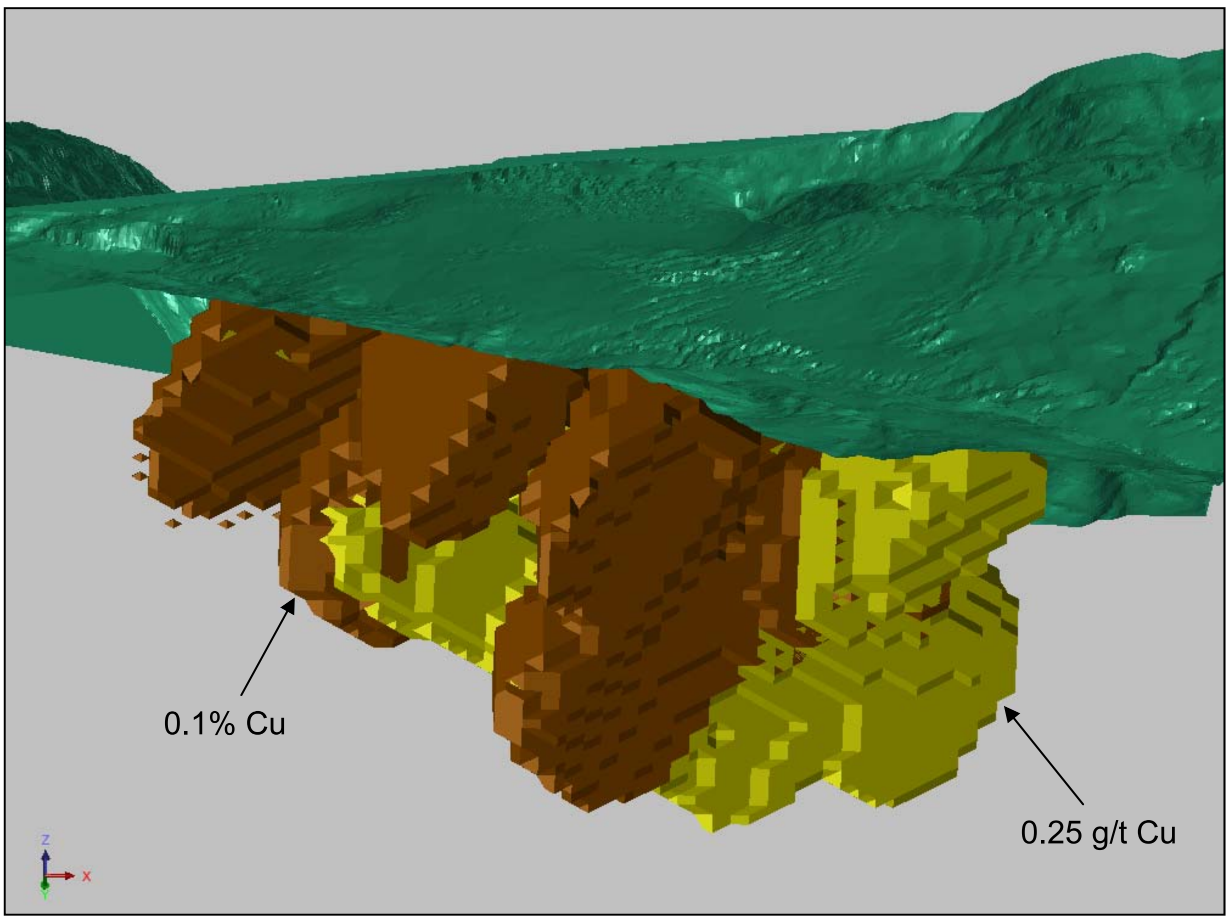


Figure 6: Isometric view showing 0.25 g/t Au and 0.1% Cu grade shells of the Iron Cap deposit (looking north)

A vertical cross-section toward the center of the deposit showing lithology, alteration, structures and grade shells is presented in Figure 7. The lithological units within the area of potential block cave mining (above the underground extraction level) are primarily altered volcanics that lie beneath the STF.

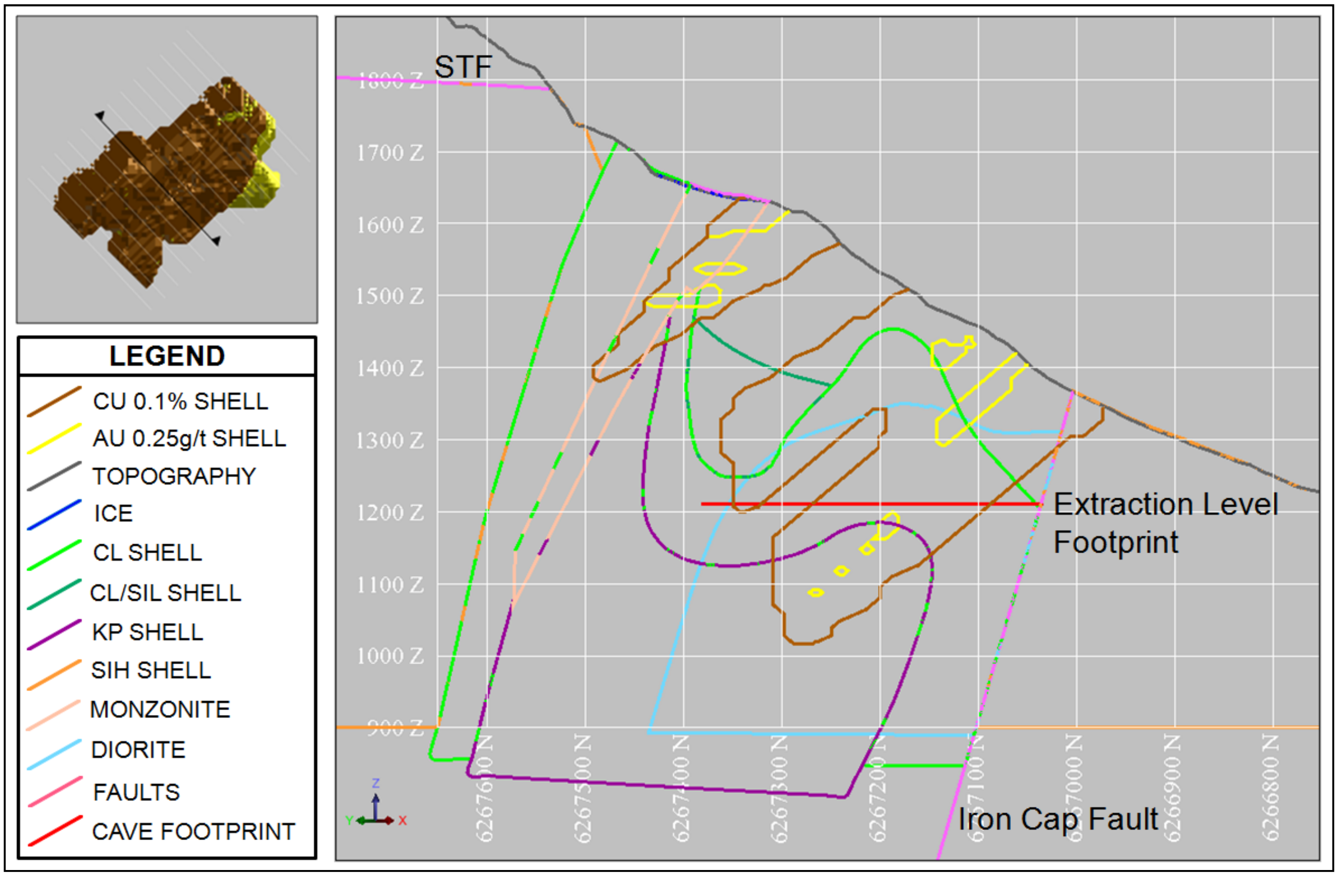


Figure 7: Vertical cross-section of the Iron Cap deposit showing lithology, alteration, faults, and 0.25 g/t Au and 0.1% Cu grade shells

Primary alteration types in the Iron Cap zone are phyllic or quartz-sericite-pyrite (QSP) and chloritic, with overprinting from silica flooding and hydrothermal brecciation. These alteration types are generally associated with the mineralized zone and immediately surrounding rock.

Outside of the mineralized and QSP altered area of the Iron Cap zone, the alteration types are dominantly potassic, siliceous, and hornfels. Geotechnical studies carried out by Bruce Geotechnical Consultants (BGC 2011) for preliminary pit designs did not identify any correlations between these alteration types and rock mass quality.

A summary of the lithological units and primary alteration types in the Iron Cap zone is contained in Table 2.

Table 2: Iron Cap Lithological Units and Primary Alteration Types

Geologic Unit	Lithology	Alteration Types	Comments
"Mitchell Intrusives" (Jurassic)	Feldspar Porphyry, Monzonite, Andesite, Diorite	Potassic, Hornfels	Above the core of the Iron Cap zone, there is a relatively large intrusive body located within the Hazelton Group volcanic. The upper slope of the Mitchell Valley has a large percentage of volcanic rocks. There are also intrusives located within the mineralized zone of the Iron Cap deposit.
Hazelton Group (Jurassic)	Volcaniclastic, Tuff, Volcanics, Sedimentary	Phyllic (QSP), Hydrothermal Brecciation, Intermediate Argillic, Chloritic, Silicic	The mineralized zone of the Iron Cap deposit is a mixture of highly altered and mineralized volcanics and intrusives belonging stratigraphically to the Hazelton Group.
		Chloritic, Propylitic, Hornfels, Potassic, Silicic	The Hazelton Group rocks are located in the footwall of the STF. Alteration in this unit can be intense, as the core of the deposit is located in it.
Stuhini Group (Triassic)	Volcaniclastic, Tuff, Volcanics,	Phyllic (QSP), , Intermediate Argillic, Chloritic, Propylitic, Silicic	The Stuhini Group is located in the STF hanging wall. It represents a back-arc basin package and is the host rock of the intrusives. Alteration in this unit can be intense close to the STF, where the core of the Sulphurets zone begins.

Taken from BGC (2011).

There are a number of regionally significant structures identified in the Iron Cap zone. These include the STF and the Iron Cap normal fault, as well as bedding, and foliation. An isometric view of the deposit showing the surface topography, mineralization, STF, and Iron Cap fault is shown in Figure 8.

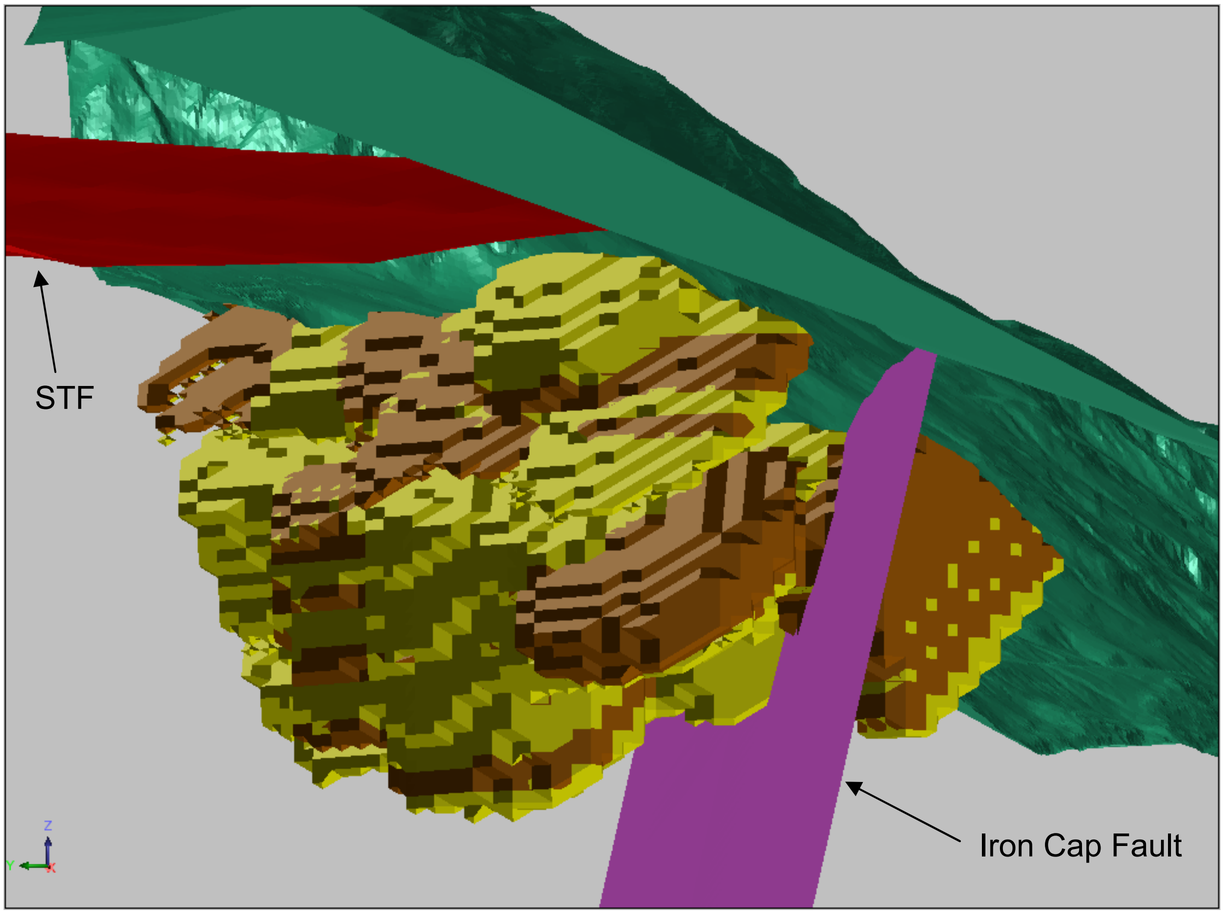


Figure 8: Isometric view showing 0.25 g/t Au and 0.1% Cu grade shells, Sulphurets Thrust Fault and Iron Cap Fault (looking east)

The Iron Cap fault dips steeply to the north and is located at the south end of the proposed block cave footprint. Based on rock quality data from exploration borehole IC-10-034, which intersects the fault at an elevation of approximately 1,210 m, this structure is not anticipated to be geotechnically significant and does not require additional design considerations.

Bedding is not very evident in the Hazelton Group rocks that contain the mineralization. The orientation of the foliation is variable and typically dips to the south at moderate to steep angles.

3.0 MINERAL RESOURCES

The mineral resources were provided by Seabridge as a text file to Golder. These were then forwarded to Gemcom for used in the Footprint Finder (FF) and PCBC modelling. The block model contained Gold (Au), Silver (Ag), Copper (Cu) and Molybdenum (Mo) grades as well as a Net Smelter Return (NSR) value based on the NSR formula in the pre-feasibility update (PFU) (Seabridge 2011). The model also contained measured, indicated and inferred grades. The inferred grades were set to zero and are not included in this pre-feasibility study.

3.1 NSR Cut-Off

The underground block cave mining cost was determined from first principals and is discussed further in Section 11.3. The PFU report provided the general and administration (G&A) and milling costs as shown in Table 3.

Table 3: Components of the NSR Cut-Off

Item	US$/t Milled
Block Cave Mining[1]	6.00
Milling, G&A and Site Service	9.57
Total	**15.57**

[1] The mining cost used to determine the resources discussed in this section was a preliminary one. More details on the mining cost can be found in Section 11.3.

3.2 Block Cave Resources

The following definitions are applicable to this report.

- *Geological resources* are as presented in the PFU (Seabridge 2011) and include all of the measured and indicated mineral resources.
- *Mineral inventory* is the portion of the geological resources above the NSR cut-off.
- *Dilution* is defined as material with zero grade within the footprint that is mined including the inferred material.
- *Block cave resources* are the quantity of measured and indicated material within the footprint that have an NSR > $15.57, as determined by PCBC and includes dilution.
- *Recovery* is the ratio of block cave resources to the mineral inventory, and represents the proportion of potentially economic material recovered in the mine plan.

In block caving, the recovery is affected by the quantity of planned and unplanned dilution that reaches the drawpoint, which is governed in part by draw control (the mines ability to maintain a constant cave back and reduce fines/dilution migration to the drawpoint). Good draw control can be hindered by lost drawpoints due to very large muck, drawpoint collapse due to ground and/or mining conditions, or poor mine planning (over-mucking from certain drawpoints and not enough from others creating an uneven cave back).

The Iron Cap deposit contains 362 M tonnes of geological resources grading 5.4 g/t Ag, 0.44 g/t Au, 0.21% Cu and 37 ppm Mo. This resource was evaluated using Gemcom's FF software (the FF results will be discussed in Section 4.0) to evaluate the economic potential for a block cave mine, and Gemcom's PCBC software (discussed in Section 10.0) to produce a schedule of mined tonnes and grade (the block cave resources). The block cave resources are estimated to be 193 M tonnes including 5% waste dilution (zero grade). A summary of the Iron Cap mineral resources can be found in Table 4 and Table 5.

Table 4: Summary of the Geological and Block Cave Resources

Category	Tonnes (millions)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological Resources[1]	362	5.4	0.44	0.21	37
Mineral Inventory	321	6.3	0.52	0.23	23
Block Cave Resources[2,3]	**193**	**5.3**	**0.45**	**0.20**	**21**
Dilution	16	0	0	0	0
Recovery	60%				
Total Dilution	5%				

[1] Geological Resources presented in Table 1.1 of the PFU (Seabridge 2011).

2 PCBC includes column mixing with dilution and shutting of columns (drawpoints) when NSR < $15.41 so a portion of the diluted mineral inventory is not recovered.

3 Block Cave Resources can be considered as Probable Mineral Reserves within the complete pre-feasibility study report.

Table 5: Mineral Resources Recovered at Drawpoints.

Category	Tonnes (millions)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Measured	0	0	0	0	0
Indicated	177	5.8	0.49	0.21	23
Measured and Indicated	**177**	**5.8**	**0.49**	**0.21**	**23**
Inferred	6	4.6	0.33	0.19	43
Waste	10	0	0	0	0
Total	193	5.3	0.45	0.20	21

4.0 PRELIMINARY MINING ASSESSMENT

Open pit mining methods were used in the PFU to evaluate the mining potential of this deposit. However, due to various environmental concerns such as pre-stripping the ice cap and creating additional potentially acid generating waste, Seabridge decided to assess other options.

The location, dimensions, and dip of the mineralized material at Iron Cap indicated that it was a candidate for block caving. Block caving is a low cost underground mining method and has the potential to achieve very high production rates. However, it requires a significant investment of time and money prior to the start of production mining. Because of the potential for low operating costs and high production rates with block caving, other underground mining methods were not investigated.

Gemcom's Footprint Finder (FF) was used initially to investigate the possibility of mining Iron Cap as a block cave. FF provides estimates of the value of columns of the block model at different elevations. The goal is to determine at which elevation a caving footprint would be the most successful (i.e., the widest) and the most profitable. FF is a coarse tool used to evaluate the potential for a deposit to be mined by block caving. Additional information concerning the FF module is presented in the Golder report "Block Cave Mine Study" (Golder 2011a).

4.1 Footprint Finder Inputs

Footprint Finder requires a block model of the mineralized material includes a value attribute, such as grade or NSR, and cost inputs to evaluate the potential profitability of caving a mineral deposit. FF used the NSR block model discussed in Section 3.0. Table 6 shows the key input values that were used. The mining costs used estimated prior to the more detailed cost estimate being undertaken and at this stage, were mostly based on experience. The exception was "Other operating costs" which were based on the PFU (Seabridge 2011). Additional details concerning the inputs and their definitions are presented in the Golder report "Block Panel Caving Conceptual Study for the KSM project" Golder (2011b).

Table 6: Key Input Values Used in Footprint Finder to Evaluate the Block Caving Potential of Iron Cap

Footprint Finder Input	Value
Incremental Horizontal Capital Cost	\$ 1,075 per m^2
Incremental Vertical Capital Cost	\$ 112,000 per m
Fixed Capital Costs	\$ 100M
Mining Operating Cost	\$ 5.40 per tonne
Other Operating Costs (milling, G&A)	\$ 8.41 per tonne
Maximum Column Height	500 m

4.2 Footprint Finder Results

A summary of the FF results is shown in Figure 9. A footprint at elevation 1,150 m has the most tonnage ($1,292M and 215M tonnes), while a footprint at 1,210 m has the most value ($1,385M and 207M tonnes). The Iron Cap block cave design was based on the 1,210 m elevation footprint, with tonnage and grade summary presented in Table 7 and the footprint geometry shown in Figure 10.

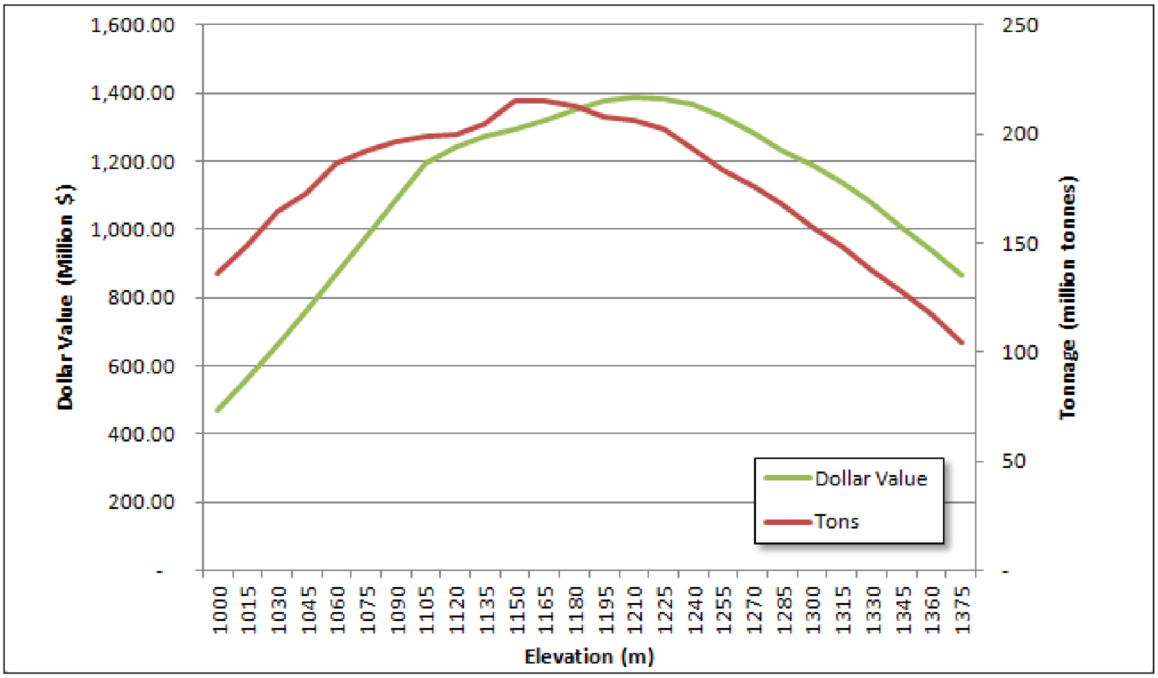


Figure 9: Summary graph of the footprint finder results for the Iron Cap deposit

Table 7: Summary of the Footprint Finder Results for the Footprint Chosen (1210 M Elevation)

Elevation (m)	Tonnage (Mtonnes)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
1,210	207	0.47	0.2	5.63	21.85

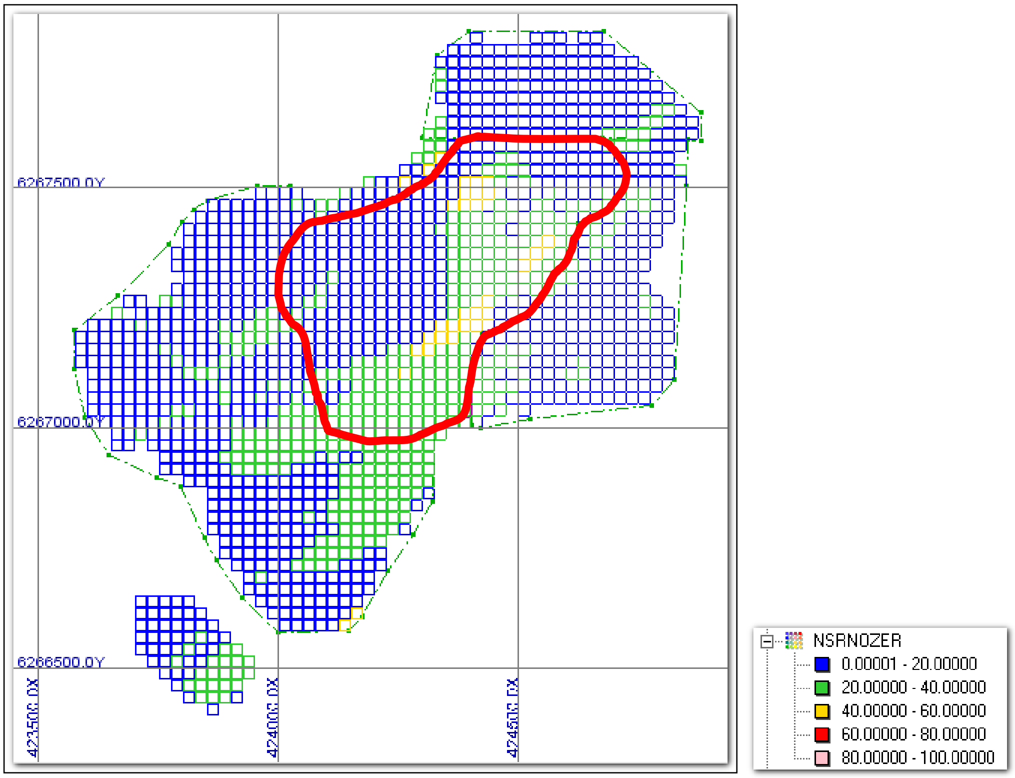


Figure 10: Iron Cap footprint (in red) with the dollar value of columns of the geological resource block model at 1210 m elevation

5.0 GEOTECHNICAL CHARACTERIZATION

The characterization of the rock mass has focused on the rock in and around the extraction and undercut levels of the proposed block cave mine and on the mineralized rock above this that will be caved. Rock within 50 m of the ground surface is expected to be of poorer quality due to weathering. This rock will not have a significant impact on the caving response of the mineralized rock, and geotechnical information from this rock has not been included in the characterization of the rock mass that will be block caved.

Characterization of the rock was based on core photographs and data collected for exploration drillholes, detailed geotechnical data collected for drilling programs carried out by BGC in 2010 (BGC 2011), and an interpreted geological model provided to Golder by Seabridge.

There are a total of 41 exploration holes and three geotechnical holes in the Iron Cap deposit. The borehole locations are shown in Figure 11. Only those holes that are near or that intersect the mineralized rock above the proposed block cave extraction level (El. 1,210 m) are considered here. Geotechnical boreholes are shown in red.

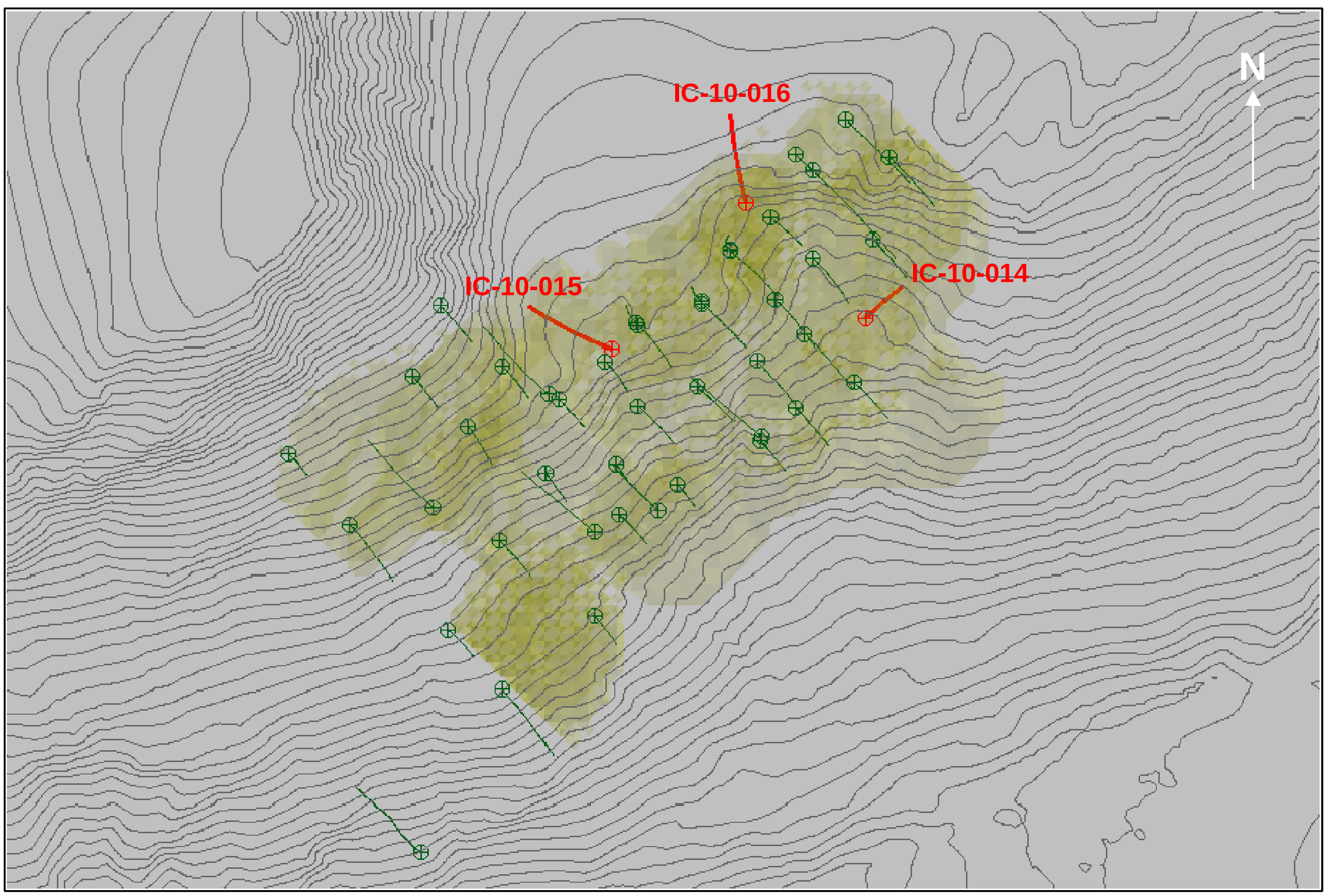


Figure 11: Iron Cap exploration and geotechnical boreholes and 0.25 g/t Au grade shell

Some of the block cave mine infrastructure (i.e., ramps, raises, conveyor drifts, etc.) will be located outside of the immediate area of mineralization. This infrastructure, including the access ramp, main conveyor, and ventilation drifts, is shown in Figure 12. For the purpose of this report, the rock outside the immediate area of mineralization where some of the infrastructure is located is referred to as “host” rock. The host rock that the mine infrastructure will be excavated in has been assessed based on data collected from nearby drillholes, where available.

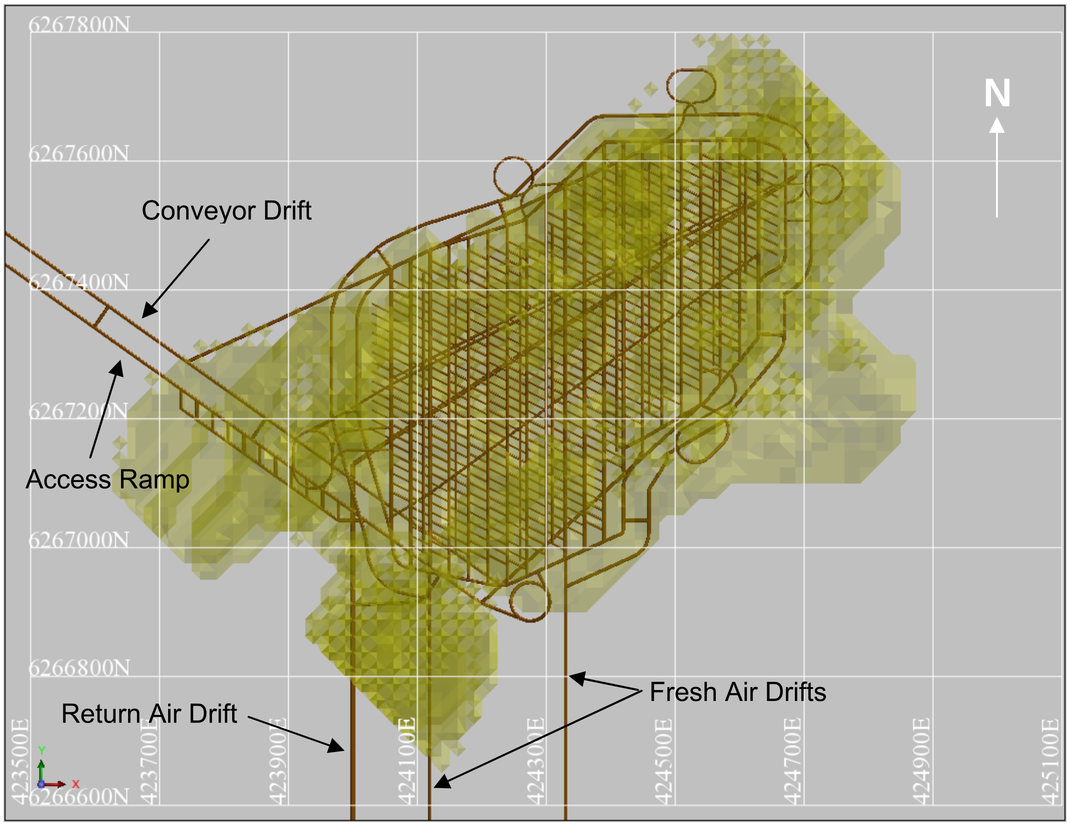


Figure 12: Plan showing mine infrastructure and 0.25 g/t Au grade shell

Figure 12 is included here for illustration purposes only to indicate where the mine infrastructure is planned relative to the mineralized material. Details of the mine design will be discussed in later sections of this report.

The key components of the rock mass characterization are summarized below. A more detailed description of the rock mass characterization, and the data on which it is based, is contained in the Golder geotechnical characterization report (Golder 2012a). Further site characterizations and geotechnical conditions are presented in BGC’s report (BGC 2011).

5.1 Rock Mass Rating

The geotechnical boreholes were logged for rock quality according to the Rock Mass Rating (RMR_{76}) system (Bieniawski 1976). Detailed criteria for the rating system are presented in Table 8 below. Core photographs representative of the range of rock quality indicated by the drilling are presented in the Golder geotechnical characterization report (Golder 2012a).

Table 8: Rock Mass Rating System (Bieniawski 1976)

Rating	Description
0 – 20	Very poor rock
20 – 40	Poor rock
40 – 60	Fair rock
60 – 80	Good rock
80 – 100	Very good rock

The exploration boreholes were only logged for rock quality designation (RQD) data, while the geotechnical boreholes were logged for both RQD and RMR. Comparison between RQD and RMR data for the geotechnical boreholes indicated a good correlation between RQD and RMR. Since the rock is generally strong and fractures are unaltered, RMR is most strongly influenced by the degree of fracturing (i.e., RQD). Data from the 2010 geotechnical boreholes (IC-10-014, IC-10-015 and IC-10-016) were used to develop a correlation between RQD and RMR. This is discussed in more detail in the Golder geotechnical characterization report (Golder 2012a). This was then applied to the exploration boreholes to estimate RMR values from RQD.

Figure 13 shows a typical cross-section with both correlated and logged RMR data. A complete set of cross-sections is contained in the Golder geotechnical characterization report.

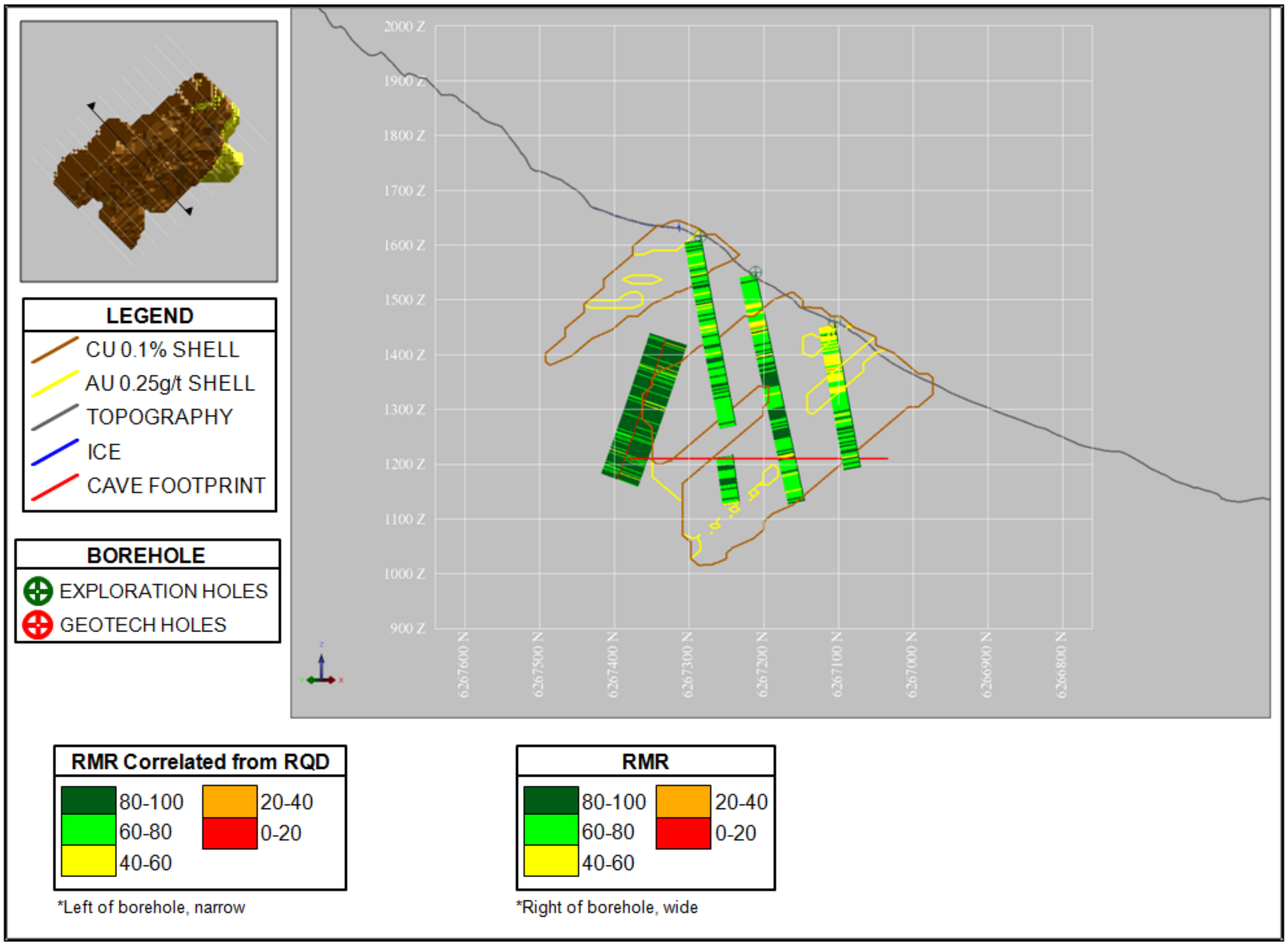


Figure 13: Vertical cross-section showing correlated RMR and logged RMR

The average RMR for the mineralized rock above the extraction level (El. 1,210 m) was determined to be approximately 70. This is in general agreement with the average RMR values reported for the STF footwall rock in BGC's report (BGC 2011). The rock conditions are classified as "good," according to the ratings shown in Table 8. RMR values are higher for geotechnical holes IC-10-015 and IC-10-016 (average RMR value of approximately 82), than for IC-10-014 (average RMR value of 64). This is discussed in more detail in the Golder geotechnical characterization report (Golder 2012a).

The host rock adjacent to the ventilation drifts to the south of the proposed block cave footprint appears to be of good quality, based on correlated RMR data from exploration borehole IC-10-044 and core photographs from exploration borehole IC-05-05. There are no geotechnical data available to assess the quality of the rock mass that the access ramp and conveyor drift will be excavated in to the northwest that connect to the Mitchell-Tiegan tunnel. However, geological interpretations suggest that the rock is good quality volcanic and sediments that are unaltered.

5.2 Intact Strength

A total of eight Unconfined Compressive Strength (UCS) tests were conducted as part of the Open Pit Slope Design PFS (BGC 2011). All eight samples were collected from the Iron Cap mineralized zone. UCS values ranged from 63 to 155 MPa (equivalent to ISRM field strength ratings of R4 to R5), with an average UCS of 102 MPa. Detailed test results are available in BGC's report (BGC 2011). These UCS values are in good agreement with UCS values obtained from laboratory testing performed on samples from the Mitchell project area, as described in the Golder report titled "2011 Geotechnical and Hydrogeological Field Investigations, Mitchell Project" (Golder 2012b).

Field intact rock strength estimates were logged by BGC for the 2010 geotechnical boreholes using the International Society for Rock Mechanics standard field identification methods (ISRM 1981). Logged ISRM strength estimates are generally consistent with laboratory test results (BGC 2011).

5.3 Fracture Orientations

Acoustic and optical televiewer survey data were reconciled with discontinuities logged in the geotechnical boreholes to develop stereographic projections. Detailed descriptions and stereographic projections of fracture orientations are available in BGC's report (BGC 2011).

Figure 14 shows a stereographic projection of combined structural orientation data from IC-10-014, IC-10-015 and IC-10-016. Data are referenced to true north. The plot indicates a prominent joint set steeply dipping to the south-southeast, and a less prominent joint set dipping at intermediate angles to the west. Note that these plots may displace some data bias due to the orientation of the boreholes. The bias may have resulted in fewer northeast dipping structures being identified.

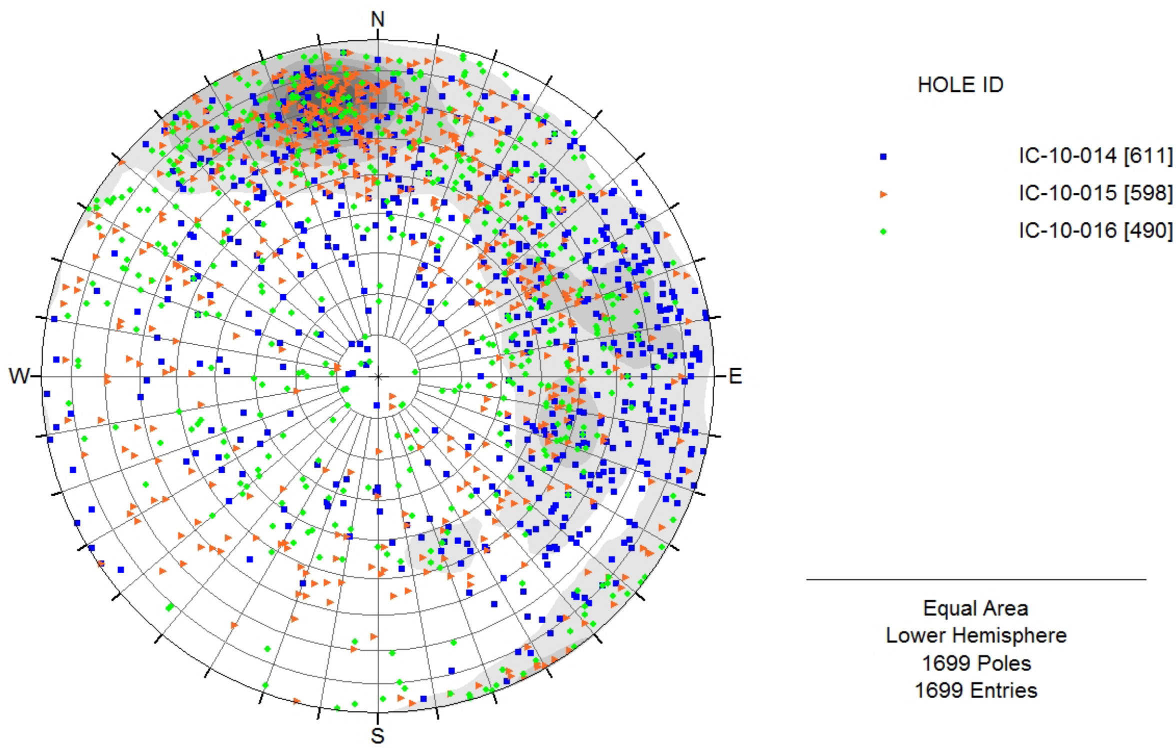


Figure 14: Stereographic projection plot showing open features classified by borehole.

5.4 Fracture Intensity

Fracture intensity is characterized by the fracture frequency logged per interval, defined as:

$$\text{Fracture Frequency } (/m) = \frac{\text{Number of Fractures in Interval}}{\text{Length of Interval}}$$

Only data from the geotechnical boreholes (IC-10-014, IC-10-015 and IC-10-016) were included in the fracture intensity characterization. Two of the holes (IC-10-015 and IC-10-016) have a relatively low fracture frequency, while the other geotechnical hole (IC-10-014) has a higher fracture frequency. Down-hole plots showing fracture frequency for the geotechnical boreholes are shown in Figure 15.

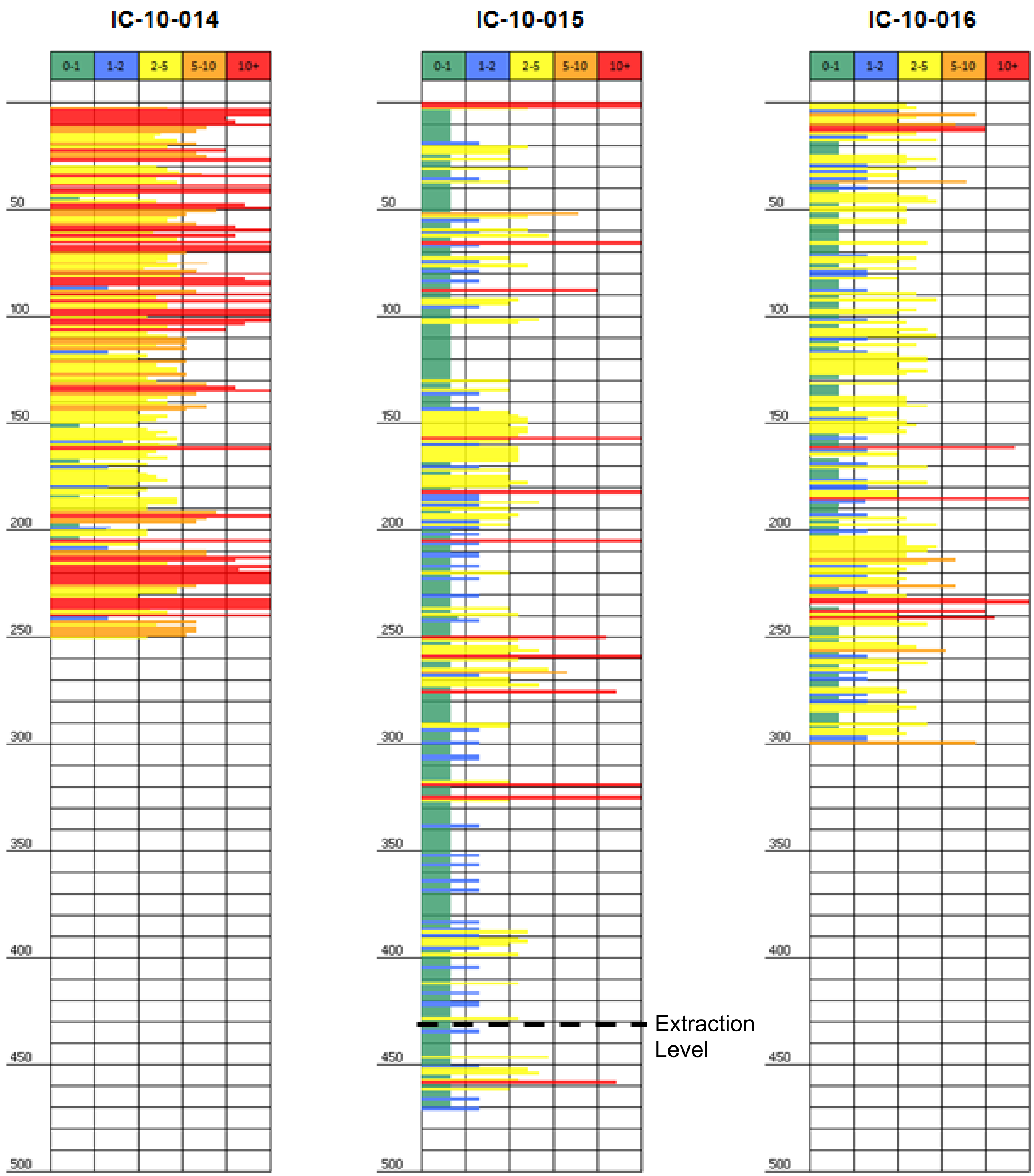


Figure 15: Down-hole plots showing fracture frequency data from geotechnical boreholes

In general, the exploration boreholes exhibit high fracture frequencies similar to IC-10-014. However, there is a low level of confidence in the fracture frequency data collected from these holes. Of particular significance is the fact that logging data from exploration holes at the Mitchell deposit suggested a much higher fracture frequency than the geotechnical holes indicated. The reason for this is uncertain, but the fracture frequency recorded for exploration holes may have been affected by drilling procedures, core handling, and logging methods. It is uncertain whether this is also the case at Iron Cap, but for the present the fracture frequencies from the exploration holes are not being relied on.

Correlations of fracture frequency with other geotechnical/geological parameters were evaluated in great detail. This included an assessment of the possible effect of alteration type on fracture frequency. Rock with pervasive silicification was found to be more fractured than other alteration types, but the difference in fracture frequency between the alteration types is less pronounced when data from IC-10-14 is excluded. At this stage, there is insufficient data available to assess whether the high fracture frequency in IC-10-014 is anomalous or representative of the rock mass above the proposed block cave mine footprint. Additional data will need to be collected as part of future studies to confirm the quality of the rock mass in the Iron Cap deposit.

The median fracture frequency for the mineralized rock above the extraction level (El. 1,210 m) in IC-10-015 and IC-10-16 is approximately 1.3 fractures per metre. The median fracture frequency for IC-10-014 alone is 4.7 fractures per metre.

5.5 Fracture Persistence

No fracture persistence data have been collected at Iron Cap. However, Golder conducted geotechnical mapping along four traverses on rock outcrops near the Mitchell deposit in June 2011. Detailed methodology, analyses and results are provided in the Mitchell field investigation report (Golder 2012b).

Two of the traverses had dominant phyllic (QSP) alteration, and two had dominant phyllic alteration with stockwork quartz veining (QSPSTW). Mapped features were characterized by the number of termination ends visible in the outcrop (i.e., 0, 1 or 2). Most features had a persistence of 3 m or less, as shown in Figure 16. However, the data are limited and strongly influenced by the size of the outcrops that were mapped (approximately 12 m by 2 m). It is recognized that there may be more continuous structures in the rock mass than indicated by the data, particularly intermediate or steeply dipping structures that would have been truncated by the mapping window. An allowance was made for this in developing the fracture model of the rock mass discussed in Section 5.6.1. The distribution of features for which either no terminations were visible (termination = 0), one end of the structure was visible (termination = 1), or both ends of the structure were visible in the mapping window (termination = 2) is summarized in Table 9.

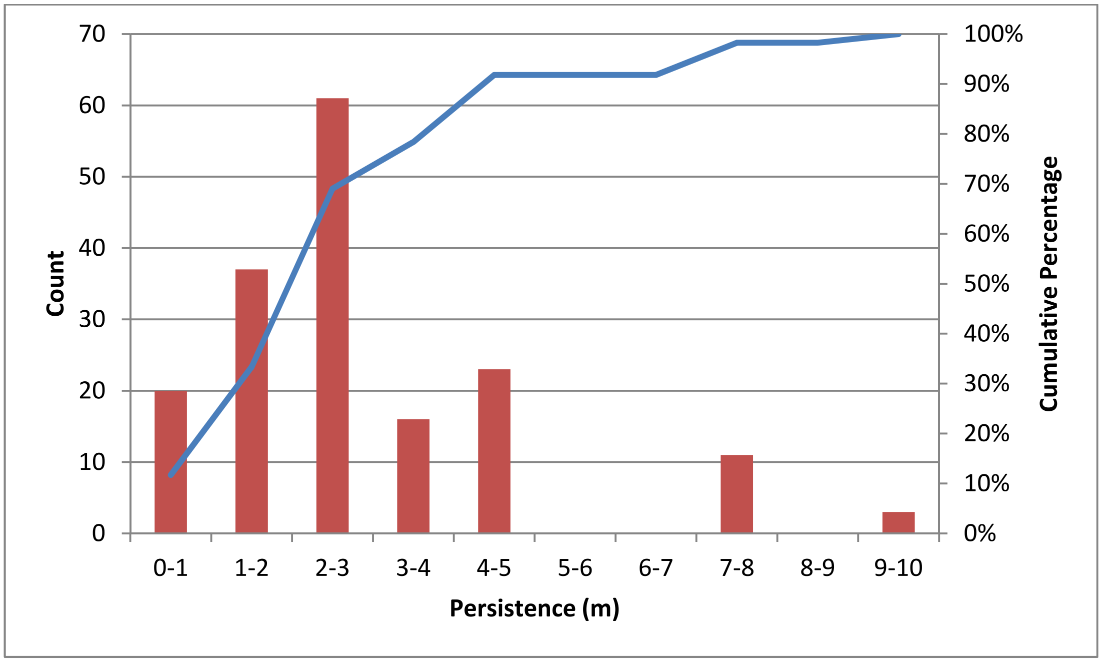


Figure 16: Persistence distribution of all mapped features

Table 9: Distribution of Termination of Mapped Features

Termination	Number of Mapped Features
0	12
1	30
2	26

5.6 Estimate of Insitu Block Size

An estimate of the range of in situ block sizes within the rock mass was developed based on the fracture characteristics discussed above and a Discrete Fracture Network (DFN) model created using the Golder FracMan software. DFN modelling is a methodology of creating a geologically realistic model of the fracture network based on stochastically defined structures. The models depict the geometry and connectivity of the fracture network and provide an indication of the geometry of the intact rock blocks.

5.6.1 DFN Model Input and Verification

The input data used to construct the model was as follows:

- distribution of fracture orientations obtained from borehole televiewer data from IC-10-014, IC-10-015, and IC-10-016;
- distribution of fracture spacing from boreholes within the Iron Cap deposit (IC-10-014, IC-10-015, and IC-10-016); and
- distribution of fracture lengths from fracture geometry information collected from outcrop mapping at the Mitchell project area during the 2011 field program (Golder 2012b).

Details on these input parameters can be found in the Golder geotechnical characterization report (Golder 2012a).

A 5x5x5 m DFN model constructed from the field data is shown in Figure 17. Fracture geometry within the model was found to be in good agreement with the field data on which it was based.

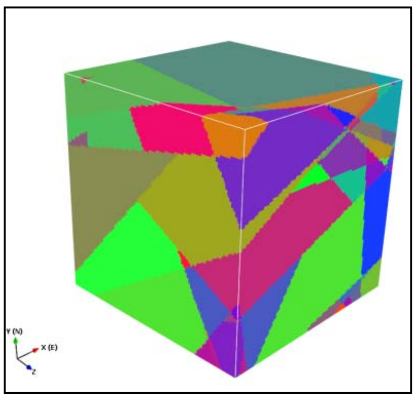


Figure 17: Example of Iron Cap 5x5x5 m DFN model

5.6.2 Results

The distribution of block sizes indicated by the DFN model is presented in Figure 18. The median block size is approximately 2.5 m^3. This represents a fairly coarse block size for caving mining. The implications of this are discussed in Section 6.2.

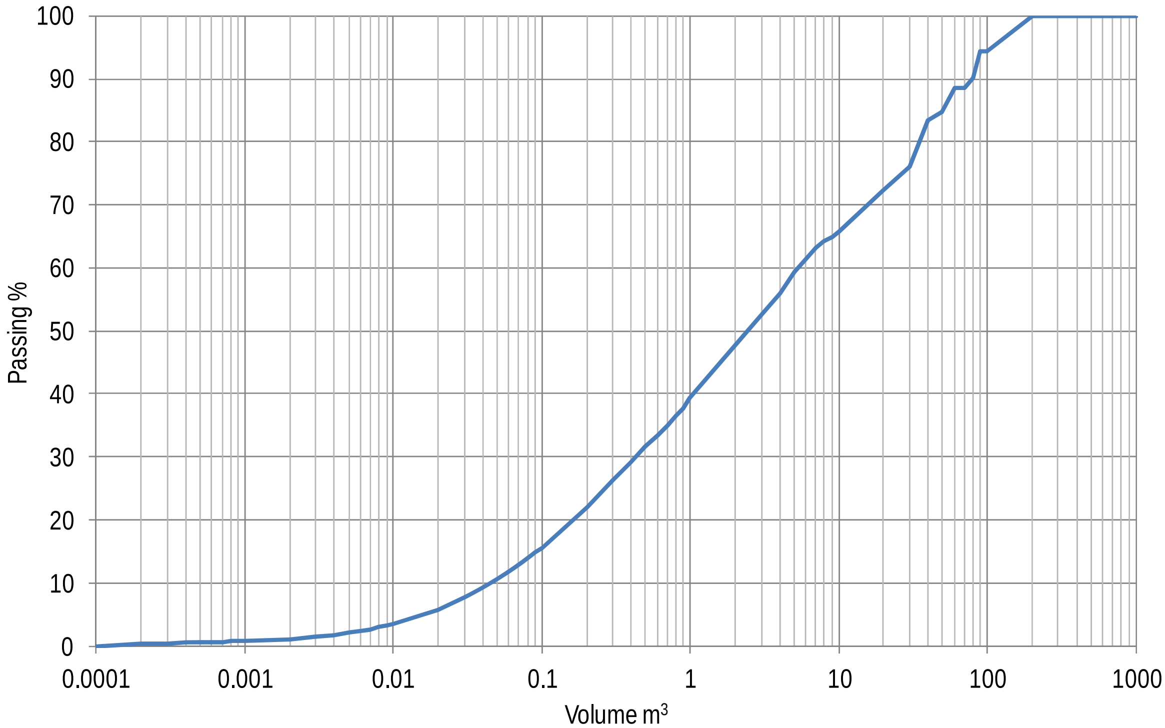


Figure 18: Block size percent passing averaged curve estimated for the Iron Cap deposit

5.7 In Situ Stress

No in situ stress testing has been conducted at Iron Cap. However, hydraulic fracturing testing was performed in borehole M-11-122 to evaluate the in situ stress at Mitchell. Detailed methodology, analyses and test results are provided in the Mitchell field investigation report (Golder 2012b).

The results of the testing at Mitchell suggest that the maximum horizontal stress may be as high as 2 to 4 times the vertical stress (estimated from overburden loading) and the minimum horizontal stress is estimated as 1 to 2 times the vertical overburden stress. In situ horizontal stresses are likely to be lower at the Iron Cap deposit due to the topography (the deposit is located in a hillside rather than in the valley bottom where stresses may be concentrated, as is the case for Mitchell).

5.8 Hydrogeological Characterization

Hydrogeological testing was conducted at Iron Cap during the 2010 field investigation (BGC 2011). Based on the test results, the hydraulic conductivity of the bedrock was found to decrease with depth from a maximum of 3 by 10^{-6} m/s to a minimum of 8 by 10^{-9} m/s, although values varied by up to two orders of magnitude at any given depth. Detailed hydrogeological test results are available in BGC's report (BGC 2011).

5.9 Discussion

The Iron Cap deposit appears to be composed of strong, moderately fractured rock. Rock quality variations are most commonly attributed to variations in fracture frequency as the strength of the rock mass does not vary significantly within the deposit.

The fracture frequency is higher for Iron Cap than the nearby Mitchell deposit, resulting in a corresponding lower predicted median in situ block size of 2.5 m^3 compared to approximately 6 m^3 for the Mitchell deposit.

There are several gaps in data that have been identified in the geotechnical and hydrogeological studies. These gaps will need to be addressed as part of future studies if the project is advanced to the next level of study. These gaps include the following:

- There are only three geotechnical holes in the Iron Cap deposit, and one of these three holes suggests significantly more fractured rock than the other two. With only three holes, there are insufficient data to confidently determine which of the existing holes most accurately represent the character of the rock mass as a whole. For the current level of study, properties from the three holes have been averaged. Future studies will need to include additional geotechnical drilling data to obtain a better spatial understanding of the fracture intensity in the deposit.
- Geotechnical logging at Iron Cap has focused on collecting parameters relevant to open pit design. Future logging should include collecting data tailored to assessing the caving geomechanics of the deposit (i.e., rock fabric, microdefects, etc.).
- To date there has been no geotechnical mapping directly relevant to the Iron Cap deposit. Assumptions regarding fracture persistence for this study have been based on limited mapping data in the area of the Mitchell deposit. Where possible, geotechnical mapping of relevant rock exposures in the area of the Iron Cap deposit should be carried out as part of future studies.
- There have been no fracture propagation assessments applicable to preconditioning designs or in situ stress interpretations developed at the Iron Cap deposit. Measurements carried out in the Mitchell deposit may not accurately reflect the fracture propagation and stress environment at Iron Cap because of the effects of surface topography. Future drilling programs should include hydraulic fracturing tests.

After this data collection is complete, more sophisticated numerical models should be developed to evaluate design aspects of the block cave (e.g., likelihood of stress-related fracturing, magnitude of abutment stresses, etc.).

6.0 CAVING GEOMECHANICS

6.1 Cavability

As indicated in Section 5.1, the quality of the rock mass at Iron Cap is rated as good. No major structural features have been identified that might influence the caving mechanism and the progression of the cave in any significant manner.

In situ stresses at the nearby Mitchell deposit have been estimated from hydraulic fracturing tests discussed in Section 5.7. The results of the testing suggest that the maximum horizontal stress may be as high as 2 to 4 times the vertical stress (estimated from overburden loading), and the minimum horizontal stress is estimated as 1 to 2 times the vertical overburden stress. In situ horizontal stresses are likely to be lower at the Iron Cap deposit due to the topography (the deposit is located in a hillside rather than in the valley bottom where stresses may be concentrated, as is the case for Mitchell). Given the strength of the rock mass (~100 MPa) and the relatively shallow depth of the deposit, stress-induced fracturing of the rock mass is unlikely to significantly contribute to caving

A preliminary assessment of the cavability of the rock mass was made using Laubscher's Stability Chart (Laubscher 1999) and the Extended Mathews' Stability Graph (Trueman and Mawdesley 2003). Both methods involve assessing cavability based on experience at other mining operations with rock of similar quality. Both assessments were based on average or "typical" geotechnical properties for the rock above the extraction level (El. 1,210 m).

Laubscher Stability Chart

The Laubscher Stability Chart relates the rock quality and stress conditions for a given deposit, characterized by the Modified Rock Mass Rating (MRMR), to the hydraulic radius of the opening. MRMR was estimated to be approximately 48 for the Iron Cap deposit. Parameters used to estimate MRMR are outlined in Table 10.

Table 10: MRMR Rating Classification

Parameter	Description	Rating
Intact rock strength	102 MPa	10
RQD	93%	14
FF/m	2 joint sets, average spacing 1.6 m	22
Joint condition		21
Large scale	Moist, straight	70%
Small scale	Moist, rough undulating	75%
Joint wall alteration	No alteration	100%
Joint filling	None	100%
RMR		**67**
Adjustments		
Weathering	None	100%
Joint orientation	3 joints, 2 inclined	80%
Mining induced stresses	Stress difference in cave back	90%
Blast effects	None	100%
MRMR		**48**

As shown in Figure 19, the minimum hydraulic radius (HR) of the undercut required to initiate caving based on Laubscher's Stability chart is approximately 25 m. This equates to an approximate area of 100 m by 100 m.

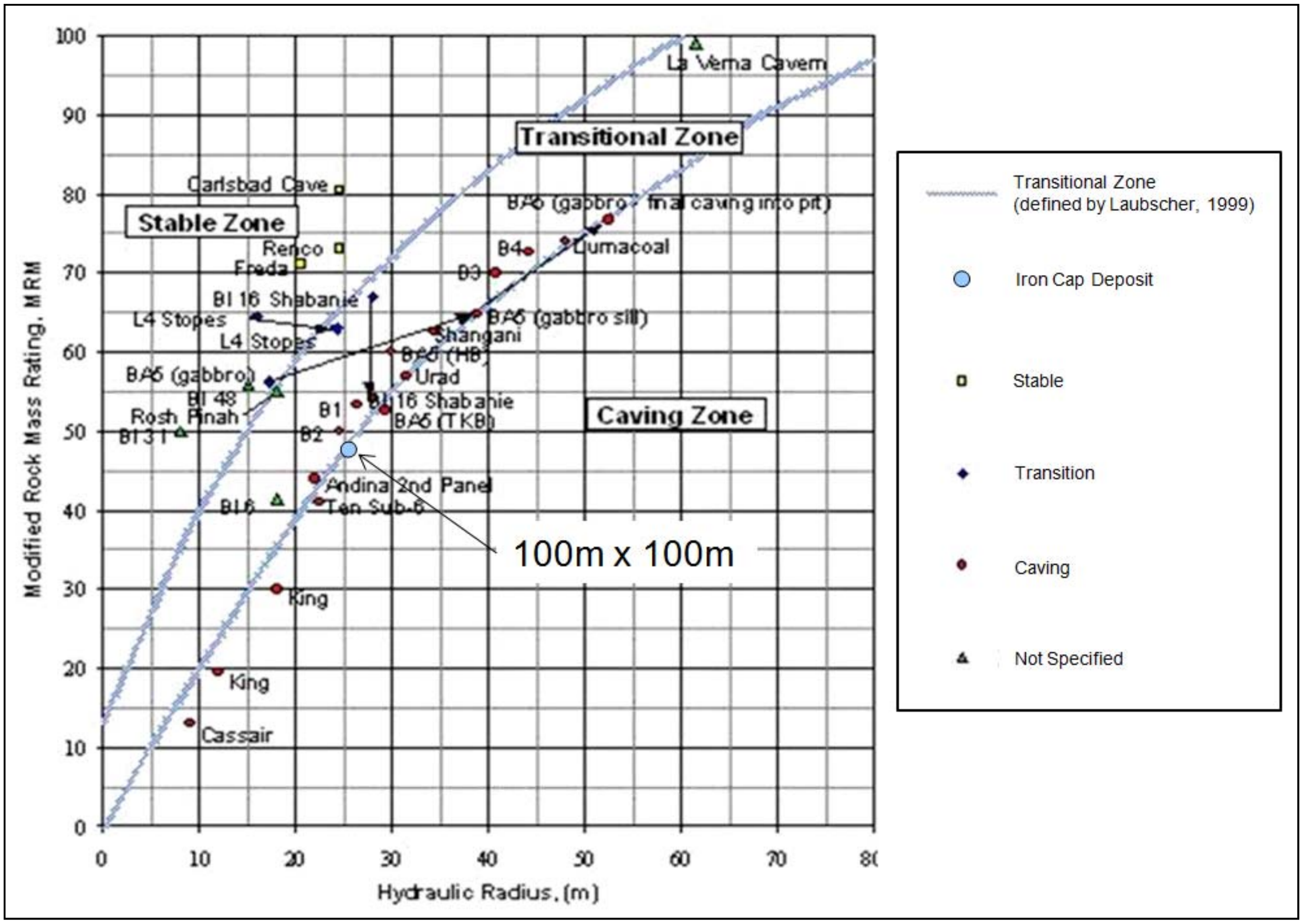


Figure 19: Cavability assessment using Laubscher's method (Laubscher 1999)

Extended Mathews Stability Graph

The Mathews method of assessing cavability uses the stability number (N) to characterize the rock quality and stress conditions of the deposit. The estimated stability number (N) for Iron Cap is 1.6. A summary of the parameters used to estimate N is contained in Table 11.

Table 11: Q' and N Rating Classification

Parameter	Description	Rating
Q'	(RQD/Jn) × (Jr/Ja)	18
Factor A [1]	$\sigma_c / \sigma_1 \approx 1$	0.1
Factor B [2]	Dominant joint set dipping at approximately 75 degrees	0.9
Factor C	Horizontal cave back	1
N	Q' x A x B x C	**1.6**

Notes:

[1] Average intact rock strength (σ_c) estimated from UCS testing of Iron Cap rock core samples. Average maximum induced compressive stress (σ_1) estimated from numerical modelling.

[2] Joint orientation estimated from stereographic projections produced from Iron Cap televiewer data.

As shown in Figure 20, the minimum hydraulic radius (HR) of the undercut required to initiate caving based on the Extended Mathews analysis is approximately 55 m. This equates to an approximate area of 220 m by 220 m. This is somewhat larger than the area indicated by the Laubscher method, which is indicative to some degree of limited experience in caving good quality rock of this nature.

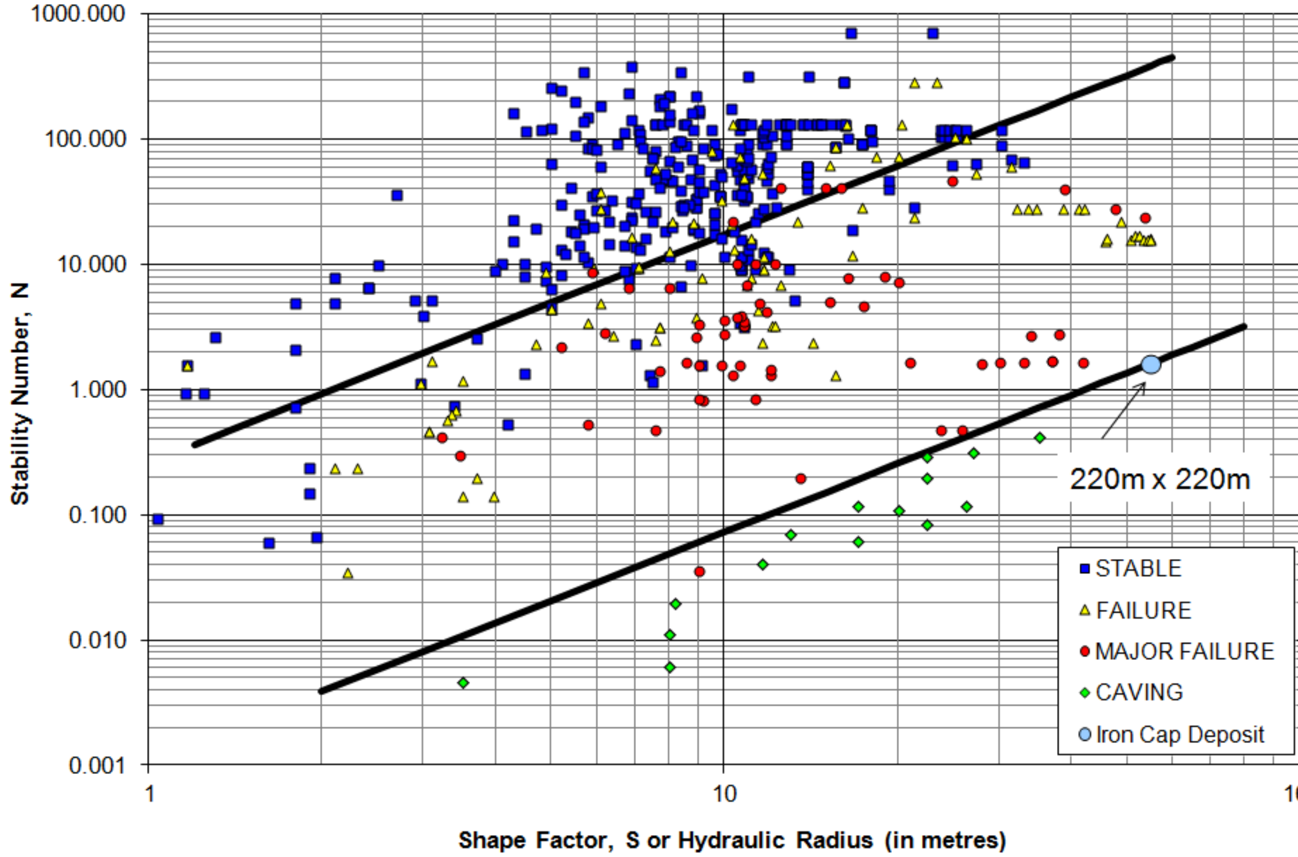


Figure 20: Cavability assessment using Mathews' method (Trueman and Mawdesley 2003)

The cavability assessments made using Laubscher's and Mathews' methods indicate that the size (diameter) of the footprint required to initiate and propagate caving is between approximately 100 m and 220 m. This footprint size is significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized, continuous nature of the deposit, suggest that the Iron Cap deposit is amenable to cave mining.

6.2 Fragmentation

The fragmentation of the rock mass as it caves and is drawn down to the drawpoints is a fundamental aspect of the design of a block cave mine. The resulting fragmentation size affects the diameter of the drawcone (Isolated Draw Zone, IDZ) that develops above a drawpoint as material is drawn down. Coarse fragmentation results in large diameter drawcones, while fine material results in narrow, slender drawcones (Figure 21). Interaction and overlapping of neighbouring drawcones is required to ensure efficient ore extraction.

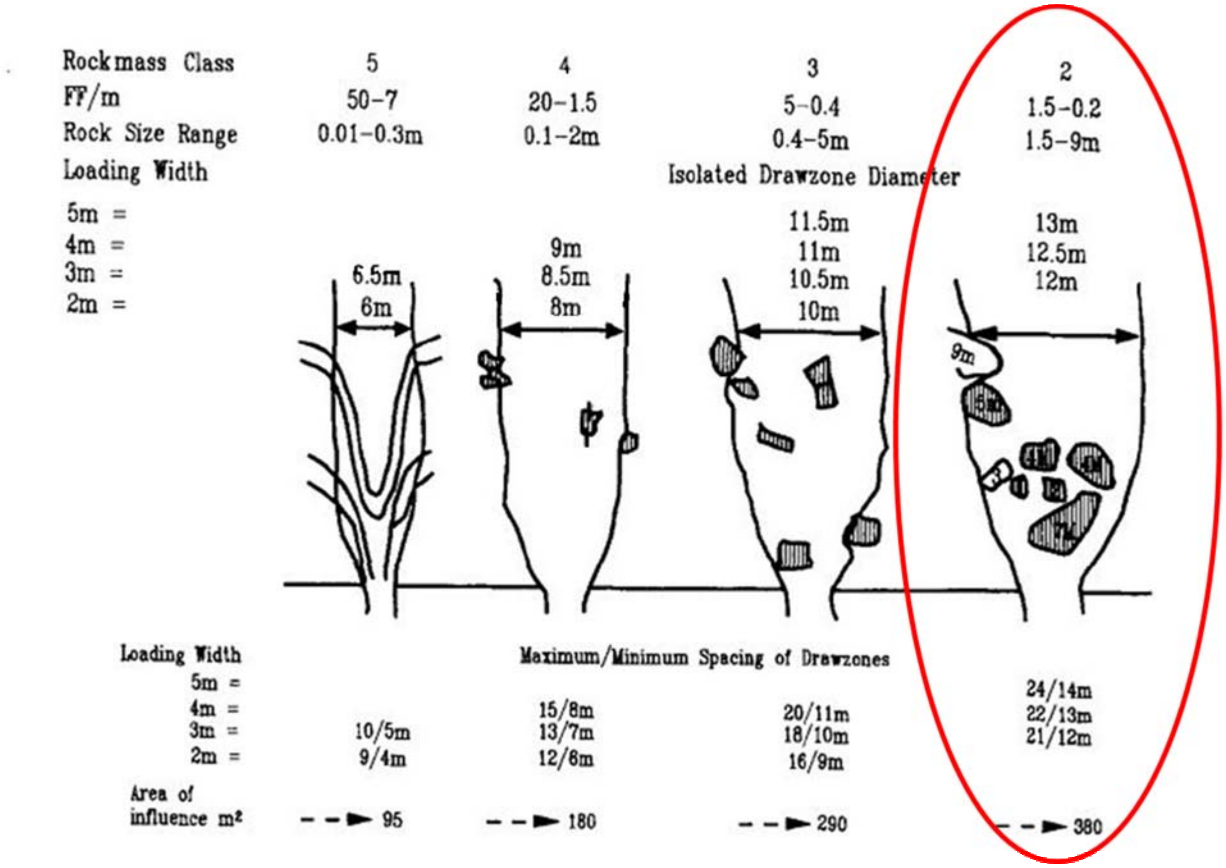


Figure 21: Maximum/minimum spacing of drawzones based on isolated drawzone diameter (Laubscher 1994)

Drawpoint spacing is typically governed by the size of a drawcone. Large diameter drawcones allow the spacing between the drawpoints to be increased, thereby reducing the number of drawpoints and the capital cost of developing the draw level. Achieving a larger spacing between drawpoints also reduces the time required to develop a given footprint area, resulting in an increased production rate. However, large sized blocks reporting to the drawpoints also increase the potential for drawpoint blockages, requiring secondary rock breaking at the drawpoints. This can inhibit production significantly and increase mine operating costs.

The first step in assessing the fragmentation of the rock reporting to the drawpoints is to estimate the in situ size of the blocks formed by the intersection of discontinuities in the rock mass. There will be further attrition of these blocks as the rock is drawn toward the drawpoints. However, it is very difficult to estimate the attrition as a result of secondary breakage, and under the prevailing conditions, fragmentation estimates are typically based on an initial assessment of the pre-caving in situ block size.

An estimate of the range of in situ block sizes for the Iron Cap deposit was developed based on a Discrete Fracture Network (DFN) model created using the Golder FracMan software (discussed in Section 5.6).

The distribution of block sizes indicated by the DFN model was presented earlier in Figure 18. The median block size is approximately 2.5 m^3. This represents a coarse block size for cave mining. A comparison between the Iron Cap deposit and estimates of block sizes at some other block caving mines is shown in Figure 22 (Butcher and Thin 2007). A number of the mines with comparably large block sizes experienced difficulties as a result of excessive secondary blasting requirements, and this adversely impacted the productivity at these mines to varying degrees.

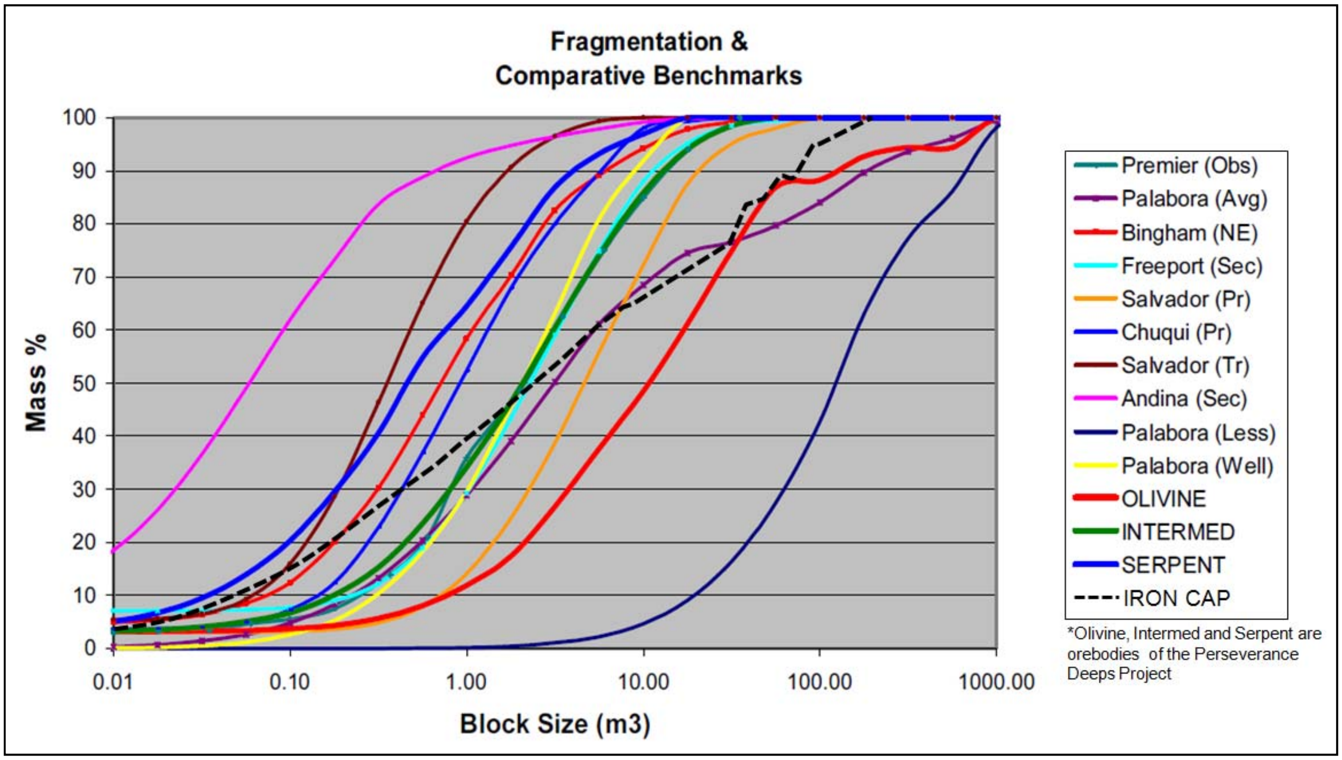


Figure 22: Comparison between the estimated block size at the Iron Cap deposit and existing block caving operations (Butcher and Thin 2007)

The factors that reduce the block size reporting to the drawpoints (from the in situ block size estimate) include the following:

- the degree to which the rock is further fractured and disturbed by the induced stresses in the back of the cave;
- the breakage of the rock as it displaces from the back of the cave; and
- the attrition that occurs as the rock is drawn toward the drawpoints.

It is very difficult to quantify the effect of attrition as the rock is drawn down through the cave zone. Experience has indicated that, in caving mines operating under similar rock conditions to those at Iron Cap, fragmentation of rock drawn down more than approximately 100 m is generally good. For this study, it was assumed that fragmentation of the initial 100 m of draw height is approximately equal to the estimated in situ block size and above this only limited secondary blasting will be required.

The common definition of oversize where secondary blasting is required is 2 m^3. As shown in Figure 18, a significant proportion of the rock has block sizes greater than this. Without some remediation measure being adopted, such large sized blocks will require significant secondary blasting, and there will likely be a significant adverse impact on production and significant damage to the drawpoints that will require ongoing rehabilitation.

As a result of this, it is proposed that the rock be preconditioned by hydrofracturing. The cost and schedule to do this have been incorporated into this study. However, there are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. It is also difficult to obtain definitive field data that demonstrates the degree of improvement obtained. The results from these mines are encouraging, however, and there is sufficient experience to indicate that such fragmentation concerns do not represent a fatal flaw at Iron Cap. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning in enhancing fragmentation needs to be addressed via production and cost risks (as discussed in Sections 11.0 and 12.0).

6.3 Drawpoint Geometry

Fragmentation of the rock is expected to be coarse, even with preconditioning being used. As indicated in Figure 21, this will result in relatively large isolated drawcone diameters of 13 m or more for a loading width of 5 m. An important objective is to maintain full interaction between individual neighbouring draw columns. The present experience in other operating cave mines is that a 15 m by 15 m drawpoint layout performs well under these coarse fragmentation conditions. Some caving mines operating in good quality rock have successfully expanded the layout to approximately 17 m by 17 m. The available information on fracture intensity at Iron Cap is somewhat uncertain due to the limited geotechnical drilling that has been undertaken, so it was considered prudent for this initial study to adopt the slightly more conservative 15 m by 15 m spacing. This aspect needs to be investigated further, and there may be an opportunity in the future to adopt an expanded layout.

6.4 Subsidence

The caving mining will draw down the mineralized rock, and a significant surface depression will develop on surface above the production footprint in the form of a crater. The top section of the crater will form a relatively steep escarpment (inclined at approximately 60 to 70 degrees) that is marginally stable but comprised of nominally in place dilated rock. Beneath this, surface and sidewall material that has progressively sloughed from the rim of the crater will rill down to the bottom of the crater at about 40 degrees.

The disturbance from caving mining will extend beyond the rim of the crater at any time in the form of wide and relatively deep open cracks. These cracks will be less prominent and more widely spaced with distance away from the rim. The crater and disturbance zone will progressively expand in size as more and more of the deposit is extracted.

Under some circumstances, the progressive development of the crater will destabilize adjacent natural slopes as it expands outwards. As discussed in more detail later, this is not thought likely to occur at Iron Cap.

For this pre-feasibility assessment, an empirical approach has been used to estimate the profile of the crater and the extent of the cracking and disturbance zone beyond the rim. These assessments have been based primarily on the geometry of the profiles at some of the caving mines in Chile in rock of similar quality and with similar structural conditions to those in and around the Mitchell deposit. Based on this, estimates of the geometry of the crater rim and the outer disturbance zone are presented on Figure 23. These profiles represent the conditions that exist at the completion of caving mining.

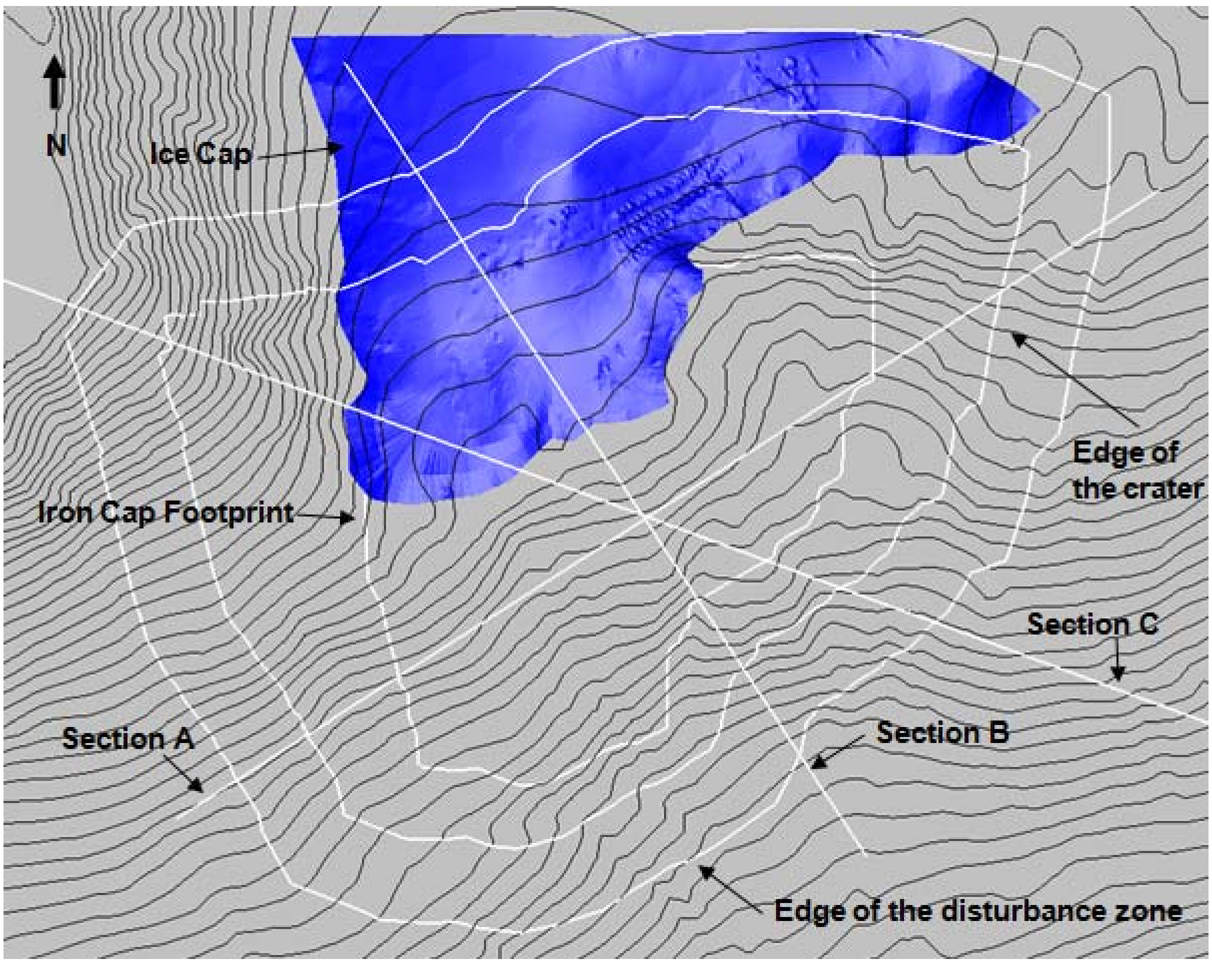


Figure 23: Topography map on top of the Iron Cap deposit showing the ice cap and the steep escarpment to the east and three section lines.

The profiles shown in Figure 24 were developed from estimates of caving angles of approximately 70 degrees to the horizontal up from the footprint of the caved rock, with the cracking or disturbance zone extending approximately 100 m laterally beyond the rim of the crater. This disturbance zone represents the outer limit of visible open cracks on the ground surface. Small but measurable subsidence may extend beyond this again, but no estimates have been made of the extent and nature of these comparatively small vertical displacements. The disturbance profiles for the conditions shown in Figure 23 are presented as cross-sections in Figure 24. These represent the conditions at the completion of mining.

Figure 24-Section A shows the surface topography beyond the disturbance zone in the east-west direction and Figure 24-Section B shows the surface topography in the north–south direction. As indicated earlier, there is a potential for the up-gradient slopes beyond the disturbance zone to be de-stabilized by the progressive development of the crater. However the slopes shown in these two figures are relatively flat, and given this, plus the absence of major structural features that might generate large scale sliding failures, failures of the natural slopes are very unlikely.

Figure 24-Section C shows the surface topography beyond the disturbance zone in the north-west south-east direction. This represents the conditions at the completion of mining. In this case, the surface profile slopes away from the crater on the north-west side, and again the crater will not likely generate any slope failures. In this case however, the crater will under-cut a steep escarpment that is aligned north-east south-west as it expands towards the north-west. This may generate slope failures within the escarpment at these intermediate stages of mining. This low grade broken rock will fall into the crater and mix with the other rock that has sloughed from the edge of the crater. The combined low grade rock will completely or partially cover the mineralized rock as it is being drawn down, and some of this will report to the drawpoints as dilution. This dilution has been accounted for in developing the production and grade schedules for the Iron Cap caving operation.

For this pre-feasibility study, the development of the crater and the stability of the adjacent natural slopes have been empirically assessed as separate responses, although in actual fact they occur as an integrated and composite response to the draw of the mineralized rock during production. Further assessments based on numerical analysis studies of the combined response of the development of the crater and potential instabilities of the natural slopes may be required as part of higher level assessments of the impact of the proposed mining on the ground surface if more accurate estimates of the resulting surface profiles are deemed to be necessary.

As shown in Figure 23, the ice field up slope from the Iron Cap deposit encroaches on the caving footprint at the production level, and will be undercut at the north and north-west extension of the crater. The profile of the ice in a north-south direction is shown in Figure 24-Section B. The ice that is undercut will cave into the crater and it will either melt or some of it will be mucked as ice. The volume of ice that is undercut is estimated to be approximately 3.4 million cubic metres. An allowance for this volume of water potentially melting and flowing down into the workings has been included in the water management system discussed later in this report.

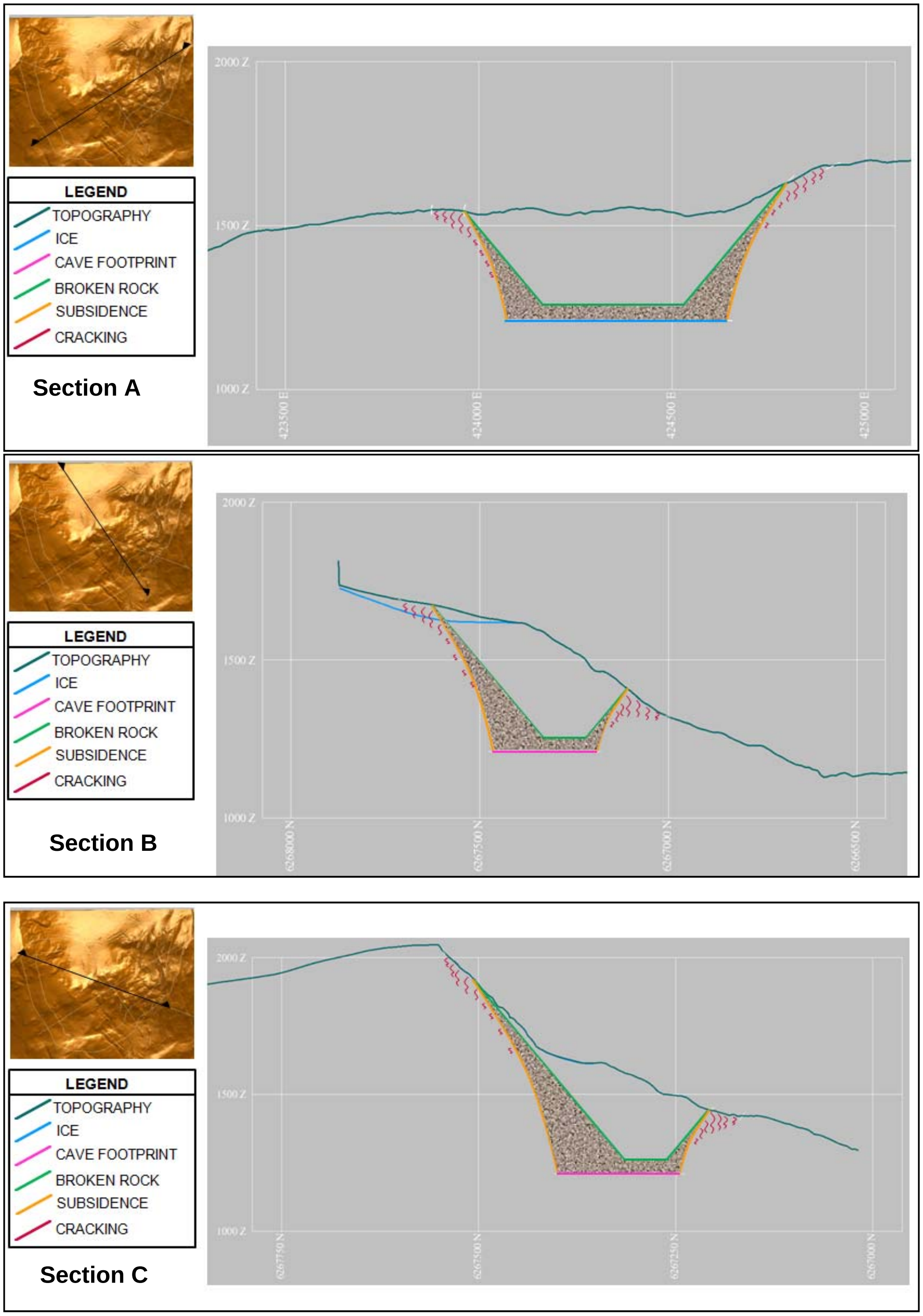


Figure 24: Three sections (A, B and C) of the Iron Cap cave zone showing to possible extent of the cave induced fracturing and subsidence

7.0 MINE DESIGN

The Iron Cap block cave design was based on modelling from FF and PCBC software (FF was discussed in Section 4.0 and PCBC will be discussed in Section 10.0). FF modelling indicated that the optimum footprint for the Iron Cap deposit is at an elevation of 1,210 m. It is approximately 545 m wide in the north-south direction, 570 m wide in the east-west direction, and has an average depth of 400 m. PCBC modelling indicated that the block cave could produce 15 million tonnes per year, requiring development of 120 new drawpoints per year. The mine design requires approximately 64 km of drifts and raises, including a 5% contingency to account for the excavations of detailed design items such as service bays, sumps and electrical substations.

The mine design was primarily influenced by rock fragmentation and material handling. The drawpoints and extraction drifts are spaced to accommodate the estimated large fragmentation of mineralized material, and the crusher location and drawpoint orientation were chosen to ensure an efficient Load-Haul-Dump (LHD) haulage route.

Four main levels are required to cave the Iron Cap deposit and include the preconditioning level, undercut level, extraction level, and conveying level. The design also includes a return air drift located below the conveying level. The location of these levels is shown in Figure 25. Appendix A contains the detailed drawings for all levels.

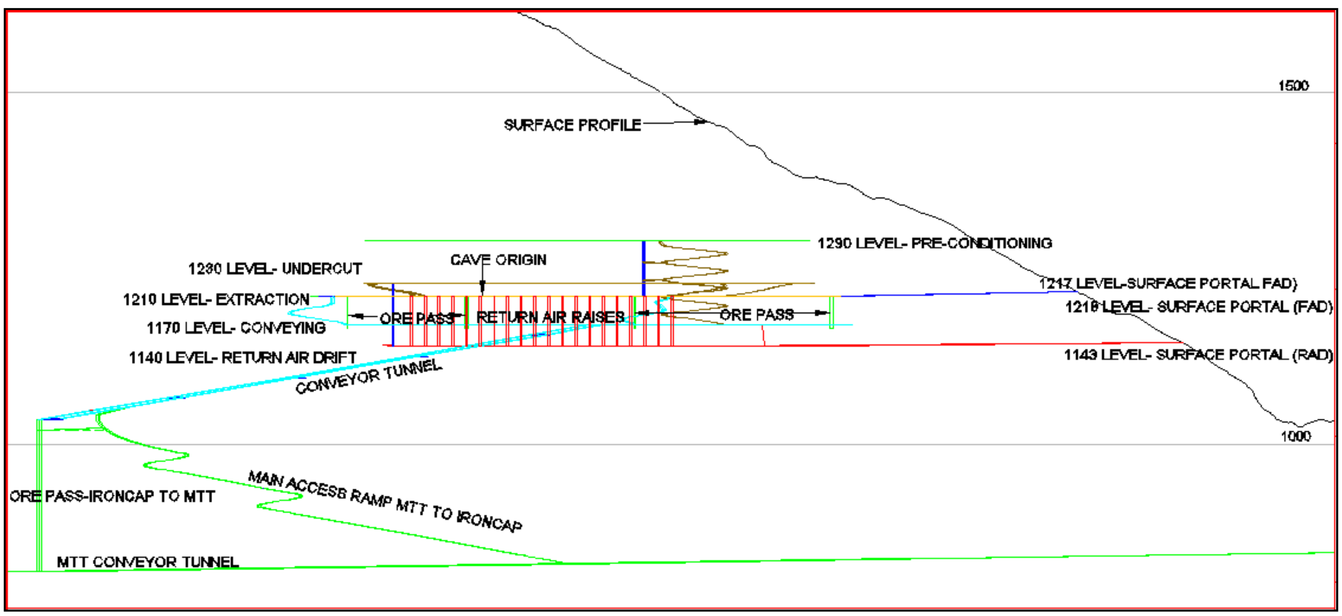


Figure 25: Section view (looking east) of the Iron Cap mine design

7.1 Underground Access

Personnel, material, and supplies will access the underground through a drift driven from the Mitchell Tiegen Tunnel (MTT). The access ramp from MTT will be driven at 15% to elevation 1,170 m, after which it will be driven at 10% to reach the Iron Cap footprint. A conveyor drift (the conveyor drift will be discussed in section 7.5) will be developed parallel to the access ramp, and the two will be connected every 300 m to provide emergency egress and a ventilation loop during construction.

The total length of the access ramp is 3.4 km. It is designed to be 5.5 m by 5.5 m wide to allow all of the underground equipment, including the crusher parts, to be transported underground (underground equipment is discussed in Section 7.8).

Two fresh air portals and one exhaust portal are planned on the north slope of the Mitchell valley. These tunnels may act as an alternative access to the underground from the surface in case of emergency. The fresh air tunnels will connect to surface at a grade of 2% with an average length of 420 m. The return air portal will be located approximately 150 m downstream of the fresh air portals to prevent air contamination. A perimeter drift is constructed around the entire mine footprint to provide fresh air to the mine workings. Additional information concerning the ventilation design can be found in Section 8.1.

Figure 26 shows a plan view of the proposed mine layout and major infrastructure including access to the MTT. The estimated length of access ramp and conveyor tunnels is shown in Table 12.

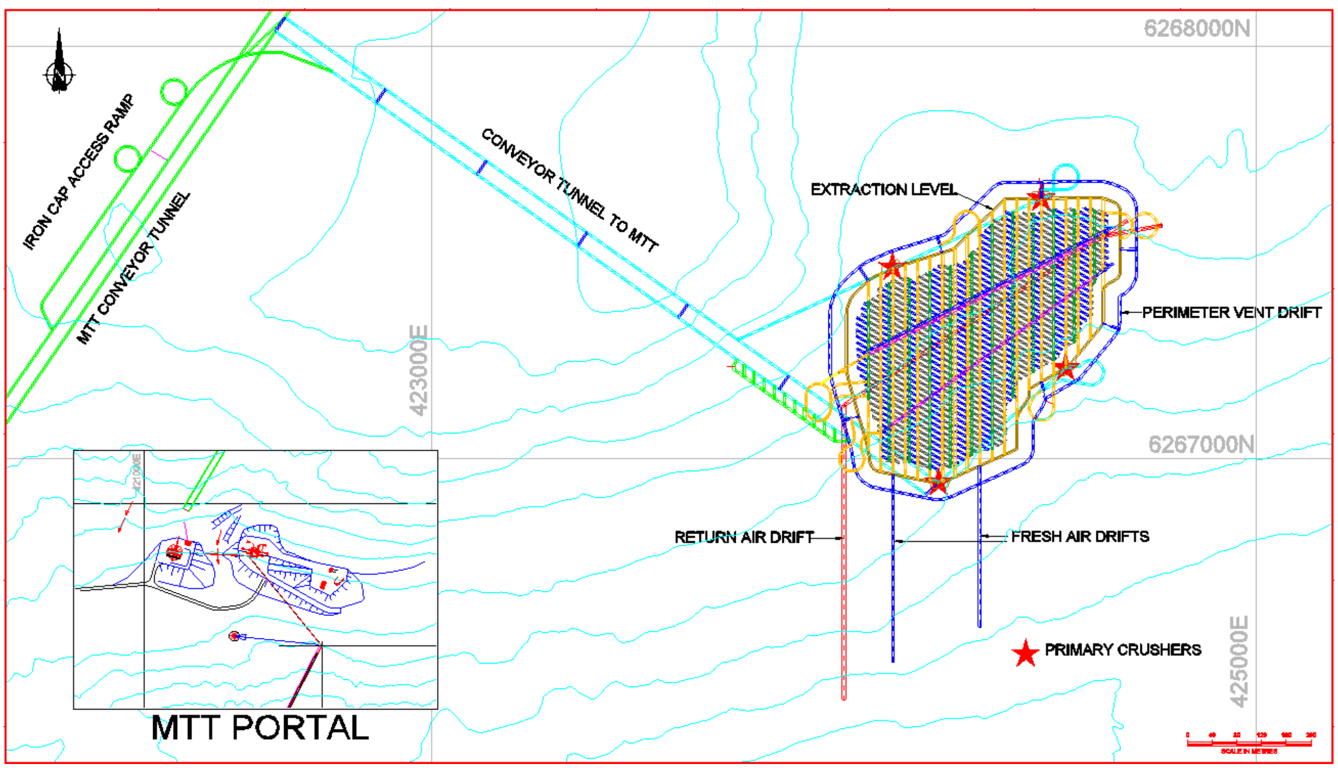


Figure 26: Plan view of the proposed Iron Cap underground mine layout

Table 12: Design Lengths of the Conveyor and Access Ramps

Item	Length (m)
Access Ramp from MTT	3,400
Conveyor Tunnels	3,300

7.1.1 Emergency Egress

The Iron Cap design has multiple drifts that can act as an emergency egress. The primary emergency egress will be the conveyor tunnel or access tunnel, whichever is accessible. If both tunnels are inaccessible, it will be possible to exit the mine through one of the fresh air drifts; however, the fan at the portal will have to be shut off because, during normal operations, the wind speed in the tunnel will be too high for personnel entry.

7.2 Preconditioning Level

A preconditioning (PC) level is planned to provide access for in situ fracturing of the rockmass prior to caving. A plan view of this level is shown in Figure 27. From this level, a series of holes will be drilled and hydrofracturing will be used to generate cracks within the future cave zone.

The PC level design is based on one PC hole having a 25 m radius of influence. The drilling pattern consists of two 25 m deep, 64 mm diameter holes, drilled on 50 m centres. Both holes will be vertical, one through the back (uphole) and the other through the floor of the drift (downhole). Hydrofracturing of the rock will occur at 1 m intervals down each hole.

The PC drifts are spaced 50 m apart east-west to cover the entire cave footprint area. Each of the PC drifts is connected to the main drift at two points that will provide access and ventilation. The level is designed to be 4 m wide by 4 m high to accommodate the drilling equipment necessary for the PC holes. It is located 60 m above the extraction level and is accessed via a ramp that connects to the perimeter drift and undercut level. A total of 7,200 m of PC drifts will be needed for the Iron Cap mine.

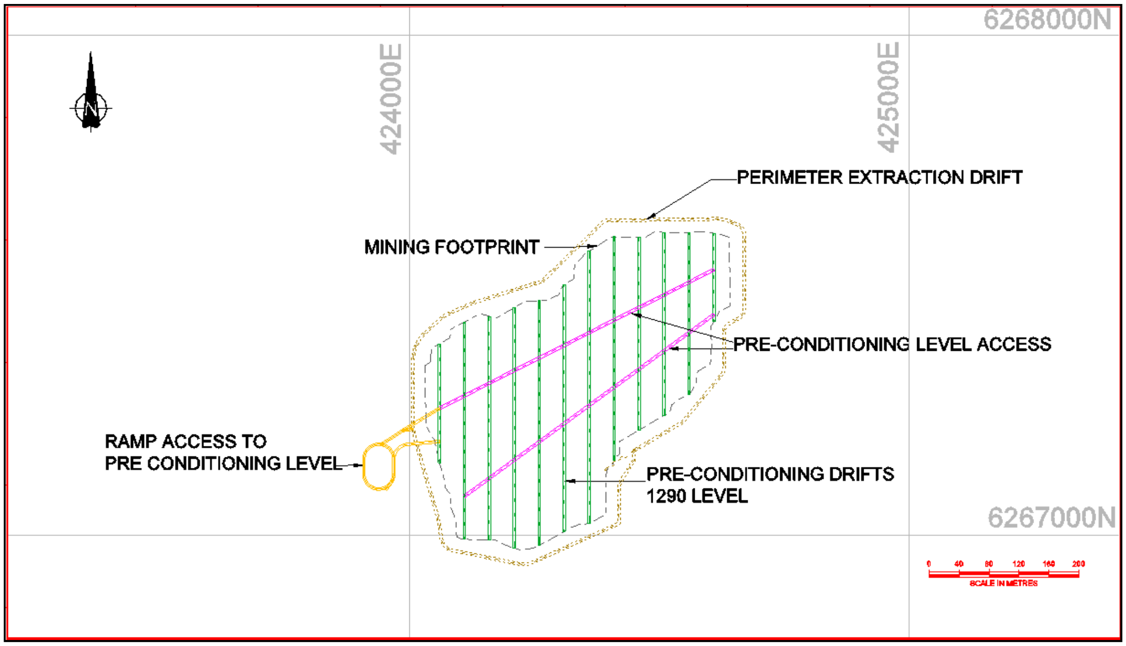


Figure 27: Plan view of the proposed preconditioning level

7.3 Undercut Level

Blasting from the undercut (UC) level initiates and propagates the cave. A plan view of this level is shown in Figure 28. Undercutting will be done using the drilling patterns shown in Figure 29, which consist of rings spaced 2 m apart, each containing twenty-one 64 mm diameter holes and approximately 140 m of drilling. Experience at other block caving operations, with rock quality similar to what is expected in the Iron Cap deposit, suggests that this drilling pattern of alternating parallel and angled holes creates sufficient void to start the caving process.

The proposed drilling pattern requires that the UC drifts are parallel to the extraction drifts. The UC drifts are 20 m above the extraction level and 15 m apart. Two crosscuts, 160 m apart, will provide access to the UC drifts. To accommodate the drilling equipment necessary, the UC is designed to be 4 m wide and 4 m high. A total of approximately 9.4 km of UC drifts will be required for the Iron Cap mine including the access ramps.

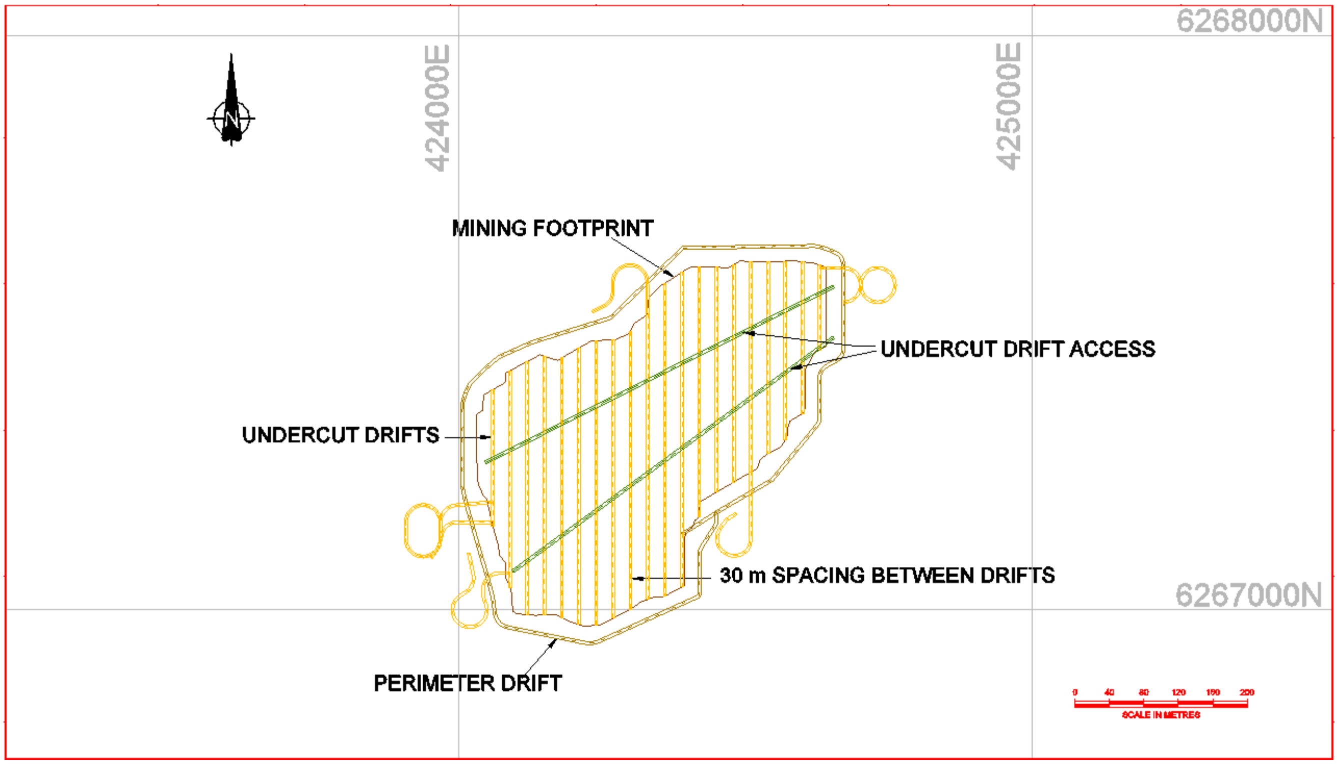


Figure 28: Plan view of the undercut level

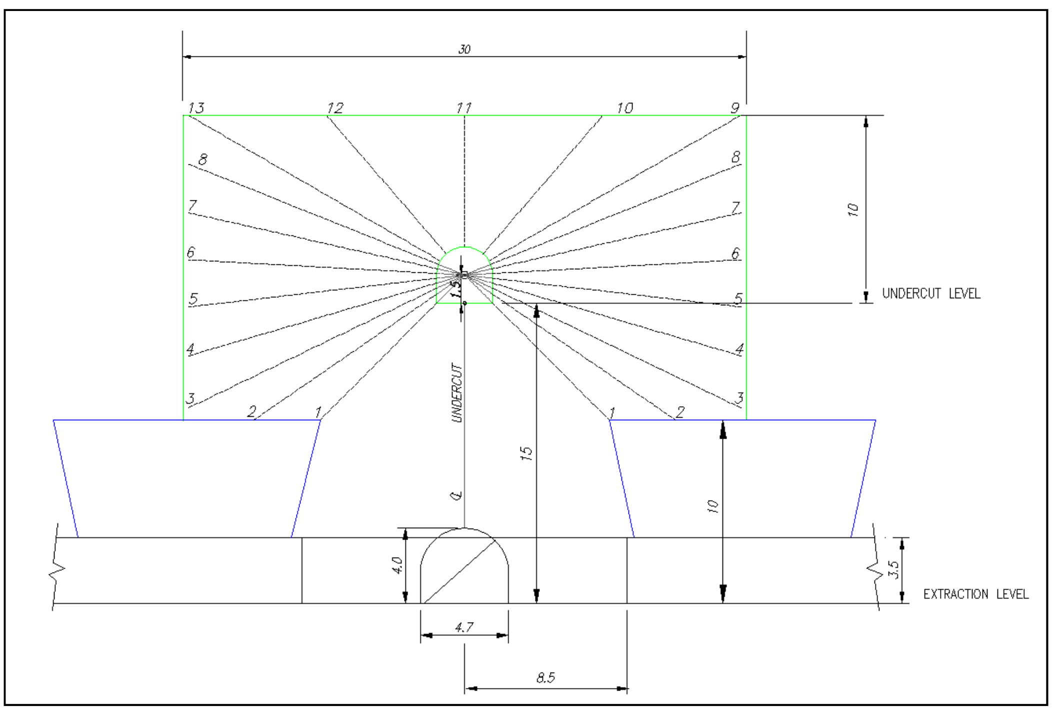


Figure 29: Schematic cross section showing the relationship between the undercut and the extraction levels, and the drill pattern used to develop the cave undercut

7.4 Extraction Level

The dimensions of the extraction level were designed to accommodate the estimated fragmentation from the Iron Cap cave and to be as productive as possible. To allow for the appropriate overlap between drawcones (a complete definition of block caving terms can be found in Appendix B), the extraction drifts (positioned in the north-south direction) are spaced 30 m apart and the crosscuts (positioned in a northwest-southeast direction) are spaced 15 m apart as shown in Figure 30. The spacing is designed from drift centreline to centreline and creates a 15 m by 15 m drawpoint layout. The extraction level drifts have a typical cross-section of 5 m by 5 m, and the drawpoints have a typical section of 4.5 m wide by 3.5 m high. The extraction drifts, undercut and conveyor tunnels will be accessed through internal ramps strategically positioned for access and ventilation purposes.

The drawpoints are 60 degrees from the axis of the extraction drift and are offset 15 m from each other. This design is based on the El Teniente mine in Chile (a large and mature block cave operation). The access angle allows for efficient entrance and exit by the underground LHD machines and the offset also reduces the impact of a mudrush. Figure 30 shows a diagram of the relationship between the extraction drifts, drawpoints and drawbells. In addition, the floors of the extraction drift and drawpoints are designed to be concreted, which will increase the speed and productivity of the LHDs as well as reduce equipment maintenance.

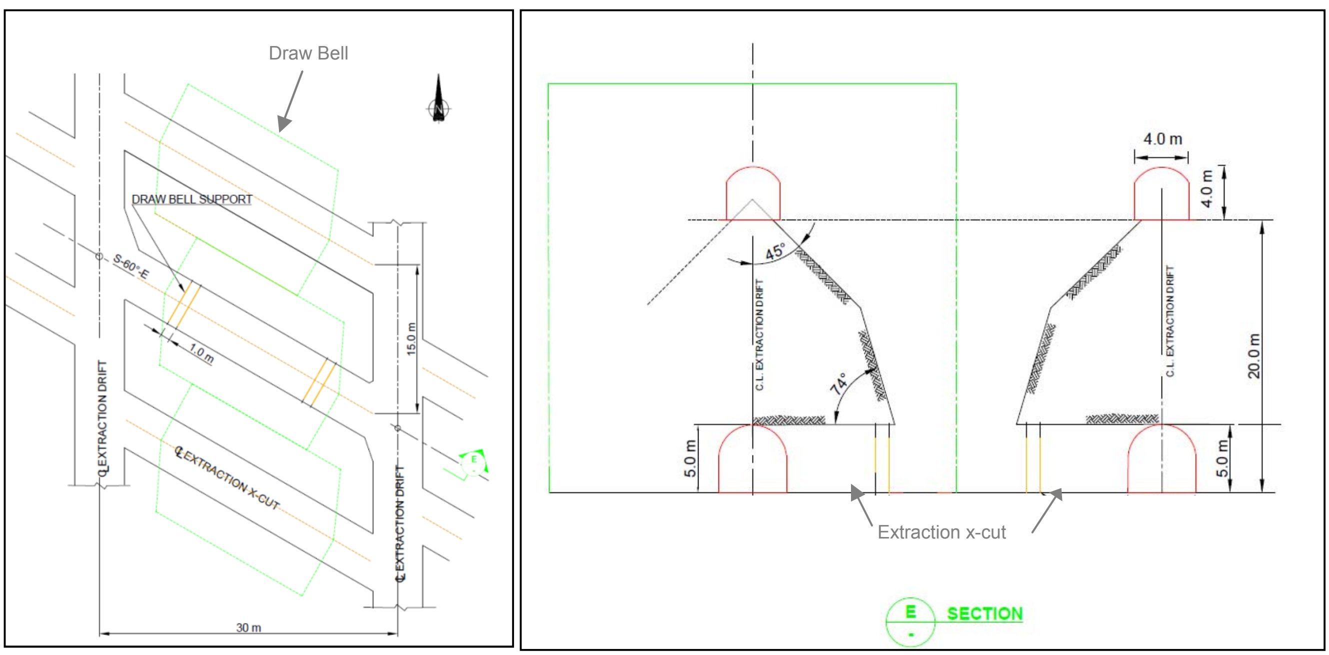


Figure 30: Diagram showing the relationship between the drawbells, drawpoints (extraction x-cut) and extraction drifts

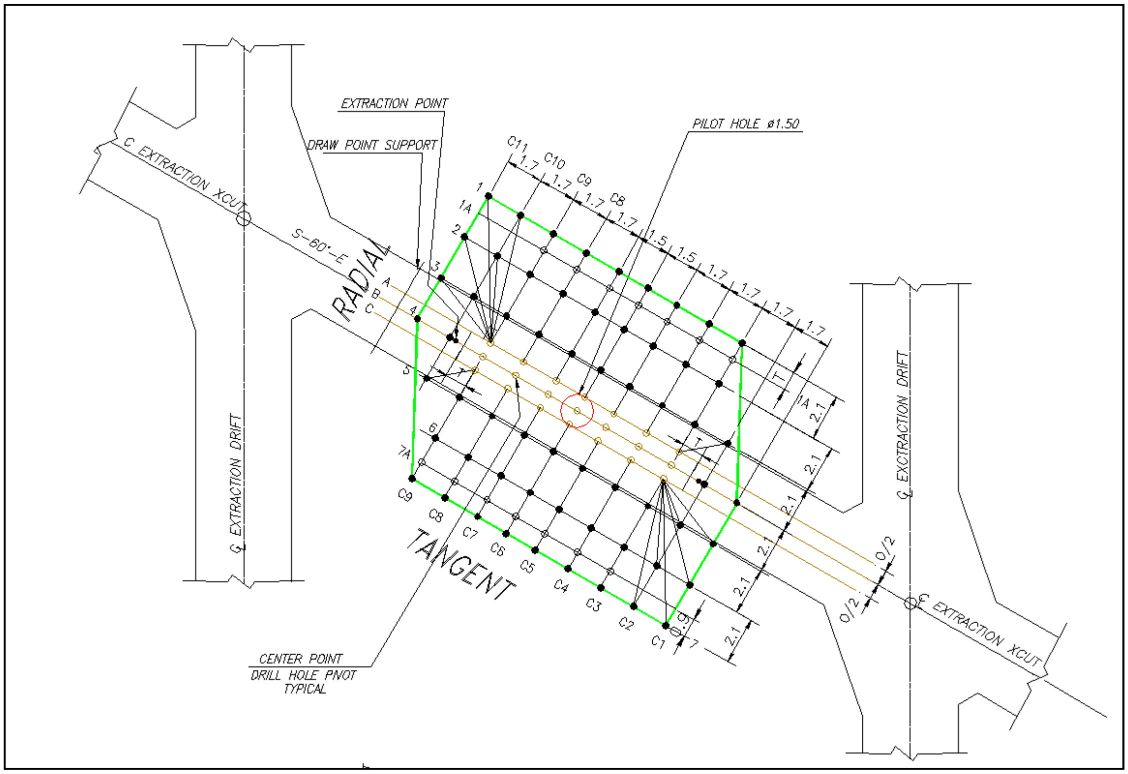
The majority of the main ventilation infrastructure is also located on the extraction level. It consists of two fresh air portals, two fresh air drifts, a perimeter ventilation drift, multiple internal ventilation raises, and a return air drift. The internal ventilation raises are located below and approximately in the middle of the extraction drifts, which allows multiple workplaces in one extraction drift. More information concerning the ventilation system can be found in Section 8.1. A breakdown of the horizontal and vertical lengths that make up the extraction level is shown in Table 13.

Table 13: Estimated Lengths of the Various Openings on the Extraction Level

Item	Drift Dimensions (W x H, m)	Length (m)
Ramps	5 x 5	3,100
Extraction Drifts	5 x 5	9,600
Drawpoints	4.5 x 3.5	16,400
Perimeter Drift	5 x 5	2,300
Return Air Drift	7.5 x 7.5	1,600
Fresh Air Drifts	5 x 5	3,900
Exhaust Air Raises (diameter)	2.0 – 4.0	1,535
Fresh Air Raises (diameter)	2.0 – 4.0	300

7.4.1 Drawbell Excavation and Final Drawpoint Support

The drawbell excavation and drawpoint setup is based on the El Teniente design, which matches well with the undercut blasting design. The drill pattern for the proposed drawbell excavation is shown in Figure 31 and contains approximately 95 holes and 500 m of drilling. The final support for the drawpoints includes steel sets and shotcrete, spaced 1 m apart and 5 m back from the brow. Additional information concerning the ground support can be found in Section 7.6.



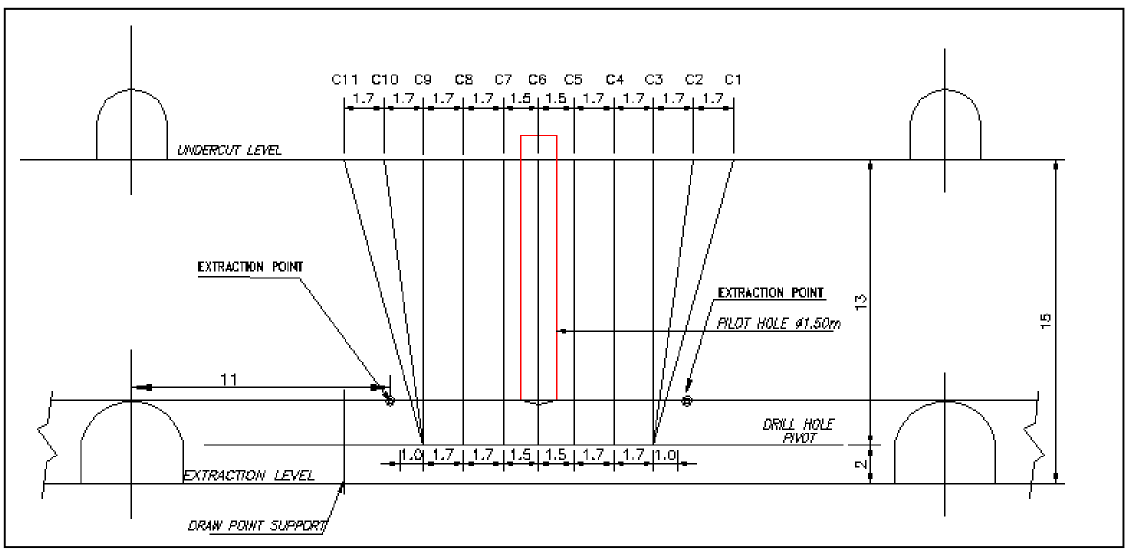


Figure 31: Plan and section view of the proposed drilling and blasting pattern for the drawbells used in the El Teniente layout

7.5 Conveying Level

The conveying level consists of two drifts located on the north and south sides of the footprint (Conveyor No. 1 and Conveyor No.2) and a third (Conveyor No. 3) drift that is parallel to the MTT access ramp (Figure 32). Short, perpendicular crosscuts between the drift for Conveyor No. 3 and the access ramp are planned for increased development productivity and emergency egress. The conveying levels are designed to be 5.5 m wide by 4.4 m high and have a total length of 3,200 m. The conveying level is located 40 m (floor to floor) below the extraction level. A plan view of the crushing and conveying level is shown in Figure 32.

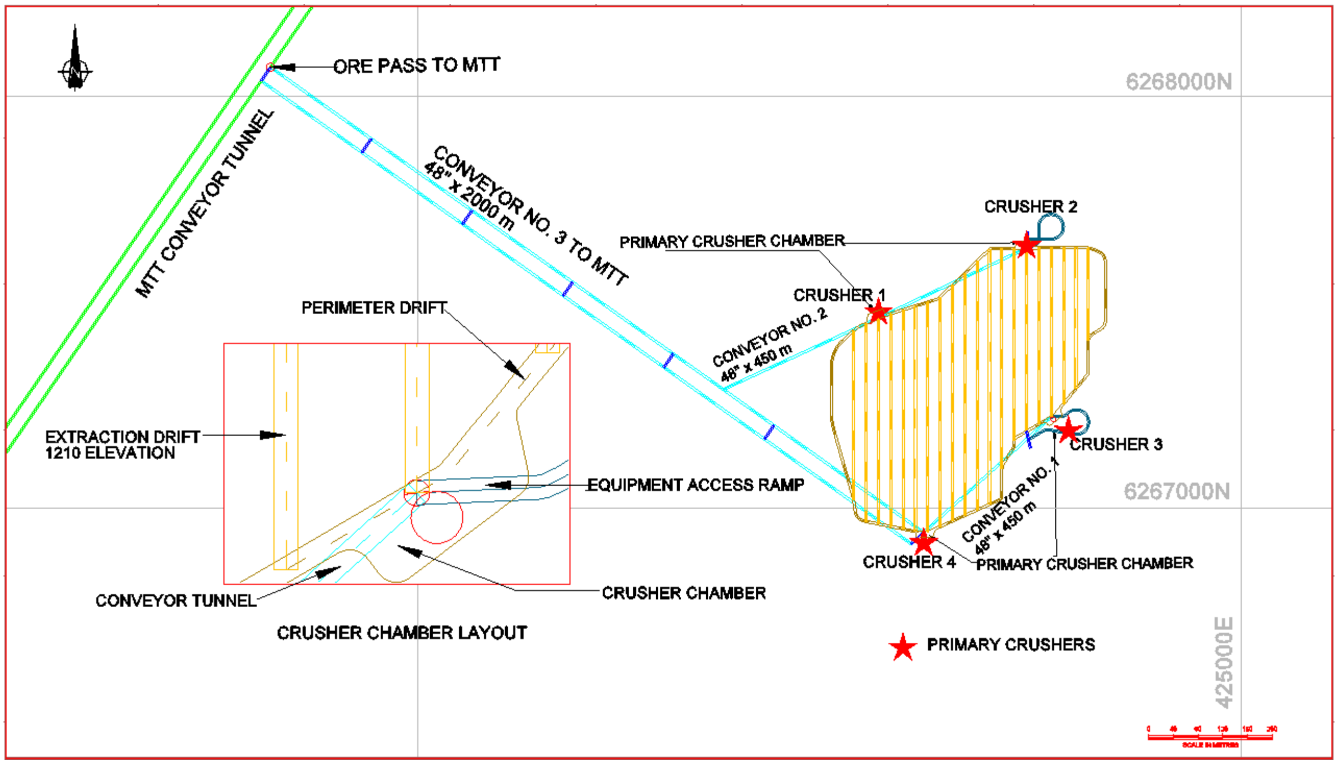


Figure 32: Location of the proposed conveying level (cyan) including the extraction drift (yellow) and the general location of the primary crushers (star) as reference

7.6 Ground Support Design

Ground support requirements for different development and infrastructure excavations have been estimated based on experience at other operations with similar rock quality and verified using empirical ground support design charts proposed by Grimstad and Barton (1993). The charts relate rock mass quality (Q), excavation span, and service use of excavation to ground support requirements.

The “equivalent dimension” of each excavation is used for support design and is defined as the ratio of the excavation span to the Excavation Support Ratio (ESR). The ESR is a factor of safety term dependent on the intended service use of the excavation. An ESR value of 1.6 has been used for the permanent ground support design, as recommended for permanent entry mining excavations (Grimstad and Barton 1993).

Rock mass quality was estimated from core logging data collected in the central boreholes, as discussed in Section 5.1. Q-values were estimated using the Norwegian Geotechnical Institute's (NGI) Q-system of rock mass classification (Barton et al. 1974). The system develops a numerical estimate of the quality of the rock mass based on the following expression:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

Where: *RQD* = rock quality designation

J_n = joint set number

J_r = joint roughness number

J_a = joint alteration number

J_w = joint water reduction factor

SRF = stress reduction factor

The rock quality classes defined in the Q-system (Barton et al. 1974) are summarized in Table 14.

Table 14: Q-System (Barton et al. 1974)

Rating	Description
0.001 – 0.01	Exceptionally Poor Rock
0.01 – 0.1	Extremely Poor Rock
0.1 – 1	Very Poor Rock
1 – 4	Poor Rock
4 – 10	Fair Rock
10 – 40	Good Rock
40 – 100	Very Good Rock
100 – 400	Extremely Good Rock
400 – 1000	Exceptionally Good Rock

Estimates of Q have been based on logged parameters from the 2010 field program (BGC 2011). An SRF value of 2 was assumed (appropriate for high stress, tight rock conditions) and a J_w of 1 (moist, low flow). The average estimated Q-value for the rock mass above where excavations will be created is approximately 9, indicating fair rock conditions.

Figure 33 shows the approximate Q-values for Iron Cap plotted on the empirical ground support chart for the different excavations that may require support.

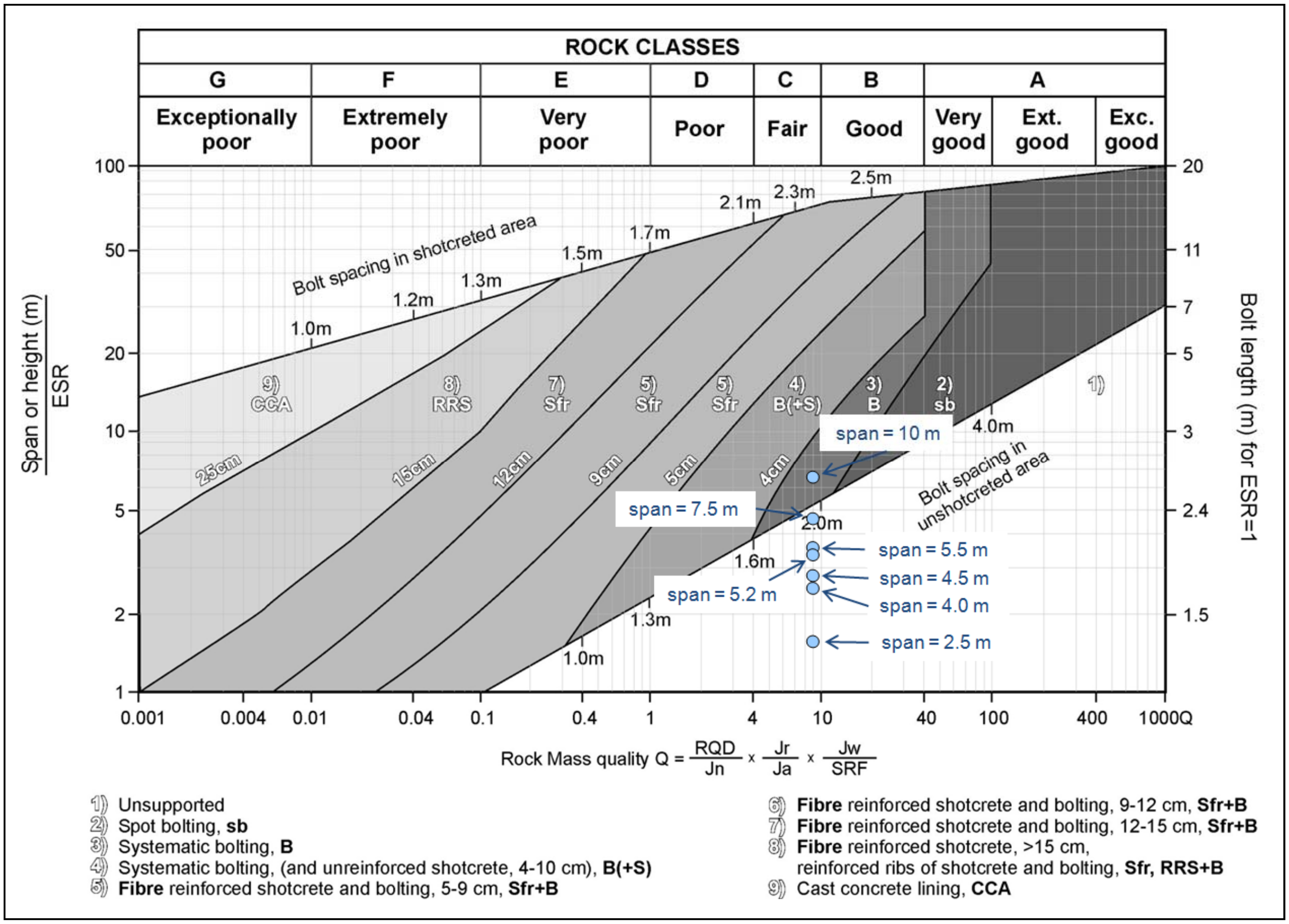


Figure 33: Empirical ground support design chart (Grimstad and Barton 1993)

The types of support recommended for the Iron Cap mine infrastructure are summarized in Table 15. The excavations listed in the table below are shown in Figure 26.

Table 15: Ground Support Recommended for Iron Cap Mine Infrastructure

Description	Span (m)	Height (m)	Type of Support
Drive (access ramp, haulage level, secondary breakage level)	5	5	2.4 m bolts on a 1.2 m pattern, mesh
Drawpoints	4.5	3.5	1.8 m bolts on a 1.2 m pattern, installed to back and walls as close to sill as possible 50 mm of mesh reinforced shotcrete Secondary support will likely consist of welded steel H-beams encased in concrete
Extraction Drifts	5	5	2.4 m bolts on a 1.2 m pattern, installed to back and walls as close to sill as possible 50 mm of mesh reinforced shotcrete
UC and PC Drifts	4.0	4.0	1.8 m bolts on a 1.2 m pattern, mesh
Conveyor Drive	5.5	4.4	2.4 m bolts on a 1.2 m pattern, mesh
Return Air Drive	7.5	7.5	2.4 m bolts on a 1.2 m pattern, mesh
Crusher and Shops	5.5	7.5	3.0 m bolts on a 1.2 m pattern 50 mm of mesh reinforced shotcrete
Internal Vent Raise	2.5	-	

Note that these ground support recommendations are preliminary and are intended for pre-feasibility level costing purposes only. A more detailed evaluation of the requirements for each specific excavation should be undertaken as part of future studies.

7.7 Material Movement

It is estimated that the Iron Cap deposit will be able to generate 45,000 tonnes of ore per day and between 600 and 1,800 tonnes per day of ore or waste from development (depending on the stage of development). The ore will be hauled directly from the drawpoints to one of four 107 x 165 cm (42" by 65") gyratory crushers installed on the extraction level perimeter drift. To maintain trafficability, a 0.3 m thick reinforced concrete sill will be installed on the extraction level.

The crushed material will be transported by one of two 914 mm conveyor belts which both feed a 1.22 m conveyor. The latter conveyor will transport the production material to a 2,000 tonne surge bin before being fed onto the MTT conveyor and transported to the plant site. A process flow diagram of the material movement system as well as plan and section views of the crushers can be found in Appendix C.

All material from development activities will be trucked to one of the crushers and will not be separated into ore or waste. The majority of the mine development at Iron Cap will be in mineralized rock that is above the NSR cutoff. The remaining quantity below the NSR cutoff is small enough relative to the production of the overall KSM site that, if mixed with the production ore, will have minimal impact on the overall ore grade. Separating the waste from the ore stream is not practical for the proposed mine.

7.8 Mobile Equipment

The mobile equipment in this design is typical of that used in underground mines and is outlined below in three categories: production, development and service. The production equipment comprises those pieces directly related to moving ore to the crushers (LHDs and secondary rock breakers). The development equipment includes LHDs and trucks as well as the AnFo loaders and ground support machines. The service equipment is used for construction and mine maintenance. The quantity of each equipment type in each category will be discussed in Section 11.2.3.

The development equipment was chosen to efficiently excavate the variety of drift dimensions planned for the mine. The face drill is a two-boom jumbo capable of drilling faces with a cross-sectional area ranging from 8 m^2 to 60 m^2, that can accommodate the small PC and UC drifts (16 m^2) as well as the large Return Air Drift (RAD) (57 m^2). The development LHD is 4.6 m^3 and has been matched with the 18 m^3 (40 tonne) development truck to ensure efficient face cleaning and truck haulage for each round.

Vertical development in the mine is required for the mine ventilation system. The raises are less than 100 m long with diameters varying between 2 and 4 meters. Raisebore drilling was chosen to excavate the planned raises because they are efficient at excavating this range of opening and because circular openings offer less resistance to airflow.

A variety of ground support equipment will be required to install ground support. Bolters are required to bolt and screen the back and walls of the development headings, and concrete mixers and shotcrete sprayers are required to supply concrete/shotcrete where needed. For example, the final drawpoint support requires shotcrete, and the floor of the extraction drifts and drawpoints require concrete.

An 8.6 m^3 production LHD was chosen because it is the largest LHD that can fit within the 15 m by 15 m El Teniente drawpoint layout. With the proposed configuration there is approximately 11 m between the brow of the drawpoint and the centreline of the extraction drift, and this machine is sized appropriately. The production drills were chosen because they are the smallest drill that can drill the specified pattern required to blast the undercut and drawpoints (longest hole is 16 m).

Multiple secondary rock breakers and block holers have been included in the design. The secondary rock breakers consist of an LHD frame with a rockbreaker attachment in place of a bucket. These machines are flexible and quite mobile. The block holers are designed for rocks that are too big to move or hang-ups that develop in a drawpoint. These units can setup, drill and load remotely keeping the operator in a safe location.

Both large and small personnel carriers are included in the design. The proposed Iron Cap mine is located approximately 23 km from the MTT access portal. It is envisioned that the large personnel carriers will be used at shift change to transport the workforce to and from the mine. The small personnel carriers will be used by staff to access the mine. The remaining mobile equipment (AnFo loader, grader, scissor lift, and boom truck) will be used as service vehicles and to install and maintain mine services (e.g., air and water pipes, ventilation ducting, and pumps). The scissor lifts and boom trucks will also be used to help with the construction of drawpoints.

At peak operation, Iron Cap will require a fleet of approximately 67 pieces of mobile equipment (Table 16). The actual quantity of equipment underground at any one time will vary depending on development and production activities and on the equipment replacement schedule. This list of equipment was used in the design of the mine services as discussed in the Section 8.0.

Table 16: Peak Mobile Equipment Requirements for the Iron Cap Mine (Year 7)

Fleet	Equipment	Number
Production	**Production Drill**	8
	LHD	14
Development	**Face Drill**	3
	Bolter	5
	LHD	3
	Truck	3
	Raisebore Machine	1
Service	**ANFO Loader**	2
	Scissor Lift	4
	Boom Truck	2
	Blockholer	2
	Mobile Rock Breaker	2
	Shotcrete Sprayer	2
	Concrete Mixer	2
	Grader	2
	Small Personnel Carrier	5
	Large Personnel Carrier	7
TOTAL		**67**

7.9 Mine Workforce

The mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life. Table 17 shows a list of the positions required at Iron Cap and the peak labour quantity separated into five categories; management, technical, maintenance, development and production. Note, peak labour occurs 5 years before peak production.

Table 17: Peak Labour Quantities by Mine Position (Year 7)

Job Title	Peak Quantity
Management	
Mine Manager	1
Underground Superintendent	1
Technical Superintendent	1
Clerical/Admin	8
Technical	
Senior Engineer	8
Engineer	17
Technologist	20
Trainers	8
Geologists	8
Safety Technician	8
Maintenance	
Planner	4
Maintenance Supervisor	4
Stores Person	8
Electrician (In-house)	32
Mechanics (In-house)	28
Development Crew	
Shift Captain	4
Shift Supervisor	8
Jumbo Operator	12
Bolter Operator	20
Loader Operator	12
Truck Operator	12
Crane Truck	8
Shotcrete Sprayer Operator	8
Concrete Mixer Operator	8
Grader	8
Construction	32
Labourer\Trainee	44
Production Crew	
Shift Captain	3
Shift Boss	12
Loader Operator	56
Secondary Breaker Operator	8

Job Title	Peak Quantity
Block Holer Operator	8
Grader Operator	8
Crusher/Conveyor Operator	20
Production Driller	32
Anfo Loader	8
Construction	32
Labourer \Trainee	10
Total	**548**

The estimate required labour required was primarily based on the quantity of mobile equipment. One operator per shift was assumed for all mobile equipment. Mechanics and electricians were estimated using a factor of 0.4 mechanics/electricians per piece of major mobile equipment (such as LHDs and trucks). Additional electricians were included to account for the production drills and the large quantity of installed equipment (crushers and conveyors). Finally, an estimate of the trainee/labourer was included for development and blasting helpers. The mine staff was estimated considering the amount of work that would be required to start and maintain a block cave mine of this size.

7.10 Summary and Contingencies

Table 18 contains a list of the different excavations, their dimensions and total estimated LOM lengths. Also included in the table are the quantities of rehabilitation (i.e., any existing drift that has to be repaired because of damage that occurred during the mining process) and contingency (i.e., design allowance for openings that were not included in the design such as sumps, and overbreak allowance for poorly blasted rounds or poor survey).

Table 18: Iron Cap Drift Dimensions

Excavation Type	Width (m)	Height (m)	Length (m)
Fresh Air Drifts	5	5	3,900
Excavation Level	5	5	9,600
PC Level	4.0	4.0	7,200
Drawpoints	4.5	3.5	16,400
UC Level	4.0	4.0	9,400
Return Air Drift	7.5	7.5	1,600
Internal Ramps	5	5	3,100
Shops	5.5	7.5	600
Mitchell Tegan Tunnel Access	5.5	5.5	3,400
Conveyor Level	5.5	4.4	3,300
Internal Ventilation Raises	2.0 – 4.0	---	1,500
		Rehabilitation (10%)	6,000
		Contingency (5%)	3,000
		Total vertical	1,500
		Total lateral	60,700

8.0 MINE SERVICES

8.1 Ventilation

8.1.1 Design Parameters

The ventilation design for the Iron Cap deposit was modelled using Ventsim Visual Standard and according to best practices established in Malcolm J. McPherson's textbook, *Subsurface Ventilation & Environmental Engineering*. The required airflow for the Iron Cap mine is 526 m^3/s based upon the total diesel equipment used on each mining level including a minimum 20% contingency. The ventilation system provides 0.012 m^3/s of usefully employed air per tonne mined, which is very close to the 0.013 m^3/s per tonne benchmark set for well-ventilated block cave operations (De Souza 2008). The total airflow requirement of 526 m^3/s is sufficient to appropriately dilute all noxious gases and particulate matter produced by the mining equipment and activities on each mining level.

To achieve the required airflow of 526 m^3/s, two 670 kW (900 HP) surface fans with variable frequency drives are required. These fans will consume approximately 630 kWh per fan at a total fan pressure of 1,902 Pa. The modelled network efficiency is 77.5%.

The Iron Cap ventilation model is designed to operate as a positive pressure or forced air system to facilitate mine air heating during the winter months and to prevent any air being drawn into the mine through the caved material. The fan operating points and volumes are sufficient for estimation purposes as part of the pre-feasibility study. Additional ventilation engineering is required to establish final operating points for the various stages of mine development.

Heating of mine air in the winter months is included in the design and cost estimates for Iron Cap. Heating of the mine air will be done by mine heaters located at each of the three main fan installations. Based upon Environment Canada temperature data for Stewart, British Columbia, the mine air heaters will have to provide approximately 24 million BTU per hour (MMBTUH) to heat 526 m^3/s from a low of -6°C (average January low) to 3°C. Heat calculations did not include any consideration of the heat transferred from the strata and mining equipment but should be investigated in the next level of study.

The total airflow requirements were based upon air quantities of 0.063 m^3/s per kilowatt of diesel equipment (i.e., 100 cfm/bhp), equipment utilization, and engine utilization. Equipment utilization was calculated based upon production requirements and availability while diesel utilization was estimated based on the engine's average working load of the engine. The minimum contingency of 20% per mining area is to account for any additional air losses. A pie chart indicating the total breakdown of air quantities, including air leakage to surface, can be found in Figure 34. Air leakage to surface through the caved material accounted for approximately 25 m^3/s. This is 15% of the total airflow, which is approximately equal to the amount of leakage reported at the Henderson block cave operation in Colorado (Nelson 2011, pers. comm.). A 20% contingency has been included in the airflow requirements for each level and accounts for areas that are not normally modelled at this level of study including sumps, refuge stations and crusher stations. Transient air losses, which include leakage to surface and contingencies, were approximately 21% of the total air quantity.

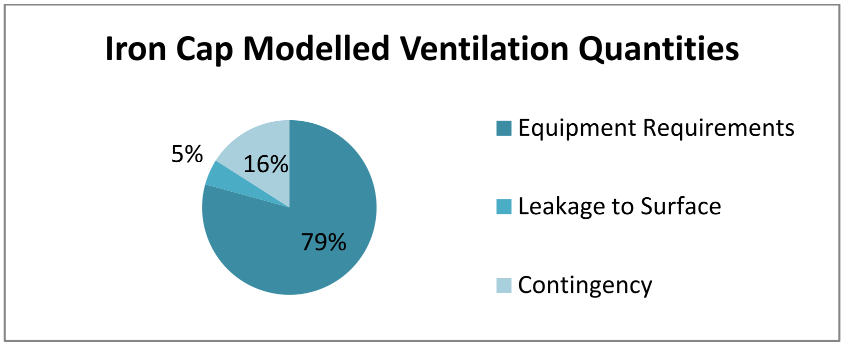


Figure 34: Iron Cap ventilation breakdown based upon 526 m3/s of total airflow

Modelling of the mine indicated that total mine resistance will be 0.00689 Ns^2/m^8. Friction factors used for the modelling were assumed to be typical for hard rock mining applications and can be found in Table 19. Airway shock losses were assigned automatically by Ventsim. These values were reviewed and set manually where deemed necessary. The modelled air velocities and design criteria can be found in Table 20. Air velocities on each level will be discussed in more detail in the design portion of this section.

Table 19: Iron Cap Ventilation Model – Friction Factors

Drift Type	Friction Factor (kg/m^3)	Comments
Typical Drifts	0.0120	Average blasted
Ventilation Raises	0.0050	Raise bored airways
Conveyor Drifts	0.0208	Due to conveyor in drift

Table 20: Iron Cap Ventilation Model – Air Velocities

	Design Criteria		Model	
Area	**Min.**	**Max.**	**Min.**	**Max.**
Working Faces	0.5	4.0	0.5	0.9
Conveyor Drifts	1.0	5.0	1.0	2.1
Mine Intake Drifts	2.5	10	8.7	8.7
Main Return Drifts	1.0	15.0	3.1	7.9
Ventilation Raises	2.5	20.0	10.2	18.4

Drift and raise dimensions vary throughout the mine. In the ventilation model it was assumed that all drifts would have a square profile and that raises would have a round profile. An overall list of the drift dimensions by mine area can be found in Table 21, and a list of raise dimensions can be found in Table 22.

Table 21: Iron Cap Ventilation Model – Drift Dimensions

Drift Type	Level	Width (m)	Height (m)
Fresh Air Drifts	1210	5	5
Excavation Level	1210	5	5
PC Level	1290	4.0	4.0
UC Level	1230	4.0	4.0
Conveying Level	1170	5.5	4.4
Return Air Drift	1140	7.5	7.5
Mitchell Tegan Tunnel Access	1000	5.5	5.5

Table 22: Iron Cap Ventilation Model – Raise Dimensions

Ventilation Raise Description	Diameter (m)	Quantity
Extraction Level to the RAD	2.0	20
Preconditioning Level to the Extraction Level	2.2	1
Undercut Level to the RAD	4.0	1
Crushing Level to the MTT	2.0	1
Crushing Level Ramp to the MTT	2.0	1
FAD to the Crushing Level	2.0	1

The air quantities on each mine level in the model were dictated by the assumed diesel equipment that will be active on each level during late stage development and production, using a conversion of 0.063 m^3/s per engine kW. These assumptions are based upon development occurring throughout the mine, in addition to production activities on the extraction level.

A breakdown of the ventilation requirements for the extraction level is shown in Table 23. The ventilation requirements for the remaining levels, except the Return Air Drift (RAD), can be found in Appendix D. The RAD ventilation assumptions were not included here as this will be a single development heading without any production related equipment requirements. The RAD will be ventilated by return air during late stage development and production.

Table 23: 1210 Extraction Level Ventilation Requirements

Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
Production Drill	2	74	79%	25%	2
LHD	16	352	78%	75%	207
Face Drill	1	120	80%	25%	2
Bolter	2	115	71%	25%	3
Truck	3	405	52%	75%	30
Anfo Loader	1	111	43%	75%	2
Scissorlift	1	95	83%	50%	2
Crane Truck	1	95	63%	50%	2
Mobile Rockbreaker	2	95	63%	75%	6
Secondary Rockbreaker	2	75	63%	75%	4
Shotcrete Sprayer	2	96	63%	75%	6
Concrete Mixer	2	155	63%	75%	9
Grader	2	114	63%	100%	9
Toyota	3	96	75%	75%	10
Personnel Carrier	3	130	25%	50%	3
Subtotal					295
Contingency					20%
Total required					**354**
Modelled quantity (excluding leakage)					**354**
Modelled leakage through cave					20

8.1.2 Design

A general ventilation flowchart of the Iron Cap ventilation model is outlined in Figure 35. The goal of the system is to ensure that a maximum amount of fresh air reaches each level in the mine during late stage development and production to meet equipment requirements while minimizing development costs, ventilation costs, leakage, and recirculation.

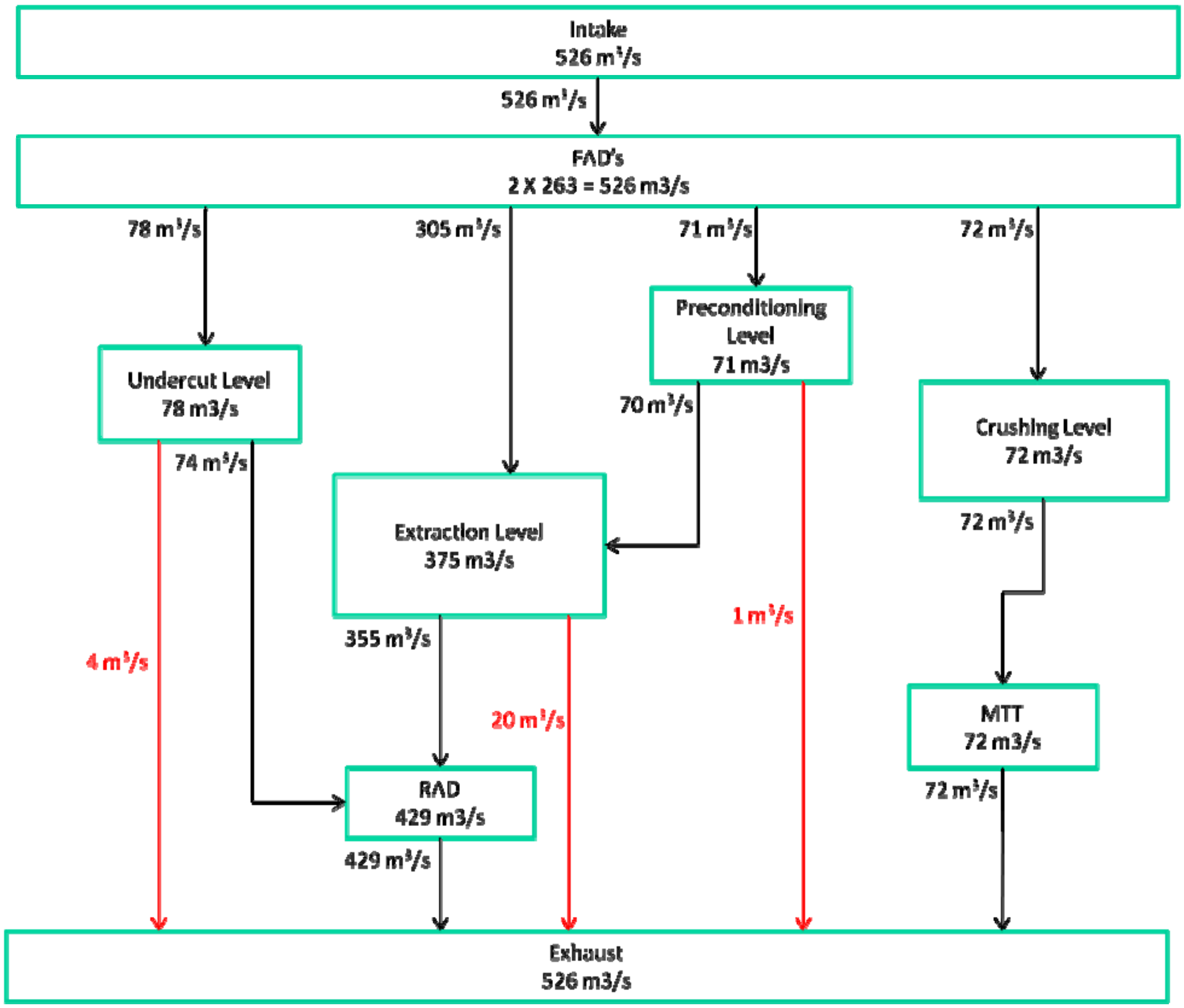


Figure 35: Iron Cap ventilation flowchart; red indicates leakage to surface

Fresh air is drawn from the surface though two intake adits. These drifts enter the mine at the same elevation as the extraction level, at 1,210 metres above sea level. The two intakes then feed into a perimeter ventilation drift that surrounds the extraction level. This perimeter drift distributes fresh air to each of the mine levels though drifts, ramps, and ventilation raises. The two intake adits and the ventilation perimeter drift comprise the Fresh Air Drifts (FAD).

Air from the FAD travels to the undercut (UC) level (1230 Level) via three ramps which are collared off crosscuts between the fresh air and perimeters drifts. The ramp collars are located on the crosscuts so that mine personnel do not have to enter into the FAD where air speeds are as high as 8.1 m/s. The air flows through the UC level and down a 4.0 m ventilation raise to the RAD (1140 Level).

The preconditioning (PC) level (1290 Level) is ventilated by fresh air from one of the ramps that also ventilates the UC level. The southwest ramp enables air from the FAD to travel up to the UC level and then continue on to the PC level. Air from the PC level then circulates around the level and down a 2.0 m diameter ventilation raise to the extraction level. The ventilation raise requires a fan to pull air up the preconditioning ramp, through the level, and down the raise to the extraction level.

The extraction level is ventilated through the ventilation raise from the PC level and nine crosscut drifts which connect the FAD to the extraction level. In order to ensure that the air flowing through the extraction level and down each drift is well balanced to meet production needs, it is important that the location of air inlets to the level, such as the nine crosscuts and air from the PC level, are strategically located to optimize flow down each extraction drift. The nine crosscuts are regulated with timber bulkheads to ensure the appropriate amount of air enters the extraction level.

In the current ventilation model 19% of the air on the extraction level is second pass air from the PC level. This was deemed acceptable because the PC level will have minimal activity once production begins. Recirculation will only be happening during mine development when the extraction level ventilation requirements are greatly reduced. When production commences, the air that was being drawn up to the PC level will go directly to the extraction level through the nine crosscuts connecting the FAD to the extraction level.

The air on the extraction level travels along each extraction drift from both directions and down a centrally located 2.0 m diameter ventilation raise to the RAD. There are currently 20 ventilation raises from this level to the RAD; one for each extraction drift. The ventilation raises are regulated to control the movement of air within the level and the extraction drift. The used air from the extraction level travels directly to the RAD which then exhausts the air to surface.

The crushing level (1170 Level) is ventilated by two ramps connected to the FAD. These ramps are both regulated to provide a total of 72 m^3/s of fresh air which eventually exhausts out to the Mitchell Tegan Conveyor Tunnel (MTCT). The interaction of the Iron Cap ventilation system with the MTCT was not modelled in this study. However, it is reasonable to assume that the MTCT will have flow-through ventilation into which the air from the Iron Cap mine crushing level will flow. It was also assumed that the MTCT does not contain regular traffic and the airspeed will not be significantly increased by the additional airflow.

The current ventilation model isolates the majority of the conveyor drift on the crushing level from the travel way through the use of ventilation doors. The majority of air entering the level is directed to the conveyor drift in an effort to increase the airspeed. The average airspeed in the conveyor drift ranges from 1.0 m/s to 2.1 m/s. Common rules of thumb indicate that conveyor drift airspeeds are in the range of 2.5 to 5.0 m/s with a minimum airspeed of 1.0 m/s. Increasing the airspeed in the conveyor drift at Iron Cap could be very expensive and may be unnecessary. Further study with respect to airspeeds and dust concentrations in the conveyor drift is recommended.

Over 80% of the air exiting the mine will do so via the RAD portal, and southwesterly winds, which are not common, could pose a problem for the quality of the air entering the mine should some of the return air be recirculated back through the mine. The RAD and FAD portal locations were selected with reference to the predominant southeasterly wind direction. However, the RAD and FAD portal locations and proximity with regard to return air dispersion in changing wind directions should be investigated in more detail.

The FAR and RAD portal locations are on the side of the Iron Cap Mountain and as such are susceptible to avalanches. For protection, an avalanche shed will be built to cover and protect the main fan infrastructure, including the fans, plenums, evase, heaters, and propane storage. Also, an escapeway will be installed so that personnel can exit the mine safely, with the additional purpose of providing access to clear snow build-up from the air intakes as necessary.

There are many ventilation controls within the Iron Cap Ventilation Model including ventilation doors and regulators. The majority of the air will be controlled using regulators installed above ventilation doors and/or modulating ventilation doors which can be used to regulate the ventilation flow through the doors. Mechanical ventilation dampers and traditional board regulators will be used to control airflow to and from ventilation raises.

Development ventilation of the Iron Cap deposit will be done using standard axial mine fans of various sizes. Large 112 kW development fans can provide approximately 26 m^3/s of air to an active mining face 800 m away (the current design does not have any planned development situations that would require a single heading development length of more than 800 m). This system would use 1.5 m diameter rigid and flexible duct and would cover the air requirements for one scoop and one truck.

Special consideration will be required with respect to the staging of the ventilation system as connections are made within and between levels. For example, it will be necessary to use the crusher excavations for ventilation of the crushing level development in order to ensure development headings have appropriate ventilation and that single headings do not exceed 800 m.

The Iron Cap ventilation system has been designed to be flexible, efficient, and capable of providing an adequate supply of fresh air to every area of the mine and to properly dilute the airborne contaminants produced in the mine. Additional information is available in Appendix D.

8.1.3 Future Work

There are a number of items in the Iron Cap ventilation model that will require further modelling, consideration, and review for a feasibility level study. These items include:

- the staging of the ventilation system with regards to the development schedule and time phases;
- a more in-depth review of transient air losses throughout the mine;
- further study into dust generation and concentrations in each mining area; and
- review of the current portal locations with regards to possible air recirculation.

Ventilation requirements are predominantly dependent on the mobile equipment fleet employed. Therefore, any changes to mining production rates and equipment feet will also require a review of the current ventilation model.

8.2 Dewatering

The mine water handling system is designed to handle the water that originates from the groundwater and surface inflows (including possible water from ice that caves into the crater when the ice field is undercut), and water that is introduced to the mine for operational purposes.

It was previously proposed to mine the Iron Cap deposit by open pit mining as discussed in the PFU (Seabridge 2011). As part of the studies for this, BGC estimated the natural groundwater inflows to the pit and the quantities of water that would be generated from the surface wells and horizontal drains required to meet the depressurization requirements to ensure the stability of the pit slopes. The overall impact from the proposed pit mining on the groundwater system is estimated to be larger than from the caving mining. This is particularly so when depressurization is included as well, which is not actually required for the caving mining, but which will occur to some degree just from natural inflows to the subsidence crater. Accordingly, the total groundwater flow (natural inflow and extracted water) from the pit study was considered to be a conservative estimate of the groundwater inflow to the underground workings. The estimated groundwater inflow is 5,140 m^3/d.

At the time of completing this pre-feasibility assessment, estimates by others of the surface inflows into the crater at Iron Cap were not available. These surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. In future studies, the water management system will need to be enhanced to cater for this additional inflow. The additional capital and operating cost for this enhanced system is not likely to be very significant compared with current costs, and not including them will not materially affect this pre-feasibility study.

As indicated earlier, the volume of ice that it is estimated will be undercut by the crater is approximately 3.4 million cubic metres. The ice that is undercut will cave into the crater over an estimated period of approximately 4 years. Assuming that this ice melts before it is mucked from the drawpoint, the average inflow to the mine workings is approximately 2,500 m^3/d.

The ice overlying the deposit forms part an icefield that extends to the north. Detailed investigations of possible water that may be pooled or that is flowing beneath the ice have not been undertaken, and no allowance for additional inflows associated with the ice cover that might flow into the crater, further investigations of this need to be undertaken as part of future studies of the project.

The underground water management system at Iron Cap is currently designed to handle 7,640 m^3/d. This caters for the groundwater inflow and the ice melt.

To provide for good drainage, the majority of the underground drifts have been graded so that water will run towards the MTT or towards a central collection sump located near the return air tunnel. The water draining towards the MTT will be collected and added to the existing “dirty” water system. The water from the sump will be drained out of the return air tunnel into the Mitchell Valley, where it will be collected in the existing surface water infrastructure. During an emergency flood event, the water will flow out of the return air tunnel.

8.3 Mine Water

It is estimated that the mine will require 28 m^3/hr of process water for the bolters and the development and production drills. In addition, it is estimated that 114 m^3/hr will be required by the conveyor fire suppression systems. This water will be supplied through the main access tunnel from the MTT through four 40 kW pumps. The water will be delivered to the working face through 0.2 m steel pipe with 10 mm wall thickness.

8.4 Mine Power

The mine power and communications design was completed by Neil Brazier of WN Brazier Associates Inc and the complete report can be found in Appendix E. It is estimated that the Iron Cap mine will require approximately 9,200 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans. A summary of the major contributors to the peak electrical load is shown in Table 24. The underground communications will be provided through a combination of Leaky-Feeder and Personal Electronic Device (PED) emergency warning system.

Table 24: A Summary of the Major Contributors to the Peak Electrical Load at Iron Cap

	Capacity (kW)	Quantity	Total Capacity (kW)	Efficiency	Demand Factor	Utilization (%)	Running Load (kW)
Jumbo	150	5	750	93%	0.8	75%	484
Bolter	56	6	336	92%	0.8	70%	205
Drills	56	8	448	100%	0.8	70%	251
Raisebore	242	1	242	91%	0.75	75%	150
Shotcrete	56	2	112	91%	0.7	75%	65
Pumps	56	6	336	92%	0.75	80%	219
Surface Fans	746	2	1492	93%	0.95	100%	1,524
U/G Fans	56	15	840	92%	0.9	75%	616
Air Compressors	373	2	746	100%	0.9	50%	336
Heating	200	1	200	100%	1	50%	100
Surface Miscellaneous	250	1	250	100%	0.4	100%	100
Conveyor 2	234	1	239	96%	78%	90%	166
Conveyor 1	407	1	462	95%	80%	80%	282
Conveyor 3	1562	1	3227	95%	90%	87%	2,384
Crusher	789	4	3183	95%	85%	75%	1,896
Lighting and Small Power	75	6	450	100%	1	50%	225
Heat Tracing	200	1	200	100%	1	35%	70
Refuge Stations	30	4	120	100%	0.6	100%	72
U/G Shop	60	1	60	100%	0.5	100%	30
Misc. Monorails	7	8	56	91%	0.85	5%	3
Misc. Sumps	15	8	120	91%	0.85	15%	17
Total							**9,200**

The main power will be supplied to the underground from the Mitchell No. 2 substation through a 25 kV cable hung from the back of the access ramp and conveyor tunnel to create a ring main style system. Each of the main levels will have a 25 kV line which will be stepped down to the required voltage by skid mounted dry-type transformers. Equipment that draws larger loads (e.g., ventilation fans, conveyors and crushers) will be equipment with a permanent transformer.

8.5 Compressed Air

It is estimated that the Iron Cap mine will require 362 m^3/hr of compressed air. This will be supplied by two compressors located underground but outside the active working area. One compressor has been sized to handle the estimated requirements and the second compressor will act as a backup.

Compressed air will be piped to the working face through 0.15 m steel pipe with a 10 mm wall thickness.

8.6 Support Infrastructure

The support infrastructure for the Iron Cap mine includes surface buildings and underground excavations that support the mine operations. The surface buildings including the change house, mine offices and warehouses will be part of the greater KSM complex, located in the Teigan valley, and were not part of the scope of this report. Underground, a small warehouse will be established next to the shop (located off of the main access tunnel near the north-east crusher). The underground will also be equipped with portable refuge stations located close to where the majority of the active mining will be occurring and small permanent refuge stations located at each of the crushers. These refuge stations will also act as underground offices.

9.0 MINE DEVELOPMENT SCHEDULE

The mine development schedule was created using Surpac's Minesched software package. The development schedule was separated into three phases. The first is the pre-production phase which involves developing the main access ramp and conveyor drift. The second phase, ore production, involves creating enough openings to start and ramp-up production from the cave. The third and final phase begins once the mine has reached steady-state production and the development fleet is only required to create enough openings to maintain production.

The underground mine will be operating 24 hours a day, 7 days a week and 365 days a year. Phase 1 development rates of one round per day per heading were assumed. Once the footprint is reached and there are many headings available, the development rate is increased to six rounds per day in Phase 2. Phase 3 begins after enough development has been completed to start the cave. At this time the development crews are reduced to excavate one round per day so that openings are excavated with a "just-in-time" philosophy. Each horizontal round is 3.7 m long, and each vertical "round" is 6 m long. The mine development schedule is shown in Figure 36. The schedule indicates that the first set of drawpoints will be ready in Year 5; therefore production cannot start until Year 6.

Rehabilitation of the lateral and vertical development was also estimated and is shown Figure 36. The quantity of rehabilitation required is estimated at approximately 10% of the development advance rate. The rehabilitation has been delayed until after the start of steady state production, as it is anticipated that this is when the majority of the degradation of the drifts will occur as a result of secondary blasting and changes in the stress field. The x-axis in Figure 36 is shown in Iron Cap project years and is not related to the overall site schedule.

It is estimated that Phase 1 development will produce an average of 216,000 tonnes per year (for 5 years) and it will be hauled to a waste dump in either Mitchell or Teigan valley. Waste generated after Year 5 will be sent to the mill as part of the ore stream.

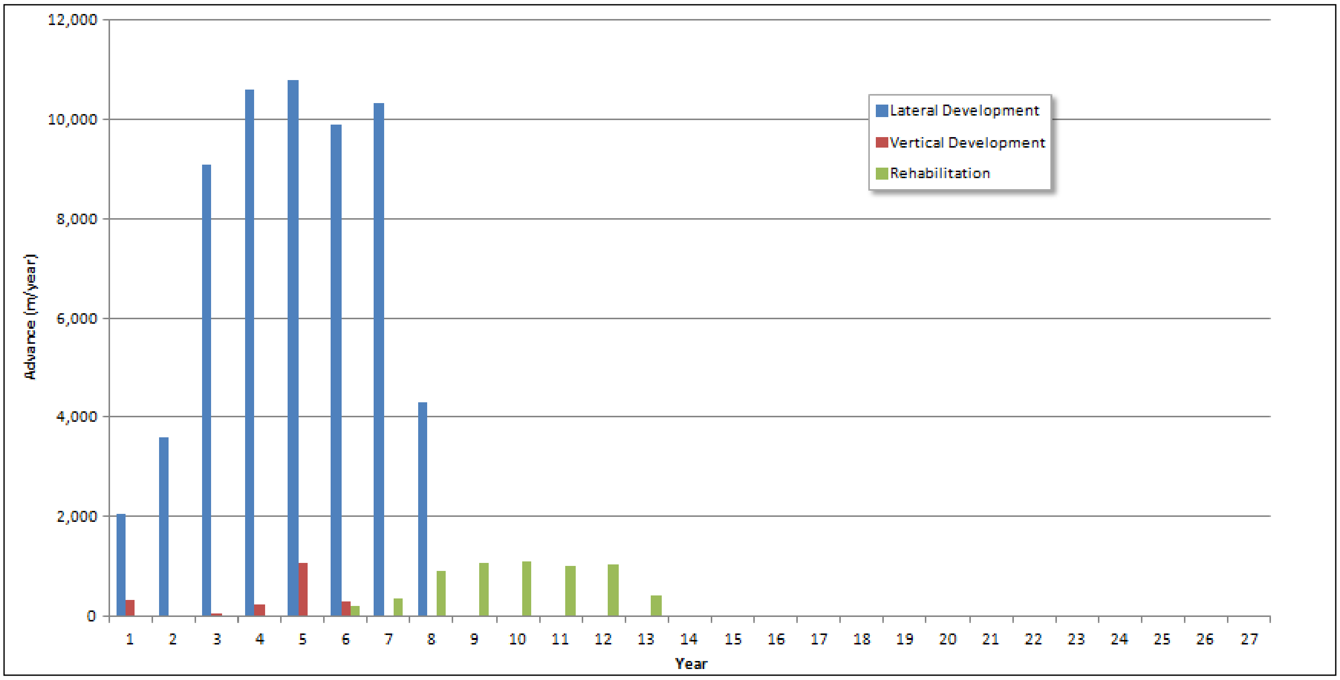


Figure 36: A chart showing the advance of lateral and vertical development and the quantity of rehabilitation estimated to be required

9.1 Mine Development Workforce

The development workforce was estimated based on the quantity of work required to construct and produce from the underground mine on an annual basis. Figure 37 shows the annual development labour and the quantity of development and rehabilitation per year. The development workforce includes all site personnel until the start of production in Year 8. After Year 8, only the labour and staff directly involved in the development of the mine are considered part of the development workforce (e.g., the jumbo operator, development truck driver, the development shift boss, and development planning engineer); the remainder are accounted for in the production workforce discussed in Section 10.3. This distinction is significant in terms of the split between mine operating cost and capital cost discussed in Section 11.3.

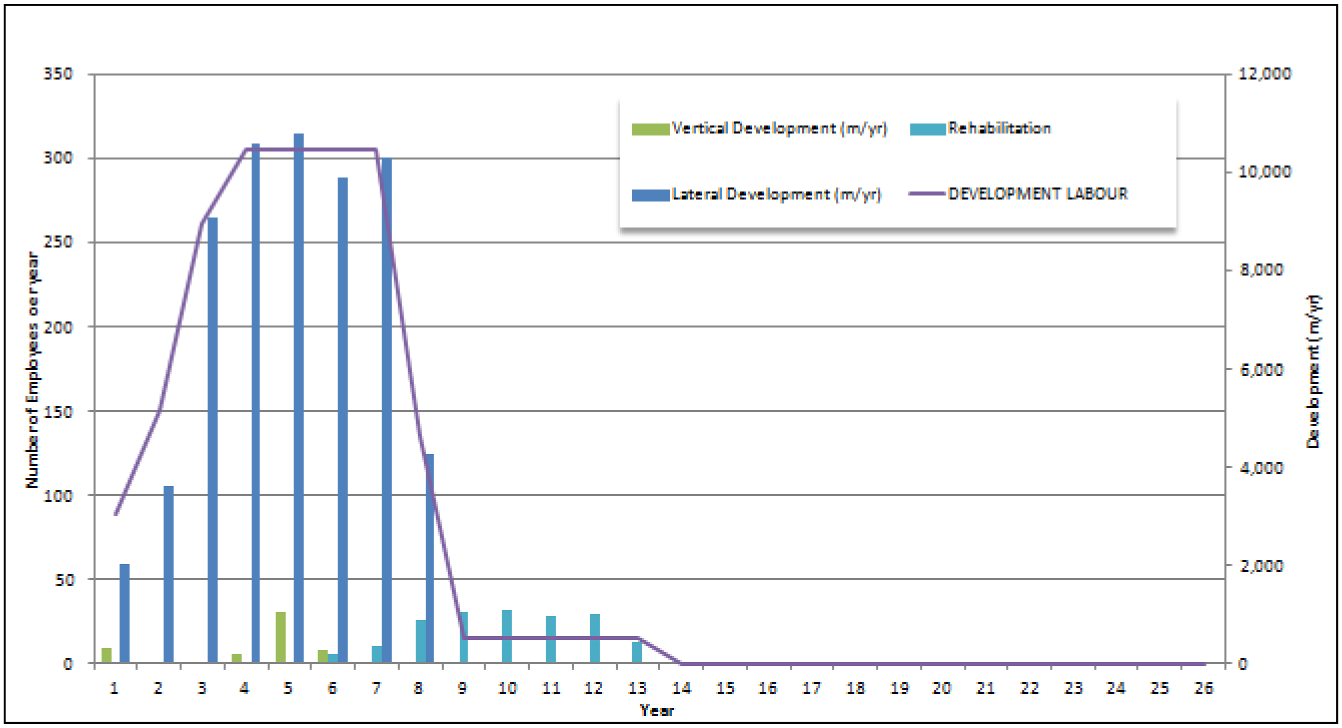


Figure 37: Chart showing the yearly development labour and the amount of vertical and horizontal development, and rehabilitation required per year

10.0 MINE PRODUCTION SCHEDULE

The mine production schedule was developed using Gemcom's PCBC software (information concerning the development and calibration of PCBC can be found on www.gemcomsoftware.com). PCBC is industry-recognized software that has been used for over 20 years to estimate production and grade profiles from different block cave mines around the world.

10.1 PCBC Input Parameters

PCBC requires certain input parameters which govern the rate at which mine production ramps up, the maximum production rate, and when a drawpoint is no longer profitable. These include the draw rate curve (more information on the draw rate curve can be found in Section 10.1.1), drawpoint construction rate, maximum production target, and the drawpoint spacing. Additional input parameters required include a cave material mixing algorithm, the drawcone layout, and the minimum and maximum height of draw. These were all based on recommendations from Gemcom and on published industry standards. PCBC also requires the block model (with an NSR attribute), surface topography for the area, and certain financial parameters to determine when a drawpoint is no longer profitable.

The key PCBC input parameters are detailed in Table 25. The mining and development costs were developed from first principals and are discussed in Section 11.3. The discount rate, milling and General and Administration costs were obtained from the PFS (Seabridge 2011).

Table 25: The List of Key PCBC Input Parameters

Item	Value	Unit
Mining Cost	6	dollars per tonne
Milling and G&A Costs	7.09	dollars per tonne
Discount Rate	5	%
Drawpoint Construction Rate	120	drawpoints per year
Yearly Production Rate	15,000,000	tonnes per year
Maximum Height of Draw	500	m
Drawpoint Spacing	15 x 15	m
Drawpoint Layout	El Teniente	

The ramp-up and maximum yearly mine production rates are determined by the drawpoint construction rate, and the initial and maximum drawpoint production rate. The drawpoint production rate, also known as the draw rate, is inputted in PCBC as the production rate curve (PRC). The values chosen for these items are based on industry averages adjusted to suit the expected situation at Iron Cap. In particular, the initial and maximum drawpoint production rates were reduced to simulate a production environment with large fragmentation. The draw rate and PRC are discussed in the following sections.

These input parameters were used in PCBC to evaluate the potential maximum yearly mine production rate from the Iron Cap footprint, which was found to be 20 million tonnes per year. After comparisons to planned, operating and closed block caving mines in a similar geological environment, this value was lowered to 15 million tonnes per year, which adds conservatism to the overall block caving design.

The maximum height of draw governs the tallest column that can be mined if the drawpoint material is still profitable. This parameter relates to the wear that develops at a drawpoint. At a certain height, the drawpoint becomes so damaged as a result of stress and the quantity of material that passes through it that it must be closed. Currently, 500 m is an accepted industry value, but the industry is trending to taller columns as more competent rock masses are caved and improved ground support techniques are developed.

A 15 m by 15 m drawpoint layout was used to accommodate the expected larger fragmentation from Iron Cap. This is based on empirical evidence collected by Laubscher (Laubscher 1994). This spacing influences capital costs as well as production rates and material mixing. As discussed in Section 6.3, future studies need to include more detailed assessments of the estimates of fragmentation and the selected drawpoint spacing. Such studies may lead to the conclusion that good interaction between draw zones can be still established and maintained with an expanded (and therefore more economical) drawpoint spacing.

It was assumed that sloughing of peripheral waste rock would occur into the crater and cover the upper surface of the material being drawn down. This was modeled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution.

10.1.1 Draw Rate

The overall draw rate expressed in millimetres per day is a useful reference indicator that allows a comparison to be made between production rates at various caving mines. Figure 38 shows the production and draw rate from a selection of active and historic block cave mines. Due to the large fragmentation that is estimated to report to the drawpoints at Iron Cap, particularly during the early stages of mining, a draw rate of 200 mm/day was chosen as a maximum cap in the PCBC analyses. As will be discuss in Section 10.2, an average draw rate of 110 mm/day is required to reach production goals at Iron Cap.

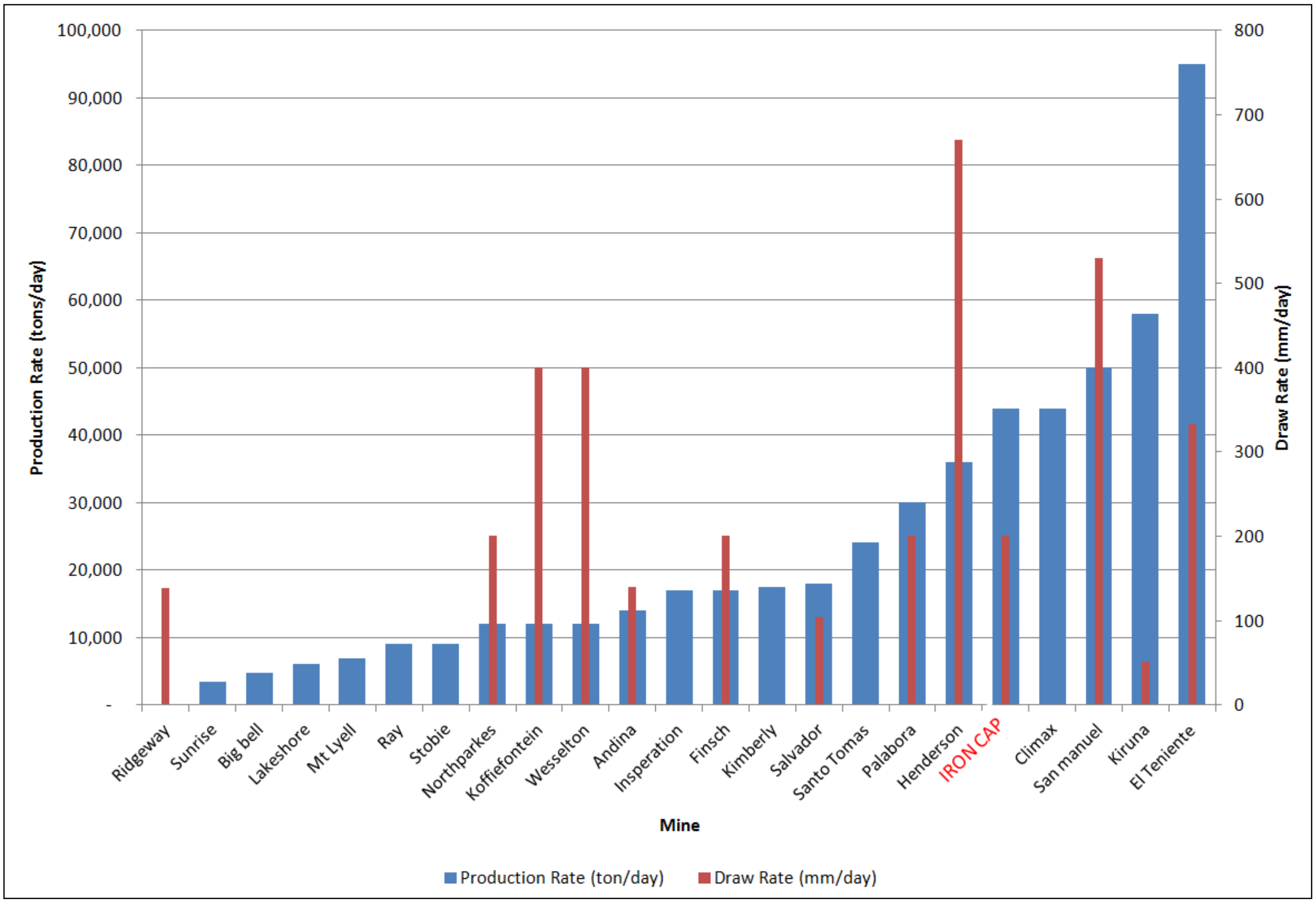


Figure 38: Draw and production rates for a selection of block and panel cave mines (after Woo, Eberhardt and van As, 2009)

10.1.2 Production Rate Curve (PRC)

The time required to reach the theoretical maximum production rate of one drawpoint is another influential parameter in PCBC. This rate is defined by the graph in Figure 39. It shows that, initially it is assumed at Iron Cap that a drawpoint can produce at a rate of 60 mm/day and that this can steadily increase until 50% of the column is mined. Then the drawpoint can produce up to the set maximum of 200 mm/day. This PRC matches actual production achievements at large fragmentation block cave mines such as Palabora, where the amount of secondary breaking required decreases after the first 100 m of a column is drawn (Ngidi 2007). Section 10.1.1 discusses the estimated draw rates per year for the Iron Cap mine.

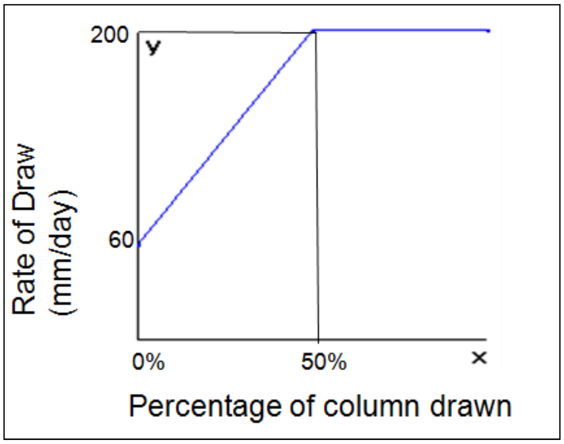


Figure 39: Production Rate Curve that is used to describe the draw rate of one drawpoint

10.2 Results of PCBC Analysis

The production schedule determined from PCBC for Iron Cap is shown in Figure 40. Iron Cap is estimated to have a production ramp-up period of four years, steady state production at 15 million tonnes per year for nine years, and then ramp-down production for another seven years. A breakdown of the production and average grade schedule can be found in Appendix F. The total production and average grade over the life of the mine is shown in Table 26. The period prior to production in Year 6is considered pre-production. This is when the majority of the ventilation and material movement infrastructure is excavated and installed. This is also the time when the development of most of the undercut and preconditioning levels occurs.

Table 26: Total Production and Average Grades for the Proposed Iron Cap Mine

Total Production	Au (gpt)	Cu (%)	Mo (ppm)	Ag (g/t)
193,360,000	0.45	0.20	21	5.32

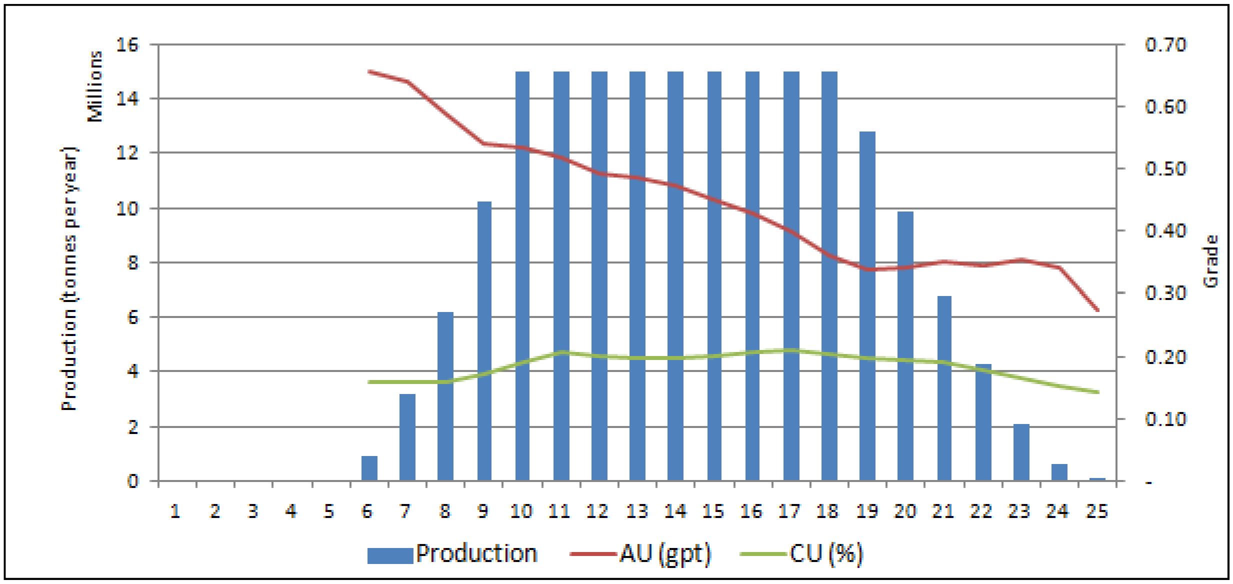


Figure 40: Yearly PCBC production schedule showing gold and copper grades

Drawpoint construction, maintenance, and utilization are important factors governing the ability of the mine to reach production targets. The PCBC production schedule discussed above was developed assuming a construction rate of 120 drawpoints per year. Figure 41 shows the number of active drawpoints in the mine over time, with new drawpoints being progressively established and old drawpoints being closed when the value of the material being mined does not exceed the NSR cut-off discussed in Section 3.1.

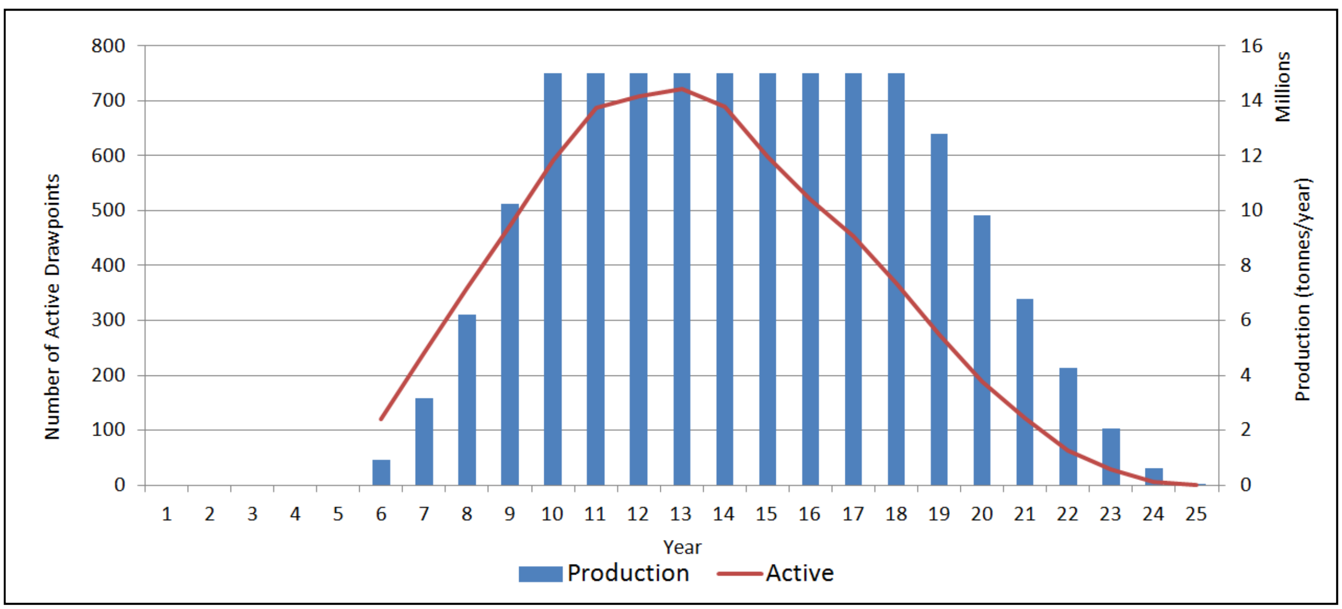


Figure 41: Chart showing the number of active drawpoints and production per year

Fragmentation is also expected to be a major influence in Iron Cap's ability to meet production goals. Many design factors have been included to manage the impact of large fragmentation, including the use of LHDs to haul directly to cone crushers, the use of multiple secondary breakers, and pre-conditioning the rockmass.

Another design parameter used was to limit the maximum production from one drawpoint to the maximum rate of draw. While the maximum draw rate was set at 200mm/day, the actual draw indicated by the PCBC analyses to achieve the production of 15 million tonnes per year is shown in Figure 42. This shows that the actual draw rate for much of the mining period is much less than the maximum at about year 23 is approximately 180 mm/day and is lower than the theoretical maximum input value of 200 mm/day. The average draw rate is 110 mm/day, which means that there are roughly twice as many drawpoints available as required to meet production targets.

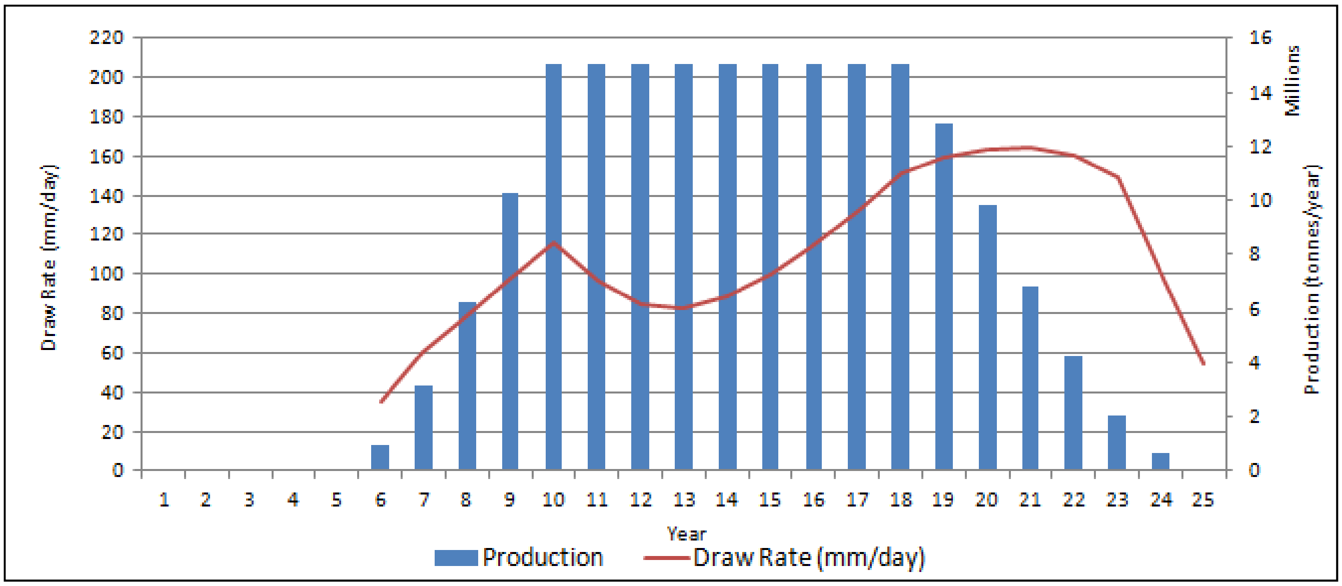


Figure 42: Yearly draw rate and production for Iron Cap

10.3 Mine Production Workforce

The production workforce includes equipment operators, mechanics, electricians, and all staff required to plan the mining processes, including engineers, technicians, and geologists after the mine starts production in Year 8. The production workforce also includes a construction crew, trainees and/or unskilled labour. The size of the production workforce is dependent on the quantity of mobile equipment and the stage of the mine life. This is shown in Figure 43. The sharp increase in the workforce initially represents the staff and labour that is planning the mine, and the maintenance and drilling labour that form part of the development workforce required to prepare the cave (drilling and blasting from the undercut and preconditioning levels). The separation of the production and development workforce is has a significant influence on the split between the mine operating cost and capital cost discussed in Section 11.3. The workforce starts to ramp down before production starts because certain positions are mainly required early in the mine life (e.g., secondary breaker operators, production drillers and construction workers).

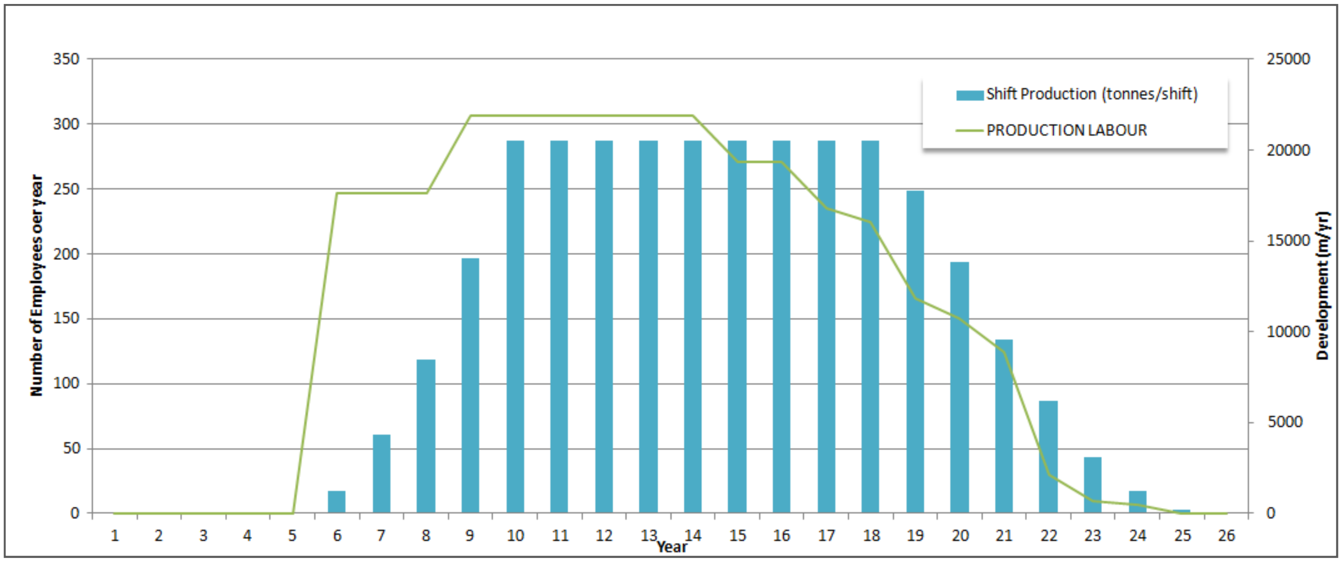


Figure 43: Yearly production workforce distribution

11.0 MINE COSTS

This section contains a description of the mine capital (CAPEX) and mine operating (OPEX) costs. Labour cost contributes to both the CAPEX and OPEX and is therefore discussed in a separate section.

11.1 Labour

The labour costs are based on estimated KSM project rates as provided by Wardrop (Wong 2010, pers. comm.). Where necessary, the rates were adjusted to reflect underground mining experience. For example, an additional category called “underground premium” was created and the “bonus” category was increased for underground workers. Different rates were applied to staff and labour. Table 27 shows the relevant mark-ups that were used to account for burdens, bonus, and remote and underground premiums.

Table 28 contains a list of the major labour categories showing annual base rates and “all-in” costs. The positions are separated into staff and labour. The staff category consists mainly of technical, supervisory and administration roles, while the labour category consists of the underground workers, including equipment operators, miners, mechanics, and electricians. There is provision for a construction crew that will be responsible for constructing the drawpoints. The mine will operate 365 days per year with mine labour (including the underground staff) working a 2 week in and 2 week out schedule and some surface staff (including engineers and geologists) may work a 4 days in and 3 days out schedule. It was assumed that all major installations (e.g., crushers, conveyors, main ventilation fans, and other mine infrastructure) will be completed by contractors.

Table 27: Breakdown of the Various Labour Mark-Ups

	Staff	Labour
Burden	35%	35%
Remote premium	10%	10%
Bonus	20%	40%
Underground premium	0%	15%

Table 28: The Yearly Base Rate and All-In Rate for the Different Mine Positions

	Level	Base Rate (per year)	All-in Rate (per year)
Staff	Mine Manager	$180,000	$297,000
	Chief Engineer	$180,000	$297,000
	Senior Engineer	$140,000	$231,000
	Mine Engineer	$115,000	$190,000
	Administration	$55,000	$91,000
	Mining Technologist	$75,000	$124,000
	Shift Captain	$120,000	$190,000
	Production Supervisor	$110,000	$182,000
	Shop Foreman	$110,000	$182,000
Labour	Operators	$84,000	$164,000
	Labourer	$65,000	$126,000
	Construction	$78,000	$152,000
	Mechanics	$92,000	$180,000
	Electricians	$99,000	$192,000
	Contract Labour	n/a	$200,000

The development and production workforce are indicated separately in Figure 44 for the life of the mine. The workforce ramps up as development headings become available and production starts. It reaches a maximum of 548 employees between years 5 and 8. After this the development is substantially completed and the crews are reduced. The size of the workforce begins to decrease prior to the production ramp down because less equipment is required to maintain production from the cave (e.g., production drillers, secondary breakers). After Year 18, the total workforce is directly influenced by the production ramp down.

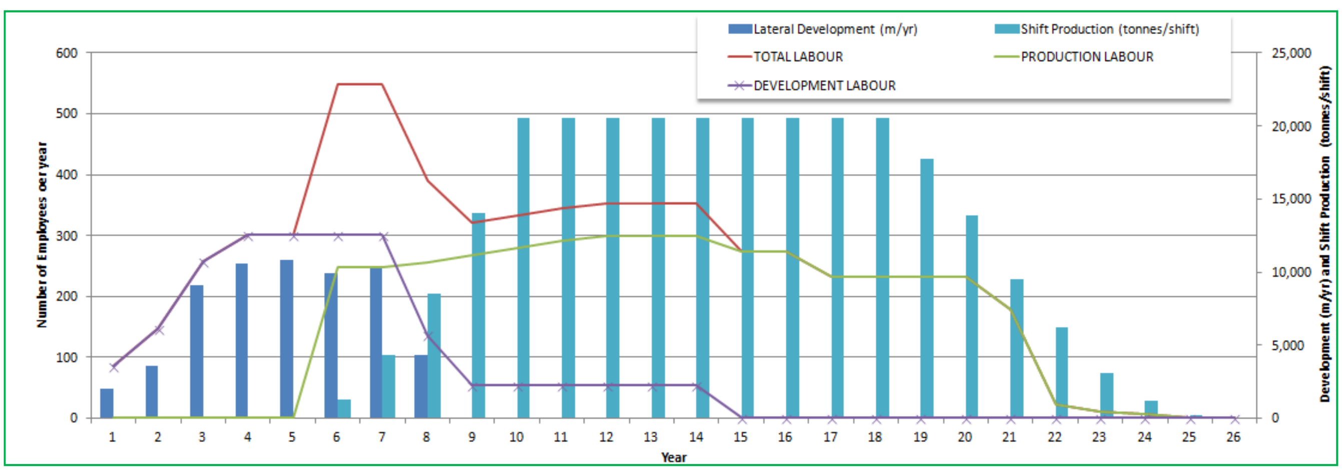


Figure 44: Distribution of the workforce and development advance per year and the production rate per shift

11.2 Mine Capital Costs

The mine capital costs include all equipment and excavations required to prepare, initiate, and maintain the cave. The cost includes excavation and hydrofracturing from the PC level, excavation and construction of the drawpoints, blasting of the undercut, and the associated infrastructure required to move mined material to surface.

The costs were developed from first principles using a detailed cost model. The costs for mine equipment and consumables were obtained from supplier quotations in 2011, Golder's database or from Wardrop. Mine contractor quotes were obtained in January 2012 and used to confirm derived cost estimates for all mine development work including lateral and vertical excavations. Wardrop estimated the equipment and installation costs for the major equipment used in the material handling system.

This discussion below describes each of the different openings that are in the design and describes the associated costs. Unless otherwise indicated, the costs presented here do not include labour costs, which were discussed in Section 11.1.

11.2.1 Mine Development

As described in Section 7.0, the proposed Iron Cap design has various sized openings with different purposes. Table 29 shows each of the different openings and the associated cost per meter. These rates include the cost of materials and equipment to create the opening. The cost for the ongoing activities within the opening (i.e., the cost to blast the undercut after it has been excavated or the cost to drill, blast, and excavate a drawbell) will be described in a separate section. The mine development costs include standard ground support (bolts and mesh) on the back and walls, and concrete floors in the extraction drifts and drawpoints (ground support requirements are discussed in Section 7.6). An example of the detailed cost calculation for a meter of development can be found in Appendix I.

An estimated rehabilitation cost of $1,200 per metre was used in this study. The actual cost of rehabilitating a drift will vary greatly depending on the extent of the damage and on the timing of the rehabilitation (i.e. is the rehabilitation considered preventative, completed to prevent further damage to the drift, or reactionary, done after the damage has occurred).

Table 29: Summary of the Unit Cost of the Various Development Sizes Proposed for the Iron Cap Mine

Description	Unit of measure	Unit Cost (ex. Labour)	Unit Cost (incl. Labour)
5.5 m x 5.5 m Drive [(1)]	m	$2,000	$4,200
3.5 m x 4.5 m Drawpoint	m	$2,700	$5,000
5.5 m x 5.5 m Extraction Drifts	m	$4,200	$6,600
4.0 m x 4.0 m Undercut and Preconditioning	m	$1,200	$3,800
5.5 m x 4.4 m Conveyor Drive	m	$2,600	$5,000
7.5 m x 7.5 m Return Air Drive	m	$3,400	$6,500
Rehabilitation	m	$1,200	$3,500
Internal vent raise (2.5 m diameter)	m	$5,500	$8,000

[(1)] This item refers to the majority of underground excavations, such as the perimeter drifts, the MTT Access and the internal ramps.

11.2.2 Block Cave Infrastructure

The block cave infrastructure includes the cost of the ongoing activity inside a drift including preconditioning the rockmass, drilling and blasting the undercut, and drilling, blasting and supporting the drawpoints and drawbells. Cost estimates for the designs are shown in Table 30. These costs do not include labour.

Table 30: Summary of the Rock Infrastructure Capital Costs

Item	Unit	Cost ($)
Preconditioning	$/m of PC drift	$ 8
Undercut Blasting	$/m of UC drift	$ 1,250
Drawbell Excavation and Drawpoint Support	$ per set of drawpoints (2 drawpoints)	$ 78,000

11.2.3 Mobile Equipment

Table 31 shows the list and unit cost of the mobile equipment required for the Iron Cap mine. A five year replacement schedule has been included in the life of mine capital cost estimates. It is estimated that over the 27 year mine life a total of $38 million will be required for development equipment, $73 million for production equipment, and $14 million for support equipment.

Table 31: Iron Cap Mobile Equipment Requirements.

Equipment	Unit Cost
Jumbo Drill Rig	$ 986,000
Development Haul Truck	$ 948,000
Development LHD	$ 1,150,000
Bolter	$ 800,000
ANFO Loader	$ 400,000
Scissor Lift	$ 382,000
Production LHD	$ 1,150,000
Raisebore Machine	$ 4,100,000
Production Drill Rig	$ 997,000
Grader	$ 235,000
Big Personnel Carrier	$ 295,000
Small Personnel Carrier	$ 145,000
Mobile Rockbreaker	$ 224,000
Block Holer	$ 577,000
Shotcrete Sprayer	$ 627,000
Concrete Mixer	$ 442,000
Boom Truck	$ 329,000

11.2.4 Stationary Equipment

Table 32 shows the list of unit and installation costs of the major stationary equipment required for the Iron Cap mine. For cost estimating purposes the replacement/refit of the conveyors and crushers is done every ten years. It is estimated that Iron Cap will require $105M of stationary equipment over the life of the mine.

Table 32: Iron Cap Stationary Equipment and Installation Costs

Equipment	Quantity	Total Cost ($M)
Crusher	4	$3.5
Crusher Installation	4	$13.5
Conveyor (incl. installation)	1	$30.7
Transfer Station to MTT	1	$5.8
Dewatering System	1	$0.52
Air Compressor	2	$0.4

11.2.5 Surface Equipment

Table 33 shows the list of unit and installation costs of the surface equipment required for the Iron Cap mine. The electrical distribution system estimate was developed by WN Brazier Associates (details can be found in Appendix F).

Table 33: Iron Cap Surface Equipment Installed Costs

Equipment	Quantity	Total Installed Cost ($M)
Surface fans	4	$1.0
Mine air heaters	1	$5.0
Ventilation bulkheads	1	$1.6
Electrical distribution system	1	$45.0
Propane tank farm	1	$1.3
Portals	3	$0.3

11.2.6 Closure

The closure costs were included as a one-time $10 million dollar expense at the end of the mine life and it excludes any benefit that may be realised by selling stationary equipment (crushers, fans, conveyor belts), mobile equipment (LHDs, trucks or jumbos) or services (electrical wiring). The closure cost includes the following items:

- Remove the mobile equipment from the mine by driving it out and either salvaging it or placing it in a landfill or designated dump site.

- Leave the major infrastructure such as crusher, rockbreakers and conveyors (including belting) in the mine. All oils would be drained from the motors and gears.
- Leave all electrical cable and piping.
- Remove all extraneous oils and lubricants, such as those in electrical gear (transformers etc.), and any explosives and chemicals.
- Remove the surface ventilation fans and either salvage or dispose of them in landfill.
- Seal all openings to surface, except the Return Air Drift at Iron Cap, with cement plugs. These would contain drainpipes to allow water to drain for collection by the surface drainage and treatment systems.

The RAD at Iron Cap would be left open to provide a water outflow for a large flood event. This water would be captured with the existing surface drainage plans. The adit would be covered with a secure grate to prevent access.

11.2.7 Life of Mine Capital Cost Schedule

Capital costs include the purchase and installation of all equipment and the excavation of all the underground workings. Figure 45 shows the life of mine capital cost for the Iron Cap mine, which is estimated to be $1.5 billion. This includes approximately $509 million in pre-production capital expense over the first 5 years of the mine life and an average sustaining capital cost of $46 million over the remaining 21 years. The life of mine capital cost is shown in a table in Appendix G.

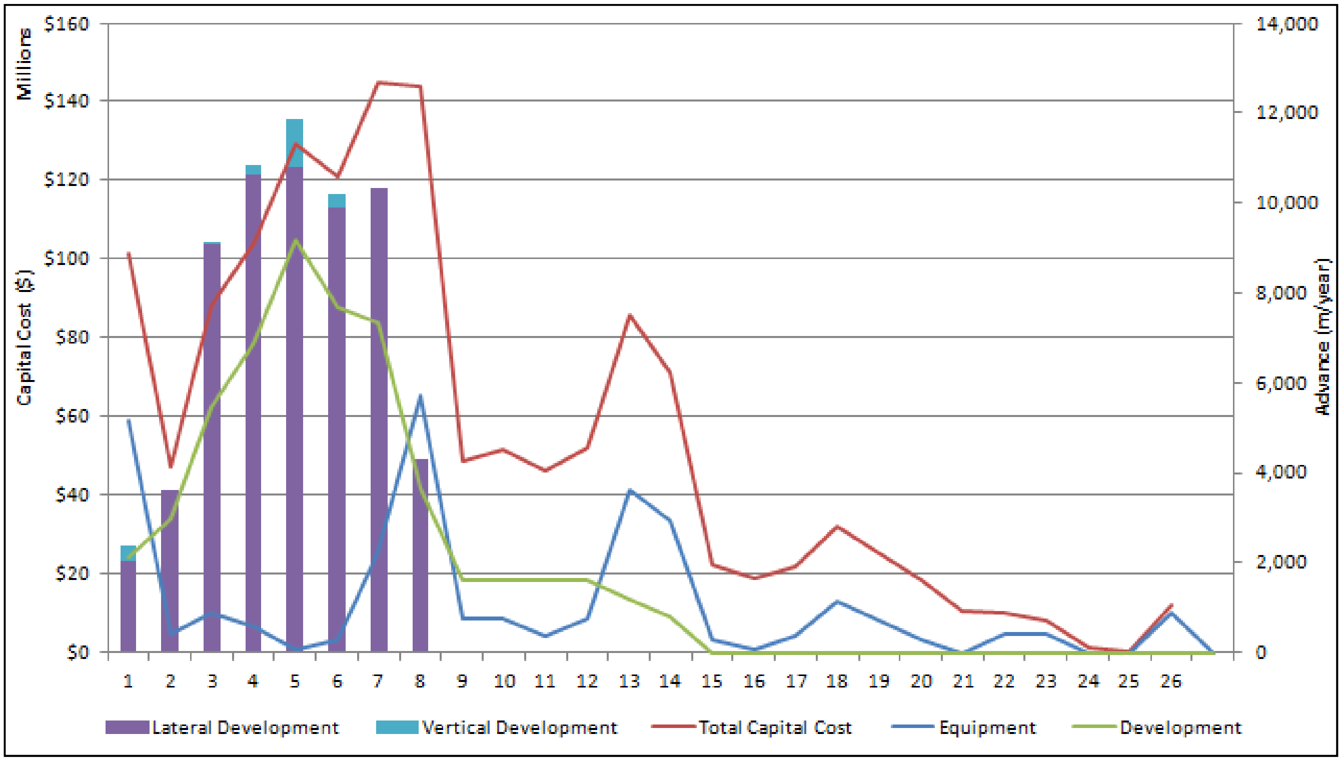


Figure 45: Distribution of capital cost of the life of the Iron Cap mine

11.3 Mine Operating Costs

A preliminary cost estimate of $6/tonne was used in the PCBC analysis to produce a tonne and grade production schedule for Iron Cap. This schedule was then input to the whole KSM complex production schedule developed by others. The mine operating cost presented in this section differs slightly from the preliminary one used because additional refinements were made. However, the difference did not warrant additional modelling to develop revised schedules.

The mine operating cost (OPEX) consists of the equipment and labour that is required to move material from the drawpoint to the MTT conveyor tunnel and the fixed costs to run the mine. This includes the use of the LHDs, secondary breakers, crushers and conveyors, and the labour required to plan and execute the mining plan (mine labour comprises about 56% of the total Iron Cap mine OPEX). Included in the fixed costs are items that are not affected by the production rate, such as the ventilation fans, pumps and the general mine expenses such as office supplies. Table 34 shows a breakdown of the average life of mine OPEX. Appendix H contains a detailed breakdown of the operating expense.

Table 34: A Summary of the Iron Cap Mine Operating Costs

Activity	OPEX ($/tonne mined)
Production Mucking (LHD)	$ 0.82
Crushing	$ 0.65
Conveyors	$ 0.39
Block Holers	$ 0.16
Mobile Rockbreakers	$ 0.12
Labour	$ 3.45
Rehabilitation	$ 0.04
Fixed	$ 0.53
Total	**$ 6.15**

Figure 46 shows the distribution of the unit OPEX over the life of the mine. The unit OPEX is higher in the first years due to the relatively high number of personnel on site producing at a comparatively small production rate. This is typical for an underground mine and even more applicable to block caving because of the high amount of development required before production can commence and the long ramp-up period to achieve the planned production rate. The influence of the workforce on the OPEX ranges from approximately 75% in the early years to approximately 35% in the later years. The impact of varying the labour cost by +/- 25% is shown in Table 35.

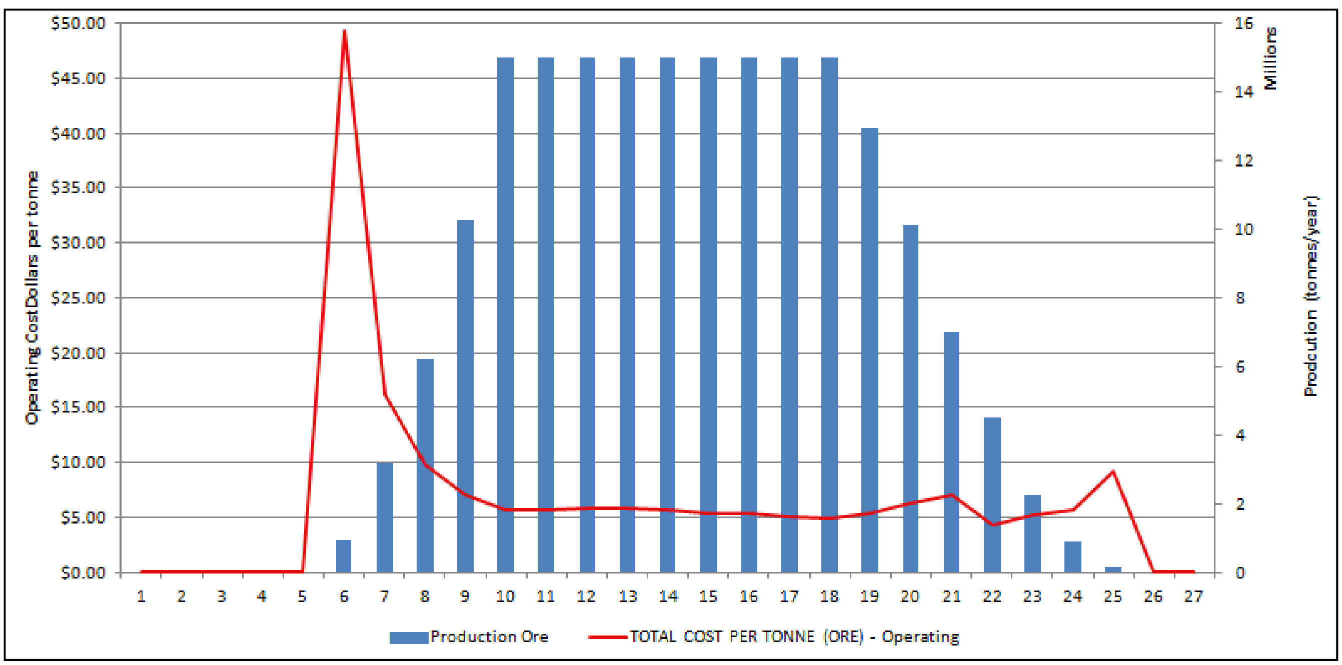


Figure 46: A chart showing the variability of the estimated OPEX over the life of the mine relative to the mine production

Table 35: Sensitivity of Labour Cost on OPEX

Item	Labour at -25%	Labour at Base Case	Labour at +25%
OPEX	5.46	5.86	6.23

11.3.1 OPEX Sensitivity

The mine OPEX is a key parameter used in PCBC to determine the profitability of a drawpoint. The influence of increasing or decreasing the OPEX by 25% on the block cave resources was investigated with additional PCBC runs, and the results are presented in Table 36. The block cave resources are not overly sensitive to OPEX as a 25% change to OPEX only changes the block cave resources by 5%.

Table 36: The Influence of Mine OPEX on Block Cave Resources

OPEX ($/tonne)	Tonnes	Au (g/t)	Cu (%)	Mo (ppm)	Ag (g/t)
6.00	197,300,448	0.448	0.194	21.465	5.258
4.50 (-25%)	207,351,744	0.437	0.191	21.792	5.178
7.50 (+25%)	187,015,520	0.459	0.196	21.168	5.331

11.4 Contingencies

Contingencies were applied to each cost item in the database and were calculated for the project based on a weighted average. The contingencies range from a low of 10% for fuel and power costs to a high of 25% for labour rates. A contingency of 20% was applied to the capital purchase of equipment and 15% to the maintenance cost of the equipment. Overall project contingency is estimated at 22%.

12.0 PROJECT OPPORTUNITIES AND RISKS

The following bullet points summarize the main opportunities and risks to block caving the Iron Cap deposit.

12.1 Project Opportunities

- A pre-feasibility study requires the use of Measured and Indicated resources only, which is approximately 207 million tonnes inside the Iron Cap footprint. There is approximately 142 million tonnes of Inferred material inside the footprint that was not included in this study (grades and tonnages are shown in Table 37). Upgrading this material to the Indicated category would improve the project economics. The results presented in the Table 37 are from the Footprint Finder program and differ from the Block Cave Resources presented in Table 4.

Table 37: Summary of the Footprint Finder Results

Categories	Tonnage (Mtonnes)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Measured and Indicated	207	0.47	0.20	5.63	21.85
Measured, Indicated, and Inferred	349	0.42	0.20	5.12	55.73

- The footprint determined by FF to provide the maximum value (elevation 1210 m) is shown in Figure 48. This will result in the extraction of approximately 50% of the geological resource. The opportunity is still available to establish other additional draw horizons at other elevations to potentially mine more of the resource. FF was used to investigate this further.

Figure 47 shows three additional polygons that might be candidates to recover additional resources that were investigated using FF.

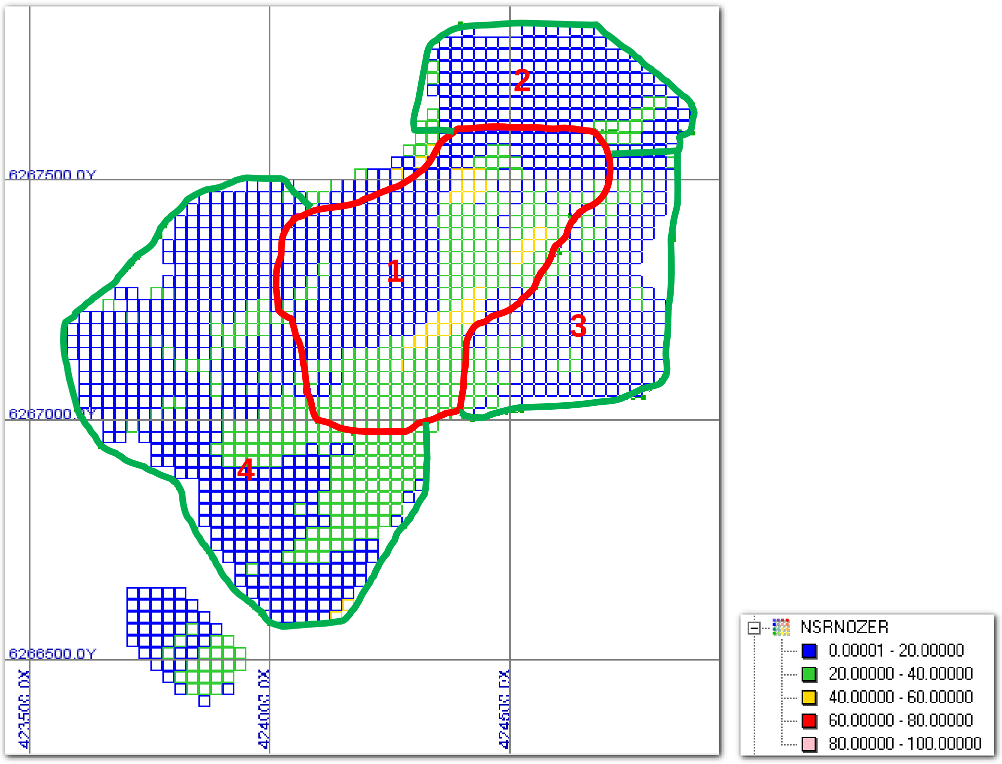


Figure 47: A section at 1210 m elevation of the Footprint Finder model showing the proposed block cave footprint, the selected footprint (1) and three possible additional footprints (2, 3 and 4) at other elevations

The results of the additional FF investigations are summarized in Table 38. These indicate that only Area 4 with a potential draw horizon at elevation 1315 m has potential to add value to the Iron Cap mine. The concept would be to excavate Areas 1 and 4 from two different elevations either concurrently or sequentially. There would be a number of sequencing issues that would need to be addressed in doing this. No investigations of mining this additional resource were carried out for this study.

Table 38: Summary of the Footprint Finder Results for Additional Mining Areas Outside of the Selected Footprint

Footprint Polygon	Elevation (m)	Tonnage (Mtonnes)	Value ($M)	Comment
1	1210	207	1,385	Selected case in this study
2	1285	3.9	14	Not viable -discarded
3	1360	26.7	70	Not viable - discarded
4	1315	95.2	365	Potential added value

- Iron Cap is one of the lower grade deposits at the KSM site and it is not scheduled to be exploited until later in the overall project life. This delay will provide time for the development of new technologies that may improve the Iron Cap economics and it will also allow time for investigations to add potential reserves (it is understood that the Iron Cap deposit is open at depth).

- Only limited geotechnical information is available on the quality and degree of fracturing of the rock mass. If further drilling information indicates that the degree of fracturing is less than presently assumed and the rock is generally more competent, which some of the current drilling information suggests, it may be possible to expand the drawpoint layout from the current 15 m by 15 m.

12.2 Project Risks

- Only very limited geotechnical drilling has been undertaken of the deposit. This drilling was focused towards obtaining information for the design of the open pits and the locations of the holes were not optimum for the design of a caving operation. The available geotechnical information, together with a good understanding of the geological conditions and knowledge from the adjacent Mitchell deposit, are considered suitable for this pre-feasibility assessment, but additional information will be required to advance the project to a higher level of study.

- The fragmentation of the rock at the drawpoints is estimated to be coarse to very coarse (the in situ block size is >2 m^3). The mine has been designed to accommodate the predicted fragmentation size. However, the predictions may not be accurate. Experience at Palabora indicates that the ability to properly predict primary, secondary and tertiary fragmentation of the rockmass during the caving process is difficult. If the size of the average drawpoint material at Iron Cap is larger than expected, then production will suffer.

- There is an ice cap on the mountain that contains the Iron Cap deposit. The mine may be operating far enough into the future that the ice will have already melted and will have little impact on the block cave mine. If such melting has not occurred, caving of the ice into the crater is not expected to pose a problem. However if there is water beneath the ice, that is presently not thought likely, then sudden inrushes may occur. The possibility of such water being present needs to be investigated further.

- Production at Iron Cap will be mucked with LHDs hauling directly to the crushers. A literature search on block cave mines using similar approaches indicates that the maximum average haul distance is approximately 250 m. This is the average haul distance at Iron Cap. There are many impacts that could potentially delay an LHD and reduce its productivity (e.g., other equipment in the drift, large hang-ups, and delays at the crusher). If the LHDs don not perform as well as anticipated, then production targets may not be reached and costs will increase.

- The LHD haul and fragmentation estimation are key factors influencing productivity. It is also recommended that additional study on the productivity of the LHDs and extraction level design be undertaken to confirm the estimated productivities. This additional work should include industry benchmarking and the use of mine simulation software tools.

- The current ventilation design has the return air portal located downwind of the fresh air portals. However, 30% of the time, the wind is blowing in the opposite direction. If the exhaust air is not properly dispersed by the time it reaches the fresh air intakes then a recirculation problem may occur.

- The Iron Cap deposit is estimated to have approximately 100 workers per shift at peak operations and it will be producing 45,000 tpd of material. This will add to the traffic that is already planned for the MTT. A review of the tunnel logistics including traffic and material flow and handling “clean” and “dirty” water flows should be completed.

- There are inherent risks associated with block caving. These include the following:
  - There is the potential to develop air gaps between the cave back and the caved material. These air gaps may result in air blasts which could cause significant damage to the underground infrastructure and hazards to employees underground. There are mitigation measures that can be implemented including a microseismic system and boreholes to measure the progression of the cave.
  - Mud-rushes at the drawpoints are a risk once the cave breaks through to the surface. This is particularly important during the annual spring thaw, when meltwater from snow and ice that accumulates on the broken material at the surface can migrate through to the underground drawpoints. The current drawpoint design has attempted to mitigate some of this risk by offsetting the drawpoints. Operational restrictions may need to be applied at certain times of the year which may adversely impact production.
  - The nature of a block cave causes stresses to be redistributed. This may result in damage to the existing drifts as a result of stress induced fracturing and/or major seismic energy releases. Further investigations of this need to be undertaken.
  - Excessive rehabilitation of the drifts from adverse stress conditions may have a significant impact on the profitability of the mine. In addition, it is anticipated that there will be a significant amount of secondary blasting of oversize material, which could damage drift infrastructure (piping and wiring in the drift). Rehabilitating these drifts will be expensive, not only because of the cost to complete the repairs, but because of the potential loss of productivity. The current design attempts to mitigate these risks by assigning a rehabilitation cost (10% of development) to the OPEX and by having approximately twice as many drawpoints available as needed to meet production targets.

13.0 CONCLUSIONS

The Iron Cap deposit is a large, massive deposit making it suitable for block caving. Analyses using FF and PCBC indicate that an economical block cave operation can be developed with a caving footprint approximately 243,000 m^2 in size.

The deposit is comprised of strong, moderately fractured rock. The strength of the rock mass does not vary significantly within the deposit, and rock quality variations are most commonly attributed to variations in fracture frequency.

The cavability assessments made using Laubscher's and Mathews' methods indicate that the size (diameter) of the footprint required to initiate and propagate caving is significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the large size of the deposit, indicates that the Iron Cap deposit is amenable to cave mining.

The DFN modelling shows that the median in situ block size is approximately 2.5 m^3. This is a large block size for cave mining, and the fragmentation is expected to be coarse. Mitigation measures have been incorporated into the design including secondary rock breakers and the LHDs hauling directly to the crushers. As well, preconditioning of the rockmass by hydraulic fracturing is proposed as a risk reduction measure. The potentially coarse fragmentation has been taken into account in developing the mine design, estimating the production rate over time and estimating operating costs.

The available information on fracture intensity at Iron Cap is somewhat uncertain because of the limited geotechnical drilling that has been undertaken, and it was considered prudent for this initial study to adopt the slightly conservative drawpoint layout of 15 m by 15 m. This aspect needs to be investigated further with further geotechnical drilling being undertaken. Depending on the results, there may be an opportunity in the future to adopt an expanded layout.

The design produces a block cave resource of approximately 192 million tonnes with an average grade of 5.3 g/t Ag, 0.45 g/t Au, 0.20% Cu and 21.6 ppm Mo. The mining operation will require approximately 62 km of openings to be excavated.

Detailed production, development, and capital and operating cost schedules have been developed for the project based on the pre-feasibility level design presented. The costs were developed from first principles and vendor quotes, and are considered accurate to +/-25%.

The mining operating cost was estimated to be $6.15 per tonne and the total mining cost including a 22% contingency was estimated to be $13.41 per tonne.

The pre-production capital expense is approximately $879M and the yearly capital expense is $27.5M per year for the remaining 19 years of mine life. The total mine life is expected to be 25 years, including 5 years of pre-production development, 4 years of production ramp-up, and 7 years of production ramp-down.

14.0 CLOSURE

We trust that this report meets your requirements at this time. If you have any questions or comments please contact the under-signed.

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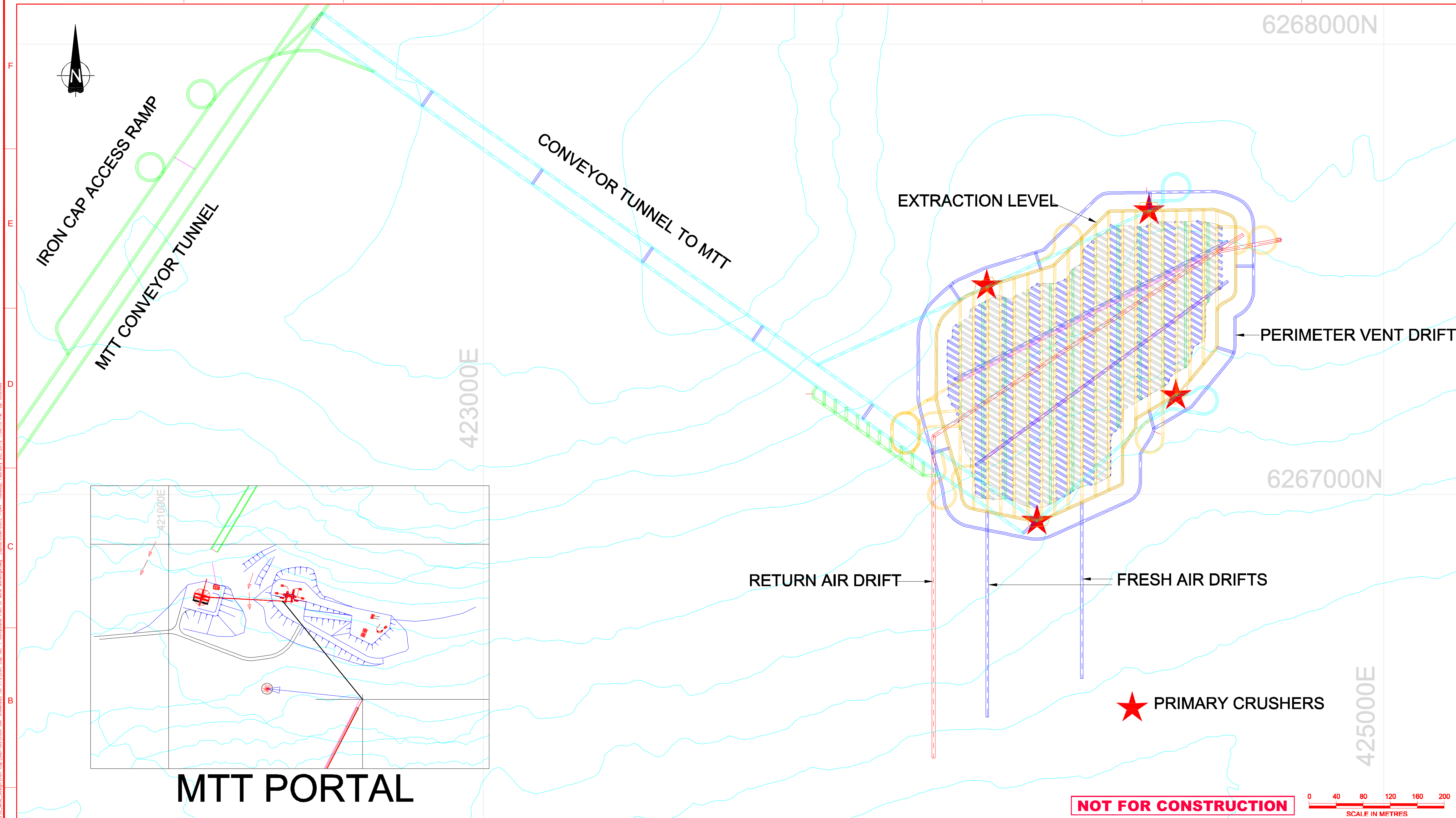
APPENDIX A

Mine Level Drawings



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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	IRON CAP COMPOSITE PLAN OF UNDERGROUND WORKINGS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE

IRON CAP COMPOSITE PLAN

PROJECT No.	PHASE No.	TASK No.
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ORIGINAL SHEET SIZE	DRAWING No.	REVISION
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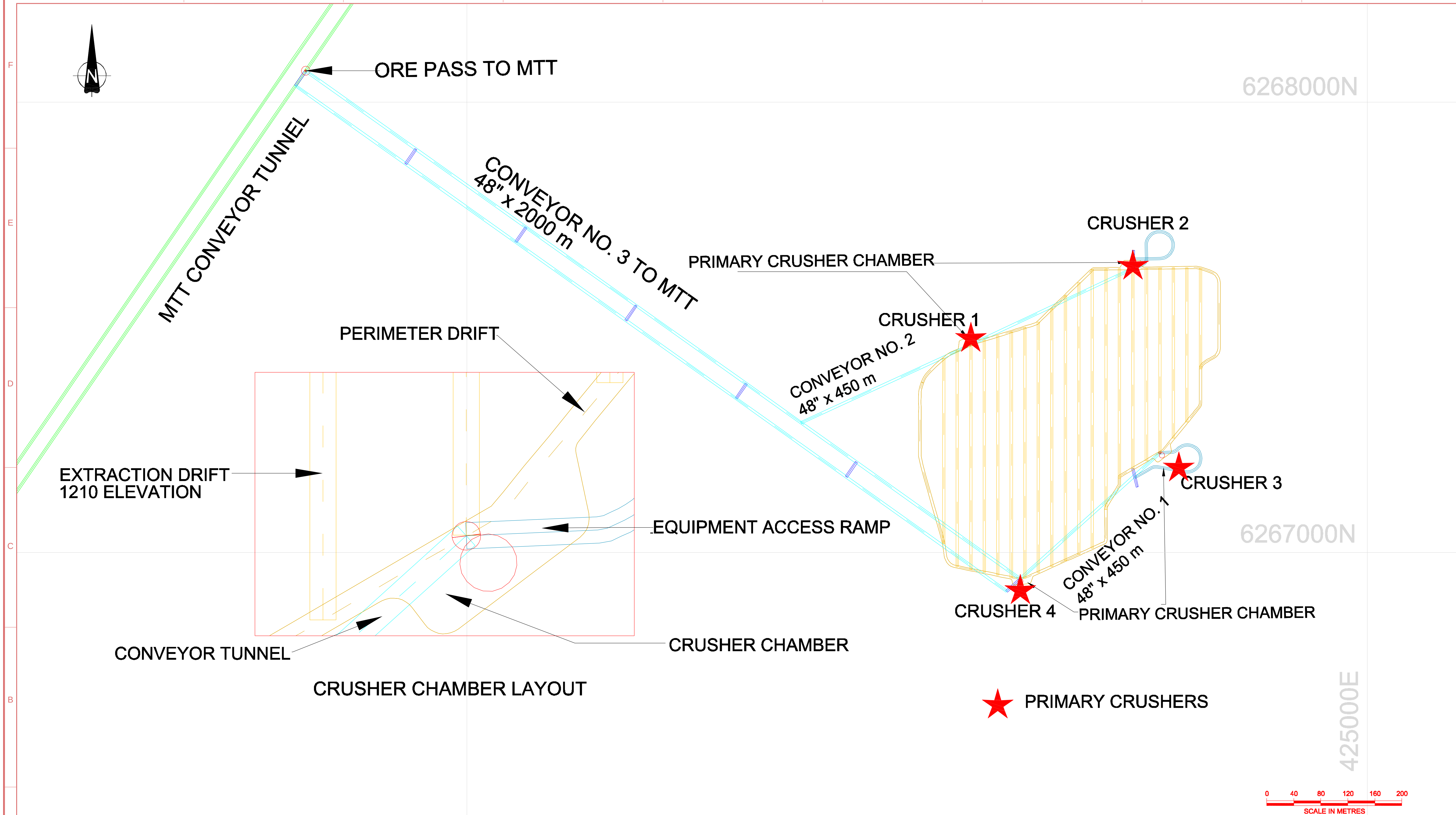

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	ORE CONVEYING AND CRUSHING	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

DRAWING TITLE: ORE CONVEYING AND CRUSHING

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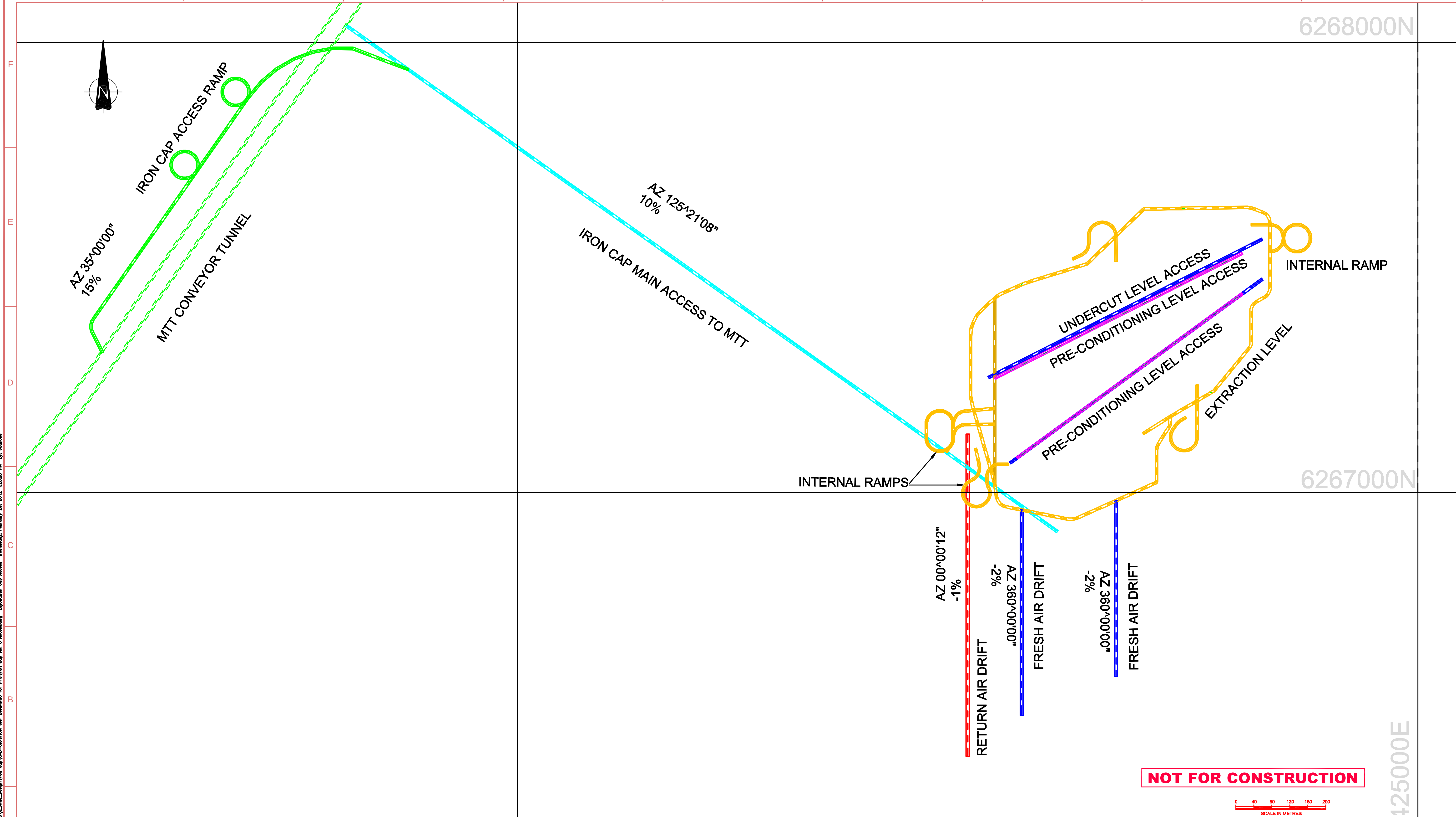

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	IRON CAP ACCESS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

DRAWING TITLE: **IRON CAP ACCESS**

PROJECT No.	PHASE No.	TASK No.
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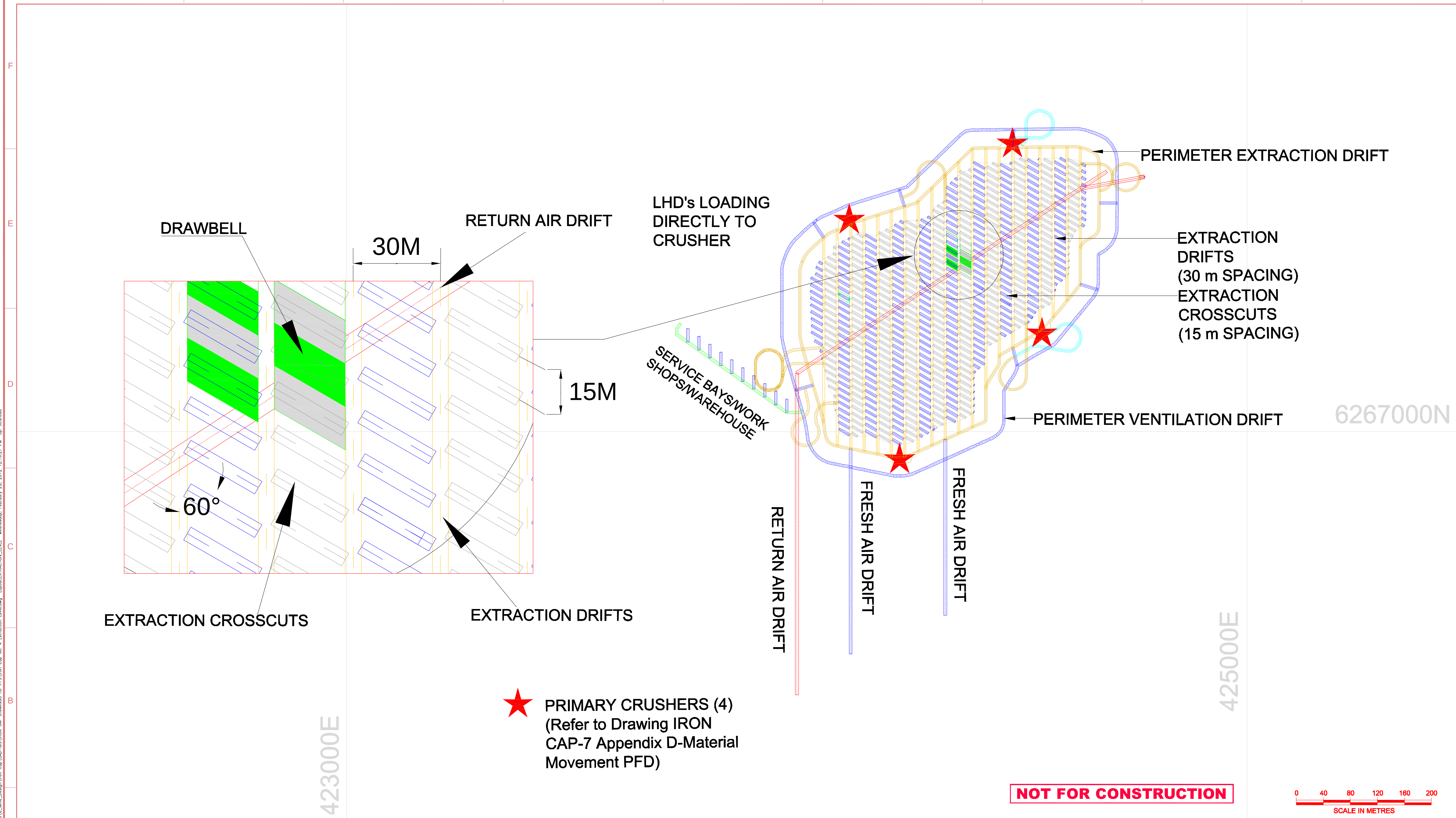

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	EXTRACTION LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD

CONSULTANT: Golder Associates

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PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

DRAWING TITLE: EXTRACTION LEVEL 1210 LEVEL

PROJECT No.	PHASE No.	TASK No.
1114390002	6000	NA

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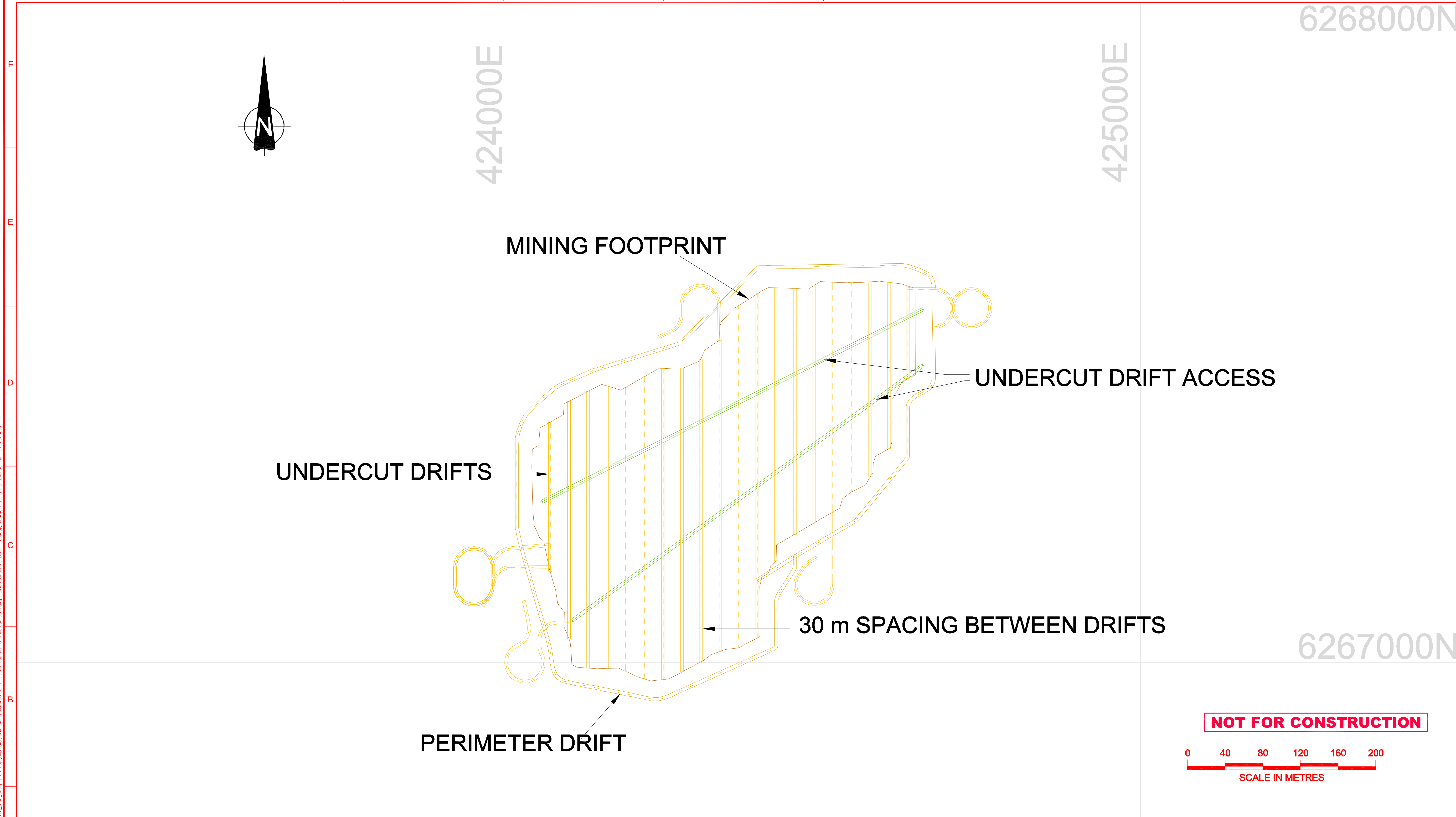

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	UNDERCUT LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT
SEABRIDGE GOLD INC

PROJECT
IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT
Golder Associates

DRAWING TITLE
UNDERCUT LEVEL
1230 LEVEL

PROJECT No.	PHASE No.	TASK No.
1114390002	6000	NA

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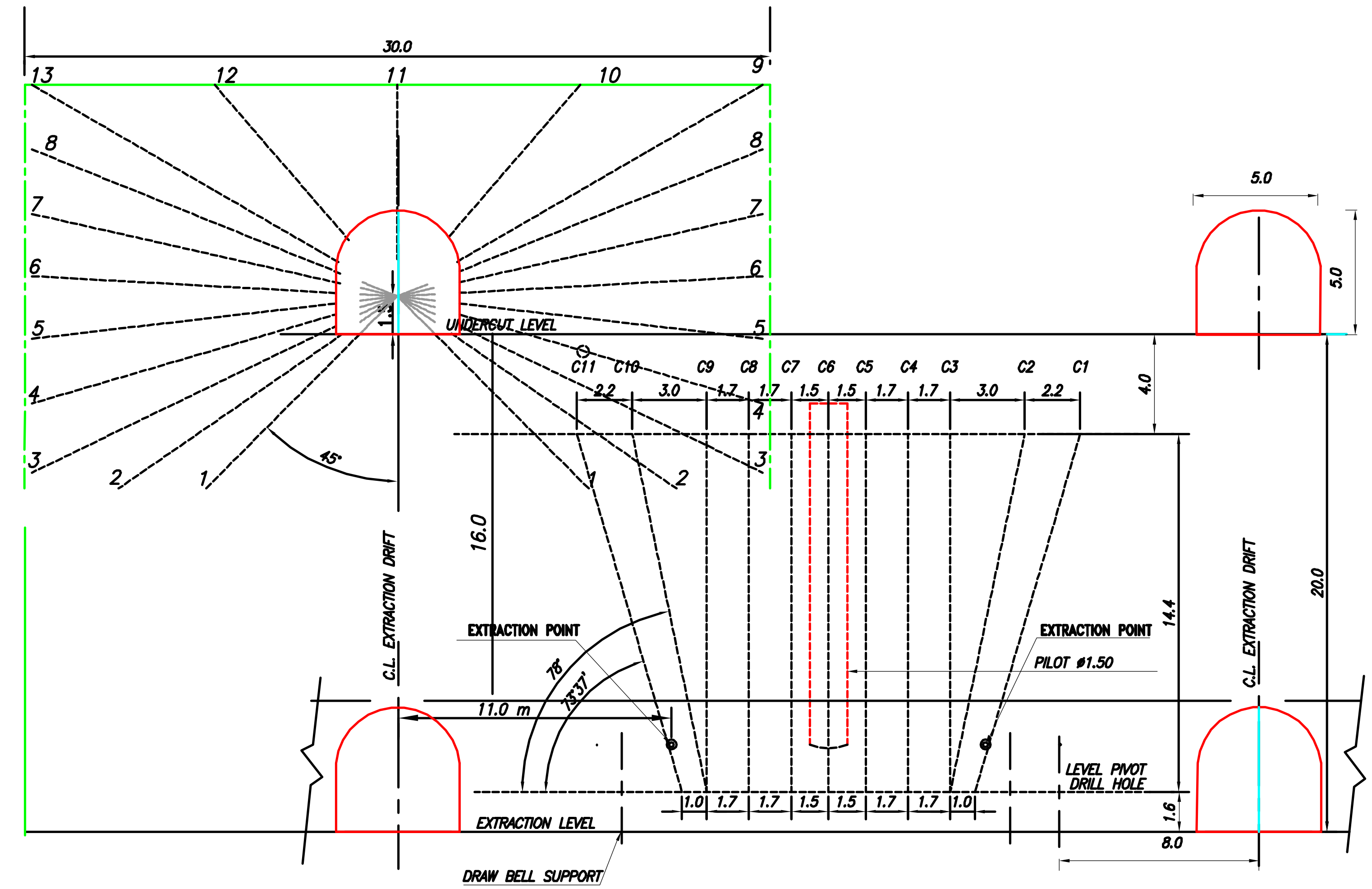

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	DRILLING DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

DRAWING TITLE: TYPICAL UNDERCUT AND DRAWBELL DRILLING LAYOUTS (15 m x 15 m PILLAR)

PROJECT No.: 1114390002 | PHASE No.: 6000 | TASK No.: NA

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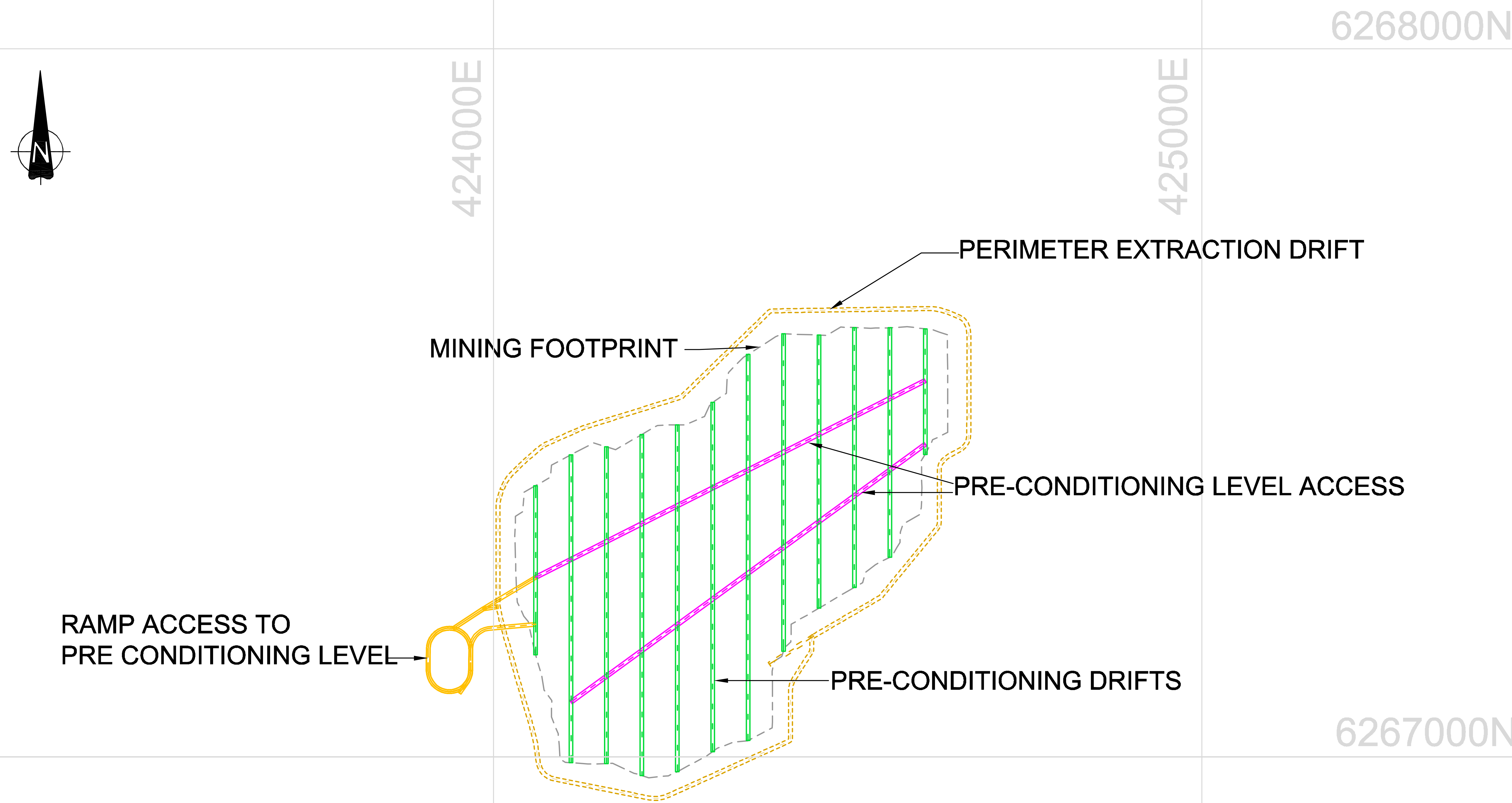

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	IRON CAP ACCESS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: **PRE-CONDITIONING LEVEL**
1290 LEVEL

PROJECT No. 1114390002 | PHASE No. 6000 | TASK No. NA

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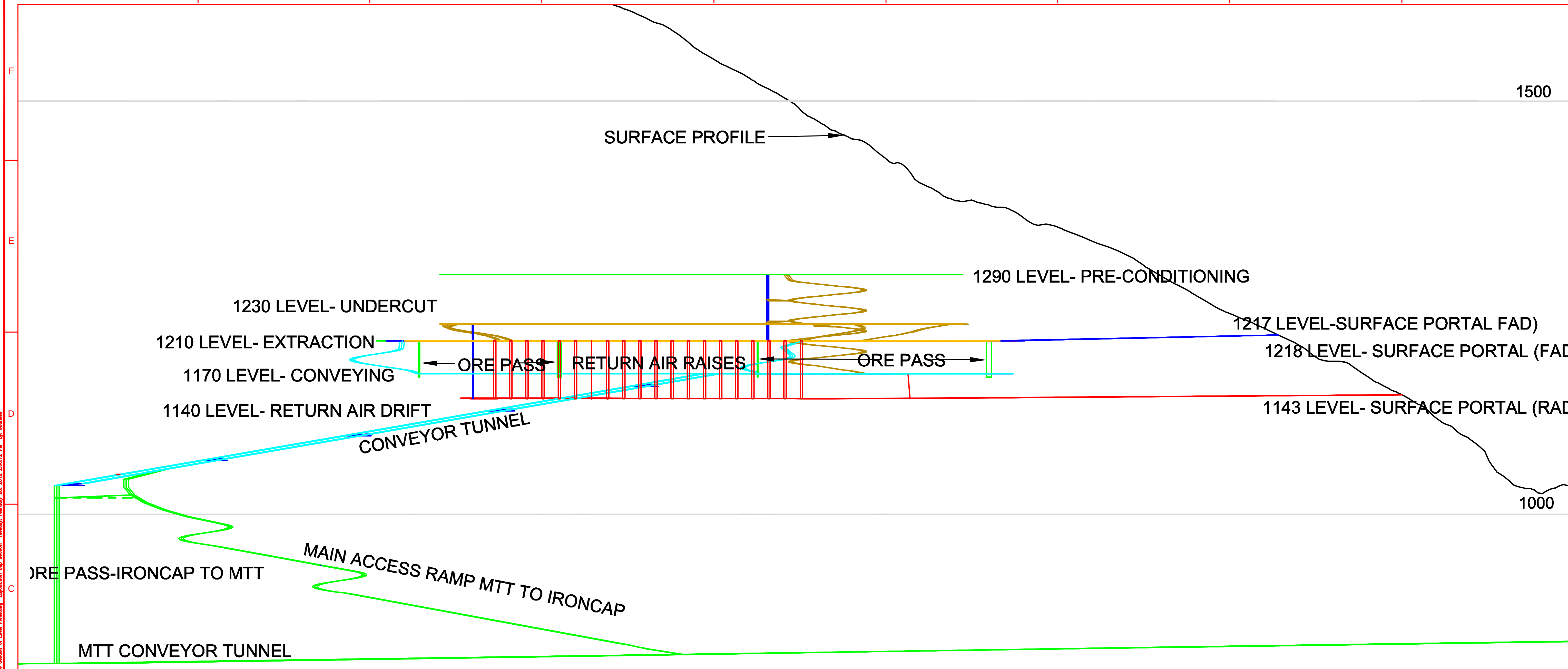

NOTE: LEVEL ELEVATION DETERMINED BY LEVEL NAME
EXAMPLE: 1140 LEVEL- RETURN AIR DRIFT = 1140 METER ELEVATION

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	IRON CAP COMPOSITE SECTION UNDERGROUND WORKINGS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: **IRON CAP COMPOSITE SECTION**

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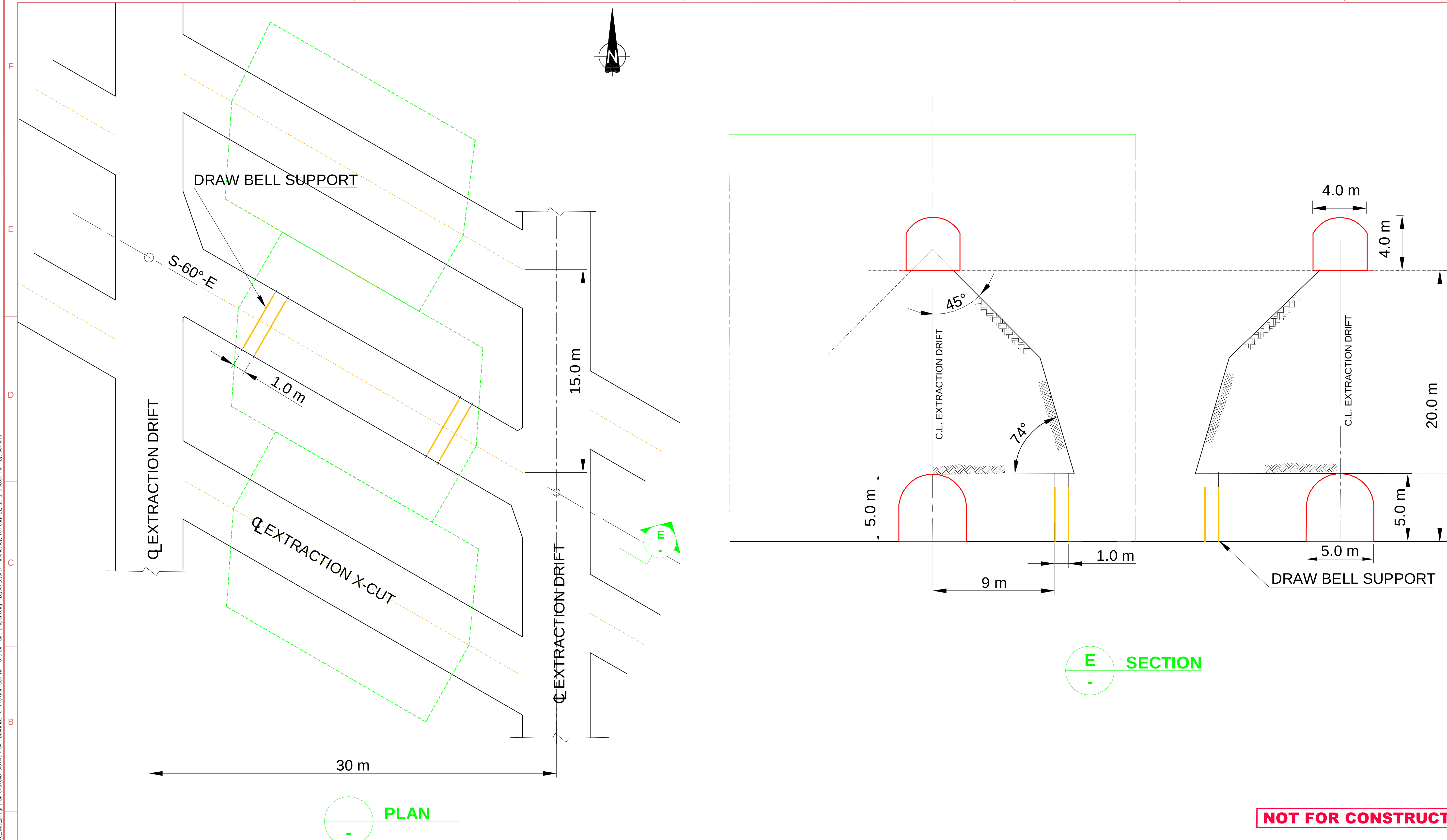

PLAN

SECTION

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	DRAWPOINT DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: **DRAWPOINT PLAN AND SECTION**

PROJECT No.	PHASE No.	TASK No.
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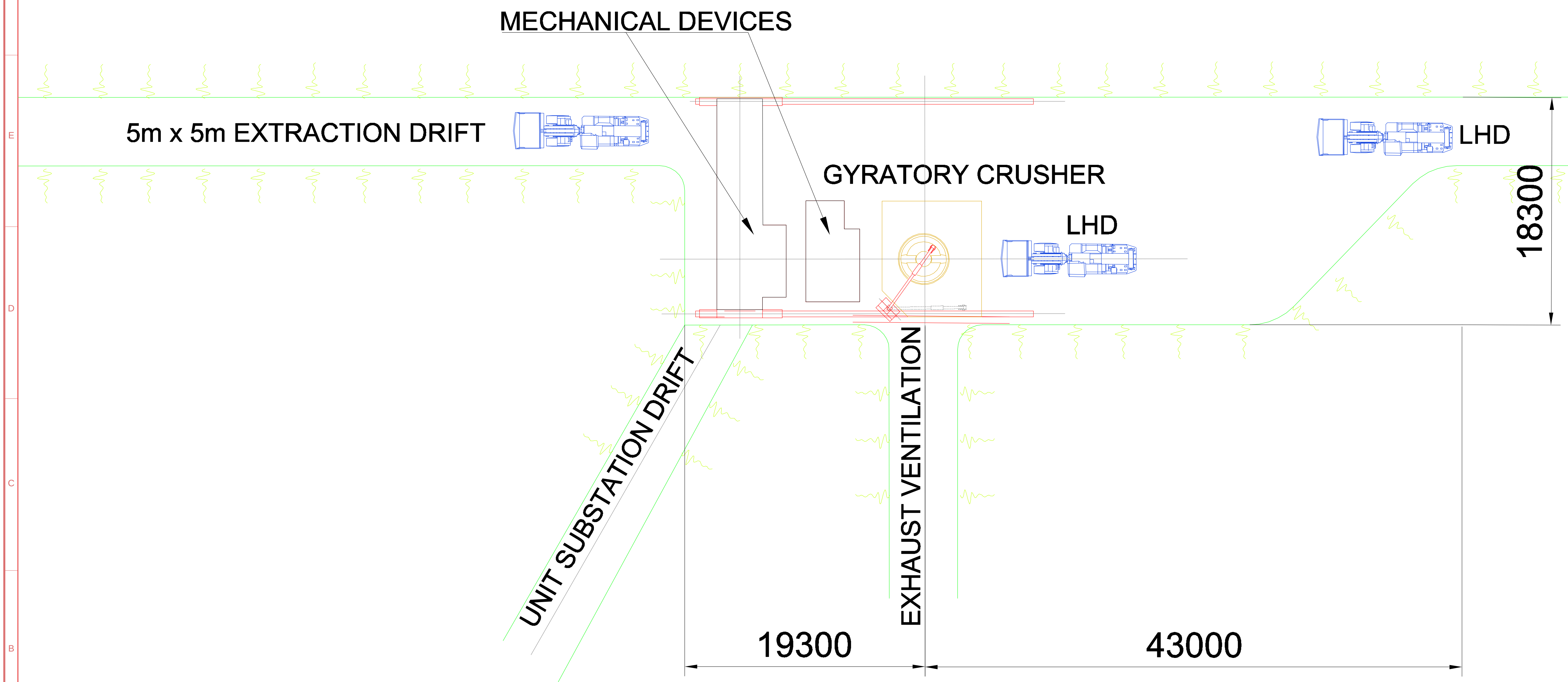

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	ORE CRUSHING	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

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DRAWING TITLE: **TYPICAL IRON CAP CRUSHER-PLAN VIEW**

PROJECT No.	PHASE No.	TASK No.
1114390002	6000	NA

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
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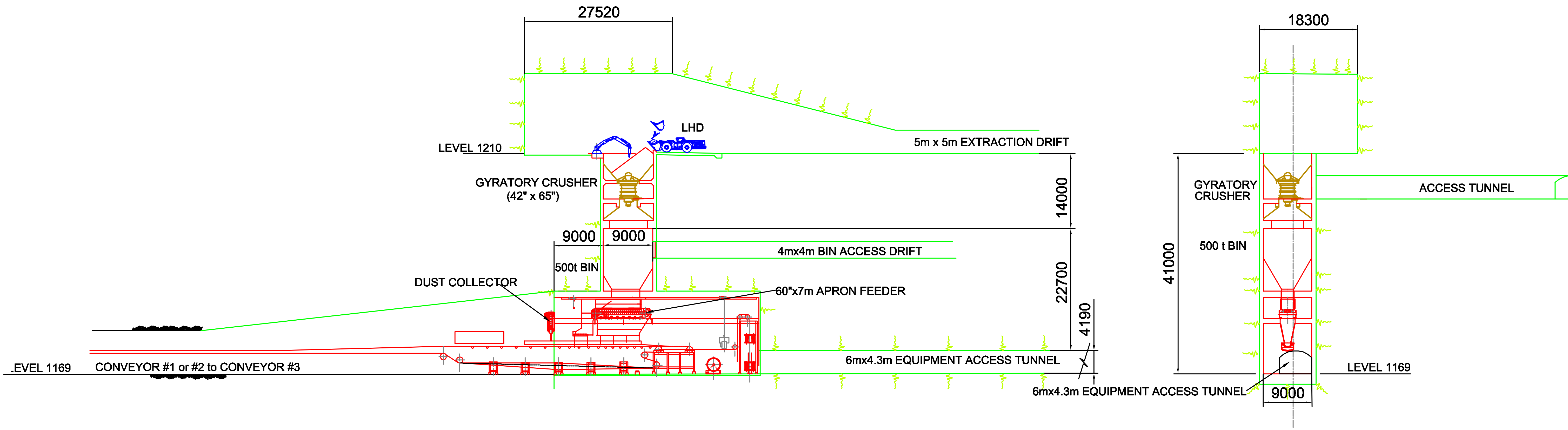

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	ORE CRUSHING	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: IRON CAP CRUSHER SECTION

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PROJECT No.	PHASE No.	TASK No.
1114390002	6000	NA

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	IRON CAP-12	1

APPENDIX B

Block Cave Definitions

TECHNICAL MEMORANDUM

DATE February 20, 2012 **PROJECT No.** 11-1439-0002

TO File

FROM Donald Tolfree **EMAIL** dtolfree@golder.ca

BLOCK CAVING DEFINITION

Block cave mining is a low cost bulk underground mining method in which the block of ore to be mined is undercut by drilling and blasting, and some of the blasted material is progressively removed to create a void. This causes the rock mass above the undercut to fail, and the failed material displaces and dilates into the void created by the undercut. Drawbells excavated beneath the undercut are used to extract the broken ore, precipitating further failure of the intact rock, and displacement and dilation of the ore. Continued extraction of the ore over a sufficiently large area allows the failure of the rock mass to propagate upward to ground surface as a block cave. The vast majority of the ore block is not directly accessed or fragmented by drilling and blasting, making this a low cost bulk mining method.

The three main horizons in a block cave mine are the undercut level, the extraction or production level, and the haulage level. A fourth level, the "pre-conditioning" level, may also be developed if geotechnical assessments indicate that the natural cavability of the mineralized material will produce material at the drawpoints that is too large to handle. Typically, this level is located above the undercut. Figure 1 is a schematic that shows the relationship between the different underground horizons used in a block cave mine. Some common block caving terms that will be used throughout the report are:

- Drawcone – theoretical zone of influence of one drawbell inside the caved material;
- Drawbell – the blasted area between the undercut level and the extraction level. The drawbell guides the broken ore to the individual drawpoints; and
- Drawpoint – the drawpoint is located in an extraction drift and provides access to the caved material to allow for removal with mechanised equipment.

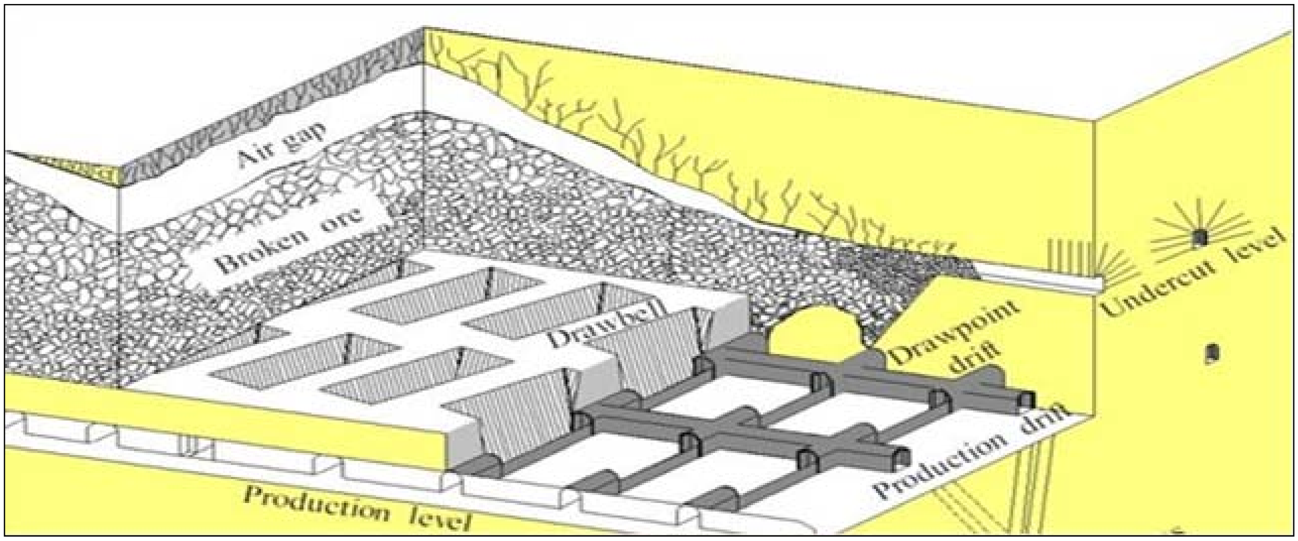

Figure 1: A schematic showing the relationship between the extraction drift (production drift), the drawpoints, drawbells and the undercut level. (Flores, 2004).

The use of the term "block cave" in this study is a generic term for the mining method described above. There are variations within block caving, such as panel caving. Block caving is used to refer to a mining method where all the drawbells are blasted within a relatively short time period relative to the mine life. The material is then extracted from all the drawbells to draw the cave down evenly over the entire footprint. Panel caving is applied when a strip or panel of drawbells is developed and ore is produced from these drawbells. As this producing unit is drawn down, another producing unit is brought into production and the earlier drawbells are closed. Panel caving is normally applied where there is a large available footprint, which is the case for the Mitchell deposit.

GOLDER ASSOCIATES LTD.

Donald Tolfree, P.Eng.
Mine Engineer

Dave Sprott, P.Eng.
Associate, Senior Mine Engineer

DT/DS/aw

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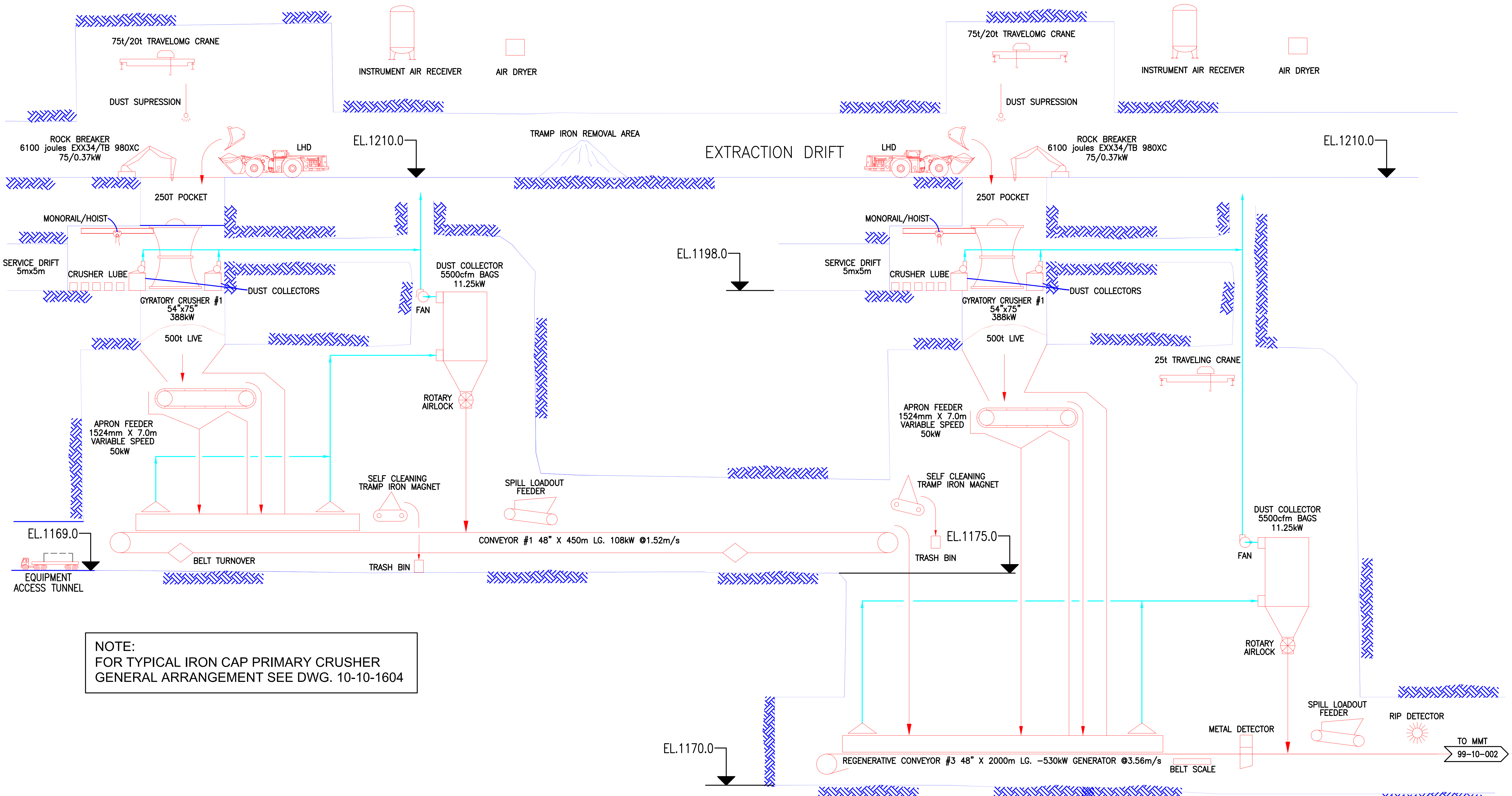
APPENDIX C

Material Movement Process Flow Diagrams



NOTE:
FOR TYPICAL IRON CAP PRIMARY CRUSHER
GENERAL ARRANGEMENT SEE DWG. 10-10-1604

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C	ISSUED FOR COST ESTIMATE		BW		HB	2011/11/30

REFERENCE DRAWING	REF. DWG No.

THIS DRAWING IS THE CONFIDENTIAL PROPERTY OF SEABRIDGE GOLD INC. AND SHALL NOT BE DUPLICATED WITHOUT PERMISSION.

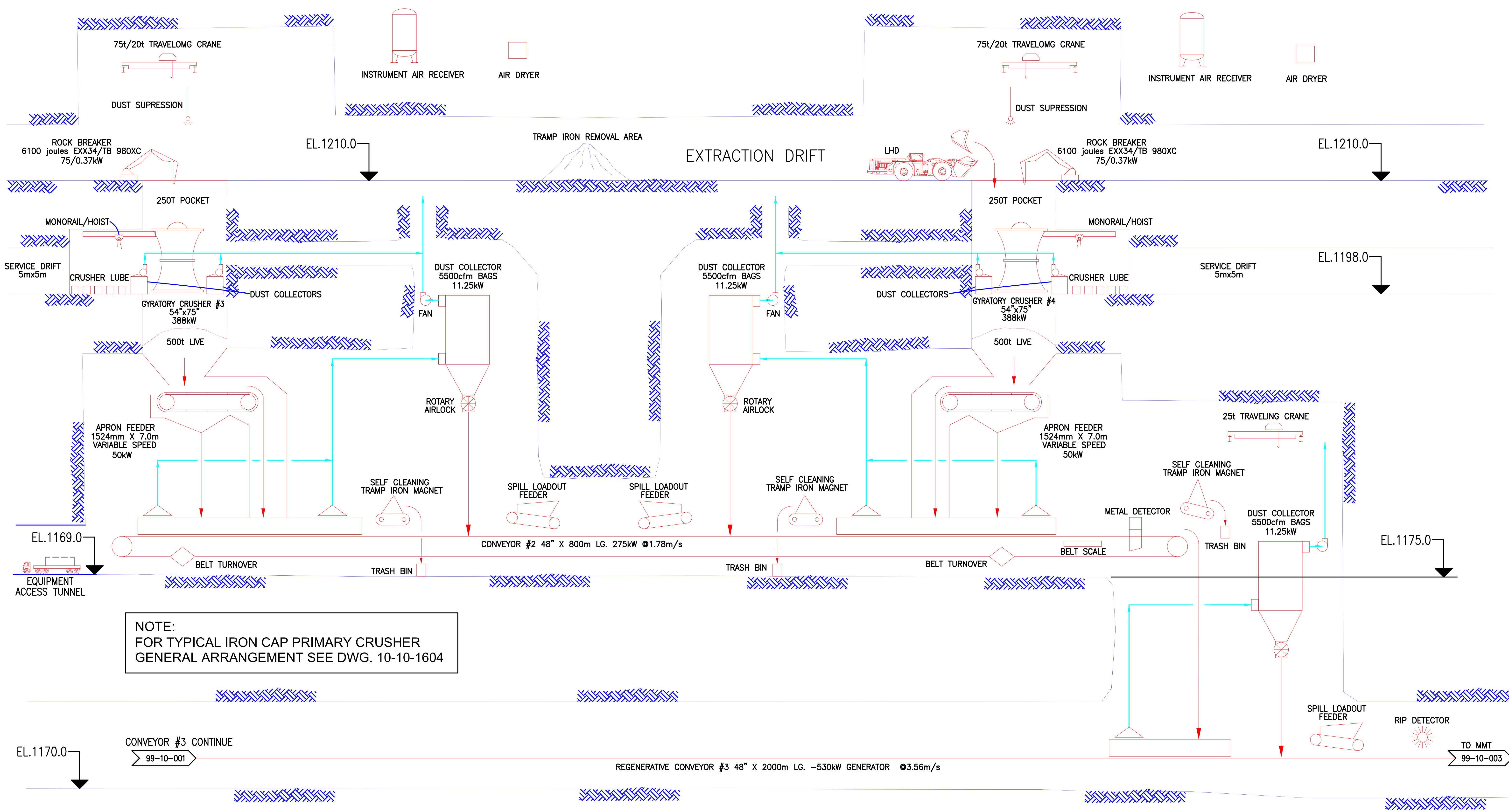
		DATE:
SCALE:	NONE	
DESIGNED:	H. Bosche	Nov 25 2011
DRAWN:	Bill Wong	Nov 25 2011
CHECKED:		
APPROVED:		

CLIENT: SEABRIDGE GOLD INC.

BOSCHE VENTURES LTD

TITLE: IRON CAP MUCK FLOW
4,5000t/d CRUSHING/CONVEYING
FLOWSHEET #1

PROJECT No.: | DRAWING No.: 99-10-001 | REV No.: C



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REFERENCE DRAWING	REF. DWG No.

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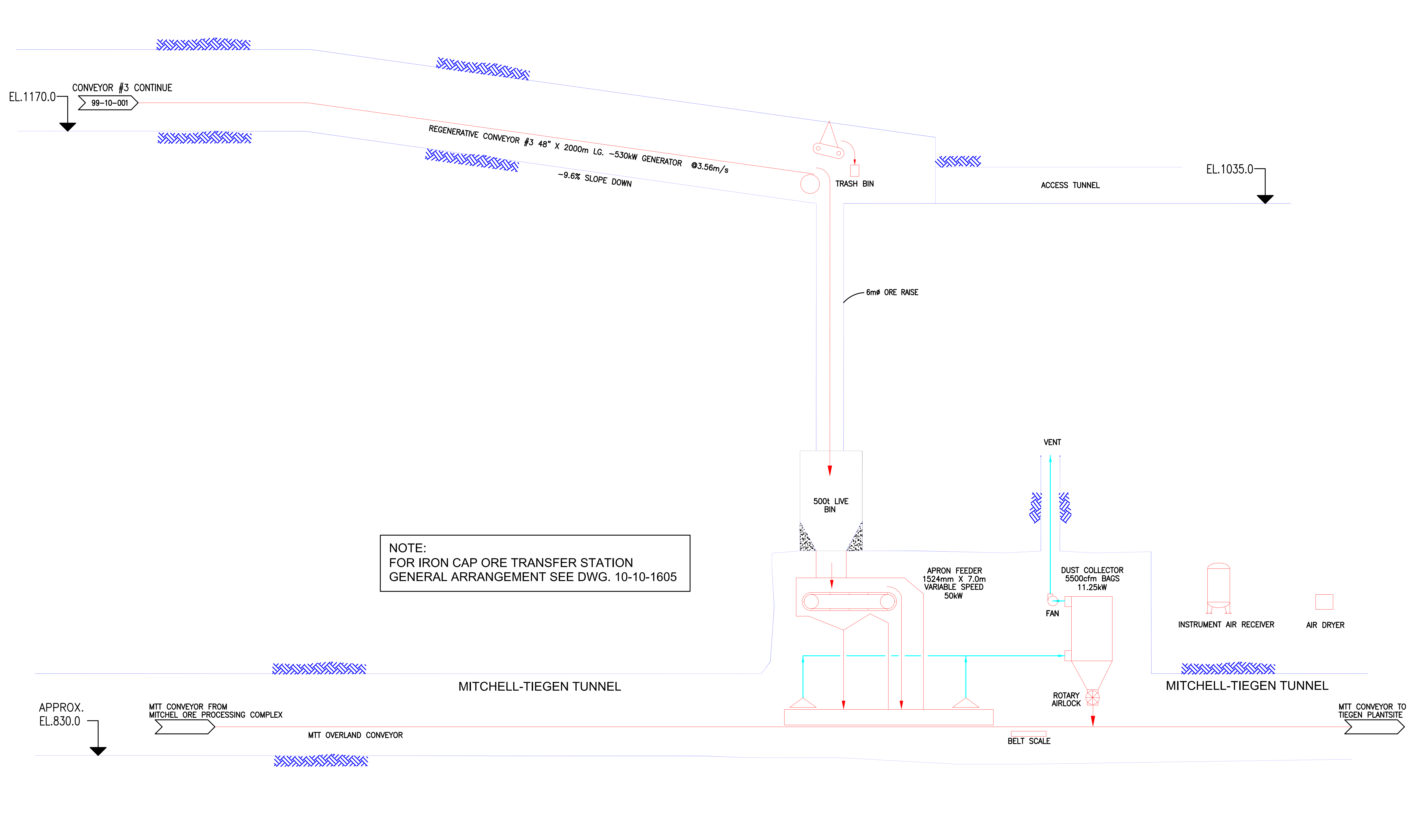
		DATE:
SCALE:	NONE	
DESIGNED:	H. Bosche	Nov 25 2011
DRAWN:	Bill Wong	Nov 25 2011
CHECKED:		
APPROVED:		

CLIENT: SEABRIDGE GOLD INC.

TITLE: IRON CAP MUCK FLOW 4,5000t/d CRUSHING/CONVEYING FLOWSHEET #2

PROJECT No. | DRAWING No. 99-10-002 | REV No. C



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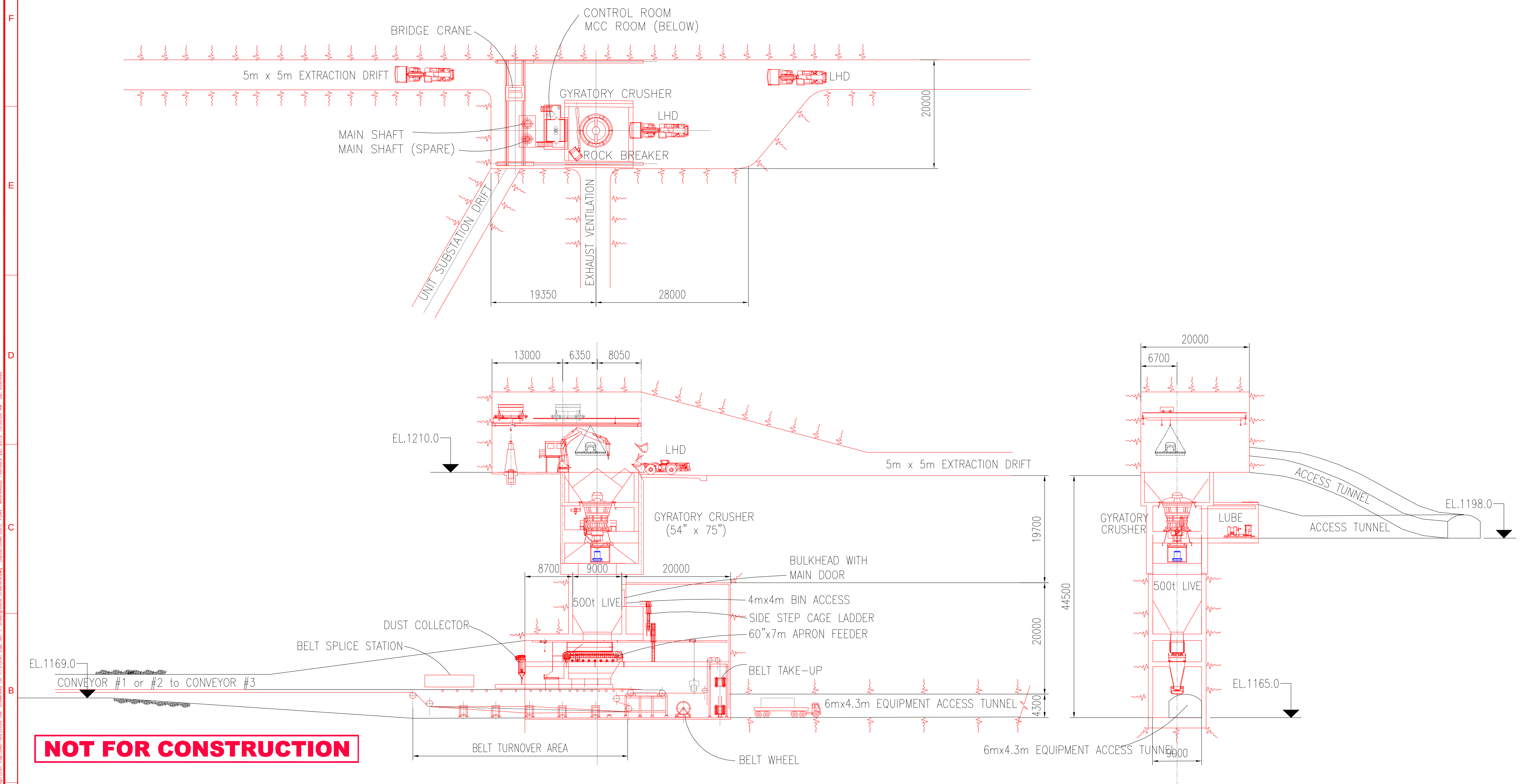
		DATE:
SCALE:	NONE	
DESIGNED:	H. Bosche	Nov 25 2011
DRAWN:	Bill Wong	Nov 25 2011
CHECKED:		
APPROVED:		

CLIENT: SEABRIDGE GOLD INC.

TITLE: IRON CAP MUCK FLOW
4,5000t/d CRUSHING/CONVEYING
FLOWSHEET #3

PROJECT No.: | DRAWING No.: 99-10-003 | REV No.: C



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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	IRON CAP CRUSHERS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

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DRAWING TITLE: **IRON CAP UNDERGROUND CRUSHERS**

PROJECT No.	PHASE No.	TASK No.
1114390002	6000	NA

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	IRON CAP-7	1

APPENDIX D

Ventilation - Airflow Calculations and Level Diagrams

Table D-1: 1290 Preconditioning Level Ventilation Requirements

Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m3/s)
Production Drill	2	74	79%	25%	2
Face Drill	1	120	80%	25%	2
Bolter	2	115	71%	25%	3
LHD	1	352	70%	75%	12
Truck	2	405	52%	75%	20
Anfo Loader	1	111	43%	75%	2
Scissorlift	1	95	83%	50%	2
Toyota	1	96	75%	75%	3
Subtotal					46
Contingency					20%
Total Required					**56**
Modelled Quantity (Excluding Leakage)					**56**
Modelled Leakage Through Cave					1

Table D-2: 1230 Undercut Level Ventilation Requirements

Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m3/s)
Production Drill	2	74	79%	25%	2
Face Drill	1	120	80%	25%	2
Bolter	2	115	71%	25%	3
LHD	1	352	70%	75%	12
Truck	3	405	52%	75%	30
Anfo Loader	1	111	43%	75%	2
Scissorlift	1	95	83%	50%	2
Crane Truck	1	95	63%	50%	2
Toyota	2	96	75%	75%	7
Subtotal					62
Contingency					20%
Total Required					**75**
Modelled Quantity (Excluding Leakage)					**75**
Modelled Leakage Through Cave					4

Table D-3: 1170 Crushing Level Ventilation Requirements

Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m3/s)
Face Drill	1	120	80%	25%	2
Bolter	2	115	71%	25%	3
LHD	1	352	70%	75%	12
Truck	3	405	52%	75%	30
Anfo Loader	1	111	43%	75%	2
Scissorlift	1	95	83%	50%	2
Crane Truck	1	95	63%	50%	2
Toyota	2	96	75%	75%	7
Subtotal					60
Contingency					20%
Total Required					**72**
Modelled Quantity (Excluding Leakage)					**72**
Modelled Leakage Through Cave					-

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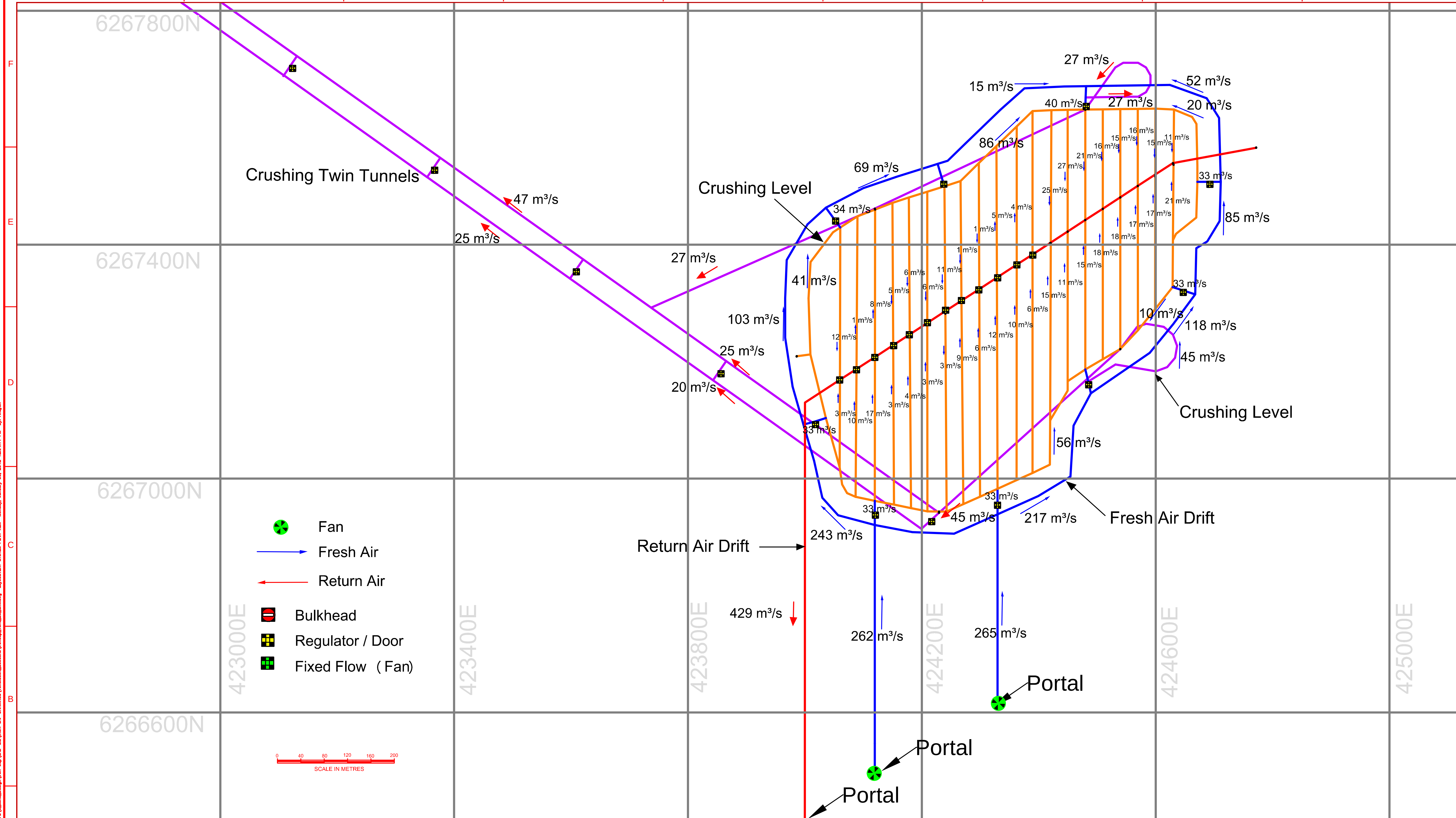

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SP-1001-01	VENTILATION SYSTEM	A
NUMBER	SUBJECT	REVISION
SPECIFICATIONS		

REFERENCE TITLE
REFERENCES

STAMP/SEAL

CLIENT
SEABRIDGE GOLD

PROJECT
IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT
Golder Associates

DRAWING TITLE
IRON CAP VENTILATION
EXTRACTION LEVEL
PLAN VIEW

PROJECT No.	PHASE No.	TASK No.
1114390002	7000	NA

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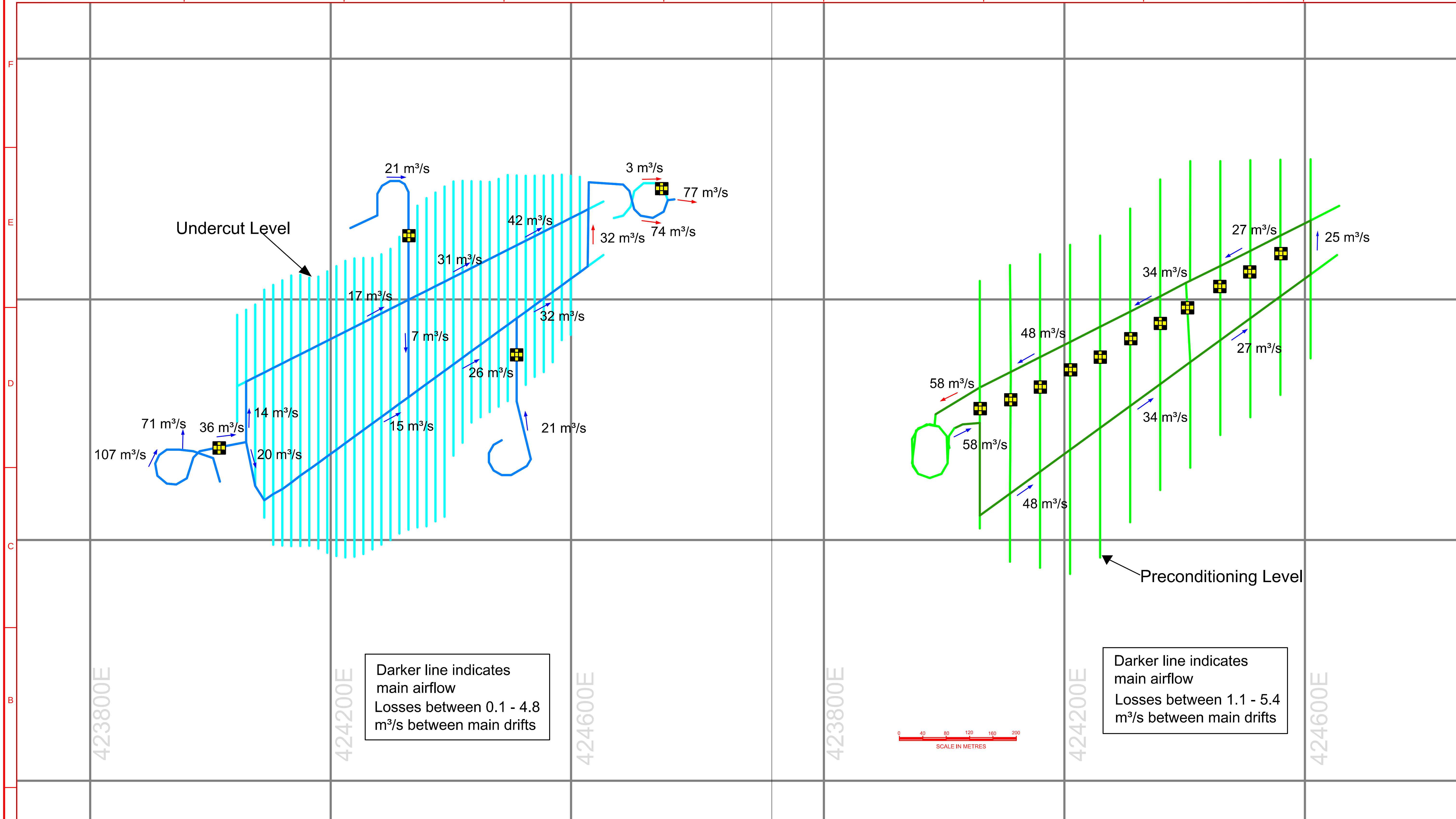

Fan	Bulkhead
Fresh Air	Regulator / Door
Return Air	Fixed Flow (Fan)

REV	MM/DD/YY	REVISION	DES	CHK	DRW	CHK	RVW
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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	VENTILATION SYSTEM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD

PROJECT: IRON CAP BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE:

IRON CAP VENTILATION
UNDERCUT AND PRECONDITIONING LEVEL
PLAN VIEW

PROJECT No.	PHASE No.	TASK No.
1114390002	7000	NA

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	IRON CAP-5	1

APPENDIX E

Electrical Design - WN Brazier & Associates Ltd.

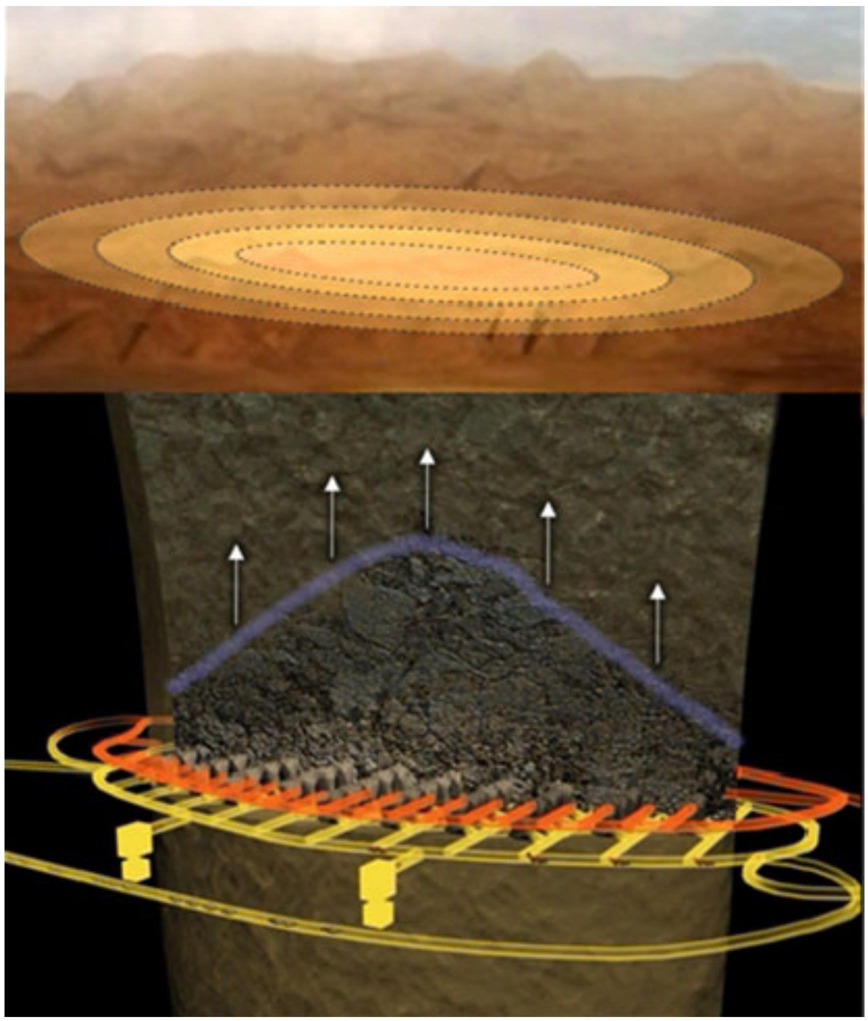
WN BRAZIER ASSOCIATES INC

SEABRIDGE GOLD INC.

KSM PROJECT

GOLDER ASSOCIATES

IRON CAP BLOCK CAVE ELECTRICAL ESTIMATE



Rev. A – December, 2011
Rev. B – January, 2012

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1.0 INTRODUCTION

1.1 General

This pre-feasibility level report, covering the electrical capital costs for the operations phase of the proposed Iron Cap block cave option, has been prepared for Golder Associates Ltd. as requested and directed by Seabridge Gold Inc.

The cost of the initial construction stage power distribution system, to run mining equipment and vent fans, and other equipment as required to drive the initial drifts is not included in this budget and is assumed to be included in mine development costs.

Refer to Section 6 for a cost estimate summary and Appendix A for the estimating spreadsheets.

Figure 1.1 -1 Mitchell Valley

2.0 BASIS OF ESTIMATE

2.1 Golder Associates Ltd.

Golder Associates have provided the basic design information for the proposed block-caving alternative as required to assemble the electrical estimate. This included an electrical load list and basic mine plan. The Iron Cap mine is designed to produce 40,000 tpd.

Harold Bosche, of Bosche Ventures, provided additional information and drawings covering the ore handling conveyor and crusher systems.

All costs are in fourth quarter, 2011, Canadian dollars. As the 7% provincial PST is to be reintroduced, it is included in costs where applicable (such as for a few items of non-production equipment, namely lighting and heating equipment where it will apply once reintroduced). It is to be noted that as the reinstated PST would only apply to lighting and other non-process equipment, the capital cost impact is relatively small, but operating power purchase costs (for the operation phase) will be increased by 7 percent. This has been shown in the revised operating power costs per kilowatt hour.

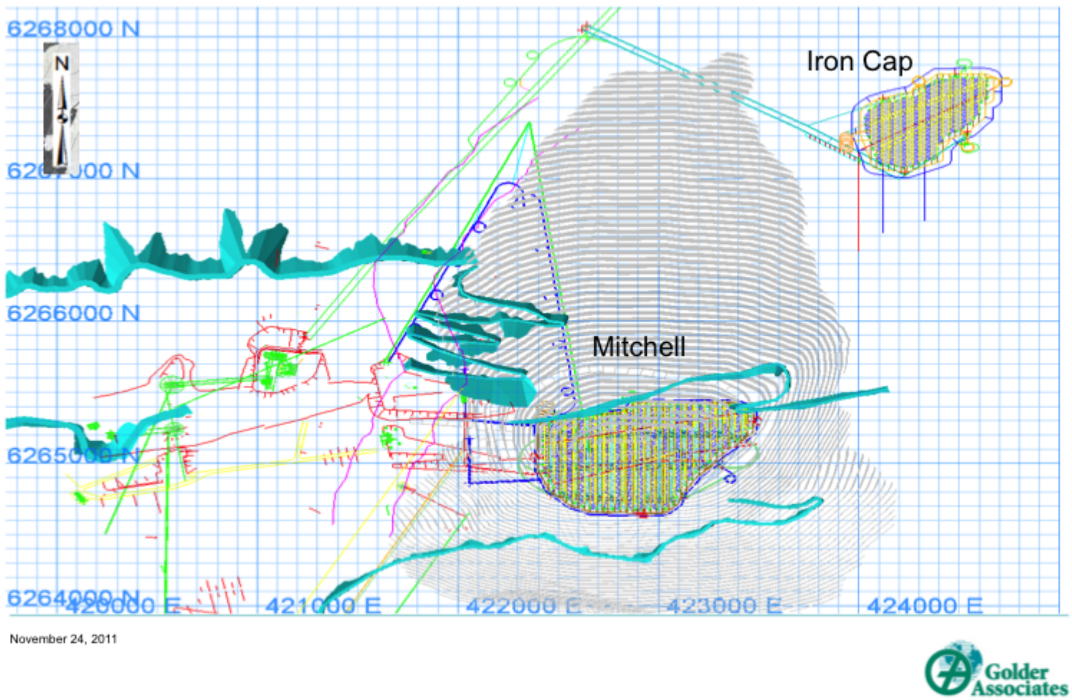
In general, the power supply electrical equipment for the various mine areas have been estimated assuming the installations are all separate. At the design stage it may be possible to group equipment and realize some savings.

The Iron Cap estimate has moved from a scoping level study to the prefeasibility level.

2.2 General Arrangement

The following diagram from Golder illustrates the general mine layout. The Golder plans as included herein form the basis of the electrical estimate.

Figure 2.2 – 1 Mitchell - Iron Cap General Layout (From Golder)



The preconditioning level is accessed from the perimeter drift, with power provided for development activities from the main ring main power system in the perimeter drift.

Figure 2.2 - 2 Mitchell & Iron Cap Plan View (From Golder)

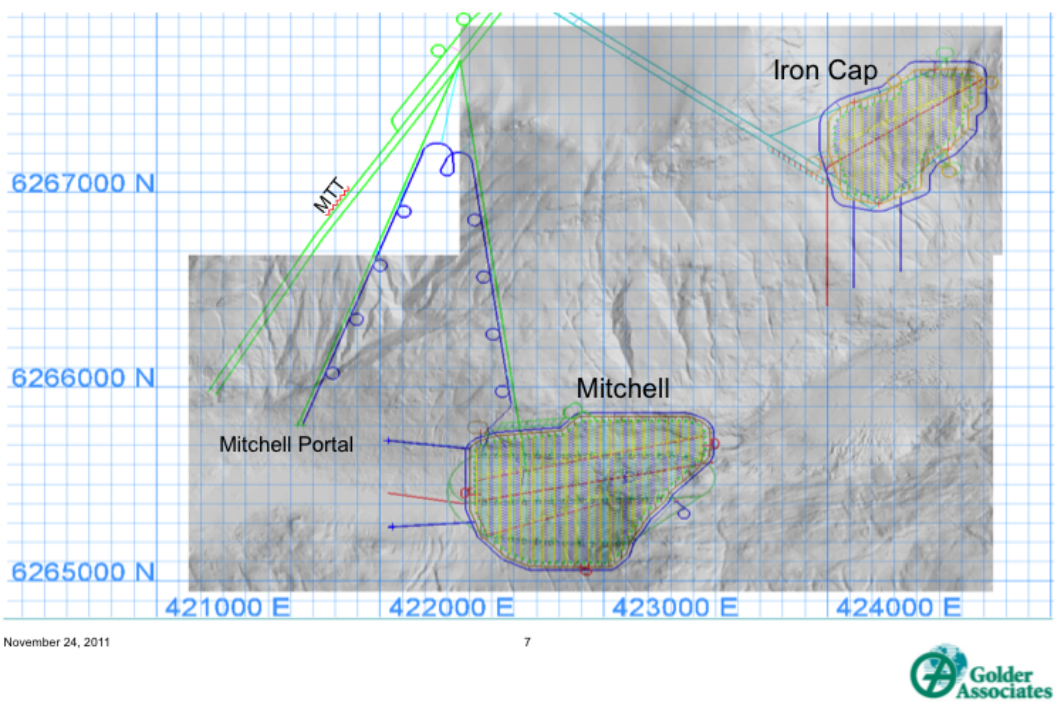


Figure 2.2 – 3 Iron Cap Mine Design (From Golder)

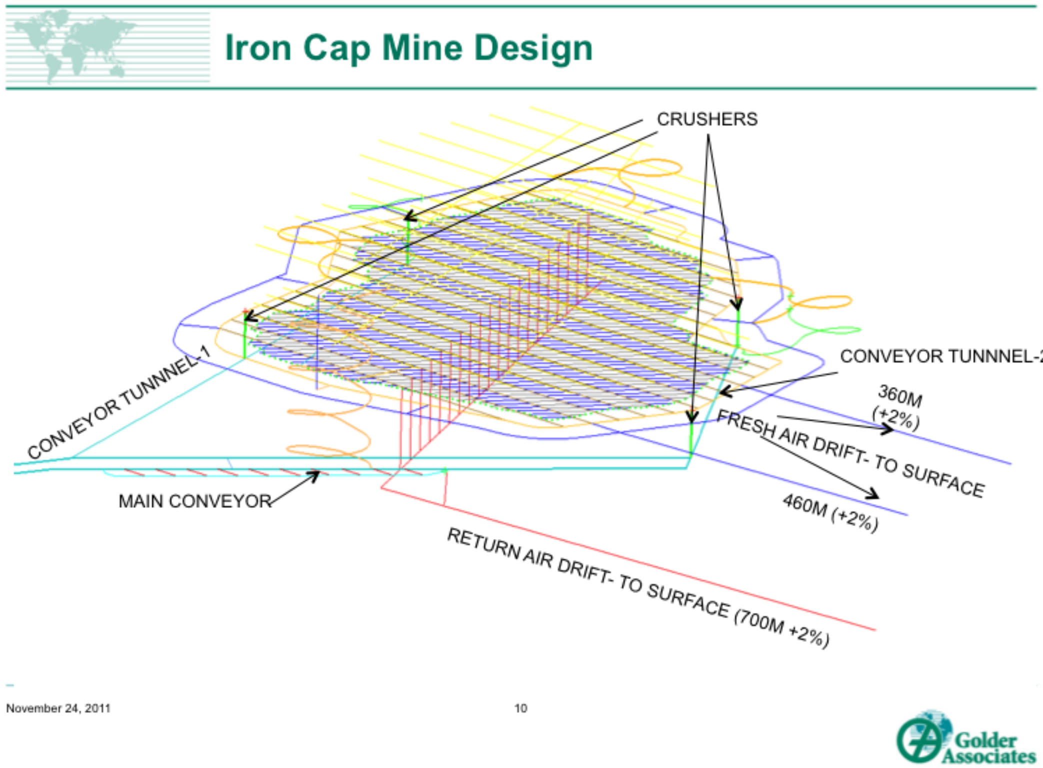
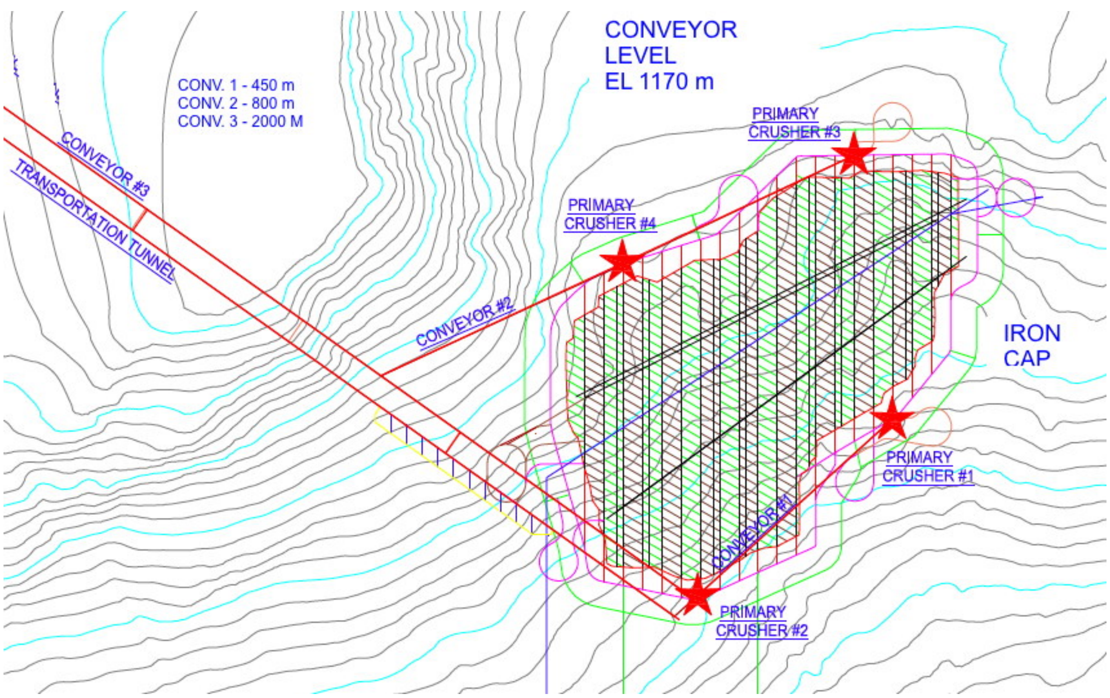


Figure 2.3 - 4 Iron Cap Conveyors (From Bosche Ventures Drawing No. 10-10-1601 Rev. C)



2.3 Items Included

The principal electrical estimated costs include:

- Underground mine power supply cables (for a ring man system).
- Ring main (circuit breaker) units.
- Underground mine switchgear and unit substations (dry type transformers).
- Power supply cables, transformers and switchgear for conveyors and crusher drives.
- Ring main units and unit substations for mine electrical equipment, vent fans, pumps, etc.
- Power and control for main air compressors, vent fans, mine air heaters and stench gas system.
- Communication system.
- Crusher and conveyor power, control, and instrument systems.
- Modular control rooms (knocked down for transport).

2.4 Items Not Included

The cost estimate does not include:

- Electrical temporary installations for the development phase.
- The 600 volt trailing cables, end boxes, etc. as used in the actual operating phase mining (assumed to be in the mining cost).
- The per total cost of electric power as used in mining operations. (This is assumed to be factored into the mining costs.) This report does, however, provide the per kW.h cost to be used.
- The cost of the KSM overall site power distribution system that is allowed for elsewhere.
- Surface facilities including any required emergency generators, etc. (These are being estimated by Wardrop).
- A mine main water pumping system, other than power for small local pumps and an allowance for power for fire water pumps.
- Any electrical costs associated with the mine water after its pumped from the mine.

3.0 ELECTRICAL DESIGN

3.1 General

It is understood that the proposed block cave installation would constitute a large-scale 40,000 tpd mining operation, and as such a ring main style power distribution is planned, rather than a simple radial distribution as one often sees in small mines. With a ring main system there are (at least) two primary power supply cables and each major load, such as underground unit substations, are fed via a 3-breaker ring main unit. Normally, the power supply loop is closed throughout the mine. However, if fault develops in a section of the power cable (say due to a rock fall), the ring main breakers on either side will open to isolate the fault and the mine will remain energized. Essentially all major electrical loads are served by a loop and they can receive power from either side. A suitable protective relaying scheme is used to ensure all circuit breaker tripping is coordinated to ensure continuity of supply.

Although this study has been based on a large amount of diesel equipment (LHDs), the included power supply ring main system and distribution equipment will support a much higher use of electrically powered equipment. As this proposed mine has access to utility power currently costing in the range of C$0.05 per kilowatt hour, electric equipment may be cost effective compared to current diesel fuel prices, especially as ventilation and mine air heating costs could be expected to be reduced with the use of more electrical equipment.

3.2 Mine Main Power Distribution Cable

The basic, initial mine power distribution system will include cables for drifts as listed in the figure below.

Figure 3.2-1 Mine Access Distances

Drift	Length (m)
Conveyor Tunnels	3266
Access Tunnels	3327
Perimeter Drift	2266
Undercut Level	750
Preconditioning Level	750
Vent Drifts	360 + 460 = 820

The main underground power distribution would be sourced from the mine 25 kV distribution circuit breakers in the Mitchell No. 2 (GIS) Substation (as included in the plant site power supply electrical cost estimates).

As the plantsite distribution is 25 kV, the underground mine cables and switchgear will also be 25 kV, thus saving the cost of a step-down substation, and reducing cable sizes and voltage drops for the large underground mine (due to the higher voltage). Underground dry type unit substations will step the voltage down to 4160 volts for conveyor and crusher drives and 600 volts for mining equipment, etc.

In summary to ensure a reliable power system, the Iron Cap mine power supply is via two 25 KV ring main feeders, one running down the conveyor 3 tunnel from the main Mitchell – Teigen conveyor tunnel and the other down the access ramp. Additional lengths of 25 kV cable have been allowed from the power source at the Mitchell side main step-down (GIS) No. 2 Substation.

The 25 kV power supply system will be resistance grounded with rapid tripping to eliminate high earth potentials caused by line to ground faults in the power supply system.

The 4.16 kV and 600 volt systems would also be resistance grounded with rapid tripping to eliminate equipment shock hazard, all in full compliance with the BC

Mines Act. In addition, all trailing cables would of course have pilot check wires with feeders from unit substations and end (distribution) boxes having sensitive ground fault tripping.

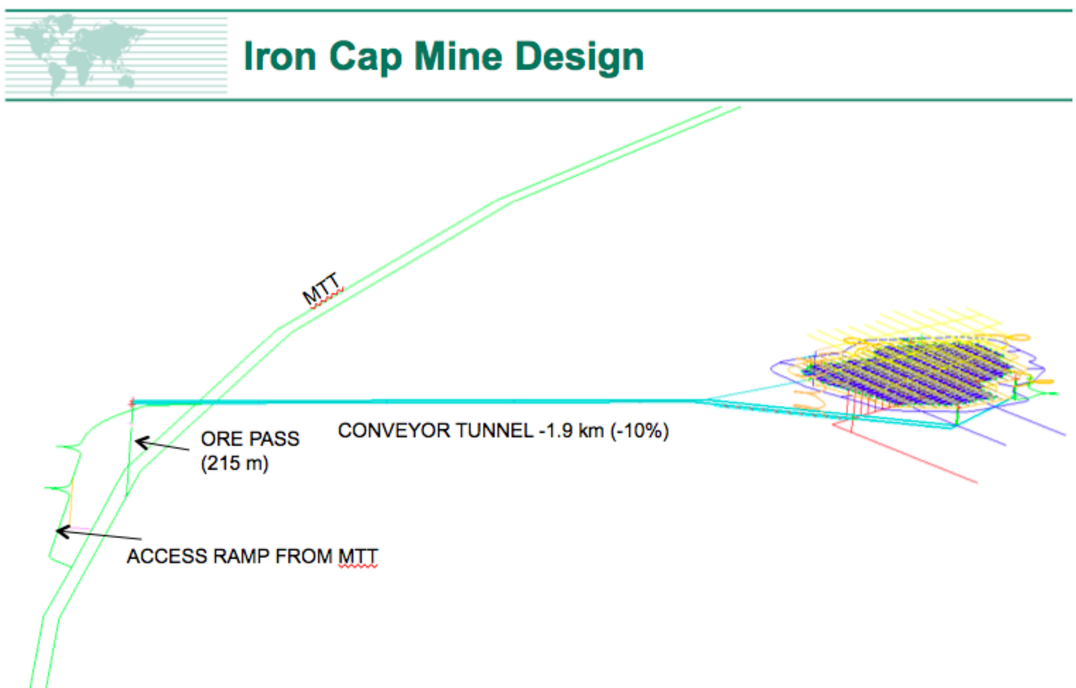
As previously mentioned herein, it is understood that the current mining plans and estimates have more diesel powered production equipment than might be in the final mine development case, given that electric power is around 5 cents per kW.h while diesel fuel is approaching a dollar per litre. The electrical system that has been included for herein would support a future shift to more electrical loads.

The entire installation would be in accordance with the BC Mines Act and Regulations and Parts 1 and 5 of the Canadian Electrical Code.

3.3 Service Ramp

The main mine access, which generally runs in parallel with conveyor 3, will carry one of the two main 25 kV ring main power cables.

Figure 3.3 – 1 Service Ramp



3.4 Pre-Conditioning Level

This level will require power during development. During this period it is understood that electro-hydraulic long-hole pre-conditioning drills that will be used to drill pre-conditioning bores. Portable substations and main 0.6/1 kV Teck power supply cables and end boxes are all included for use in this area during initial operations. Local 600 volt trailing cables from the unit substations or end boxes are assumed to be included in the mining costs with the equipment.

In summary, for drilling and fracturing operations power supply cables and substations are allowed for in this study. The basic estimate includes cable and unit substations for two drifts as part of the basic costs. It is assumed this equipment and cable will be re-used as successive areas are fractured. The cost to re-use and repair/replace the cable and equipment during these successive operations is assumed to be included either in mining costs or sustaining capital and is not in this estimate.

Note, the 600 volt trailing cable to the equipment is assumed to be in the mining costs with the equipment.

3.5 Haulage Level

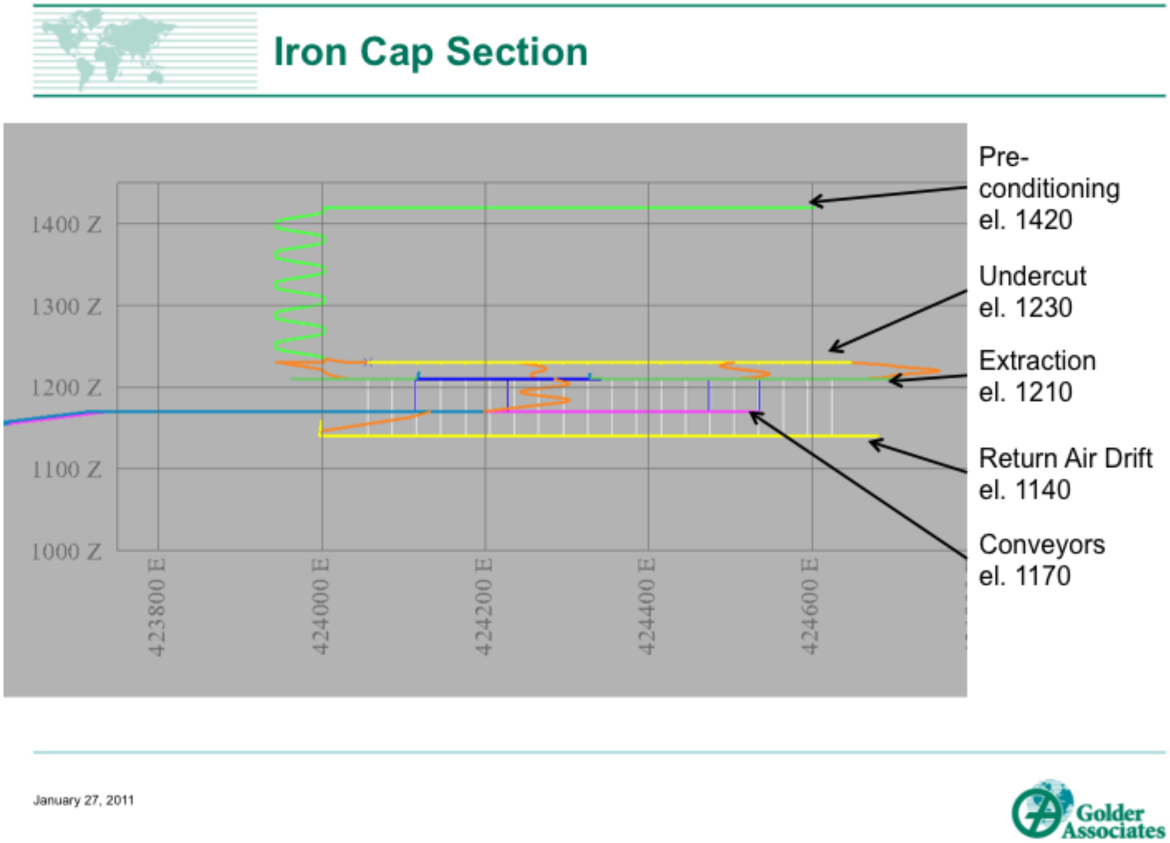
Golder have advised: “The Iron Cap mine won’t have a haulage level; the LHDs will be dumping directly to the crushers.”

3.6 Extraction Level

It is understood that for the extraction level (extraction drift horizon with the draw-points shown in orange on the diagrams) that diesel LHD’s have been allowed for at this stage.

Main power supply cables, several ring main units, unit substations, 0.6/1 kV Teck cable and end boxes have been allowed for on this level.

Figure 3.6 – 1 Sectional View (From Golder)



3.7 Conveyor Power And Control Design

Power Supply

Separate portable unit substations are provided for all the conveying equipment that presents a large load. The main 25 kV power supply is from the mine ring main system.

Ancillary Loads

At each conveyor drive station a dry type step-down 25 kV to 600 volt transformer and MCC, all skid mounted, are provided for ancillary loads. In addition, lighting transformers and low voltage circuit breaker panels are included.

Power and Control Wiring

The supply and installation of local power and control wiring, cable tray, etc. is included.

Conveyor Lighting

An allowance is made for lighting at drive stations and at transfer points, but not along the entire length of the conveyor system.

Control System

The cost for a conveyor PLC control system including cables, pilot devices, PLC hardware, HMIs and programming are included.

Conveyor Drives

Normally drive motors etc. as required are best included with electrical, not mechanical. However, for this preliminary estimate it is understood that the motors have been included with the conveyor equipment.

Refer to the drive list for motor HP, etc.

Skid mounted unit substations and 4160 volt motor starters are included.

Four quadrant regenerative VFDs have been included for downhill conveyor No. 3. The VFDs are specifically designed for underground mine installation.

Belt Scales

Conveyors 1, 2 and 3 are equipped with belt scales.

Metal Detectors

Three metal detectors are allowed for.

Commissioning

The electrical costs associated with conveyor commissioning are included.

3.8 Crusher Station Power And Control

General

There are four crusher stations. Each crusher station includes the crusher, lube system, service crane, dust collector and discharge apron feeder. Note, the cost for power and control will not vary significantly if the size of the crusher motors changes.

Power Supply

The crusher stations would be fed off of the underground ring main system. A 2000 kVA, 25 kV to 4160 volt unit sub would be provided to run each of the crushers (and other possible local equipment) with a 1000 kVA, 600 volt unit substation provided to supply the feeders, crusher auxiliaries, etc.

Power and Control Wiring

The supply and installation of local crusher power and control wiring, cable tray, etc. is included.

Crusher Area Lighting

An allowance is made for local area lighting.

Control System

The cost for a crusher PLC control system including a local control station, cables, pilot devices, HMI running Wonderware, PLC hardware and programming is included.

Crusher Drives

Drive motors themselves are assumed to be included with the crusher equipment.

Control House

A pre-fabricated local control house suitable for underground installation is included. (This will be “knocked down” for transport.)

3.9 Pumping

Local mine pumps would be powered from the 600 volt unit substations, the pumps are by mechanical, this cost code covers local wiring only. As per Golder:

- Iron Cap is designed to have water flow (minimal pumping) to the MTT.
- Mitchell will be design to have the water flow to a central collection point underground, then pumped to surface.
- Block cave inflows are from the pit inflows for the Iron Cap and Mitchell pits from the previous PFS.

At this stage a main pumping station has not been included.

3.10 Mine Ventilation Fans And Air Heating

The surface fans are located at the ends of two vent drifts as shown in the figure below and are powered by cables running down the drifts, fed from the mine ring main system. VFD fan drives, hardened for underground service, have been allowed for.

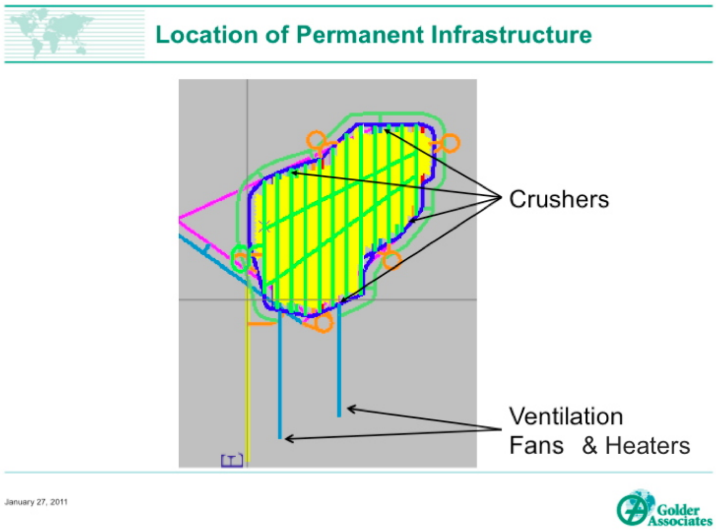
It is assumed that the fan motors are included with the fans by others, but as noted above this estimate includes the VFDs (variable speed drives) that are complete with an integral step-down transformer from 25 kV.

The propane fired mine air heaters are located near the vent fans. The heater quote to Golder included: “heaters equipped with transformers that will step the voltage down from 4160 to whatever is required (575 volts and 120 volts).”

Electrical power has also been allowed for the propane fueled mine air heaters and controls have been included for the stench gas system. It is noted that the mine air heaters were quoted as 4160 volt supply whereas the actual supply is 25 kV. However, this will not make a significant difference in the total cost.

Control cables and a stench gas system has been included.

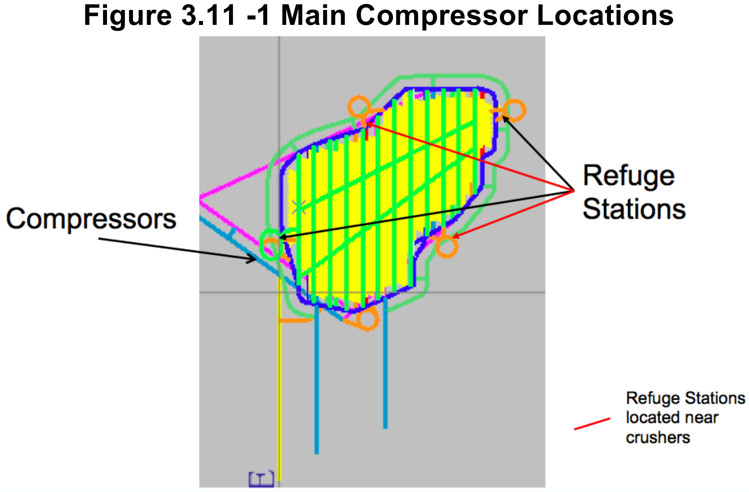
Figure 3.10-1 Vent Fan Locations (From Golder)



3.11 Air Compressors

As per Golder "At least two compressors are to be located outside the mining area with pipe compressed air piped through the underground to feed the equipment, booster stations are not uncommon. For Iron Cap, we have spec'd two 2200 cfm, 500 hp (~380 kw) compressors that will meet the requirements. Since access to outside from Iron Cap is difficult and the majority of the equipment requiring compressed air will be located around the footprint, it is suggest that the compressors be located close to the footprint. In addition to the main compressors, smaller duplex air compressors have been located at each crusher station.

Figure 3.11 -1 Main Compressor Locations



3.12 Engineering

Engineering is included in the indirect cost estimate for electrical design of the permanent facilities.

3.13 Construction Management

Construction management costs are included in the indirect cost estimate.

3.14 Electrical Spares

Electrical spare part costs are included in the indirect cost estimate.

4.0 MINE POWER CONSUMPTION

4.1 Total Iron Cap Load Quantities

The mining load quantities as provided by Golder are shown in Figure 3.12-1 below.

Figure 4.1 -1 Equipment Quantities From Golder, Updated

Equipment	Unit Power (kW)	Quantity	Utilization	Power Factor	Load Factor	Estimate Power Consumption (kW)
Jumbo	150	5	75%	80%	90%	405
Bolter	56	6	75%	80%	90%	181
Drills	56	8	75%	80%	90%	242
Raisebore Machine	242	1	75%	80%	90%	131
Pumps	56	6	75%	75%	90%	170
Surface Fans	750	2	100%	95%	90%	1,283
Underground Fans (est.)	56	15	100%	70%	90%	529
Surface Buildings	--	--	--	--	--	--
Conveyor 1	94	2	90%	80%	90%	122
Conveyor 2	150	1	90%	95%	90%	115
Conveyor 3	2400	1	90%	95%	90%	1,847
Underground Crushers	450	4	90%	95%	90%	1,385
Shop	60	1	50%	90%	90%	24
Refuge Stations	30	4	75%	80%	90%	65
Main Compressor	380	2	75%	90%	90%	462
Subtotal						6,961
Contingency (+25%)						1,740
Total						9,000

4.2 Electrical Load List

The below Excel spreadsheet includes and estimate of the total project running load. This includes all conveyors #1, 2 and 3 but the undersigned does not know if conveyors 2 and 3 will both be running at the start of production.

Figure 4.2 -1 Iron Cap Load List And Calculations

NAME	HP	KW	QTY	TOTAL KW	EFFIC-IENCY	DEMAND FACTOR	UTILIZATION, %	RUNNING LOAD, KW
JUMBO		150	5	750	93%	0.8	75%	484
					100%			
BOLTER		56	6	336	92%	0.8	70%	205
DRILLS		56	8	448	100%	0.8	70%	251
RAISEBORE		242	1	242	91%	0.75	75%	150
MACHINE					100%			
SHOTCRETE		56	2	112	91%	0.7	75%	65
SPRAYER					100%			
PUMPS		56	6	336	92%	0.75	80%	219
					100%			
SURFACE FANS	1000	746	2	1492	93%	0.95	100%	1,524
U/G FANS		56	15	840	92%	0.9	75%	616
MAIN COMPRESS.	500	373	2	746	100%	0.9	50%	336
MISC. HEATING		200	1	200	100%	1	50%	100
SURFACE MISC.		250	1	250	100%	0.4	100%	100
CONVEYOR 2	250	186.5	1	187	94%	0.85	90%	152
			2		94%	0.8	90%	
		0			94%	0.9	90%	
CV2 BELT MAGNET	20	14.92	1	15	94%	0.8	90%	11
CV 2 SPILL FEEDERS	50	37.3	1	37	100%	0.7	10%	3
					100%	0.7	10%	
CONVEYOR 1	400	298.4	1	298	94%	0.8	90%	229
CV1 DUST COLLECOR	50	37.3	1	37	94%	0.9	90%	32
CV 1 SPILL FEEDERS	75	55.95	2	112	100%	0.7	10%	8
CV 1 BELT MAGNET	20	14.92	1	15	93%	0.9	90%	13
CONVEYOR 3	1600	1200	2	2400	100%	0.8	90%	1,728
CV3 DISCH FEEDER	150	111.9	1	112	100%	0.8	90%	81
CV3 DUST COLLECT.	50	37.3	1	37	100%	0.9	90%	30
CV 3 APRON FEEDER	200	152.8	4	611	100%	0.9	90%	495
CV3 AIR COMP	10	7.46	2	15	100%	0.9	90%	12
CV 3 BELT MAGNET	20	14.92	1	15	93%	0.9	90%	13
CV 3 SPILL FEDER	50	37.3	1	37	93%	0.9	70%	25
U/G CRUSHER	600	447.6	4	1790	93%	0.9	70%	1,213
U/G CRUSHER LUBE	50	37.3	4	149	91%	0.86	70%	99
CRUSHER CRANES		20	4	80	100%	0.5	10%	4
CRUSH DUPLX COMP	10	7.46	8	60	93%	0.95	50%	30
CRUSH APRON FDR	150	111.9	4	448	91%	0.8	90%	354
CRUSH ROCK BKR	100	74.6	4	298	100%	0.7	5%	10
CRUSHER BELT MAG	20	14.92	4	60	91%	0.9	90%	53
CRUSH DUST COLL	50	37.3	4	149	100%	0.9	90%	121
CRUSHER SPILL FDR	50	37.3	4	149	100%	0.8	10%	12
					100%			
LTG & SMALL POWER		75	6	450	100%	1	50%	225
HEAT TRACING		200	1	200	100%	1	35%	70
REFUGE STATIONS		30	4	120	100%	0.6	100%	72
U/G SHOP		60	1	60	100%	0.5	100%	30
					100%			
MISC. MONO RAILS		7	8	56	91%	0.85	5%	3
MISC. SUMP PUMPS		15	8	120	91%	0.85	15%	17
					100%			
					100%			
TOTALS				13,870				**9,193**
				CONNECTED NAMEPLATE LOAD, KW				RUNNING LOAD, KW
						ANNUAL GW.h = IRON CAP BLOCK CAVE		**80.53**

The above load calculations are based on all 4 crushers operating and includes for a reasonable amount of ancillary equipment in addition to the main loads. This list assumes haulage is by diesel powered equipment.

4.3 Power Cost

The power cost for the study depends on whether the process plat uses HPGR grinding and thus the whole plant receives BC Hydro PowerSmart credits or whether grinding employs lower efficiency SAG milling, with no discounts in billing. The latter is assumed to be the case so the appropriate power cost, including the cost of site transformer and transformer losses and the upcoming PST, is C$0.054 per Kilowatt-hour.

Figure 4.3 - 1 Power Cost

YEAR	HPGR POWER COST	POWER COST SAG	PERCENT INCREASE	HPGR COST WITH 7% PST	SAG POWER COST WITH 7% PST
2009		$0.0410			
2010		$0.0434	5.9%		
2011	0.042	$0.0469	8.1%		
2012	0.045	$0.051	8.1%	0.048	0.054
THE FOLLOWING RATES ARE PROJECTED AT THE CURRENT 8.1% RATE OF INCREASE					
2013	0.049	0.055	8.1%	0.052	0.059
2014	0.053	0.059	8.1%	0.056	0.063
2015	0.057	0.064	8.1%	0.061	0.069
2016	0.062	0.069	8.1%	0.066	0.074

5.0 MINE COMMUNICATIONS

5.1 General

Communication systems external to the mine are included in the KSM plantsite estimates. The estimates included in this report cover the Iron Cap mine and conveyor systems.

A communications system proposal was received from Mine Site Technologies (Canada) Inc. for the Iron Cap mine communications system. They proposed both Leaky Feeder and their "Impact" Ethernet radio systems. The prices herein have been based on a Leaky Feeder scheme.

A proposal was also included for a PED emergency warning system for the mine.

A fibre-optic control interconnection is also allowed between the crusher and conveyor control systems and the surface installation. This would also provide telephone and internet/email communications links at these locations.

5.2 Basic Leaky Feeder Equipment

The following Figure 5.1-1 includes all basic equipment required for a leaky feeder system. The leaky feeder supply and install is shown separately in another table, such that it can be included in mine costs as the mine is developed. The supply and installation costs are transferred to the master electrical estimating sheet.

The capital cost of $ 366,746 shown in the table below is a base cost for the system. This cost is transferred to the overall electrical cost spreadsheet. Additional costs for the drift installation is shown in the subsequent table.

FIGURE 5.1 - 1 LEAKY FEEDER EQUIPMENT, BASIC SUPPLY				
QTY	PART #	DESCRIPTION	UNIT PRICE	TOTAL PRICE
LEAKY FEEDER SYSTEM				
1	VC-HEADEND	Leaky Feeder Headend (3 channel system)	$46,375	$46,375
1	VC-ANTENNA	Leaky Feeder Antenna System	$11,842	$11,842
20	V-CA75-DS	Leaky Feeder Cable - 350m roll	$1,090	$21,800
18	VC-1004-INF2	Infinity II Line Amplifiers	$1,155	$20,790
5	VC-1007-INF	Infinity Line Branch 3 Way	$821	$4,105
1	V-1007-INF2U	Pilot Tone Generator - Uplink	$390	$390
2	V-KSKSR	KS-KS Rotational Adaptor	$47	$94
2	V-KSNFA	KS - N Female Adaptor	$45	$90
1	VC-1011	Line Termination Kit	$111	$111
2	V-LF32V-PSKIT	32V Leaky Feeder Power Supply w/Inserter	$3,045	$6,090
1	STP-ANT	Stope Antenna (times this by number of drifts)	$1,773	$1,773
1	VC-1007-INF	Infinity Line Branch 3 Way (times this by number of drifts)	$821	$821
1	VC-1020	Telephone Interconnect System (option)	$4,727	$4,727
1	VC-2000KIT	VDV Amplifier Diagnostics Kit (option)	$5,090	$5,090
8	UPS-BCKP	UPS Backup System (option)	$3,636	$29,088
CABLING		(THESE COST WILL BE TABULATED PER METRE OF DRIFT, WITH LABOUR ADDED)		
1	CLN-SFT-CBL	Customer Programming Software / Cloning Cables	$408	$408
	LMR-600	LMR-600 Low Loss Cabling and Connectors (500m)	$4,331	
	ANM-1610 (L-Com)	LMR-600 Connector	$31	
	RG-213	RG-213 Cable (300m)	$135	
	RG-58U	RG58U (100 ft)	$133	
	V-N-CONN	N Type Splice Connector	$20	
	V-N-M-CONN	N Type Male Connectors	$18	
	CB-934	CAB Hooks, Per 1200	$2	
	MISC-CON	Lot, Consumables	$100	
LABOUR		(ONE OFF)		
	LABOUR	Radio Licensing Administration Fee	$250	
	LABOUR	In-Shop Setup and Configuration/Custom Cabling/Radio programming	$1,500	
	LABOUR	On-site Commissioning	$15,000	
	LABOUR	Leaky Feeder Training	$5,000	
RADIOS				
2	FR-5000	FR5000 VHF Repeaters	$2,291	$4,582
1	UR-FR5000	UR-FR5000 VHF Repeater Modules	$1,745	$1,745
BASE STATIONS				
7	IC-F5021	Icom IC-F5021 VHF 128ch Variable Power Analog Mobile	$513	$3,591
7	SEC-1223	Samplex SEC-1223 Desktop Power Supply	$218	$1,526
EQUIPMENT RADIOS		(FOR 66 VEHICLES)		
66	IC-F5021	Icom IC-F5021 VHF 128ch Variable Power Analog Mobile	$513	$33,858
66	WHP-ANT-KT	Mobile Antenna Kit	$1,226	$80,916
PORTABLE RADIOS		(FOR 2 SHIFTS)		
140	IC-F3021S	Icom IC-F3021S 8 Channel Variable Power VHF Portable Radio (For 2 shifts)	$404	$56,560
140	3021-ANT-KT	3021 Tunable Antenna Kit	$28	$3,920
14	BP-232N	Spare 2000 mAh, 7.4V Li-Ion battery packs	$87	$1,218
140	HM-159L	HM-159L Noise Cancelling Speaker Microphone	$87	$12,180
24	BC-197 #22	6 unit multi charger with cups and power supply whole radio or batteries	$544	$13,056
(THIS COST IS TRANSFERRED TO THE MINE BASIC ELECTRICAL CAPEX)			TOTAL	**$366,746**

5.3 Leaky Feeder Installation

The costs shown below were developed from unit costs in the preceding table and are intended to be applied on a per unit (metre) basis. The estimate includes an initial amount of cable for the start of production, which has been transferred to the Electrical cost estimate summary spreadsheet. The below table also shows a per metre number for future installations as the mine is developed. These costs are not tabulated in the electrical cost estimate summary spreadsheet. It is intended that these costs would be added into sustaining capital.

FIGURE 5.3 - 1 LEAKY FEEDER INSTALLATION

QTY	UNIT	DESCRIPTION	LABOUR UNIT	TOTAL HRS	LABOUR COST	UNIT MATERIAL	MATERIAL COST	TOTAL COST
		LABOUR RATE/HR:	$110		(INCLUDES CONSTRUCTION EQUIPMENT)			
INITIAL INSTALLATION								
1000	m	LMR -600 Cable	0.1	100	$11,000	$8.66	$8,662	$19,662
4	ea	LMR -600 Terminations	4	16	$1,760	$31.00	$124	$1,884
1000	m	3/8 Inch Steel Messenger, Cable Hooks, Etc.	0.05	50	$5,500	$3.00	$3,000	$8,500
1	lot	Misc. cable, etc.	100	100	$11,000	$5,000.00	$5,000	$16,000
TOTAL LEAKY FEEDER PER Km							MATERIAL	$46,046
							LABOUR	$55,000
						TOTAL COST PER KM		$101,046
(FOR FUTURE MINE DEVELOPMENT ADD THIS AMOUNT PER KM OF DRIFT)								
0.8	km	Conveyor 1 Drift	500	400	$44,000	$46,046.00	$36,837	$80,837
0.4	km	Conveyor 2 Drift	500	200	$22,000	$46,046.00	$18,418	$40,418
4	km	Conveyor 3 Drift	500	2000	$220,000	$46,046.00	$184,184	$404,184
				0				
8	KM	Initial Drifts	500	4000	$440,000	$46,046.00	$368,368	$808,368
				6600	$726,000		$607,807	$1,333,807
(THIS AMOUNT IS TRANSFERRED TO THE OVERALL ELECTRICAL BUDGET)								TOTAL

5.4 PED System

The figure below shows the cost of PED equipment. The supply costs are transferred to the master electrical estimating spreadsheet where the PED installation costs are also shown.

FIGURE 5.4 - 1 PED EQUIPMENT, BASIC SUPPLY				
QTY	PART #	DESCRIPTION	UNIT PRICE	TOTAL PRICE
TRANSMISSION SYSTEM				
1	P-TX-CABC-000L	PED 110 V Headend Fully Equipped (Long Loop)	$43,391	$43,391
1	P-SEMA-000	Smart External Modulator	$4,364	$4,364
1	P-S010	PEDcall Software Licence	$7,249	$7,249
1		UPS	$4,500	$4,500
ANTENA SYSTEM				
13	P-XLP1X10	Single Core PED antenna (500m roll)	$5,279	$68,627
13	P-SP2KIT	Antenna Splice Kits	$355	$4,615
RECEIVERS				
140	CL-92M-PC-T8-01L	Caplamp, battery and PED ICCL with Integrated Tracking Tag and LED Headpiece	$1,755	$245,700
1	CL-CHGR-30L	Thirty Unit Charging Rack with charging boards for LED Headpiece	$7,000	$7,000
66	AutoPEDs	Vehicle PEDS	$2,590	$170,940
(THIS AMOUNT TRANSFERRED TO ELECTRICAL BUDGET)			TOTAL	$556,386

The installation cost for the above PED equipment, in particular the main loop antenna, is included on the electrical estimating spreadsheet.

Note, the personal PED receivers also include a cap lamp and battery.

The PED price is all-inclusive. Sustaining capital would only be required for more vehicles, more miners and to replace faulty and destroyed equipment.

6.0 ESTIMATE SUMMARY

6.1 General

The electrical power, control, communications and instrument cost estimate is summarized below. All sums are in fourth quarter 2011 Canadian dollars. Refer to the attached Appendix A spreadsheet for estimate details including man-hours and a breakdown of the costs..

6.2 Conveyors 1, 2 And 3

The estimated conveyor electrical and instrumentation cost is: **$8,158,790**

6.3 Main Power Distribution For Mine

The main power supply estimate is: **$9,559,993**

The basic main perimeter power supply system which is a ring main utilizing three breaker ring main units so that each load has two sources of supply. This estimate also includes two main power feeds to the mine from the Mitchell Substation No.2.

Power supply for Fire Pumps (allowance) is **215,625**

6.4 Pre-Conditioning Level Electrical Power Supply Costs

The basic preconditioning electrical power supply is included in Cost Code 2, Power Supply.

Pre-conditioning basic costs are: **$1,518,463**

6.5 Extraction Level Electrical Costs

The basic extraction level costs included in the estimate are: **$5,711,838**

These cost will be repeated as the mine develops.

6.6 Crusher Station Electrical Costs

The crusher electrical and installation costs are:

- Crusher Station 1: **$5,711,838**
- Crusher Station 2: **$2,633,568**
- Crusher Station 3: **$2,633,568**
- Crusher Station 4: **$2,633,568**

6.7 Refuge Station Outfitting Costs

The refuge cost allowance, excluding excavation and other civil costs are:

- Refuge Station 1: **$100,483**
- Refuge Station 2: **$100,483**
- Refuge Station 3: **$100,483**
- Refuge Station 4: **$100,483**

6.8 Main Vent Fans And Power Supply To Mine Air Heaters, Electrical Costs

The electrical, instrumentation and power supply including VFDs for the two main vent fans, the stench gas system, and power to the two mine air heater houses (but excluding the entire heater equipment supply and install is: **$1,214,225**

6.9 Main Air Compressors

Electrical cost for the two main air compressors is: **$571,729**

6.10 Pumping Electrical

There is no separate budget for water pumping at this time.

6.11 Mine Communications (Leaky Feeder & PED)

The supply and installation cost for the basic system is: **$2,463,814**

For sustaining capital for future communication needs on a per km basis, please refer to Section 5 of this report.

6.12 Total Direct Costs

The total direct costs are: **$40,350,678**

6.13 Engineering, Construction Management (Electrical) And Spares, Indirects

The total indirect engineering, construction management and spare parts costs are: **$7,588,500**

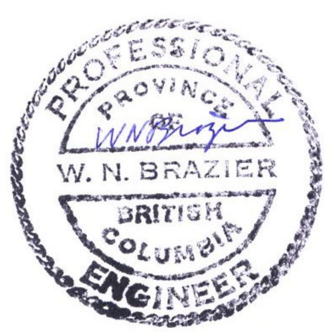
6.14 Grand Total (Electrical, instrumentation & Communications)

Total direct and indirect costs are: **$47,939,178**

For Sustaining Capital, refer to the report and estimate details.

W.N. Brazier

W.N. Brazier, P.Eng.
Jan. 31, 2012



7.0 APPENDIX A - ESTIMATE SPREADSHEET

KSM IRON CAP BLOCK CAVE SCOPING LEVEL ELECTRICAL COST ESTIMATE

NOTES

1) COSTS IN 3RD QTR. 2011 C$
2) IN 2012 (OR 2013) PST WILL APPLY TO LIGHTING AND HEATING (NON-PROCESS EQUIPMENT). THIS HAS BEEN INCLUDED IN THE BASE PRICE.
3) MOTORS ARE ASSUMED PROVIDED WITH CONVEYORS, ETC. BUT VFDs WHERE REQUIRED, ARE INCLUDED HEREWITH.
4) ELECTRICAL SWITCHGEAR, TRANSFORMERS, RING MAIN UNITS, ETC. ARE UNDERGROUND MINING TYPE EQUIPMENT.
5) THE MAIN MINE POWER SUPPLY IS A 25 KV (MATCHES SITE DISTRIBUTION) RING MAIN SYSTEM.

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi- *plier*	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
					1.25	$105						-		-	-	-	-	-	
COST CODE 1 - CONVEYORS CV1, CV2 & CV3					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
CONVEYOR POWER, CONTROL & INSTRUMENTATION					1.25	$105						-		-	-	-	-	-	
(Conveyors CV1,2 & 3, These						$105													
conveyors are powered from the mine																			
25 kV ring main system)					1.25							-		-	-	-	-	-	
CABLE 1/0 5 kV TECK	350	M		0.40	1.25	$105	2.0	70.0				175	18,375	24,500	-	700	-	43,575	
CABLE 4/0 5 kV TECK	350	M		0.45	1.25	$105	2.0	145.0				197	20,672	50,750	-	700	-	72,122	
CV1 & CV2 CABLE 2/0 25 kV	0	M		0.49	1.25	$105	2.0	120.0				-		-	-	-	-	-	
CV3 RING MAIN CABLE, 3C, 25kV,						$105	2.0												
133%, 350 MCM (part of ring main)	2,200	M		0.52	1.25			195.0				1,430	150,150	429,000	-	4,400	-	583,550	
Cable and messenger, clips, etc.						$105	1.0												
(Perimeter)	3,500	M		0.20	1.25			4.0				875	91,875	14,000	-	3,500	-	109,375	
(Rock bolts by others)					1.25	$105						-		-	-	-	-	-	
14,056					1.25	$105						-		-	-	-	-	-	
TERMINATIONS	1	LOT		300.00	1.25	$105		15,000.0				375	39,375	15,000	-	-	-	54,375	
					1.25	$105						-		-	-	-	-	-	
RING MAIN UNITS (Conveyors)	4	ea		75.00	1.25	$105	750.0		113,000			375	39,375	-	452,000	3,000	-	494,375	
25 KV JBs	4	ea		25.00	1.25	$105		1,000.0	5,500			125	13,125	4,000	22,000	-	-	39,125	
UNIT SUBS, 25 KV- 4.16 KV For						$105													
Conveyors					1.25							-		-	-	-	-	-	
2000 KVA C/W MOTOR STARTERS						$105	1500.0												
(CV1 & CV2)	2	EA		175.00	1.25			16,600.0	332,000			438	45,938	33,200	664,000	3,000	-	746,138	
2500 KVA C/W MOTOR STARTERS						$105	1500.0												
(CV3, two dives)	2	EA		175.00	1.25			18,978.2	379,563			438	45,938	37,956	759,126	3,000	-	846,020	
						$105						-		-	-	-	-	-	
600 VOLT UNIT SUB C/W MCC FOR						$105	1000.0												
AUX DRIVES, 500 KVA	4	EA		200.00	1.25				175,000			1,000	105,000	-	700,000	4,000	-	809,000	
CONVEYOR 600 VOLT TECK CABLE					1.25	$105						-		-	-	-	-	-	
(Sized for spaced at 50 HP & >					1.25	$105						-		-	-	-	-	-	
Pilot device wiring 3#14	1,500	m		0.13	1.25	$105		4.76				244	25,594	7,140	-	-	-	32,734	
Pilot Devices 4#14	1,000	m		0.13	1.25	$105		5.65				163	17,063	5,650	-	-	-	22,713	
4#14		m		0.13	1.25	$105		5.65				-		-	-	-	-	-	
CV1, Cv2 & CV 3 Pull Cord sw4#14	4,800	m		0.13	1.25	$105		5.65				780	81,900	27,120	-	-	-	109,020	
6c#14		m		0.13	1.25	$105		7.97				-		-	-	-	-	-	
Control cable local s/s 6#14	2,250	m		0.13	1.25	$105		7.97				366	38,391	17,933	-	-	-	56,323	
Misc. motors 3c# 12	500	m		0.13	1.25	$105		5.86				81	8,531	2,930	-	-	-	11,461	
3c#12		m		0.13	1.25	$105		5.86				-		-	-	-	-	-	
3c#12		m		0.13	1.25	$105		5.86				-		-	-	-	-	-	
CV 1 Dust Coll fan Aux.3#12	150	m		0.13	1.25	$105		5.86				24	2,559	879	-	-	-	3,438	
CV 2 Dust Coll fan Aux.3#12	150	m		0.13	1.25	$105		5.86				24	2,559	879	-	-	-	3,438	
CV 3 Dust Coll fan Aux.3#12	150	m		0.13	1.25	$105		5.86				24	2,559	879	-	-	-	3,438	
4#12		m		0.13	1.25	$105		6.95				-		-	-	-	-	-	
Misc. 3#10	500	m		0.15	1.25	$105		7.87				94	9,844	3,935	-	-	-	13,779	
Conveyor station Air Comp., 4@ 10 HP					1.25	$105													
3#10	600	m		0.15				7.87				113	11,813	4,722	-	-	-	16,535	
4#10		m		0.15	1.25	$105		10.27				-		-	-	-	-	-	
3#6		m		0.15	1.25	$105		19.30				-		-	-	-	-	-	
3#6		m		0.15	1.25	$105		19.30				-		-	-	-	-	-	
3#6		m		0.15	1.25	$105		19.30				-		-	-	-	-	-	
Welding Outlets & Belt Splice 3#6	2,700	m		0.15	1.25	$105		19.30				506	53,156	52,110	-	-	-	105,266	
CV1 Dust Coll Fan, 50 HP3#6	200	m		0.15	1.25	$105		19.30				38	3,938	3,860	-	-	-	7,798	
CV2 Dust Coll Fan, 50 HP3#6	200	m		0.15	1.25	$105		19.30				38	3,938	3,860	-	-	-	7,798	
CV3 Dust Coll Fan, 50 HP3#6	200	m		0.15	1.25	$105		19.30				38	3,938	3,860	-	-	-	7,798	
3c#8		m		0.17	1.25	$105		14.60				-		-	-	-	-	-	
3c#8		m		0.17	1.25	$105		14.60				-		-	-	-	-	-	
3c#8		m		0.17	1.25	$105		14.60				-		-	-	-	-	-	
3c#8		m		0.17	1.25	$105		14.60				-		-	-	-	-	-	
CV1 Belt Magnet 3c#8	200	m		0.17	1.25	$105		14.60				43	4,463	2,920	-	-	-	7,383	
CV2 Belt Magnet 3c#8	200	m		0.17	1.25	$105		14.60				43	4,463	2,920	-	-	-	7,383	
CV3 Belt Magnet 3c#8	200	m		0.17	1.25	$105		14.60				43	4,463	2,920	-	-	-	7,383	
3c#4	0	m			1.25	$105		17.00				-		-	-	-	-	-	
Conv 2 spill feeder 1, 75HP, 3c#4	150	m		0.19	1.25	$105		17.00				36	3,741	2,550	-	-	-	6,291	
Conv 1 spill feeder 1, 75HP, 3c#4	150	m		0.19	1.25	$105		17.00				36	3,741	2,550	-	-	-	6,291	
Conv 1 spill feeder 2, 75HP, 3c#4	150	m		0.19	1.25	$105		17.00				36	3,741	2,550	-	-	-	6,291	
Conv 3 spill feeder 1, 75HP, 3c#4	150	m		0.19	1.25	$105		17.00				36	3,741	2,550	-	-	-	6,291	
3c#4	0	m			1.25	$105		17.00				-		-	-	-	-	-	
3c#2	0	m			1.25	$105		24.50				-		-	-	-	-	-	
3C#1/0	0	m			1.25	$105		40.00				-		-	-	-	-	-	
3c#2/0	0	m			1.25	$105		75.00				-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
Conv. 3 Discharge Apron Fdr 3c#2/0	200	m		0.25	1.25	$105		75.00				63	6,563	15,000	-	-	-	21,563	
3#3/0	0	m			1.25	$105		85.00				-		-	-	-	-	-	
3#4/0	0	m			1.25	$105		112.00				-		-	-	-	-	-	
3c250	0	m			1.25	$105		159.00				-		-	-	-	-	-	
3C300	0	m			1.25	$105		192.00				-		-	-	-	-	-	
4c250	0	m			1.25	$105		198.00				-		-	-	-	-	-	
3c350	0	m			1.25	$105		225.00				-		-	-	-	-	-	
4c350	0	m			1.25	$105		246.50				-		-	-	-	-	-	
3c#500	0	m		0.55	1.25	$105		253.00				-		-	-	-	-	-	
4c#500	0	m		0.55	1.25	$105		334.00				-		-	-	-	-	-	
1c3500	0	m		0.23	1.25	$105		108.20				-		-	-	-	-	-	
3c750	0	m		0.23	1.25	$105		419.26				-		-	-	-	-	-	
Cable Terminations	1	lot		500.00	1.25	$105		25,000.0				625	65,625	25,000	-	-	-	90,625	
					1.25	$105						-		-	-	-	-	-	
INSTRUMENT CABLE					1.25	$105						-		-	-	-	-	-	
INST Cable, 8TR-#16 300V ST-OS					1.25	$105													
STC AIA PVC	900	m		0.25				17.00				281	29,531	15,300	-	-	-	44,831	
Terminations	8	ea		2.00	1.25	$105		18.00				20	2,100	144	-	-	-	2,244	
4 Pair AIC	1,500	m		0.20	1.25	$105		5.20				375	39,375	7,800	-	-	-	47,175	
Terminations	100	ea		1.00	1.25	$105		15.00				125	13,125	1,500	-	-	-	14,625	
Inst JBs, etc.	1	lot		100.00	1.25	$105		2,500.00	8,500			125	13,125	2,500	8,500	-	-	24,125	
2 pair AIC	1,000	m		0.20	1.25	$105		4.20				250	26,250	4,200	-	-	-	30,450	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Conveyor 3 VFDs, 4 quadrant regenerative, designed for						$105													
underground, 1600 HP each	2	ea		200.00	1.25			15,781.3	315,625			500	52,500	31,563	631,250	-	-	715,313	
POWER & CONTROL FOR:					1.25	$105						-		-	-	-	-	-	
Conveyor Belt Magnets					1.25	$105						-		-	-	-	-	-	
Conveyor Cleanup Feeders					1.25	$105						-		-	-	-	-	-	
Conv. Dust Collector fan					1.25	$105						-		-	-	-	-	-	
Conveyor Dust Collector Aux.					1.25	$105						-		-	-	-	-	-	
Conveyor Drive station air comp.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
One lot control stations, JBs, etc.	1	lot		350.00	1.25	$105		7,500.0	15,000			438	45,938	7,500	15,000	-	-	68,438	
					1.25	$105						-		-	-	-	-	-	
Conv. 3 Control program, AC Teck	1	lot			1.25	$105				150,000		-		-	-	-	150,000	150,000	
					1.25	$105						-		-	-	-	-	-	
CABLE TRAY, Local at drive stations, feed points, etc. Along conveyor roper cable is supported from the back on a messenger or is on the conveyor tables (control cable). For all 3 conveyors.						$105								-	-	-	-	-	
					1.25							-							
6 inch	800	m		0.45	1.25	$105		75.0				450	47,250	60,000	-	-	-	107,250	
12 inch	1,600	m		0.60	1.25	$105		82.0				1,200	126,000	131,200	-	-	-	257,200	
18 inch	800	m		0.70	1.25	$105		95.0				700	73,500	76,000	-	-	-	149,500	
24 inch	200	m		0.85	1.25	$105		110.0				213	22,313	22,000	-	-	-	44,313	
Supports	1	lot		400.00	1.25	$105		16,000.0				500	52,500	16,000	-	-	-	68,500	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Fibre Optic Control Cable System, c/w						$105	0.1												
messenger cable, etc.	3,840	m			1.25			6.5				-		24,960	-	192	-	25,152	
Fibrotic splices, panels, etc.	1	lot		500.00	1.25	$105	5000.0	10,000.0	15,000			625	65,625	10,000	15,000	5,000	-	95,625	
					1.25	$105						-		-	-	-	-	-	
PLC CONTROL SYSTEM	1	lot		1500.00	1.25	$105	2500.0	25,000.0	250,000			1,875	196,875	25,000	250,000	2,500	-	474,375	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
PLC Programming	1	lot			1.25	$105		5,000.0		48,000.00		-		5,000	-	-	48,000	53,000	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
CV2 PULL CORD SW	12	ea		4.00	1.25	$105		25.0	500			58	6,125	292	5,833	-	-	12,250	
CV2 BELT RIP DETECTOR	1	ea		16.00	1.25	$105		100.0	2,500			20	2,100	100	2,500	-	-	4,700	
CV2 SPEED SW.	1	ea		20.00	1.25	$105		200.0	1,500			25	2,625	200	1,500	-	-	4,325	
CV2 MISALIGNMENT SW	6	ea		4.00	1.25	$105		25.0	300			30	3,150	150	1,800	-	-	5,100	
CV1 PULL CORDS	22	ea		4.00	1.25	$105		25.0	500			112	11,725	558	11,167	-	-	23,450	
CV1 BELT RIP DETECTOR	1	ea		16.00	1.25	$105		100.0	2,500			20	2,100	100	2,500	-	-	4,700	
CV1 SPEED SW.	1	ea		20.00	1.25	$105		200.0	1,500			25	2,625	200	1,500	-	-	4,325	
CV1 MISALIGNMENT SW	11	ea		4.00	1.25	$105		25.0	300			55	5,775	275	3,300	-	-	9,350	
CV3 PULL CORDS	54	ea		4.00	1.25	$105		25.0	500			272	28,525	1,358	27,167	-	-	57,050	
CV3 BELT RIP DETECTOR	3	ea		16.00	1.25	$105		100.0	2,500			60	6,300	300	7,500	-	-	14,100	
CV3 SPEED SW.	2	ea		20.00	1.25	$105		200.0	1,500			50	5,250	400	3,000	-	-	8,650	
CV3 MISALIGNMENT SW	26	ea		4.00	1.25	$105		25.0	300			130	13,650	650	7,800	-	-	22,100	
					1.25	$105						-		-	-	-	-	-	
BELT SCALES	3	EA		150.00	1.25	$105		2,500.0	21,000			563	59,063	7,500	63,000	-	-	129,563	
METAL DETECTORS	3	EA		120.00	1.25	$105		2,500.0	12,500			450	47,250	7,500	37,500	-	-	92,250	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
LOCAL LIGHTING AT CONVEYOR FEEDS, DRIVE STATIONS AND TRANSFER POINTS	8	lot		250.00	1.25	$105	2500.0	15,000.0	10,000			2,500	262,500	120,000	80,000	20,000	-	482,500	
					1.25	$105						-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
Conveyor 1 Feed, Drive & Discharge					1.25	$105						-		-	-	-	-	-	
Ltg trans & panel	3	lot		80.00	1.25	$105			4,000			300	31,500	-	12,000	-	-	43,500	
Fixtures and supports					1.25	$105						-		0.00	-	-	-	-	
Fixtures and supports					1.25	$105						-		0.00	-	-	-	-	
Cable 3c#12	3,000	m		0.13	1.25	$105	0.5	5.86				488	51,188	17,580.00	-	1,500	-	70,268	
Cable 4c#12	1,500	m		0.13	1.25	$105	0.5	6.95				244	25,594	10,425.00	-	750	-	36,769	
Cable 4c#10	3,000	m		0.15	1.25	$105	0.5	10.27				563	59,063	30,810.00	-	1,500	-	91,373	
Cable					1.25	$105						-		-	-	-	-	-	
Terminations	3	lot		100.00	1.25	$105	0.5		5,000			375	39,375	-	15,000	2	-	54,377	
Fittings, boxes and plug outlets	3	lot		150.00	1.25	$105	0.5		3,500			563	59,063	-	10,500	2	-	69,564	
Supports, etc.	3	lot		50.00	1.25	$105	0.5	5,000.0				188	19,688	15,000	-	2	-	34,689	
					1.25	$105						-		-	-	-	-	-	
Grounding (Conveyors)					1.25	$105						-		-	-	-	-	-	
Ground wire, 2/0 AWG	2,000	m		0.07	1.25	$105	$2.0	7.00				175	18,375	14,000	-	4,000	-	36,375	
Ground Connectors, compression	40	ea		0.75	1.25	$105		30.00				38	3,938	1,200	-	-	-	5,138	
Ground wire, 4/0	500	m		0.1	1.25	$105	$2.0	11.00				63	6,563	5,500	-	1,000	-	13,063	
Ground Connectors, compression	10	ea		0.75	1.25	$105		35.0				9	984	350	-	-	-	1,334	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
For mine communication system see					1.25	$105						-		-	-	-	-	-	
leaky feeder and PED estimate					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Allowance for control room communication over fibre optic system (VoIP)	1	lot		500.00	1.25	$105		6,500.0	25,000			625	65,625	6,500	25,000	-	-	97,125	
						$105						-		-			-	-	Area Subtotal
SUBTOTALS, COST CODE 1, CONVEYORS 1, 2 & 3 ELECTRICAL AND INSTRUMENTATION											-	24,589	2,581,863	1,484,738	3,835,443	58,747	198,000		$8,158,790
COST CODE 2 - MAIN POWER SUPPLY- MINE RING MAIN												-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
NOTE, THIS COST CENTRE INCLUDES FOR A 25 KV RING MAIN POWER SUPPLY IN THE RAMP, PERIMETER DRIFT AND TO UNDERCUT AND PRECONDITIONING LEVELS.					1.25	$105						-		-	-	-	-	-	
TWO RUNS OF RING MAIN CABLE IN MAIN CONVEYOR TUNNEL AND SERVICE RAMP TO POINT WHERE CONV. TUNNEL FROM IRON CAP MEETS THE MAIN TUNNEL AND WHERE RAMP DAYLIGHTS, 25KV, 133%, 3c 350 MCM, Teck 90	4,700	m		0.45	1.25	$105	2.0	195.0				2,644	277,594	916,500	-	9,400	-	1,203,494	
TWO RUNS OF RING MAIN CABLE FROM MITCHELL SUBSTATION DOWN MAIN MITCHELL - TEIGEN CONVEYOR TUNNEL AND ON SURFACE TO SERVICE RAMP ADIT, 25KV, 133%, 3c 350 MCM, Teck 90	4,500	m		0.45	1.25	$105	2.0	195.0				2,531	265,781	877,500	-	9,000	-	1,152,281	
MESSENGER (For above cable, rock bolts by others)	9,200	m		0.15	1.25	$105	1.0	4.0				1,725	181,125	36,800	-	9,200	-	227,125	
CONVEYOR TUNNELS 3266 m (SEE CONVEYOR SECTION)					1.25	$105						-		-	-	-	-	-	
ACCESS TUNNELS 3327 m, main 25 kV ring main cable, 3c 350 MCM	3,450	m		0.45	1.25	$105	3.0	195.0				1,941	203,766	672,750	-	10,350	-	886,866	
PERIMETER DRIFT, 2266 m, 25 kV, 3c 350 MCM	2,240	m		0.45	1.25	$105	3.0	195.0				1,260	132,300	436,800	-	6,720	-	575,820	
UNDERCUT LEVEL, 750 m, 25 kV, 3c 350 MCM	800	m		0.45	1.25	$105	3.0	195.0				450	47,250	156,000	-	2,400	-	205,650	
PRECONDITIONING LEVEL, 750 m, 25 kV, 3c 350 MCM	800	m		0.45	1.25	$105	3.0	195.0				450	47,250	156,000	-	2,400	-	205,650	
VENT DRIFT POWER CABLES, 360 + 460 m = 820m, 4.16 kV, 3c#2/0	950	m		0.45	1.25	$105	3.0	115.0				534	56,109	109,250	-	2,850	-	168,209	
					1.25	$105						-		-	-	-	-	-	
162.9					1.25	$105						-		-	-	-	-	-	
26.1					1.25	$105						-		-	-	-	-	-	
Messenger & clips for cables in drifts (rock bolts by others)	8,240	M		0.15	1.25	$105	1.0	4.0				1,545	162,225	32,960	-	8,240	-	203,425	
Terminations	1	lot		950.00	1.25	$105	3000.0	30,000.0				1,188	124,688	30,000	-	3,000	-	157,688	
HV Junction Boxes, etc.	1	lot		300.00	1.25	$105	1500.0	26,000.0				375	39,375	26,000	-	1,500	-	66,875	
(4783 m long drift)					1.25	$105						-		-	-	-	-	-	
RING MAIN UNIT (for mine equipment, ring main units for conveyors, etc. shown in other areas)	10	EA		100.00	1.25	$105	1000.0	2,000.0	113,000			1,250	131,250	20,000	1,130,000	10,000	-	1,291,250	
					1.25	$105						-		-	-	-	-	-	
UNIT SUB, 25 KV-600V, 1500 KVA (to supply mining equipment, fan, conveyor and crusher unit subs shown in other areas)	10	EA		180.00	1.25	$105	2500.0	9,450.0	189,000			2,250	236,250	94,500	1,890,000	25,000	-	2,245,750	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Grounding					1.25	$105						-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
Ground wire, 2/0 AWG	5,000	m		0.07	1.25	$105	$2.0	7.00				438	45,938	35,000	-	10,000	-	90,938	
Ground Connectors, compression	167	ea		0.75	1.25	$105		30.00				156	16,406	5,000	-	-	-	21,406	
Ground wire, 4/0	17,440	m		0.1	1.25	$105	$2.0	11.00				2,180	228,900	191,840	-	34,880	-	455,620	
Ground Connectors, compression	581	ea		0.75	1.25	$105		35.0				545	57,225	20,347	-	-	-	77,572	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
HEAT TRACE ALLOWANCE FOR WATER PIPES IN MAIN DRIFTS (excludes pipe insulation)	1	LOT		1500.00	1.25	$105	2500.0	50,000.0	75,000			1,875	196,875	50,000	75,000	2,500	-	324,375	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
						$105								-			-	-	Area Subtotal
SUBTOTALS, COST CODE 2, IRON CAP GENERAL MINE POWER DISTRIBUTION												23,336	2,450,306	3,867,247	3,095,000	147,440	-		$9,559,993
						$105						:		-					
COST CODE 3 - FIRE PUMP POWER SUPPLY					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
ALLOWANCE FOR POWER					1.25	$105						-		-	-	-	-	-	
SUPPLY FOR FIRE PUMPS	1	lot		500.00	1.25	$105		50,000.0	100,000			625	65,625	50,000	100,000	-	-	215,625	
INCLUDING POWER SUPPLY					1.25	$105						-		-	-	-	-	-	
(No reliable details available)					1.25	$105						-		-	-	-	-	-	
THIS DOES NOT INCLUDE THE					1.25	$105						-		-	-	-	-	-	
PUMPS, MOTOR, FIRE					1.25	$105						-		-	-	-	-	-	
PUMP CONTROLLERS OR					1.25	$105						-		-	-	-	-	-	
POSSIBLE HEAT TRACE.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
						$105								-			-	-	Area Subtotal
SUBTOTAL COST CODE 3 - FIRE PUMP POWER SUPPLY											-	625	65,625	50,000	100,000	-	-		215,625
COST CODE 4 PRECONDITIONING																			
POWER SUPPLY PRE-CONDITIONING					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
NOTE, 25 KV RING MAIN CABLE IS IN COST CODE 2					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105	2.0												
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	2,000	m		0.49				120.0				1,225	128,625	240,000	-	4,000	-	372,625	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING					1.25	$105	1000.0	2,000.0	113,000										
SUPPLY	4	ea		75.00								375	39,375	8,000	452,000	4,000	-	503,375	
					1.25	$105						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600					1.25	$105	2500.0	9,450.0	189,000										
VOLTS, 1500 kva	2	ea		100.00								250	26,250	18,900	378,000	5,000	-	428,150	
END (DISTRIBUTION) BOXES	2	ea		10.00	1.25	$105		950.0	9,500			25	2,625	1,900	19,000	-	-	23,525	
600 VOLT TECK CABLE 3c 250 MCM	800	m		0.20	1.25	$105	2.0	165.0				200	21,000	132,000	-	1,600	-	154,600	
and MESSENGER					1.25	$105						-		-	-	-	-	-	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
TERMINATIONS, ETC.	1	lot		150.00	1.25	$105	1500.0	15,000.0				188	19,688	15,000	-	1,500	-	36,188	
					1.25	$105						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 4 PRE-CONDITIONING											-	2,263	237,563	415,800	849,000	16,100	-		$1,518,463
COST CODE 5 - EXTRACTION LEVEL					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
POWER SUPPLY PRE-CONDITIONING					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
NOTE, 25 KV 350 MCM RING MAIN CABLE IS IN COST CODE 2					1.25	$105						-		-	-	-	-	-	
					1.25	$105	2.0												
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	0	m		0.49				120.0				-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING					1.25	$105	1000.0	2,000.0	113,000										
SUPPLY	14	ea		75.00								1,313	137,813	28,000	1,582,000	14,000	-	1,761,813	
					1.25	$105						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600					1.25	$105	2500.0	8,750.0	175,000										
VOLTS, 1000 kva	14	ea		100.00								1,750	183,750	122,500	2,450,000	35,000	-	2,791,250	
END BOXES	14	ea		10.00	1.25	$105		850.0	8,500			175	18,375	11,900	119,000	-	-	149,275	
MISC. 5 KV CABLE, ETC.	0	lot			1.25	$105						-		-	-	-	-	-	
600 VOLT TECK CABLE 3c 250 MCM					1.25	$105													
and MESSENGER	4,900	m		0.20				165.0				1,225	128,625	808,500	-	-	-	937,125	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-		-		-	
TERMINATIONS, ETC.	1	lot		150.00	1.25	$105	1500.0	15,000.0				188	19,688	15,000	-	1,500	-	36,188	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 5 EXTRACTION LEVEL											-	7,100	745,500	1,416,700	5,000,000	68,100	-		$5,711,838
					1.25	$105						-		-	-	-	-	-	
COST CODE 6 - CRUSHER STATION #1																			
(Typical 1 of 4)					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
MOTORS					1.25	$105						-		-	-	-	-	-	
Assumed supplied with equipment					1.25	$105						-		-	-	-	-	-	
RING MAIN UNIT	1	ea		100.00	1.25	$105	2000.0	3,000.0	115,000			125	13,125	3,000	115,000	2,000	-	133,125	
UNIT SUBSTATION - 2000 KVA 25-4.16 kV with starters	1	ea		250.00	1.25	$105	2500.0	6,500.0	345,000			313	32,813	6,500	345,000	2,500	-	386,813	
UNIT SUB & AUX MCC FOR	1	ea		180.00	1.25	$105	2500.0	5,000.0	286,000			225	23,625	5,000	286,000	2,500	-	317,125	
600 VOLT DRIVES, 500 KVA					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
POWER CABLE					1.25	$105						-		-	-	-	-	-	
(SP = Spaced, RF= Random Fill)					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
5 KV TECK					1.25	$105						-		-	-	-	-	-	
Crusher Motor #2 SP	200	m			1.25	$105		74.2				-		14,840	-	-	-	14,840	
Terminations	4	ea		4.00	1.25	$105		150.0				20	2,100	600	-	-	-	2,700	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
600 /1KV CABLE					1.25	$105						-		-	-	-	-	-	
Lube system motors # 8 RF	300	m			1.25	$105		8.9				-		2,682	-	-	-	2,682	
Rock Breaker # 2 SP	100	m			1.25	$105		29.0				-		2,900	-	-	-	2,900	
Apron feeder # 2/0 SP	150	m			1.25	$105		75.0				-		11,250	-	-	-	11,250	
Monorail hoist # 10	100	m			1.25	$105		6.7				-		670	-	-	-	670	
Bridge Crane #6	100	m			1.25	$105		14.6				-		1,460	-	-	-	1,460	
2 Air compressors #8	200	m			1.25	$105		8.9				-		1,788	-	-	-	1,788	
					1.25	$105						-		-	-	-	-	-	
Welding Outlets # 6	300	m			1.25	$105		14.6				-		4,380	-	-	-	4,380	
Belt splice station #6	75	m			1.25	$105		14.6				-		1,095	-	-	-	1,095	
spill feeder (see conveyor loads)					1.25	$105						-		-	-	-	-	-	
Dust Collector Fan					1.25	$105						-		-	-	-	-	-	
Dust Collector Aux.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Lighting Transformer 45 kVA	2	ea		25.00	1.25	$105		200.0	2,100			63	6,563	400	4,200	-	-	11,163	
Control Transformer 30 kVA	1	ea		25.00	1.25	$105		200.0	1,800			31	3,281	200	1,800	-	-	5,281	
Unit Heaters 10 kW 600 V	5	ea		8.00	1.25	$105		200.0	1,950			50	5,250	1,000	9,750	-	-	16,000	
Jan 3/12 copper 3.43/lb.					1.25	$105						-		-	-	-	-	-	
Misc. 600V/1kV Teck					1.25	$105						-		-	-	-	-	-	
3#14	1,500	m		0.13	1.25	$105		4.76				244	25,594	7,140	-	-	-	32,734	
4#14	1,500	m		0.13	1.25	$105		5.65				244	25,594	8,475	-	-	-	34,069	
6#14	1,500	m		0.13	1.25	$105		7.97				244	25,594	11,955	-	-	-	37,549	
	1,500	m		0.13	1.25	$105						244	25,594	-	-	-	-	25,594	
3#12	1,500	m		0.13	1.25	$105		5.86				244	25,594	8,790	-	-	-	34,384	
4#12	1,500	m		0.13	1.25	$105		6.95				244	25,594	10,425	-	-	-	36,019	
3#10	500	m		0.15	1.25	$105		7.87				94	9,844	3,935	-	-	-	13,779	
4#10	500	m		0.15	1.25	$105		10.27				94	9,844	5,135	-	-	-	14,979	
3#6	500	m		0.15	1.25	$105		19.30				94	9,844	9,650	-	-	-	19,494	
3#8	200	m		0.17	1.25	$105		14.60				43	4,463	2,920	-	-	-	7,383	
3c#4	0	m			1.25	$105		17.00				-		-	-	-	-	-	
3c#2	0	m			1.25	$105		24.50				-		-	-	-	-	-	
3C#1/0	0	m			1.25	$105		40.00				-		-	-	-	-	-	
3#3	0	m			1.25	$105		33.00				-		-	-	-	-	-	
3#4	0	m			1.25	$105		46.00				-		-	-	-	-	-	
3#3/0	0	m			1.25	$105		85.00				-		-	-	-	-	-	
3#4/0	0	m			1.25	$105		112.00				-		-	-	-	-	-	
3c250	0	m			1.25	$105		159.00				-		-	-	-	-	-	
3C300	0	m			1.25	$105		192.00				-		-	-	-	-	-	
4c250	0	m			1.25	$105		198.00				-		-	-	-	-	-	
3c350	0	m			1.25	$105		225.00				-		-	-	-	-	-	
4c350	0	m			1.25	$105		246.50				-		-	-	-	-	-	
3c#500	0	m		0.55	1.25	$105		253.00				-		-	-	-	-	-	
4c#500	0	m		0.55	1.25	$105		334.00				-		-	-	-	-	-	
1c3500	0	m		0.23	1.25	$105		108.20				-		-	-	-	-	-	
3c750	0	m		0.23	1.25	$105		419.26				-		-	-	-	-	-	
MISC. 5 KV CABLE	0	m			1.25	$105						-		-	-	-	-	-	
	0	m			1.25	$105						-		-	-	-	-	-	
3c2/0	0	m			1.25	$105		70.00				-		-	-	-	-	-	
3C250	0	m			1.25	$105		192.78				-		-	-	-	-	-	
3c300	0	m			1.25	$105		119.00				-		-	-	-	-	-	
3c350	0	m			1.25	$105		245.00				-		-	-	-	-	-	
3c400	0	m			1.25	$105						-		-	-	-	-	-	
3c500	0	m			1.25	$105		199.00				-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
	0	m			1.25	$105						-		-	-	-	-	-	
MISC. 25 KV 133% CABLES	0	m			1.25	$105						-		-	-	-	-	-	
3c4/0	0	m			1.25	$105		87.00				-		-	-	-	-	-	
3c400	0	m			1.25	$105		115.00				-		-	-	-	-	-	
3c2/0	0	m			1.25	$105		114.00				-		-	-	-	-	-	
3c300	0	m			1.25	$105		172.00				-		-	-	-	-	-	
3c500	0	m			1.25	$105		257.00				-		-	-	-	-	-	
	0	m			1.25	$105						-		-	-	-	-	-	
	0	m			1.25	$105						-		-	-	-	-	-	
CONTROL CABLE					1.25	$105						-		-	-	-	-	-	
3#14 Teck	2,000	m		0.13	1.25	$105		2.9				325	34,125	5,800	-	-	-	39,925	
6#14 Teck	1,500	m		0.13	1.25	$105		5.0				244	25,594	7,500	-	-	-	33,094	
12#14 Teck	500	m		0.15	1.25	$105		10.0				94	9,844	5,000	-	-	-	14,844	
20#14 Teck	200	m		0.17	1.25	$105		16.0				43	4,463	3,200	-	-	-	7,663	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
CONTROL CABLE TERMINATIONS					1.25	$105						-		-	-	-	-	-	
3#14 Teck	100	ea		0.50	1.25	$105		100.0				63	6,563	10,000	-	-	-	16,563	
6#14 Teck	76	ea		0.75	1.25	$105		100.0				71	7,481	7,600	-	-	-	15,081	
12#14 teck	50	ea		1.00	1.25	$105		125.0				63	6,563	6,250	-	-	-	12,813	
20#14 teck	50	ea		1.50	1.25	$105		130.0				94	9,844	6,500	-	-	-	16,344	
					1.25	$105						-		-	-	-	-	-	
INSTRUMENT CABLE					1.25	$105						-		-	-	-	-	-	
INST Cable, 8TR-#16 300V ST-OS					1.25	$105													
STC AIA PVC	150	m		0.25				17.0				47	4,922	2,550	-	-	-	7,472	
Terminations	4	ea		2.00	1.25	$105		18.0				10	1,050	72	-	-	-	1,122	
4 Pair AIC	1,500	m		0.20	1.25	$105		4.2				375	39,375	6,300	-	-	-	45,675	
Terminations	32	ea		1.00	1.25	$105		15.0				40	4,200	480	-	-	-	4,680	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
CABLE TRAY					1.25	$105						-		-	-	-	-	-	
150	200			0.55	1.25	$105		75.0				138	14,438	15,000	-	-	-	29,438	
300	400			0.60	1.25	$105		82.0				300	31,500	32,800	-	-	-	64,300	
450	100			0.70	1.25	$105		95.0				88	9,188	9,500	-	-	-	18,688	
600	200			0.85	1.25	$105		110.0				213	22,313	22,000	-	-	-	44,313	
900	100			1.10	1.25	$105		120.0				138	14,438	12,000	-	-	-	26,438	
Fittings @ 15%	1	lot			1.25	$105						-	13,781	-	-	-	-	13,781	
Tray Support steel@ 10%	1	lot			1.25	$105						-	9,188	-	-	-	-	9,188	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
BOXES AND FITTINGS, etc.	1	lot		750.00	1.25	$105		35,000.0	5,000			938	98,438	35,000	5,000	-	-	138,438	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Grounding					1.25	$105						-		-	-	-	-	-	
Ground wire, 2/0 AWG	1,000	m		0.07	1.25	$105		7.00				88	9,188	7,000	-	-	-	16,188	
Ground Connectors, compression	33	ea		0.75	1.25	$105		35.00				31	3,281	1,167	-	-	-	4,448	
Ground wire, 4/0	200	m		0.1	1.25	$105		11.00				25	2,625	2,200	-	-	-	4,825	
Ground Connectors, compression	7	ea		0.70	1.25	$105		30.0				6	613	200	-	-	-	813	
LOCAL LIGHTING AT CRUSHER	1	lot		550.00	1.25	$105	2500.0	40,000.0				688	72,188	40,000	-	2,500	-	114,688	
					1.25	$105						-		-	-	-	-	-	
CRUSHER E-HOUSE/CONTROL					1.25	$105	2500.0												
ROOM	1	ea		500.00				10,000.0	175,000			625	65,625	10,000	125,000	2,500	-	203,125	
					1.25	$105						-		-	-	-	-	-	
CONTROL SYSTEM INCLUDING PLC,					1.25	$105	2500.0												
PROGRAMMING, CABLE. PILOT																			
DEVICES, ETC.	1	lot		1600.00				75,000.0	100,000			2,000	210,000	75,000	100,000	2,500	-	387,500	
					1.25	$105						-		-	-	-	-	-	
INSTRUMENTATION					1.25	$105						-		-	-	-	-	-	
General	1	lot		500.00	1.25	$105		2,500.0	50,000			625	65,625	2,500	50,000	-	-	118,125	
CCTV	1	LOT		200.00	1.25	$105		3,500.0	25,000			250	26,250	3,500	25,000	-	-	54,750	
					1.25	$105						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 6 CRUSHER STATION 1											-	10,229	1,097,009	455,309	1,066,750	14,500	-		2,633,568
COST CODE 7 CRUSHER STATION 2					1.25	$105						10,229	1,097,009	455,309	1,066,750	14,500	-	2,633,568	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 7 CRUSHER STATION 2											-	10,229	1,097,009	455,309	1,066,750	14,500	-		$2,633,568
																		-	
COST CODE 8 CRUSHER STATION 3					1.25	$105				-		-	-	-	-	-		-	
					1.25	$105				-		10,229	1,097,009	455,309	1,066,750	14,500		2,633,568	
										-		-	-	-	-	-		-	Area Subtotal
SUBTOTAL COST CODE 8 CRUSHER STATION 3										-	-	10,229	1,097,009	455,309	1,066,750	14,500	-		$2,633,568
																		-	
COST CODE 9 CRUSHER STATION 4					1.25	$105				-		10,229	1,097,009	455,309	1,066,750	14,500		2,633,568	
					1.25	$105				-		-	-	-	-	-		-	
										-		-	-	-	-	-		-	Area Subtotal
SUBTOTAL COST CODE 9 CRUSHER STATION 4										-	-	10,229	1,097,009	455,309	1,066,750	14,500	-		$2,633,568
					1.25	$105						-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
COST CODE 10					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 10										-	-	-	-	-	-	-	-		$0
					1.25	$105						-		-	-	-	-	-	
COST CODE 11, REFUGE STATION 1					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Transformer, breaker panel, etc.	1	lot		44.00	1.25	$105		1,000.0	6,500			55	5,775	1,000	6,500	-	-	13,275	
					1.25	$105						-		-	-	-	-	-	
Light fixtures, vapour tight fluorescent	10	ea		2.50	1.25	$105		20.0	225			31	3,281	200	2,250	-	-	5,731	
Emergency	2	ea		3.00	1.25	$105		20.0	400			8	788	40	800	-	-	1,628	
Door HID	1	ea		2.50	1.25	$105		20.0	325			3	328	20	325	-	-	673	
Heaters	3	ea		5.00	1.25	$105		25.0	1,500			19	1,969	75	4,500	-	-	6,544	
Communications	1	lot		30.00	1.25	$105	500.0	2,500.0	2,500			38	3,938	2,500	2,500	500	-	9,438	
					1.25	$105						-		-	-	-	-	-	
Allowance for air, safety equip. furniture, etc.	1	lot		100.00	1.25	$105	500.0	5,000.0	15,000			125	13,125	5,000	15,000	500	-	33,625	
					1.25	$105						-		-	-	-	-	-	
Outlets, boxes, etc.	1	lot		30.00	1.25	$105		2,500.0				38	3,938	2,500	-	-	-	6,438	
					1.25	$105						-		-	-	-	-	-	
Cable, 600 V Teck					1.25	$105						-		-	-	-	-	-	
3#12	250	m		0.13	1.25	$105		5.9				41	4,266	1,465	-	-	-	5,731	
4#12	100	m		0.13	1.25	$105		7.0				16	1,706	695	-	-	-	2,401	
					1.25	$105						-		-	-	-	-	-	
Terminations	1	lot		40.00	1.25	$105		1,500.0				50	5,250	1,500	-	-	-	6,750	
					1.25	$105						-		-	-	-	-	-	
Grounding	1	lot		40.00	1.25	$105	500.0	2,500.0				50	5,250	2,500	-	500	-	8,250	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
REFUGE STATION 2					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 11, REFUGE STATION 1											-	473	49,613	17,495	31,875	1,500	-		$100,483
COST CODE 12 REFUGE STATION 2					1.25	$105						-		-	-	-	-	-	
					1.25	$105						473	49,613	17,495	31,875	1,500	-	100,483	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 12, REFUGE STATION 2											-	473	49,613	17,495	31,875	1,500	-		$100,483
					1.25	$105						-		-	-	-	-	-	
COST CODE 13 REFUGE STATION 3					1.25	$105						-		-	-	-	-	-	
					1.25	$105						473	49,613	17,495	31,875	1,500	-	100,483	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 12, REFUGE STATION 3											-	473	49,613	17,495	31,875	1,500	-		$100,483
					1.25	$105						-		-	-	-	-	-	
COST CODE 14 REFUGE STATION 4					1.25	$105						473	49,613	17,495	31,875	1,500	-	100,483	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 14, REFUGE STATION 4											-	473	49,613	17,495	31,875	1,500	-		$100,483
					1.25	$105						-		-	-	-	-	-	
COST CODE 15, MAIN VENT FANS (2) AND AIR HEATERS POWER (2)					1.25	$105						-		-	-	-	-	-	
MAIN VENT FANS																			
(NOTE CABLE DOWN DRIFT FOR VENT FANS IS INCLUDED IN COST CODE 2 - MAIN POWER SUPPLY.)					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
MOTORS ASSUMED SUPPLIED WITH FANS					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
VFDs, DESIGNED FOR U/G MINE INSTALLATION, c/w step-down xformer from 25 kV	2	ea		175.00	1.25	$105	2500.0	15,750.0	315,000			438	45,938	31,500	630,000	5,000	-	712,438	
					1.25	$105						-		-	-	-	-	-	
LOCAL CABLE TO MOTORS, 3c 2/0 5 kV	200	M			1.25	$105						-		-	-	-	-	-	
	162.9				1.25	$105						-		-	-	-	-	-	
Terminations, boxes, hardware, supports, etc.	1	lot		150.00	1.25	$105	2500.0	12,000.0				188	19,688	12,000	-	2,500	-	34,188	
					1.25	$105						-		-	-	-	-	-	
Stench gas system	1	lot		100.00	1.25	$105	1000.0	2,500.0	11,000			125	13,125	2,500	11,000	1,000	-	27,625	
					1.25	$105						-		-	-	-	-	-	
Control system fibre optic cable c/w messenger	1,000	m		0.50	1.25	$105	2.0	7.0				625	65,625	7,000	-	2,000	-	74,625	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
Fibre optic terminations, panels, equip. etc.	1	lot		250.00	1.25	$105	2500.0	12,000.0	15,000			313	32,813	12,000	15,000	2,500	-	62,313	
Controls and Instruments	1	lot		200.00	1.25	$105	1500.0	6,000.0	7,000			250	26,250	6,000	7,000	1,500	-	40,750	
Misc. control & instrument cable, etc.	1	lot		250.00	1.25	$105	500.0	10,000.0	5,000			313	32,813	10,000	5,000	500	-	48,313	
					1.25	$105						-		-	-	-	-	-	
POWER SUPPLY TO MINE AIR HEATERS						$105													
					1.25							-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
HV JUNCTION BOX (FROM FANS)	2	EA		100.00	1.25	$105		5,500.0				250	26,250	11,000	-	-	-	37,250	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
CABLE 3C # 2/0, 25 kV	200	M		0.49	1.25	$105	2.0	120.0				123	12,863	24,000	-	400	-	37,263	
					1.25	$105						-		-	-	-	-	-	
CONTROL (ALARM) CONNECTIONS	1	LOT		200.00	1.25	$105	1000.0	10,000.0	10,000			250	26,250	10,000	10,000	1,000	-	47,250	
MISC. BOXES, CABLES, TERMINATIONS, ETC.	1	LOT		350.00	1.25	$105	1500.0	15,000.0	5,000			438	45,938	15,000	5,000	1,500	-	67,438	
					1.25	$105						-		-	-	-	-	-	
Grounding					1.25	$105						-		-	-	-	-	-	
Ground wire, 2/0 AWG	500	m		0.07	1.25	$105	$2.0	7.00				44	4,594	3,500	-	1,000	-	9,094	
Ground Connectors, compression	10	ea		0.75	1.25	$105		30.00				9	984	300	-	-	-	1,284	
Ground wire, 4/0	500	m		0.1	1.25	$105	$2.0	11.00				63	6,563	5,500	-	1,000	-	13,063	
Ground Connectors, compression	10	ea		0.75	1.25	$105		35.0				9	984	350	-	-	-	1,334	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 15 MAIN VENT FANS AND AIR HEATER POWER (2)											-	3,435	360,675	150,650	683,000	19,900	-		$1,214,225
					1.25	$105						-		-	-	-	-	-	
COST CODE 16 MAIN AIR COMPRESSORS POWER & CONTROL					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Note, power supply is from ring main system in the drift. See Cost Code 2)					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
RING MAIN UNITS	1	ea		100.00	1.25	$105	750.0		113,000			125	13,125	-	113,000	750	-	126,875	
					1.25	$105						-		-	-	-	-	-	
2000 KVA C/W MOTOR STARTERS	1	EA		200.00	1.25	$105	1500.0	13,750.0	275,000			250	26,250	13,750	275,000	1,500	-	316,500	
					1.25	$105						-		-	-	-	-	-	
LOCAL TECK POWER CABLE	1	lot		75.00	1.25	$105		9,000.0				94	9,844	9,000	-	-	-	18,844	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
Grounding					1.25	$105						-		-	-	-	-	-	
Ground wire, 2/0 AWG	250	m		0.07	1.25	$105	$2.0	7.00				22	2,297	1,750	-	500	-	4,547	
Ground Connectors, compression	5	ea		0.75	1.25	$105		30.00				5	492	150	-	-	-	642	
Ground wire, 4/0	200	m		0.1	1.25	$105	$2.0	11.00				25	2,625	2,200	-	400	-	5,225	
Ground Connectors, compression	4	ea		0.75	1.25	$105		35.0				4	394	140	-	-	-	534	
					1.25	$105						-		-	-	-	-	-	
CONTROL (ALARM) CONNECTIONS	1	LOT		200.00	1.25	$105	1000.0	10,000.0	10,000			250	26,250	10,000	10,000	1,000	-	47,250	
MISC. BOXES, CABLES,	1	LOT		250.00	1.25	$105	1500.0	12,000.0	5,000										
TERMINATIONS, ETC.												313	32,813	12,000	5,000	1,500	-	51,313	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
⋮												-					-	-	Area Subtotal
SUBTOTAL COST CODE 16, MAIN AIR COMPRESSORS POWER & CONTROL											-	1,087	114,089	48,990	403,000	5,650	-		$571,729
												-		-			-	-	
COST CODE 17 MINE COMMUNICATIONS SYSTEM (LEAKY FEEDER & PED)												-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
NOTE, THE DETAILED ESTIMATES ARE INCLUDED IN THE REPORT WRITE-UP.					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
LEAKY FEEDER SYSTEM (BASIC INITIAL INSTALLATION)	1	LOT		6600.00	1.00	$110	5000.0	607,807.0	366,746			6,600	726,000	607,807	366,746	5,000	-	1,705,553	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
PED MATERIAL & EQUIPMENT	1	LOT		1500.00	1.25	$105	5000.0		556,386			1,875	196,875	-	556,386	5,000	-	758,261	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 17, MINE COMMUNICATIONS:											-	8,475	922,875	607,807	923,132	10,000	-		$2,463,814
Direct costs include project contractor profit																			
												MH	C$	C$	C$	C$	C$	C$	C$
SUBTOTAL, DIRECT COSTS											0	113,715	12,064,983	9,933,146	19,283,075	389,937	198,000	40,314,490	$40,350,678
																			(Checksum) 41,869,140
COST CODE 17 ENGINEERING, CONSTRUCTION MANAGEMENT AND SPARES												-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	

Description	Quantity	Units	Unit Weight kg	Unit MH	Productivity U/G Multi-plier	Labour Rate $105	Equip Rental Unit $	Material Unit Cost $	Equipment Unit Cost $	Subcont. Unit Rate	Weight Total	MHrs Total	Labour Total Cost	Material Total Cost $	Equipment Total Cost $	Equipment Rental Total Cost $	Subcontract Total Cost $	Total Direct Cost	Area Sub-Totals
SPARE PARTS	5.0%	OF EQUIPMENT			1.25	$105						-		-	625,000	-	-	625,000	
					1.25	$105						-		-	-	-	-	-	
CONSTRUCTION MANAGEMENT					1.25	$105						-		-	-	-	-	-	
					1.25	$105													
(Note special (average) labour rates)												-		-	-	-	-	-	
Construction Offices & Office Equipment, communications, vehicle rental etc.	24	mth			1.25	$105	12,500	2,500	4,000			-		60,000	96,000	300,000	-	456,000	
					1.25	$105						-		-	-	-	-	-	
Construction Management FOR	20	mth			1.25	$150.0	7,500	10,000											
electrical & inst.				800								20,000	3,000,000	200,000	-	150,000	-	3,350,000	
QA/QC, Electrical & Instrumentation	20	mth		200	1.25	$125.0	3,500	5,000				5,000	625,000	100,000	-	70,000	-	795,000	
Monitoring of electrical construction					1.25	$105						-		-	-	-	-	-	
COMMISSIONING					1.25	$105						-		-	-	-	-	-	
Vendor Reps (in addition to proposal)	1	lot			1.25	$105				200,000		-		-	-	-	200,000	200,000	
Commissioning personnel	3	mth		400	1.25	$150.0	5,000	7,500				1,500	225,000	22,500	-	15,000	-	262,500	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
ENGINEERING (POWER)					1.25	$105						-		-	-	-	-	-	
	1	lot			1.25	$105				1,900,000									
System electrical and instrument design												-		-	-	-	1,900,000	1,900,000	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-	-	-	-	-	
					1.25	$105						-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 17, ENGINEERING, CONSTRUCTION MANAGEMENT AND SPARES											-	26,500	3,850,000	382,500	721,000	535,000	2,100,000		$7,588,500

NOTES

- FREIGHT IS INCLUDED IN EQUIPMENT COSTS
- OVERHEAD AND PROFIT INCLUDED

EXCLUSIONS

- OWNER'S COSTS
- ENVIRONMENTAL COSTS
- CONTINGENCY (15% WOULD BE SUGGESTED)
- SUSTAINING CAPITAL

TOTAL CONSTRUCTION COST (DIRECT & INDIRECT)	0.00	$47,902,990	$47,939,178

APPENDIX F

Development and PCBC Production Schedule (including grade and tonnes) by Year

DEVELOPMENT SCHEDULE

	Project Year		1	2	3	4	5	6	7	8	9	10	11	12	13
	Year		0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name															
Conveyors															
Conveyors		3,258	157.501	1236.15	1864.17						0	0	0	0	0
Subtotal (Inhouse)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	15%	**$20,117,045**	$972,569	$7,633,226	$11,511,250	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$3,017,557***	$145,885	$1,144,984	$1,726,687	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
MTT Access															
MTT Access		3,430	1,837.50	1,592.17	0.00	0.00	0.00	0.00		0.00	0.00	0.00	0.00	0.00	0.00
Subtotal (Inhouse)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	15%	**$21,178,194**	$11,346,556	$9,831,637	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$3,176,729***	$1,701,983	1,474,745.61	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawpoints															
Drawpoints		16,424	0	0	0	0	1442.176	2540.198	8143.478	4297.736	0	0	0	0	
Subtotal (Inhouse)	23%	**$41,626,754**	$0	$0	$0	$0	$3,655,298	$6,438,313	$20,640,225	$10,892,918	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$9,709,769***	$0	$0	$0	$0	$852,627	$1,501,787	$4,814,495	$2,540,859	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Exctraction Drifts															
Exctraction Drifts		9,584	0	0	0	1096.667	7036.85	1359.39	91.454	0	0	0	0	0	0
Subtotal (Inhouse)	25%	**$39,384,026**	$0	$0	$0	$4,506,421	$28,915,802	$5,586,001	$375,802	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$9,704,190***	$0	$0	$0	$1,110,378	$7,124,829	$1,376,386	$92,597	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Ventilation Drifts															
Exhaust Tunnels		1,612	0	0	735.097	876.744									
Fresh Air Tunnels		3,864	53.889	0	0	1008.22	585	1630.445	586.573		0	0	0	0	
Subtotal (Inhouse)	21%	**$6,371,556**	$0	$0	$2,905,815	$3,465,741	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$1,352,445***	$0	$0	$616,797	$735,648	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Undercut Excavation															
Undercut Excavation		9,379	0	0	4835.224	530.224	0	2658.006	1355.479				0	0	0
Subtotal (Inhouse)	15%	**$11,736,868**	$0	$0	$6,050,836	$663,526	$0	$3,326,249	$1,696,257	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$1,789,821***	$0	$0	$922,726	$101,185	$0	$507,238	$258,672	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Preconditioning Level															
Preconditioning Level		7,157	0	0	0	5638.211	1519.247	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	15%	**$8,956,898**	$0	$0	$0	$7,055,700	$1,901,197	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$1,365,888***	$0	$0	$0	$1,075,963	$289,924	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Perimeter															
Perimeter		2,258	0	0	188.9	1357.201	216.736	495.264		0	0	0	0	0	0
Subtotal (Inhouse)	25%	**$9,278,982**	$0	$0	$776,227	$5,577,006	$890,611	$2,035,137	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$2,286,333***	$0	$0	$191,262	$1,374,170	$219,446	$501,456	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Shop															
Shop		619	0	325.749	293.143							0	0	0	0
Subtotal (Inhouse)	22%	**$2,076,247**	$0	$1,092,816	$983,430	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$447,051***	$0	$235,302	$211,749	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Internal Ramps															

DEVELOPMENT SCHEDULE

		Project Year	1	2	3	4	5	6	7	8	9	10	11	12	13
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name															
Internal Ramps		3,107	0.00	455.93	1,184.60	106.44	0.00	1,212.84	146.89	0.00	0.00	0.00	0.00	0.00	0.00
Subtotal (Inhouse)	25%	***$12,766,001***	$0	$1,873,518	$4,867,742	$437,362	$0	$4,983,790	$603,588	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$3,145,532***	$0	$461,633	$1,199,408	$107,766	$0	$1,228,002	$148,723	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Vertical Development															
RAR Raise		0													
FAR Raises		0													
Internal Ventilation Raise		1,535	114.409	0	20.11	140	1050	210		0	0	0	0		
Crusher Chambers		375	215.369	0	0	80	0	80		0	0	0	0	0	0
Subtotal (Inhouse)	25%	***$11,947,478***	$2,620,632	$0	$111,160	$1,512,399	$5,803,958	$1,899,330	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$2,973,939***	$649,852	$0	$27,734	$375,860	$1,448,094	$472,399	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawbell and Drawpoint Blasting															
Drawbell and Drawpoint Blasting		1,015					120	120	120	120	120	120	120	120	55
Subtotal (Inhouse)	24%	***$79,246,773***	$0	$0	$0	$0	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$4,294,160
Subtotal (Contractor)	0%	***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$19,226,105***	$0	$0	$0	$0	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$1,041,809
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Undercut Blasting															
Undercut Blasting		9,379	0	0	4835	530	0	2658	1355	0	0	0	0	0	0
Subtotal (Inhouse)	15%	***$5,929,020***	$0	$0	$3,056,653	$335,188	$0	$1,680,295	$856,885	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$889,353***	$0	$0	$458,498	$50,278	$0	$252,044	$128,533	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Pre-conditioning - Hydrofracturing															
Pre-conditioning - Hydrofracturing		7,157	0	0	0	5638	1519	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	15%	**$55,830**	$0	$0	$0	$43,980	$11,851	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$8,375***	$0	$0	$0	$6,597	$1,778	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL LATERAL DEVELOPMENT METERS		**60,691**	2,049	3,610	9,101	10,614	10,800	9,896	10,324	4,298	0	0	0	0	0
TOTAL VERTICAL DEVELOPMENT METERS		**1,910**	330	0	20	220	1,050	290	0	0	0	0	0	0	0
TOTAL REHABILITATION METERS (opex cost)	10%	**6,069**						205	361	910	1,061	1,080	990	1,032	430
TOTAL CONTINGENCY DEVELOPMENT	5%	**3,035**	102	181	455	531	540	495	516	215	0	0	0	0	0
TOTAL UNDERCUT METERS		**9,379**	0	0	4,835	530	0	2,658	1,355	0	0	0	0	0	0
TOTAL DRAWBELLS		**1,015**	0	0	0	0	120	120	120	120	120	120	120	120	55
TOTAL DEVELOPMENT ORE TONNES		**2,583,293**	47,187	63,147	330,990	458,117	630,493	449,651	420,946	182,761	0	0	0	0	0
TOTAL DEVELOPMENT INCREMENTAL TONNES	do not use	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT WASTE TONNES		**680,692**	134,132	186,746	229,125	130,690	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT TONNES		**3,263,985**	181,319	249,892	560,115	588,807	630,493	449,651	420,946	182,761	0	0	0	0	0
TOTAL CONTAINED AU		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL INHOUSE COST	23%	**$231,041,845**	$2,748,832	$2,099,398	$18,337,894	$24,217,448	$51,211,703	$35,937,397	$34,187,802	$20,530,906	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$4,294,160
TOTAL CONTRACTOR COST	15%	**$41,295,238**	$12,319,125	$17,464,864	$11,511,250	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	22%	**$272,337,084**	$15,067,957	$19,564,261	$29,849,144	$24,217,448	$51,211,703	$35,937,397	$34,187,802	$20,530,906	$9,369,077	$9,369,077	$9,369,077	$9,369,077	$4,294,160
TOTAL INHOUSE CAPITAL CONTIGENCY		**$52,443,375**	$649,852	$461,633	$3,416,424	$4,931,248	$12,207,957	$8,112,350	$7,716,058	$4,813,896	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$1,041,809
TOTAL CONTRACTOR CAPITAL CONTIGENCY		**$6,194,286**	$1,847,869	$2,619,730	$1,726,687	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL CONTIGENCY		**$58,637,661**	$2,497,721	$3,081,363	$5,143,112	$4,931,248	$12,207,957	$8,112,350	$7,716,058	$4,813,896	$2,273,037	$2,273,037	$2,273,037	$2,273,037	$1,041,809

Iron Cap Production Schedule - March 26, 2012

Period	Current tons	To date	Dil %	AUIDW	CUIDW	MOIDW	AGIDW	AUREC	CUREC	MOREC	AGREC	NSR	AUEQ
Yr1	922,184	922,184	0.0	0.66	0.16	19.58	11.65	67.35	79.51	7.76	78.49	32.32	1.064055
Yr2	3,164,686	4,086,870	0.0	0.64	0.16	17.70	8.58	66.76	78.48	6.00	72.86	30.03	1.045221
Yr3	6,194,987	10,281,857	0.0	0.59	0.16	17.84	5.93	65.82	77.72	5.73	67.98	27.07	0.997467
Yr4	10,248,411	20,530,268	0.0	0.54	0.17	18.26	5.06	65.23	78.02	6.06	66.21	25.91	0.978925
Yr5	15,000,000	35,530,268	0.5	0.53	0.19	18.35	5.38	64.21	78.47	5.85	66.63	27.17	1.023381
Yr6	14,999,999	50,530,268	5.0	0.52	0.21	19.49	5.72	61.11	76.78	6.36	65.54	28.13	1.051452
Yr7	14,999,998	65,530,264	7.6	0.50	0.20	19.68	5.01	58.44	74.12	6.67	62.31	26.85	1.014466
Yr8	14,999,998	80,530,264	7.4	0.49	0.20	19.06	4.61	57.52	72.90	6.30	61.02	26.16	0.995281
Yr9	14,999,999	95,530,264	5.3	0.48	0.20	20.28	4.62	57.77	73.59	6.83	61.68	25.78	0.982159
Yr10	14,999,999	110,530,264	5.1	0.45	0.20	21.03	4.75	57.53	73.58	7.19	61.78	25.33	0.964196
Yr11	14,999,999	125,530,264	4.1	0.43	0.21	23.68	4.78	58.40	75.45	8.41	62.84	25.15	0.961335
Yr12	15,000,000	140,530,256	4.4	0.40	0.21	25.80	4.95	58.59	76.26	9.26	63.96	24.65	0.94038
Yr13	15,000,000	155,530,256	5.6	0.36	0.20	26.87	5.29	57.79	74.99	9.21	64.33	23.37	0.882774
Yr14	12,962,386	168,492,640	6.8	0.34	0.20	23.91	5.60	57.70	74.24	8.13	65.02	22.51	0.842492
Yr15	10,106,065	178,598,720	7.9	0.34	0.20	22.55	5.92	57.99	74.45	8.02	65.63	22.56	0.836835
Yr16	6,969,435	185,568,144	10.2	0.35	0.19	22.70	6.18	57.56	73.72	8.54	65.82	22.75	0.835968
Yr17	4,520,357	190,088,496	13.0	0.34	0.18	21.93	6.30	56.06	71.51	8.33	65.51	21.91	0.796964
Yr18	2,245,305	192,333,808	13.0	0.35	0.16	21.83	6.49	56.79	71.10	8.73	68.07	21.38	0.770326
Yr19	891,290	193,225,104	17.5	0.33	0.15	21.13	6.16	54.02	66.84	9.81	65.72	19.56	0.701647
Yr20	135,098	193,360,192	25.4	0.29	0.13	18.63	5.23	48.77	60.64	8.13	58.88	17.17	0.62332
	193,360,195			0.45	0.20	21.47	5.32						

APPENDIX G

Life of Mine Capital and Operating Cost Schedule

COST SCHEDULE														
			Project Year	1	2	3	4	5	6	7	8	9	10	11
			Year	0	0	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**													
Production Ore	tonnes		193,360,195	0	0	0	0	0	922,184	3,164,686	6,194,987	10,248,411	15,000,000	14,999,999
Production Incremental	tonnes		0	0	0	0	0	0	0	0	0	0	0	0
Production Waste	tonnes		0	0	0	0	0	0	0	0	0	0	0	0
Production Total	tonnes		193,360,195	0	0	0	0	0	922,184	3,164,686	6,194,987	10,248,411	15,000,000	14,999,999
NSR ($/t)	$/t			0.00	0.00	0.00	0.00	0.00	32.32	30.03	27.07	25.91	27.17	28.13
AUGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	0.66	0.64	0.59	0.54	0.53	0.52
CU%	%			0.00%	0.00%	0.00%	0.00%	0.00%	15.94%	15.76%	16.00%	17.12%	19.13%	20.81%
AGGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	11.65	17.70	17.84	18.26	18.35	19.49
MOPPM (ppm)	ppm			0.00	0.00	0.00	0.00	0.00	19.58	8.58	5.93	5.06	5.38	5.72
Lateral Development	m		60,691	2,049	3,610	9,101	10,614	10,800	9,896	10,324	4,298	0	0	0
Vertical Development	m		1,910	330	0	20	220	1,050	290	0	0	0	0	0
Total Development	m		62,601	2,379	3,610	9,121	10,834	11,850	10,186	10,324	4,298	0	0	0
CAPITAL COSTS - CAPEX	**Accuracy (%)**													
In house														
Development Equipment		20%	$22,641,816	$4,668,842	$3,116,214	$5,818,381	$1,982,484	$800,293	$2,935,832	$986,000	$784,987	$381,898	$784,987	$381,898
Production Equipment		20%	$77,406,950	$0	$0	$0	$0	$0	$0	$7,434,617	$8,584,156	$4,598,156	$3,448,617	$0
Support Equipment		20%	$14,649,678	$295,468	$1,126,408	$0	$2,432,102	$0	$295,468	$1,344,368	$2,627,343	$295,468	$0	$0
Stationary Equipment		20%	$91,349,934	$1,320,000	$0	$2,107,500	$2,000,000	$0	$0	$15,704,984	$18,882,484	$3,500,000	$4,820,000	$3,715,000
Additional Equipment/Infrastructure		25%	$126,736,490	$53,024,214	$300,000	$1,996,600	$0	$96,600	$96,600	$0	$34,269,262	$0	$0	$0
Contractor														
Development Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CAPITAL COSTS		22%	$332,784,868	$59,308,524	$4,542,622	$9,922,481	$6,414,586	$896,893	$3,327,900	$25,469,969	$65,148,232	$8,775,522	$9,053,604	$4,096,898
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		22%	$332,784,868	$59,308,524	$4,542,622	$9,922,481	$6,414,586	$896,893	$3,327,900	$25,469,969	$65,148,232	$8,775,522	$9,053,604	$4,096,898
DEVELOPMENT COSTS - CAPEX	**Accuracy (%)**													
Fixed Development Costs														
In house		21%	$13,147,496	$717,823	$1,698,403	$2,916,587	$3,907,341	$3,907,341	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs														
In house		24%	$233,118,092	$2,748,832	$3,192,214	$19,321,325	$24,217,448	$51,211,703	$35,937,397	$34,187,802	$20,530,906	$9,369,077	$9,369,077	$9,369,077
Contractor		25%	$41,295,238	$12,319,125	$17,464,864	$11,511,250	$0	$0	$0	$0	$0	$0	$0	$0
Labour														
In house		25%	$348,085,151	$13,045,336	$23,609,288	$41,027,873	$48,729,487	$48,729,487	$48,729,487	$48,729,487	$21,398,362	$9,014,391	$9,014,391	$9,014,391
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		24%	$594,350,739	$16,511,991	$28,499,905	$63,265,785	$76,854,276	$103,848,531	$84,666,884	$82,917,289	$41,929,267	$18,383,468	$18,383,468	$18,383,468
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$41,295,238	$12,319,125	$17,464,864	$11,511,250	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		24%	$635,645,978	$28,831,116	$45,964,769	$74,777,035	$76,854,276	$103,848,531	$84,666,884	$82,917,289	$41,929,267	$18,383,468	$18,383,468	$18,383,468
PRODUCTION COSTS - OPEX	**Accuracy (%)**													
Fixed Production Costs														
In house		21%	$102,131,975	$0	$0	$0	$0	$0	$3,907,341	$4,816,729	$5,267,949	$5,591,594	$5,834,328	$5,834,328
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			9%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	9%	9%	9%	8%	7%	7%
Variable Production Costs														
In house		23%	$420,942,450	$0	$0	$0	$0	$0	$2,227,759	$7,216,929	$14,381,990	$23,236,308	$33,417,129	$33,304,016
Percentage			35%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	5%	14%	24%	33%	40%	39%
Labour														
In house		25%	$666,213,600	$0	$0	$0	$0	$0	$39,325,139	$39,325,139	$40,636,849	$42,604,415	$44,571,981	$46,539,546
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			56%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	87%	77%	67%	60%	53%	54%
TOTAL IN HOUSE PRODUCTION COSTS		24%	$1,189,288,025	$0	$0	$0	$0	$0	$45,460,239	$51,358,797	$60,286,789	$71,432,317	$83,823,437	$85,677,890
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$1,189,288,025	$0	$0	$0	$0	$0	$45,460,239	$51,358,797	$60,286,789	$71,432,317	$83,823,437	$85,677,890
TOTAL COSTS	**Accuracy (%)**													
TOTAL IN HOUSE COST		24%	**$2,116,423,633**	$75,820,515	$33,042,527	$73,188,266	$83,268,862	$104,745,424	$133,455,023	$159,746,054	$167,364,288	$98,591,307	$111,260,509	$108,158,256
TOTAL CONTRACTOR COST		25%	**$41,295,238**	$12,319,125	$17,464,864	$11,511,250	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST		24%	**$2,157,718,871**	$88,139,640	$50,507,391	$84,699,516	$83,268,862	$104,745,424	$133,455,023	$159,746,054	$167,364,288	$98,591,307	$111,260,509	$108,158,256
TOTAL IN HOUSE CONTINGENCY			**$504,430,058**	$18,071,121	$7,875,382	$17,443,748	$19,846,365	$24,965,106	$31,807,774	$38,073,999	$39,889,735	$23,498,329	$26,517,916	$25,778,523
TOTAL CONTRACTOR CONTINGENCY			**$10,323,810**	$3,079,781	$4,366,216	$2,877,812	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY			**$514,753,868**	$21,026,938	$12,049,241	$20,206,248	$19,864,946	$24,988,479	$31,837,553	$38,109,645	$39,927,080	$23,520,328	$26,542,743	$25,802,657
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$13.82**	-	-	-	-	-	$144.72	$50.48	$27.02	$9.62	$7.42	$7.21
TOTAL COST PER TONNE (ORE) - Operating			**$6.15**	-	-	-	-	-	$49.30	$16.23	$9.73	$6.97	$5.59	$5.71

COST SCHEDULE														
			Project Year		12	13	14	15	16	17	18	19	20	21	22
			Year		0	0	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**													
Production Ore	tonnes		193,360,195	14,999,998	14,999,998	14,999,999	14,999,999	14,999,999	15,000,000	15,000,000	12,962,386	10,106,065	6,969,435	4,520,357
Production Incremental	tonnes		0	0	0	0	0	0	0	0	0	0	0	0
Production Waste	tonnes		0	0	0	0	0	0	0	0	0	0	0	0
Production Total	tonnes		193,360,195	14,999,998	14,999,998	14,999,999	14,999,999	14,999,999	15,000,000	15,000,000	12,962,386	10,106,065	6,969,435	4,520,357
NSR ($/t)	$/t			26.85	26.16	25.78	25.33	25.15	24.65	23.37	22.51	22.56	22.75	21.91
AUGPT (gpt)	gpt			0.50	0.49	0.48	0.45	0.43	0.40	0.36	0.34	0.34	0.35	0.34
CU%	%			20.25%	19.75%	19.75%	20.01%	20.79%	21.07%	20.34%	19.66%	19.53%	19.13%	17.85%
AGGPT (gpt)	gpt			19.68	19.06	20.28	21.03	23.68	25.80	26.87	23.91	22.55	22.70	21.93
MOPPM (ppm)	ppm			5.01	4.61	4.62	4.75	4.78	4.95	5.29	5.60	5.92	6.18	6.30
Lateral Development	m		60,691	0	0	0	0	0	0	0	0	0	0	0
Vertical Development	m		1,910	0	0	0	0	0	0	0	0	0	0	0
Total Development	m		62,601	0	0	0	0	0	0	0	0	0	0	0
CAPITAL COSTS - CAPEX		**Accuracy (%)**												
In house														
Development Equipment		20%	$22,641,816	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment		20%	$77,406,950	$7,434,617	$8,584,156	$4,598,156	$3,448,617	$0	$3,448,617	$8,584,156	$4,598,156	$3,448,617	$0	$4,598,156
Support Equipment		20%	$14,649,678	$1,126,408	$743,468	$2,136,634	$0	$578,900	$904,175	$743,468	$0	$0	$0	$0
Stationary Equipment		20%	$91,349,934	$107,500	$30,764,967	$0	$0	$0	$0	$3,607,500	$3,500,000	$1,320,000	$0	$0
Additional Equipment/Infrastructure		25%	$126,736,490	$0	$0	$26,953,214	$0	$0	$0	$0	$0	$0	$0	$0
Contractor														
Development Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CAPITAL COSTS		22%	$332,784,868	$8,668,525	$40,092,591	$33,688,004	$3,448,617	$578,900	$4,352,792	$12,935,124	$8,098,156	$4,768,617	$0	$4,598,156
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		22%	$332,784,868	$8,668,525	$40,092,591	$33,688,004	$3,448,617	$578,900	$4,352,792	$12,935,124	$8,098,156	$4,768,617	$0	$4,598,156
DEVELOPMENT COSTS - CAPEX		**Accuracy (%)**												
Fixed Development Costs														
In house		21%	$13,147,496	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs														
In house		24%	$233,118,092	$9,369,077	$4,294,160	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Labour														
In house		25%	$348,085,151	$9,014,391	$9,014,391	$9,014,391	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		24%	$594,350,739	$18,383,468	$13,308,551	$9,014,391	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		24%	$635,645,978	$18,383,468	$13,308,551	$9,014,391	$0	$0	$0	$0	$0	$0	$0	$0
PRODUCTION COSTS - OPEX		**Accuracy (%)**												
Fixed Production Costs														
In house		21%	$102,131,975	$5,834,328	$5,834,328	$5,827,018	$5,827,018	$5,827,018	$5,827,018	$5,616,719	$5,616,719	$5,609,409	$5,609,409	$5,609,409
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			9%	7%	7%	7%	7%	7%	8%	8%	8%	9%	12%	29%
Variable Production Costs														
In house		23%	$420,942,450	$33,357,540	$32,603,425	$32,065,605	$32,065,605	$32,065,605	$32,065,607	$32,065,607	$27,709,785	$21,603,807	$14,898,611	$9,663,198
Percentage			35%	38%	38%	37%	39%	39%	43%	43%	39%	34%	31%	50%
Labour														
In house		25%	$666,213,600	$47,851,257	$47,851,257	$47,851,257	$43,455,947	$43,455,947	$36,831,810	$36,831,810	$36,831,810	$36,831,810	$28,247,860	$3,906,316
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			56%	55%	55%	56%	53%	53%	49%	49%	52%	58%	58%	20%
TOTAL IN HOUSE PRODUCTION COSTS		24%	$1,189,288,025	$87,043,125	$86,289,010	$85,743,879	$81,348,570	$81,348,570	$74,724,434	$74,514,136	$70,158,314	$64,045,026	$48,755,880	$19,178,923
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$1,189,288,025	$87,043,125	$86,289,010	$85,743,879	$81,348,570	$81,348,570	$74,724,434	$74,514,136	$70,158,314	$64,045,026	$48,755,880	$19,178,923
TOTAL COSTS		**Accuracy (%)**												
TOTAL IN HOUSE COST		24%	**$2,116,423,633**	$114,095,117	$139,690,152	$128,446,274	$84,797,187	$81,927,470	$79,077,226	$87,449,260	$78,256,470	$68,813,643	$48,755,880	$23,777,079
TOTAL CONTRACTOR COST		25%	**$41,295,238**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST		24%	**$2,157,718,871**	$114,095,117	$139,690,152	$128,446,274	$84,797,187	$81,927,470	$79,077,226	$87,449,260	$78,256,470	$68,813,643	$48,755,880	$23,777,079
TOTAL IN HOUSE CONTINGENCY			**$504,430,058**	$27,193,519	$33,293,860	$30,613,985	$20,210,628	$19,526,657	$18,847,328	$20,842,725	$18,651,708	$16,401,097	$11,620,514	$5,667,048
TOTAL CONTRACTOR CONTINGENCY			**$10,323,810**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY			**$514,753,868**	$27,218,978	$33,325,030	$30,642,646	$20,229,549	$19,544,938	$18,864,973	$20,862,238	$18,669,170	$16,416,452	$11,631,394	$5,672,353
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$13.82**	$7.61	$9.31	$8.56	$5.65	$5.46	$5.27	$5.83	$6.04	$6.81	$7.00	$5.26
TOTAL COST PER TONNE (ORE) - Operating			**$6.15**	$5.80	$5.75	$5.72	$5.42	$5.42	$4.98	$4.97	$5.41	$6.34	$7.00	$4.24

COST SCHEDULE																
		Project Year	23	24	25	26	27	28	29	30	31	32	33	34	35	36
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**															
Production Ore	tonnes	193,360,195	2,245,305	891,290	135,098	0	0	0	0	0	0	0	0	0	0	0
Production Incremental	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Production Waste	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Production Total	tonnes	193,360,195	2,245,305	891,290	135,098	0	0	0	0	0	0	0	0	0	0	0
NSR ($/t)	$/t		21.38	19.56	17.17	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
AUGPT (gpt)	gpt		0.35	0.33	0.29	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
CU%	%		16.30%	14.51%	13.14%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
AGGPT (gpt)	gpt		21.83	6.16	5.23	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MOPPM (ppm)	ppm		6.49	21.13	18.63	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Lateral Development	m	60,691	0	0	0	0	0	0								
Vertical Development	m	1,910	0	0	0	0	0	0								
Total Development	m	62,601	0	0	0	0	0	0								
CAPITAL COSTS - CAPEX	**Accuracy (%)**															
In house																
Development Equipment	20%	$22,641,816	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment	20%	$77,406,950	$4,598,156	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment	20%	$14,649,678	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment	20%	$91,349,934	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Additional Equipment/Infrastructure	25%	$126,736,490	$0	$0	$0	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor																
Development Equipment	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CAPITAL COSTS	22%	$332,784,868	$4,598,156	$0	$0	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CAPITAL COSTS	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS	22%	$332,784,868	$4,598,156	$0	$0	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
DEVELOPMENT COSTS - CAPEX	**Accuracy (%)**															
Fixed Development Costs																
In house	21%	$13,147,496	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs																
In house	24%	$233,118,092	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor	25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Labour																
In house	25%	$348,085,151	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS	24%	$594,350,739	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR DEVELOPMENT COSTS	25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS	24%	$635,645,978	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
PRODUCTION COSTS - OPEX	**Accuracy (%)**															
Fixed Production Costs																
In house	21%	$102,131,975	$5,146,624	$1,816,150	$878,540	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage		9%	44%	36%	71%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!					
Variable Production Costs																
In house	23%	$420,942,450	$4,799,803	$1,905,317	$288,800	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage		35%	41%	38%	23%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!					
Labour																
In house	25%	$666,213,600	$1,868,605	$1,324,105	$70,743	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage		56%	16%	26%	6%	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!	#DIV/0!					
TOTAL IN HOUSE PRODUCTION COSTS	24%	$1,189,288,025	$11,815,033	$5,045,573	$1,238,083	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR PRODUCTION COSTS	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS	24%	$1,189,288,025	$11,815,033	$5,045,573	$1,238,083	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COSTS	**Accuracy (%)**															
TOTAL IN HOUSE COST	24%	**$2,116,423,633**	$16,413,189	$5,045,573	$1,238,083	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR COST	25%	**$41,295,238**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	24%	**$2,157,718,871**	$16,413,189	$5,045,573	$1,238,083	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CONTINGENCY		**$504,430,058**	$3,911,932	$1,202,566	$295,086	$2,383,408	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CONTINGENCY		**$10,323,810**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY		**$514,753,868**	$3,915,595	$1,203,692	$295,362	$2,385,639	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST PER TONNE (ORE) - Capital & Operating		**$13.82**	$7.31	$5.66	$9.16	-	-	-	-	-	-	-	-	-	-	-
TOTAL COST PER TONNE (ORE) - Operating		**$6.15**	$5.26	$5.66	$9.16	-	-	-	-	-	-	-	-	-	-	-

COST SCHEDULE												
			Project Year	37	38	39	40	41	42	43	44	45
			Year	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**											
Production Ore	tonnes		193,360,195	0	0	0	0	0	0	0	0	0
Production Incremental	tonnes		0	0	0	0	0	0	0	0	0	0
Production Waste	tonnes		0	0	0	0	0	0	0	0	0	0
Production Total	tonnes		193,360,195	0	0	0	0	0	0	0	0	0
NSR ($/t)	$/t			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
AUGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
CU%	%			0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
AGGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MOPPM (ppm)	ppm			0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Lateral Development	m		60,691									
Vertical Development	m		1,910									
Total Development	m		62,601									
CAPITAL COSTS - CAPEX		**Accuracy (%)**										
In house												
Development Equipment		20%	$22,641,816	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment		20%	$77,406,950	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment		20%	$14,649,678	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment		20%	$91,349,934	$0	$0	$0	$0	$0	$0	$0	$0	$0
Additional Equipment/Infrastructure		25%	$126,736,490	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor												
Development Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Support Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Stationary Equipment		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CAPITAL COSTS		22%	$332,784,868	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		22%	$332,784,868	$0	$0	$0	$0	$0	$0	$0	$0	$0
DEVELOPMENT COSTS - CAPEX		**Accuracy (%)**										
Fixed Development Costs												
In house		21%	$13,147,496	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs												
In house		24%	$233,118,092	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0
Labour												
In house		25%	$348,085,151	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		24%	$594,350,739	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$41,295,238	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		24%	$635,645,978	$0	$0	$0	$0	$0	$0	$0	$0	$0
PRODUCTION COSTS - OPEX		**Accuracy (%)**										
Fixed Production Costs												
In house		21%	$102,131,975	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			9%									
Variable Production Costs												
In house		23%	$420,942,450	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			35%									
Labour												
In house		25%	$666,213,600	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			56%									
TOTAL IN HOUSE PRODUCTION COSTS		24%	$1,189,288,025	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$1,189,288,025	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COSTS		**Accuracy (%)**										
TOTAL IN HOUSE COST		24%	**$2,116,423,633**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR COST		25%	**$41,295,238**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST		24%	**$2,157,718,871**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CONTINGENCY			**$504,430,058**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CONTINGENCY			**$10,323,810**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY			**$514,753,868**	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$13.82**	-	-	-	-	-	-	-	-	-
TOTAL COST PER TONNE (ORE) - Operating			**$6.15**	-	-	-	-	-	-	-	-	-

APPENDIX H

Detailed Breakdown of Non-labour Operating Cost

NON LABOUR PRODUCTION OPERATING COSTS

PRODUCTION OPERATING COSTS SUMMARY

Equipmnent	Hourly Machine Capacity (tonnes)	Op Cost ($/tonne)	Accuracy (%)	$/hr
Production LHD	213	CAD 0.821	22%	214.52
Production Haul Truck	Not Used	Not Used	na	0.00
Crusher	639	CAD 0.649	25%	516.59
Conveyors	2556	CAD 0.391	25%	1250.00
Block Holer	291	CAD 0.161	20%	55.82
Mobile Rockbreaker		CAD 0.116	22%	0.00

Production Equipment

LHD

						CAD$/t	Accuracy	Contingency
Type:	Production LHD	Equipment:	Sandvik LH517					
Bucket Capacity	tonnes	21						
	m^3	8.6						
	Volumetric Fill Factor	90%						
	Swell factor	1.25						
	tonnes at volume	16.72	Uses ore density					
Loading time	mins	1						
Tipping Time	mins	1						
Tramming Time								
Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)					
Flat Laden	225	10	1.35					
Gradient Laden		5.8	0.00					
Flat/Down Gradient Empty	225	10	1.35					
Gradient Empty		7.5	0.00					
Total Tramming			2.70					
Total Cycle Time	mins	4.70						
Hourly Machine Capacity	tonnes/hour	213.43						
Number of Operating LHDs	each	12.00	80% mechanical availability					
Daily Machine Capacity	tonnes/day	46,000						
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
45	100%	45	CAD 45.00			CAD 0.21	15%	CAD 0.03
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	CAD 0.00
Maintenance Cost per hour								
CAD 130.22						CAD 0.61	25%	CAD 0.15
Operating Cost per tonne						**CAD 0.82**	22%	**CAD 0.18**

Haul Trucks

						CAD$/t	Accuracy	Contingency
Type:	n/a	Equipment:	#N/A					
Bucket Capacity	tonnes	80						
	m^3	37						
	Swell factor	1.25						
	tonnes at volume	79.92	Uses ore density					
Loading time	mins	2						
Tipping Time	mins	2						
Tramming Time								
Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)					
Flat Laden	3140	15	12.56					
Gradient Laden		6	0.00					
Flat/Down Gradient Empty		20	0.00					
Gradient Empty		14	0.00					
Total Tramming			12.56					
Total Cycle Time	mins	16.56						
Hourly Machine Capacity	tonnes/hour	Not Used						
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
#N/A	#N/A	#N/A	#N/A	Assumes Engine on when loading		#N/A	15%	#N/A
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	15%	#N/A
Maintenance Cost per hour								
#N/A						#N/A	#N/A	#N/A
Operating Cost per tonne						**Not Used**	na	**na**

Stationary Equipment

Crusher/Sizer

						CAD$/t	Accuracy	Contingency
Type:	Crusher	Equipment:	Sandvik CG820 (54x75)					
Nominal Throughput	tonnes/hr	638.8888889	2 shifts per day, 4 crushers operating					
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	15%	CAD 0.00
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
450	75%	95%	90%	289	CAD 14.43	CAD 0.02	15%	CAD 0.00
Maintenance Cost per hour								
CAD 400.00						CAD 0.63	25%	CAD 0.16
Operating Cost per tonne						**CAD 0.65**	25%	**CAD 0.16**

Conveyor 1

	CAD$/t	Accuracy	Contingency
Operating Cost per tonne	**CAD 0.39**	25%	**CAD 0.10**

NON LABOUR PRODUCTION OPERATING COSTS

PRODUCTION OPERATING COSTS SUMMARY

Equipmnent	Hourly Machine Capacity (tonnes)	Op Cost ($/tonne)	Accuracy (%)	$/hr
Production LHD	213	CAD 0.821	22%	214.52
Production Haul Truck	Not Used	Not Used	na	0.00
Crusher	639	CAD 0.649	25%	516.59
Conveyors	2556	CAD 0.391	25%	1250.00
Block Holer	291	CAD 0.161	20%	55.82
Mobile Rockbreaker		CAD 0.116	22%	0.00

Secondary Breaking Equipment

Block Holer

Type:	Block Holer	Equipment:	Maclean BH-3	
Number of Block Drawpoints per shift		15		
Volume of boulder	m3	51.66		
Density	t/m3	2.7		
Face area of boulder	m2	15.75		
Blast hole coverage	m2/hole	2		
Number of Blast Holes	holes/boulder	7		
Total number of holes per shift	holes	105		
Total drill meters per shift	m	344		
Nominal Throughput	tonnes/hr	291		

Drilling

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost	CAD $/shift	Accuracy	Contingency
Drill Bit:	Bit 45mm x R32	Item	1	900	CAD 93.38	CAD 35.73	*15%*	*CAD 5.36*
Rod:	Jumbo Drill Rod 4300 mm	Item	1	2500	CAD 416.82	CAD 57.42	*15%*	*CAD 8.61*
Coupling:	Coupling R38	Item	1	2500	CAD 67.95	CAD 9.36	*15%*	*CAD 1.40*
Shank Adaptor:	Shank Adapter T-38	Item	1	7000	CAD 598.87	CAD 29.46	*15%*	*CAD 4.42*
Reamer:	Reamer 102mm x R32	Item	0	1500	CAD 233.90	CAD 0.00	*15%*	*CAD 0.00*
Subtotal consumable cost for excavation per shift						**CAD 131.98**		***CAD 19.80***

Fuel/Electricity Costs

Type:	block holer	Equipment:	Maclean BH-3
Drilling Time Per Meter	mins	0.5	
Number of booms operating		1	
Estimated Total Drilling Time	mins	172.20	
Positioning time	mins	555	Currently estimated at 30 minutes move between setups + 1 minute per hole
Total operating time per round	mins	727.20	

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour	CAD $/shift	Accuracy	Contingency
20	25%	5	CAD 5.00	CAD 45.00	*15%*	*CAD 6.75*

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	CAD $/shift	Accuracy	Contingency
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	*15%*	*CAD 0.00*

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour	CAD $/shift	Accuracy	Contingency
CAD 158.60	CAD 1,427.40	*25%*	*CAD 356.85*

	CAD $/shift	Accuracy	Contingency
Subtotal fuel/electricity cost per shift	**CAD 1,472.40**		***CAD 363.60***
TOTAL DRILLING COST per shift	**CAD 1,604.38**		***CAD 383.40***
TOTAL DRILLING COST per tonne	**CAD 0.07**	24%	***CAD 0.02***

Blasting

Consumable Costs

Item	Type	Unit	Units required per shift	Life (if applicable)	Unit cost	CAD $/shift	Accuracy	Contingency
Bulk Explosive:	Bulk emulsion	kg	620		CAD 1.60	CAD 991.87	*15%*	*CAD 148.78*
Primer:	Pentex CD90	Item	105		CAD 2.96	CAD 310.80	*15%*	*CAD 46.62*
Detonator:	Exel LP 5m	Item	105	Assumes 1x detonators per	CAD 3.95	CAD 414.75	*15%*	*CAD 62.21*
Lead in Line:	Blasting Cable	m	105		CAD 0.68	CAD 71.26	*15%*	*CAD 10.69*
Stemming:	Stemming	tonne	0.16		CAD 27.45	CAD 4.46	*15%*	*CAD 0.67*
Subtotal consumable cost for excavation per shift						**CAD 1,793.14**		***CAD 268.97***

Fuel/Electricity Costs

Type:	ANFO Loader	Equipment:	Maclean AC-3	Used as a means to estimate costs. We won't be using this piece of equipment
Drum Capacity	m3	2		
Drums Required per excavation		1		
Pumping Rate	kg/min	60		
Powder Factor	kg/m3	0.8		
Estimated Pumping Time	mins	1		
Positioning time	mins	210	Currently estimates 2 minutes per hole	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	20	1	1.2
Gradient Laden		10	0
Flat/Down Gradient Empty	20	1	1.2
Gradient Empty		10	0
Total Tramming			2.4

Total Time per Shift — mins — 213

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour	CAD $/shift	Accuracy	Contingency
10	25%	2.5	CAD 2.50	CAD 22.50	*15%*	*CAD 3.38*

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	CAD $/shift	Accuracy	Contingency
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	*15%*	*CAD 0.00*

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour	CAD $/shift	Accuracy	Contingency
CAD 30.45	CAD 274.05	*25%*	*CAD 68.51*

	CAD $/shift	Accuracy	Contingency
Subtotal fuel/electricity cost per shift	**CAD 296.55**		***CAD 71.89***
TOTAL BLASTING COST per shift	**CAD 2,089.69**		***CAD 340.86***
TOTAL BLASTING COST per tonne	**CAD 0.09**		***CAD 340.86***
Operating Cost per shift	**CAD 3,694.07**	20%	***CAD 724.26***
Operating Cost per tonne	**CAD 0.16**	20%	***CAD 0.03***

Mobile Rockbreaker

				CAD$/shift	Accuracy	Contingency
Type:	Mobile Rockbreaker	Equipment:	Sandvik DB120			
Quantity of rockbreakers			2			

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour	CAD $/shift	Accuracy	Contingency
45	100%	45	CAD 45.00	CAD 405.00	*15%*	*CAD 60.75*

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	CAD $/shift	Accuracy	Contingency
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	*15%*	*CAD 0.00*

Maintenance Cost per hour	CAD $/shift	Accuracy	Contingency
CAD 103.41	CAD 930.69	*25%*	*CAD 232.67*

	CAD $/shift	Accuracy	Contingency
Operating Cost per shift	**CAD 2,671.38**	22%	***CAD 586.85***
Operating Cost per tonne	**CAD 0.12**	22%	***CAD 0.03***

APPENDIX I

Example of Detailed Cost Calculation for a Meter of Development

DEVELOPMENT REGIME

Regime Code:	D1 - P
Description:	5m x 5m Drive (Waste)

Excavation Dimensions		Blasts Details		Support Details	
Height (m)	5	Estimated number of holes	71	In-Row spacing (m)	1.2
Width (m)	5	Estimated number of reamer holes	4	Row separation (m)	1.2
Advance per round (m)	3.7	Estimated drill meters (m)	277.5	Side Wall Bolting	YES
Volume extracted (m3)	92.50	Estimated reamer meters (m)	14.8	Bolts per round	48
Rock Density (t/m3)	2.65	Hole Diameter (mm):	45	Resin cartridges per bolt	4
Tonnes extracted	245.13	Required Powder Factor (kg/m3):	2.5	Bolt Length (m)	2.4
Swell factor	1.25	Explosive Product Required (kg):	231	Estimated drill meters (m)	115.2
Volume hauled (m3)	115.63	Explosive Density (t/m3)	1.3	Shotcrete Thickness (mm)	0
		Stemming Density (t/m3):	1.6	Shotcrete Rebound Factor	1.2
				Shotcrete Volume per round (m3)	0
				Mesh required (Roof / Side)	ROOF+WALL
				Conrete floor required	NO
				Concrete floor thickness (m)	0.5

Drilling

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost	Cost for excavation	Accuracy	Contingency
Drill Bit:	Bit 45mm x R32	Item	1	900	CAD 93.38	CAD 28.79	15%	CAD 4.32
Rod:	Jumbo Drill Rod 4300 mm	Item	1	2500	CAD 416.82	CAD 46.27	15%	CAD 6.94
Coupling:	Coupling R38	Item	1	2500	CAD 67.95	CAD 7.54	15%	CAD 1.13
Shank Adaptor:	Shank Adapter T-38	Item	1	7000	CAD 598.87	CAD 23.74	15%	CAD 3.56
Reamer:	Reamer 102mm x R32	Item	1	1500	CAD 233.90	CAD 2.31	15%	CAD 0.35
Subtotal consumable cost for excavation						**CAD 108.65**		***CAD 16.30***

Fuel/Electricity Costs

Type:	Jumbo Drill Rig	Equipment:	DD420-60
Drilling Time Per Meter	mins	1	
Number of booms operating		2	
Estimated Total Drilling Time	mins	138.75	
Positioning time	mins	83.25	Currently estimated at 30 minutes + 0.75 minute per hole
Total operating time per round	mins	222.00	

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			Cost for excavation	Accuracy	Contingency
20	50%	10	CAD 10.00			CAD 37.00	15%	CAD 5.55

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	Cost for excavation	Accuracy	Contingency
150	75%	80%	90%	81	CAD 4.05	CAD 14.99	15%	CAD 2.25

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour	Cost for excavation	Accuracy	Contingency
CAD 95.16	CAD 352.09	15%	CAD 52.81

Subtotal operating cost for excavation	**CAD 404.08**	***CAD 60.61***
TOTAL DRILLING COST	**CAD 512.73**	***CAD 76.91***

Blasting

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost	Cost for excavation	Accuracy	Contingency
Bulk Explosive:	Bulk emulsion	kg	231		CAD 1.60	CAD 370.00	15%	CAD 55.50
Primer:	Pentex CD90	Item	142		CAD 2.96	CAD 420.32	15%	CAD 63.05
Detonator:	Exel LP 5m	Item	142	Assumes 2x detonators per hole	CAD 3.95	CAD 560.90	15%	CAD 84.14
Lead in Line:	Blasting Cable	m	100		CAD 0.68	CAD 67.86	15%	CAD 10.18
Stemming:	Stemming	tonne	0.16		CAD 27.45	CAD 4.46	15%	CAD 0.67
Subtotal consumable cost for excavation						**CAD 1,423.55**		***CAD 213.53***

Fuel/Electricity Costs

Type:	ANFO Loader	Equipment:	Maclean AC-3	Must be entered exactly as in 'SEA_FLEET_DETAILS WORKBOOK'
Drum Capacity	m3	2		
Drums Required per excavation		1		
Pumping Rate	kg/min	60		
Estimated Pumping Time	mins	4		
Positioning time	mins	142	Currently estimates 2 minutes per hole	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	200	10	1.2
Gradient Laden		10	0
Flat/Down Gradient Empty	200	10	1.2
Gradient Empty		10	0
Total Tramming			2.4

Total Time per Excavation mins 148

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			Cost for excavation	Accuracy	Contingency
10	25%	2.5	CAD 2.50			CAD 6.18	15%	CAD 0.93

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	Cost for excavation	Accuracy	Contingency
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	CAD 0.00

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour	Cost for excavation	Accuracy	Contingency
CAD 30.45	CAD 75.24	15%	CAD 11.29

Subtotal operating cost for excavation	**CAD 81.42**	***CAD 12.21***
TOTAL BLASTING COST	**CAD 1,504.96**	***CAD 225.74***

Ground Support

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost (CAD$)	Cost for excavation	Accuracy	Contingency
Bolter Drill Bit:	Bit 33 mm x R25	Item	1	900	CAD 68.48	CAD 8.77	15%	CAD 1.31
Bolter Drill Rod:	Jumbo Drill Rod 3700mm	Item	1	2500	CAD 227.42	CAD 10.48	15%	CAD 1.57
Bolter Coupling:	Coupling R32	Item	1	2500	CAD 33.80	CAD 1.56	15%	CAD 0.23
Bolter Shank Adaptor:	Shank Adapter T-38	Item	1	7000	CAD 598.87	CAD 9.86	15%	CAD 1.48
Rock Bolt:	2.4 metre x 22mm rebar bolt	Item	48		CAD 13.89	CAD 666.72	15%	CAD 100.01
Rock Bolt Plate:	Domed Plate 150 x 150 x 10 mm	Item	48		CAD 4.75	CAD 228.00	15%	CAD 34.20
Resin:	Resin Fast Set (28mm/250mm)	Cap	192		CAD 2.27	CAD 435.84	15%	CAD 65.38
Mesh:	Mesh 1200 x 3000	Item	16		CAD 25.95	CAD 415.26	15%	CAD 62.29
Shotcrete:	Shotcrete (with steel Fibres)	m^3	0		CAD 600.00	CAD 0.00	30%	CAD 0.00
Concrete Floor:	Concrete	m^3	0		CAD 616.00	CAD 0.00	30%	CAD 0.00
Subtotal cost for excavation						**CAD 1,776.47**		***CAD 266.47***

Fuel/Electricity Costs

Type:	Bolter	Equipment:	Maclean MEM928 HBM50	Must be entered exactly as in 'SEA_FLEET_DETAILS WORKBOOK'
Drilling Time Per Meter	mins	1.5		
Number of bolter booms operating		1		
Estimated Total Drilling Time	mins	172.80		
Time to install 1x bolt	mins	2		
Estimated Total Bolt Installation Time	mins	96		
Positioning time	mins	84	Currently estimated at 5 minutes between rows and 2 minutes between holes within a row	
Total operating time per round	mins	352.80		

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			Cost for excavation	Accuracy	Contingency
20	25%	5	CAD 5.00			CAD 29.40	15%	CAD 4.41

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour	Cost for excavation	Accuracy	Contingency
56	75%	80%	90%	30	CAD 1.51	CAD 8.89	15%	CAD 1.33

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour	Cost for excavation	Accuracy	Contingency
CAD 87.54	CAD 514.74	15%	CAD 77.21

Subtotal machine operating cost for excavation	**CAD 553.03**	***CAD 82.95***

DEVELOPMENT REGIME

Regime Code:	D1 - P
Description:	5m x 5m Drive (Waste)

Excavation Dimensions		Blasts Details		Support Details	
Height (m)	5	Estimated number of holes	71	In-Row spacing (m)	1.2
Width (m)	5	Estimated number of reamer holes	4	Row separation (m)	1.2
Advance per round (m)	3.7	Estimated drill meters (m)	277.5	Side Wall Bolting	YES
Volume extracted (m3)	92.50	Estimated reamer meters (m)	14.8	Bolts per round	48
Rock Density (t/m3)	2.65	Hole Diameter (mm):	45	Resin cartridges per bolt	4
Tonnes extracted	245.13	Required Powder Factor (kg/m3):	2.5	Bolt Length (m)	2.4
Swell factor	1.25	Explosive Product Required (kg):	231	Estimated drill meters (m)	115.2
Volume hauled (m3)	115.63	Explosive Density (t/m3)	1.3	Shotcrete Thickness (mm)	0
		Stemming Density (t/m3):	1.6	Shotcrete Rebound Factor	1.2
				Shotcrete Volume per round (m3)	0
				Mesh required (Roof / Side)	ROOF+WALL
				Conrete floor required	NO
				Concrete floor thickness (m)	0.5

Type: Shotcrete Sprayer Equipment: Maclean SS-2

Spray time per m3	mins	5	Normet Spec
Positioning time	mins	15	
Total operating time for excavation	mins	15	

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour				
10	25%	2.5	CAD 2.50		CAD 0.63	15%	CAD 0.09

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
56	75%	80%	90%	30	CAD 1.51	CAD 0.38	15%	CAD 0.06

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 9.67	CAD 2.42	15%	CAD 0.36

Subtotal machine operating cost for excavation ***CAD 3.42*** ***CAD 0.51***

Type: Concrete Mixer Equipment: Maclean TM-3

Drum Capacity	m3	6	
Drums required per excavation		0	Includes shotcrete required volume and concrete floor required volume
Loading Time	mins	5	Per trip
Discharge Rate	mins/m3	5	Set rate to match rate of concrete spray
Estimated Discharge Time	mins	0	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	300	10	1.8
Gradient Laden		10	0
Flat/Down Gradient Empty	300	18	1
Gradient Empty		10	0
Total Tramming			2.8

Total Time per Excavation mins 0

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			
10	100%	10	CAD 10.00	CAD 0.00	15%	CAD 0.00

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	CAD 0.00

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 6.82	CAD 0.00	15%	CAD 0.00

Subtotal machine operating cost for excavation ***CAD 0.00*** ***CAD 0.00***

Subtotal operating cost for excavation **CAD 556.45** ***CAD 83.47***

TOTAL GROUND SUPPORT COST **CAD 2,332.92** ***CAD 349.94***

Haulage

Fuel/Electricty Costs

Type: Development LHD Equipment: Sandvik LH517 Must be entered exactly as in 'SEA_FLEET_DETAILS WORKBOOK'

Bucket Capacity	tonnes	14	
	m^3	6.5	
	Swell factor	1.25	
	tonnes at volume	13.78	<= LIMITING VALUE
Cycles per round		18	
Loading time	mins	2	
Tipping Time	mins	1	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	40	15	0.16
Gradient Laden		5.8	0.00
Flat/Down Gradient Empty	40	15	0.16
Gradient Empty		7.5	0.00
Total Tramming			0.32

Total Cycle Time	mins	3.32	
Hourly Machine Capacity	tonnes/hour	249.04	
Cycle Time for excavation	mins	59.76	does not reflect fleet

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			
45	100%	45	CAD 45.00	CAD 44.82	15%	CAD 6.72

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	CAD 0.00

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 111.62	CAD 111.17	15%	CAD 16.68

Subtotal machine operating cost for excavation ***CAD 155.99*** ***CAD 23.40***

DEVELOPMENT REGIME

Regime Code:	D1 - P
Description:	5m x 5m Drive (Waste)

Excavation Dimensions		Blasts Details		Support Details	
Height (m)	5	Estimated number of holes	71	In-Row spacing (m)	1.2
Width (m)	5	Estimated number of reamer holes	4	Row separation (m)	1.2
Advance per round (m)	3.7	Estimated drill meters (m)	277.5	Side Wall Bolting	YES
Volume extracted (m3)	92.50	Estimated reamer meters (m)	14.8	Bolts per round	48
Rock Density (t/m3)	2.65	Hole Diameter (mm):	45	Resin cartridges per bolt	4
Tonnes extracted	245.13	Required Powder Factor (kg/m3):	2.5	Bolt Length (m)	2.4
Swell factor	1.25	Explosive Product Required (kg):	231	Estimated drill meters (m)	115.2
Volume hauled (m3)	115.63	Explosive Density (t/m3)	1.3	Shotcrete Thickness (mm)	0
		Stemming Density (t/m3):	1.6	Shotcrete Rebound Factor	1.2
				Shotcrete Volume per round (m3)	0
				Mesh required (Roof / Side)	ROOF+WALL
				Conrete floor required	NO
				Concrete floor thickness (m)	0.5

Type: Development Haul Truck Equipment: Sandvik TH540

Bucket Capacity	tonnes	50	
	m^3	20	
	Swell factor	1.25	
	tonnes at volume	42.40	<= LIMITING VALUE
Cycles per round		6	
Loading time	mins	13.28	
Tipping Time	mins	1	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden		15	0.00
Gradient Laden	3000	6	30.00
Flat/Down Gradient Empty	3000	15	12.00
Gradient Empty		14	0.00
Total Tramming			42.00

Total Cycle Time	mins	56.28	
Hourly Machine Capacity	tonnes/hour	45.20	
Cycle Time for excavation	mins	337.68	
Cycle Time for excavation (Engine on)	mins	258.00	does not reflect fleet

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour				
55	100%	55	CAD 55.00	Calculated on Engine On Time Only	CAD 236.50	15%	*CAD 35.48*

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	*CAD 0.00*

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 92.75	CAD 521.97	15%	*CAD 78.30*

Subtotal machine operating cost for excavation	**CAD 758.47**	***CAD 113.77***
Subtotal operating cost for excavation	**CAD 914.46**	***CAD 137.17***
TOTAL HAULAGE COST	**CAD 914.46**	***CAD 137.17***

Ventilation Provision

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost	Cost for excavation	*Accuracy*	*Contingency*
Vent Duct:	Vent Duct 1066mm	m	3.7		CAD 14.51	CAD 53.69	*30%*	*CAD 16.11*
Laterals:	Laterals 1066mm	Unit	0.12	Assumes Lateral every 30m	CAD 153.33	CAD 18.91	*30%*	*CAD 5.67*
Catenary Wire:	Catenary Wire	m	3.7		CAD 1.20	CAD 4.43	*30%*	*CAD 1.33*
Subtotal consumable cost for excavation						**CAD 77.04**		***CAD 23.11***

Fuel/Electricity Costs

Type: Scissor Lift Equipment: Maclean SL-3

Vent Installation Time/m	mins	8	Assumes 8 minutes per m
Total Vent Installation Time	mins	30	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	300	10	1.8
Gradient Laden		10	0
Flat/Down Gradient Empty	300	18	1
Gradient Empty		10	0
Total Tramming			2.8

Total Time per Excavation mins 32

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			
10	50%	5	CAD 5.00	CAD 2.70	15%	*CAD 0.41*

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	15%	*CAD 0.00*

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 14.15	CAD 7.64	15%	*CAD 1.15*

Subtotal operating cost for excavation	**CAD 10.34**	***CAD 1.55***
TOTAL VENTILATION PROVISION COST	**CAD 87.37**	***CAD 24.66***

DEVELOPMENT REGIME

Regime Code: D1 - P

Description: 5m x 5m Drive (Waste)

Excavation Dimensions		Blasts Details		Support Details	
Height (m)	5	Estimated number of holes	71	In-Row spacing (m)	1.2
Width (m)	5	Estimated number of reamer holes	4	Row separation (m)	1.2
Advance per round (m)	3.7	Estimated drill meters (m)	277.5	Side Wall Bolting	YES
Volume extracted (m3)	92.50	Estimated reamer meters (m)	14.8	Bolts per round	48
Rock Density (t/m3)	2.65	Hole Diameter (mm):	45	Resin cartridges per bolt	4
Tonnes extracted	245.13	Required Powder Factor (kg/m3):	2.5	Bolt Length (m)	2.4
Swell factor	1.25	Explosive Product Required (kg):	231	Estimated drill meters (m)	115.2
Volume hauled (m3)	115.63	Explosive Density (t/m3)	1.3	Shotcrete Thickness (mm)	0
		Stemming Density (t/m3):	1.6	Shotcrete Rebound Factor	1.2
				Shotcrete Volume per round (m3)	0
				Mesh required (Roof / Side)	ROOF+WALL
				Conrete floor required	NO
				Concrete floor thickness (m)	0.5

Services Provision

Consumable Costs

Item	Type	Unit	Units required per round	Life (if applicable)	Unit cost	Cost for excavation	Accuracy	Contingency
Compressed Air Pipe:	6" Pipe s80	m	3.7		CAD 30.67	CAD 113.47	*15%*	*CAD 17.02*
Water Pipe:	4" Pipe - poly	m	3.7		CAD 27.55	CAD 101.94	*15%*	*CAD 15.29*
Low Voltage Distribution Cable:	Cable Electrical 16mm2 4 Core	m	3.7		CAD 43.08	CAD 159.40	*15%*	*CAD 23.91*
High Voltage Distribution Cable:	Cable Electrical 70mm2 4 Core	m	3.7		CAD 129.24	CAD 478.19	*15%*	*CAD 71.73*
Communications:	Leaky Feeder Cable	m	3.7		CAD 0.00	CAD 0.00	*15%*	*CAD 0.00*
Subtotal consumable cost for excavation						**CAD 852.99**		***CAD 127.95***

Fuel/Electricity Costs

Type: Scissor Lift | Equipment: Maclean SL-3

Services Installation Time/m	mins	10	Assumes 10 minutes per m
Total Vent Installation Time	mins	37	

Tramming Time

Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)
Flat Laden	300	10	1.8
Gradient Laden		10	0
Flat/Down Gradient Empty	300	18	1
Gradient Empty		10	0
Total Tramming			2.8

Total Time per Excavation mins 40

If altered use array formula CTRL-SHIFT-ENTER

Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour			
10	50%	5	CAD 5.00	CAD 3.32	*15%*	*CAD 0.50*

If altered use array formula CTRL-SHIFT-ENTER

Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
0	0%	0%	0%	0	CAD 0.00	CAD 0.00	*15%*	*CAD 0.00*

If altered use array formula CTRL-SHIFT-ENTER

Maintenance Cost per hour			
CAD 14.15	CAD 9.38	*15%*	*CAD 1.41*

Subtotal operating cost for excavation	**CAD 12.70**		***CAD 1.90***
TOTAL SERVICES PROVISION COST	**CAD 865.69**		***CAD 129.85***
TOTAL COST FOR EXCAVATION	**CAD 6,218.13**	15%	***CAD 944.27***
TOTAL COST PER METER	**CAD 1,680.57**		***CAD 255.21***
MAINTENANCE ELEMENT FOR EXCAVATION	**CAD 1,594.64**	15%	***CAD 239.20***
MAINTENANCE ELEMENT PER METER	**CAD 430.98**		***CAD 64.65***

At Golder Associates we strive to be the most respected global company providing consulting, design, and construction services in earth, environment, and related areas of energy. Employee owned since our formation in 1960, our focus, unique culture and operating environment offer opportunities and the freedom to excel, which attracts the leading specialists in our fields. Golder professionals take the time to build an understanding of client needs and of the specific environments in which they operate. We continue to expand our technical capabilities and have experienced steady growth with employees who operate from offices located throughout Africa, Asia, Australasia, Europe, North America, and South America.

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