May 31, 2012

REPORT ON

Pre-feasibility Block Cave Mine Design - Mitchell Deposit

Submitted to:
Seabridge Gold Inc.
400 - 106 Front Street East
Toronto, ON   M5A 1E1

Attention: Mr. Jim Smolik

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. Mitchell 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 1,500 m east-west (along strike) and 400 m to 1400 m north-south and are between approximately 300 m and 900 m in the vertical dimension. The deposit is massive, reasonably continuous, and in general, geometrically suitable to mine by block caving. The potential of mining the Mitchell deposit by a combination of open pit and underground methods was investigated in a previous report titled "Block Cave Mining Study" (Golder 2011a), which concluded that it was possible to mine the upper portions of the Mitchell deposit by open pit methods and the deeper portions by block caving.

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 1,747M tonnes of mineralized material grading 3.2 g/t Ag, 0.61 g/t Au, 0.17% Cu, and 59 ppm Mo. This resource was evaluated using Gemcom's Footprint Finder software to evaluate the economic potential for a block cave mine. A footprint at elevation 235 m produced the most value and resulted in 438M tonnes of block cave resources with 9% unplanned waste dilution at zero grade as shown in Table A.

Table A: Geological and Block Cave Resources for Mitchell

Category	Tonnes (million)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological resources[1]	1,747	3.20	0.61	0.17	59
Mineral inventory	757	3.54	0.56	0.17	50
Block cave resources from PCBC[2,3]	**438**	**3.48**	**0.53**	**0.16**	**34**
Dilution	39	0	0	0	0
Recovery	58%				
Dilution	9%				

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

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

[3] Block cave resources can be considered as Probable Mineral Reserves within the complete pre-feasibility study report.

The quality of the rock mass at the Mitchell deposit is rated as good. No major structural features have been identified that might influence the caving mechanism and the progression of the cave in any significant manner. Cavability assessments were made using Laubscher's and Mathews' methods which involve assessing cavability based on experience at other mining operations with rock of similar quality. Both methods indicate that the size (area) of the footprint required to initiate and propagate caving is between approximately 110 m and 220 m. These dimensions are significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized three-dimensional shape of the deposit, suggests that the Mitchell deposit is amenable to cave mining. In situ stresses have been estimated from hydraulic fracturing tests and based on high induced stresses in the cave back, as predicted by numerical modelling, it is expected that stress-induced fracturing of the rock mass may contribute to caving. More sophisticated numerical analyses are recommended to confirm and quantify stress-related impacts as part of future studies.

A significant proportion of the rock at Mitchell is predicted to have block sizes greater than 2 m^3. Without some remediation measure being adopted, such large sized blocks will require significant secondary blasting, and there will likely be a significant adverse impact on production and significant damage to the drawpoints that will require ongoing rehabilitation. As a result of this, it is proposed to precondition the rock by hydrofracturing. The cost and schedule to do this have been incorporated into this study. There are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. It is also difficult to obtain definitive field data that demonstrates the degree of improvement obtained. 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 Mitchell. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning to enhance fragmentation needs to be addressed via production and cost risks. It is also very difficult to quantify the effect of attrition as the rock is brought down within the cave except that experience has indicated that in caving mines operating under similar rock conditions to those at Mitchell, fragmentation of rock, drawn down more than approximately 100 m is generally good. For this study, it was assumed that fragmentation of the initial 100 m of draw height is approximately equal to the estimated in situ block size and, above this, only limited secondary blasting would be required.

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

The underground mine design was based on modelling using Gemcom's Footprint Finder (FF) and PCBC software. FF modelling indicated that the optimum footprint for the Mitchell deposit is approximately 728 m wide in the north-south direction, 1,022 m wide in the east-west direction, and 860 m vertically with the footprint elevation at 235 m. PCBC modelling indicated that the block cave could produce 55,000 tonnes per day, requiring the development of 120 new drawpoints per year. The final mine design includes approximately 145 km of drifts and raises, including a 5% contingency to account for the excavations of design items such as service bays, sumps, and electrical substations. The design is composed of six main types of levels including preconditioning, undercut, extraction, secondary breakage, haulage, and conveying. In addition, there are two tunnels (access ramp and conveyor) from the footprint to surface to provide for mine access and material handling. The floors of the extraction drifts and drawpoints are designed to be concreted, which will increase the speed and productivity of the Load-Haul-Dump (LHD) vehicles as well as reduce equipment maintenance. The six levels of the mine design will be accessed through internal ramps beginning on the extraction level. These ramps are strategically positioned to maintain access to the levels during caving and for ventilation purposes. There are 34 extraction drifts on the extraction level and each drift is designed with three ore passes. This reduces the average LHD haul distance to approximately 100 m and improves productivity.

Production material will be hauled from drawpoints to one of three ore passes situated within the same extraction drift. The ore passes from neighbouring extraction drifts will feed a stationary rockbreaker on the secondary breaking level, which will reduce the size of the material further and feed it to the haulage level via passes with chutes. A train on the haulage level will haul the material to one of two gyratory crushers, where it will be crushed and conveyed to the surface.

The proposed mobile equipment is typical of that used in underground mines and is comprised of those pieces directly related to moving ore to the crushers (8.6 m^3 LHDs, secondary rockbreakers, and the train), the development equipment (4.6 m^3 LHDs and 18 m^3 trucks) as well as the AnFo loaders and ground support machines. In addition, service equipment is included for construction and mine maintenance activities. At peak operation, Mitchell will require a fleet of approximately 60 pieces of mobile underground equipment. The mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life with a peak quantity of 489 personnel in Year 7.

The majority of the main ventilation infrastructure is also located on the extraction level. It consists of two fresh air raises, two fresh air drifts, a fresh air ring drift, multiple internal ventilation raises, a return air drift, and two exhaust raises. The conveying level starts beneath the cave and finishes on surface near to the main conveyor transporting material to the plant site. It is designed to accommodate both production ore and development waste material. The required airflow for the Mitchell mine to achieve a production rate of 55,000 tpd is 860 m^3/s based upon the diesel equipment utilized, air velocity considerations, and a contingency of 20% per level. Heating of the mine air in the winter months is included in the design and cost estimates. It is estimated that the Mitchell mine will require approximately 17,400 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans.

The maximum estimated groundwater inflow for the Mitchell block cave mine is 13,200 m^3/d. At the time of completing this pre-feasibility assessment, estimates by others of the surface inflows into the crater at Mitchell were not available. These surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. In future studies, the water management system will need to be enhanced to cater for this additional inflow.

The mine development schedule was separated into three phases; an initial pre-production phase which involves developing the primary access ramp and conveyor drifts; 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 is about 4,000 m, with peak development occurring during the second phase, when about 15,000 m is required per year.

The mine production schedule was developed using Gemcom's PCBC software. It was assumed that sloughing of peripheral waste rock would occur into the crater and cover the upper surface of the material being drawn down. This was modelled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution. Due to the large fragmentation that is estimated to report to the drawpoints at Mitchell, particularly during the early stages of mining, a draw rate of 200 mm/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 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. Mitchell is estimated to have a production ramp-up period of 6 years, steady state production at 20 million tonnes per year for 14 years, and then ramp-down production for another 7 years.

The average mine operating cost is estimated at $5.00/tonne and consists of the equipment and labour that is required to move material from the drawpoint to the surface conveyor portal and the fixed costs to operate the mine (Table B). This includes the use of the LHDs, secondary breakers, crushers and conveyors, and the labour required to plan and execute the mining plan. Mine labour comprises approximately 52% of the total Mitchell underground mining cost while crushing and conveying is 15%, secondary breaking is 13% and production mucking and haulage is 12%.

Table B: Underground Mine Operating Cost Breakdown

Activity	OPEX ($/tonne)	(%)
Labour	$2.60	52%
Crusher and conveying	$0.80	15%
Stationary and mobile rockbreaking	$0.61	13%
Production LHD and haulage	$0.58	12%
Fixed costs	$0.36	7%
Rehabilitation	$0.04	1%
Total	**$ 5.00**	

The 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 6 years of the mine life, are estimated at $800 million with an average sustaining capital cost of $74 million over the remaining 31 years. The life-of-mine capital costs are estimated to be $3.1 billion

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Table 1: Units Used in the Text

Unit	Definition
m	Metre
km	Kilometres
mm	Millimetres
g/t	Grams per tonne
% Grade	Grade item in % (such as Copper)
US$/t	US dollars per tonne
$	Dollars - assumed cad unless specified
M	Million
%	Percent
ppm	Parts per million
m^2	Square metres
m^3	Cubic metres
m/s	Metres per second
MPa	Mega Pascal's
FF/m	Fracture frequency per metre
°	Degrees in an angle
Q'	Modified Q (Barton's rock mass classification system)
N	Stability number
"	Inch
m^3/s	Cubic metres 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 metre per second air causes a pressure drop of one Pascal
m^3/d	Cubic metres per day
m^3/hr	Cubic metres per hour
mm/day	Millimetres per day
$/m	Dollars per metre
$M	Million dollars
$/tonne	Dollars per tonne

Unit	Definition
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 kilometres southwest of Bell II 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 Seabridge pre-feasibility study update (PFU) 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 (PFS) 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 Mitchell deposit. A similar evaluation for the Iron Cap deposit is presented in a separate report.

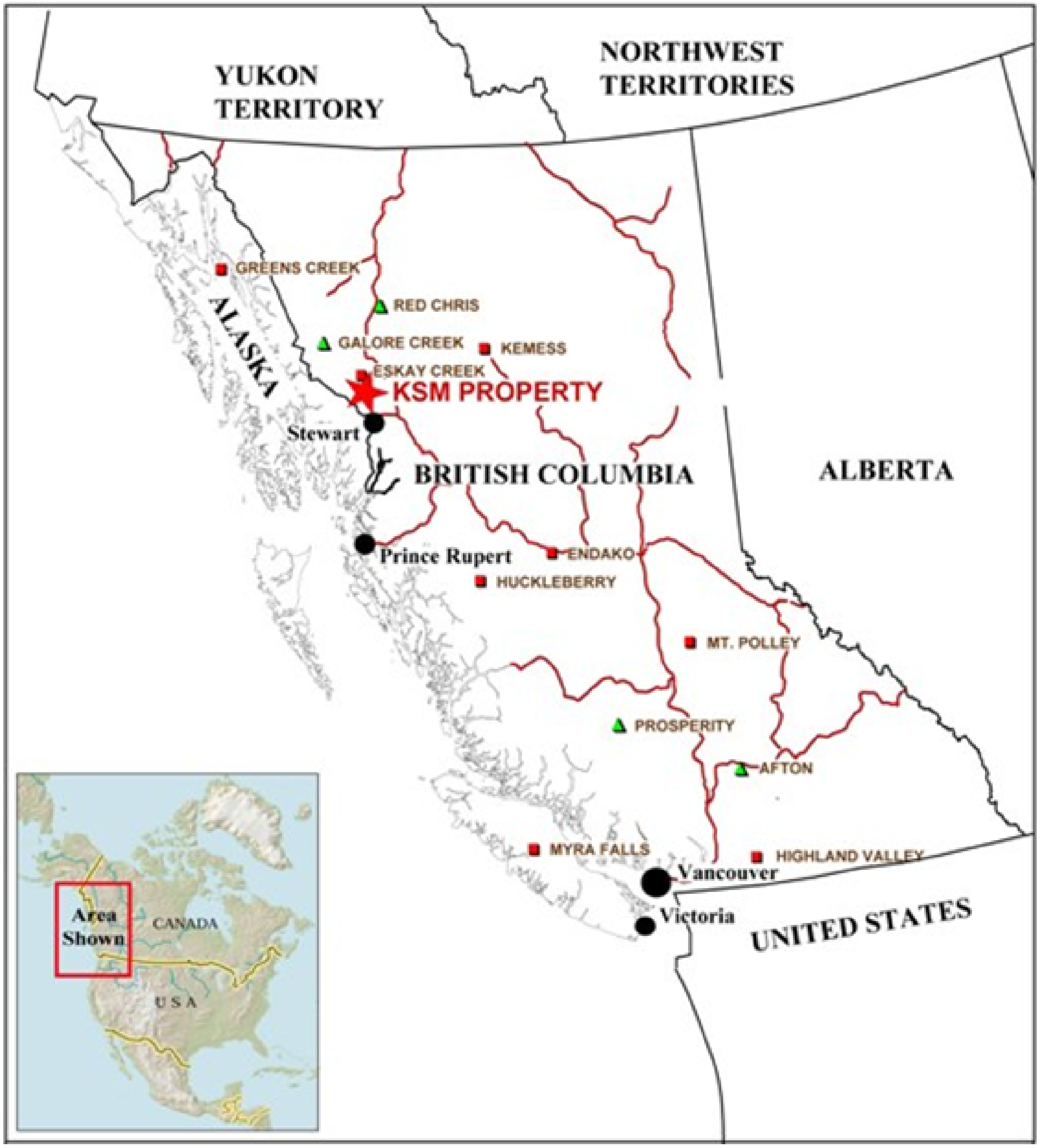

Figure 1: Location of the Mitchell, Kerr and Sulphurets (KSM) property.

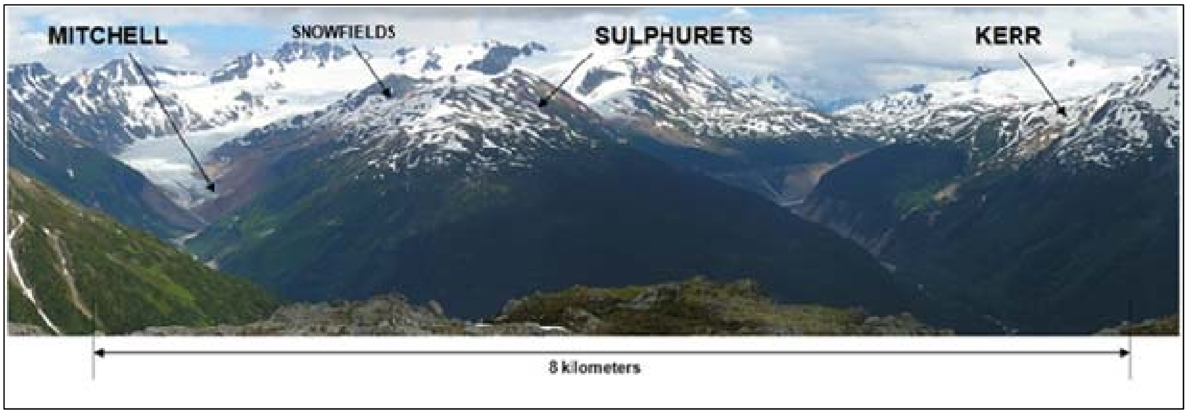

Figure 2: Aerial view of the general area of the Mitchell deposit (looking east).

1.1 Mining Concept

The Mitchell deposit is a porphyry type intrusion that has been deformed by subsequent tectonic processes, resulting in a footwall contact dipping at approximately 40 degrees to the north. The deposit outcrops at the base of the Mitchell valley just to the west of the Mitchell glacier, which has previously eroded some of the deposit at the base of the valley. Figure 3 shows the site topography and the 0.25 g/t gold (Au) grade shell.

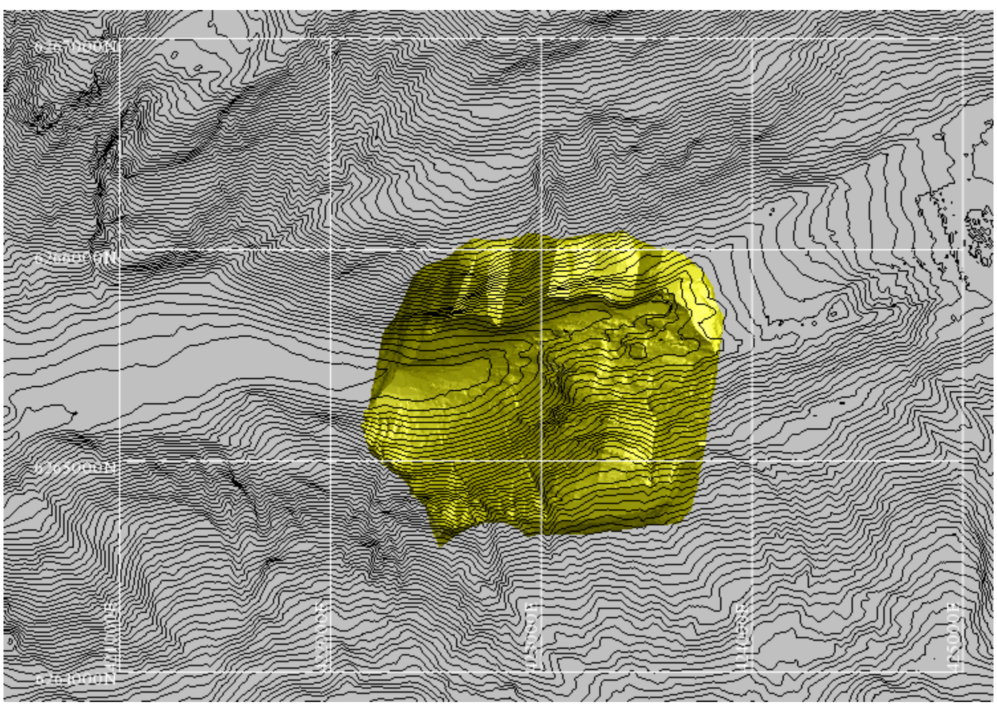

Figure 3: Site topography and 0.25 g/t Au grade shell.

The geometrical shapes of the 0.25 g/t Au or 0.1% Cu grade shells are very similar and superimpose one another. They extend approximately 1,500 m east-west (along strike) and 400 m to 1400 m north-south and are between approximately 300 m and 900 m in the vertical dimension. A "typical" cross-section through the deposit is shown in Figure 4. The deposit is large in three dimensions, and reasonably continuous. It is understood that the deposit remains open at depth.

The proposed mine plan will involve open pit mining to a designed depth followed by block cave mining from the underground. The pre-feasibility open pit design is presented as an appendix in the report entitled "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" (Seabridge 2011).

An assessment was made by Moose Mountain Technical Services (MMTS) of the limit of the pit shell at which the open pit mining cost approximately matches the underground block caving mining cost. This pit is referred to as the M685 pit. A typical cross-section showing the proposed open pit design is shown in Figure 5. No detailed optimization was carried out as to the precise transition from open pit to underground mining. This will need to be undertaken as part of the feasibility study.

The Mitchell deposit has been the focus of another report by Golder, titled "Bock Cave Mine Study" (Golder 2011a), and reports by Seabridge. The relevant Seabridge reports are:

- The pre-feasibility study evaluating the potential to mine the Mitchell deposit solely by open pit methods. This report was published on March 31, 2010 and the relevant sections to this report include:
  - Appendix G1: A geotechnical design of the open pit by Bruce Geotechnical Consultants Inc. (BGC);
  - Appendix G7: An open pit depressurization analysis by BGC;
  - Appendix F: A mine design section by MMTS;
- The mineral resource update published on January 25, 2011; and
- The pre-feasibility study update published on June 15, 2011.

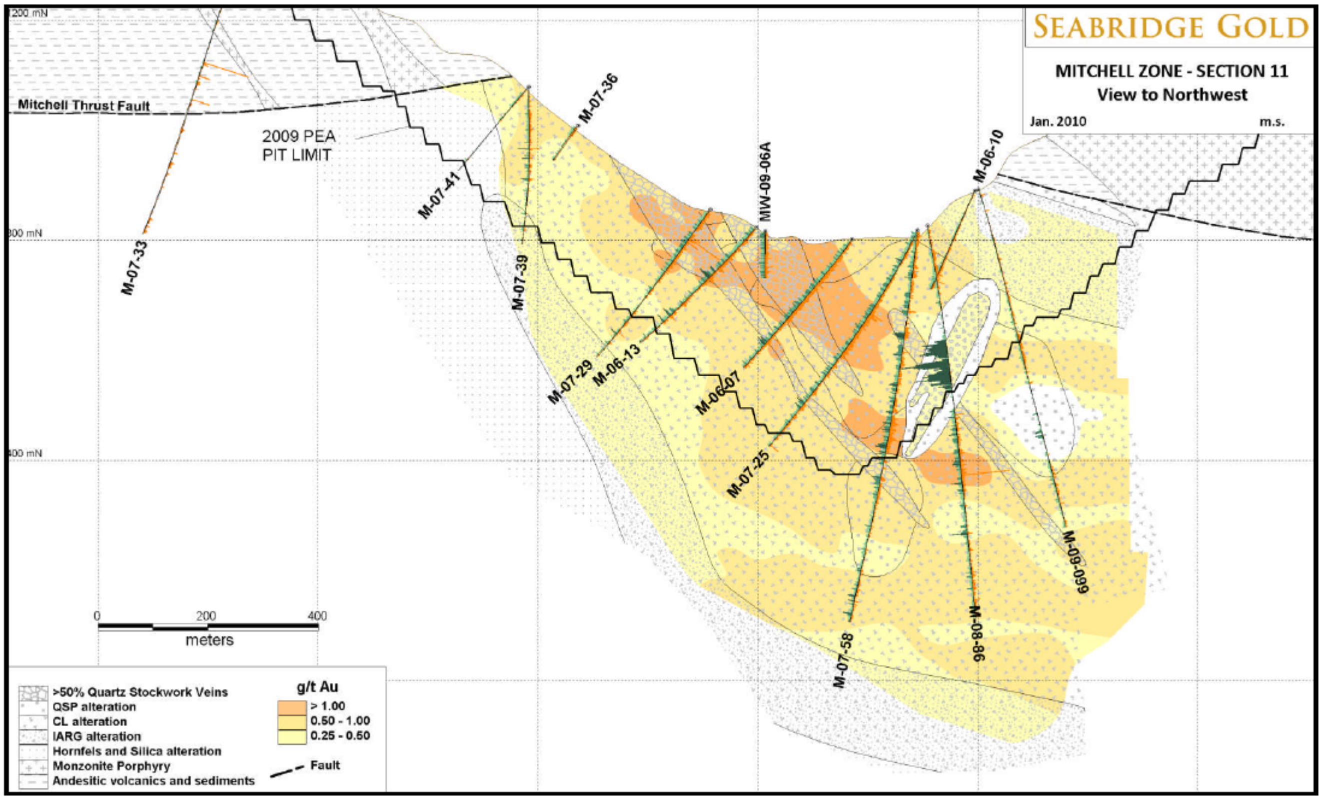

Figure 4: Typical cross-section through Mitchell deposit (Seabridge 2011).

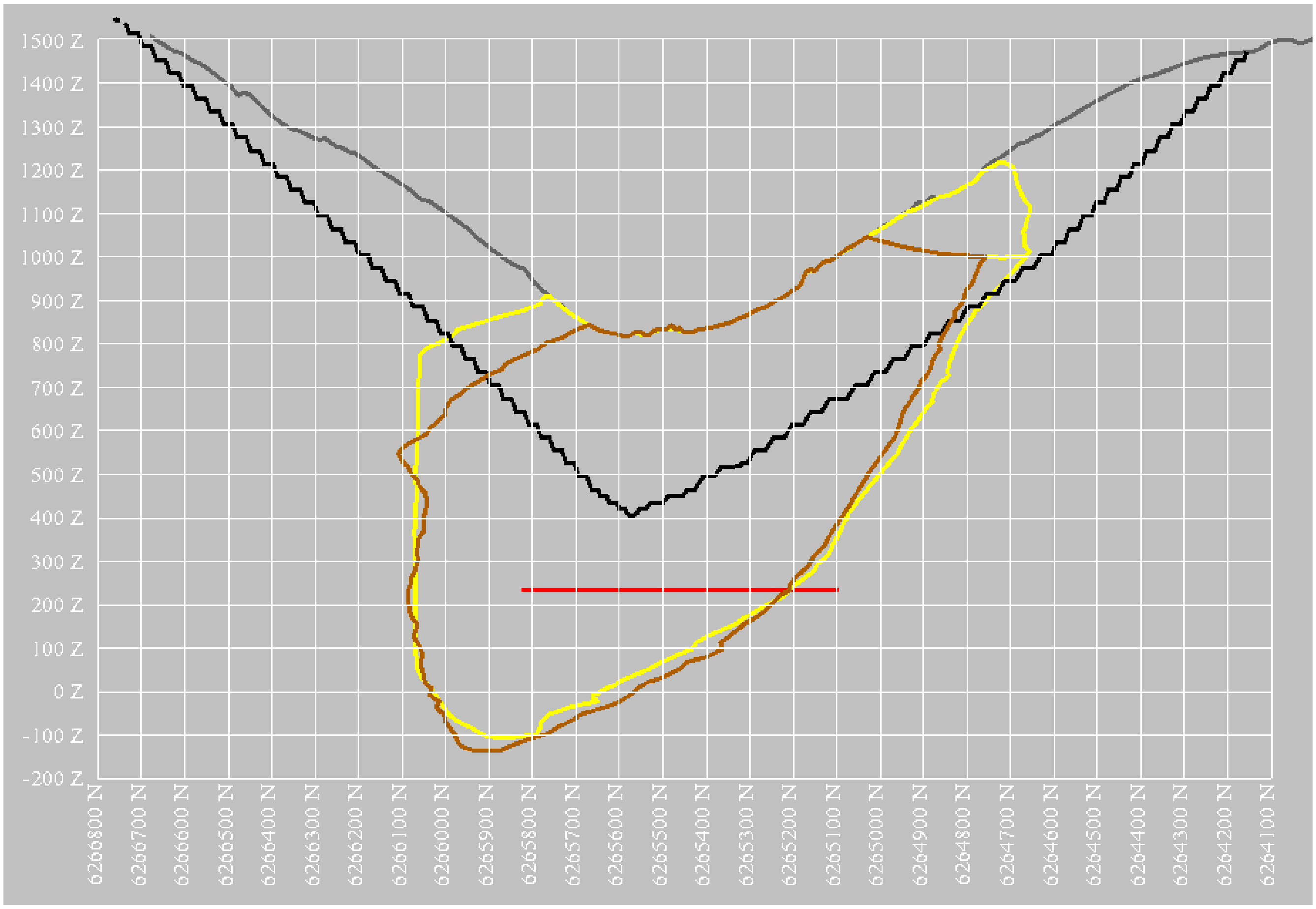

Figure 5: Typical cross-section showing proposed pit, looking east.

1.2 Scope of Work

Mitchell is part of the KSM property located in the Coast Mountains in northwestern British Columbia amid challenging topography. The property has 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 the PFS published in 2011. Golder was engaged to evaluate the potential to mine the Mitchell deposit using block caving methods to the pre-feasibility level of engineering study. This includes the following scope:

- Integration with open pit mining;
- Underground mine access including a trade-off study regarding the use of shafts versus conveyors to move broken material to surface (Appendix A);
- 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 including trade-off studies between electric and diesel Load-Haul-Dump machines, and between diesel trucks, electric trucks and trains; 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 Ore Process Control (OPC) was completed by Bosche 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) or to provide input to others (e.g., to estimate the size of camp required).

2.0 GEOLOGICAL SETTING

The Mitchell deposit is a porphyry-type intrusion that has been deformed by subsequent tectonic processes, resulting in a footwall contact dipping at approximately 40 degrees to the north. The deposit outcrops at the base of the Mitchell valley just to the west of the Mitchell glacier.

A general view of the outcrop of the Mitchell deposit and the surface expressions of relevant geological features are shown in Figure 6.

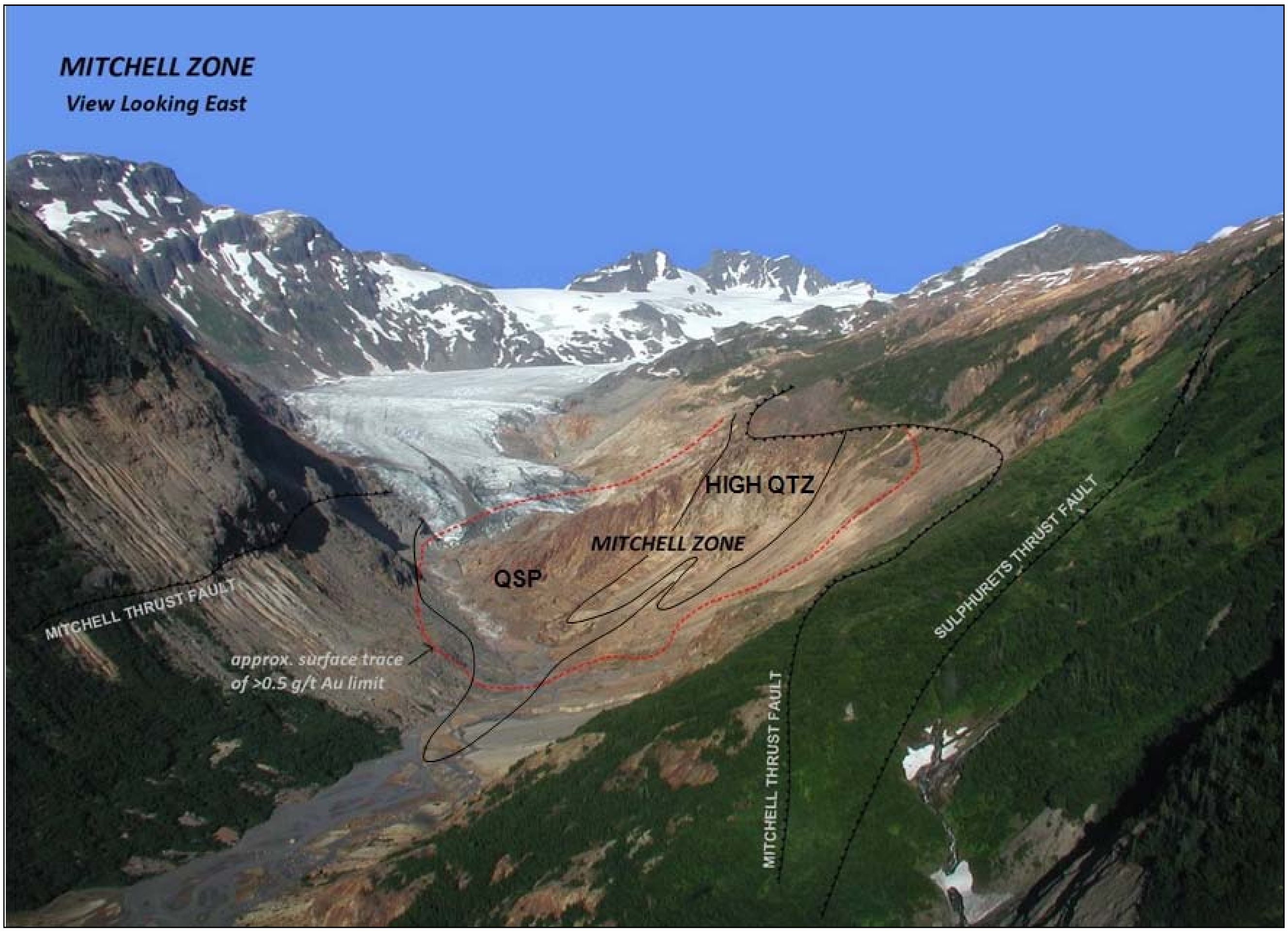

Figure 6: Aerial view showing the outcrop of the Mitchell deposit and surface expressions of relevant geological features.

The geological information for the Mitchell 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.

The geometrical shapes of the 0.25 g/t Au or 0.1% Cu grade shells are very similar and superimpose one another. The deposit extends in plan approximately 1,500 m east-west (along strike), approximately 400 m to 1400 m north-south, and approximately 300 m and 900 m vertically (Figure 7).

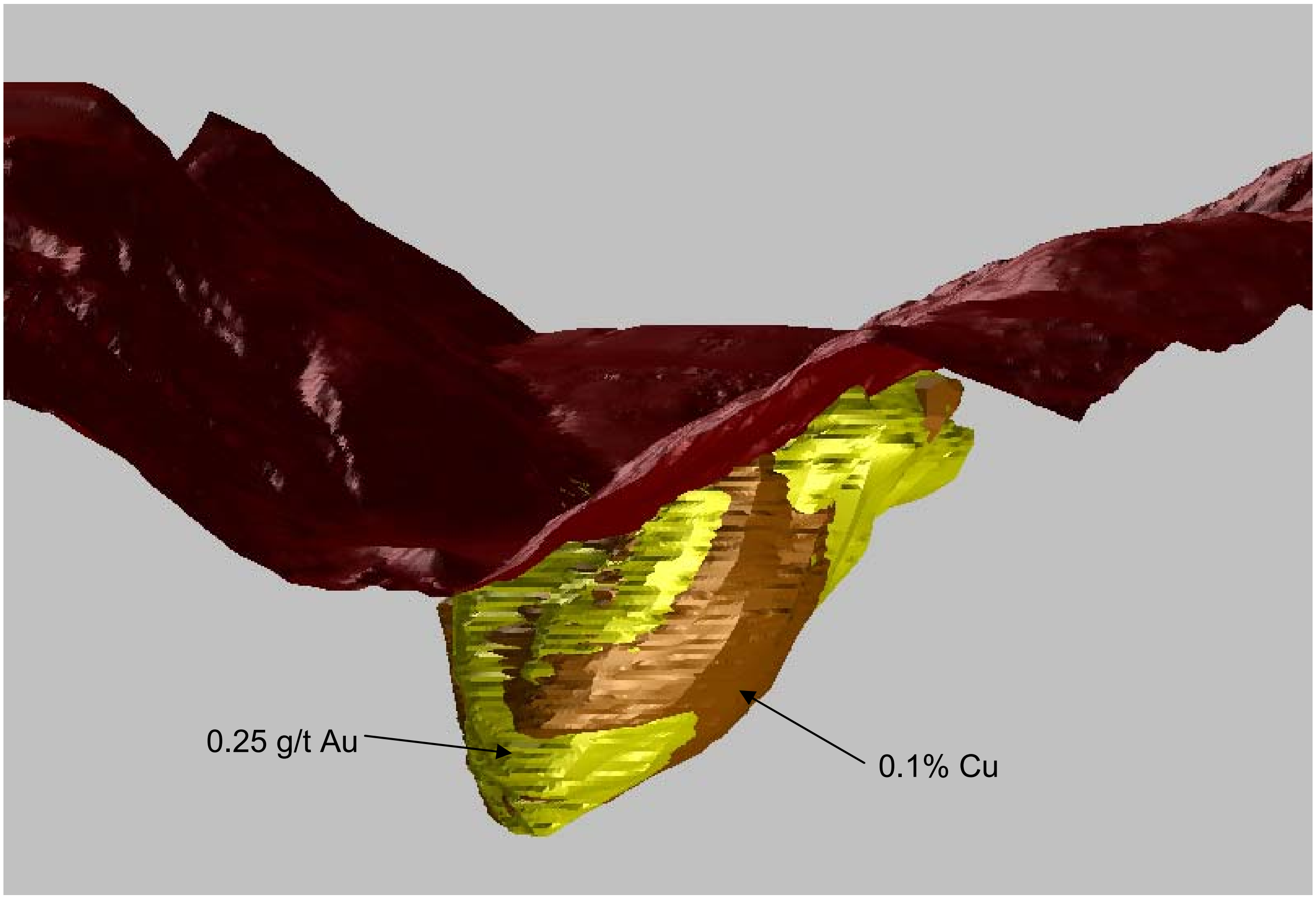

Figure 7: Isometric view 0.25 g/t Au and 0.1% Cu grade shells of the Mitchell deposit.

A vertical cross-section towards the centre of the deposit showing lithology, alteration, structure, and grade shells is presented in Figure 8. The lithological units within the area of potential block cave mining (between the floor of the proposed pit and the underground production level) are primarily altered volcanics that lie beneath the Mitchell Thrust Fault (Table 2).

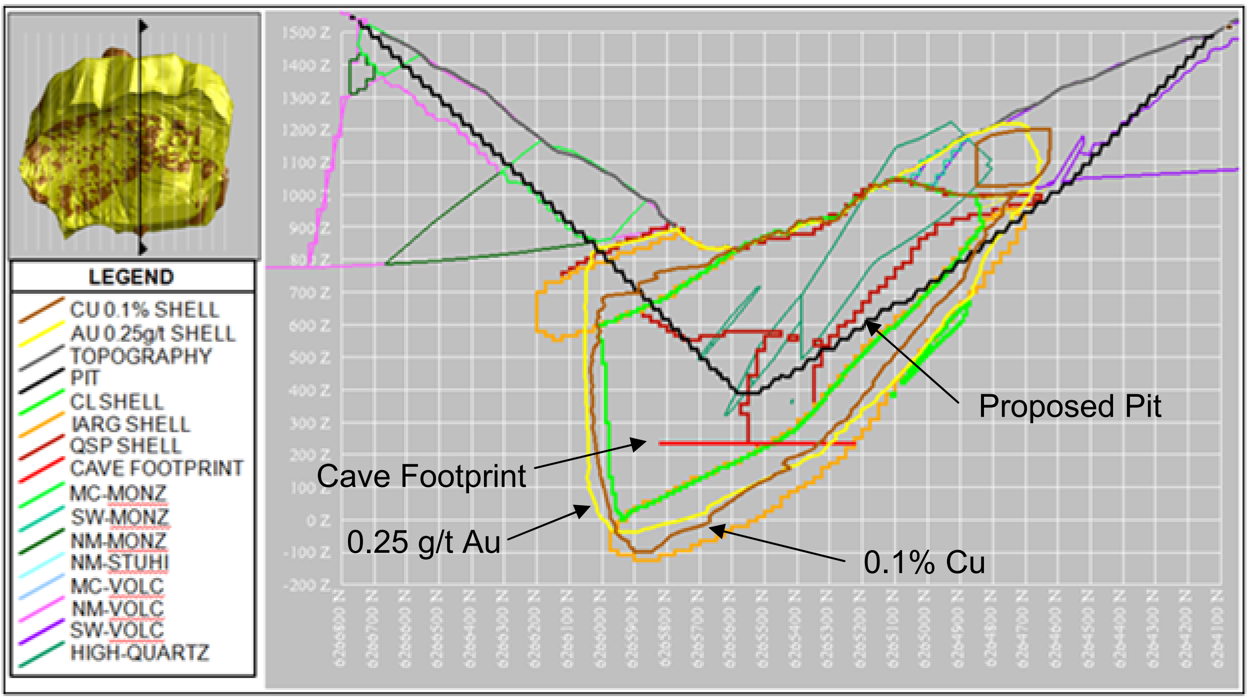

Figure 8: Vertical cross-section (423100 Easting) of the Mitchell deposit showing lithology, alteration, and 0.25 g/t Au and 0.1% Cu grade shells.

Table 2: Mitchell Lithology

Code	Description
MC-MONZ	MC Monzonite
SW-MONZ	SW Monzonite
NM-MONZ	NM Monzonite
NM-STUHI	NM Stuhini group rocks
MC-VOLC	MC Volcanic
SW-VOLC	SW Volcanic
NM-VOLC	NM Volcanic
HIGH-QUARTZ	High quartz

As indicated in Table 3, these rocks are typically associated with intermediate argillic alteration (IARG), quartz-sericite-pyrite alteration (QSP), and chlorite-propylitic alteration (CL-PR). For the purpose of this study, the logged alteration codes have been classified into the above three alteration types (IARG, QSP, and CL-PR). Alteration types that did not fit these three broad categories have been classified as “Other” as indicated in Table 3.

Table 3: Mitchell Alteration

Code	Description	Logged Codes	Percentage by Length of Logged Rock (%)
CL-PR	Chlorite-propylitic alteration	CL, CL2, CLSTW, CL2STW, PR	62.3
IARG	Intermediate argillic alteration	IARG	8.3
QSP	Quartz-sericite-pyrite alteration	QSP, QSPSTW	13.7
Other	Carbonate veining Hematization Hornfels or skarn Potassic Late quartz veins Silicic	CARB HEM HFLS, SIH, MTH KP, PKBX, QB QTVN SI, SIL, PSBX	15.7

Note: Taken from Seabridge (2011).

3.0 BLOCK CAVING RESOURCES

A mineral resource block model was provided by Moose Mountain Technical Services (MMTS) and 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. The inferred grades were set to zero and are not included in this pre-feasibility study.

3.1 NSR Cut-Off

The NSR cut-off used in this report is based on the NSR formula detailed in the PFU that was also used to determine general and administration (GA), water treatment and milling costs as detailed in Table 4. The underground mining cost was determined from first principles and is discussed further in Section 11.

Table 4: Components of the NSR Cut-off

Item	US$/t Milled
Underground Mining[1]	5.84
Milling, G&A and Site Service	9.57
Total	**15.41**

Note: [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 11.2.

3.2 Resource breakdown

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, including those mined by open pit;
- *Mineral inventory* is the portion of the potentially economic resources above the NSR cut-off located outside the pit;
- *Dilution* is defined as material with zero grade that is mined within the footprint at the 235 m elevation, including the inferred material;
- *Block cave resources* are the measured and indicated material that is mined from within the footprint at the 235 m elevation and with NSR > $15.41. It is determined by PCBC and also includes the dilution; and
- *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.

The geological resource contains 1,747M tonnes of mineralized material grading 3.2 g/t Ag, 0.61 g/t Au, 0.17% Cu, and 59 ppm Mo. This resource was evaluated using Gemcom's Footprint Finder software (the Footprint Finder results will be discussed in Section 4) to evaluate the economic potential for a block cave mine. The result is approximately 438M tonnes of block cave resources, including 9% unplanned and 10% planned dilution. A summary of the Mitchell block cave resources can be found in Table 5 and Table 6.

Table 5: Geological and Block Cave Resources Table for Mitchell

Category	Tonnes (million)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological resources[1]	1,747	3.20	0.61	0.17	59
Mineral inventory	757	3.54	0.56	0.17	50
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Dilution	39	0	0	0	0
Recovery	58%				
Dilution	9%				

Notes: [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 6: Mineral Resources Recovered at the Drawpoints

Category	Tonnes (million)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Measured	138	4.18	0.63	0.20	37
Indicated	242	4.03	0.61	0.19	39
Measured and Indicated	**381**	**4.08**	**0.62**	**0.19**	**38**
Waste	39	0	0	0	0
Inferred	18	3.92	0.47	0.16	43
Total[4]	**438**	**3.56**	**0.54**	**0.17**	**33**

Note: [4] The block cave resources reported in Table 5 have a different grade than those reported in Table 6. Table 5 represents the results from the PCBC analysis where the inferred grades were set to zero. To report the grade of the inferred material mined in the production schedule, PCBC was evaluated with the influence of the inferred grades included. The difference in grade is less than 5% and considered within the range of accuracy of this study.

4.0 PRELIMINARY MINING ASSESSMENT

The Mitchell deposit outcrops at the base of the Mitchell valley near the toe of the Mitchell glacier. Initially, only open pit mining methods were used to evaluate the mining potential of this deposit. However, pit operating costs increase significantly as the pit is deepened. The potential of mining the Mitchell deposit by a combination of open pit and underground methods was investigated in a report titled "Block Cave Mining Study" (Golder 2011a), which concluded that it was possible to mine the upper portions of the Mitchell deposit by open pit methods and the deeper portions by block caving.

Block caving is a low cost underground mining method and it has the potential to achieve very high underground production rates. However, it involves 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 to investigate the possibility of mining the Mitchell deposit 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 tool used to evaluate the potential for a deposit to be mined by block caving. Additional information concerning the FF module is presented in Golder's initial report on block caving the Mitchell deposit (Golder 2011a).

4.1 Footprint Finder Inputs

Footprint Finder requires a block model of the mineralized material, including a value attribute such as 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 7 shows the typical inputs required and the values used for Mitchell. These costs were mostly based on experience, with the exception of "Other Operating Costs" which were based on the PFU (Seabridge 2011). Additional details concerning the inputs and their definitions can be found in the Golder report "Block Panel Caving Conceptual Study for the KSM Project" (Golder 2011b).

Table 7: Input Values Used in Footprint Finder to Evaluate the Block Caving Potential of Mitchell

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
Pit shell (depth: 405 m)	LG PIT 08-UG OP Cut-off 15OCT2010.dxf

4.2 Footprint Finder Results

A summary of the FF results is shown in Figure 9. A footprint at elevation 175 m will have the most tonnage ($994M and 539M tonnes), while a footprint at 235 m will have the most value ($1,275M and 526M tonnes). The Mitchell block cave design was based on the 235 m elevation footprint, with tonnage and grade summary presented in Table 8 and the footprint geometry shown in Figure 10.

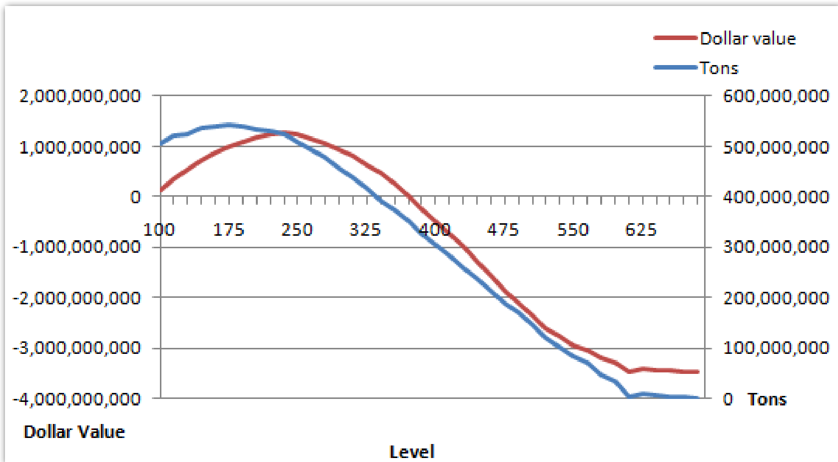


Figure 9: Summary graph of the footprint finder results for the Mitchell deposit.

Table 8: Summary of the Footprint Finder results for the Footprint Chosen 235 m Elevation

Elevation (m)	Tonnage (Mtonnes)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
235	526	0.57	0.18	3.79	39.69

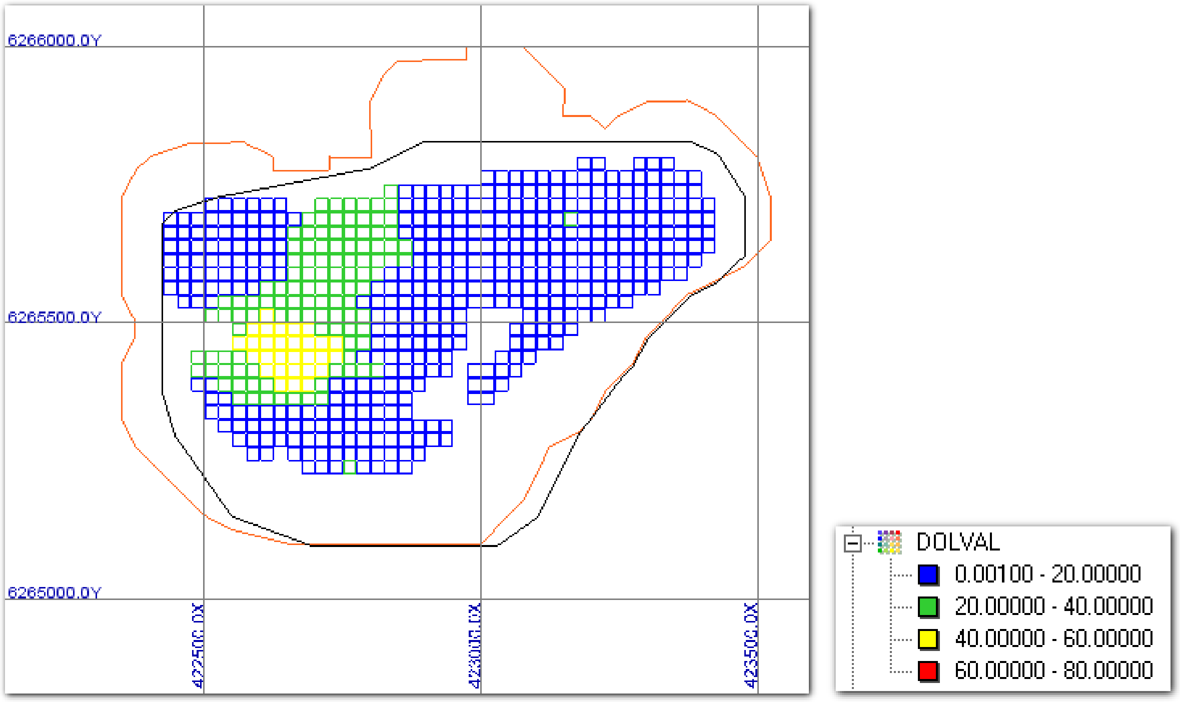


Figure 10: Outline of the Mitchell footprint (inner, black line) with the value of columns of the geological resource at 235 m elevation.

5.0 GEOTECHNICAL CHARACTERIZATION

The characterization of the rock mass has focused on the rock in and around the extraction level of the proposed block cave mine and on the mineralized rock above this that will be caved. A second area of interest involves the rock where the ramps, conveyor drifts, raises, and other mine infrastructure will be excavated to connect the production elevation to surface.

Characterization of the rock was based on core photographs and data collected for exploration drillholes, detailed geotechnical data collected for drilling programs carried out by BGC in 2009 (BGC 2010) and Golder in 2011 (Golder 2012a), outcrop mapping data (Golder 2012a), laboratory testing data (BGC 2010; Golder 2012a), and an interpreted geological model provided by Seabridge. Detailed descriptions of the available data for this study are contained in the geotechnical characterization report (Golder 2012b).

There are a total of 114 exploration holes and 14 geotechnical holes in the Mitchell deposit area. The borehole locations are shown in Figure 11. Geotechnical boreholes are shown in red.

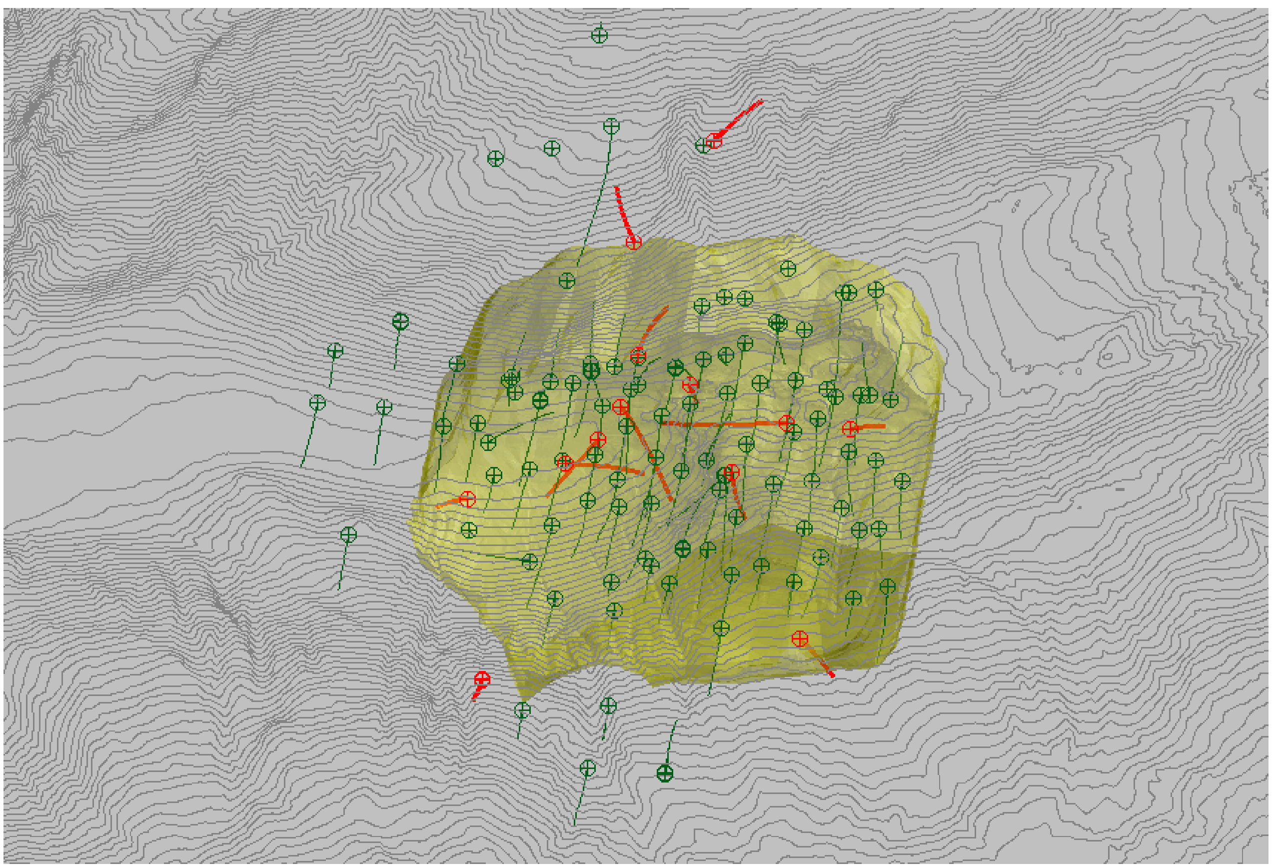


Figure 11: Mitchell exploration and geotechnical boreholes and 0.25 g/t Au grade shell.

For the purpose of this study, host rock refers to the rock mass outside of the immediate area of mineralization. The host rock in which the mine infrastructure (e.g., raises, conveyor drifts, ramps, etc.) will be excavated has been assessed based on data collected for nearby drillholes.

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 2012b). Further site characterizations and geotechnical conditions are presented in BGC's pre-feasibility report for the open pit (BGC 2010).

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 shown in Appendix B with example core photographs for each of the categories listed in Table 9 below.

Table 9: 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. A good correlation was observed between RQD and RMR for the geotechnical boreholes. Using the RQD and RMR data from the Mitchell "central" boreholes (Figure 12), a correlation was developed between the two. The correlation equation 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 geotechnical characterization report (Golder 2012b).

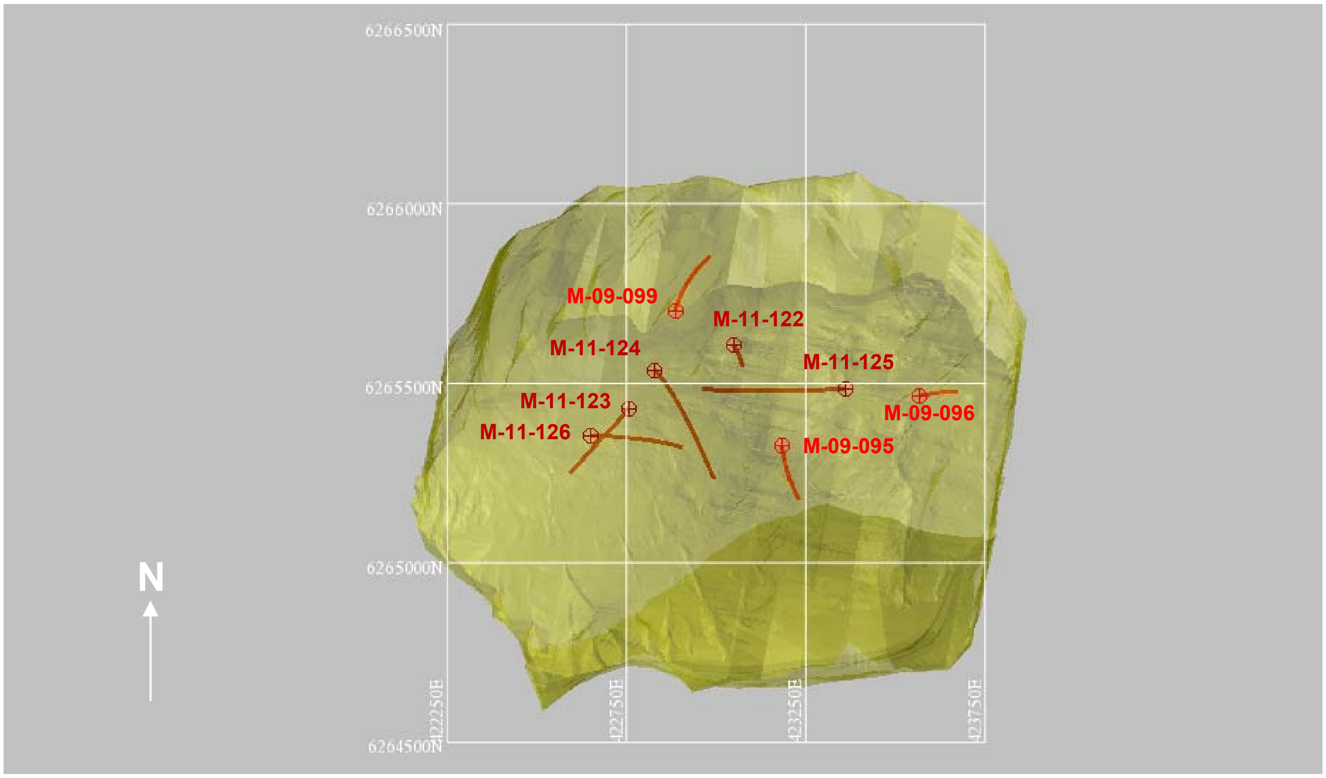


Figure 12: Central boreholes and 0.25 g/t Au grade shell.

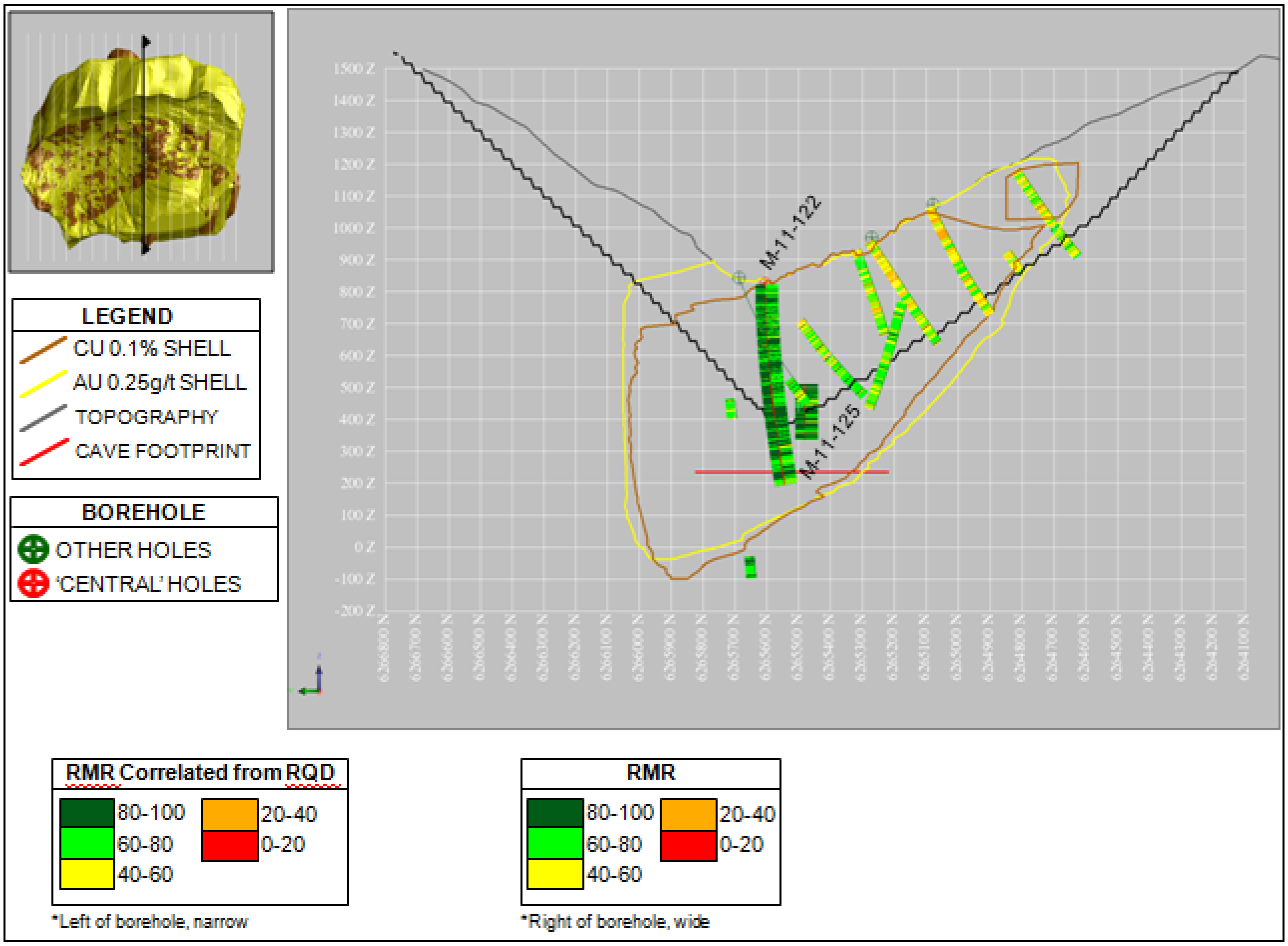


Figure 13: Vertical cross-section at Easting 423100 showing correlated RMR and logged RMR.

A brief discussion of the typical rock quality for the mineralized rock and the surrounding host rock where some of the mine infrastructure will be located is contained in the following sections.

5.1.1 Mineralized Rock

The average RMR for the mineralized rock between the pit floor (El. 405 m) and the production horizon (El. 235 m) was determined to be approximately 77. The rock conditions are classified as "good" according to the ratings shown in Table 9 and are relatively consistent across the deposit. Values are in agreement with those described in the pre-feasibility open pit study (Seabridge 2011).

5.1.2 Host Rock

Details of the anticipated rock conditions around specific infrastructure excavations are discussed in Section 7.8. The majority of the mine infrastructure is located below the Mitchell Thrust Fault (MTF). Average RMR values are similar to the mineralized rock for each alteration type and range from approximately 65 to 75, indicating good quality rock.

Rock quality is anticipated to be slightly lower for infrastructure located above the MTF (e.g., the upper portion of the ramp). Average RMR values for each alteration type range from approximately 50 to 60, indicating fair quality rock.

5.2 Intact Rock Strength

Intact rock strength has been estimated based on International Society for Rock Mechanics (ISRM) strength ratings logged for the geotechnical boreholes as well as Unconfined Compressive Strength (UCS) and Triaxial Strength tests carried out on samples from the 2009 and 2011 field programs. A detailed description of the field and laboratory strength data is presented in the geotechnical characterization report (Golder 2012b).

The intact strength is generally consistent throughout the project site. There is no distinction between the strength of the mineralized rock and that of the surrounding host rock. The data also suggest that there is no significant difference between the strength of the rock above the MTF and below the MTF.

A total of 30 UCS tests were conducted as part of the 2009 and 2011 field programs (BGC 2010; Golder 2011). UCS values ranged from 38 to 205 MPa, with an average UCS (for all alteration types) of 97 MPa. A summary of the testing results by alteration type is presented in Table 10.

Table 10: UCS Testing Results from the 2009 and 2011 Programs

Alteration Type	Number of Samples	Range (MPa)	Average (MPa)
Chloritic-propylitic (CL-PR)	22	38.3 – 176.3	93.6
Phyllic: Quartz-sericite-pyrite (QSP)	4	68.9 – 87.4	75.8
Intermediate argillic (IARG)	2	86.4 – 167.6	127.0
Other	2	93.5 – 204.8	149.1

There are not enough data to conclude whether strengths vary between alteration types. There is no obvious pattern of strength with depth.

A series of triaxial tests were carried out to estimate the failure envelope of the intact rock. All samples were generally consistent in appearance (e.g., colour, veining). Four samples were logged as CL-PR alteration (chloritic and prophylitic alteration) and two samples were logged as QSP (phyllic alteration).

The tests were carried out at confining stresses between 0.5 and 6 MPa. These stresses were based on the results of simple 3D elastic stress modelling to investigate the stresses around the block cave at various stages of cave development. The confining stress in the back of the cave is predicted to approach 6 MPa at approximately 5 m into the back. Details of the analyses are presented in Appendix C.

The estimated friction angle and cohesion for the rock mass are 47 degrees and 20 MPa, respectively.

Field intact rock strength estimates were logged for the 2009 and 2011 boreholes according to the ISRM standard field identification methods (ISRM 1981). A description of each strength category from the field logging is described in Table 11.

Table 11: Field Identification Methods for Description of Rock Strength (ISRM 1981)

Grade	Description	Field Identification	Approximate Range of UCS (MPa)
R0	Extremely weak rock	Indented by thumbnail	0.25 – 1.0
R1	Very weak rock	Crumbles under firm blows with point of a geological hammer, can be peeled by a pocket knife	1.0 – 5.0
R2	Weak rock	Can be peeled by a pocket knife with difficulty, shallow indentations made by firm blow with point of geological hammer	5.0 – 25
R3	Medium strong rock	Cannot be scraped or peeled with a pocket knife, specimen can be fractured with single firm blow of geological hammer	25 – 50
R4	Strong rock	Specimen requires more than one blow of geological hammer to fracture it	50 – 100
R5	Very strong rock	Specimen requires many blows of geological hammer to fracture it	100 – 250
R6	Extremely strong rock	Specimen can only be chipped with geological hammer	> 250

Logged ISRM strength measurements were found to be somewhat underestimated when compared to laboratory tests. The indices are useful in identifying whether any weak zones exist in the deposit, but should not be relied on for distinguishing between R3 and R4 strength rock. Downhole plots of ISRM strength indices are described in the geotechnical characterization report (Golder 2012b). Figure 14 shows ISRM strength ratings plotted downhole for a “typical” section through the orebody.

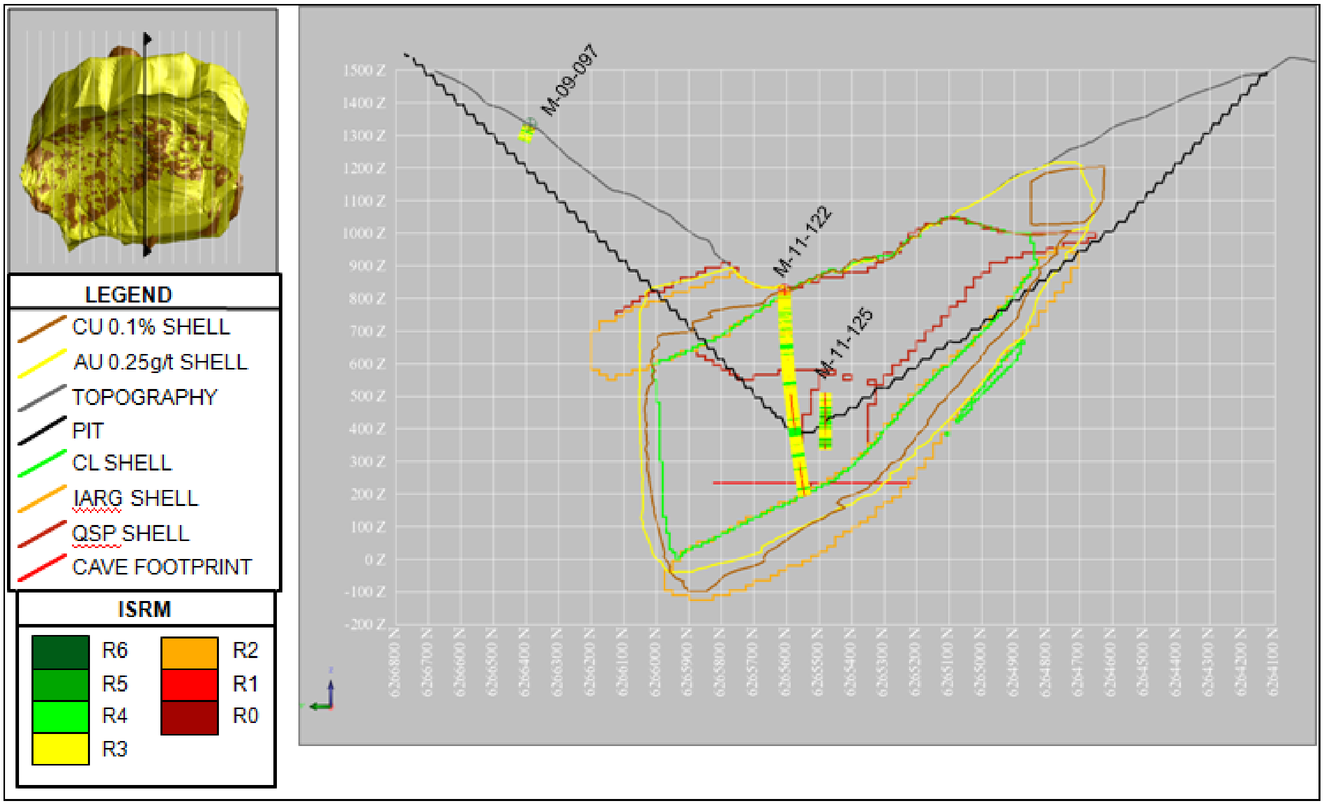


Figure 14: Vertical cross-section at Easting 423100 showing logged ISRM strength.

It is interesting to note that the rock does not appear to preferentially break along veins or foliation. Field observations indicate that the veins and foliation are not obviously planes of weakness. Qualitative observations suggest that when hit with a geological hammer, fractures are just as likely to form across veins as along veins.

5.3 Fracture Orientations

Oriented core logging was part of the 2009 and 2011 geotechnical drilling programs. Detailed descriptions and stereographic projections of fracture orientations are available in the reports for these field investigations (BGC 2010; Golder 2012a).

The oriented boreholes in the immediate area of mineralization are M-09-095, M-09-096, M-09-099, M-11-123, M-11-124, M-11-125 and M-11-126. Foliation appears to be prominent, steeply dipping approximately to the north (Figure 15). A second, less dominant joint set dips at moderate angles (30 to 60 degrees) approximately to the south.

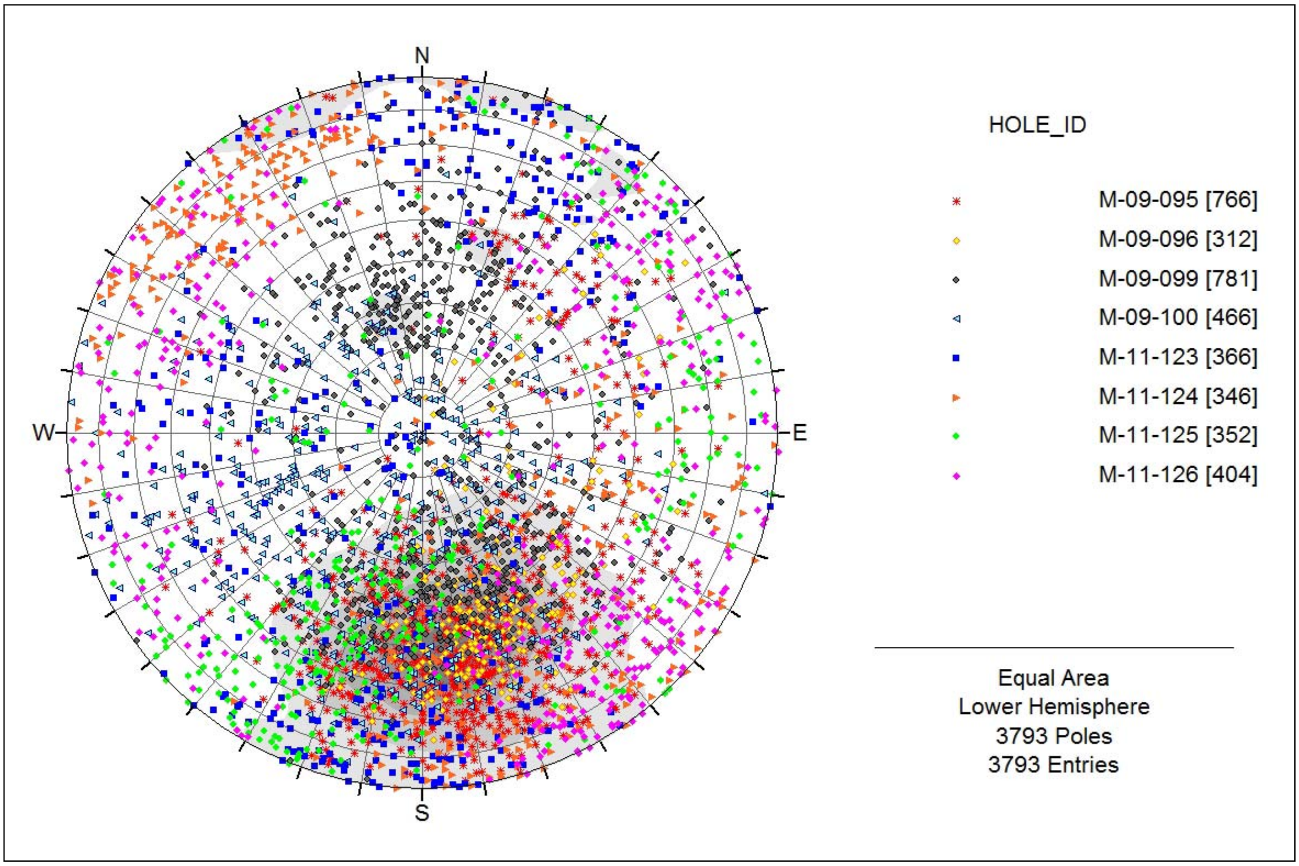


Figure 15: Stereographic projection showing open features classified by borehole.

5.4 Fracture Intensity

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

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

When assessing fracture frequency, only the boreholes in and around the footprint of the mineralized rock were considered. Portions of the holes above the floor of the proposed pit were discarded.

Fracture frequency is generally uniform throughout the Mitchell deposit. It does not appear to vary significantly by location. The average fracture frequency is approximately 1 fracture per metre.

Correlations of fracture frequency with other geotechnical/geological parameters were evaluated in great detail. This included an assessment of the effect of alteration type, rock fabric (i.e., massive, foliated, or stockwork), frequency of closed veins, and intensity of micro-defects. A slightly higher fracture frequency was identified for rock logged with the IARG alteration type and rock logged as having stockwork veining. The differences are not considered significant.

5.5 Fracture Persistence

During the 2011 field program, Golder conducted geotechnical mapping along four traverses on rock outcrops at Mitchell. Traverse locations, mapping photos, and geotechnical mapping data sheets are included in the factual report (Golder 2012a).

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 12.

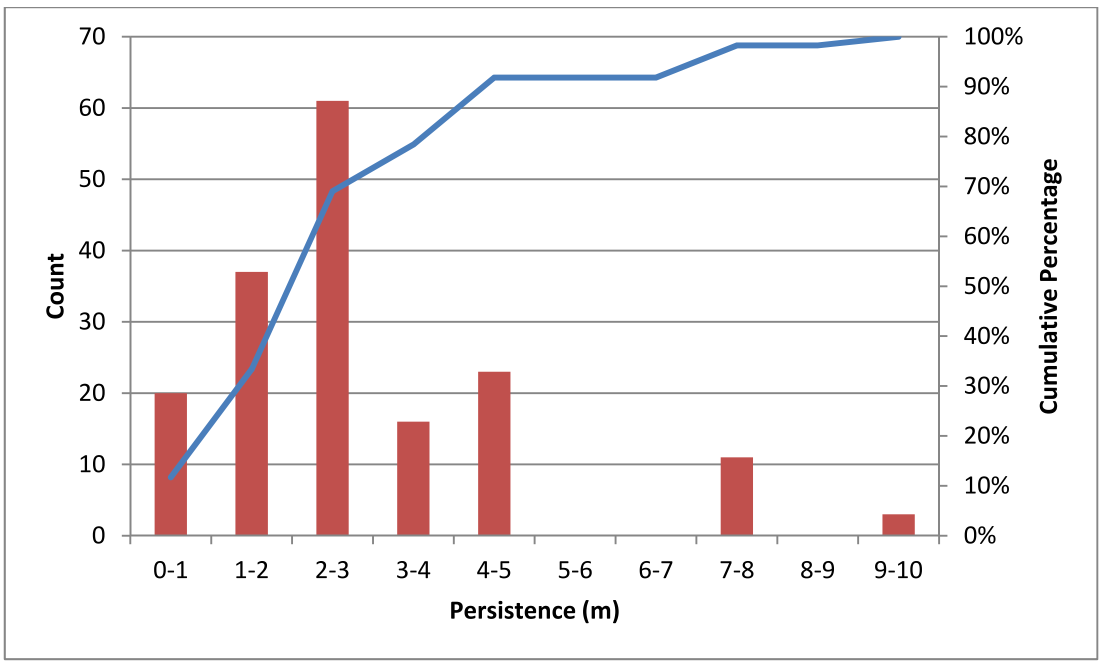


Figure 16: Persistence distribution of all mapped features.

Table 12: Distribution of Termination of Mapped Features

Termination	Number of Mapped Features
0	12
1	30
2	26

5.6 In Situ 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 as well as the geometry of the associated intact rock blocks.

5.6.1 DFN Model Input and Verification

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

- Distribution of fracture orientations obtained from borehole televiewer data from M-09-095, M-09-096, and M-09-099;
- Distribution of fracture spacing from boreholes within the Mitchell deposit (M-09-095, M-09-096, M-09-099, M-09-123, M-09-124, M-09-125, M-09-126); and
- Distribution of fracture persistence from fracture geometry information collected from outcrop mapping during the 2011 field program (Golder 2012a).

Details on these input parameters are contained in the Golder geotechnical characterization report (Golder 2012b).

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.

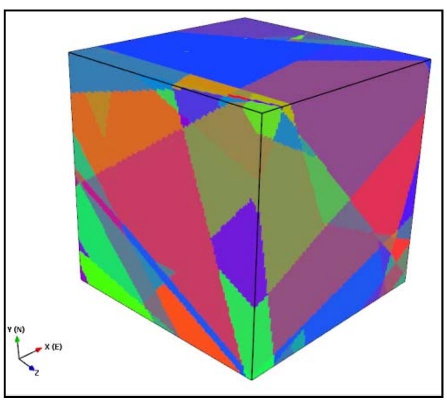


Figure 17: Example of Mitchell 5x5x5 m DFN model.

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 6 m^3. This represents a very coarse block size for caving mining. The implications of this are discussed in Section 6.2.

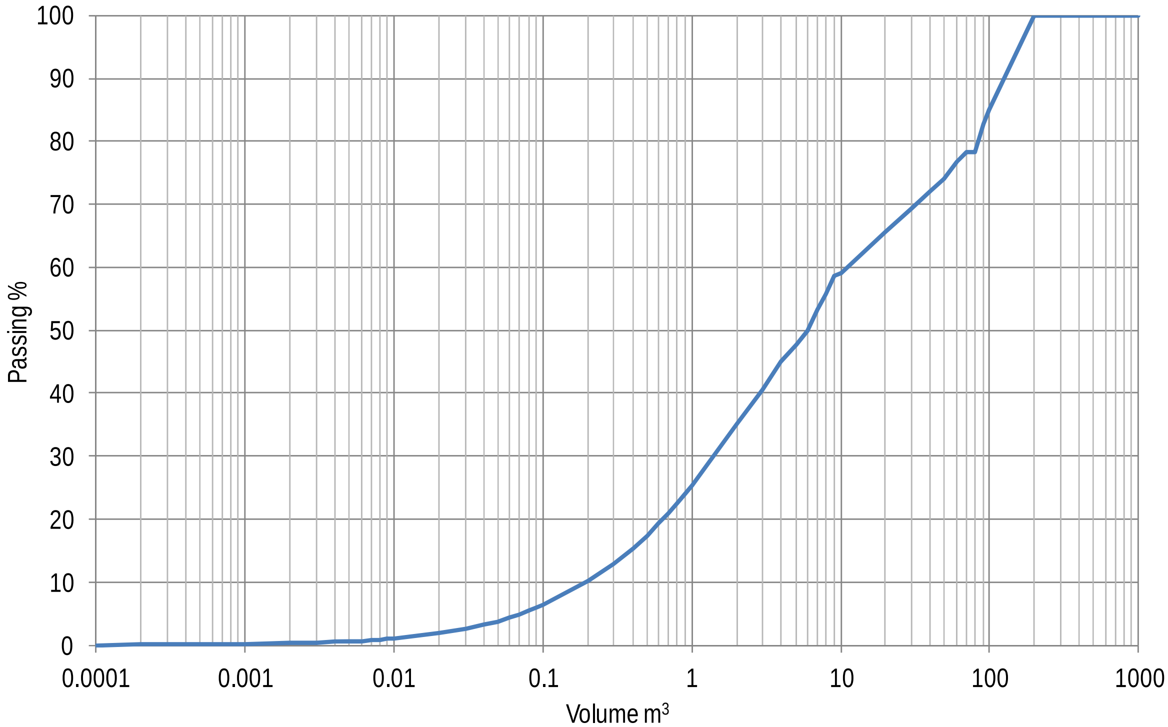


Figure 18: Estimated block size from DFN modelling.

5.7 In Situ Stress

In situ stress has been estimated from the results of hydraulic fracture tests in borehole M-11-122. Detailed methodology, analyses, and test results are provided in the factual report for the 2011 field investigation (Golder 2012a).

A summary of estimated in situ stresses from each of the seven tests is presented in Table 13. Note, however, that there are some uncertainties in the calculated principal stresses listed in this table. For example, the vertical stress is calculated based on overburden depth. The results of numerical models, however, suggest that the vertical stress in the floor of the valley (at the location of the tests) may be higher than this. The limitations of the tests are discussed in more detail in the Golder field investigation report (Golder 2012a). The calculated stresses should be considered as an indication of potential stress levels at these locations rather than an accurate estimate of the in situ stress.

Table 13: Summary of In Situ Stress Values from Hydraulic Fracturing in Borehole M-11-122

Field Test No.	Depth (m)	Alteration[1]	σ_{HMax} (MPa)	σ_{HMin} (MPa)	σ_v[2] (MPa)	Tensile Strength[3] (MPa)	Pore Pressure[4] (MPa)
7	158.0	QSP	19.5	8.6	4.4	11.2	1.6
6	384.5	CL-PR	47.2	20.5	10.7	11.6	3.8
5	442.0	CL-PR	34.8	16.0	12.3	13.3	4.5
4	511.0	CL-PR	37.7	16.5	14.2	13.1	5.2
3	570.9	CL-PR	39.3	19.5	15.9	12.1	5.8
2	604.4	CL-PR	30.3	15.0	16.8	12.4	6.1
1	608.9	CL-PR	37.9	20.3	16.9	10.9	6.1

Notes: [1] Alteration types were provided by Seabridge.

[2] Vertical stress was calculated based on the average overburden thickness over the test interval using an estimated density of 2781 kg/m^3.

[3] Determined from laboratory testing.

[4] Pore pressure was calculated based on the column of water at each test interval depth.

Hydraulic fracture orientations were collected for three intervals using impression packers. The orientations suggest that the maximum horizontal stress acts across the valley (roughly north-south) and the minor horizontal stress is oriented along the valley (roughly east-west), as would be expected.

5.8 Hydrogeological Characterization

Hydrogeological testing was carried out as part of the 2009 field program (BGC 2010) and the 2011 field program (Golder 2012a).

In 2009, BGC conducted a total of nine hydrogeological tests below the MTF in the central boreholes (M-09-095, M-09-096, and M-09-099). Hydraulic conductivity values calculated from these tests were presented in BGC's pre-feasibility report for the open pit (BGC 2010). The data indicated hydraulic conductivity values ranging from 1 x 10^{-9} to 1 x 10^{-7} m/s below the MTF. The highest hydraulic conductivities (1 x 10^{-7} m/s) were calculated from tests conducted at the highest elevations (greater than 800 metres above sea level).

In 2011, Golder conducted a total of 21 hydrogeological tests in five geotechnical boreholes (M-11-122, M-11-123, M-11-124, M-11-125, and M-11-126). The results of the hydrogeological investigation were discussed in "2011 Geotechnical and Hydrogeological Field Investigations, Mitchell Project" (Golder 2012a). Artesian conditions were observed in boreholes M-11-122, M-11-123, M-11-124, and M-11-126, with vertical static water levels ranging from 9.1 to 33.2 m above ground surface. Vertical static water levels in M-11-125 ranged from 27.2 to 35.2 m below ground surface.

Hydraulic conductivity values calculated from the 2011 hydrogeological tests ranged from 3 x 10^{-10} to 4 x 10^{-6} m/s. The results indicated a general trend of increasing hydraulic conductivity with elevation. This trend generally agrees with the 2009 data.

There is no indication of a correlation between hydraulic conductivity and RMR.

6.0 CAVING GEOMECHANICS

6.1 Cavability

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

In situ stresses 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. Simple 3D elastic numerical models were developed to estimate mining-induced stresses in the back of the cave. The initial model conditions were calibrated to the results of the in situ stress measurements. The results were presented in a technical memorandum titled "Mitchell Block Caving Stress Modelling," included here as Appendix C.

Based on the high induced stresses (ranging from approximately 72 to 127 MPa) in the cave back predicted by the numerical models, it is expected that stress-induced fracturing of the rock mass may contribute to caving. However, given the simplicity of the models at this stage of study, the benefits of potential stress failures in the cave back should be viewed as a potential upside and have not been considered in this design study. More sophisticated numerical analyses are recommended to confirm and quantify stress-related impacts as part of future studies.

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 between the block cave extraction level (El. 235 m) and the proposed pit floor (El. 405 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 51 for the Mitchell deposit. Parameters used to estimate MRMR are outlined in Table 14.

Table 14: MRMR Rating Classification

Parameter	Description	Rating
Intact rock strength	97 MPa	10
RQD	98%	15
FF/m	2 joint sets, average spacing 2.8 m	25
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		**71**
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		**51**

As shown in Figure 19, the minimum hydraulic radius (HR) of the undercut that would be predicted to cave (based on empirical case studies) is approximately 28 m. This equates to an approximate area of 110 m by 110 m.

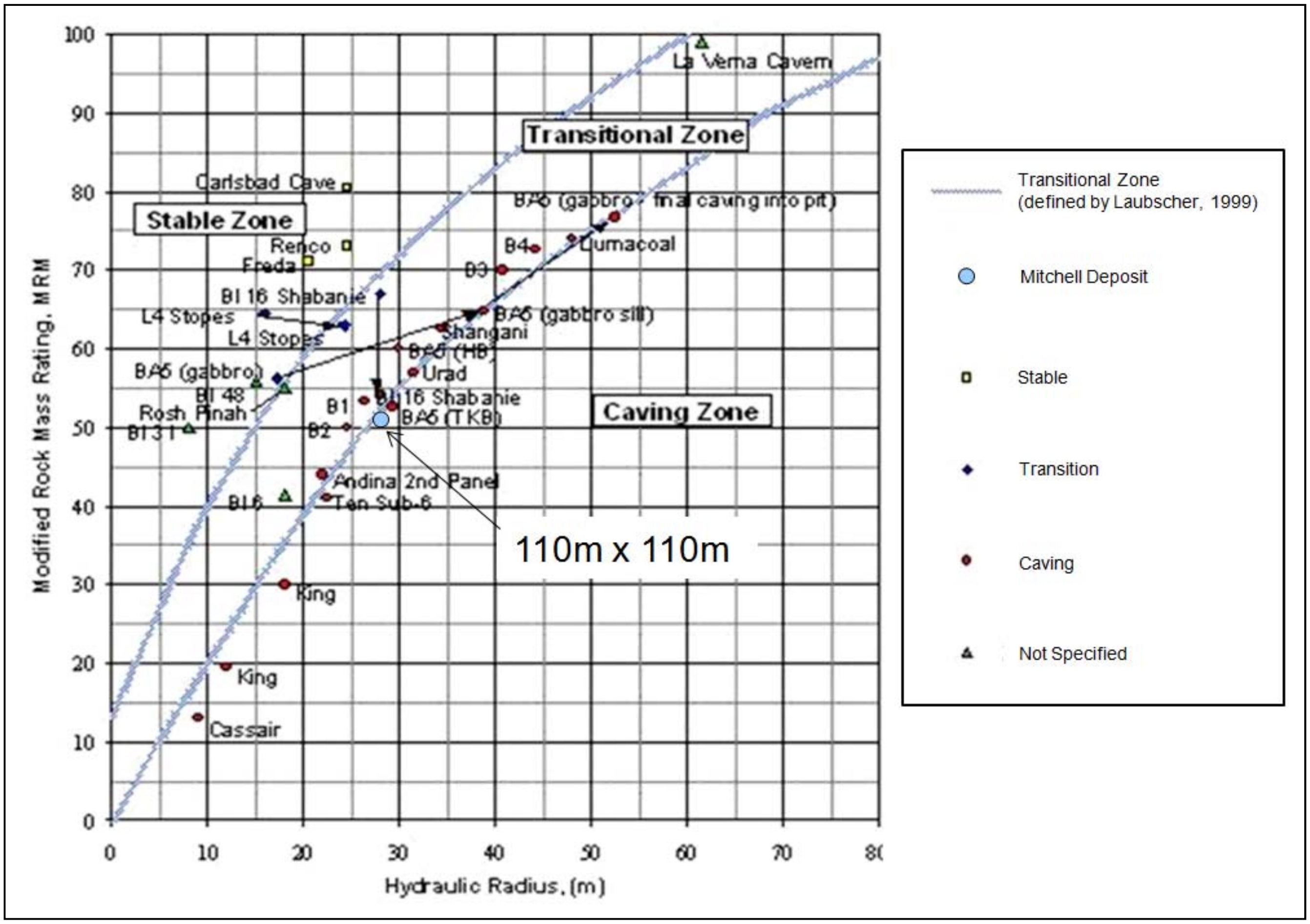


Figure 19: Cavability assessment using Laubscher's method (Laubscher 1999).

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 Mitchell is 1.6. A summary of the parameters used to estimate N is contained in Table 15.

Table 15: Q' and N Rating Classification

Parameter	Description	Rating
Q'	(RQD/Jn) × (Jr/Ja)	20
Factor A [1]	$\sigma_c / \sigma_1 \approx 1$	0.1
Factor B [2]	Dominant joint set dipping at approximately 60 degrees	0.8
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 Mitchell rock core samples. Average maximum induced compressive stress (σ_1) estimated from numerical modelling.

[2] Joint orientation estimated from stereographic projections produced from Mitchell televiewer and oriented core logging 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.

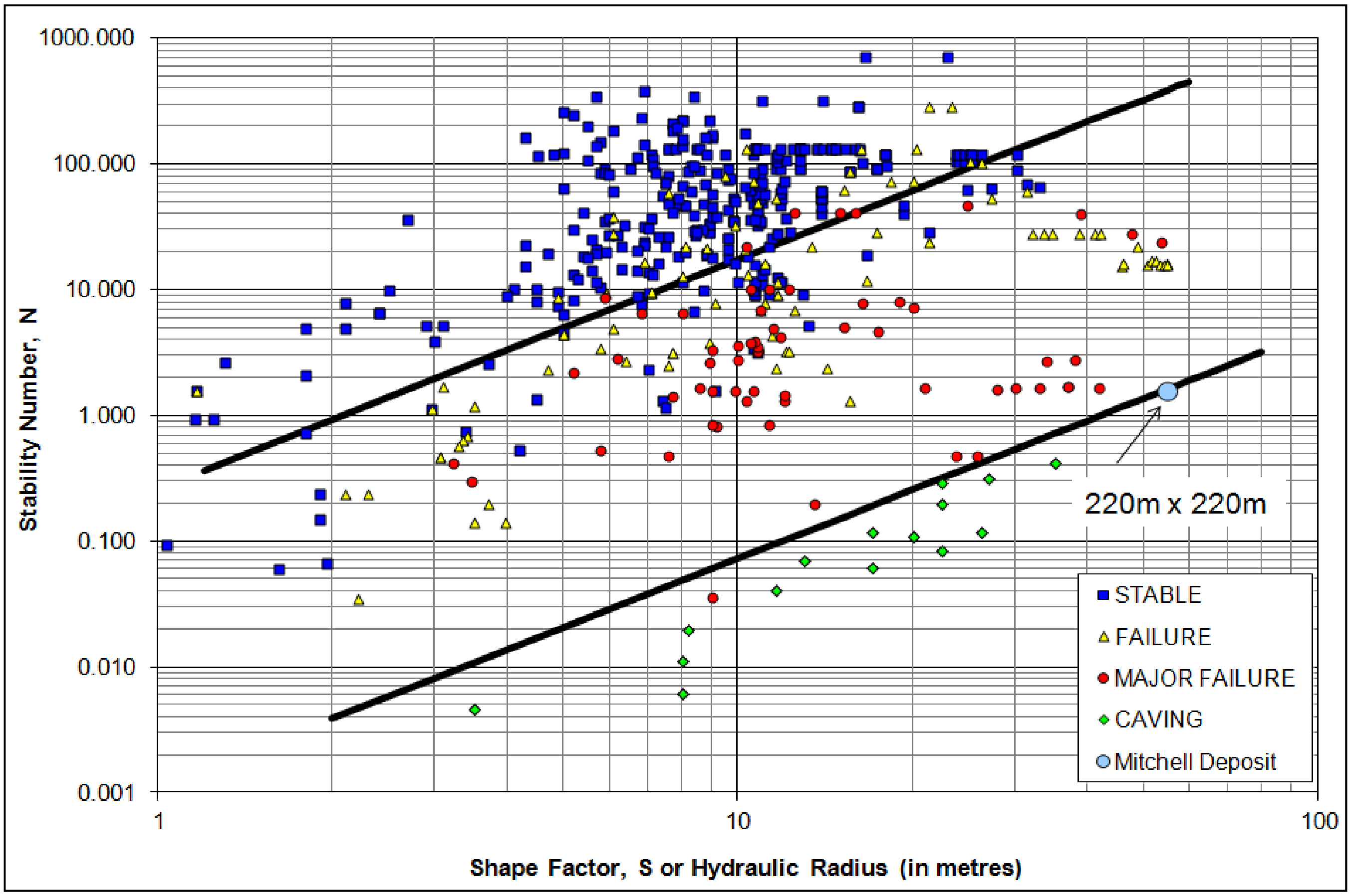


Figure 20: Cavability assessment using the Mathews extended stability graph (Trueman and Mawdesley 2003).

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 110 m and 220 m. These dimensions are significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized three-dimensional shape of the deposit, suggest that the Mitchell 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.

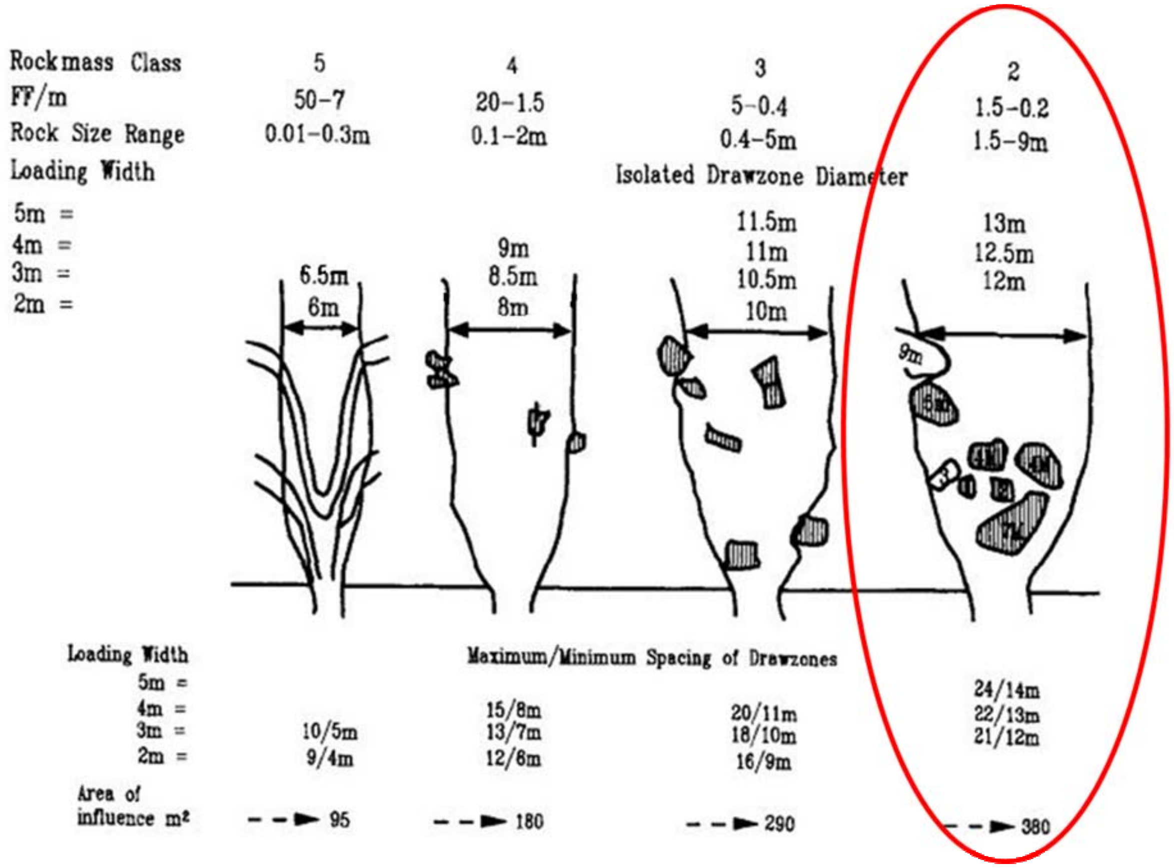


Figure 21: Maximum/minimum spacing of drawzones based on isolated drawzone diameter (Laubscher 1994).

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 towards 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 Mitchell 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 in Figure 18. The median block size is approximately 6 m^3. This represents a very coarse block size for cave mining. A comparison between the Mitchell 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 that have 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.

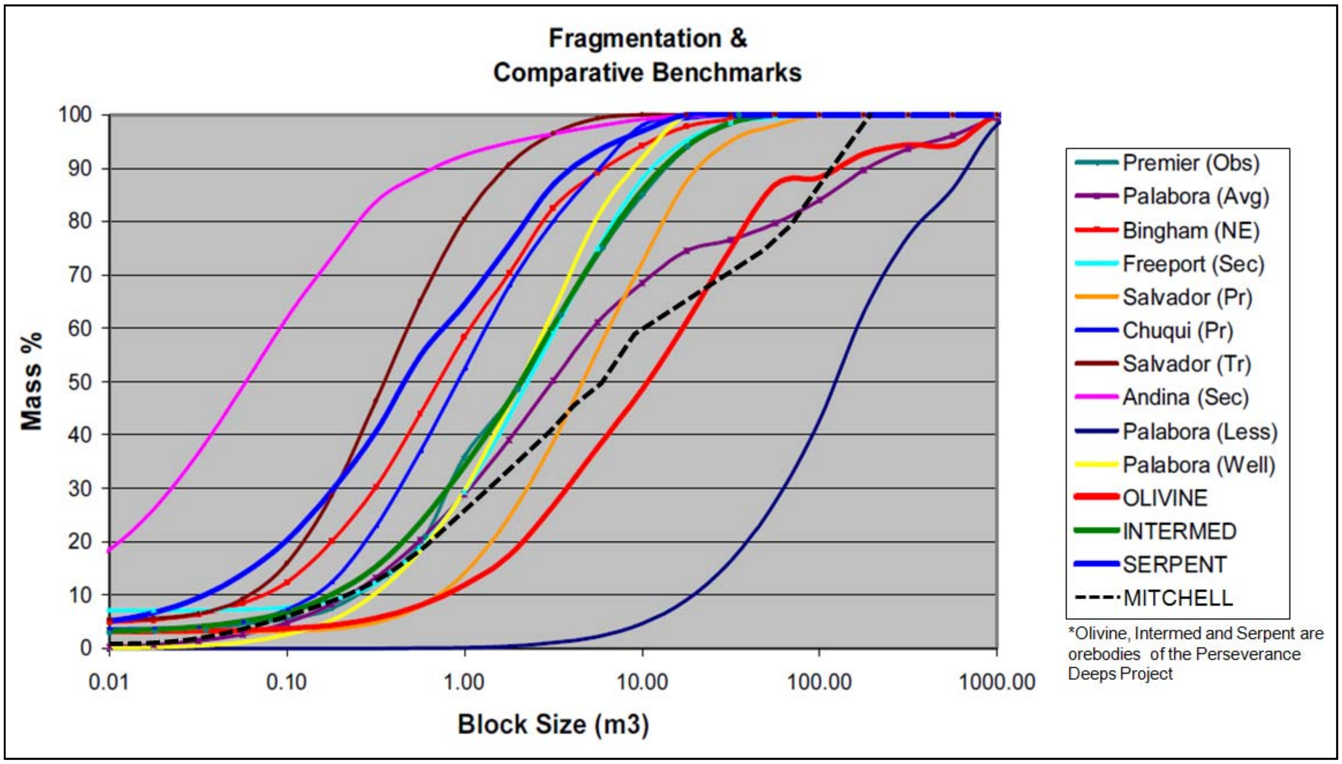


Figure 22: Comparison between the estimated block size at the Mitchell deposit and existing block caving operations (Butcher and Thin 2007).

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 towards the drawpoints.

Some preliminary numerical models have been developed to obtain an indication of the level of induced stress in the back of the cave at different stages of cave development. The 3D elastic models were constructed in Map3D™. These analyses are discussed in Appendix C. The results indicate that stresses in the back of the cave may approach the intact strength of the rock. This suggests that there may be some stress-induced fracturing that develops, but more sophisticated analyses would be required to quantify this impact. For this reason, the potential impact of stress on fragmentation has been ignored in this present study. The results do suggest that the estimates of fragmentation discussed here (based on in situ block size estimates) may be somewhat conservative.

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

The 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 to precondition the rock 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 Mitchell. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning to enhance fragmentation needs to be addressed via production and cost risks (as discussed in Sections 7 and 11).

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. The important objective is to maintain full interaction between individual neighbouring draw columns. 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 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

A preliminary evaluation of the likely extent of surface subsidence associated with the proposed block cave has been undertaken. As part of future studies (e.g., feasibility study), it will likely be necessary to undertake some detailed numerical analysis studies of the subsidence and surface disturbance that might be precipitated by the caving mining at Mitchell. This would need to incorporate the direct disturbance of the caving, the response of the pit walls, and the response of the valley walls above and to the periphery of the pit. For the current pre-feasibility study, an empirical approach is considered satisfactory.

This assessment has hypothetically assumed that the ground surface is flat and there is no pit. Under these circumstances, empirical evidence suggests that, for the type of geological/structural conditions at Mitchell, a crater typically develops on surface above and slightly laterally beyond the footprint of the production horizon of the caving mining. The top section of the crater is a relatively steep escarpment (60 to 70 degrees) that is marginally stable but comprised of nominally in place dilated rock. Beneath this is failed broken rock that has progressively sloughed from the rim of the crater. This rock rills down to the bottom of the crater at about 40 degrees. Beyond the rim/crest of the crater, significant surface cracking is evident that becomes progressively less pronounced as the distance from the crest increases. Based on the experience at other mines operating in similar geological conditions to those that exist at Mitchell, both for flat and moderately inclined ground surfaces, the potential width of the disturbed cracking zone (i.e., the distance to the last observable crack) is estimated to be approximately 100 m.

Starting with the footprint of the proposed caving mining at the production level elevation, this general model has been extrapolated up to the walls of the slopes of the open pit. As shown in Figure 23, the estimated limit of the surface subsidence lies within the outer boundary of the open pit. Cross-sections showing the estimated profiles depicting the angle of repose of failed rock, the steep escarpment just beneath the crest of the crater, the crest of the crater, and the disturbed zone of surface cracking beyond the crest are shown in Figure 24 and Figure 25.

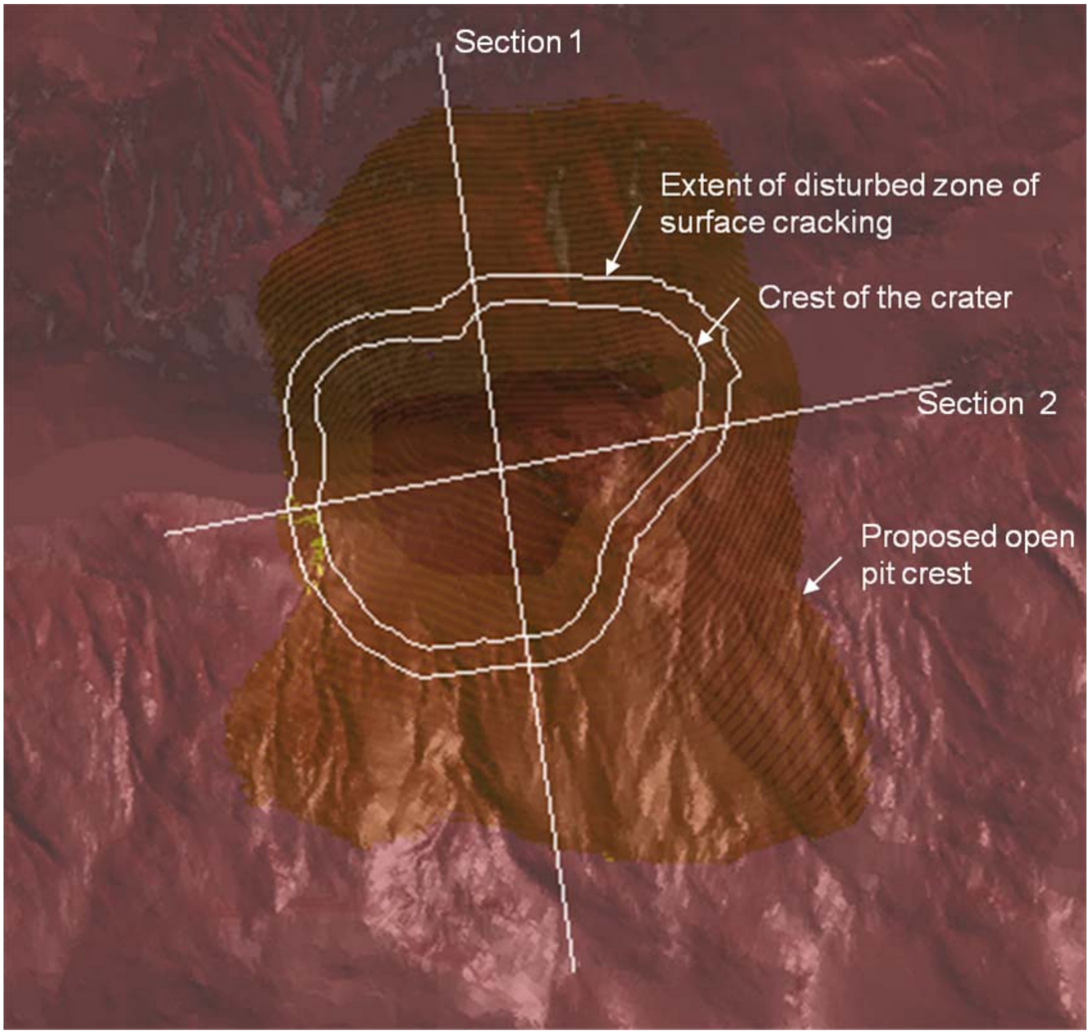


Figure 23: Estimated limit of surface subsidence.

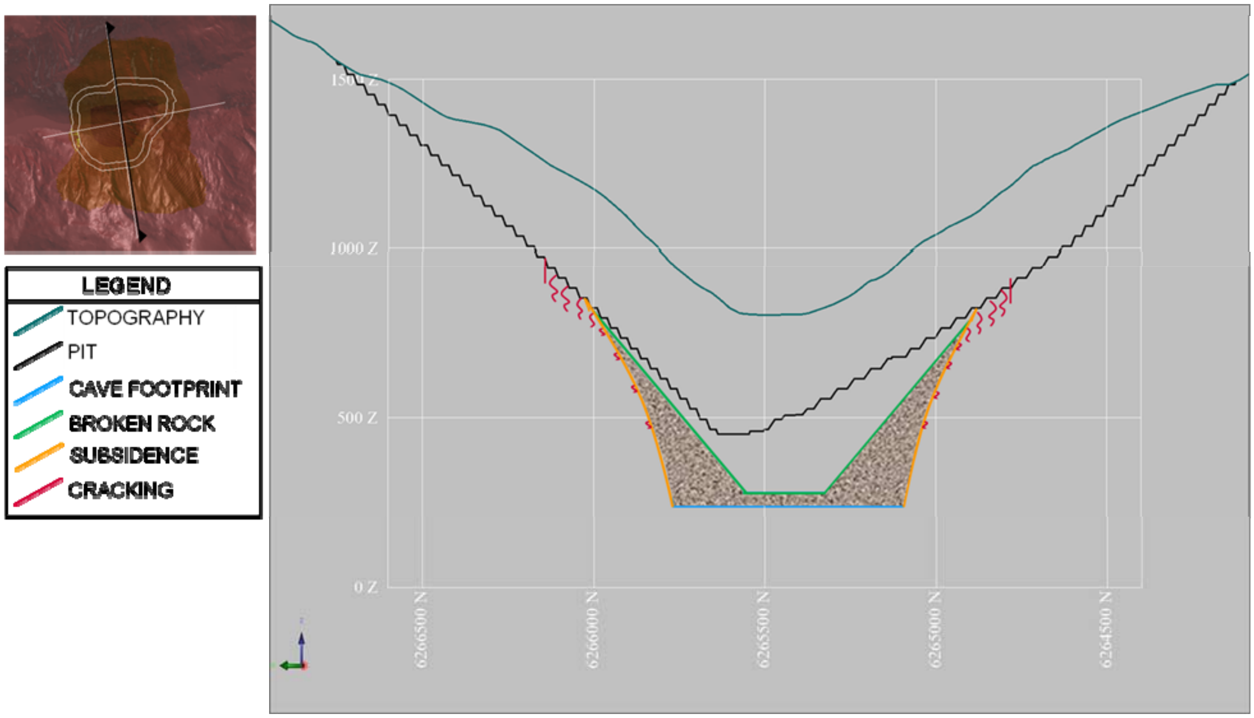


Figure 24: Schematic of subsidence profile – section 1.

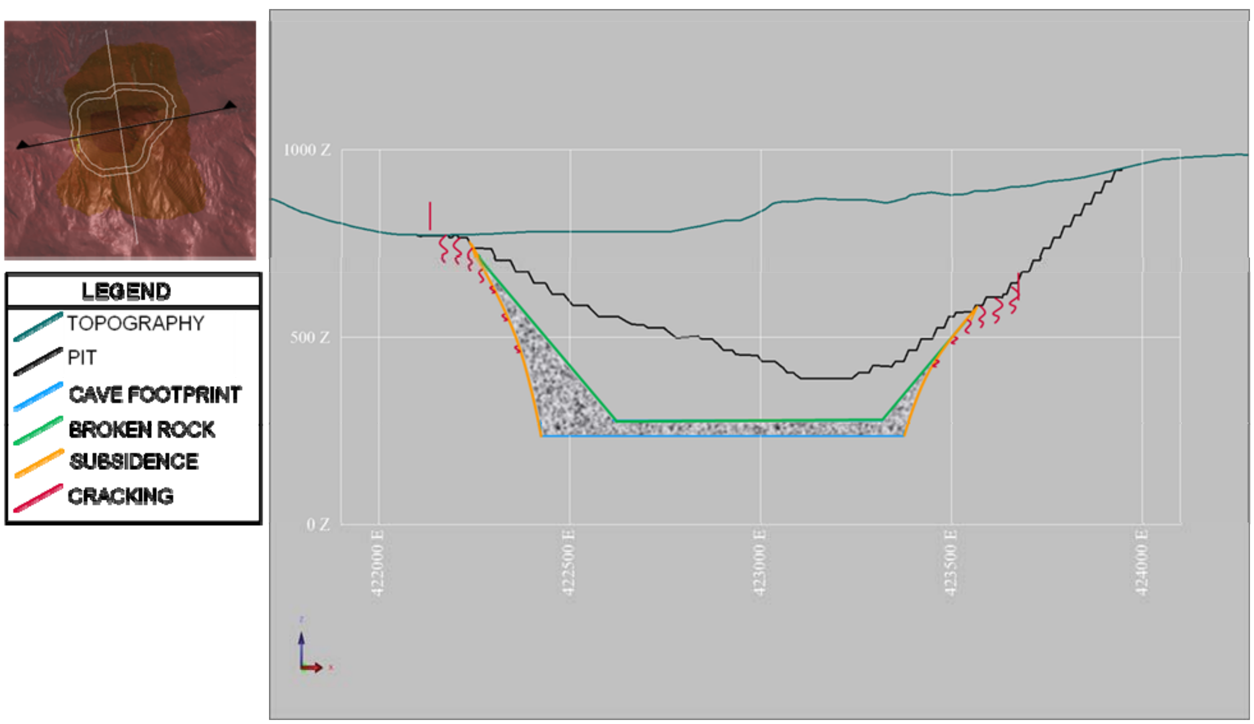


Figure 25: Schematic of subsidence profile – section 2.

Estimates of the extent of disturbance of the pit and natural stopes has been undertaken to assess whether the infrastructure on the surface is likely to be impacted in any way. This has taken into account the additional potential instability of the slopes beyond the disturbed cracking zone that the formation of the crater may precipitate. Based on these estimates, it is unlikely that any surface infrastructure will be impacted during the caving mining and after closure. In particular, there are no concerns regarding the proposed location of infrastructure to the west of the pit adjacent to the south portals of the Mitchell Teigan Tunnels (MTT), and to the excavations to the east of the pit required for the Mitchell Diversion Tunnels (MDT).

Further assessments of the possible impact of the caving mining and the formation of the crater on the stability of the adjacent slopes are presented in separate reports.

7.0 MINE DESIGN

The Mitchell deposit is large in three dimensions and reasonably continuous, which makes it suitable for a high tonnage, low cost mining method. The current design includes mining the upper portion of the deposit by open pit and the lower portion as a block cave. This section refers to the underground block cave portion of the mine design. The open pit design was completed by MMTS.

The underground mine 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 Mitchell deposit is approximately 728 m wide in the north-south direction, 1,022 m wide in the east-west direction, and 500 m vertically with the footprint elevation at 235 m. PCBC modelling indicated that the block cave could produce 55,000 tonnes per day, requiring the development of 120 new drawpoints per year. The mine design involves approximately 145 km of drifts and raises, including a 5% equivalent contingency to account for the excavation of design items such as service bays, sumps, and electrical substations.

The mine design is composed of six main types of levels including preconditioning, undercut, extraction, secondary breakage, haulage, and conveying. In addition, there are two tunnels (access ramp and conveyor) from the various underground working levels to surface. This section will describe the function of each level. Detailed drawings of each level can be found in Appendix D. Figure 26 and Figure 27 show a plan view and section view of the proposed mine layout and major infrastructure, respectively.

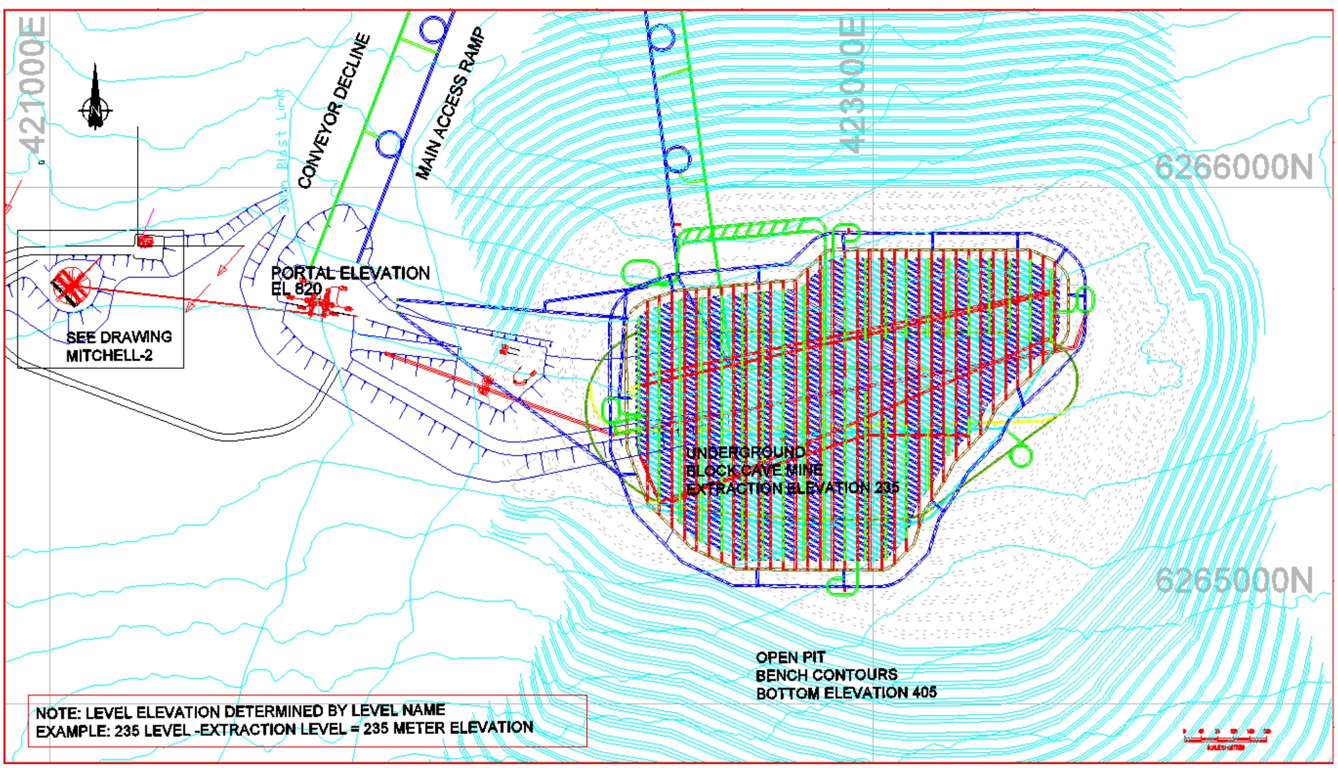


Figure 26: Plan view of the proposed underground mine layout.

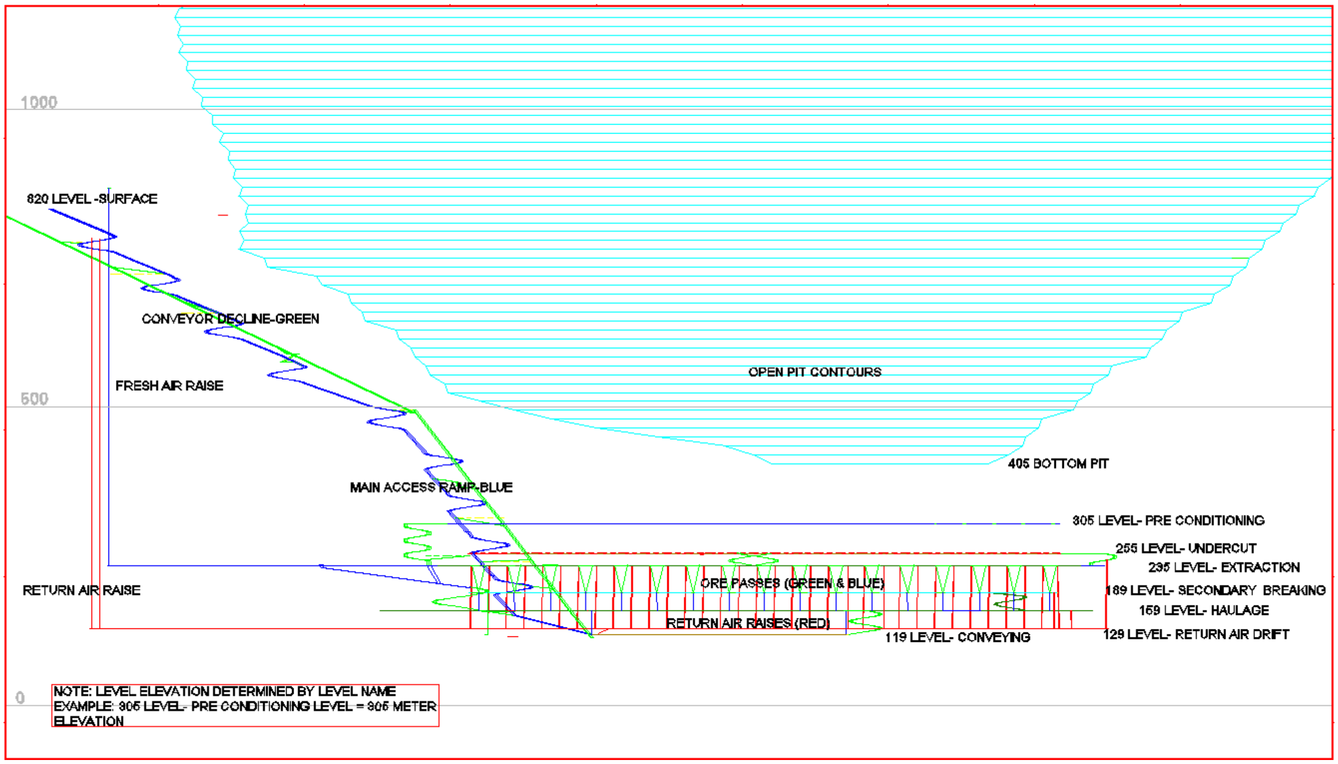


Figure 27: Section of the Mitchell mine design (looking south) showing the position of each of main levels in the mine.

7.1 Underground Access

Personnel, material, and supplies will access the underground through a main access ramp which will be developed from a portal near the Ore Process Control (OPC) area at the 820 m elevation. The access ramp from the surface is graded at 15% with a total length of 6.3 km. It is designed to be 5.0 m by 5.0 m wide to allow all of the underground equipment, including crusher parts, to be transported underground (underground equipment is listed in Section 7.10).

A second ramp to the surface will be excavated to accommodate the conveyor. The conveyor tunnel portal is 100 m away from the main access ramp portal, and both ramps will be connected every 300 m to improve ventilation and development rates during construction and to provide a secondary egress during operation. The conveyor decline will transport all mined material to surface. It is separated into two arms, each approximately 2.0 km long and grading at approximately 17%. It is designed to be 5.5 m wide and 4.5 m high, with the belt on feet, on the floor. Figure 28 shows a plan view of the proposed access ramps and tunnels. The estimated length of the access ramp and conveyor tunnels is shown in Table 16.

A trade-off study was completed comparing the cost of moving material to surface through a shaft or through a conveyor. The study assumed that a ramp to the underground would be needed in both scenarios and that one 10 m shaft or two 7 m diameter shafts would be required to move 55,000 tpd. The results of the study indicated that a conveyor would be cheaper and more flexible than a shaft. More details concerning this trade-off study can be found in Appendix A.

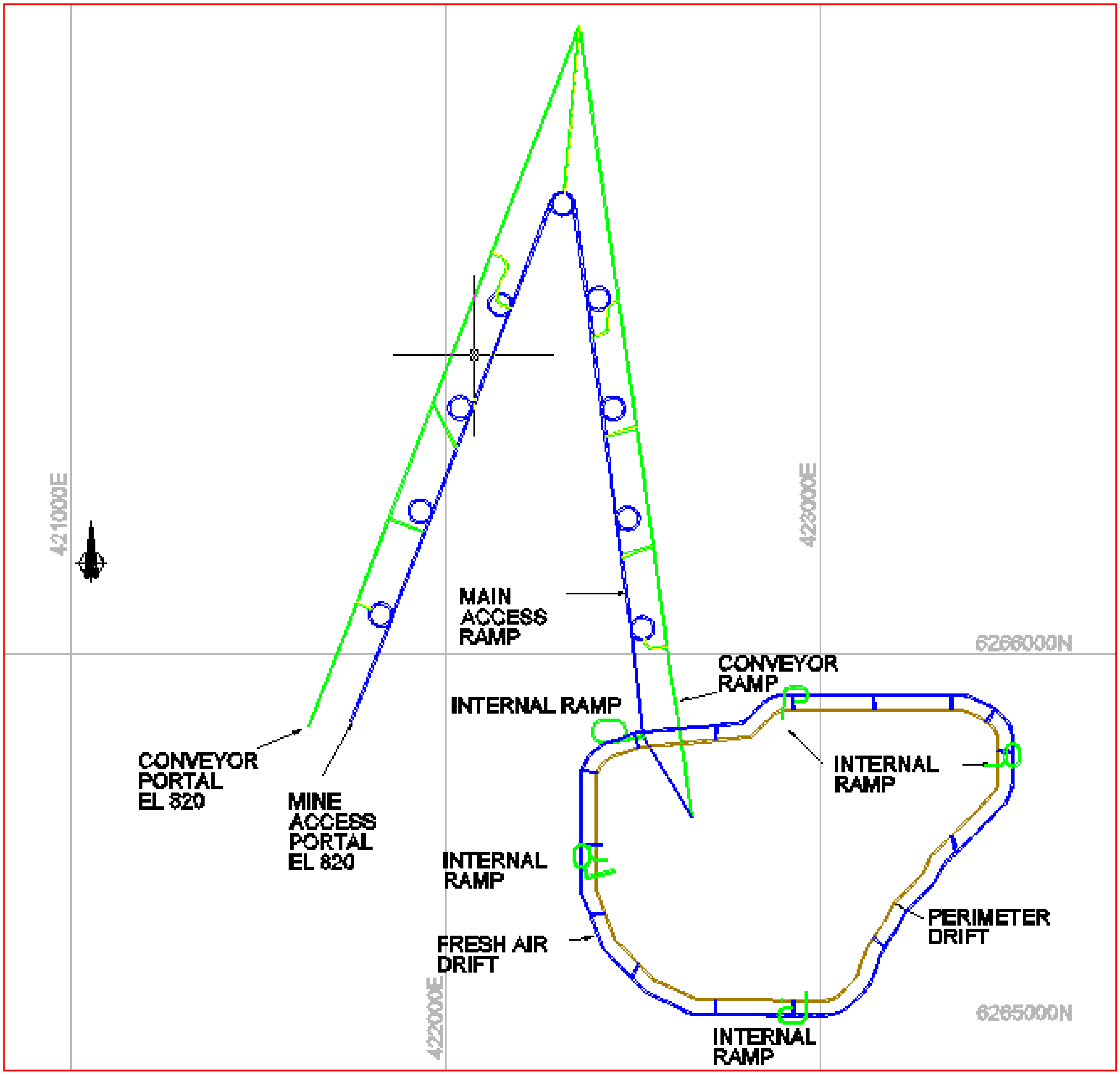


Figure 28: Plan view of the proposed Mitchell mine access.

Table 16: Design Lengths of the Conveyor and Access Ramps

Item	Length (m)
Access ramp	6,300
Conveyor tunnels	4,600

7.1.1 Emergency Egress

The Mitchell underground is designed with two portals, one each for the access ramp and conveyor drifts. These two tunnels will have opposite airflow (i.e., if one is blocked because of a fire, the other should contain clean air). Therefore, one of the tunnels will be the primary egress, while the other will be the emergency egress.

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 29. From this level, as well as from the bottom of the pit, 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 majority of the cave footprint area. Each of the PC drifts is connected to the main drift at two points that will provide access and ventilation. The drifts on the PC level are 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 11,700 m of PC drifts will be needed for the Mitchell mine. The part of the footprint area that is not covered by the PC drifts will be hydrofractured from the bottom of the pit.

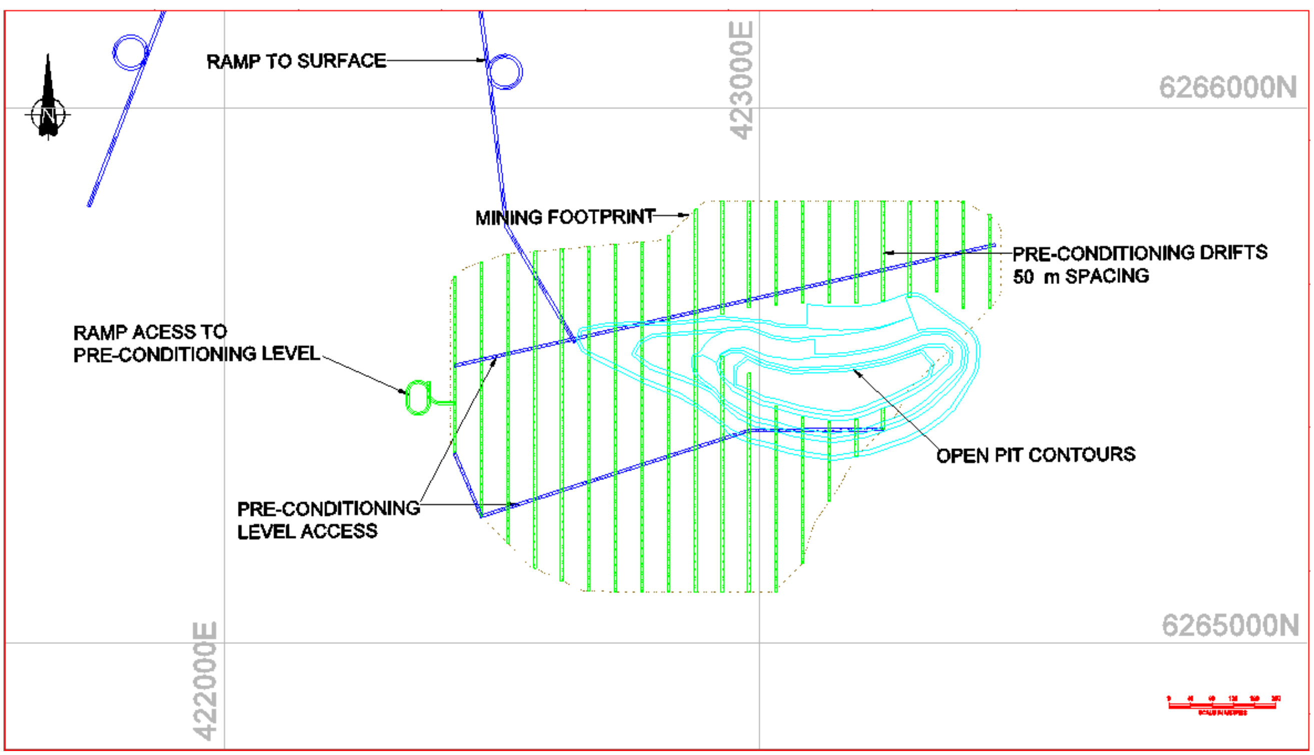


Figure 29: A plan view of the preconditioning (PC) level showing the coverage of the footprint by both the PC level and the open pit.

7.3 Undercut Level

Blasting from the undercut level (UC) initiates and propagates the cave. A plan view of this level is shown in Figure 30. Undercutting will be done using the drilling patterns shown in Figure 31, 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 mass qualities similar to what is expected in the Mitchell deposit, suggests that this drilling pattern is sufficient 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 and ventilation to the UC drifts. Figure 30 shows the position of the UC drifts relative to the cave footprint. To accommodate the drilling equipment necessary, the drifts on the UC level are designed to be 4.0 m wide and 4.0 m high. The Mitchell block cave design includes approximately 20.7 km of UC drift.

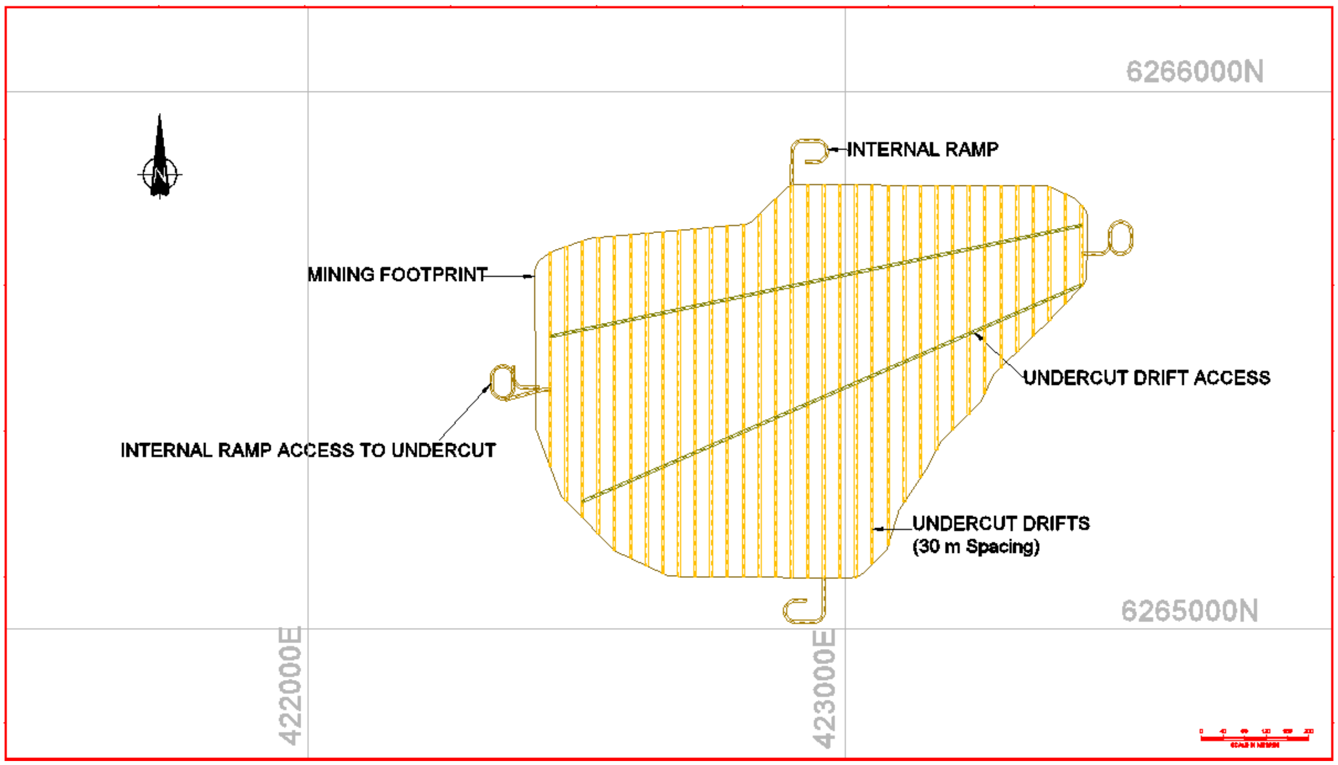


Figure 30: Plan view of the undercut level.

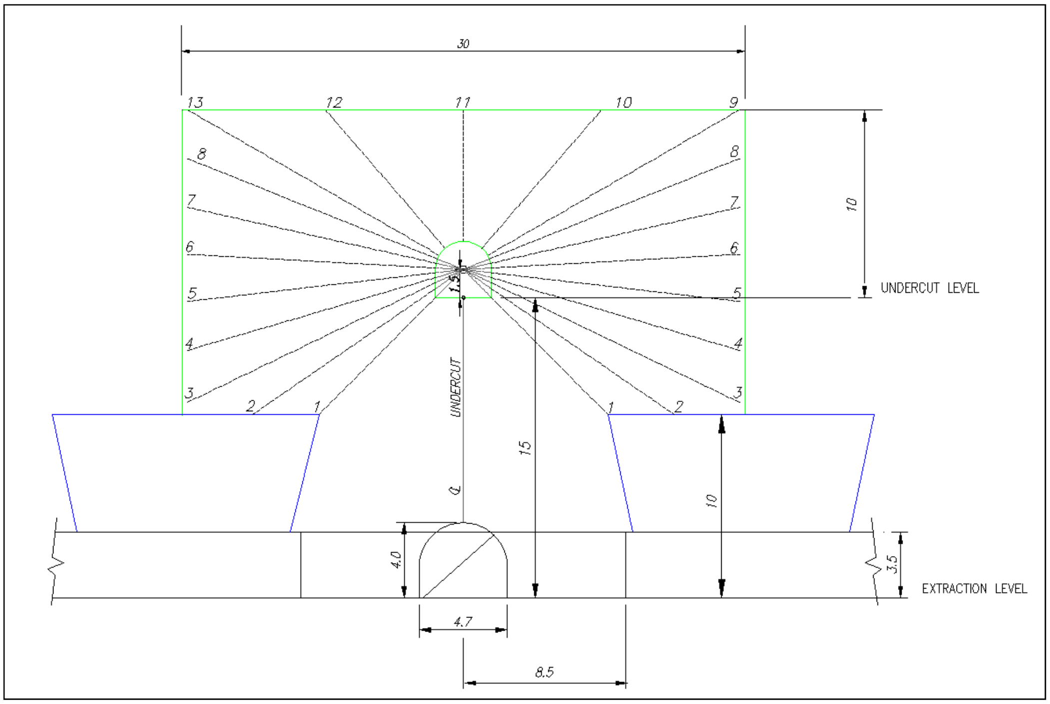


Figure 31: Schematic cross-section showing the relationship between the undercut and the extraction levels and drill pattern used to initiate the cave.

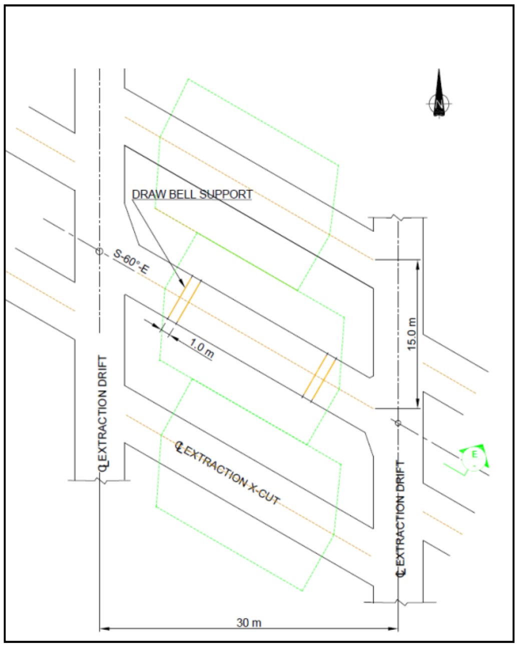
7.4 Extraction Level

The extraction level was designed to accommodate the estimated fragmentation of the Mitchell deposit cave and to be as productive as possible. To allow for the appropriate overlap between the drawcones (a complete definition of block caving terms can be found in Appendix E), 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 32. 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 cross-section of 4.5 m by 3.5 m.

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 reduces the impact of a mudrush. Figure 32 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.

The six levels of the mine design will be accessed through internal ramps beginning on the extraction level. These ramps are strategically positioned to maintain access to the levels during caving and for ventilation purposes.

There are 34 extraction drifts on the extraction level and each drift is designed with three ore passes. The average LHD haul distance is approximately 100 m which provides for good productivity. Additional information concerning the design of the material movement system can be found in Section 7.9.



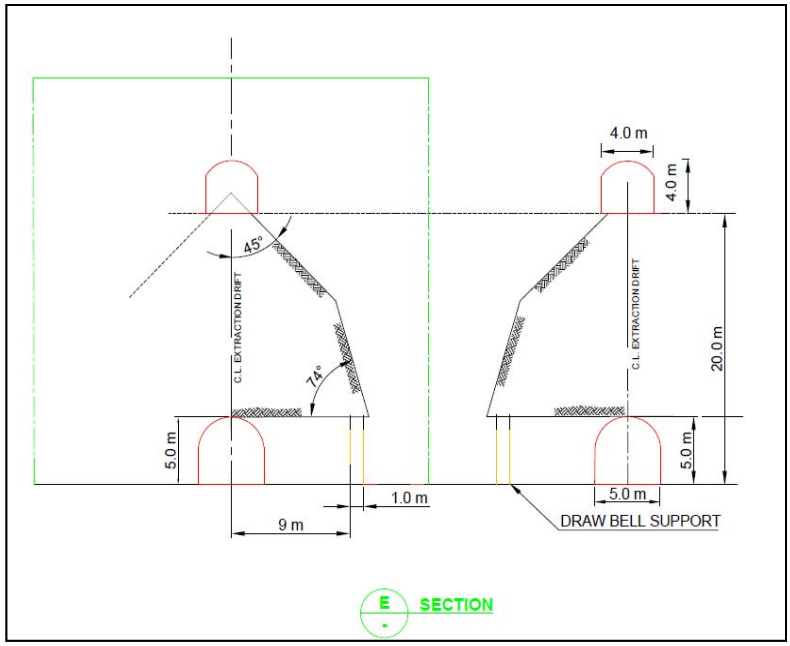


Figure 32: Diagram showing the relationship between the drawbells, drawpoints (extraction x-cut) and extraction drifts.

The majority of the main ventilation infrastructure is also located on the extraction level. It consists of two fresh air raises, two fresh air drifts, a fresh air ring drift, multiple internal ventilation raises, a return air drift, and two exhaust raises. The internal ventilation raises are located below and approximately in the middle of the footprint, which allows for 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 17.

Table 17: Estimated Lengths of the Various Drift Types for the Mitchell Deposit

Item	Length (m)
Internal ramps	2,700
Extraction drifts	20,400
Drawpoints	43,000
Perimeter drift	3,100
Return air drifts	1,700
Fresh air drifts	4,900
Ventilation air raises	6,800
Ore passes	5,500

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 33 and Figure 34 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 ground support can be found in Section 7.8.

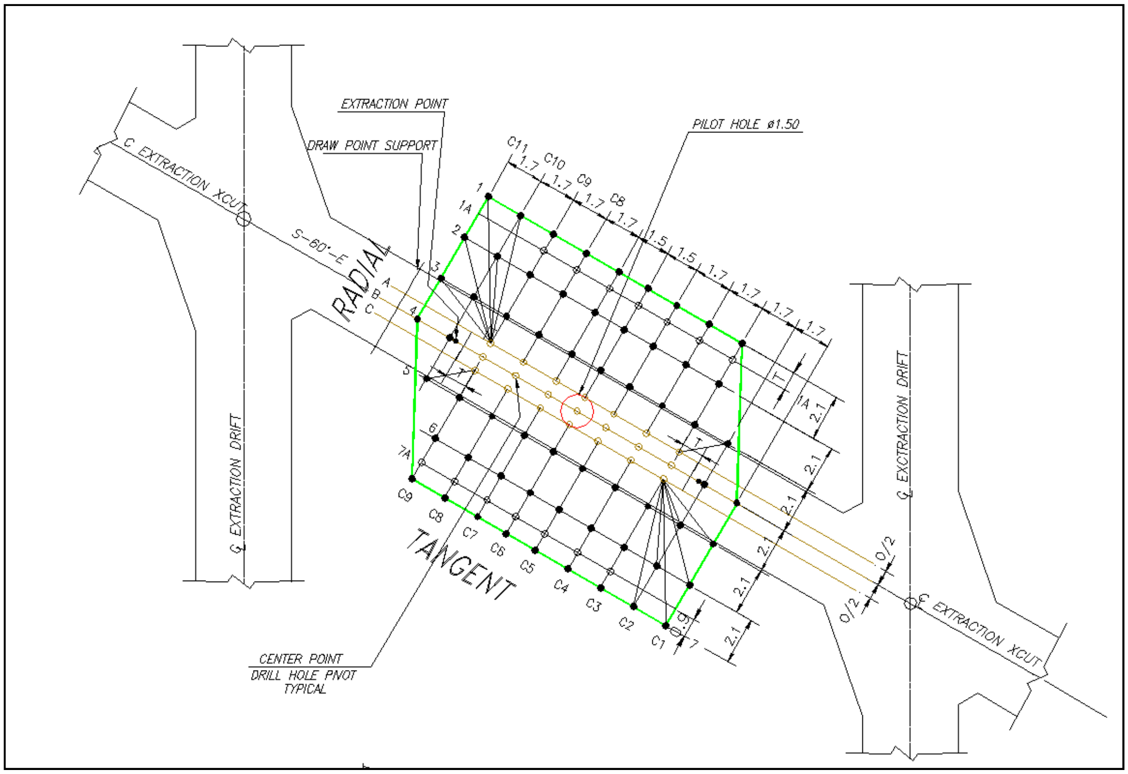


Figure 33: Plan view of the drilling pattern used for the drawbell excavation.

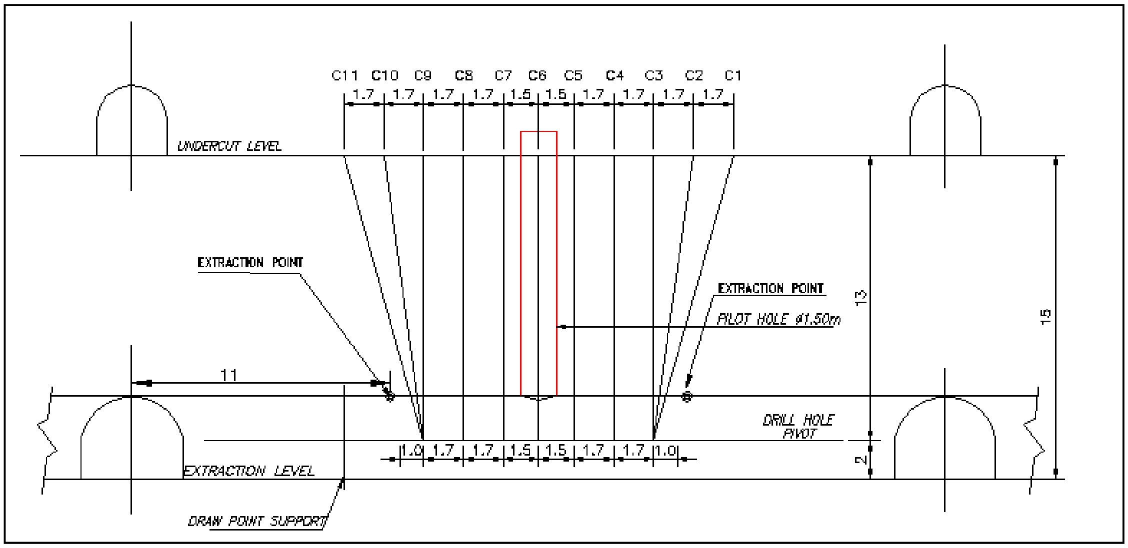


Figure 34: Plan and section view of the proposed drilling and blasting pattern for the drawbells used in the El Teniente layout.

7.5 Secondary Breakage Level

The secondary breakage level (SBL), shown in Figure 35, is designed to provide access to stationary rockbreakers. This level is located between the extraction and haulage levels, and is required to reduce the size of the broken material so it can be hauled with a train on the haulage level. It is located approximately 40 m below the extraction level and designed to be 4 m wide by 4 m high to accommodate an LHD, which is expected to be the largest piece of equipment on the level. It will contain approximately 90 secondary breaker stations, which are designed to be 15 m wide by 10 m high to accommodate the rockbreaker and the incoming and outgoing ore passes, and it will be accessed through ramps located on the north end of the footprint. The current design requires 3.6 km of SBL and 800 m of rockbreaker stations. The SBL is an important part in the material movement system designed for Mitchell. A complete description of the use of this level can be found in Section 7.9.

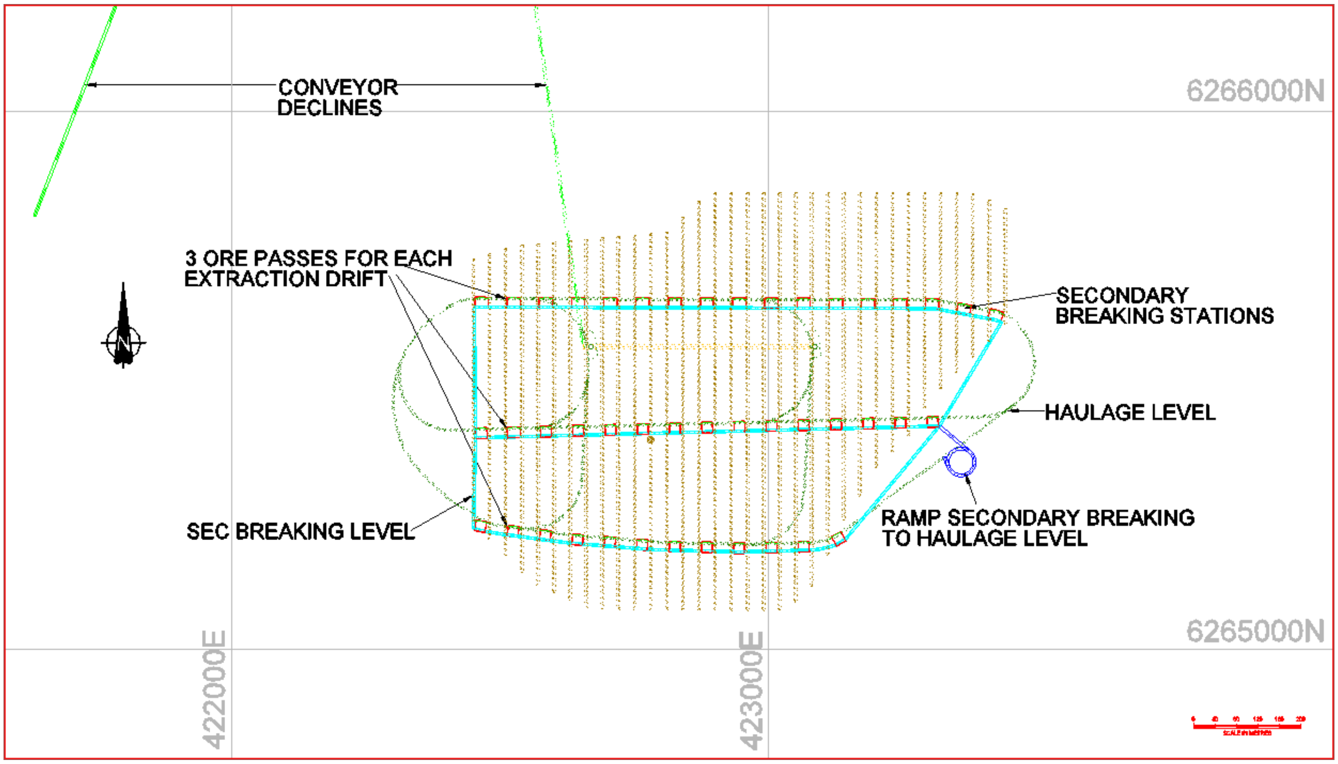


Figure 35: Plan view of the secondary breaking level, haulage level and extraction level designed for the Mitchell underground.

7.6 Haulage Level

The haulage level is designed to collect the broken material from ore passes and haul it to one of two gyratory crushers. It is designed for track haulage in three loops and it can be incrementally excavated and constructed as mining depletes one area and moves to the next. The haulage level is located 76 m (floor to floor) below the extraction level. A typical cross-section of the drift is 5 m wide by 5 m high to accommodate the loaded cars under the chutes. The haulage level has a total length of 5.5 km and will have place for approximately 90 chutes, but will only require approximately 30 at full production. It is envisioned that there will be recycling of the chutes as the cave front moves. There are two scroll-type dumps strategically positioned under the cave that will directly feed the gyratory crushers. A plan view of the haulage level is shown in Figure 36.

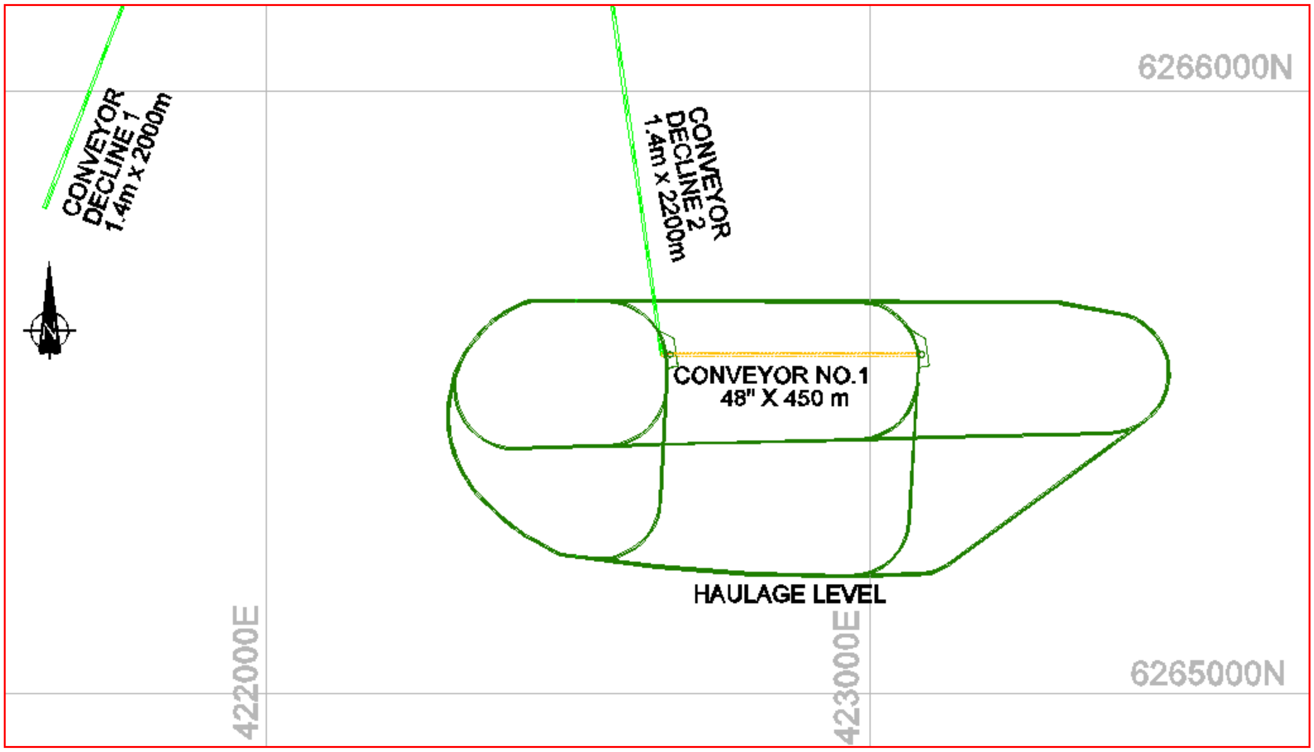


Figure 36: Plan view of the conveying and the haulage levels.

7.7 Conveyor Drift

The conveying level starts beneath the cave and finishes on surface near to the portal in the OPC area. It is designed to accommodate both production and development material (Figure 36). There are two conveyors that are fed by two 107 x 165 cm (42" X 65") gyratory crushers on the haulage level. The first conveyor is a 1.2 m wide, 450 m long belt which collects the broken material from under the haulage level and feeds a 1.4 m wide main conveyor that hauls the broken material, in two 2.1 km legs, to surface. On the surface there is a small trunk conveyor to bring the material to the OPC.

7.8 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 18.

Table 18: 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 2011 field program. An SRF value of 2 was assumed (appropriate for high stress rock conditions) and a J_w of 1 (moist, low flow).

Several figures depicting the location of the access ramp relative to the MTF (and showing the available drillhole data) are contained in Appendix F. The majority of the ramp and other mine infrastructure at Mitchell is located below the MTF. The average Q-value for rock below the MTF was estimated to be approximately 10, indicating fair to good rock conditions.

The upper portion of the proposed access ramp is located above the MTF. The rock quality is anticipated to be slightly lower above the MTF, as discussed in Section 5.1 and in BGC's pre-feasibility report for the open pit (BGC 2010). A range of Q-values for rock above the MTF was estimated using the following relationship developed by Bieniawski (1989):

$$RMR = 9\ ln(Q) + 44$$

Average RMR values for rock above the MTF range from approximately 50 to 60, which correspond to a range in Q-values of approximately 2 to 6. The rock conditions are classified as poor to fair, as per Table 18.

Figure 37 shows the approximate Q-values for rock above and below the MTF plotted on the empirical ground support chart for the different excavations that may require support.

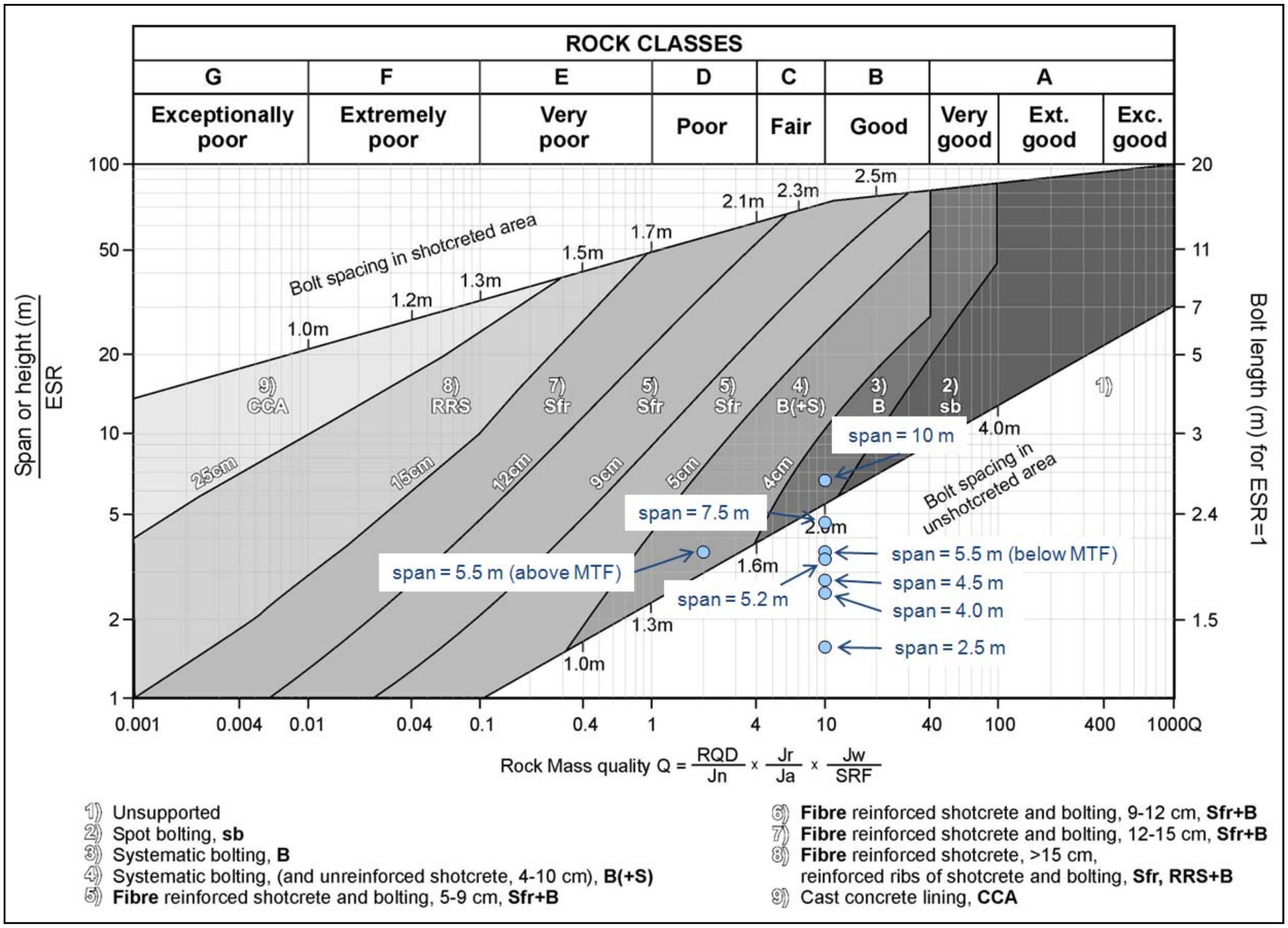


Figure 37: Empirical ground support design chart (Grimstad and Barton 1993).

The types of support recommended for the Mitchell mine infrastructure are summarized in Table 19. The excavations listed in the table below are shown in Figure 28.

Table 19: Ground Support Recommended for Mitchell Mine Infrastructure

Description	Span (m)	Height (m)	Type of Support
Access ramp (above the MTF)	5.0	5.0	2.4 m bolts on a 1.2 m pattern, mesh. 50% of the ramp above the MTF will require 50 mm of mesh reinforced shotcrete.
Access ramp (below the MTF)			2.4 m bolts on a 1.2 m pattern, mesh.
Drive (secondary breakage level, perimeter drift)	5.0	5.0	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	5.0	5.0	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.
Undercut and preconditioning 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 rockbreaker rooms	10.0	10.0	3.0 m bolts on a 1.2 m pattern. 50 mm of mesh reinforced shotcrete.
Exhaust raise	5.2	-	
Intake raise	5.2	-	
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.

7.9 Material Movement

It is estimated that the Mitchell deposit will be able to generate 55,000 tonnes of production material per day and 600 to 1,800 tonnes per day ore or waste from development (depending on the stage of development). The production material will be hauled from the drawpoint to one of three ore passes in the same extraction drift. The ore pass will be 4 m in diameter and equipped with a 1 m by 1 m grizzly. The ore passes from neighbouring extraction drifts will feed a stationary rockbreaker on the secondary breaking level, which will reduce the size of the material further by passing it through a 0.5 m by 0.5 m grizzly and feeding a 4 m diameter ore pass. This ore pass is equipped with a chute that will feed a train on the haulage level. The train will haul the material to one of two gyratory crushers, where it will be crushed and conveyed to the surface. Figure 38 shows a schematic of the material movement design.

Broken material produced through development will follow a path similar to that of the material produced from the drawpoints. It will be trucked from the active development area to one of the ore passes and will be mixed with the production material. The grade of the development material at Mitchell is such that the majority is above NSR cut-off. The remaining quantity of waste (below NSR cut-off) is small enough relative to the production of the rest of the KSM complex that it should have a minimal impact on the mill feed grade. Separating the development waste from the ore stream is not practical in the proposed mine. Process flow diagrams of the material movement system can be found in Appendix G.

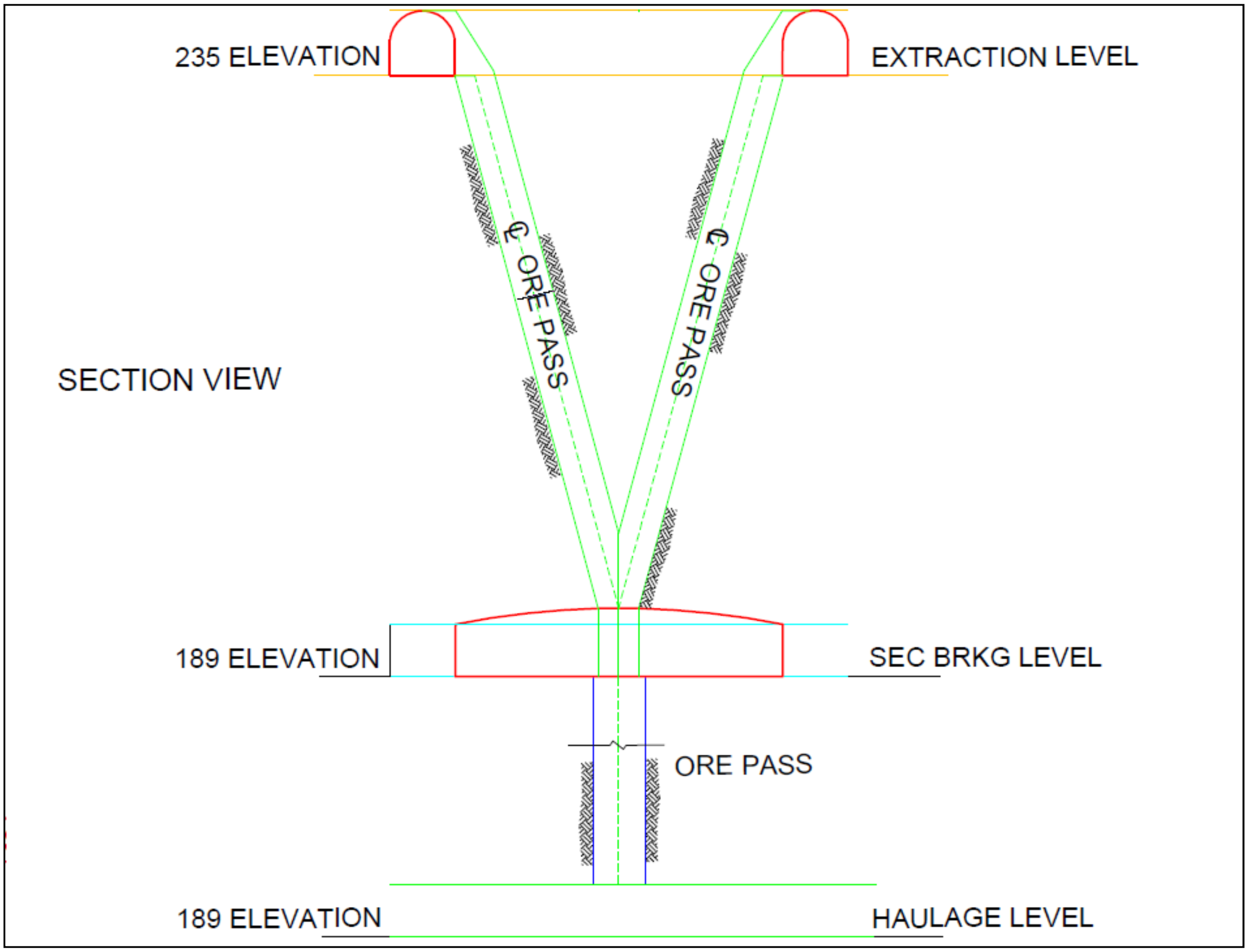


Figure 38: Schematic of the material movement system designed for Mitchell including the 4 m diameter ore passes that feed a secondary breaking level and haulage level.

7.10 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, secondary rockbreakers, and the train). The development equipment includes LHDs and trucks as well as 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, which 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.

A variety of ground support equipment will be required to install the ground support. Bolters are required to bolt and screen the back and walls of the development headings, and a concrete mixer and shotcrete sprayer are required to supply concrete/shotcrete where needed. The final drawpoint support includes the application of shotcrete, and concrete will be placed on the floor of the extraction drifts and drawpoints.

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 rockbreakers and block holers have been included in the design. The secondary rockbreakers 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 set up, drill and load remotely, keeping the operator in a safe location.

At peak operation, two 75T locomotives pulling 27, 15 m^3 cars each will be required on the haulage level. These locomotives are larger than those typically found in Canadian mines, but are successfully employed at the Kiruna and El Teniente mines. This train system was chosen because it can easily be upgraded by adding additional cars as mine production ramps up, it can haul a large amount of material at low cost, and it can be automated.

Both large and small personnel carriers are included in the design. The mine is located 23 km from the proposed camp location. 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 the drawpoints.

At peak operation, Mitchell will require a fleet of approximately 60 pieces of mobile equipment (Table 20). 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 Section 8.0.

Table 20: Peak Mobile Equipment Requirements for Mitchell Mine (Year 14)

Fleet	Equipment	Number
Production	Production drill	3
	Production LHD	10
	Production locomotive	3
	Raisebore machine	1
Development	Face drill	3
	Bolter	5
	LHD	3
	Truck	2
Service	AnFo loader	2
	Scissor lift	3
	Boom truck	2
	Block holer	2
	Mobile rockbreaker	4
	Shotcrete sprayer	2
	Concrete mixer	2
	Grader	2
	Small personnel carrier	7
	Large personnel carrier	4
TOTAL		**60**

7.11 Mine Workforce

The mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life. Table 21 shows a list of the positions required at Mitchell and the peak labour quantity separated into five categories: management, technical, maintenance, development, and production. It is noted that peak labour requirements occur prior to peak production.

Table 21: Peak Labour Quantities by Job Title (Year 7)

Job Title	Peak Labour Quantity
Management	
Underground Superintendent	1
Technical Superintendent	1
Maintenance Superintendent	1
Clerical/Admin	8
Technical	
Senior Engineer	10
Engineer	18
Technologist	20
Trainers	8

Job Title	Peak Labour Quantity
Geologists	8
Safety Technician	8
Maintenance	
Planner	4
Shop Foreman	4
Stores Person	8
Electrician (In-house)	32
Mechanics (In-house)	28
Development Crew	
Shift Captain	4
Shift Boss	4
Jumbo Operator	12
Bolter Operator	12
Loader Operator	12
Truck Operator	16
Crane Truck	8
Shotcrete Sprayer Operator	8
Agicar Operator	8
Grader	8
Construction	32
Labourer/Trainee	44
Production Crew	
Shift Captain	12
Shift Boss	12
Loader Operator	16
Train Operator	8
Secondary Breaker Operator	8
Block Holer Operator	8
Grader Operator	8
Crusher/Conveyor Operator	16
Production Driller	24
Anfo Loader	8
Construction	32
Labourer/Trainee	10
Total	**489**

The labour estimate was based primarily on the requirements to operate the mobile equipment. One operator per shift was assumed for each piece of mobile equipment. Mechanics and electricians were estimated using a common 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 trainees/labourers 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.

7.12 Contingencies

Table 22 contains a list of the different excavations, their dimensions and total estimated lengths over the life of the mine. A contingency length has been included to allow for openings that were not specifically included in the design such as sumps and electrical substations, and for the inefficiencies that result from overbreak as a result of poorly blasted rounds or poor surveying practices. Also included in the table are estimates of expected rehabilitation requirements.

Table 22: Summary of Mitchell Drift Dimensions

Drift Type	Width (m)	Height (m)	Length (m)
Fresh air drifts	5.0	5.0	4,900
Excavation level	5.0	5.0	20,400
Drawpoints	4.5	3.5	43,000
Perimeter drift	5.0	5.0	3,100
Haulage level	5.0	5.0	5,500
Secondary breaking level	4.0	4.0	3,600
Secondary breaking chambers	15	10	800
Undercut level	4.0	4.0	20,700
Preconditioning level	4.0	4.0	11,700
Return air drift	7.5	7.5	1,700
Internal ramps	5.0	5.0	2,700
Shops	5.5	7.5	1,400
Mine access	5.0	5.0	6,300
Conveyor level	5.5	4.4	4,600
Internal ventilation raises	2.0 – 4.0	---	4,200
Main ventilation raises	5.2	---	2,600
Ore passes and crusher chambers (crusher chamber dimension)	4 (10)		5,503
		Rehabilitation (10%)	13,000
		Contingency (5%)	6,500
		Total vertical	12,300
		Total lateral	130,400

8.0 MINE SERVICES

The mine services include ventilation, dewatering, power, and compressed air design. These topics are discussed in the following section.

8.1 Ventilation

The ventilation design was developed using the standard edition of Ventsim Visual, and according to best practices established in Malcolm J. McPherson's textbook, *Subsurface Ventilation & Environmental Engineering* (McPherson 1993). The airflow requirements for the majority of the mine were determined based on an industry- accepted factor of 0.063 m^3/s per engine kilowatt hour.

8.1.1 Design Parameters

The required airflow is 860 m^3/s based upon the diesel equipment utilized, air velocity considerations, and a contingency of 20% per level. The ventilation system provides 0.015 m^3/s air per tonne mined, which is slightly more than the 0.013 m^3/s per tonne benchmark for a well-ventilated block cave operations (De Souza 2008). The total airflow requirement of 860 m^3/s is sufficient to dilute all noxious gases, dust, and particulate matter produced by the mining equipment and activities in each mining area.

To achieve this airflow, two 2240 kW (3000 HP) and one 670 kW (900 HP) surface fans with variable frequency drives are required. These fans require approximately 5,160 kWh to operate. The maximum estimated total fan pressure is 5,088 Pa and the total mine resistance is 0.00615 Ns^2/m^8. The estimated network efficiency is approximately 85%.

The majority of air will be delivered into the mine through two 6.0 m diameter Fresh Air Raises (FARs); each FAR will have a fan providing 370 m^3/s and a mine heater. The other 120 m^3/s will be delivered by a fan directing air down the main ramp. The ventilation system is designed to operate as a positive pressure 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 the mine air in the winter months is included in the design and cost estimates. 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, approximately 38 million BTU per hour (MMBTUH) will be required to heat 860 m^3/s of air from a low of -6°C (average January low) to 3°C. In developing this estimate, no consideration was given to the heat produced during auto compression as air descends down the intake raises, heat transferred from the strata, or heat generated by mining equipment.

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. A minimum air velocity of 1 m/s was used in areas of comminution and haulage. Additional information about the ventilation calculations can be found in Appendix H. The quantity of air required on the various levels ranged from a low of 62 m^3/s on the PC level to a high of 389 m^3/s on the extraction level.

A pie chart indicating the total breakdown of air quantities, including air leakage to surface, is presented in Figure 39. Air leakage to surface accounted for approximately 46 m^3/s, of which 33 m^3/s was lost through the caved material. This is approximately 4% of the total airflow, which is close to the 5% leakage reported at the Henderson block cave operation in Colorado (Nelson 2011, pers. comm.).

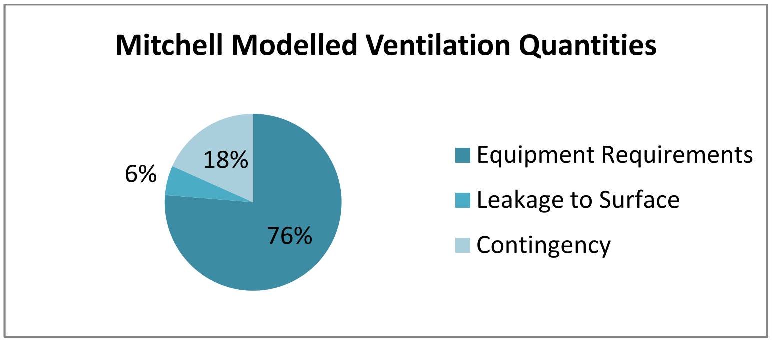


Figure 39: Mitchell ventilation breakdown based upon 860 m3/s of total airflow.

Friction factors were assumed to be typical for hard rock mining applications and are presented in Table 23. Airway shock losses were as assigned automatically by Ventsim. These values were reviewed and set manually where deemed necessary. The calculated air velocities and design criteria are presented in Table 24.

Table 23: Mitchell 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
Haulage drifts	0.0208	Due to undulations in back from ore passes

Table 24: Mitchell Ventilation Model – Air Velocities

Area	Design Criteria		Model	
	Min.	Max.	Min.	Max.
Working faces	0.5	4.0	0.6	1.0
Conveyor drifts	1.0	5.0	1.0	4.1
Main haulage drifts	1.0	6.0	1.0	3.3
Main return drifts	1.0	15.0	1.0	12.3
Ventilation raises	2.5	20.0	3.5	13.3

The ventilation of development drifts during excavation will be done using axial mine fans of various sizes. Large 112 kW fans provide the required 26 m^3/s approximately of air to an active mining faces up to 800 m away. Rigid and flexible duct of 1.5 m diameter is proposed to cover the air requirements for one LHD and one truck.

8.1.2 Airflow Design

The general airflow of the proposed ventilation system model is outlined in Figure 40.

The majority of fresh air (740 m^3/s) is forced into the mine though the two Fresh Air Raises (FAR). The FAR transports the air down approximately 650 m to two drifts that lead to the perimeter ventilation drift which surrounds the extraction level. The Fresh Air Drifts (FAD) are located on the 235 Level. The FAD feed fresh air to each mine level though drifts, ramps, and ventilation raises.

An additional 120 m^3/s of fresh air is forced down the main ramp. This air is mainly used to ventilate the access and conveyor ramps and the crushing level. It exhausts out of the Return Air Drift (RAD) on 129 Level and the conveyor ramp.

The airflow in the access and conveyor ramps is separate from the airflow in the production area for safety reasons.

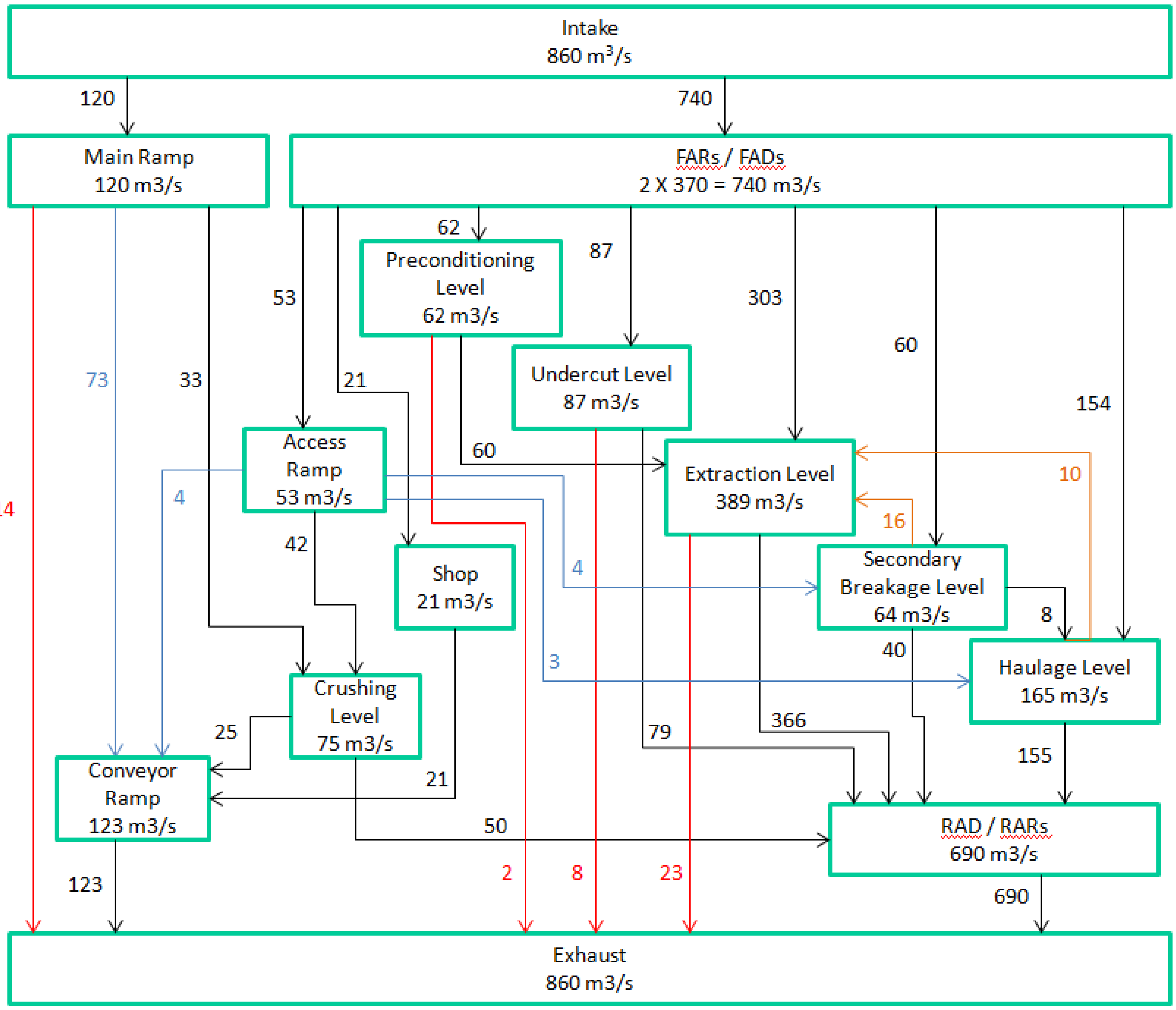


Figure 40: Mitchell ventilation flowchart; red indicates leakage to surface, blue indicates leakage across ventilation doors, and orange indicates leakage through ore passes.

8.2 Dewatering

The mine water handling system needs to be designed to handle the water that originates from the groundwater and surface inflows, and water that is introduced to the mine for operational purposes.

It was previously proposed to mine the Mitchell 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. This also provides an approximate estimate of the groundwater flows that will potentially report to the underground workings. The maximum estimated groundwater inflow is 13,200 m^3/d.

The surface water flows generated by precipitation and snow melt within the catchment area that if interception and diversion initiatives are not implemented will flow into the crater and down into the underground workings, will be significantly higher than this. At the time of completing this pre-feasibility assessment, estimates by others of these surface inflows into the crater at Mitchell were not available. The surface flows that are not intercepted and diverted will report to the drawpoints and an appropriate underground water management system will be developed to handle the flows and convey the water to surface. The design of this water management system will be completed at a later date once estimates of the surface water inflows are available.

8.3 Mine Water

It is estimated that the mine will require 30 m^3/hr of process water for the bolters and the development and production drills. In addition, 114 m^3/hr is estimated to be required by the conveyor fire suppression systems. This water will be supplied through the main access decline by gravity. The water will be delivered to the working face through 0.2 m steel pipe with 10 mm wall thickness.

8.4 Mine Power

The mine power design was completed by Neil Brazier of WN Brazier Associates Inc. It is estimated that the Mitchell mine will require approximately 17,400 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans as shown in Table 25. The underground communications will be provided through a combination of Leaky-Feeder and Personal Electronic Device (PED) emergency warning system. The complete report can be found in Appendix I.

Table 25: A Summary of the Major Contributors to the Peak Electrical Load at Mitchell

	Capacity (kW)	Quantity	Total Capacity (kW)	Efficiency	Demand Factor	Utilization (%)	Running Load (kW)
Jumbo	150	4	600	93%	0.8	75%	335
Bolter	56	5	280	93%	0.8	75%	156
Drills	56	8	448	93%	0.8	75%	250
Train + Trolley	1000	2	2000	100%	0.85	75%	1275
Raisebore	242	1	242	91%	0.75	75%	124
Rockbreaker	37	20	740	92%	0.8	65%	354
Shotcrete	56	2	112	91%	0.7	50%	36
Pumps	56	6	336	92%	0.75	80%	185
Surface fans	5150	1	5150	93%	0.95	100%	4550
U/G fans	56	15	840	92%	0.9	75%	522
Air compressors	130	2	260	100%	0.9	50%	117
Heating	200	1	200	100%	1	50%	100
Surface miscellaneous	60	1	60	100%	0.4	100%	24
Conveyor 1	323	1	323	96%	78%	90%	218
Conveyor 2	5172	1	5172	95%	80%	90%	3538
Conveyor 3	4962	1	4962	95%	90%	90%	3818
Crusher	556	2	1112	95%	85%	75%	673
Lighting and small power	250	6	1500	100%	1	50%	750
Heat tracing	500	1	500	100%	1	35%	175
Refuge stations	30	7	210	100%	0.6	100%	126
U/G Shop	150	1	150	100%	0.5	100%	75
Misc. monorails	3.7	6	22.2	91%	0.85	5%	1
Misc. sumps	18.7	6	112.2	91%	0.85	15%	13
Total							**17,415**

8.5 Compressed Air

It is estimated that the Mitchell mine will require 360 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 Mitchell 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). The underground workings 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 and at the shop. 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 primary 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. A breakdown of the yearly advance per heading can be found in Appendix K.

The underground mine will be operating 24 hours a day, 7 days a week, 365 days a year. The start of Phase 1 is scheduled according to the KSM site production plan. Phase 1 development rates of one round per day per heading were assumed. Once the extraction level is reached and a large amount of headings are available, the development rate is increased to eight 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 round is 5 m long. The mine development schedule is shown in Figure 41. The schedule indicates that the first set of drawpoints will be ready in Year 6; therefore, production cannot start until Year 7.

Rehabilitation of the lateral and vertical development was also estimated and is shown Figure 41. The amount of rehabilitation required is estimated to be equivalent to approximately 10% of the lateral development. The rehabilitation has been scheduled for after the start of steady state production when the majority of the degradation of the drifts will occur as a result of secondary blasting and changes in the stress field. The time scale in Figure 41 is shown in "Mitchell project years" and is not related to the overall site schedule.

It is estimated that Phase 1 development will produce an average of 275,000 tonnes per year (for 8 years) and this rock will be hauled to a waste dump in either Mitchell or Teigan valley. Waste generated after Year 8 will be sent to the mill as part of the ore stream.

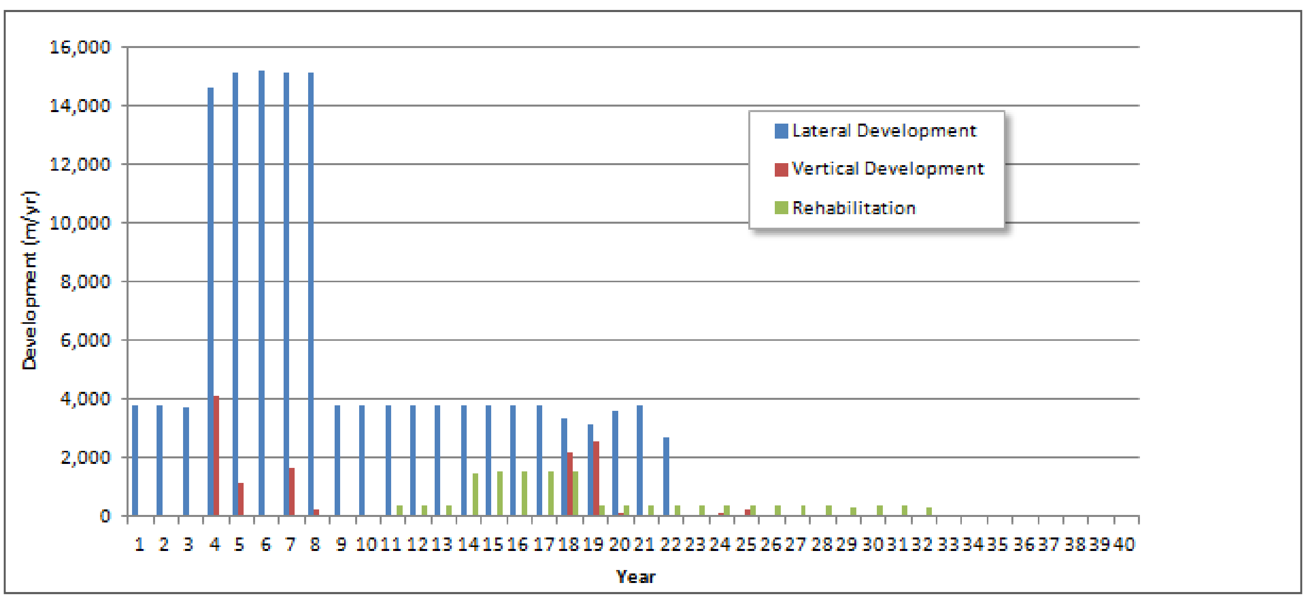


Figure 41: Chart showing the advance of lateral and vertical development and the quantity of rehabilitation estimated to be required.

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 42 shows the development and rehabilitation per year and the corresponding development labour. The development workforce includes all site personnel up to the start of production in Year 7. After Year 7, 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.

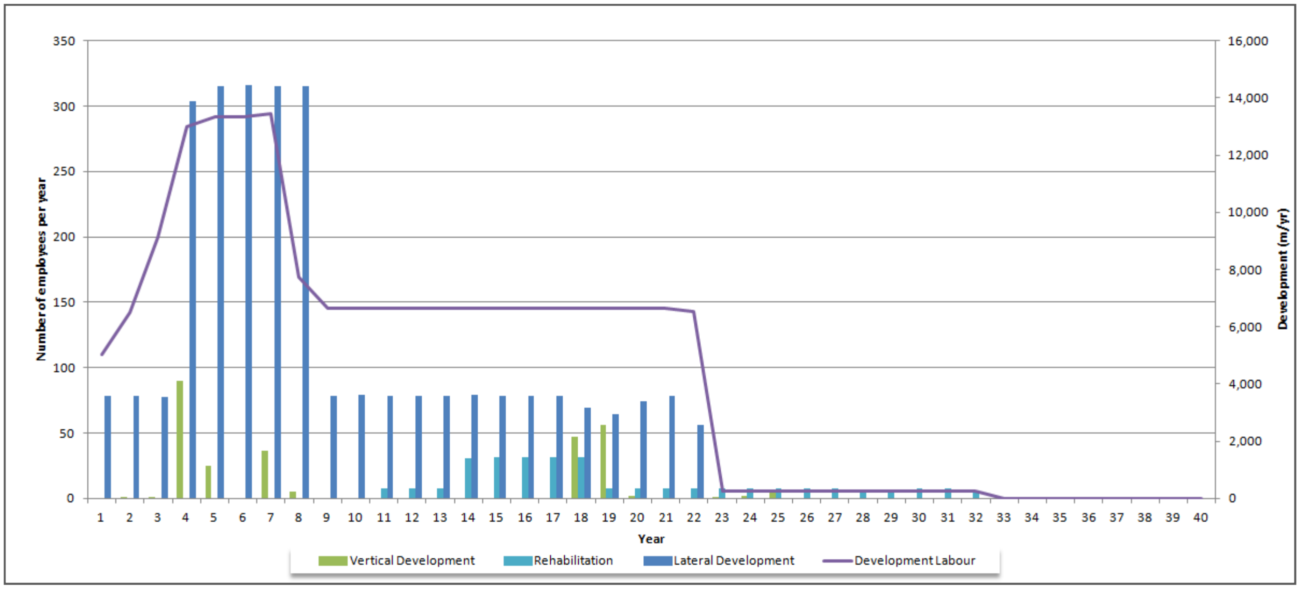


Figure 42: Chart showing the yearly development labour and the amount of vertical and horizontal development, and rehabilitation required per year.

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 at the following internet site 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. The input includes 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. 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 26. The mining and development costs were developed from first principles and are discussed in Section 11.3. The discount rate, milling, and General and Administration costs were obtained from the PFS (Seabridge 2011).

Table 26: Key PCBC Input Parameters

Item	Value	Unit
Mining cost	6	dollars per tonne
Processing and G&A costs	7.09	dollars per tonne
Development cost	1,075	\$ per m^2
Discount rate	5	%
Drawpoint construction rate	120	drawpoints per year
Yearly production rate	20,000,000	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 controlled 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 Mitchell. In particular, the initial and maximum drawpoint production rates were reduced to simulate a production environment with coarse fragmentation. The draw rate and PRC are discussed in more detail later.

These input parameters were used in PCBC to evaluate the potential maximum yearly mine production rate, and based on this, a slightly conservative production rate of 20 million tonnes per year was selected.

The maximum height of draw governs the tallest column that PCBC assumes can be drawn if the drawpoint material is still profitable. This parameter relates to the wear that develops at a drawpoint and the associated drawpoint stability. At a certain height, the drawpoint becomes so damaged as a result of stress and the quantity of material that has passed 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 Mitchell. 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 modelled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution.

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 43 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 Mitchell, particularly during the early stages of mining, a draw rate of 200 mm/day was chosen as a maximum cap in the PCBC analysis. As will be discussed in Section 10.2, an average draw rate of 108 mm/day is required to reach production targets at Mitchell.

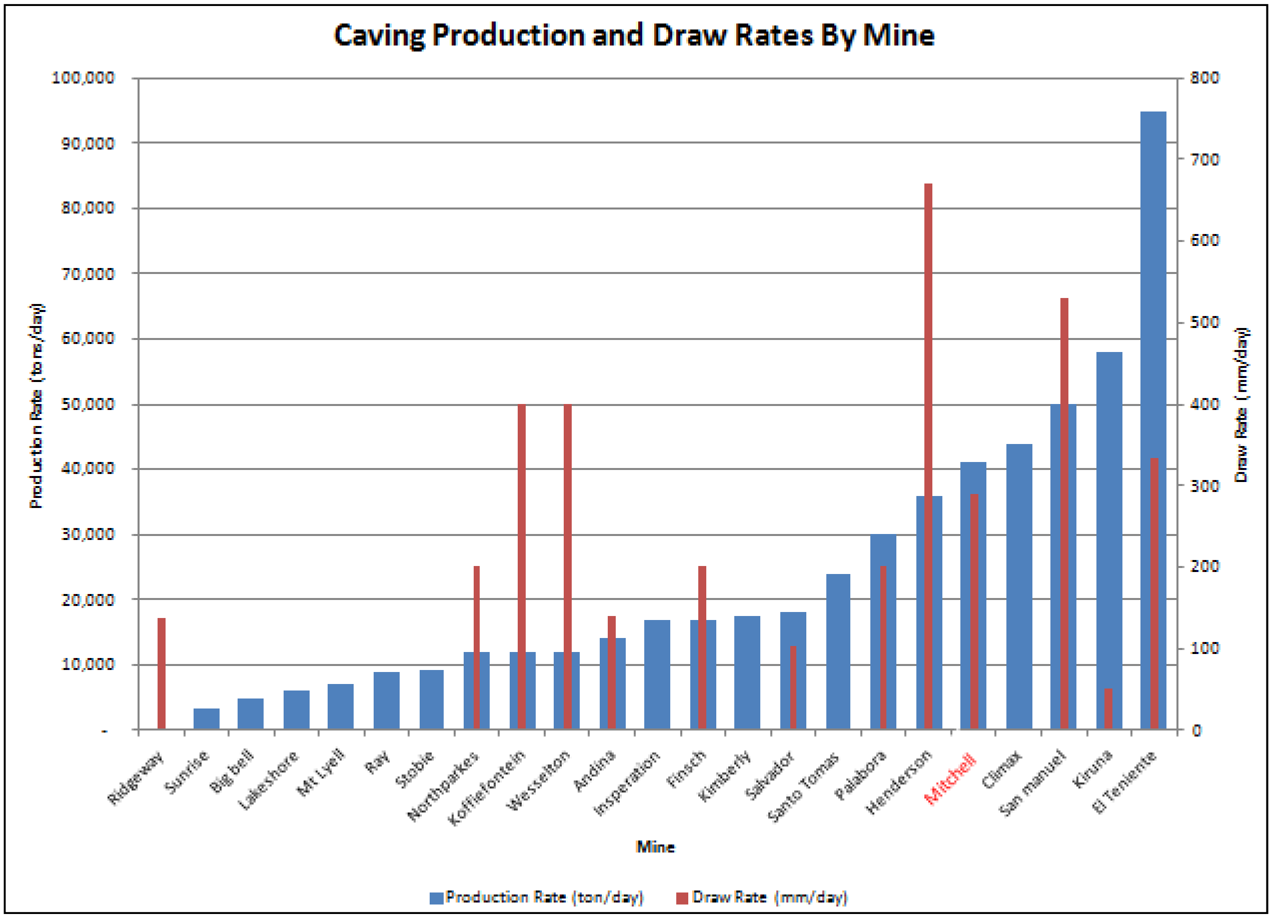


Figure 43: Draw and production rates of a selection of block and panel cave mines (after Woo, Eberhart and van As, 2009).

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 44. It shows that, initially, it is assumed at Mitchell that a drawpoint can produce 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 that is required decreases after approximately the first 100 m of a column is drawn (Ngidi 2007).

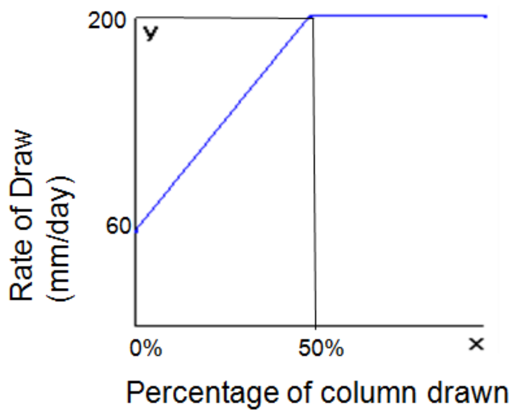


Figure 44: Production Rate Curve used to describe the rate of change of the draw rate of one drawpoint.

10.1.3 Cave Start Location

Two cave start locations were evaluated in PCBC. The first is directly underneath the pit bottom. It was considered because block cave operators have found that the rock mass "behaves better" once breakthrough to the pit/surface occurs. However, the column heights directly under the Mitchell pit bottom are short (relative to the columns on the edge of the pit) and the PCBC analysis showed that the mine would have trouble reaching maximum production targets because drawpoints were being closed faster than others could be brought online.

The second start location was considered based on the columns with the highest value. Starting the cave at this location is similar to mining the high grade material first, which is financially beneficial for a low-grade deposit. The columns around the second start location are also taller than under the Mitchell pit, and the PCBC analysis showed that the mine would be more likely to reach production targets. The analysis presented in this report used the second start location.

10.2 Results of PCBC Analysis

The production schedule determined from PCBC for Mitchell is shown in Figure 45. Mitchell is estimated to have a production ramp-up period of 6 years, steady state production at 20 million tonnes per year for 16 years and ramp down production for another 8 years. A breakdown of the production and average grade schedule can be found in Appendix K. The period prior to production in Year 7 is considered pre-production. The total production and average grade of the life of the mine is shown in Table 27.

Table 27: Total Production and Average Grades for the Proposed Mitchell Underground Mine

Total Production	AU (gpt)	CU (%)	MO(ppm)	AG (g/t)
437,966,000	0.53	0.16	34	3.48

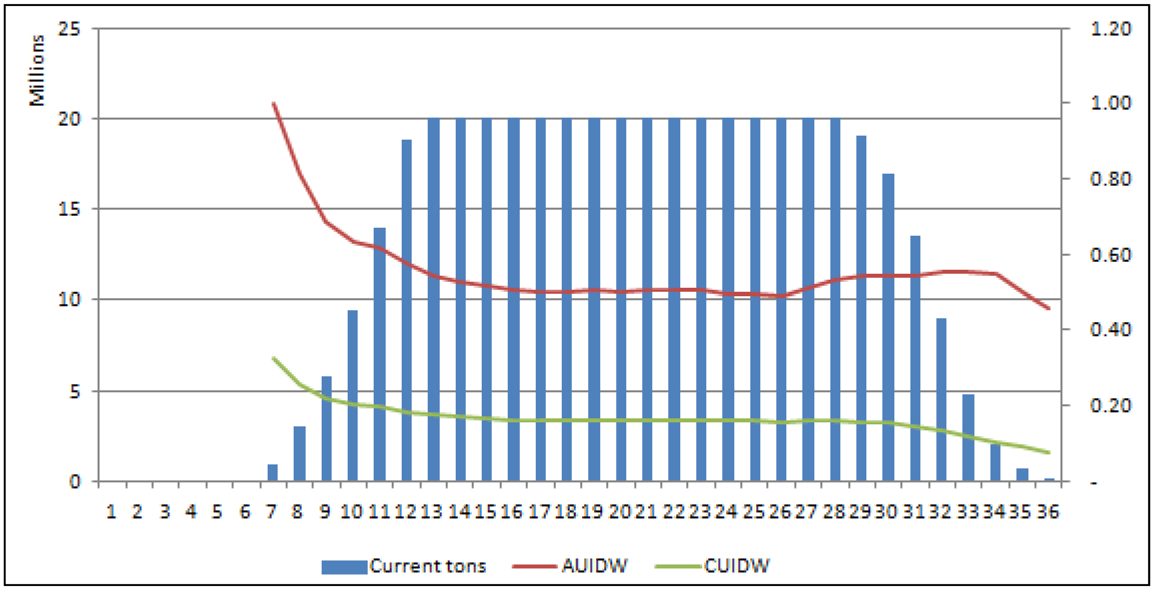


Figure 45: Yearly PCBC production schedule showing gold and copper grades.

Drawpoint construction, maintenance and utilization will be 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. As shown in Figure 46, the rate of increase in the number of active drawpoints mirrors the production ramp-up rate until a year after the production targets are met. After this point, the rate at which drawpoints become active is reduced as more drawpoints are closed than are constructed (as discussed in Section 10.1, a drawpoint is closed when the value of the material being mined does not exceed the NSR cut-off).

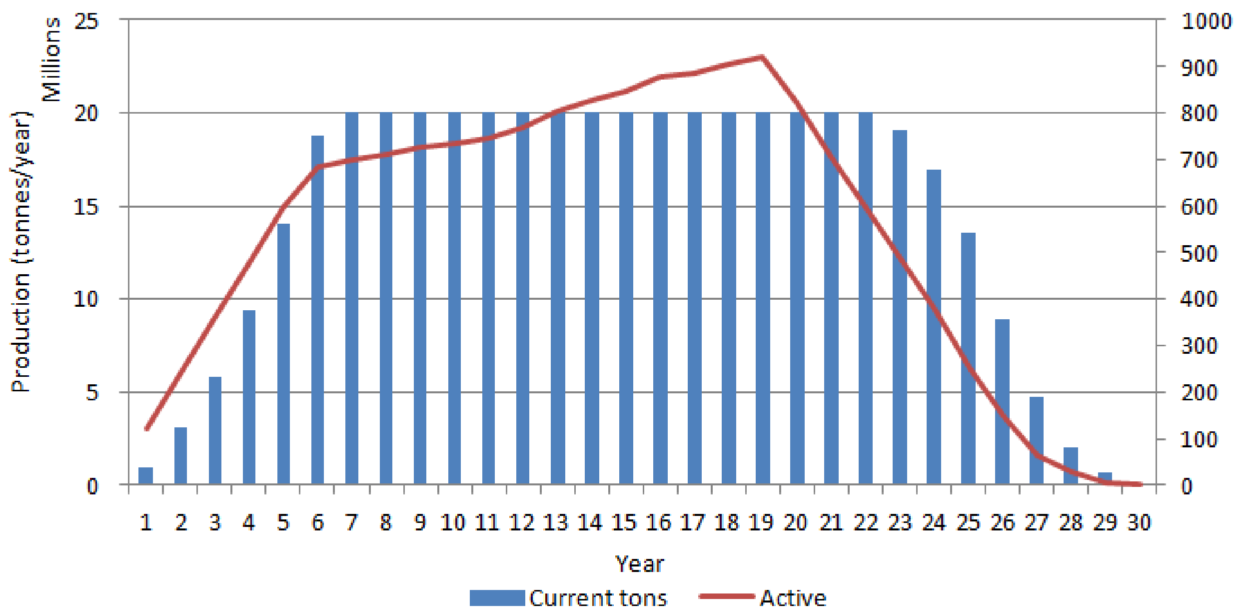


Figure 46: The number of active drawpoints and production per year.

The draw rate per drawpoint is another design parameter used to reduce the impact of the coarse fragmentation. Figure 47 shows that the maximum draw rate from the Mitchell deposit does not exceed 165 mm/day, which is lower than the accepted value of 200 mm/day. The average draw rate is 108 mm/day, which means that there are roughly twice as many drawpoints available as are required to meet production targets.

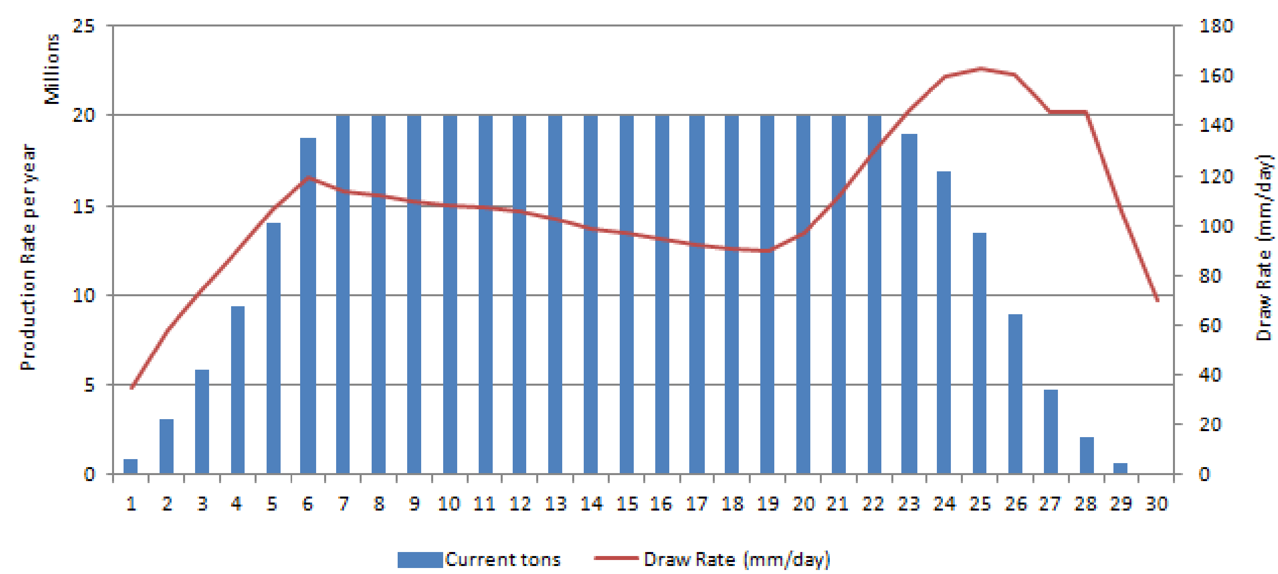


Figure 47: Average yearly draw rate (mm/day) and production (tonnes/year).

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 7. 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 48. The sharp increase in the initial workforce 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 UC and PC levels). The separation of the production and development workforce 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 to ramp down because certain positions are mainly required early in the mine life (e.g., secondary breaker operators, production drillers, and construction workers). More details concerning the workforce breakdown can be found in Appendix J.

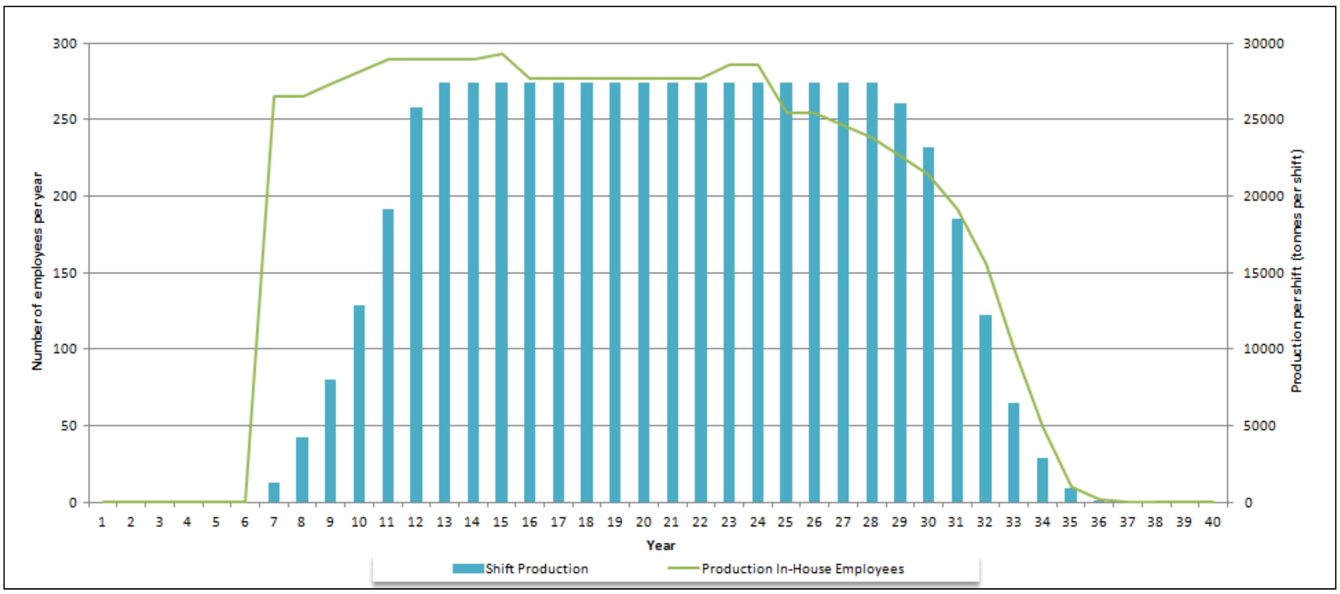


Figure 48: Yearly production workforce distribution.

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 therefore, it is discussed as a separate section.

11.1 Labour

The labour costs are based on estimated KSM project rates as provided by Wardrop (Wong 2011, 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 28 shows the relevant mark-ups that were used to account for burdens, bonus, and remote and underground premiums.

Table 29 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) working 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 constructed by contractors.

Table 28: Breakdown of the Various Labour Mark-Ups

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

Table 29: 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 distribution of the combined development and production workforce is shown in Figure 49. The workforce ramps up as development headings become available and production starts, and it reaches a maximum of 489 employees for Year 7. After this the initial development is substantially completed and the crews are reduced. The size of the workforce parallels the mine production between Years 8 and 22, after which the sustaining development is completed and workforce is reduced to a skeleton crew. After year 22, the total workforce is directly influenced by the mine production ramp down.

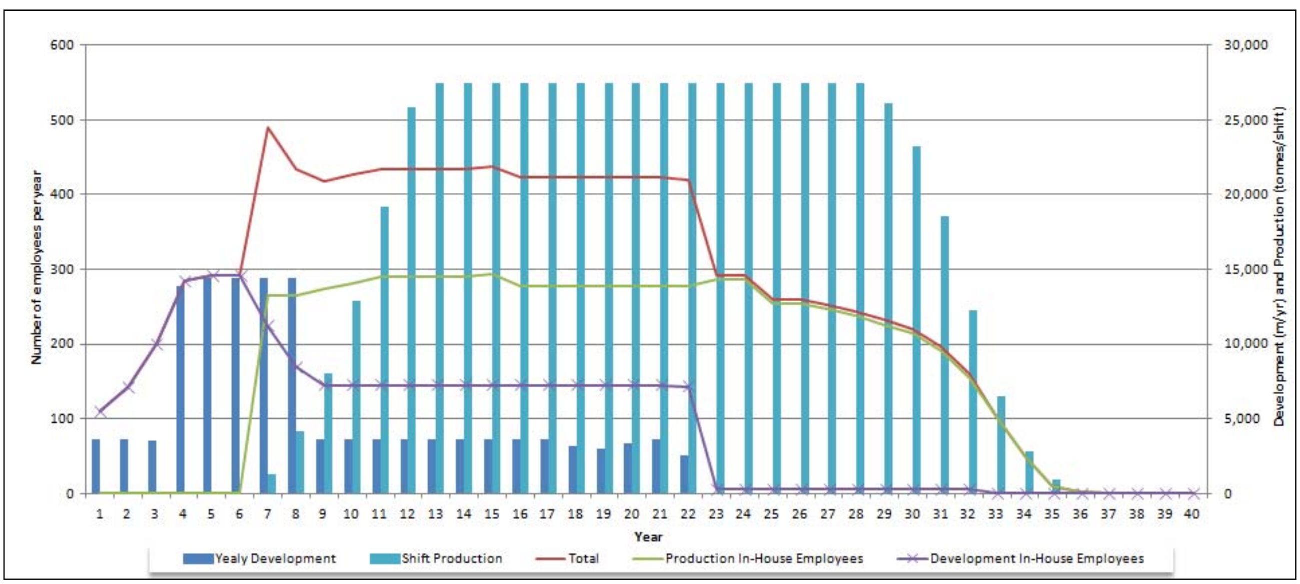


Figure 49: Yearly distribution of development, production and total workforces, and development advance and production rate per shift.

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.

The 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, the proposed Mitchell underground design has various sized openings with different purposes. Table 30 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 costs of ongoing activities once the initial development is created (i.e., the cost to blast the undercut after it has been excavated or the cost to drill, blast, and excavate a drawbell) are described in a separate section of this report. The development costs include standard ground support (bolt and mesh) in the back and walls, and concrete floors on the extraction drift and drawpoints. An example of the detailed cost calculation for a meter of development can be found in Appendix L.

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.

Table 30: Summary of the Unit Cost of the Various Development Sizes Proposed for the Mitchell Mine

Description	Unit of measure	Unit Cost (ex. Labour)	Unit Cost (incl. Labour)
5.0m x 5.0m Drive [(1)]	m	$2,000	$4,200
3.5m x 4.5m Drawpoint	m	$2,700	$5,000
5.5m x 5.5m Extraction drifts	m	$4,200	$6,500
4.0m x 4.0m Undercut and preconditioning	m	$1,200	$3,500
5.5m x 4.4m Conveyor drive	m	$2,600	$4,900
7.5m x 7.5m Return air drive	m	$3,400	$5,600
Rehabilitation	m	$1,200	$3,500
Internal vent raise	m	$5,500	$7,900
Main ventilation raise	m	$9,100	$11,400

Note: [1] This item refers to the majority of underground excavations, such as the perimeter drifts, the underground access, and the internal ramps.

11.2.2 Block Cave Infrastructure

The block cave infrastructure includes the cost of the ongoing activity in a drift once the excavation is completed 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 31. These costs do not include labour

Table 31: Summary of the Block Cave Infrastructure Capital Cost

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 32 shows the list and unit cost of the mobile equipment required for the Mitchell mine. A five year replacement schedule has been included in the life of mine capital cost estimates. It is estimated that over the 37 year underground mine life, a total of $48 million will be required for development equipment, $140 million for production equipment, and $45 million for support equipment.

Table 32: A List of the Mobile Equipment Required and Unit Costs

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
Production locomotive	$ 2,500,000
Production rail car	$ 130,000
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 33 shows the quantity and unit costs of the major stationary equipment required for the Mitchell mine. For cost estimating purposes the replacement/refit of the conveyors and crushers is done every ten years. It is estimated that the Mitchell underground will require $543M of stationary equipment over the life of the mine. Accessories include structural steel and concrete to support the equipment, ore pass chains, etc.

Table 33: Mitchell Stationary Equipment Costs

Equipment	Quantity	Unit Cost ($M)
Crusher (incl. installation)	2	$ 4.5
Crusher accessories	2	$ 9.0
Conveyor #1 (incl. installation)	1	$ 3.5
Conveyor #2 (incl. installation)	1	$ 18.0
Conveyor #3 (incl. installation)	1	$ 16.4
Conveyor accessories	1	$ 6.7
Dumps and chutes	44	$ 0.4
Stationary rockbreakers (incl. installation)	30	$ 0.25
Stationary rockbreakers accessories	90	$ 2.9
Underground fans	15	$ 0.021
Dewatering system	1	$ 0.26
Air compressor	2	$ 0.4

11.2.5 Surface Equipment

Table 34 shows a list of unit and installation costs of the surface equipment required for the Mitchell mine. The electrical distribution system estimate was developed by WN Brazier Associates (details can be found in Appendix I).

Table 34: Surface Infrastructure Cost

Equipment	Quantity	Unit Cost ($M)
Surface fans	4	$ 1.0
Ventilation bulkheads	1	$ 1.6
Electrical distribution system	1	$ 46.0
Heaters	1	$ 5.0
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 expense at the end of the mine life and it excludes any benefit that may be realized by selling stationary equipment (crushers, fans, conveyor belts), mobile equipment (LHDs, trucks or jumbos) or services (electrical wiring). The proposed closure activities and relevant cost items incorporate the following:

- 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 in the mine;
- 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; and
- Seal all openings to surface with cement plugs or barricades.

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 50 shows the life of mine capital cost for the Mitchell mine, which is estimated to be $3.1 billion. This includes approximately $800 million in pre-production capital expense over the first 6 years of the mine life and an average sustaining capital cost of $74 million over the remaining 31 years. The life of mine capital cost is shown in a table in Appendix L.

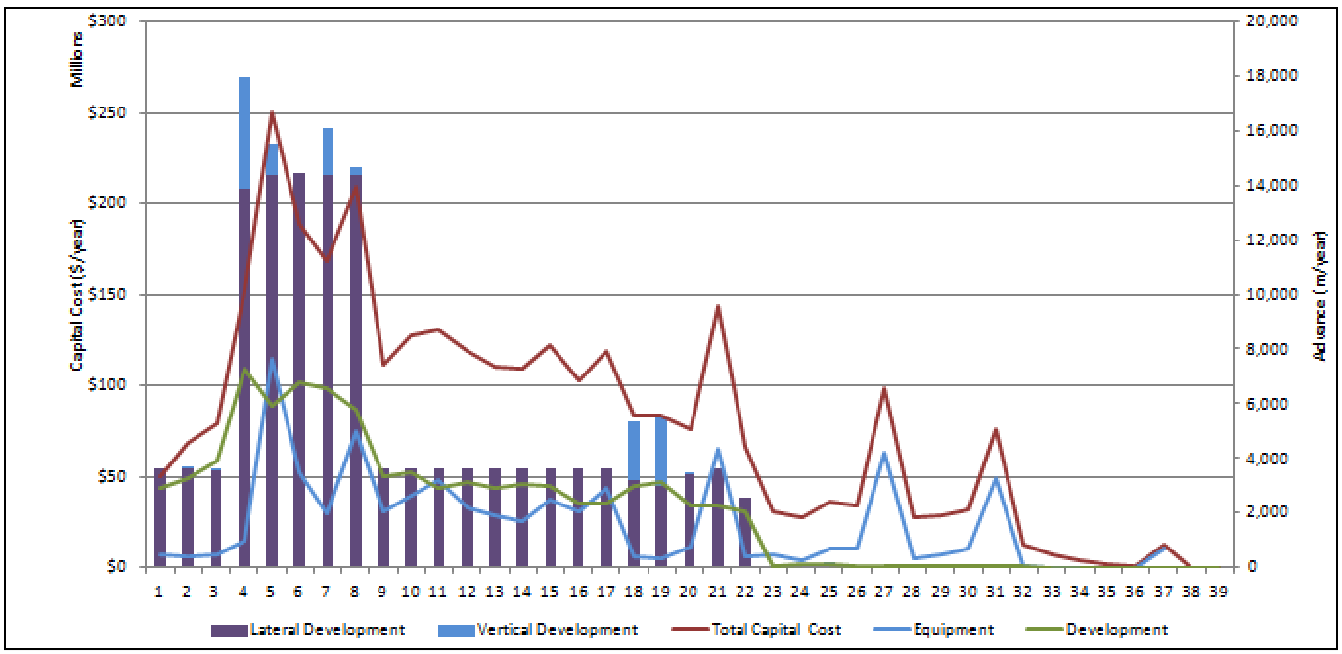


Figure 50: Yearly capital cost estimate and the quantity of lateral and vertical development.

11.3 Mine Operating Costs

A preliminary cost estimate of $6/tonne was used in the PCBC analyse to produce a tonne and grade production schedule for Mitchell. 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 differences 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 surface conveyor portal and the fixed costs to operate 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 approximately 52% of the total Mitchell underground mine OPEX). Included in the fixed costs are items that are not affected by the quantity of production, such as the ventilation fans, pumps, and the general mine expenses such as office supplies. Table 35 shows a breakdown of the average life of mine OPEX which is estimated at $5.00/tonne. Appendix M contains the detailed cost model and breakdown of the mine operating expenses.

Table 35: Summary of the Mine Operation Cost

Equipment	Mitchell OPEX ($/tonne)	Mitchell OPEX (%)
Production LHD	$0.56	11%
Production locomotive	$0.02	0%
Crusher	$0.27	5%
Conveyor 1 and 2	$0.27	5%
Conveyor 3	$0.26	5%
Block holer	$0.13	3%
Stationary rockbreaker	$0.29	6%
Mobile rockbreaker	$0.19	4%
Labour	$2.60	52%
Fixed costs	$0.36	7%
Rehabilitation	$0.04	1%
Total	**$ 5.00**	

Figure 51 shows the distribution of the OPEX over the life of the mine. The OPEX is higher in the first years due to the relatively high number of personnel on site producing at a comparatively low production rate. This is typical for an underground mine and even more applicable to block caving because of the significant amount of development required before production can commence and the long ramp-up period to achieve the planned production rate. The workforce component of the OPEX ranges from approximately 75% in the early years to approximately 35% in the later years.

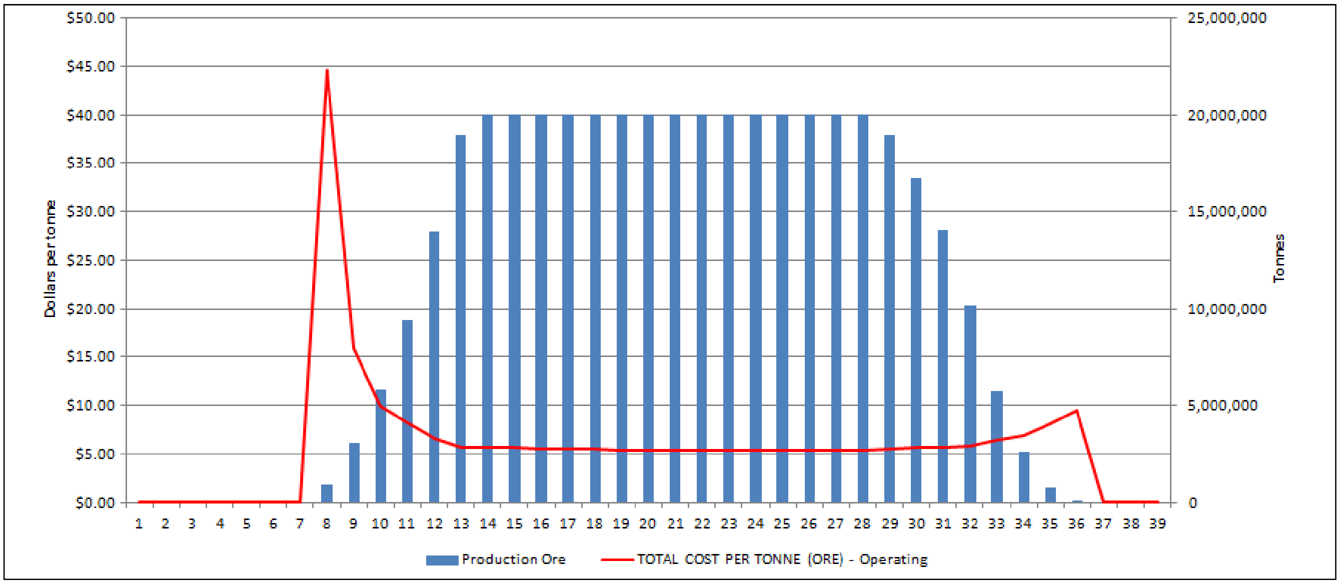


Figure 51: Chart showing the variability of the estimated OPEX over the life of the mine compared to the mine production.

The impact of varying the labour cost by +/- 25% is shown in Table 36. A 25% increase in the labour cost will result in a $0.35/tonne (7%) increase in the OPEX.

Table 36: Influence of Increasing or Decreasing the Labour Cost on the OPEX

Item	Labour at -25%	Labour at Base Case	Labour at +25%
OPEX	4.64	5.00	5.35

11.3.1 OPEX Sensitivity

The mine OPEX is a key parameter used in PCBC to determine the profitability of individual drawpoints. The influence of increasing or decreasing the OPEX by 25% on the available block cave resources was investigated with additional PCBC runs, and the results are presented in Table 37. The block cave resources are not overly sensitive to OPEX as a 25% change to OPEX only changes the block cave resources by less than 5%. Note this comparison was completed early in the study and the preliminary OPEX of $5.84/tonne was used.

Table 37: Influence of Mine OPEX on Block Cave Resources

OPEX ($/tonne)	Tons	Au (g/t)	Cu (%)	Mo (ppm)	Ag (g/t)
5.84	421,182,784	0.524	0.165	34.408	3.548
4.38 (-25%)	431,319,744	0.519	0.164	34.188	3.514
7.30 (+25%)	409,480,544	0.530	0.167	34.470	3.583

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% on the maintenance cost of the equipment. Overall project contingency is estimated to be 23%.

12.0 PROJECT OPPORTUNITIES AND RISKS

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

12.1 Project Opportunities

- A pre-feasibility study permits the use of Measure and Indicated resources only, which is approximately 421 million tonnes within the zone that is proposed to be mined. T here is also approximately 376 million tonnes of Inferred material inside this zone (grades and tonnages are shown in Table 38) that were not included in the study but would improve project economics. The results presented in Table 38 are from the Footprint Finder program and differ from the Block Cave Resources stated in Table 5.

Table 38: Summary of the Footprint Finder results.

Categories	Tonnage (Mtonnes)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Measured and Indicated	421	0.52	0.17	3.55	34.41
Measured, Indicated, and Inferred	797	0.55	0.17	3.70	41.26

- It may be possible to cave the Mitchell deposit as two individual mining zones, one vertically offset from the other, with each zone using a system of LHD's hauling directly to crushers (similar to the smaller Iron Cap mine design). The Iron Cap design is similar to the design used at Palabora and may be better suited to handling the expected coarse fragmentation. The current design of using ore passes, grizzlies and secondary rockbreakers could prove to be a bottleneck in the material movement system.

  The potential cost saving of this design concept should be investigated further. It is likely to incur larger crushing and conveying costs as more crushers and conveyors would be required. However there is likely to be less mine development (no need for the secondary breaking or haulage levels) and lower equipment costs (the stationary rockbreaker stations, chutes, and chain gates would be eliminated). The current design requires an estimated $330 million dollars of infrastructure and equipment to handle the coarse fragmentation and significant additional lead time is needed to develop the secondary breaker and haulage levels prior to the start of extraction.

- The current design is based on a conservative drawpoint layout of 15 m by 15 m. More detailed assessments of the fracture orientation and spacing information may indicate that this layout can be expanded to a 17 m by 17 m layout or some equivalent expanded layout. This will result in a significant economic benefit and should be investigated further.

- Approximately 80% of the air exiting the mine will do so via the RAD and RAR. The dimensions of the RAD are 7.5 m by 7.5 m which, due to the high velocity of the air, is a bottleneck in the system. A detailed trade-off study between the cost of the additional development and the fan capital and operating costs by year may indicate that it would be more cost-effective to widen the RAD to a larger size with smaller fans on surface.

12.2 Project Risks

- The ore pass and grizzly system may not achieve the planned production rate, particularly if the muck that is transported to the passes is coarser and requires more breakage than is currently estimated.
- The rock fracture information that has been collected at Mitchell indicates that the size of the rock blocks in-situ will be significantly greater than 2 m^3; in effect, very large. The mine design was developed with full recognition of this by using conservative estimates of factors such as the availability of drawpoints, the amount of secondary blasting required, requirements to rehabilitate damaged drawpoints, material handling delays, etc. In addition, pre-conditioning of the rock mass is also proposed using hydraulic fracturing. However, there is still a degree of uncertainty as to the possible impact of the expected coarse fragmentation at least until a mature height of individual column draw has been established. Further assessments of this uncertainty should be undertaken.
- 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 the other direction. If the exhaust air is not properly dispersed by the time it reaches the fresh air intakes, then a recirculation problem may occur.
- There are inherent risks associated with block caving. These include the following:
  - If there is not good a good understanding of the caving profile both of the uncaved back and the thickness of the caved rock, and/or there is not good production draw control, voids may develop above the drawpoints. Under some circumstances this can result in air blasts, and significant safety hazards and damage to the underground infrastructure. Good operating practices will be implemented to mitigate such concerns, along with good monitoring practices such as microseismic monitoring and borehole caving propagation monitoring.
  - Mudrushes at the drawpoints are a risk once the cave breaks through to surface. This is particularly important during the annual spring thaw, when melt-water from snow and ice that accumulates on the broken material at surface can migrate through to the underground drawpoints. The current drawpoint design has attempted to mitigate some of this risk by offsetting the drawpoints, and remote monitoring practices will be introduced on an as-needed basis until such risks are no longer present.
  - Excessive rehabilitation of the drifts may have a significant impact on the profitability of the mine. The nature of a block cave causes the stress field to change in magnitude and direction, which could result in damage to the existing drifts. In addition, it is anticipated that there will be a significant amount of secondary blasting of oversize 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 also because of the potential lost productivity. The current design attempts to mitigate these risks by assigning a rehabilitation cost to the OPEX and by having approximately twice as many drawpoints available than are needed to meet production targets.

13.0 CONCLUSIONS

The Mitchell 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 577,000 m^2 in size.

The mine design is based on a drawpoint layout 15 m by 15 m. Further detailed assessments of the fracture information and the estimated fragmentation may indicate that slightly expanded layout can be adopted. This aspect needs to be investigated further.

Even with the proposed pre-conditioning of the rock mass by hydraulic fracturing the fragmentation is expected to be relatively coarse. This has been accounted for in establishing the production rate of 55,000 tpd. As well, the cost of block holers and mobile and stationary secondary rockbreakers have been included in the cost estimate.

The Measured and Indicated resources are approximately 438 million tonnes with an average grade of 3.5 g/t Ag, 0.53 g/t Au, 0.16% Cu and 34 ppm Mo. The mining operation will require approximately 142 km of openings to be excavated.

Detailed production, development, and capital and operating cost schedules have been established. The costs were developed from first principles and vendor quotes, and are considered accurate to +/-25%.

The mining operating cost is estimated to be $5.00 per tonne and the total mining cost including a 23% contingency is estimated to be $12.09 per tonne.

The pre-production capital expense is approximately $800M and the yearly capital expense is $74M per year for the remaining 31 years. The total mine life of Mitchell is expected to be 37 years, including 6 years of pre-production development, 6 years of ramp-up, 16 years of steady state, and 8 years of ramp down production. The pre-production development period is 6 years.

14.0 CLOSURE

If you have any questions regarding the above, please contact the undersigned.

GOLDER ASSOCIATES LTD.

ORIGINAL SIGNED

Donald Tolfree, P.Eng.
Mining Engineer

ORIGINAL SIGNED

David Sprott, P.Eng.
Principal, Senior Mining Engineer

ORIGINAL SIGNED

Johnny Canosa, P.Eng.
Senior Mining Engineer

ORIGINAL SIGNED

Karen Moffitt, P.Eng.
Associate, Senior Geotechnical Engineer

ORIGINAL SIGNED

Ross Hammett, P.Eng.
Principal, Senior Civil/Mining Engineer

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15.0 REFERENCES

Barton, N., R. Lien and J. Lunde. 1974. *Engineering Classification of Rock Masses for Design of Tunnel Support.* Rock Mechanics, 6, p. 189-236.

BGC (Bruce Geotechnical Consultants). 2010. *KSM Project: Mitchell Zone - Open Pit Slope Design - FINAL.* Prepared for Seabridge Gold Inc. Submitted April 16, 2010.

Bieniawski, Z.T. 1976. *Rock Mass Classification in Rock Engineering*. In: Exploration for Rock Engineering, Proceeding of the Symposium (Edited by Bieniawski, Z.T.) 1, 97-10. Cape Town, Balkema.

Bieniawski, Z.T. 1989. *Engineering Rock Mass Classifications*. Wiley, New York.

Butcher, R.J. and I.G.T. Thin. 2007. *The Input of Choices for Predicting Fragmentation in Block Cave Projects.* 1st International Symposium on Block and Sub Level Caving Cave Mining. Cape Town, South Africa, p. 35-50.

De Souza, E.M. 2008. Mine Ventilation Course Notes. Queen's University.

Golder Associates Ltd. 2011a. *Block Cave Mine Study*. Report Number REP 0516_11. Submitted May 17, 2011.

Golder Associates Ltd. 2011b. *Block Panel Caving Conceptual Study for the KSM project*. Report Number REP 0414_11. Submitted 14 April 2011.

Golder Associates Ltd. 2012a. *2011 Geotechnical and Hydrogeological Field Investigations - Mitchell Project.* Prepared for Seabridge Gold Inc. Submitted February 15, 2012.

Golder Associates Ltd. 2012b. *Mitchell Deposit – Geotechnical Characterization.* Prepared for Seabridge Gold Inc. Submitted February 2012.

Grimstad, E. and N. Barton. 1993. *Updating of the Q-system for NMT*. Proceedings of the International Symposium on Sprayed Concrete. Norwegian Concrete Association, Oslo, p. 46-66.

ISRM (International Society of Rock Mechanics). 1981. *Rock Characterization, Testing and Monitoring.* In: Brown, E.T., ISRM Suggested Methods. Pergamon Press. Oxford, p. 211.

Laubscher, D.H. 1994. *Cave Mining – The State of the Art.* Journal of the South African Institute of Mining and Metallurgy, 94, p. 279-293.

Laubscher, D.H. 1999. *A Practical Manual on Block Caving.* Section 6 – Cavability. International Caving Study, Julius Kruttschnitt Mineral Research Centre, University of Queensland, Australia, p. 323.

McPherson, M.J. 1993. *Subsurface Ventilation and Environmental Engineering*. Available at: http://www.mvsengineering.com/index.cfm?fuseaction=menu&menu_id=5004. Accessed January 17, 2012.

Nelson, Brad. 2011. *Senior Engineer at Climax Molybdenum- Henderson Mine*. Site visit with Donald Tolfree in March, 2011

Seabridge. 2010. Kerr-Sulphurets-Mitchell (KSM) *Prefeasibility Study*. March 31, 2010.

Seabridge. 2011. Kerr-Sulphurets-Mitchell (KSM) *Prefeasibility Study Update,* NI43-101 Report, Trueman, R. and C. Mawdesley. 2003. *Predicting Cave Initiation and Propagation.* CIM Bulletin 96(1071):54-59.

Wong, Wing. 2011. *Senior Cost Estimator for Wardrop*, meeting with Donald Tolfree (Golder), September 2011.

Woo K., E. Eberhardt and A. van As. 2009. *Characterization and empirical analysis of block caving induced surface subsidence and macro deformations,* Proceedings of the 3rd CANUS Rock Mechanics Symposium, Toronto, Ontario, Canada.

APPENDIX A

Trade-off Studies

MEMORANDUM

TO File **DATE** November 2, 2011

CC David Sprott, Johnny Canosa, Donald Tolfree

FROM Andrew Lyon **PROJECT No.** 11-1439-0002

SEABRIDGE HAULAGE EQUIPMENT TRADE-OFF STUDY

Introduction

The Mitchell block cave is part Seabridge Gold's KSM project, located in Northwestern British Columbia, near the Eskay Creek Mine. The mineralized material is roughly 1,500 m wide by 600 m long and the development is located approximately 600 m below surface. A block caving study estimated the production rate to be 65,000 tonnes per day or 24 million tonnes per annum for the 42 year life of the mine. Material excavated from the extraction level is dumped into ore passes by Load Haul Dumps (LHDs), loaded on the haulage level, then dumped into a crusher and conveyed to surface. This trade-off study will focus on the type of equipment required on the haulage level. The options evaluated include:

- conventional diesel haul truck;
- electric haul truck;
- large capacity truck and trailer combination; and
- train.

Methods

The study is based on the mine parameters described above and the assumptions listed below. It was completed to a scoping level of detail and used the following equipment for each type of haulage:

- Sandvik TH680 diesel truck – 80 tonnes
- Kiruna K1050ED electric truck – 50 tonnes
- Powertrans Underground truck consisting of 1 powertruck, 2 tow trailers and 1 powertrailer – 125 tonnes
- Schalke 75T Locomotive with rail cars – 32 tonnes/car

A spreadsheet was created to list all the assumptions and common factors used for the trade-off study in order to compare them. The assumptions included diesel and electrical cost, utilization and availability, hours per day and days per year, fuel burn and haul speeds. These inputs were then used to calculate material movement per hour in the form of cycle times, which dictates the quantity of each type of equipment needed in order to move the specified 65,000 tonnes per day. These numbers, along with capital and operating costs, were then used to create a cash flow for the entirety of the mine life in order to calculate a net present cost (NPC) for each haulage option. The NPC allows for a relative comparison between each haulage option, thereby assisting in the selection of the most appropriate fleet. The cost inputs were obtained from supplier quotes and from Golder's database of costs for the Seabridge project.

Assumptions:

- Discount Factor: 5%
- Diesel cost: 1 $/L
- Electrical cost: 0.04 $kW/h
- Loaded labour cost: 46.69 $/hr
- Utilization: 85%
- Availability: 85%
- Replacement:
  - TH680 & Powertrans: every 10 years
  - Train Locomotive & Kiruna: every 15 years
- Haul Speeds: 15 km/h except electric Kiruna's – 19 km/h
- Sustaining Capital*:
  - Electrical & rail infrastructure: 1/10 total CAPEX in Operating cost per year
  - Rail cars: 1/15 total CAPEX in Operating cost per year

*The sustaining capital assumptions are for replacement costs of components as they wear out. In regards to the rail cars that equates to 4 replacements a year, and for the infrastructure its 10% of the capital per year.

Unit Capital Costs:

Sandvik	$	2,000,000
Kiruna	$	3,000,000
Powertrans		
Powertruck (x1)	$	1,050,000
Powertrailer (x1)	$	750,000
Tow Trailer (x2)	$	525,000
Total	$	2,325,000
Train		
Schalke Locomotive	$	2,500,000
Rail Car	$	130,000
Total	$	2,630,000

Infrastructure Capital Costs:

All options will use the excavated haulage drifts but the conventional trucks and powertrucks do not require additional infrastructure. The electric trucks and trains would require the following:

Electric Trucks			
Trolley line	$	800	$/m
Trolley substation*	$	125,000	$/1000 m

*Approximately 6,300 m of trolley line are required

Rail

100 lb rail cost	$	1,742	$/m
Power rail cost	$	364	$/m
Scroll dump cost	$	600,000	$

The four different haulage methods were compared based on the following categories:

- Equipment capital cost (LOM) – Capital cost of all the pieces of equipment over the life of the mine (LOM)
- Infrastructure capital cost (LOM) – only applicable for electric trucks and train. For the electric trucks this includes the trolley line and substations to boost power. For the trains this includes the rail cost, the power rail, and 2 scroll dumps.
- Operating cost – Fuel or electricity cost, maintenance, lube and labour.
- Ventilation capital cost – cost of raise boring shaft for ventilation. Specific fan prices not included.
- Ventilation operating cost – estimated using Ventsim software using the mine design and expected airflow requirements for the four methods.
- Operability and manoeuvrability – compares the flexibility of each system, based on a literature search.
- Ability to handle oversize – subjective comparison of each option's ability to handle oversize, based on a literature search.

Results

A comparison of the advantages and disadvantages of each haulage option is presented in Table 1. In the table, each of the options is ranked in each category ranging from 1 (best/highest) to 4 (lowest/worst). Based on this comparison, the preferred option would be the Kiruna electric haul trucks.

Table 1: Summary of advantages and disadvantages of various equipment types (1 is lowest/best, 4 is highest/worst).

Category	Powertrans	Sandvik TH680	Kiruna K1050ED	Schalke 75T
Ventilation requirements	4	3	1	2
Equipment unit capital cost	2	1	3	4
Equipment capital cost LOM	2	2	2	1
Infrastructure capital cost	1	1	3	4
Noise	4	3	1	1
Operating and maintenance cost	3	3	2	1
Operability & flexibility*	3	1	1	4
Ability to handle oversize*	good	poor	good	poor

***subjective ranking**

Table 2 contains a summary of the net present cost comparisons of the four options and Figure 1 contains a graph of these results.

Table 2: Comparison of total net present costs for the different options.

	Powertrans	Sandvik TH680	Kiruna K1050ED	Schalke 75T
Payload	125 tonnes	80 tonnes	50 tonnes	32 tonnes/car
Number of units	20	27	35	2 trolleys + 54 cars
Engine size per unit (kW)	640	317	91	946
Equipment capital cost LOM	$ 222,300,000	$ 212,000,000	$ 201,000,000	$ 34,656,000
Infrastructure capital cost LOM	$ -	$ -	$ 26,418,000	$ 60,754,123
Average yearly ops cost	$ 19,142,511	$ 16,267,915	$ 10,046,717	$ 1,333,379
Ventilation capital cost	$ 4,399,580	$ 3,942,481	$ 3,333,015	$ 3,333,015
Ventilation operating by yr	$ 5,500,000	$ 1,900,000	$ 700,000	$ 1,000,000
Net present cost @ 5 %	$ 556,752,100	$ 434,848,233	$ 369,094,196	$ 111,673,382

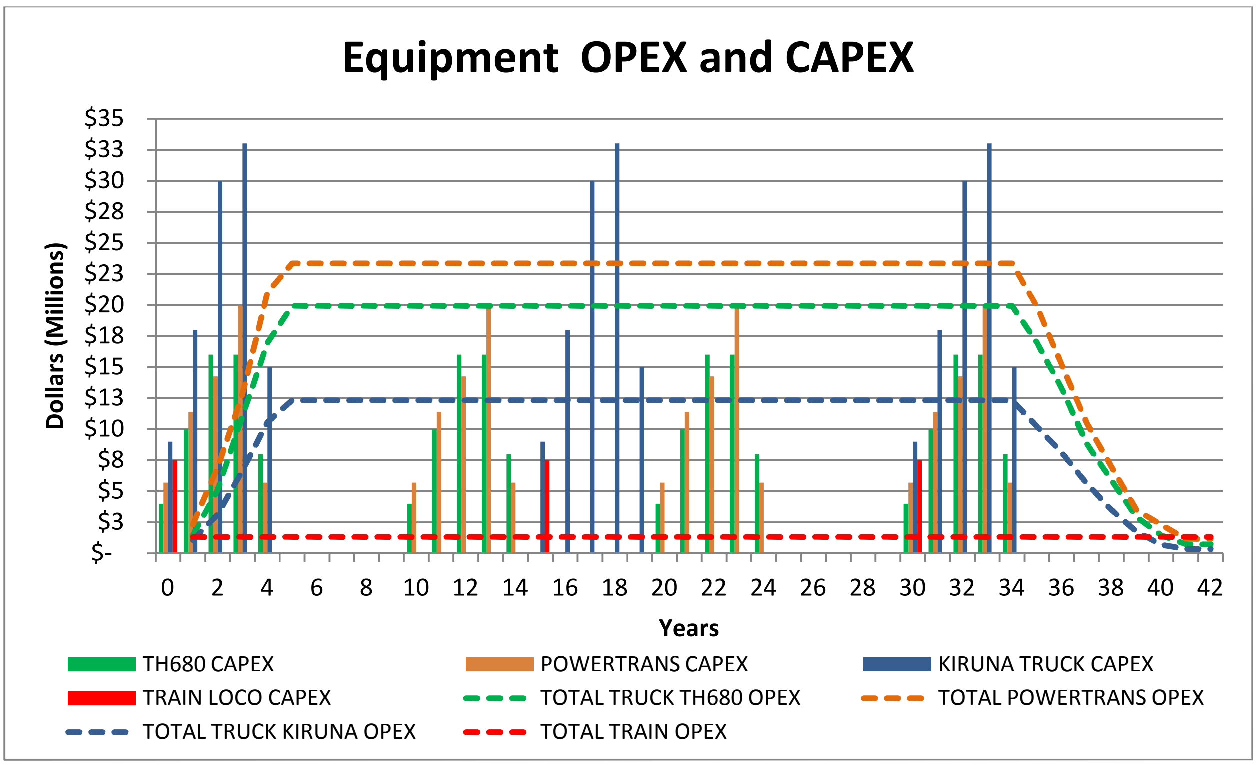

Figure 1: Yearly operating and capital expense for each of the four haulage options over the life of the mine

The results of the cost comparison show that the least expensive option over the LOM, by a significant margin, is the train. Both the LOM capital costs and annual operating costs for this option are lowest.

Discussion

Due to the relatively large ratio of engine size-to-tonnage, the Powertrans require the highest volume of ventilation air and consume the most fuel as seen in the average yearly operating cost ($19M). The larger engine size also increases the ventilation capital cost ($4.4M) and yearly operating cost ($5.5M). The Powertrans option requires fewer units (20) than conventional diesel trucks due to their larger payload.but because of their higher unit capital cost the overall Powertrans capital cost ($222M) over the LOM is the highest. The NPC of the Powertrans is approximately 557M.

The Sandvik diesel haul truck has similar but slightly lower capital cost over the life of the mine ($212M) than the Powertrans. The yearly operating cost is lower ($16M) due to the smaller engine size, and ventilation capital costs are slightly lower ($3.9M). The yearly ventilation costs are more than halved ($1.9M) because of the smaller engine. However, more vehicles (27) are required due to the reduced payload. The estimated NPC of the diesel trucks is $434M.

Since the Kiruna has the smallest payload, this option requires the most units among the wheeled equipment (35). Even with the highest price per unit, the Kiruna has the lowest equipment capital cost ($201M) due to a longer equipment life of 15 years (vs. the 10 years for the diesel trucks). In order to run the Kiruna's, overhead power lines, substations and control panels need to be installed along all the haulage drifts. This infrastructure amounts to an extra $26M over the LOM compared to the diesel trucks. On the other hand, the Kiruna has the lowest yearly operating cost, ventilation capital cost and ventilation operating cost per year. These values are $10M, $3.3M and $0.7M, respectively. The estimated NPC of the electric trucks is $369M and it ranks as the second least expensive alternative.

Lastly, the Schalke train has the largest engine, but because only 2 locomotives are needed, the ventilation requirements are as low as that of the Kiruna trucks. As with the Kiruna trucks, the Schalke train needs infrastructure installed. The train has the highest infrastructure capital cost of all the options, totalling $61M over the LOM, with much of this cost incurred at the start of the mine life. Conversely, the train option has the lowest equipment capital cost since only 2 locomotives and 54 rail cars are needed. Over the LOM, this amounts to $34M, with a yearly operating cost of only $1.3M, due to the relatively low cost of electricity and the need for such a small number of locomotives. Of the four options, the trains have the lowest estimated NPC of $111M.

Conclusion

Of the four options considered for material haulage at the Mitchell mine, the trains have the lowest NPC. Their inflexibility is not a concern because the block cave mining method is inherently inflexible and the size of the material feeding the train can be controlled through grizzlies and rockbreakers. The biggest benefit of the train is the low equipment capital cost over the LOM. Even though the locomotives are expensive, and many rail cars would have to be purchased, these expenses amount to less overall capital than the three other options. In addition, the LOM operating cost of the train is very low, mostly due to the low cost of electricity and the need for only 2 operators. The main drawback to the train option is that it has the highest initial capital expense of all the options presented in this study.

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TO Project File 11-1439-0002 Phase 2000 **DATE** August 17, 2011

CC Dave Sprott, Donald Tolfree

FROM Johnny Canosa **PROJECT No.** 111439002

MITCHELL BLOCK CAVE STUDY – DIESEL VS ELECTRIC LHD TRADE-OFF STUDY

Introduction

The Mitchell block cave is part Seabridge Gold's KSM project, located in Northwestern British Columbia, near the Eskay Creek Mine. The mineralized material is roughly 1,500 m wide by 600 m long and the development is located approximately 600 m below surface. A block caving study estimated the production rate to be 65,000 tonnes per day or 24 million tonnes per annum for the 42 year life of the mine. Material excavated from the extraction level is dumped into ore passes by Load Haul Dumps (LHDs), loaded into rail cars on the haulage level, then dumped into a crusher and conveyed to surface. Because of the quantity of LHDs required to meet production targets (12) and the relatively inexpensive electrical power at the project site, this trade-off study will compare the economics of using electric and diesel powered LHDs.

LHDs are commonly used in block cave mines and diesel LHDs have historically been the equipment of choice. However, diesel equipment has two major drawbacks including potential for poorer underground air quality due to diesel emissions and particulates and the rising cost of diesel fuel. In some regions the cost of electric power is relatively low and the use of electric LHDs is becoming more popular. Table 1 lists some high tonnage underground block cave mines and the type of LHD that they are using or are planning on using.

Table 1: A list of mines using either Diesel or Electric LHD's

Diesel LHD	Electric LHD
Codelco-El Teniente	Rio Tinto –NorthParkes
Palabora	NewCrest – Cadia East, Ridgeway2.
Finsch Mine- De Beers	
Freeport-Indonesia	

Methods

The study is completed to a scoping level of detail and is based on the mine parameters described above and the assumptions listed below. Cost inputs were obtained from the Costmine Database and were used to estimate the yearly operating cost of each type of equipment. In addition, a ventilation capital cost was estimated for each option. The yearly operating costs and capital costs were combined into a Net Present Cost value to account for discounting over the LOM.

The LH621 and LH625E were used in this study. These machines have similar capacities (approximately 22 tonnes) and productivity estimates indicate that a total of 12 units are required to reach the daily tonnage.

- Discount Factor: 5%
- Diesel cost: 1 $/L

- Electrical cost: 0.04 $kW/h
- Loaded labour cost: 31.38 $/hr
- Utilization: 85%
- Availability: 85%
- Replacement (same for both pieces):
    - Overhaul every 5 years
    - Replace every 10 years
- Speed: same for both types

Results and Discussion

A summary of the input values and the NPC results are shown in Table 2. Additional detail concerning the cost inputs can be found in the tables attached to this document.

Table 2: Cash Flow Analysis Summary

	Diesel LHD's	Electric LHD's
Equipment Capital Cost	$19,200,000	$26,760,000
Number of Units Operating	12	12
Average Yearly Ops Costs	$12,885,000	$12,873,000
Ventilation Capital Costs	$3,790,000	$3,028,000
Years	20	20
Net Present Cost @5%	$174,400,000	$176,863,000

The electric LHDs have a higher initial capital cost because the unit cost of the equipment is higher. However, they have a smaller ventilation capital cost because electric LHDs do not need as much airflow to dilute emissions. The yearly operating cost is similar for both types of equipment. According to this study, the diesel LHD has a slightly lower NPC than the electric LHD. However, the difference between the results is well within the level of accuracy of this study and so the NPC's are considered the same.

To help determine which type of equipment to use in the Mitchell pre-feasibility block cave design, a list of the pros and cons of each piece of equipment is also listed below in Tables 3 and 4.

Table 3: Pros and cons of using diesel powered LHDs

Pros	Cons
■ The anticipated maximum one-way tramming distances for LHD's on the extraction level will be 890 m. The average haul distance will be less, approximately 200 m. However, part of the large fragmentation risk mitigation includes multiple ore passes in one extraction drift. It is conceivable that an LHD will have to haul across the footprint	■ More emission of gas and heat ■ Higher noise levels

Pros	Cons
(890 m) during the mine life, although it may be infrequent. ■ Recent developments in water-cooled diesel engine technology has resulted in much higher fuel efficiencies over air-cooled diesel LHD's (28.7 L/hr versus 34L/hr). ■ Lower initial capital costs	

Table 4: Pros and cons of using electric powered LHDs

Pros	Cons
■ Emit very low noise and no exhaust fumes, thus ensuring a better work environment ■ Lower ventilation capital cost	■ Experience at Northparkes Mine indicates that the electric LHD can have a maximum cable reel length of 336 m. This would limit flexibility on the extraction level and would not allow the machine to take advantage of some of the large fragmentation risk mitigation measures. ■ High initial capital costs ■ The electric LHD costs do not include back-up power battery packs to enable mobility when a power transmission outlet is not available or when there is a power outage, again limiting flexibility.

Conclusions:

It was decided that the diesel LHDs should be used in the Mitchell pre-feasibility block cave design. The decision is based on the increased flexibility of the diesel LHD and the fact that the electric LHD would not be able to take full advantage of the fragmentation risk mitigation factors included in the overall mine design (the cable reel on an electric LHD has a maximum range of approximately 350 m). This decision is based on the logistics of using the equipment underground and not on the NPC. The NPC's for both options were the same. If the mine design were modified to eliminate the drawbacks of using an electric LHD (i.e. shorter maximum tram distances), it is recommended that this study be revisited.

DATE January 16, 2012 **PROJECT No.** 11-1439-0002

TO Project File 11-1439-0002 Phase 2000

CC D.Tolfree, D. Sprott

FROM Clyde Cooper **EMAIL** clyde.cooper@gmail.com

SEABRIDGE GOLD MITCHELL PROJECT – SHAFT VERSUS CONVEYOR TRADE-OFF STUDY

Introduction

The Mitchell block cave is part Seabridge Gold's KSM project, located in Northwestern British Columbia, near the Eskay Creek Mine. The mineralized material is roughly 1,500 m wide by 600 m long and the development is located approximately 600 m below surface. A block caving study estimated the production rate to be 65,000 tonnes per day or 24 million tonnes per annum for the 42 year life of the mine. Material excavated from the extraction level is dumped into ore passes by Load Haul Dumps (LHDs), loaded onto rail cars on the haulage level, then dumped into a crusher and conveyed to surface.

A scoping level trade-off study was conducted to determine whether shaft or conveyor haulage would be most cost effective for the Mitchell mine. The study evaluated three options: a conveyor decline, one ten meter diameter shaft, and two seven meter diameter shafts. The three options were evaluated based upon their respective development, infrastructure purchase, and construction costs. (The operating costs for each item were not considered because they are similar enough that it would not impact the results of the study.) These costs were then discounted over each construction period to create a Net Present Cost (NPC) for each option. The results of the study can be found in Table 1.

Table 1: Trade-Off Study Results

Option	NPC @ 5%
Conveyor	$ 61,014,000
10 m Shaft	$ 104,761,000
2 x 7 m Shafts	$ 118,340,000

Conveyor production rates were based upon having a 54" belt deliver up to 3,000 tonnes per hour in order to meet the peak production requirement of 65,000 tpd. Conveyor ramp development costs were based upon the Mitchell cost model created by Golder. Conveyor purchase costs were based upon a 2011 quote from Sandvik obtained for the Iron Cap cost model. Conveyor installation costs were assumed to be equal to the excavation cost as a timely quote was unable to be obtained for the trade-off study.

It was assumed that a 10 meter diameter shaft would be able to meet or exceed peak production requirements based upon other mining projects such as Oyu Tolgoi's Shaft No. 3 (10 m diameter) which is designed to provide a total capacity of 55,606 tpd (AMEC Minproc, 2010). It was assumed that the 7 meter diameter shafts would be able to provide at least 27,500 tpd each as mines such as Palabora in South Africa can hoist 30,000 tpd from a 7.4 m diameter shaft (Taljaard and Stephenson, 2000). Shaft infrastructure and installation costs were based upon a quote from Thyssen Mining, Costmine, and the Mitchell cost model.

The shaft options included a smaller conveyor from the footprint to the shaft bottom. A preliminary subsidence evaluation was completed to provide the expected subsidence crater boundary location. The shafts were positioned outside this boundary and a conveyor was used to transport the material to the shaft pocket.

Mine hoists and shaft loading stations were not included in the shaft cost estimate. The goal of this trade-off study was to determine the most cost effective method of moving broken material to surface. The addition of these items would only penalize the shafts further and would not change the outcome.

The conclusion of the study was that developing and installing a conveyor from surface to the underground crushers would be the cheapest option to achieve the required production targets. The NPC for the conveyor option was just over 40% less than the ten meter shaft option which had the next lowest NPC. The conveyor option was much cheaper due to the lower cost of development. Further studies should include a detailed installation quote for the conveyor and the annual operating costs for each system over the life of the project.

AMEC Minproc (2010). *Oyu Tolgoi Project Technical Report*. Retrieved January 16, 2012, from http://www.ivanhoemines.com/i/pdf/IDP10_June062010.PDF

Taljaard, J.J. and Stephenson, J.D. (2000). State-of-art shaft system as applied to Palabora underground mining project. *The Journal of The South African Institute of Mining and Metallurgy*, November/December 2000, 427-436. Retrieved January 16, 2012, from http://www.saimm.co.za/Journal/v100n07p427.pdf

Clyde Cooper
Mining Engineer

Donald Tolfree
Mining Engineer

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Design Details

Conveyor

Item	Length (m)	Unit cost ($/m)	Total Cost ($)	Unit Construction	Total Construction Time (years)
Drift	4,150	5,000	20,750,000	3.70	3.07
Connection Tunnels	1,140	5,000	5,700,000	3.70	0.84
Conveyor Installation	4,150	5,000	20,750,000	5.00	2.27
Conveyor Purchase	4,150	4,375	18,156,250		
Shaft (1x10m)					
Shaft #1 (10m)	631	132,335	83,503,436	225.14	2.80
Permenant Headframe			8,700,000		
Shaft Stations	316	5,000	1,577,500	3.70	0.23
Conveyor Drifts	1,065	5,000	5,325,000	3.70	0.79
Conveyor Installation	1,065	5,000	5,325,000		
Conveyor Purchase	1,065	4,375	4,659,375		
Shaft (2x7m)					
Shaft #1 (7m)	631	79,730	50,309,449	266.26	2.37
Shaft #2 (7m)	631	79,730	50,309,449	266.26	2.37
Permenant Headframe			8,700,000		
Shaft Stations	631	5,000	3,155,000	3.70	0.47
Conveyor Drifts	1,065	5,000	5,325,000	5.00	0.58
Conveyor Installation	1,065	5,000	5,325,000		
Conveyor Purchase	1,065	4,375	4,659,375		

		Construction Time							
Conveyor	**Total Cost ($)**	**Year1**	**Year2**	**Year3**	**Year4**	**Year5**	**Year6**	**Year7**	**Year8**
Drift	$20,257,500	$6,752,500	$6,752,500	$6,752,500	$492,500				
Connection Tunnels	$5,700,000	$2,850,000	$2,850,000						
Conveyor Installation			$6,916,667	$6,916,667	$6,916,667				
Conveyor Purchase	$18,156,250		$9,078,125	$9,078,125					
Total	*$ 44,113,750*	*$ 9,602,500*	*$ 25,597,292*	*$ 22,747,292*	*$ 7,409,167*				
Discount Rate	**5%**								
Conveyor NPC	**$ 61,013,654**								

Shaft	**Total Cost ($)**	**Year1**	**Year2**	**Year3**	**Year4**	**Year5**	**Year6**	**Year7**	**Year8**
Shaft #1 (10m)	$83,503,436	$30,061,237	$30,061,237	$23,380,962					
Permenant Headframe	$8,700,000				$8,700,000				
Shaft Stations	$6,752,500		$3,376,250	$3,376,250					
Conveyor Drifts	$2,850,000				$2,850,000				
Conveyor Installation	$5,325,000					$5,325,000			
Conveyor Purchase	$4,659,375			$4,659,375					
Total	*$ 111,790,311*	*$ 30,061,237*	*$ 33,437,487*	*$ 31,416,587*	*$ 11,550,000*	*$ 5,325,000*			
Discount Rate	**5%**								
10m Shaft NPC	**$ 104,760,448**								

Shaft	**Total Cost ($)**	**Year1**	**Year2**	**Year3**	**Year4**	**Year5**	**Year6**	**Year7**	**Year8**
Shaft #1 (7m)	$50,309,449	$21,129,969	$21,129,969	$8,049,512					
Shaft #2 (7m)	$50,309,449					$21,129,969	$21,129,969	$8,049,512	
Permenant Headframe	$17,400,000				$8,700,000				$8,700,000
Shaft Stations	$3,155,000		$1,577,500	$1,577,500					
Conveyor Drifts	$5,325,000				$5,325,000				
Conveyor Installation	$5,325,000					$5,325,000			
Conveyor Purchase	$4,659,375			$4,659,375					
Total	*$ 136,483,273*	*$ 21,129,969*	*$ 22,707,469*	*$ 14,286,387*	*$ 14,025,000*	*$ 26,454,969*	*$ 21,129,969*	*$ 8,049,512*	*$ 8,700,000*
Discount Rate	**5%**								
Two 7m Shafts NPC	**$ 118,339,674**								

APPENDIX B

Geotech – G1

Table B-1: Rock Mass Rating (RMR_{76}) System

	Parameter		Ranges of Values						
1	Strength of intact rock material	Point load strength index	> 8 MPa	4-8 MPa	2-4 MPa	1-2 MPa	For this low range uniaxial		
		Uniaxial compressive strength	> 200 MPa	100-200 MPa	50-100 MPa	25-50 MPa	10-25 MPa	3-10 MPa	1-3 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	<25%		
	Rating		20	17	13	8	3		
3	Spacing of joints		>3 m	1-3 m	0.3 – 1 m	50 – 300 mm	<50 mm		
	Rating		30	25	20	10	5		
4	Condition of joints		Very rough surfaces Not continuous No Separation Hard joint wall rock	Slightly rough surfaces Separation <1 mm Hard joint wall rock	Slightly rough surfaces Separation <1 mm Soft joint wall rock	Slickensided surfaces OR Gouge <5 mm thick OR joint	Soft gouge >5 mm thick OR Joints open >5 mm continuous joints		
	Rating		25	20	12	6	0		
5	Groundwater	Inflow per 10 m per tunnel length	None		<25 litres / min	25-125 litres / min	>125 litres / min		
		Raito joint water pressure / major principal stress	0		0.0 – 0.2	0.2 – 0.5	>0.5		
		General conditions	Completely dry		Moist only (interstitial water)	Water under moderate pressure	Server water problems		
	Rating		10		7	4	0		

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VERY POOR ROCK (RMR = 0-20)
M-11-125: 705.07 – 705.58 m

POOR ROCK (RMR = 20-40)
M-11-125: 76.70 – 78.50 m

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY MITCHELL PROJECT, BRITISH COLUMBIA
TITLE	EXAMPLE CORE PHOTOGRAPHS OF VERY POOR AND POOR ROCK

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	MV	10FEB12	SCALE NTS	REV.--
CADD	MV	10FEB12	B-1	
CHECK	KMM			
REVIEW	RDH			

FAIR ROCK (RMR = 40-60)

M-11-125: 188.25 – 190.88 m

GOOD ROCK (RMR = 60-80)

M-11-125: 297.00 – 299.01 m

PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
MITCHELL PROJECT, BRITISH COLUMBIA

TITLE

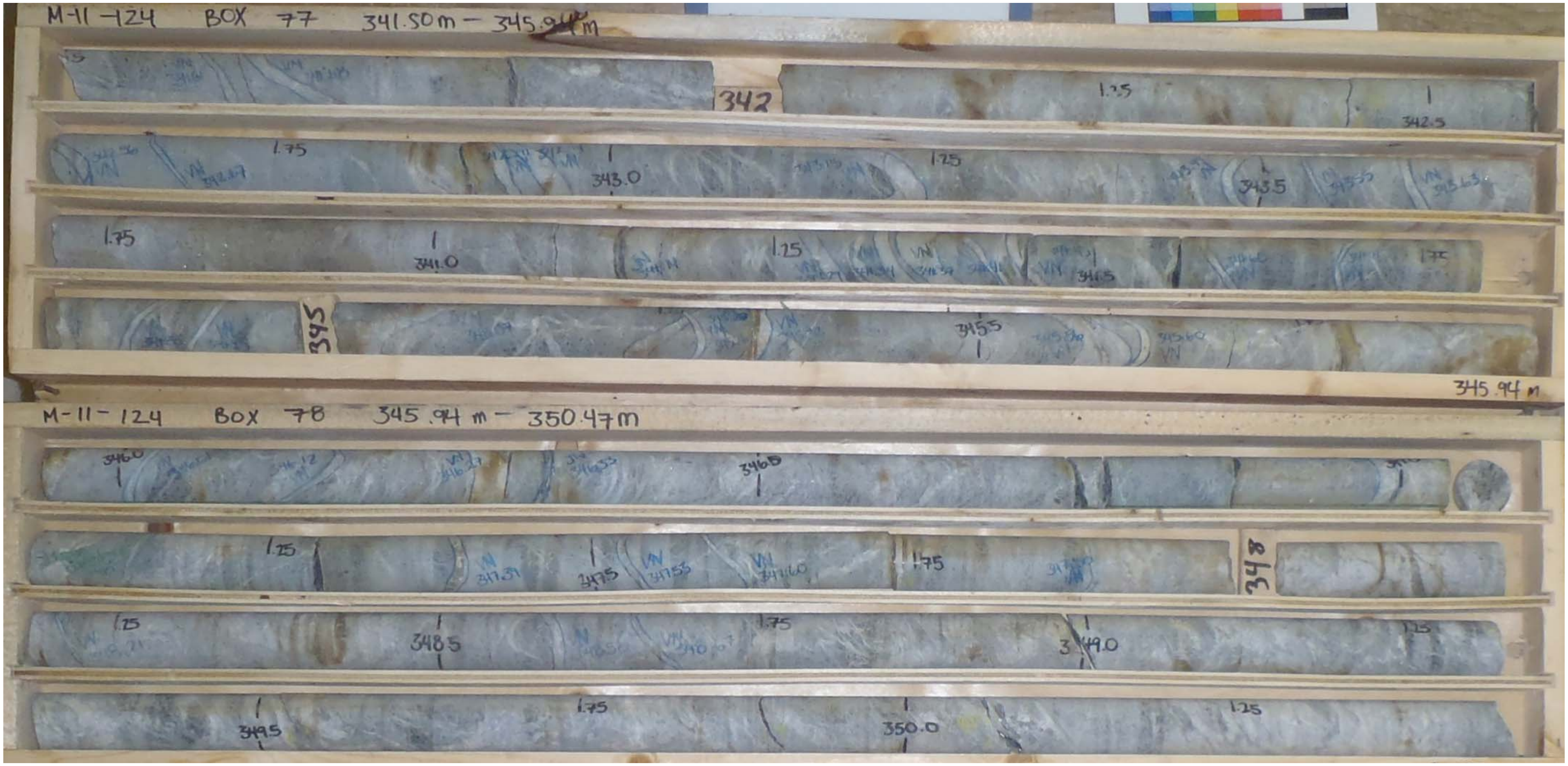
EXAMPLE CORE PHOTOGRAPHS OF FAIR AND GOOD ROCK

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	MV	10FEB12	SCALE NTS	REV.--
CADD	MV	10FEB12	**B-2**	
CHECK	KMM			
REVIEW	RDH			

VERY GOOD ROCK (RMR = 80-100)

M-11-124: 341.50 – 350.47 m



PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
MITCHELL PROJECT, BRITISH COLUMBIA

TITLE

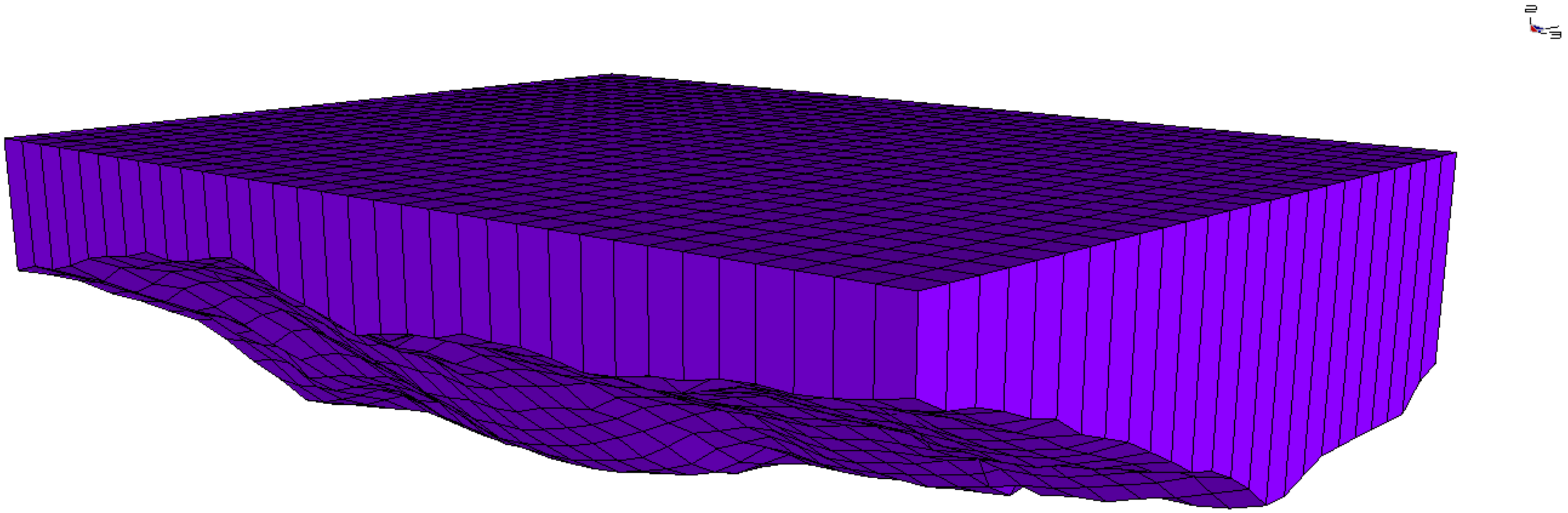
EXAMPLE CORE PHOTOGRAPH OF VERY GOOD ROCK

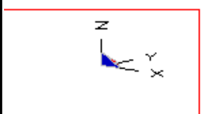
Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000
DESIGN	MV	10FEB12	SCALE NTS REV. --
CADD	MV	10FEB12	
CHECK	KMM		B-3
REVIEW	RDH		

APPENDIX C

Geotech - G2

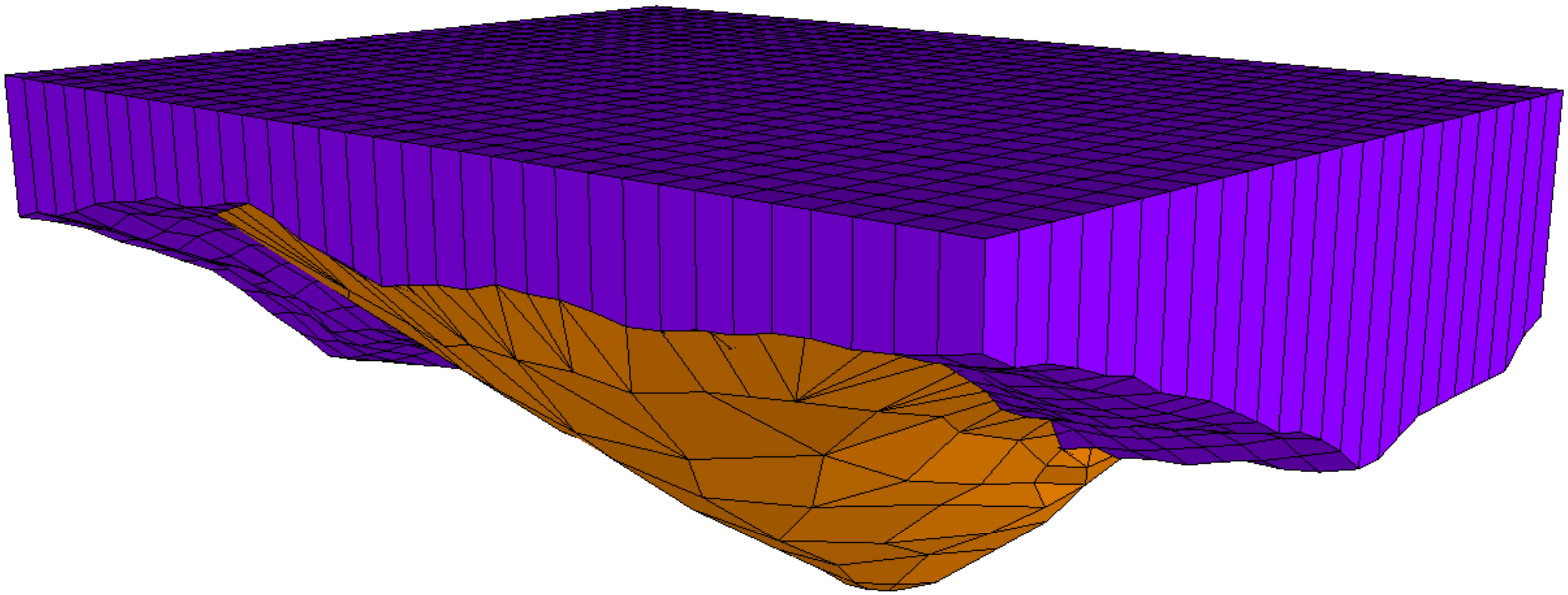




PROJECT	**SEABRIDGE GOLD INC.** **KSM CONCEPTUAL STUDY** **BRITISH COLUMBIA**
TITLE	**MITCHELL MAP3D MODEL** **STEP 1**

PROJECT No. 11-1439-0002			PHASE No. 10000
DESIGN	CY	26JAN12	SCALE NTS REV. --
CADD	CY	26JAN12	**FIGURE C-1**
CHECK	KMM		
REVIEW	RDH		

Golder Associates

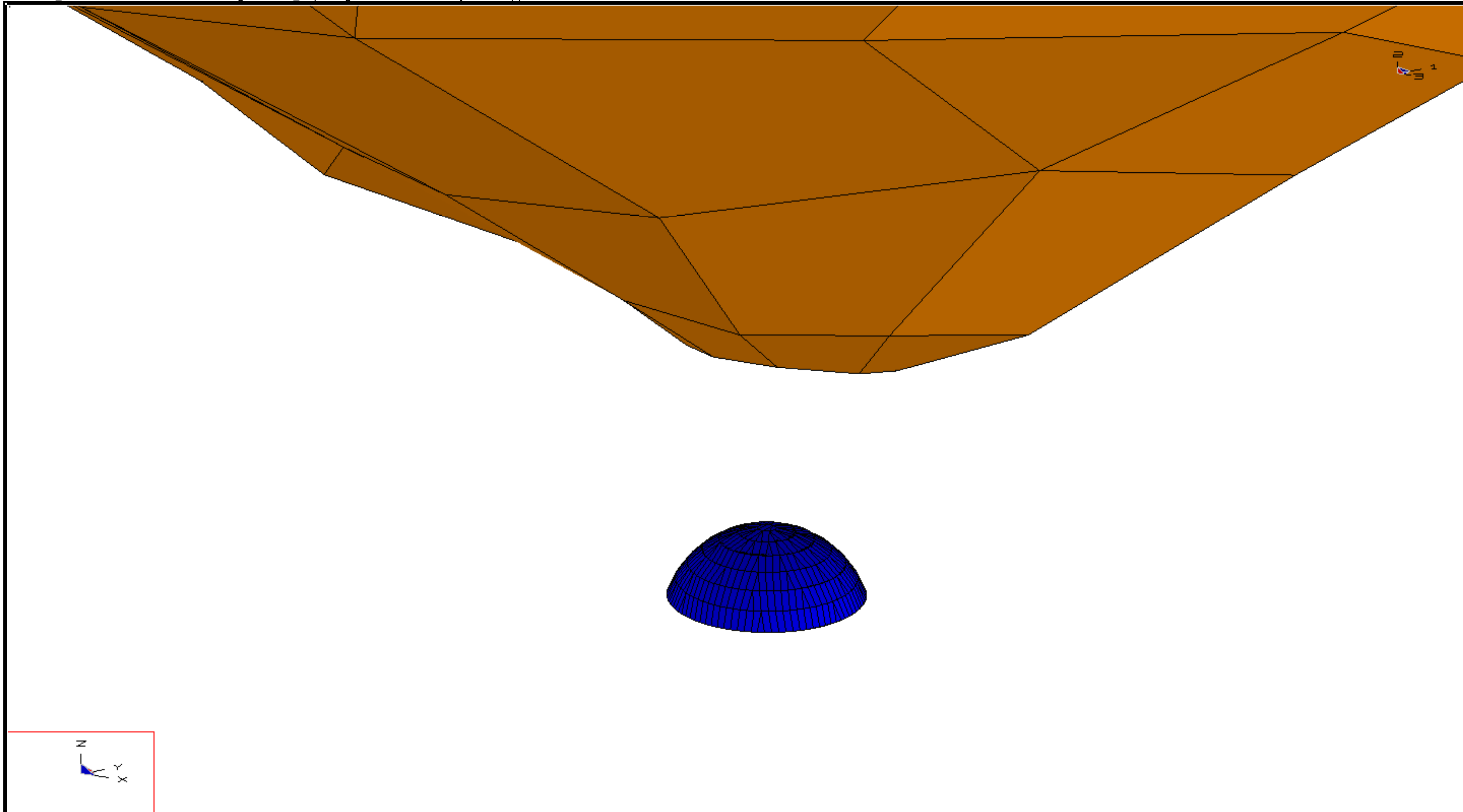


PROJECT
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE
MITCHELL MAP3D MODEL
STEP 2

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	26JAN12	SCALE NTS	REV.--
CADD	CY	26JAN12	FIGURE C-2	
CHECK	KMM			
REVIEW	RDH			



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL STEP 3

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	26JAN12	SCALE NTS	REV. --
CADD	CY	26JAN12	FIGURE C-3	
CHECK	KMM			
REVIEW	RDH			

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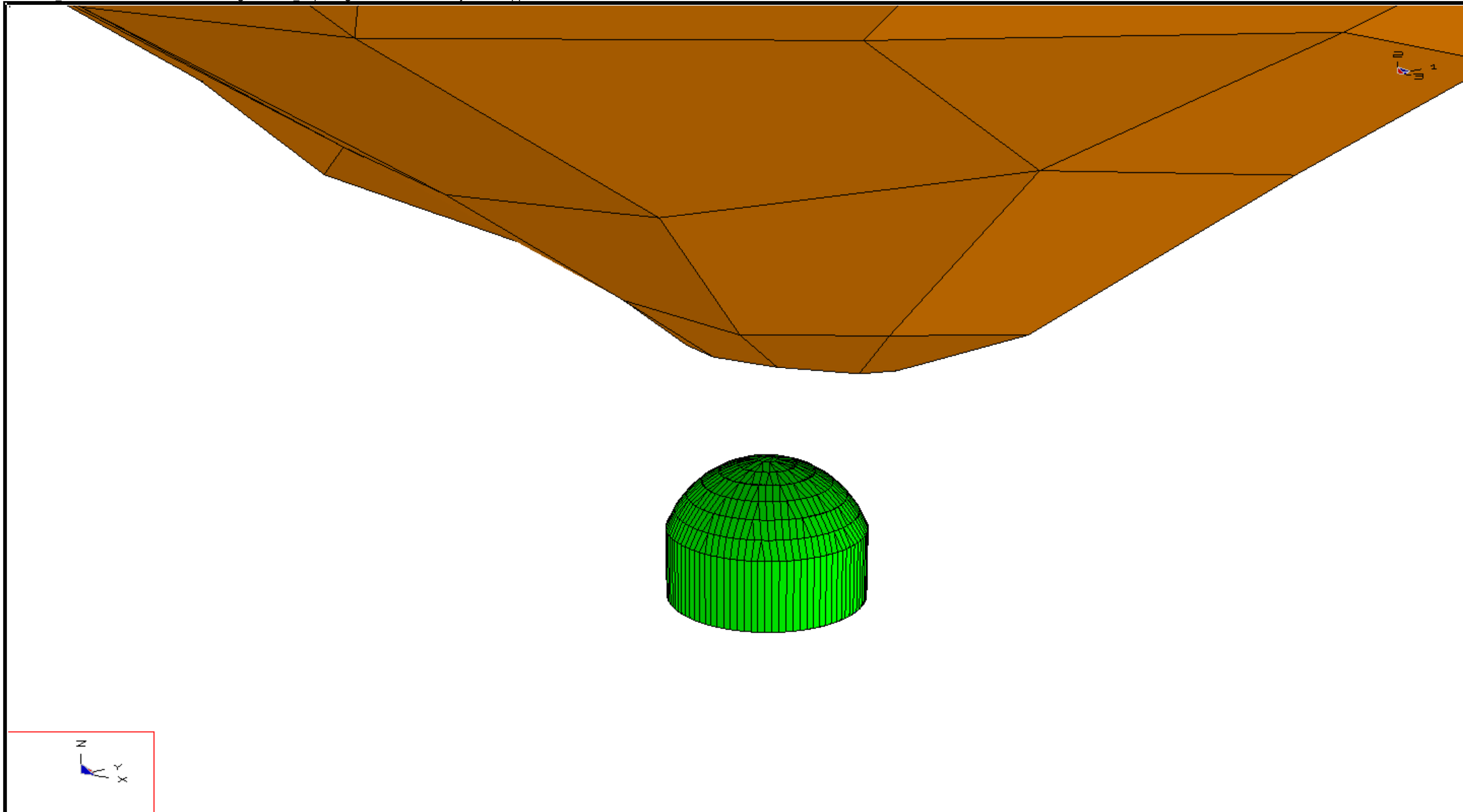

PROJECT
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE
MITCHELL MAP3D MODEL
STEP 4

PROJECT No. 11-1439-0002			PHASE No. 10000
DESIGN	CY	26JAN12	SCALE NTS REV. --
CADD	CY	26JAN12	**FIGURE C-4**
CHECK	KMM		
REVIEW	RDH		

Golder Associates

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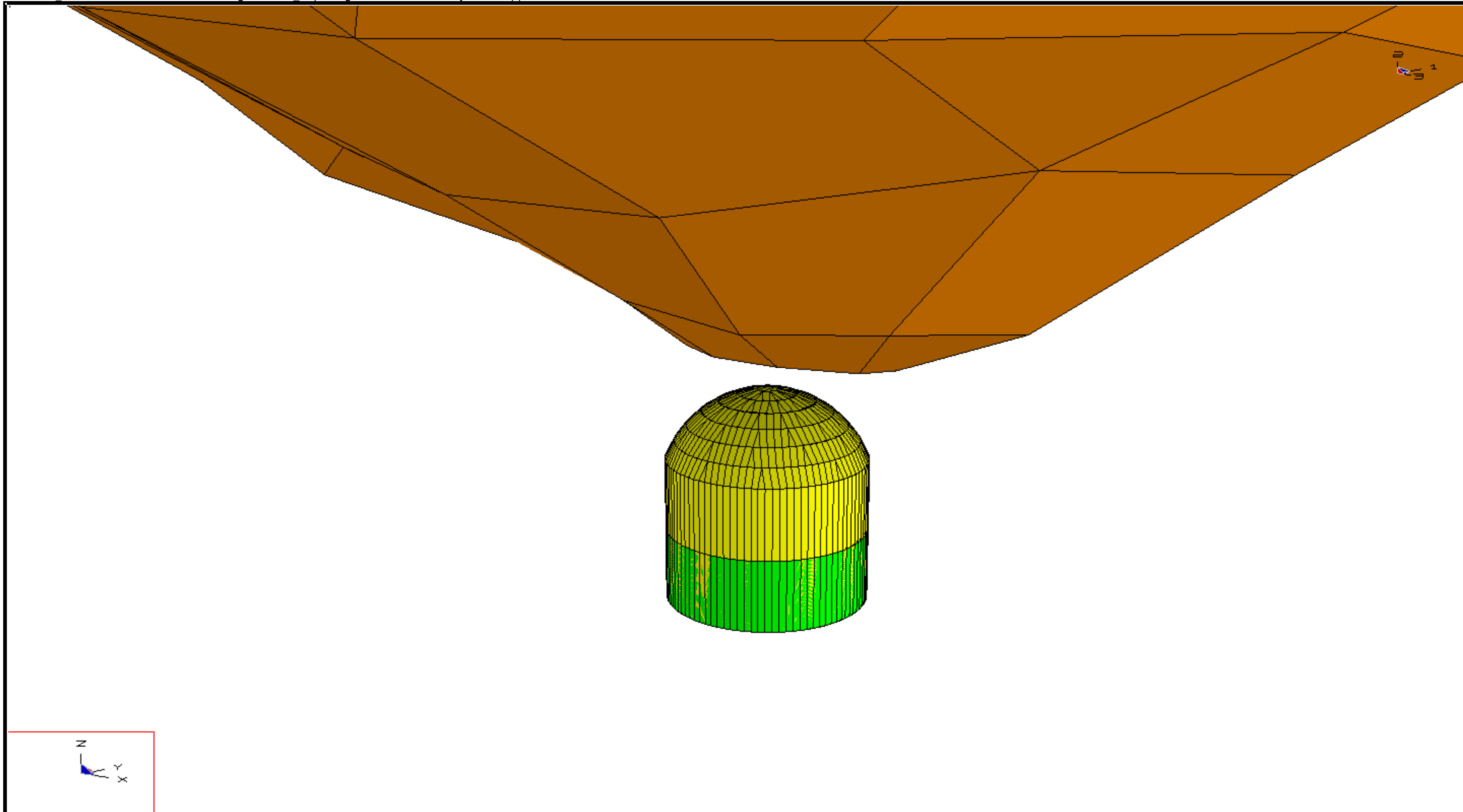

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL STEP 5

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
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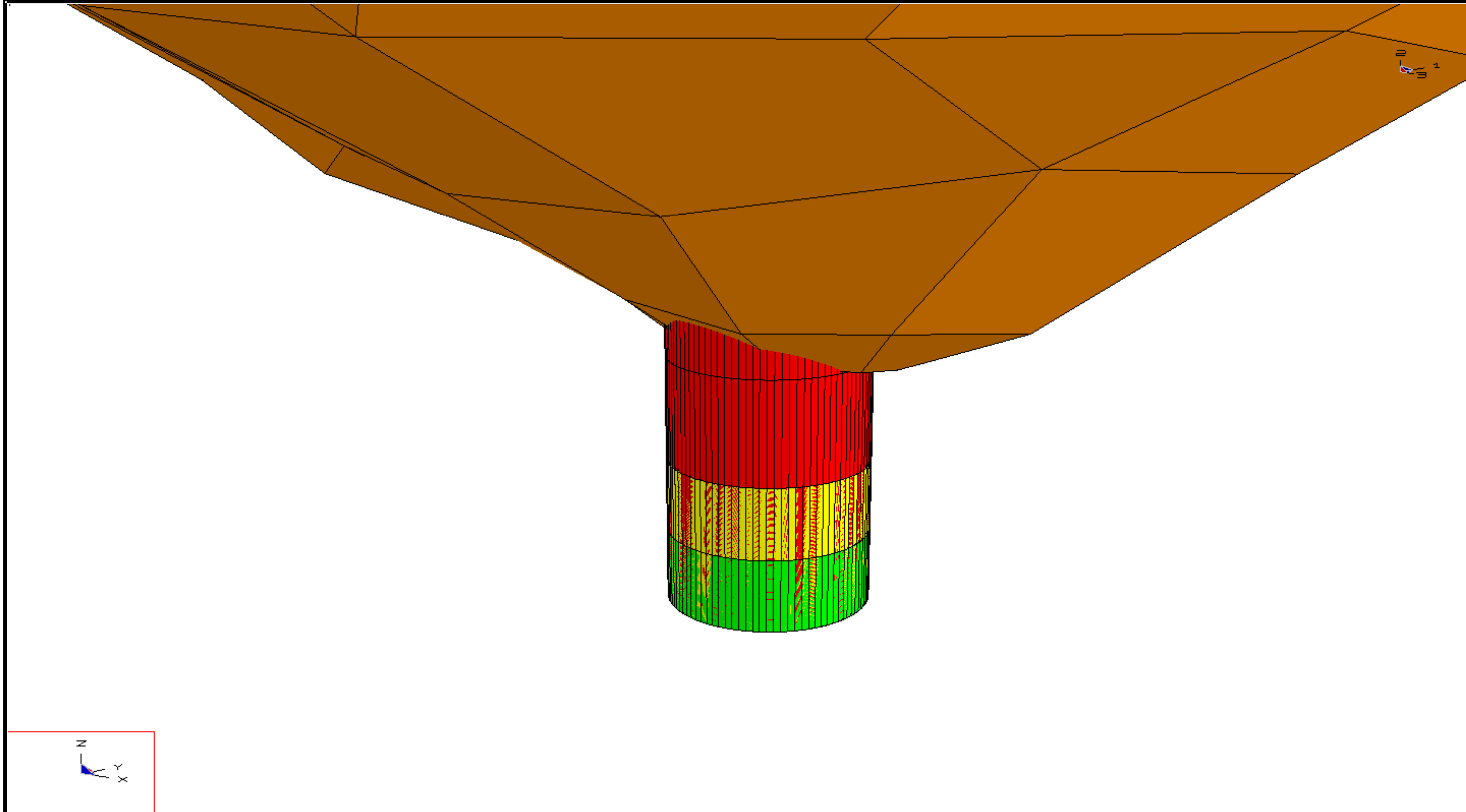
CADD	CY	26JAN12	FIGURE C-5	
CHECK	KMM			
REVIEW	RDH			



PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE

MITCHELL MAP3D MODEL
STEP 6

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	26JAN12	SCALE NTS	REV. --
CADD	CY	26JAN12	**FIGURE C-6**	
CHECK	KMM			
REVIEW	RDH			

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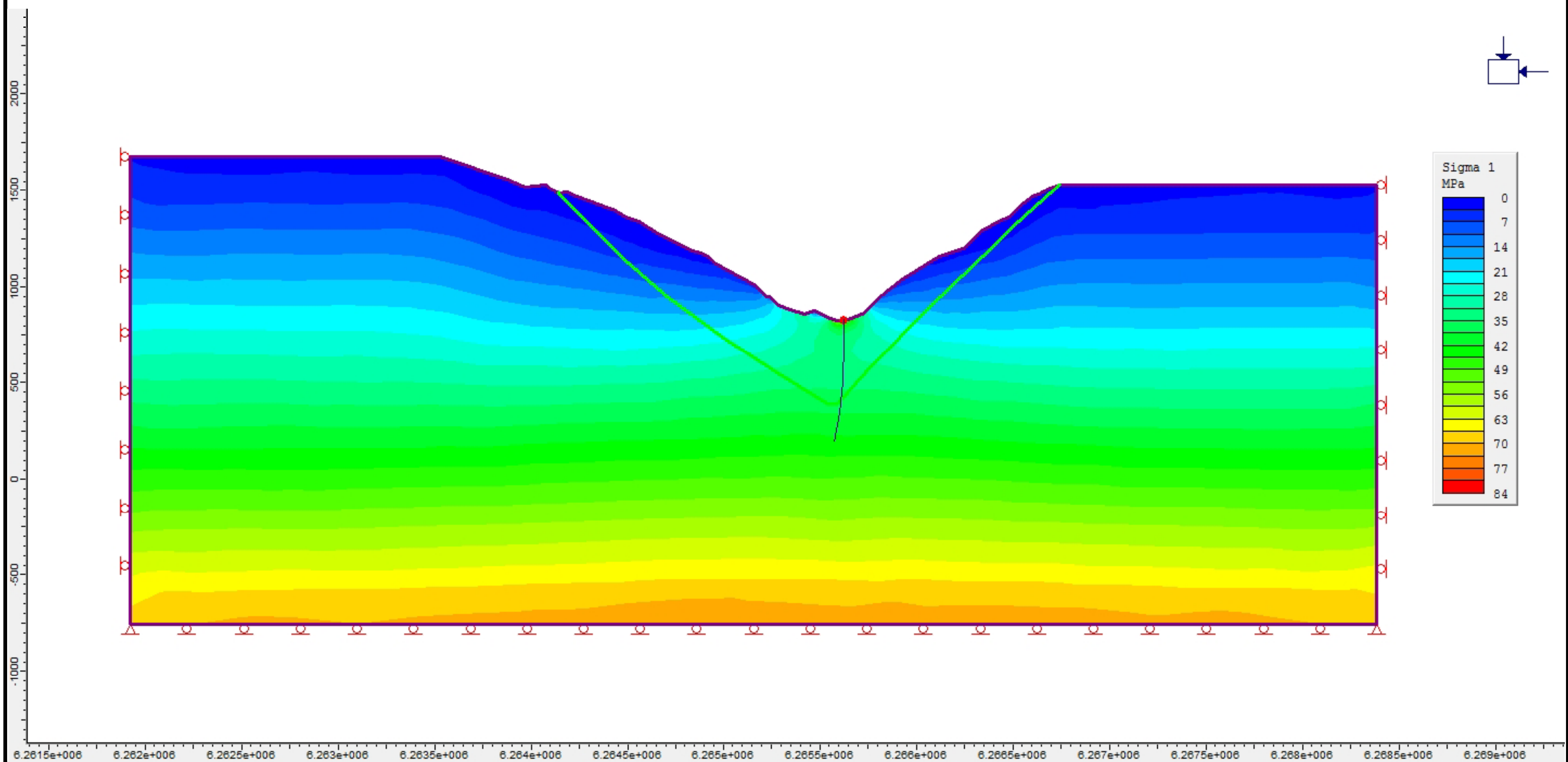

PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE

MITCHELL PHASE2 MODEL RESULTS
RESULTS – SIGMA1

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	26JAN12	SCALE NTS	REV.--
CADD	CY	26JAN12	**FIGURE C-7**	
CHECK	KMM			
REVIEW	RDH			

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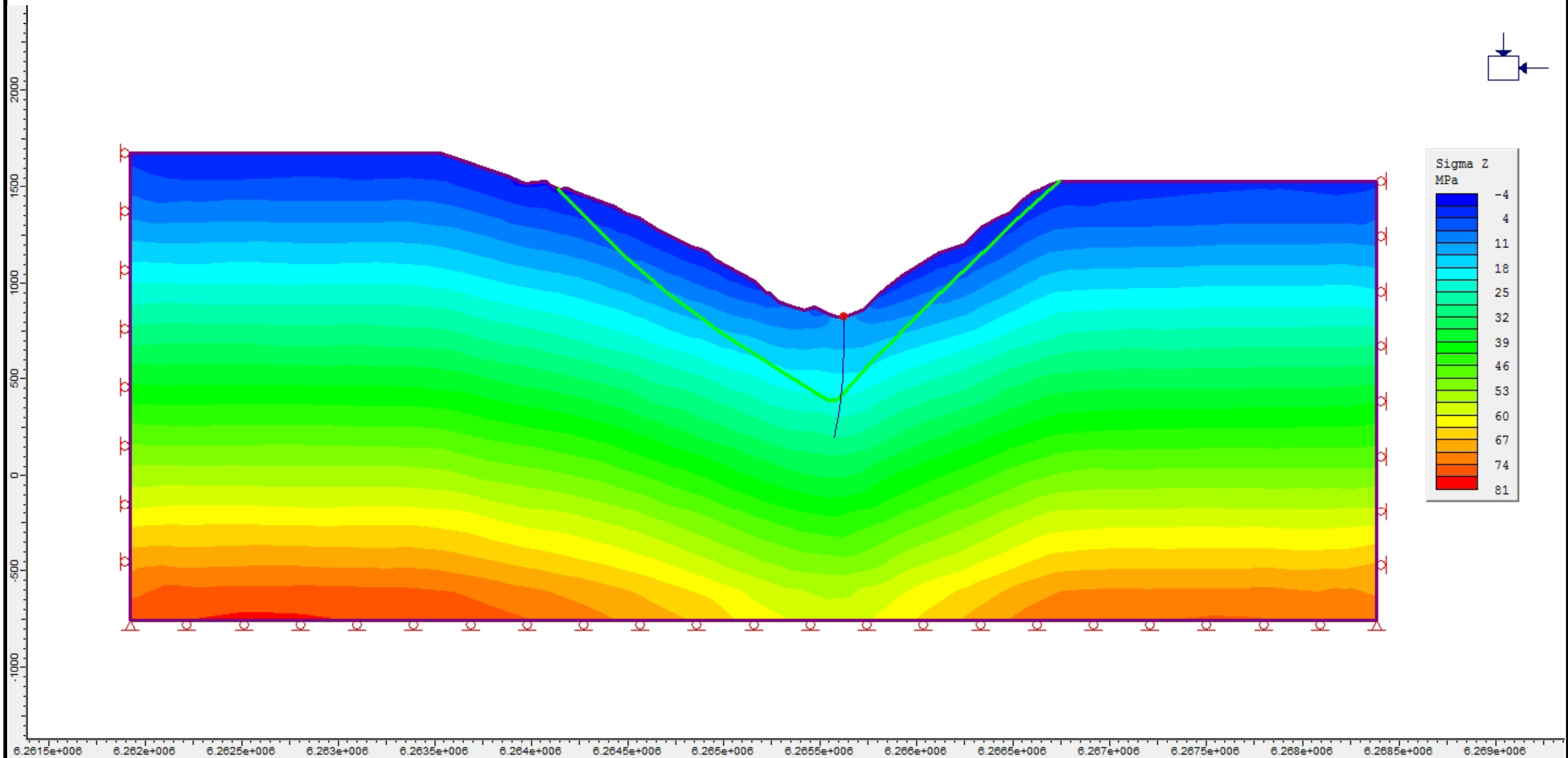
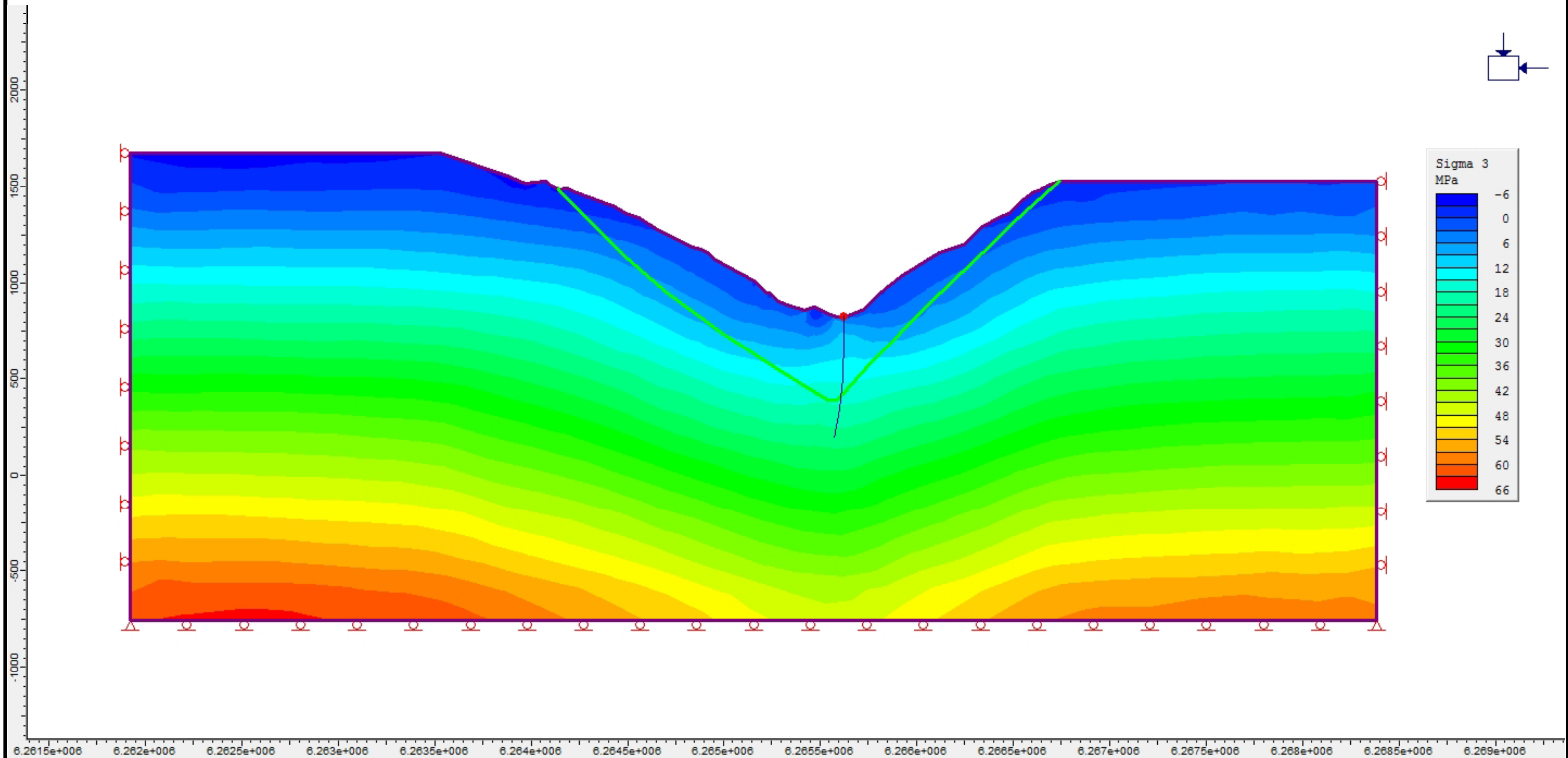


PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL PHASE2 MODEL RESULTS RESULTS – SIGMA2

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	26JAN12	SCALE NTS	REV.--
CADD	CY	26JAN12	FIGURE C-8	
CHECK	KMM			
REVIEW	RDH			

O:\Active_2011\1439\11-1439-0002 Seabridge PFS\10_Reporting\Mitchell Stress Analysis TM\Appendices



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL PHASE2 MODEL RESULTS RESULTS – SIGMA3

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
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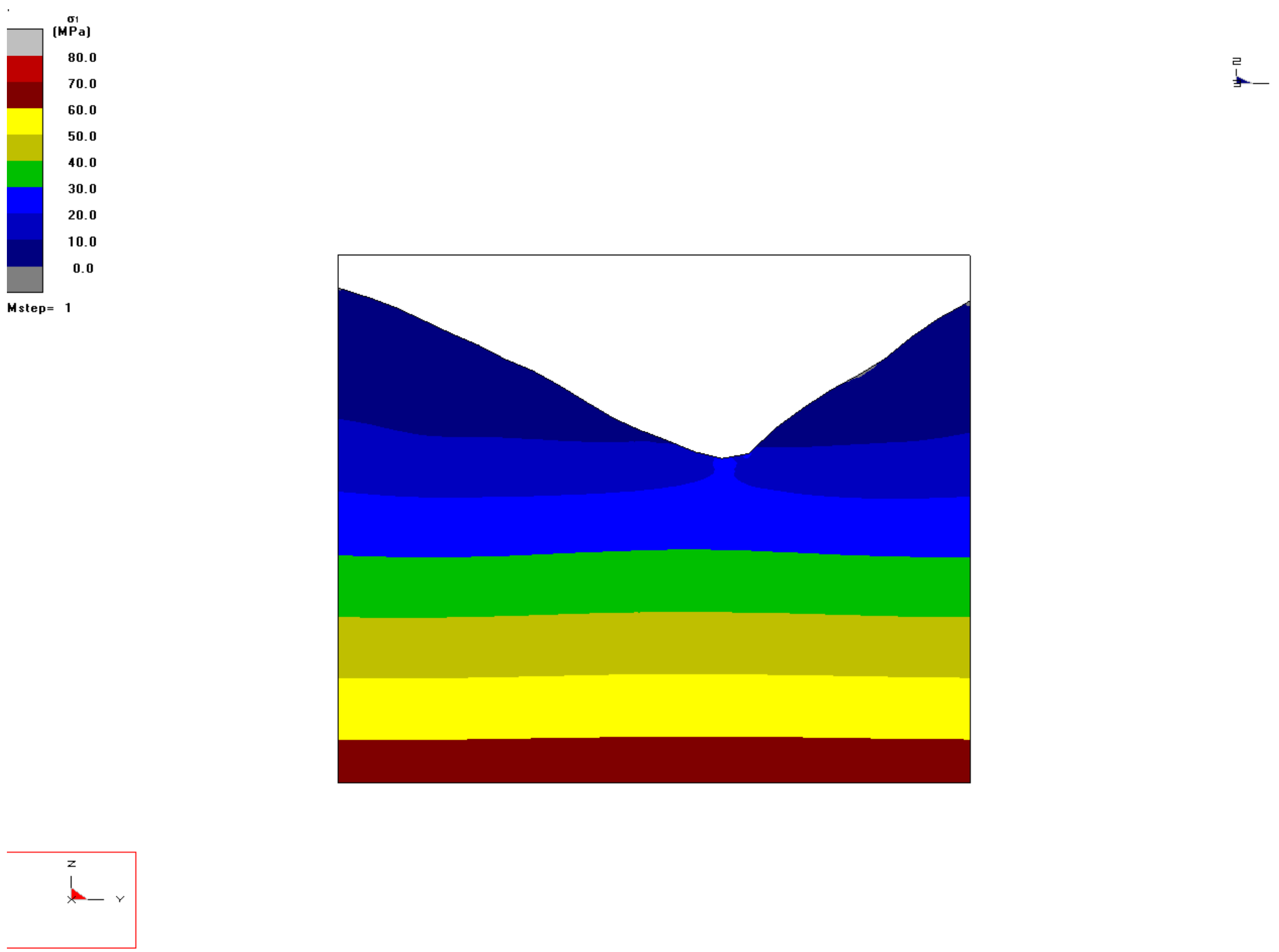
CADD	CY	26JAN12	FIGURE C-9	
CHECK	KMM			
REVIEW	RDH			



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PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE

MITCHELL MAP3D MODEL RESULTS
RESULTS – SIGMA1 – STEP 1

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
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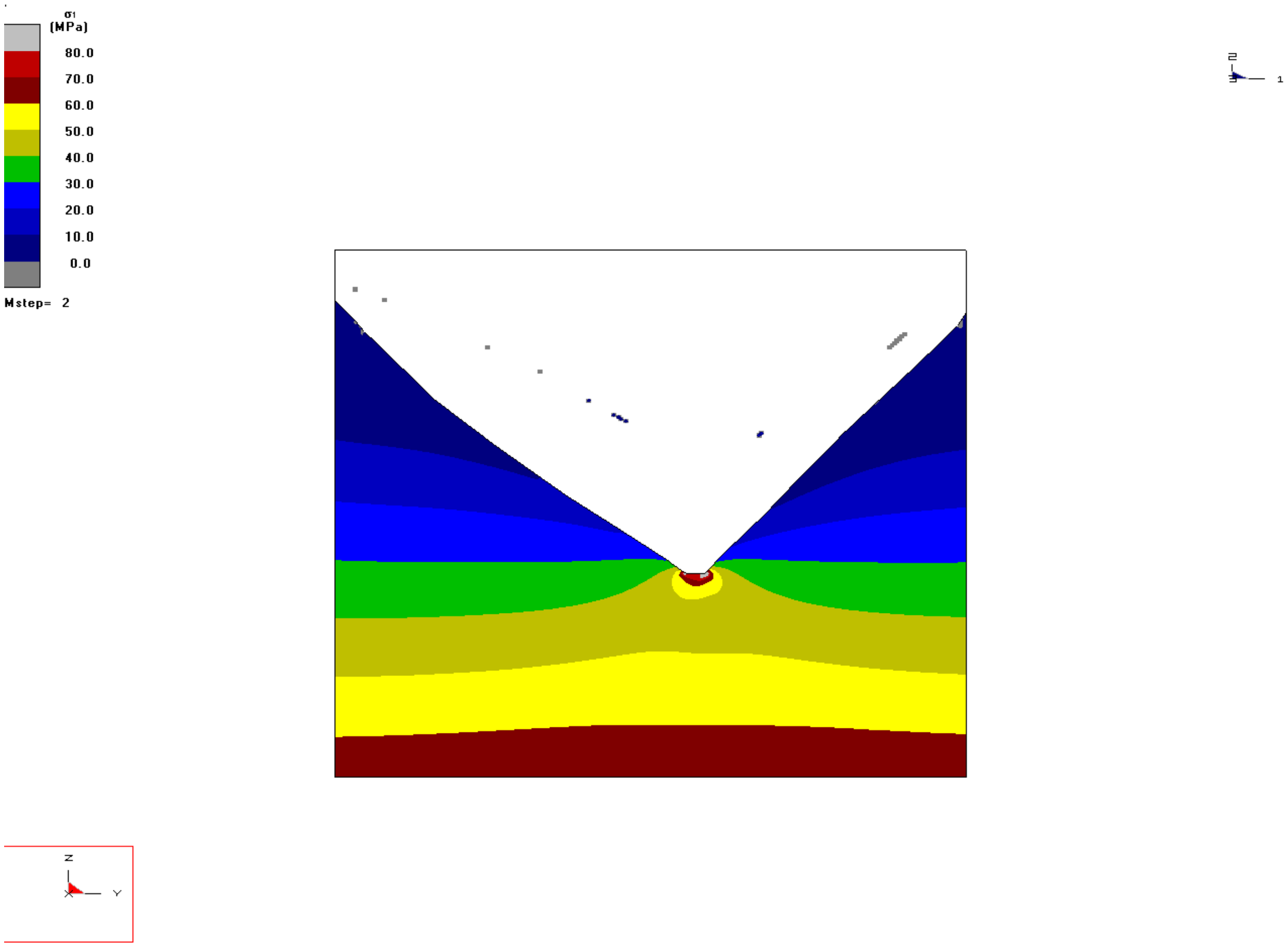
CHECK	KMM			
REVIEW	RDH			



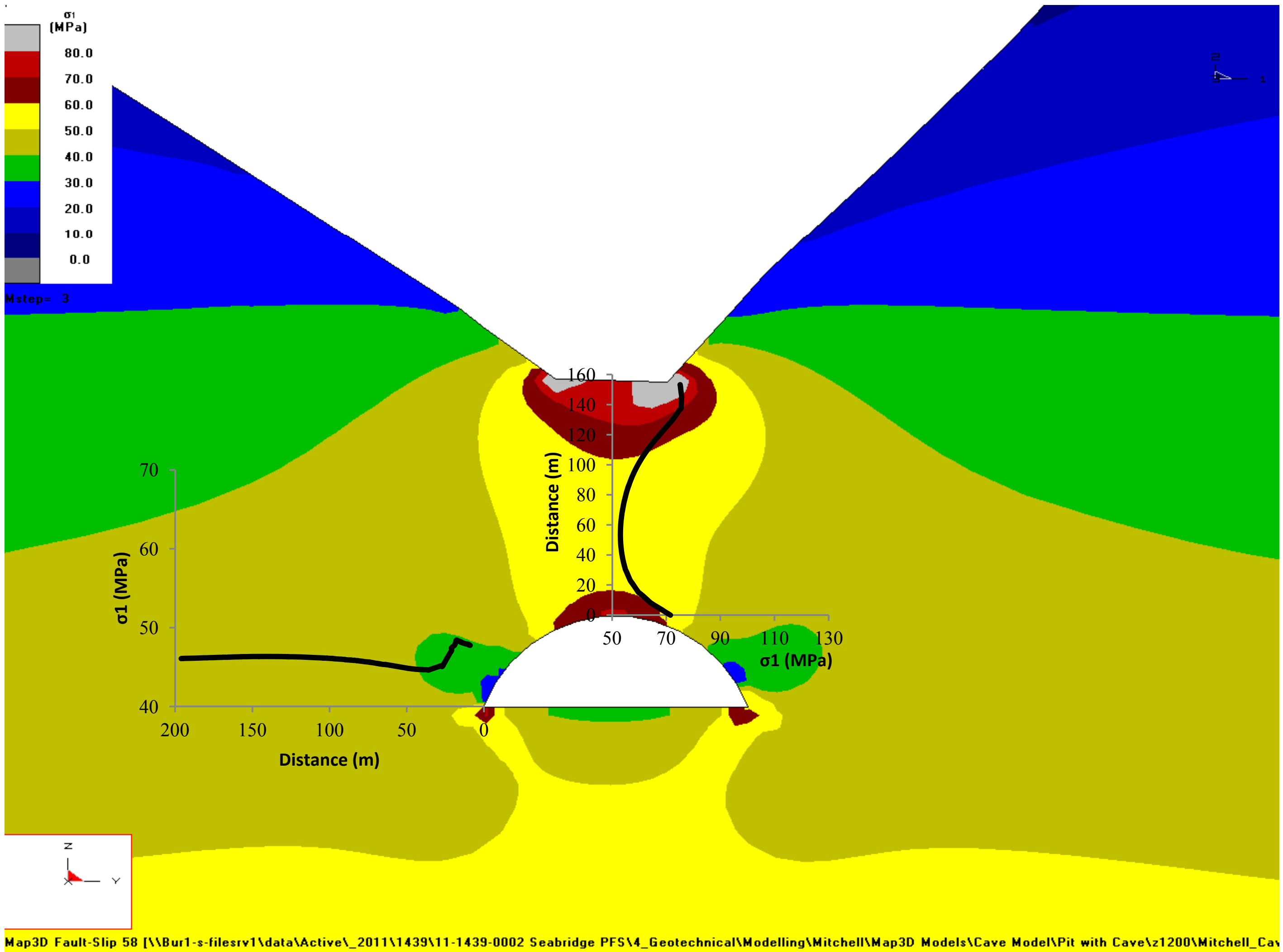
Map3D Fault-Slip 58 [\\Bur1-s-filesrv1\data\Active_2011\1439\11-1439-0002 Seabridge PFS\4_Geotechnical\Modelling\Mitchell\Map3D Models\Cave Model\Pit with Cave\z1200\Mitchell_Cav

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA1 – STEP 2

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	FIGURE C-11	
CHECK	KMM			
REVIEW	RDH			

O:\Active_2011\1439\11-1439-0002 Seabridge PFS\10_Reporting\Mitchell Stress Analysis TM\Appendices



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA1 – STEP 3

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	FIGURE C-12	
CHECK	KMM			
REVIEW	RDH			

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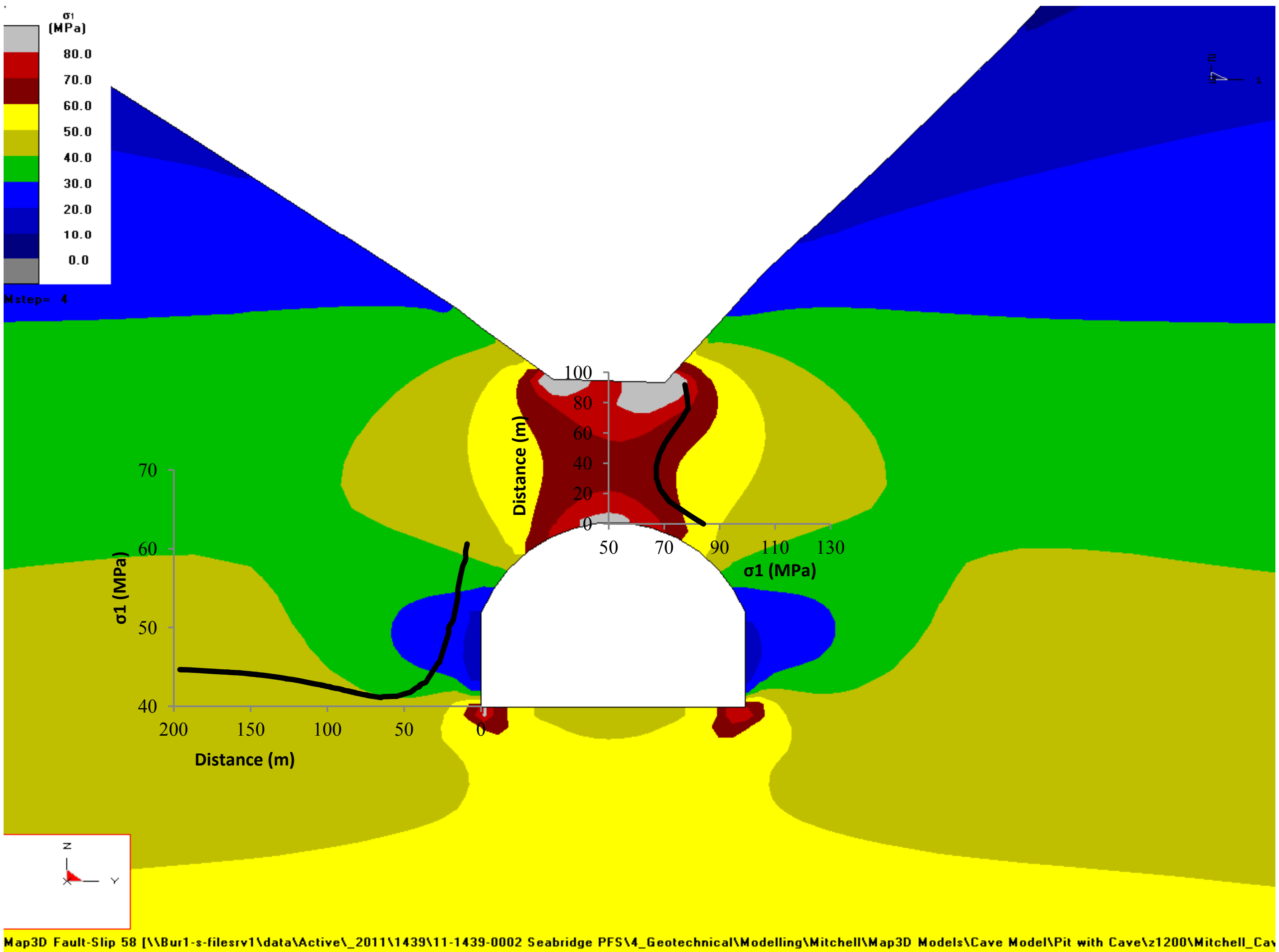

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA1 – STEP 4

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000
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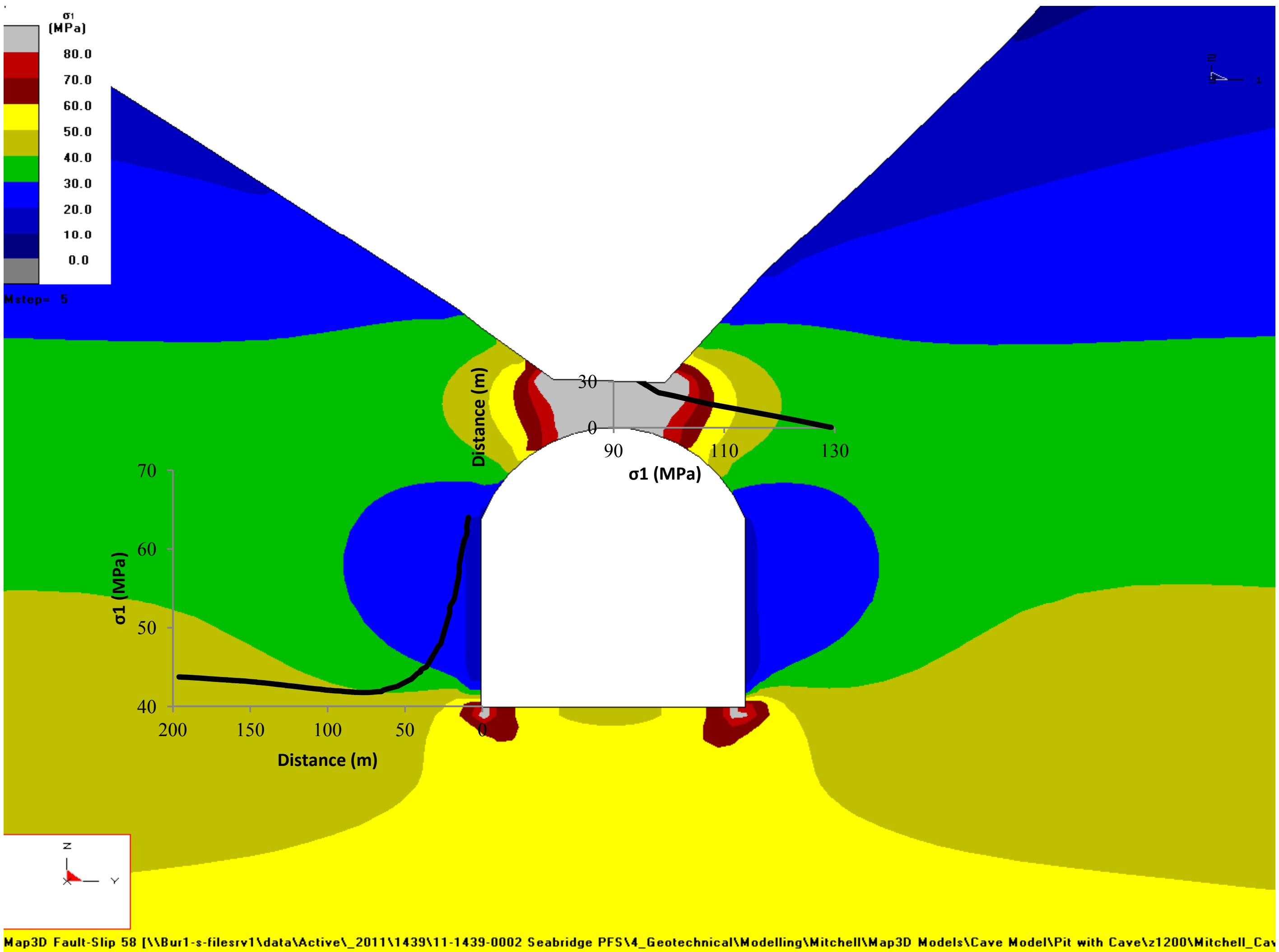
CADD	CY	27JAN12	FIGURE C-13
CHECK	KMM		
REVIEW	RDH		



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA1 – STEP 5

PROJECT No. 11-1439-0002			PHASE No. 10000
DESIGN	CY	27JAN12	SCALE NTS REV. --
CADD	CY	27JAN12	FIGURE C-14
CHECK	KMM		
REVIEW	RDH		

Golder Associates

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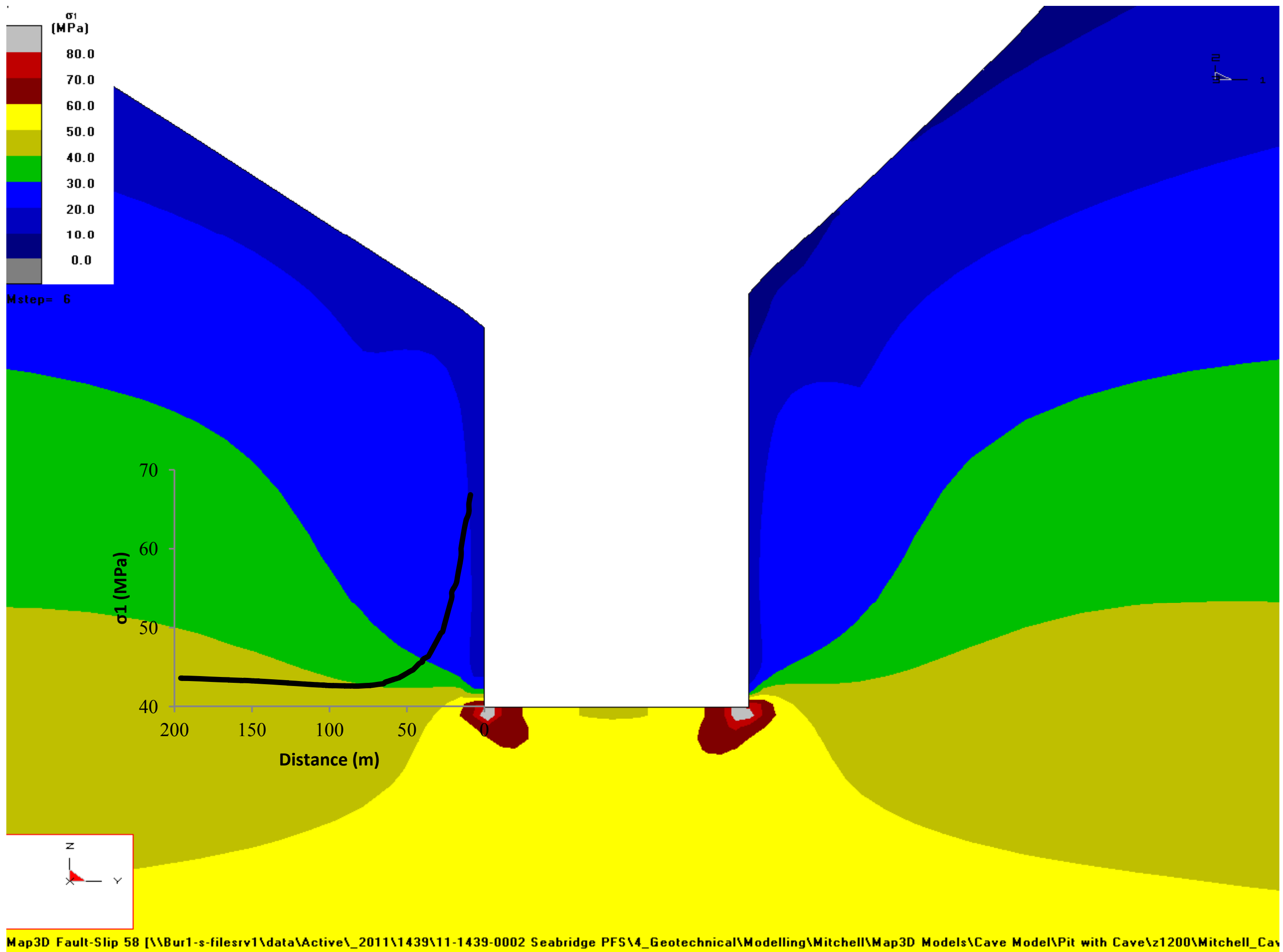


PROJECT

SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

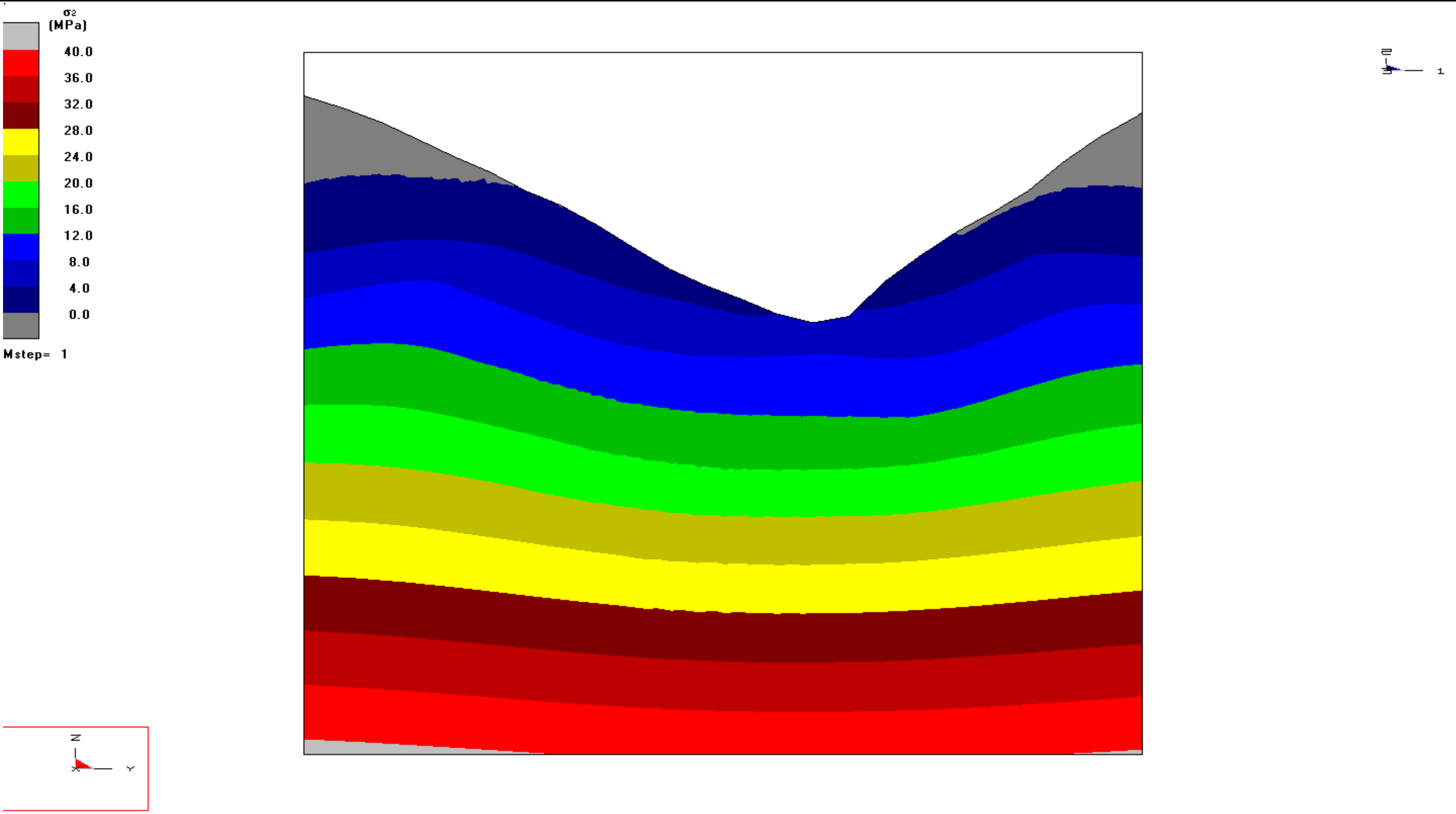
TITLE

MITCHELL MAP3D MODEL RESULTS
RESULTS – SIGMA1 – STEP 6

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000
DESIGN	CY	27JAN12	SCALE NTS REV. --
CADD	CY	27JAN12	
CHECK	KMM		**FIGURE C-15**
REVIEW	RDH		

O:\Active_2011\1439\11-1439-0002 Seabridge PFS\10_Reporting\Mitchell Stress Analysis TM\Appendices



Map3D Fault-Slip 58 [O:\Active_2011\1439\11-1439-0002 Seabridge PFS\4_Geotechnical\Modelling\Mitchell\Map3D Models\Cave Model\Pit with Cave\z1200\Mitchell_Cave_Model_1200m_Pit_Cave_z1200.001][2011

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA			
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA2 – STEP 1			
Golder Associates	PROJECT No. 11-1439-0002			PHASE No. 10000
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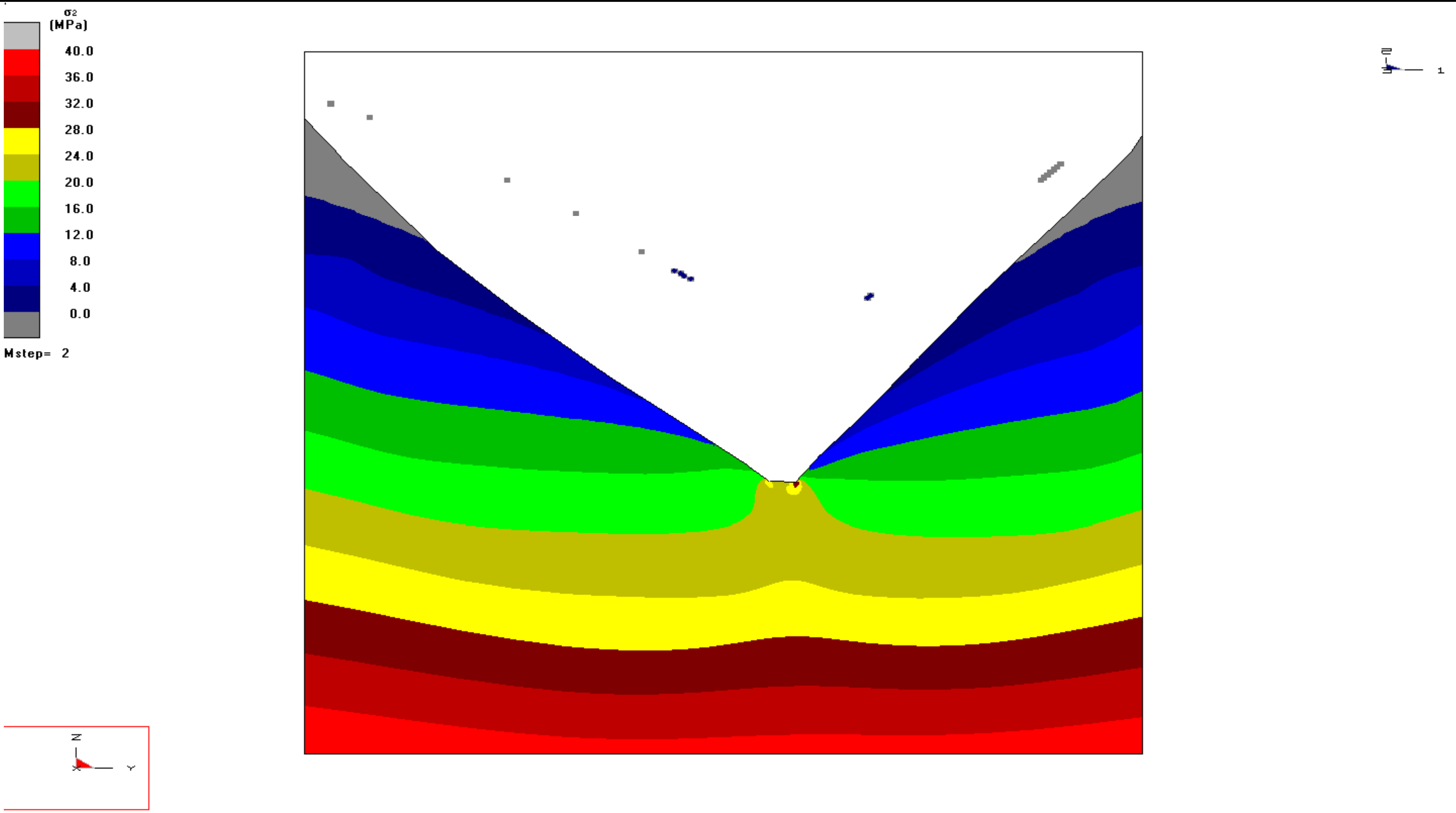
	CADD	CY	27JAN12	FIGURE C-16
	CHECK	KMM		
	REVIEW	RDH		

O:\Active_2011\1439\11-1439-0002 Seabridge PFS\10_Reporting\Mitchell Stress Analysis TM\Appendices



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PROJECT
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE
MITCHELL MAP3D MODEL RESULTS
RESULTS – SIGMA2 – STEP 2

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	FIGURE C-17	
CHECK	KMM			
REVIEW	RDH			

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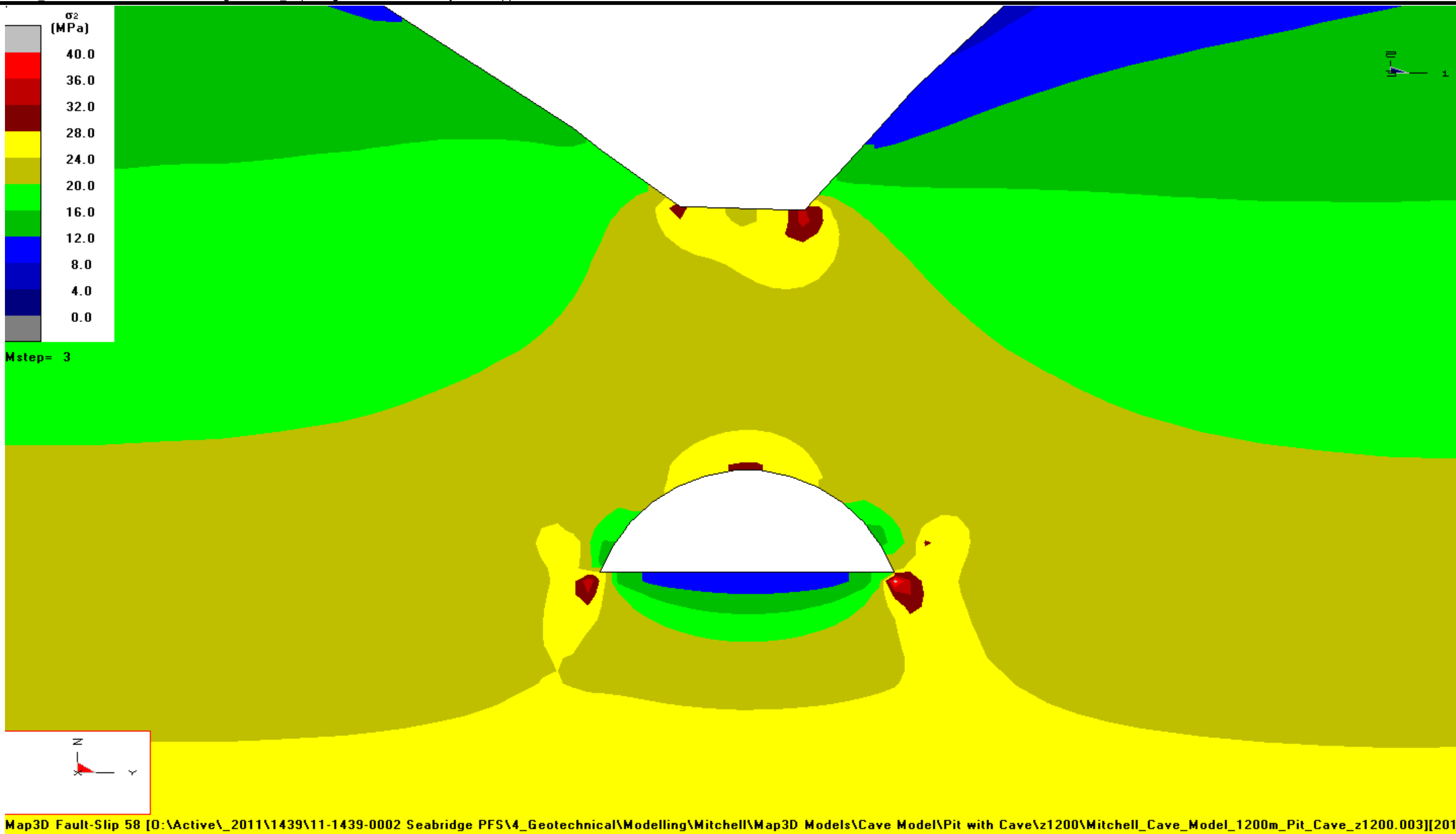


PROJECT

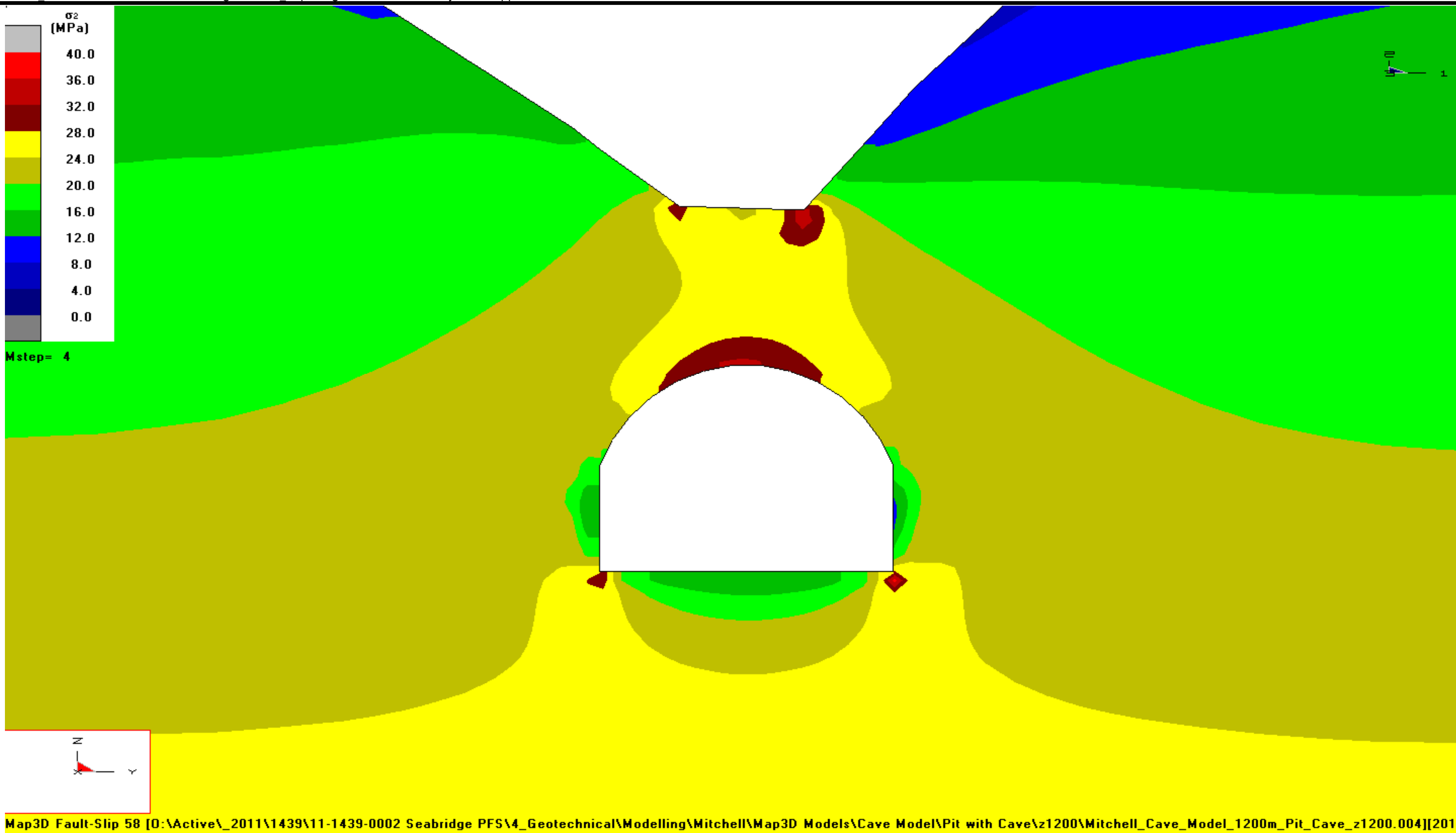
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE

MITCHELL MAP3D MODEL RESULTS
RESULTS – SIGMA2 – STEP 3

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	**FIGURE C-18**	
CHECK	KMM			
REVIEW	RDH			



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA2 – STEP 4

Golder Associates				
PROJECT No.11-1439-0002			PHASE No. 10000	
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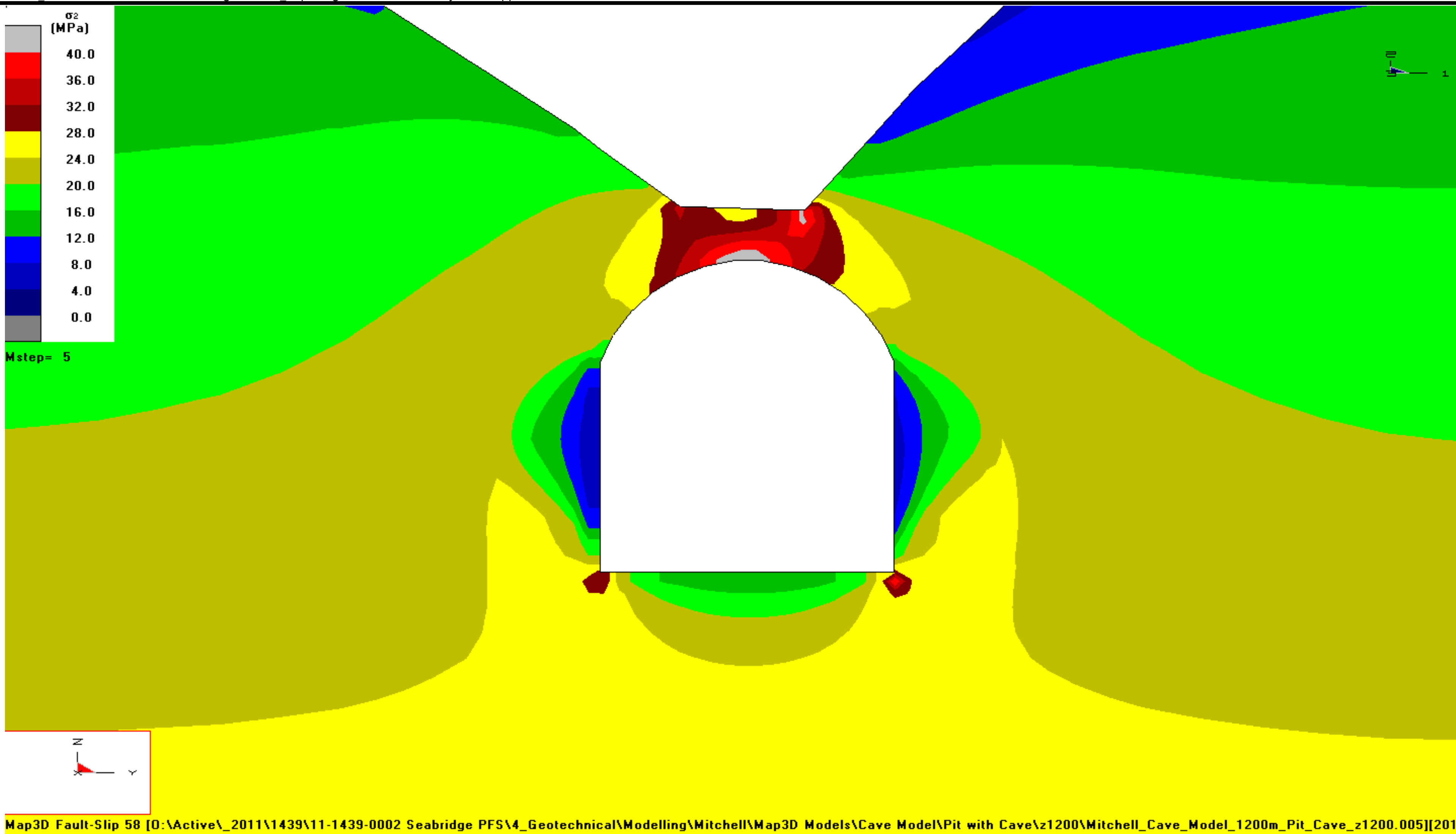
CADD	CY	27JAN12	FIGURE C-19	
CHECK	KMM			
REVIEW	RDH			



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA2 – STEP 5

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	FIGURE C-20	
CHECK	KMM			
REVIEW	RDH			

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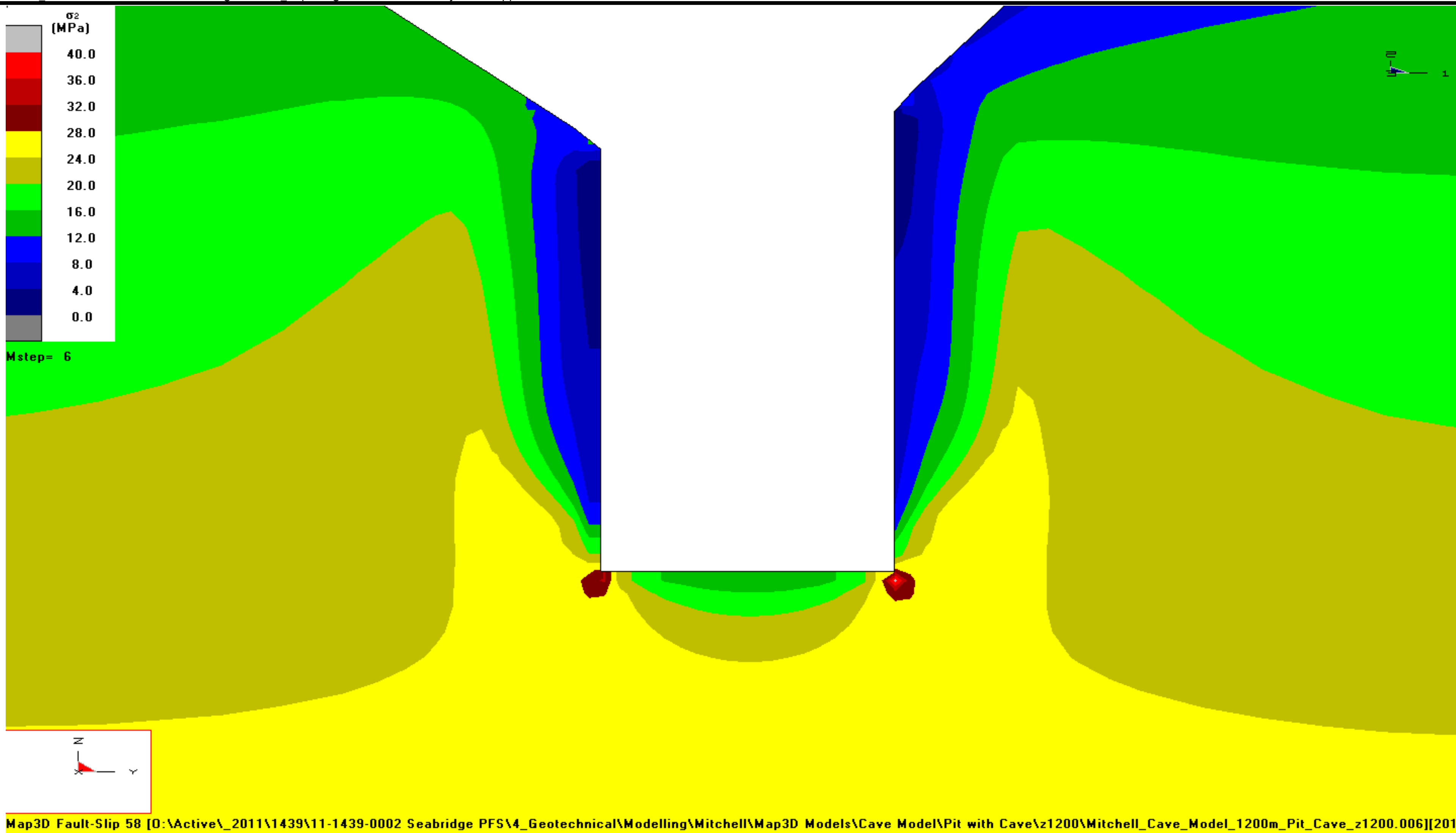


PROJECT

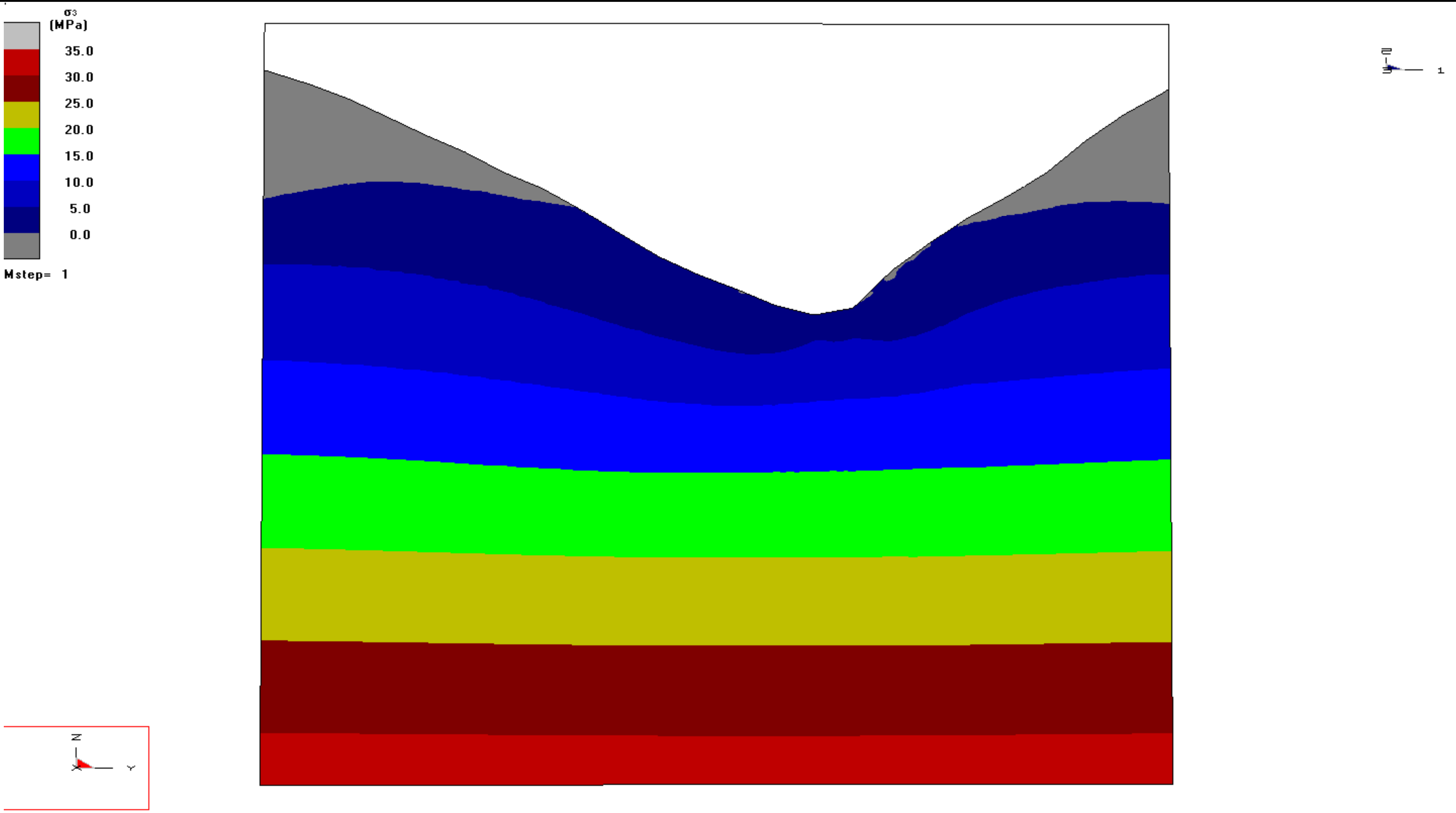
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
BRITISH COLUMBIA

TITLE

MITCHELL MAP3D MODEL RESULTS
RESULTS – SIGMA2 – STEP 6

Golder Associates

PROJECT No. 11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV. --
CADD	CY	27JAN12	**FIGURE C-21**	
CHECK	KMM			
REVIEW	RDH			



Map3D Fault-Slip 58 [O:\Active_2011\1439\11-1439-0002 Seabridge PFS\4_Geotechnical\Modelling\Mitchell\Map3D Models\Cave Model\Pit with Cave\z1200\Mitchell_Cave_Model_1200m_Pit_Cave_z1200.001][2011

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA			
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 1			
Golder Associates	PROJECT No.11-1439-0002			PHASE No. 10000
	DESIGN	CY	27JAN12	SCALE NTS REV.--
	CADD	CY	27JAN12	FIGURE C-22
	CHECK	KMM		
	REVIEW	RDH		

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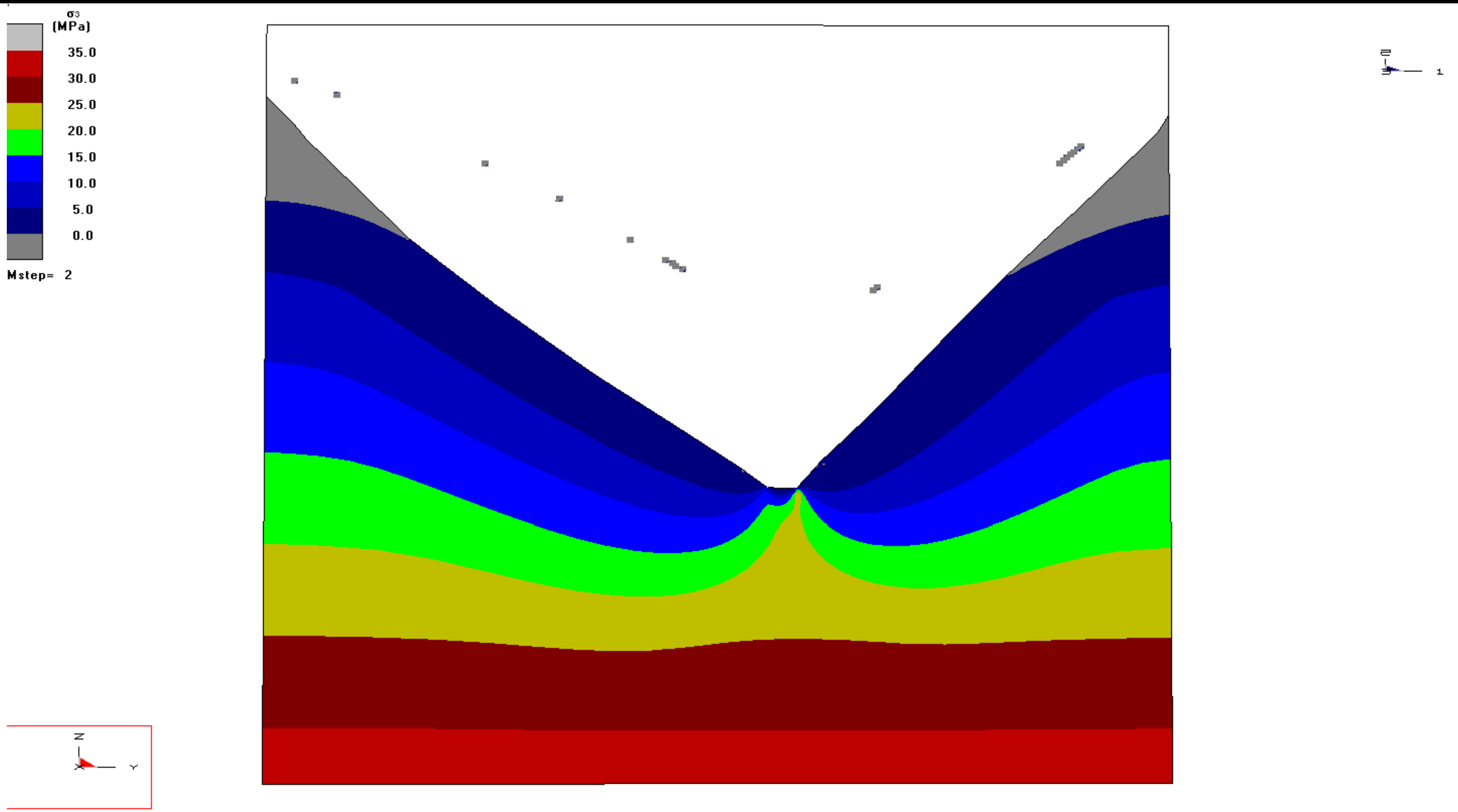
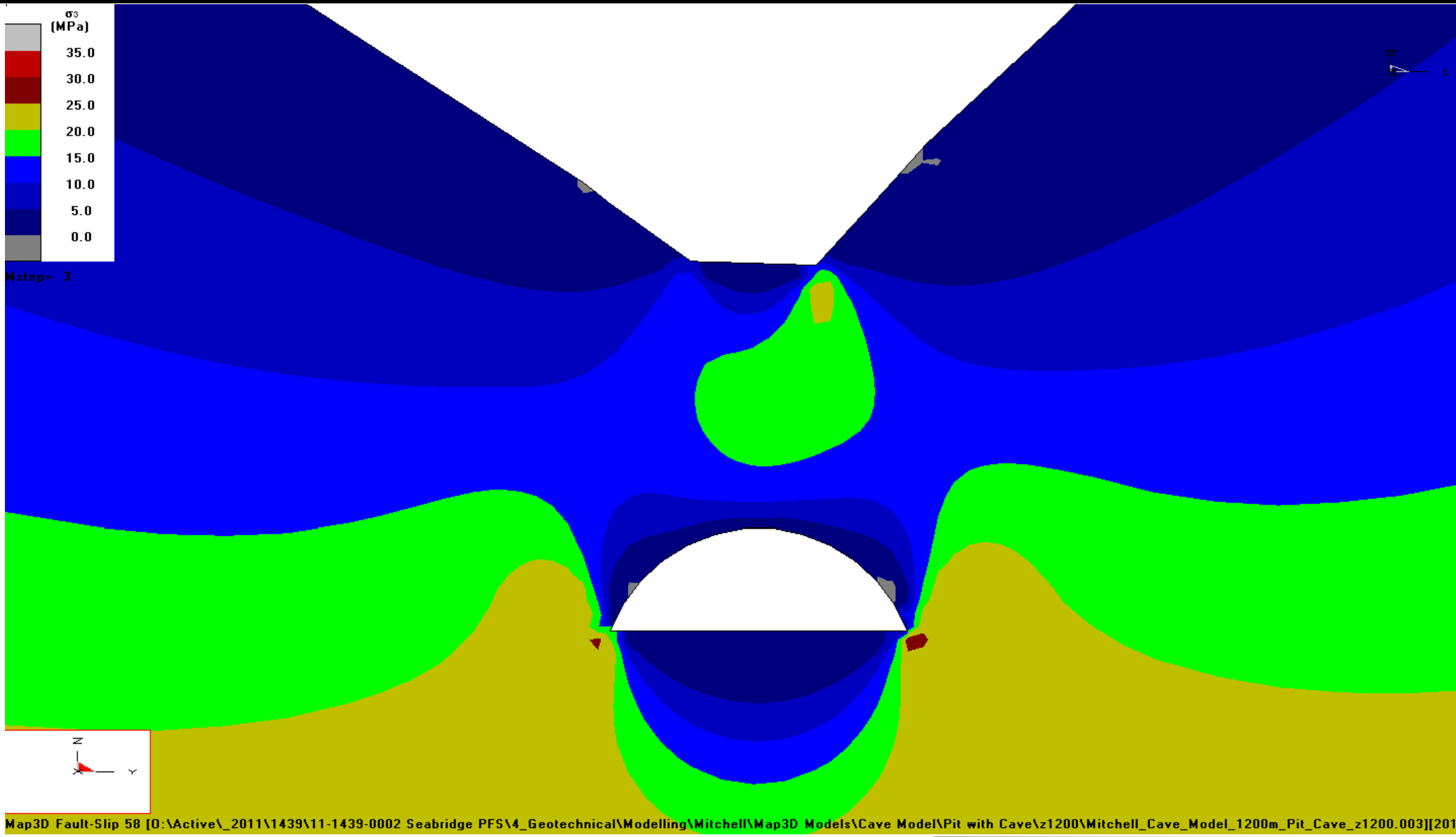


Map3D Fault-Slip 58 [O:\Active_2011\1439\11-1439-0002 Seabridge PFS\4_Geotechnical\Modelling\Mitchell\Map3D Models\Cave Model\Pit with Cave\z1200\Mitchell_Cave_Model_1200m_Pit_Cave_z1200.002][2011

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA			
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 2			
Golder Associates	PROJECT No. 11-1439-0002			PHASE No. 10000
	DESIGN	CY	27JAN12	SCALE NTS REV. --
	CADD	CY	27JAN12	FIGURE C-23
	CHECK	KMM		
	REVIEW	RDH		



PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 3

Golder Associates				
PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV.--
CADD	CY	27JAN12	FIGURE C-24	
CHECK	KMM			
REVIEW	RDH			

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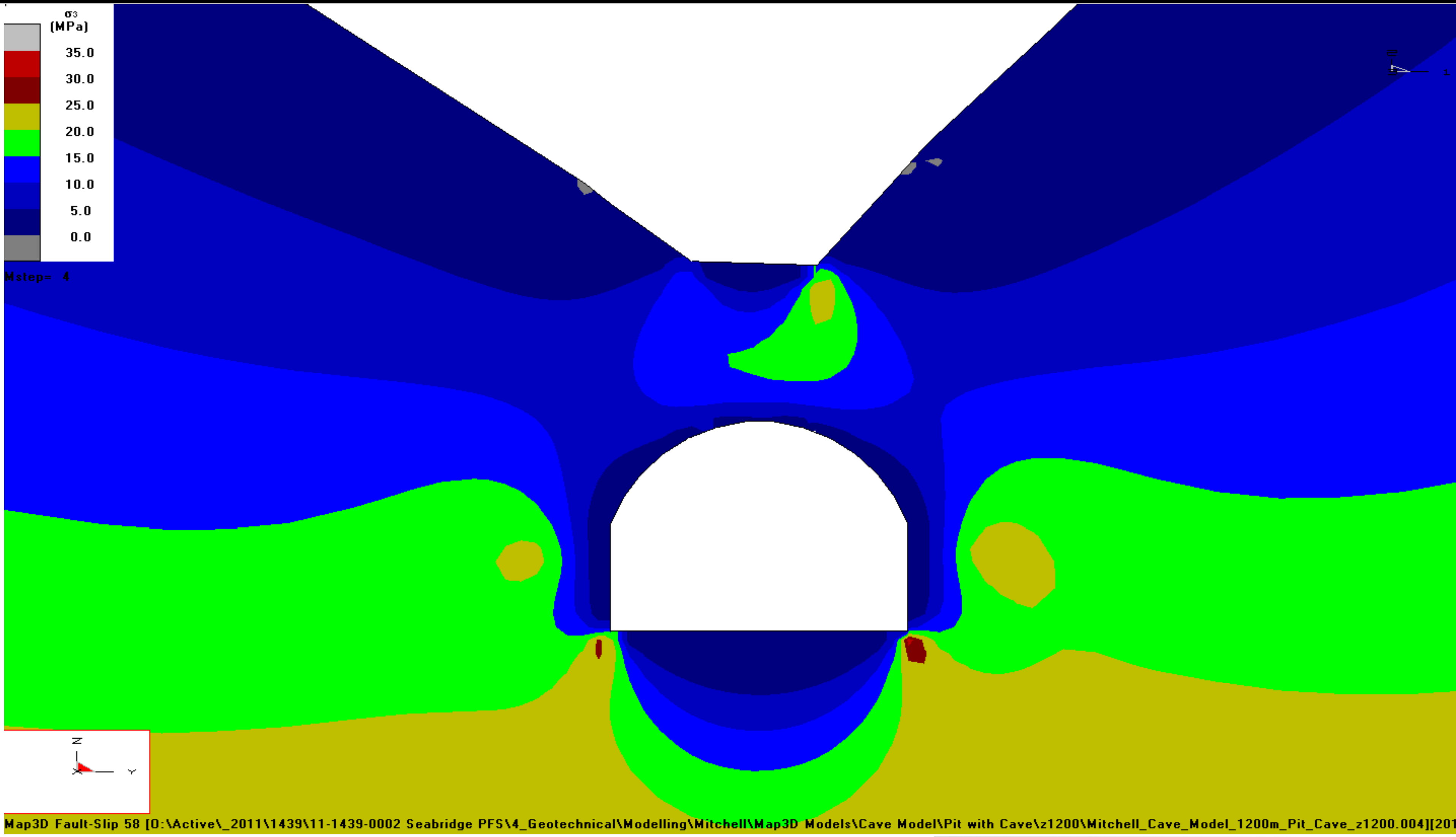

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 4

Golder Associates				
PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV.--
CADD	CY	27JAN12	FIGURE C-25	
CHECK	KMM			
REVIEW	RDH			

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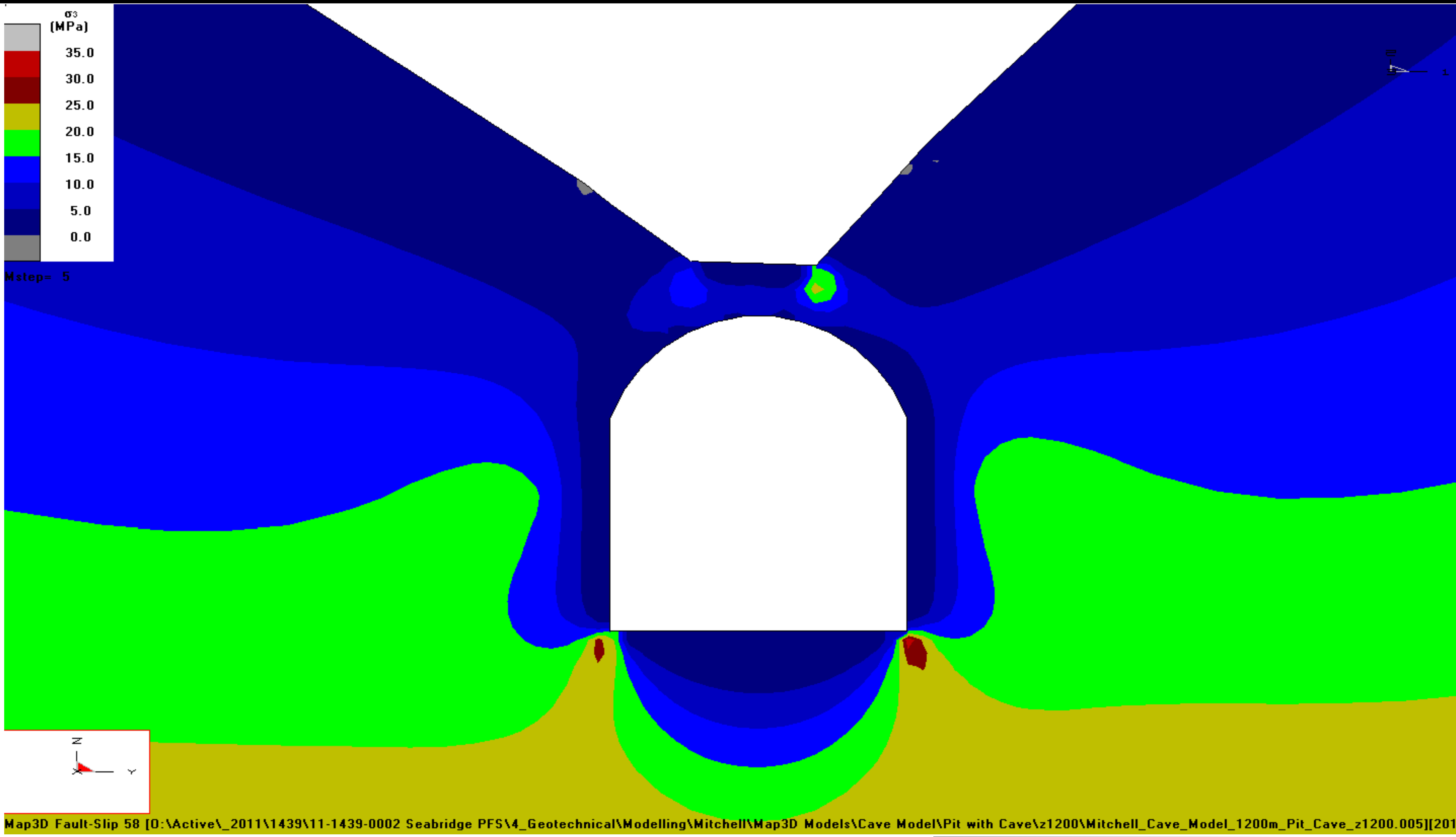

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 5

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
DESIGN	CY	27JAN12	SCALE NTS	REV.--
CADD	CY	27JAN12	FIGURE C-26	
CHECK	KMM			
REVIEW	RDH			

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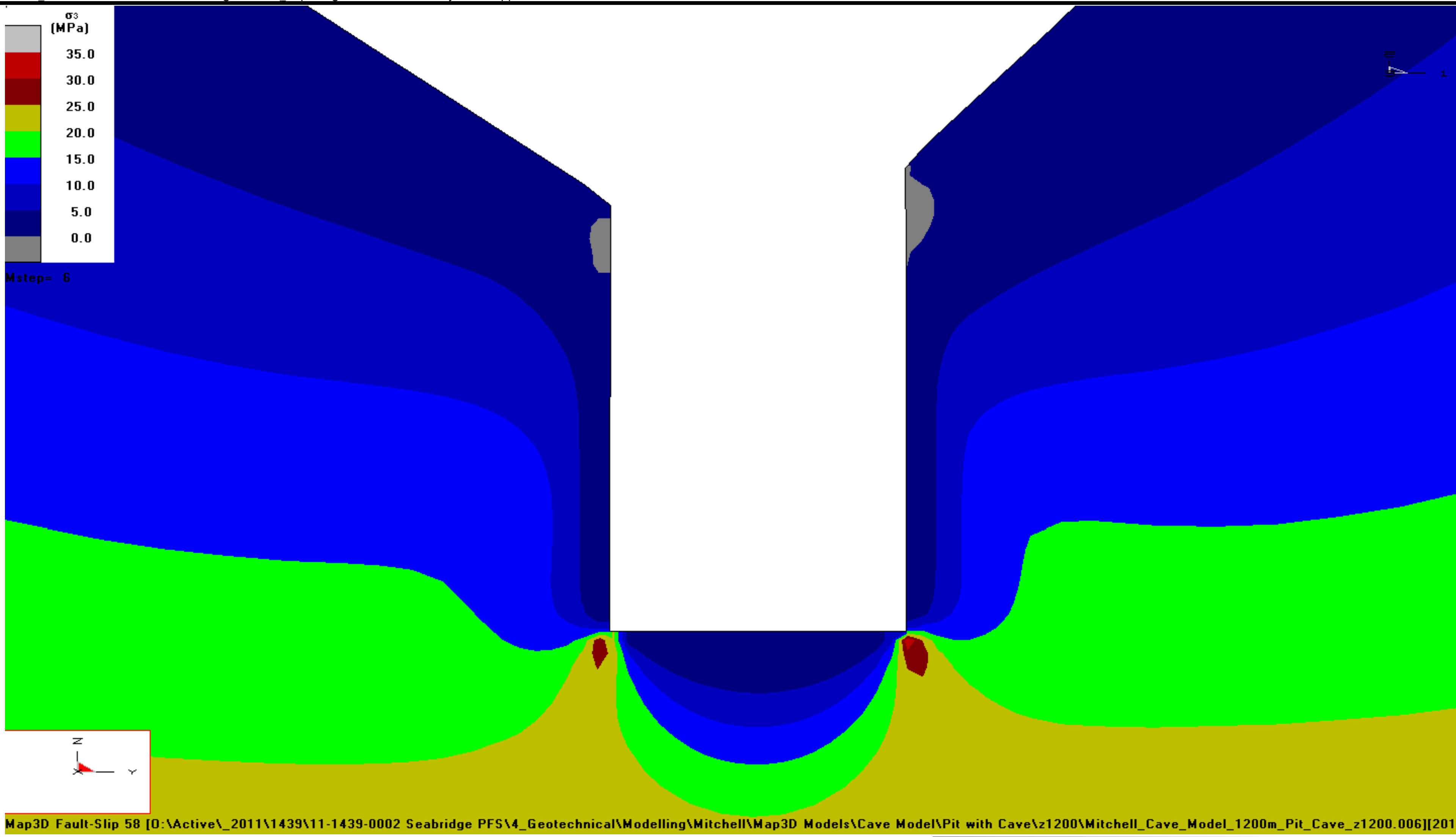
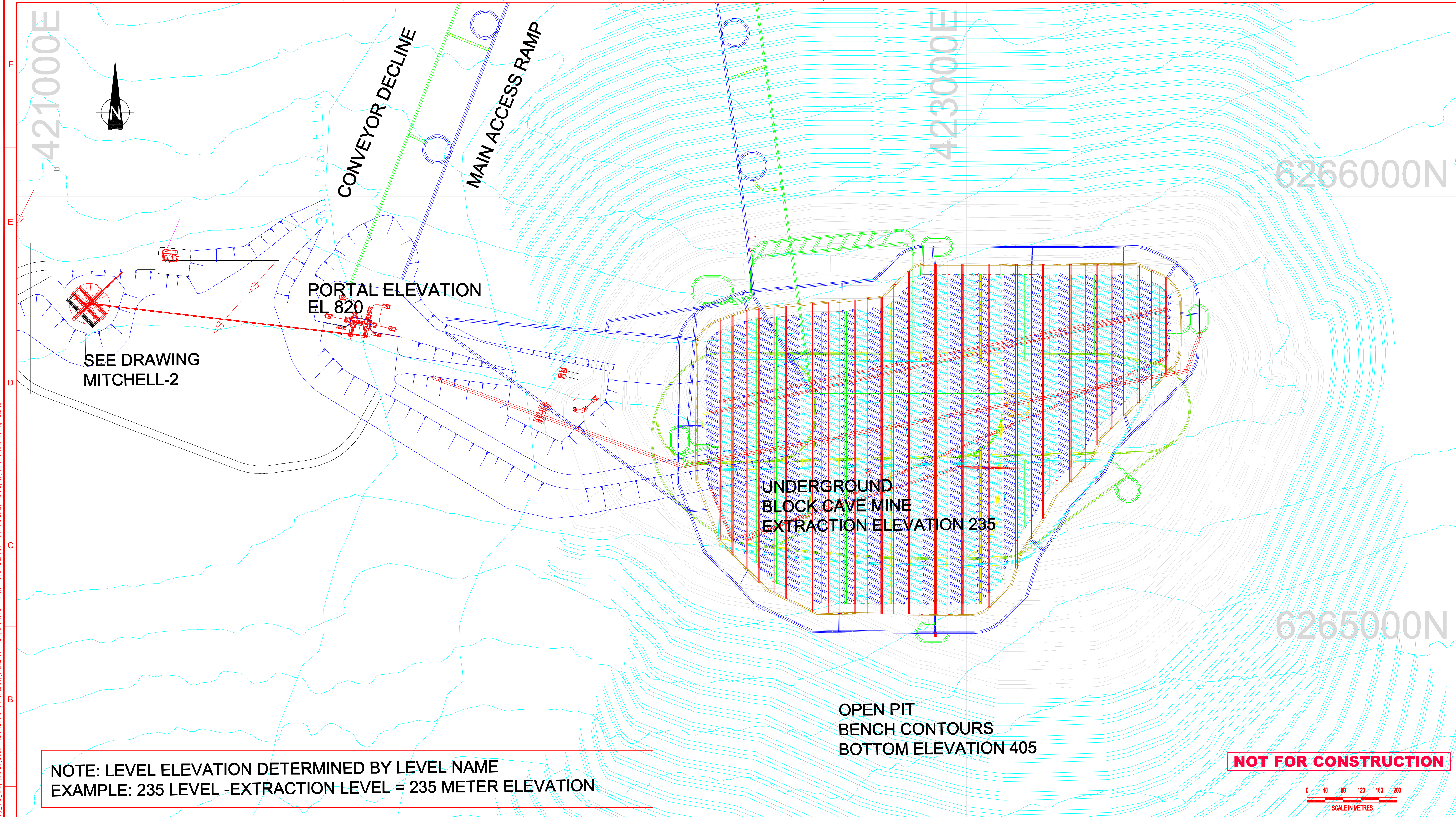


PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY BRITISH COLUMBIA			
TITLE	MITCHELL MAP3D MODEL RESULTS RESULTS – SIGMA3 – STEP 6			
Golder Associates	PROJECT No.11-1439-0002			PHASE No. 10000
	DESIGN	CY	27JAN12	SCALE NTS REV.--
	CADD	CY	27JAN12	FIGURE C-27
	CHECK	KMM		
	REVIEW	RDH		

APPENDIX D

Detailed Level Drawings



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REVISIONS

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

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT
SEABRIDGE GOLD INC

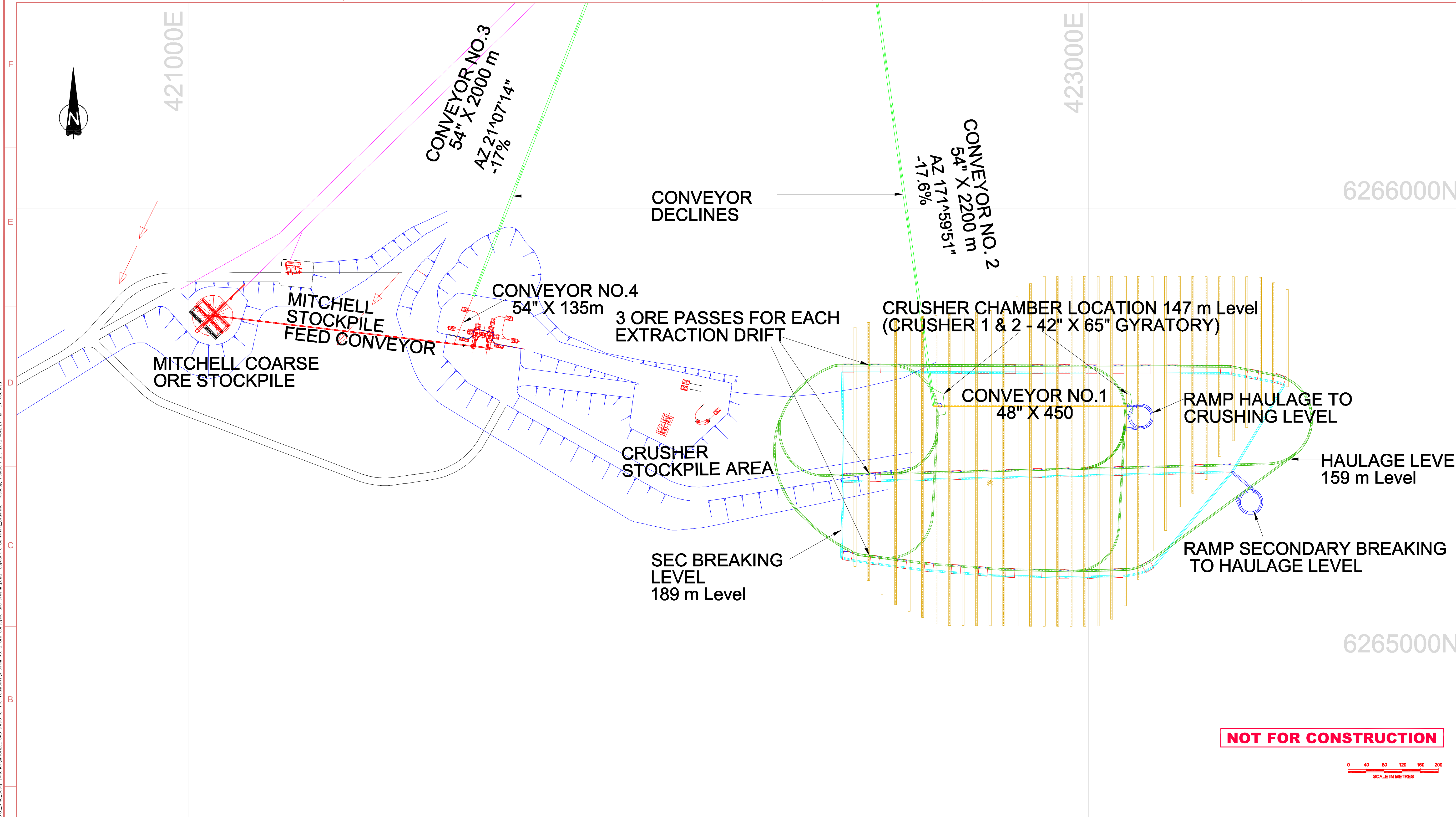
CONSULTANT
Golder Associates

PROJECT
MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE
MITCHELL COMPOSITE PLAN

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-1	1



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

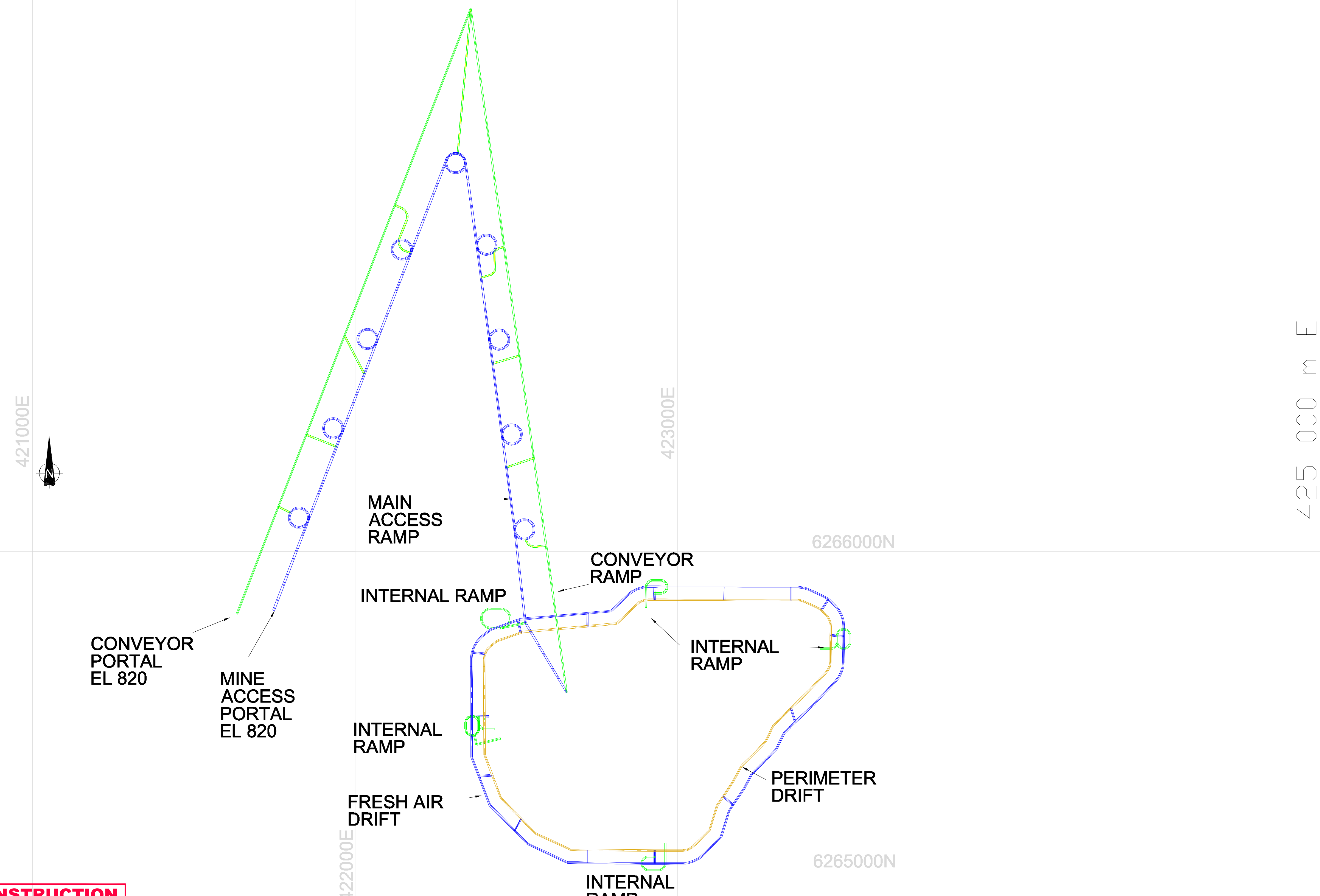
CONSULTANT: Golder Associates

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: ORE CONVEYING AND CRUSHING

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-2	1



NOT FOR CONSTRUCTION

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	MITCHELL MINE ACCESS	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

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PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: **MITCHELL MINE ACCESS**

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

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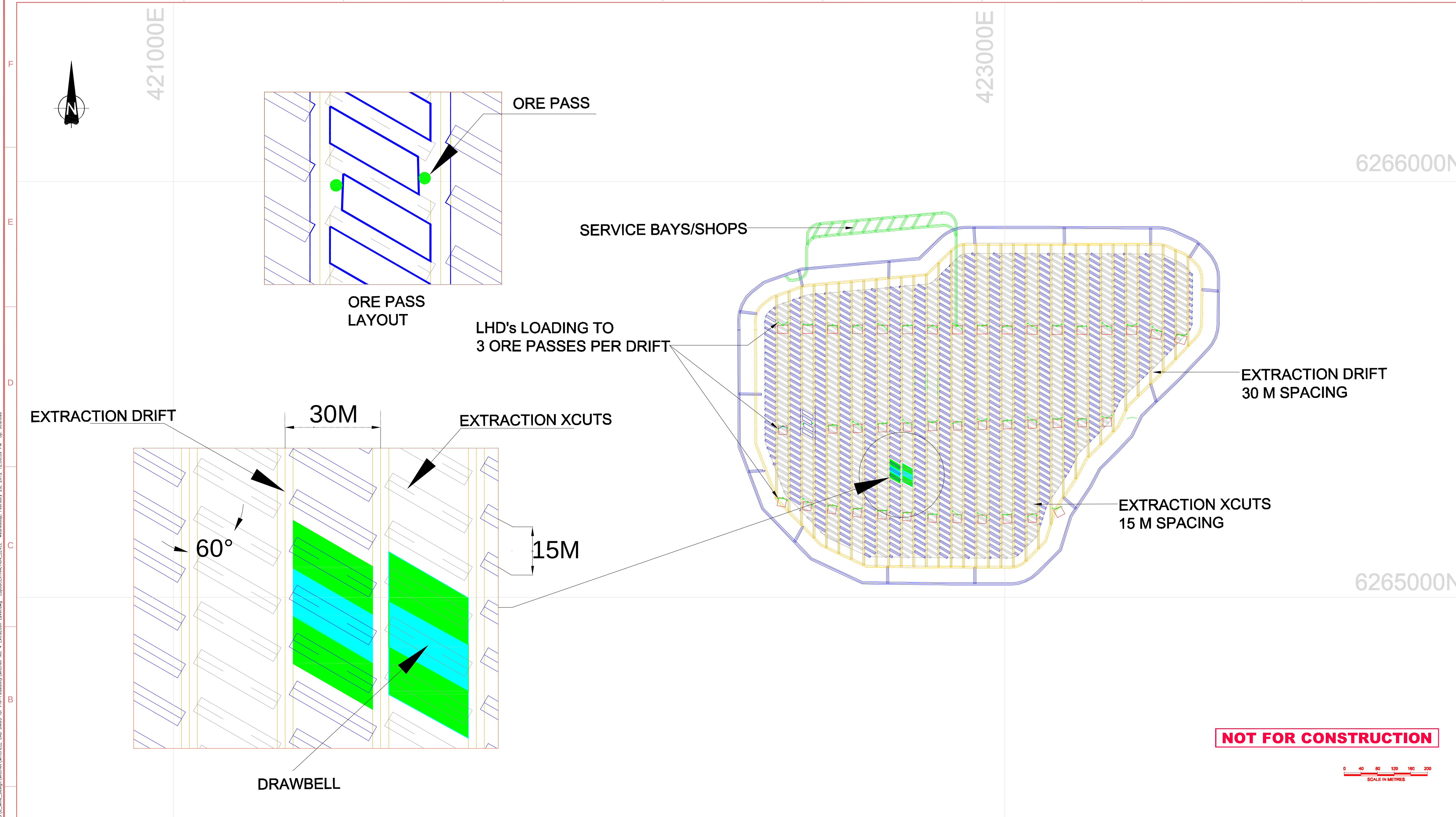

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	EXTRACTION LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: EXTRACTION LEVEL 235 LEVEL

PROJECT No.	PHASE No.	TASK No.
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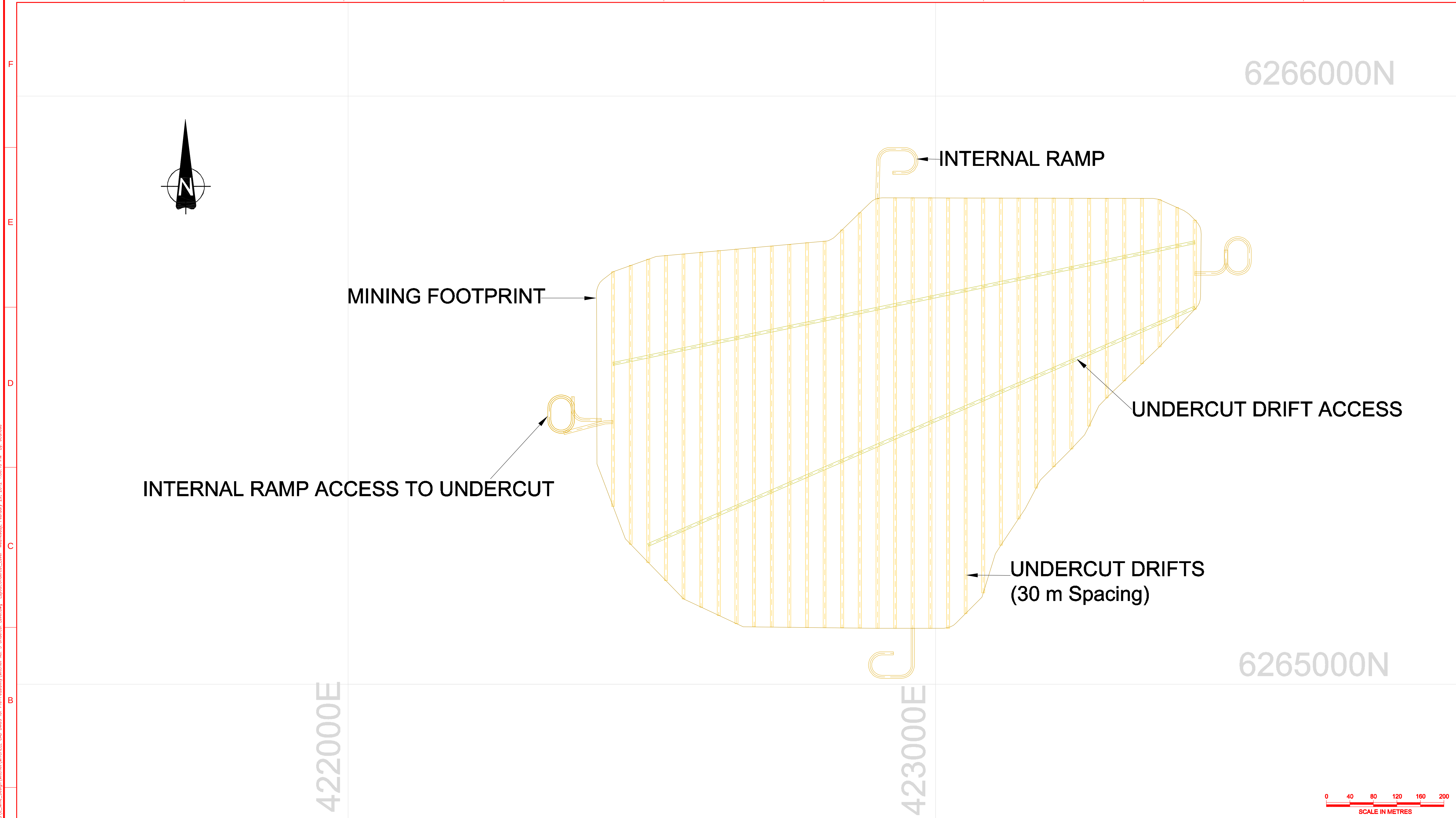

CLIENT	PROJECT
SEABRIDGE GOLD INC	MITCHELL BLOCK CAVE MINE DESIGN
CONSULTANT: Golder Associates	DRAWING TITLE: UNDERCUT LEVEL 255 LEVEL

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

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

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REVISIONS

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SPECIFICATIONS

REFERENCE TITLE

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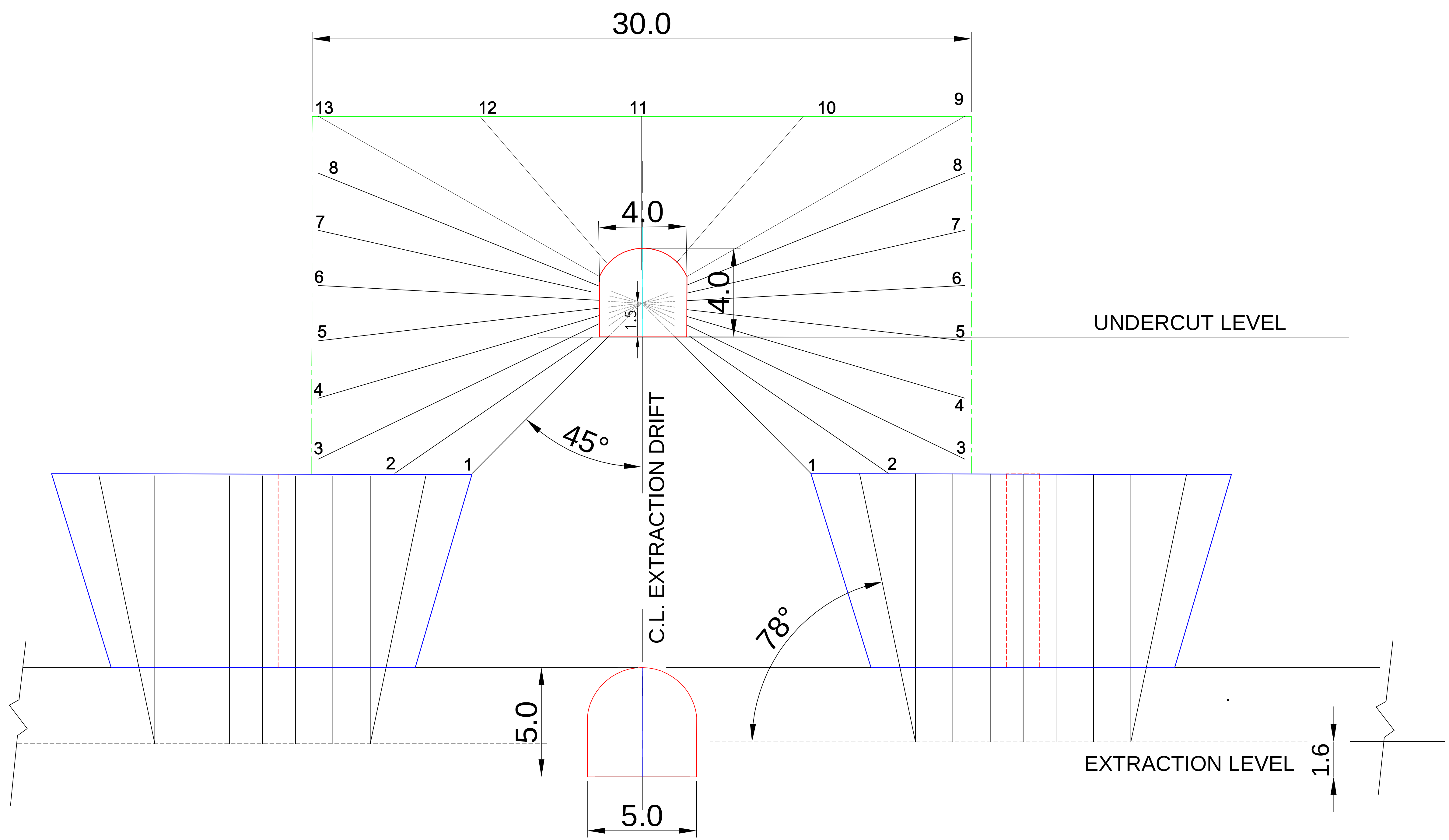

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	DRILLING DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: **TYPICAL UNDERCUT AND DRAWBELL DRILLING LAYOUTS**

PROJECT No.	PHASE No.	TASK No.
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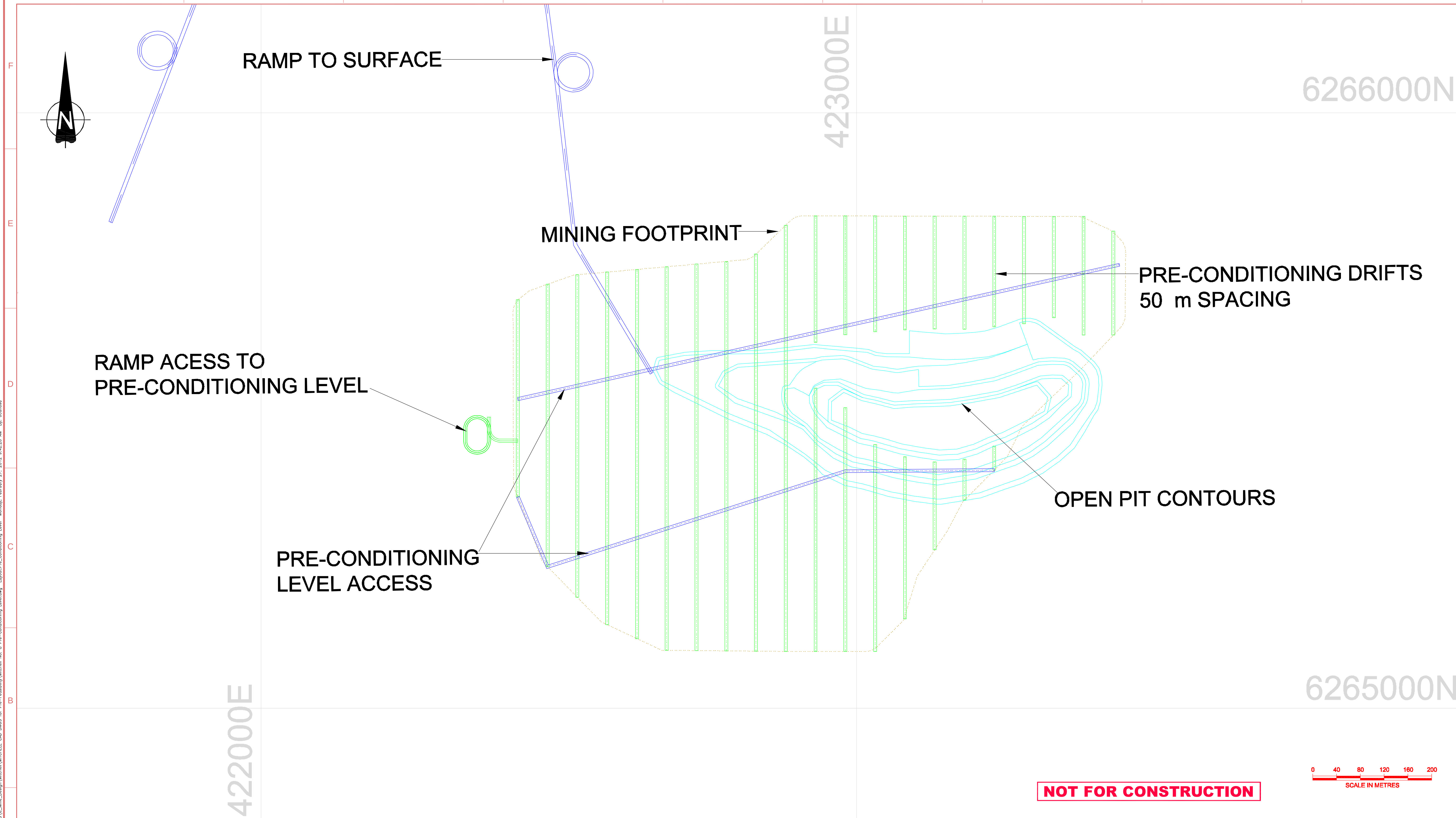

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	PRE-CONDITIONING LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: PRE-CONDITIONING LEVEL 305 LEVEL

PROJECT No.	PHASE No.	TASK No.
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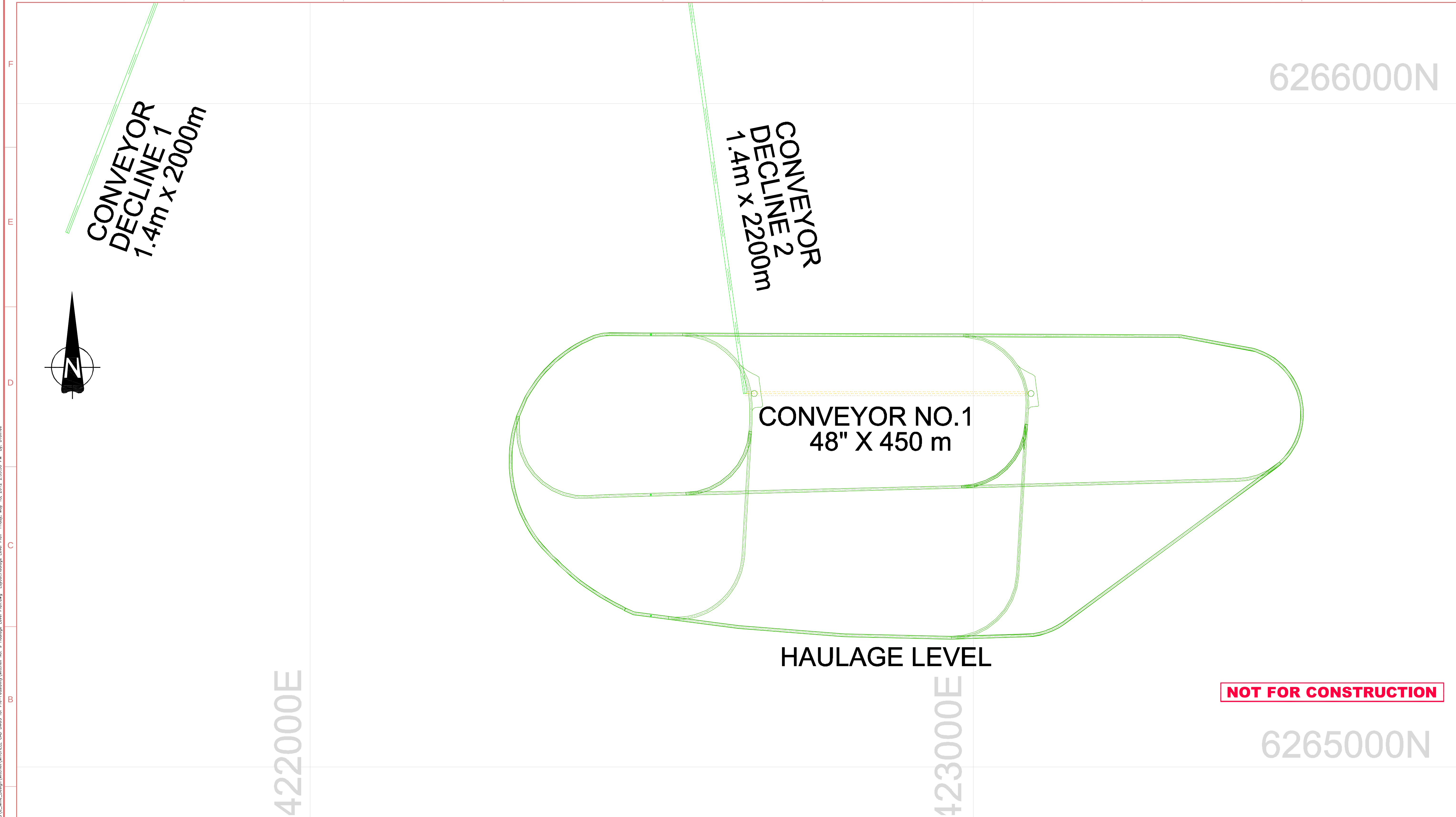

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	MITCHELL HAULAGE LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

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PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: HAULAGE LEVEL
159 LEVEL

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-9	1

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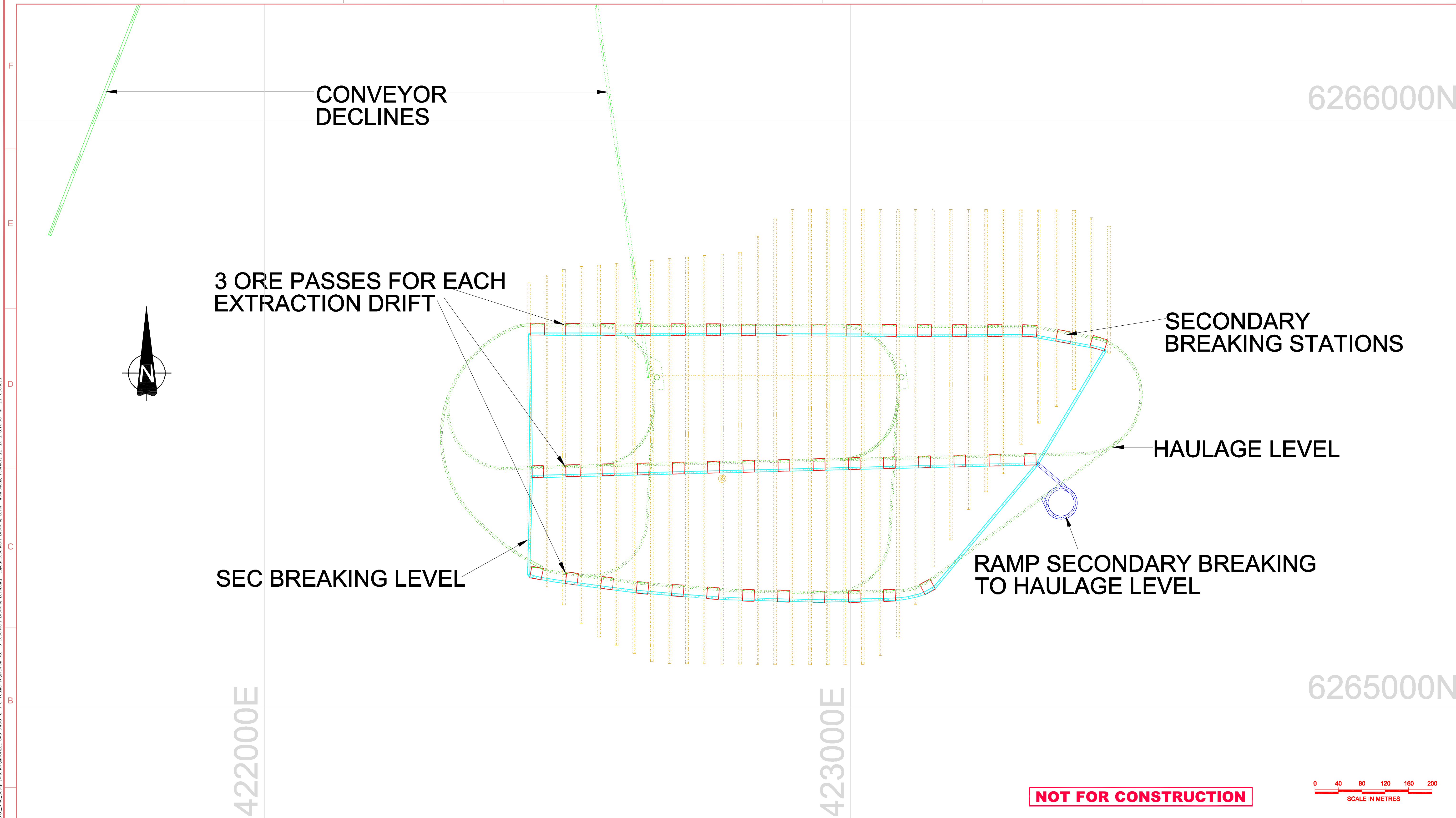

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	SECONDARY BREAKING LEVEL	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: SECONDARY BREAKING LEVEL
189 LEVEL

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

ORIGINAL SHEET SIZE: ANSI D | DRAWING No. MITCHELL-10 | REVISION 1

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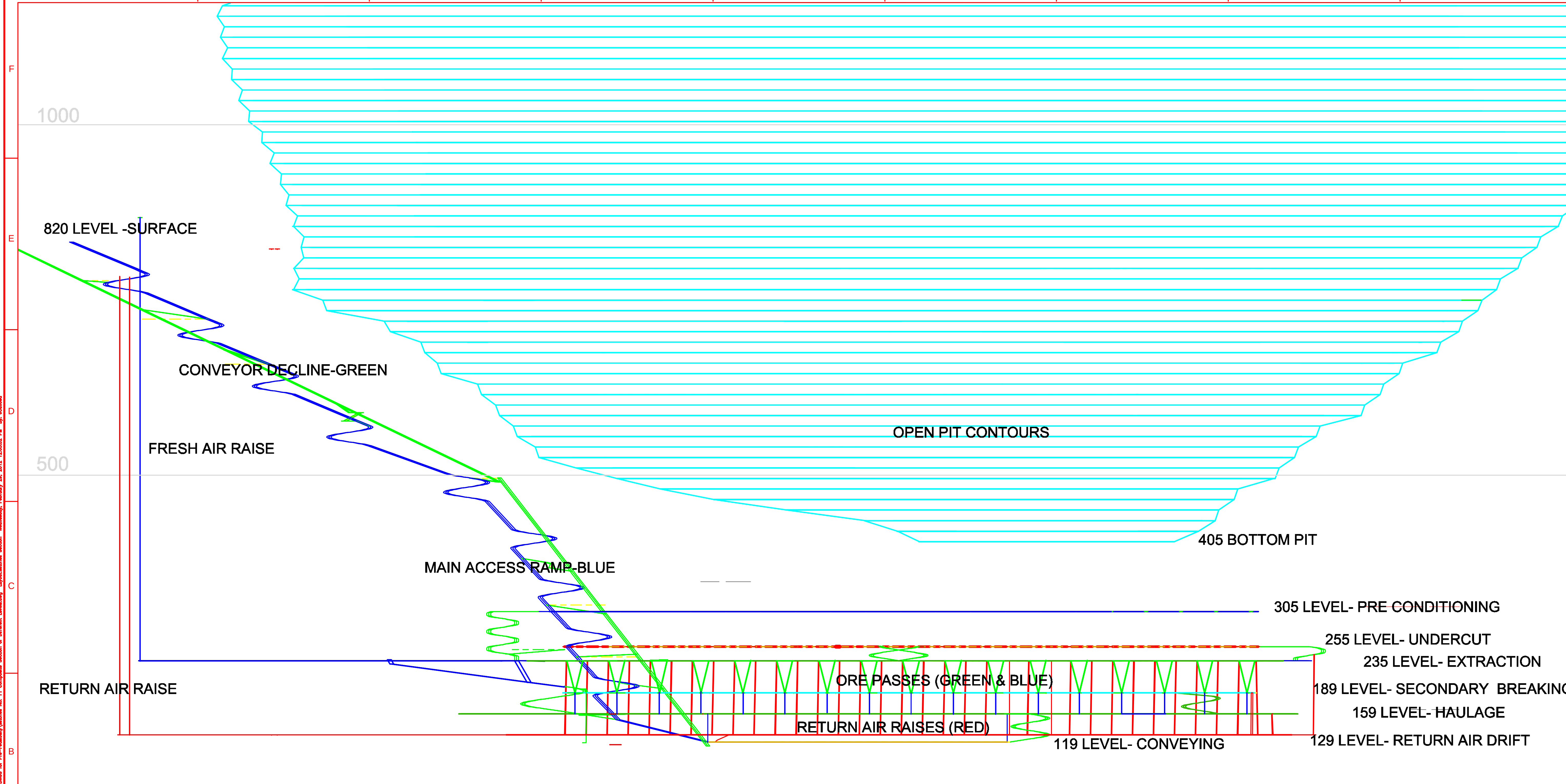

NOTE: LEVEL ELEVATION DETERMINED BY LEVEL NAME
EXAMPLE: 305 LEVEL- PRE CONDITIONING LEVEL = 305 METER ELEVATION

NOT FOR CONSTRUCTION

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REVISIONS

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

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

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PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: MITCHELL COMPOSITE SECTION

PROJECT No.	PHASE No.	TASK No.
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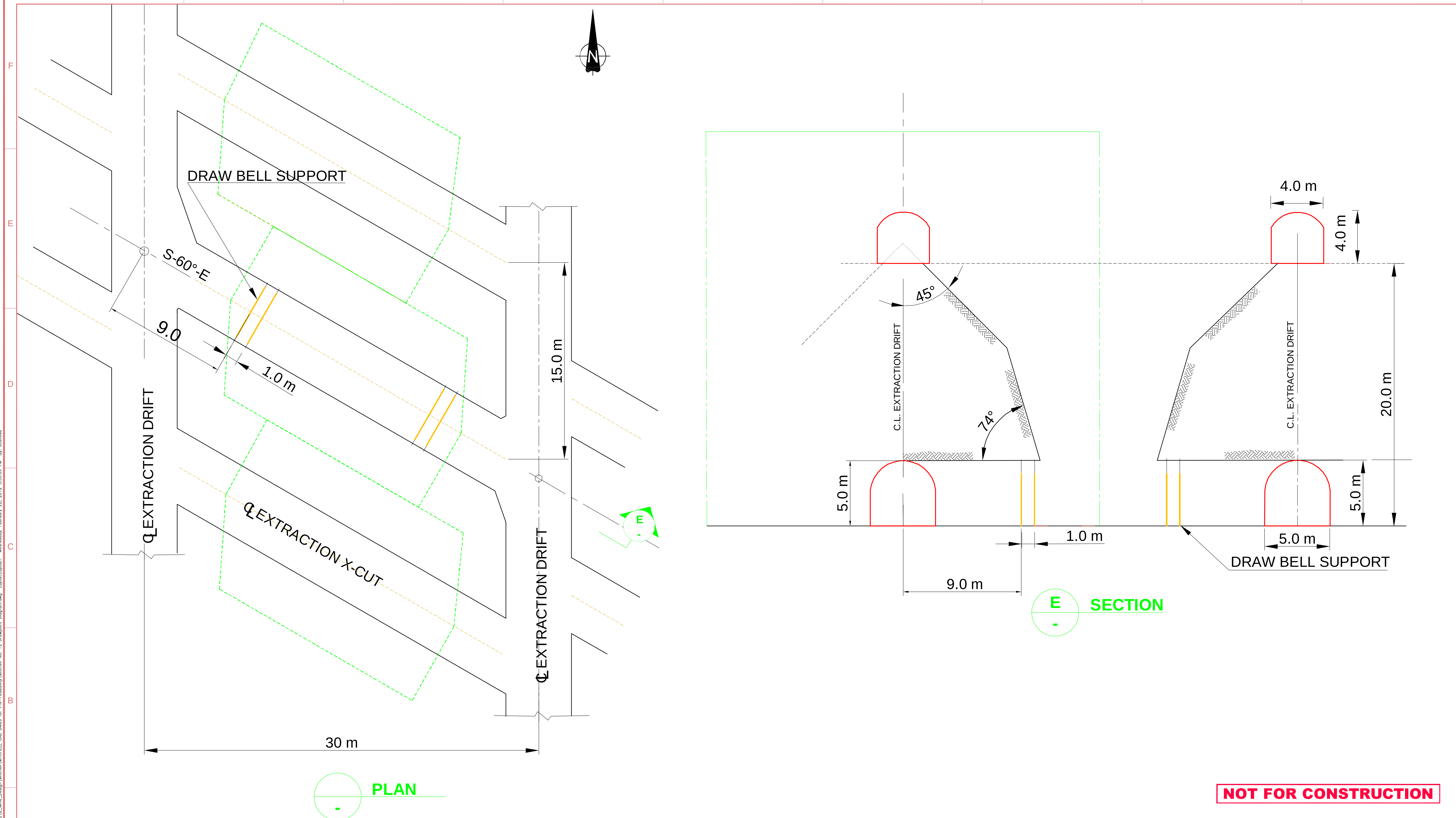

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REFERENCE TITLE
REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

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PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: **DRAWPOINT PLAN AND SECTION**

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-12	0

APPENDIX E

Definition of Block Cave Terms

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.

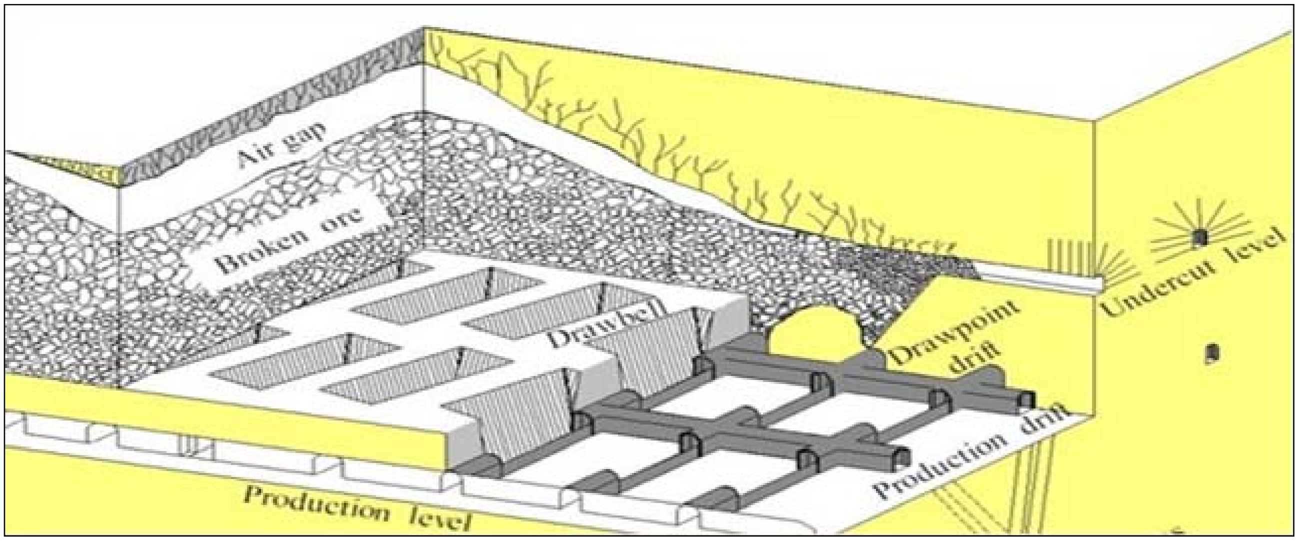

Figure 1: A schematic showing the relationship between the extraction drift (production drift), the drawpoints, drawbells and the undercut level. (Flores, 2004).

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.
Intermediate Geotechnical Engineer

David Sprott, P.Eng.
Associate, Senior Mining Engineer

DT/DS/aw

\\bur1-s-filesrv2\final\2011\1439\11-1439-0002\1114390002-005-r-rev0\appendix e - block cave definitions\block caving definitions.docx

APPENDIX F

Geotech - G3

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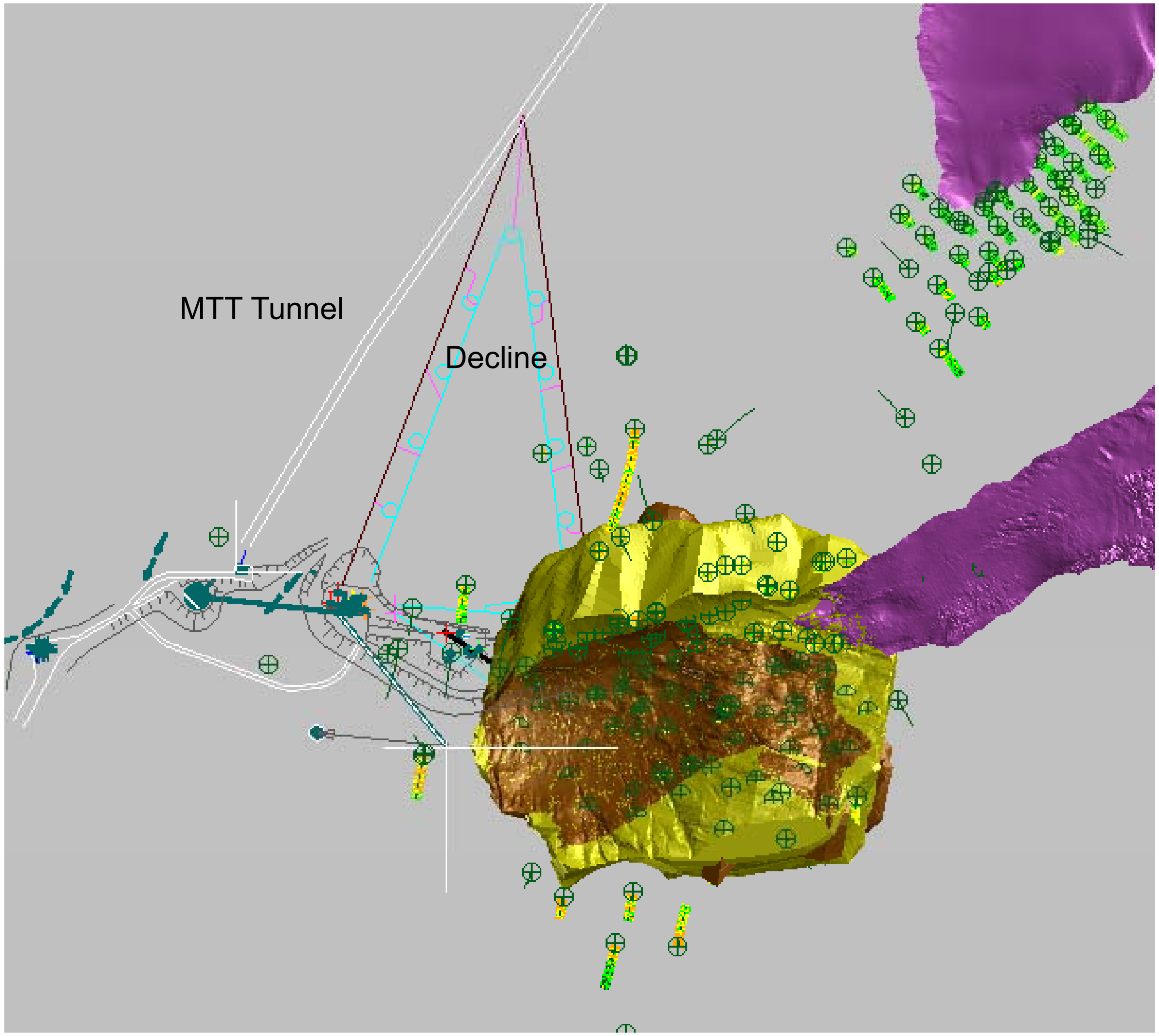

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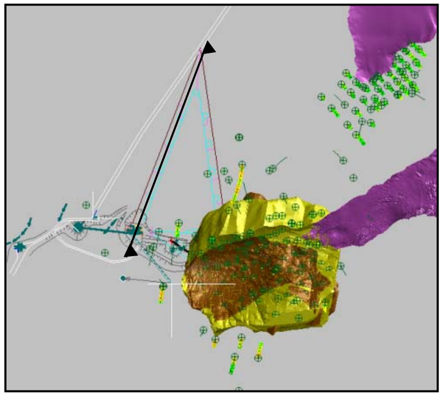

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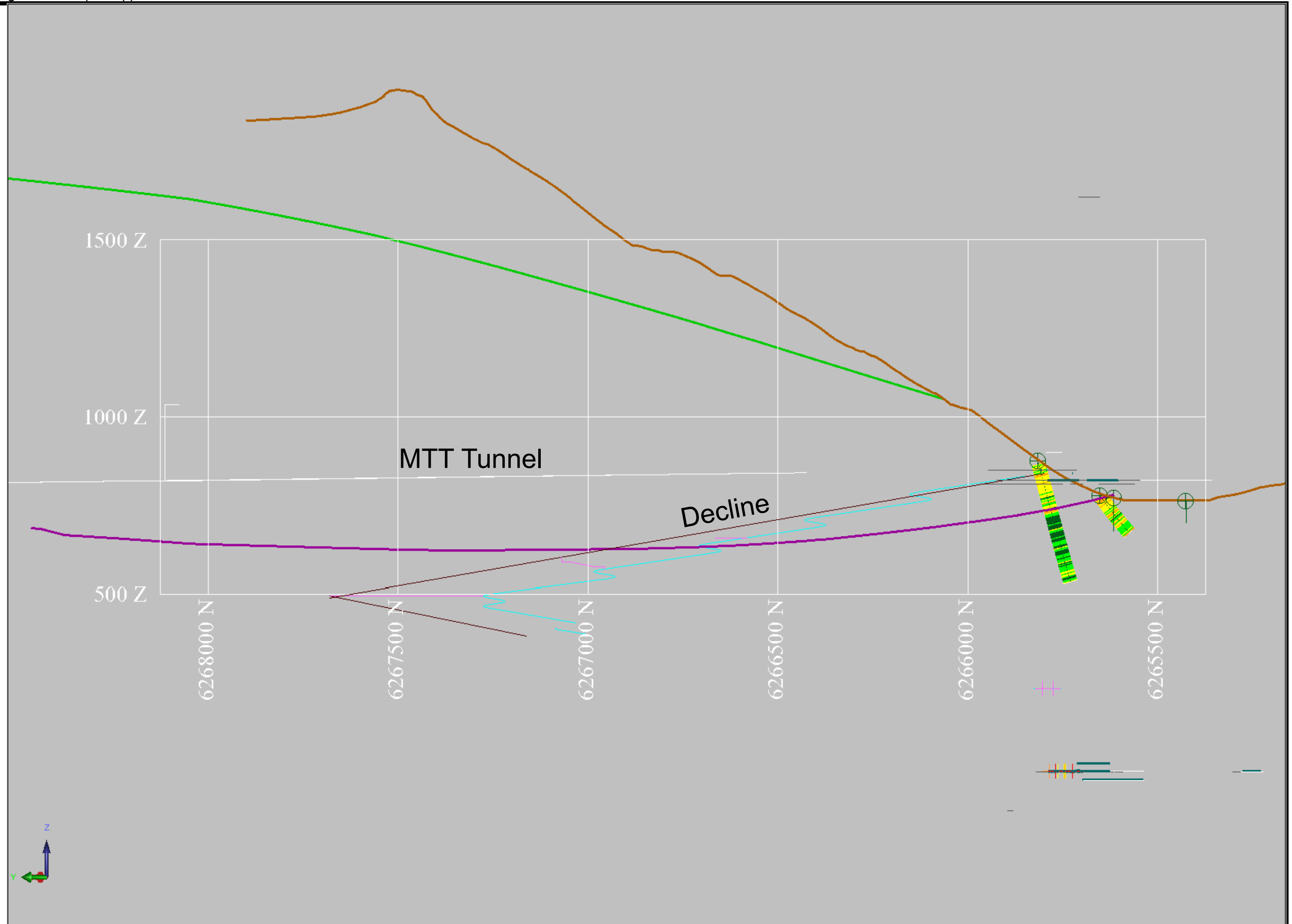
- AU 0.25g/t SHELL
- TOPOGRAPHY
- SULPHERETS THRUST FAULT
- MITCHELL THRUST FAULT



RMR Correlated from RQD

80-100	20-40
60-80	0-20
40-60	

600 m thick section

PROJECT
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
MITCHELL PROJECT, BRITISH COLUMBIA

TITLE
MITCHELL DEVELOPMENT
SECTION 01

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000	
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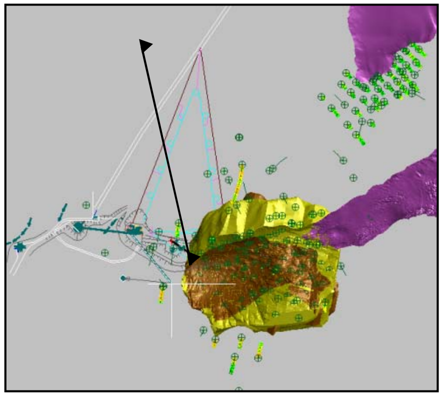

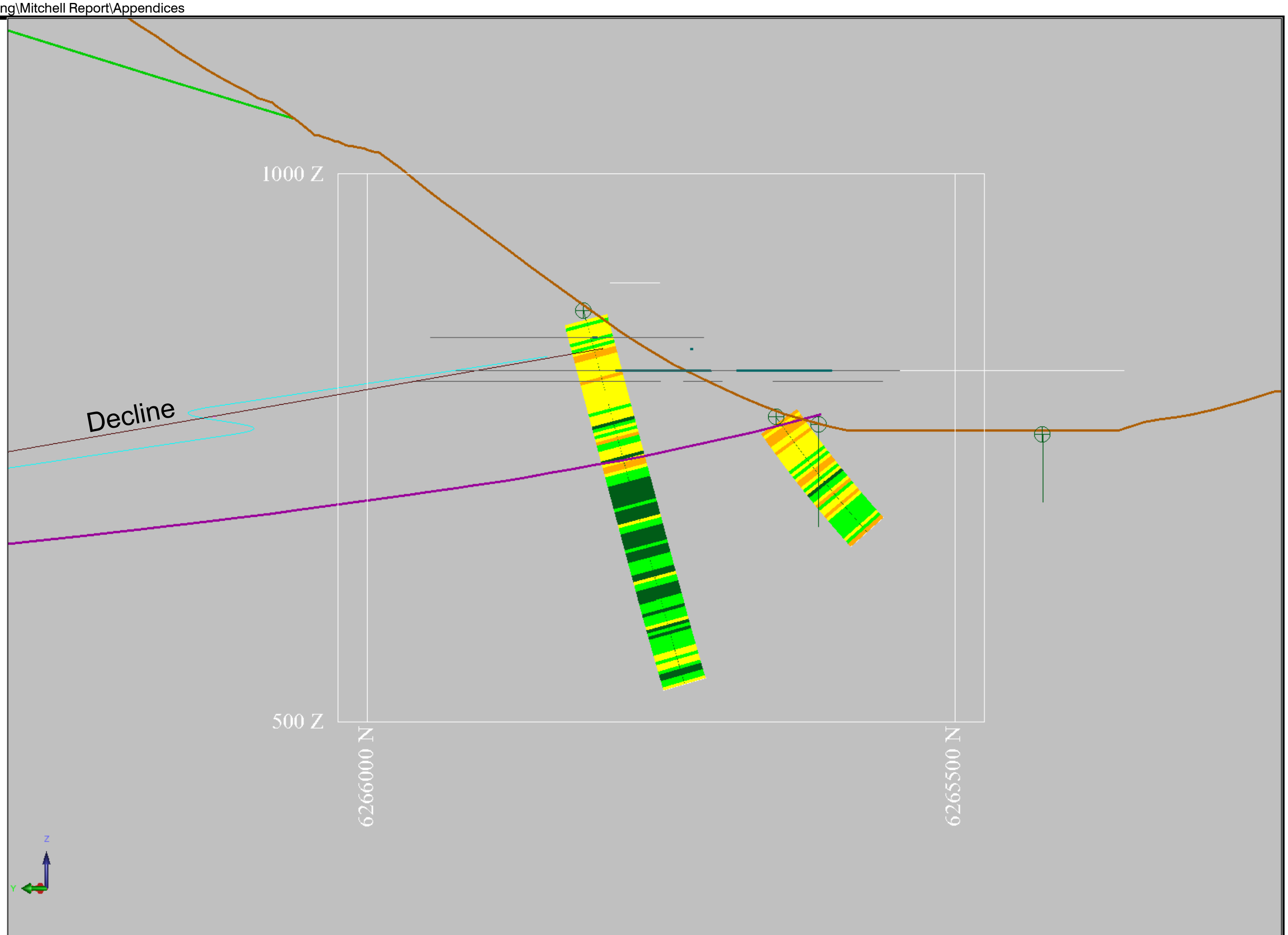

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- CU 0.1% SHELL
- AU 0.25g/t SHELL
- TOPOGRAPHY
- SULPHERETS THRUST FAULT
- MITCHELL THRUST FAULT

RMR Correlated from RQD

80-100	20-40
60-80	0-20
40-60	

600 m thick section

PROJECT	SEABRIDGE GOLD INC. KSM CONCEPTUAL STUDY MITCHELL PROJECT, BRITISH COLUMBIA
TITLE	MITCHELL DEVELOPMENT SECTION 01

Golder Associates

PROJECT No.11-1439-0002			PHASE No. 10000
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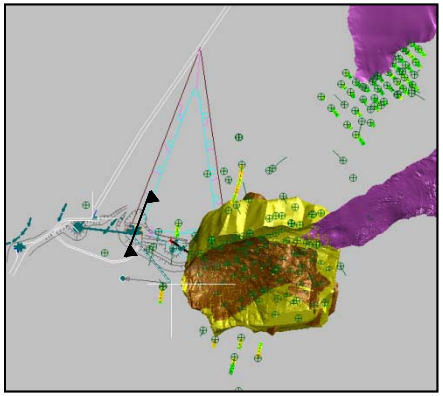

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- AU 0.25g/t SHELL
- TOPOGRAPHY
- SULPHERETS THRUST FAULT
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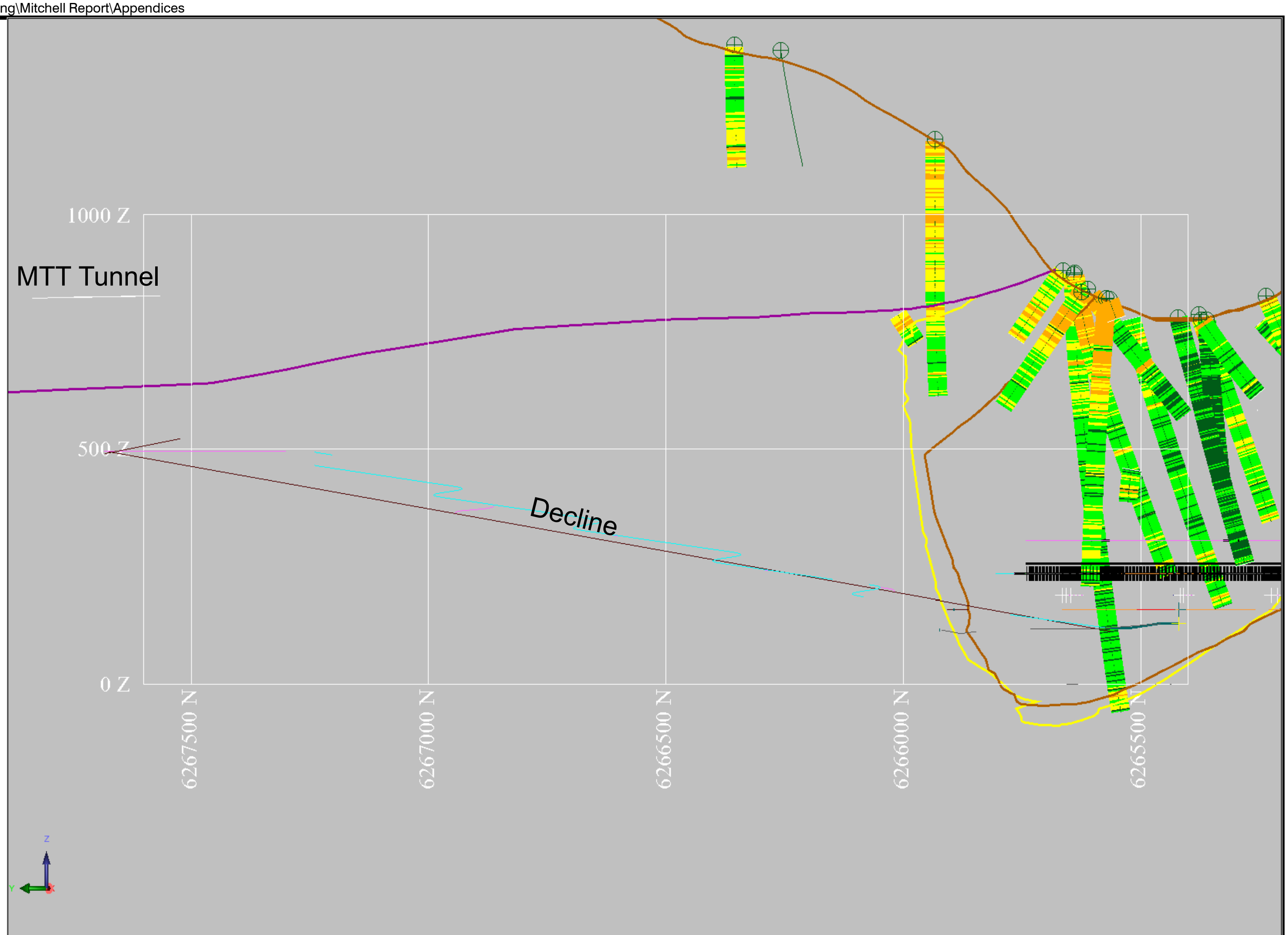

RMR Correlated from RQD

80-100	20-40
60-80	0-20
40-60	

200 m thick section

PROJECT
SEABRIDGE GOLD INC.
KSM CONCEPTUAL STUDY
MITCHELL PROJECT, BRITISH COLUMBIA

TITLE
MITCHELL DEVELOPMENT
SECTION 02

Golder Associates

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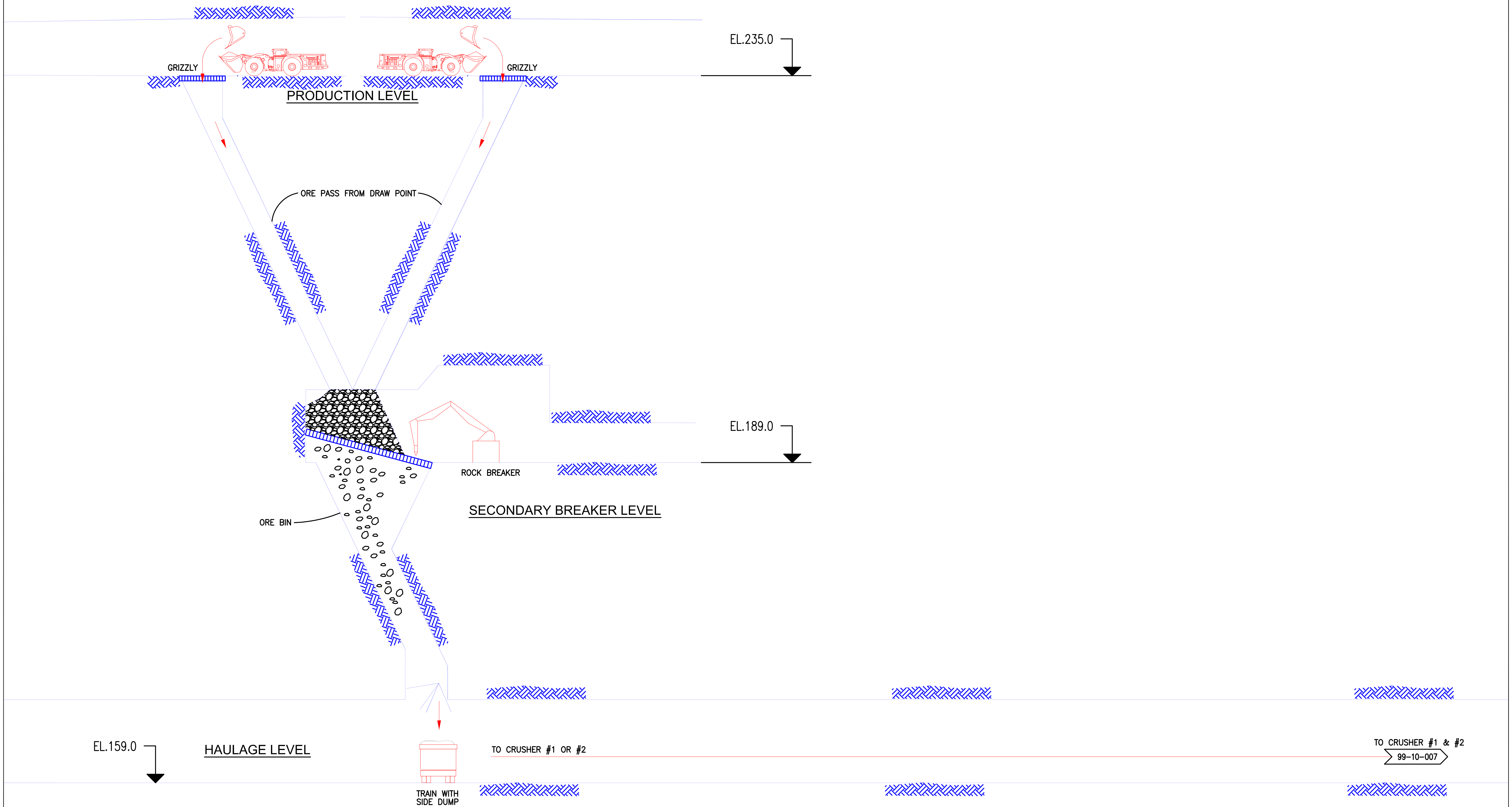
APPENDIX G

Material Movement Process Flow Diagram



TYPICAL ORE PASS (50 LOCATIONS TOTAL)

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

CLIENT: SEABRIDGE GOLD INC.

BOSCHE VENTURES LTD

TITLE: MITCHELL MUCK FLOW 55,000t/d CRUSHING/CONVEYING FLOWSHEET #1

PROJECT No.	DRAWING No.	REV No.
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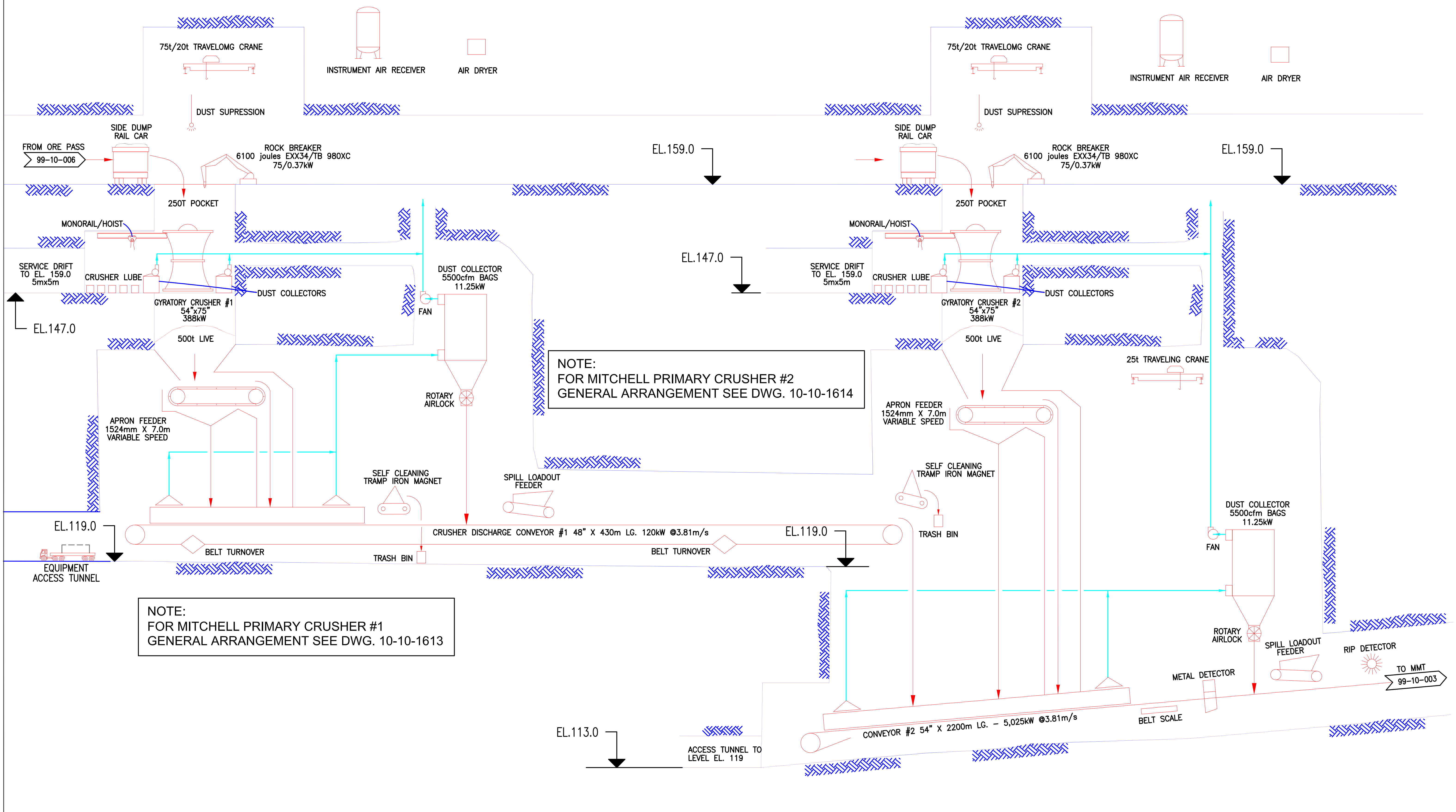

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THIS DRAWING IS THE CONFIDENTIAL PROPERTY OF SEABRIDGE GOLD INC. AND SHALL NOT BE DUPLICATED WITHOUT PERMISSION.

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

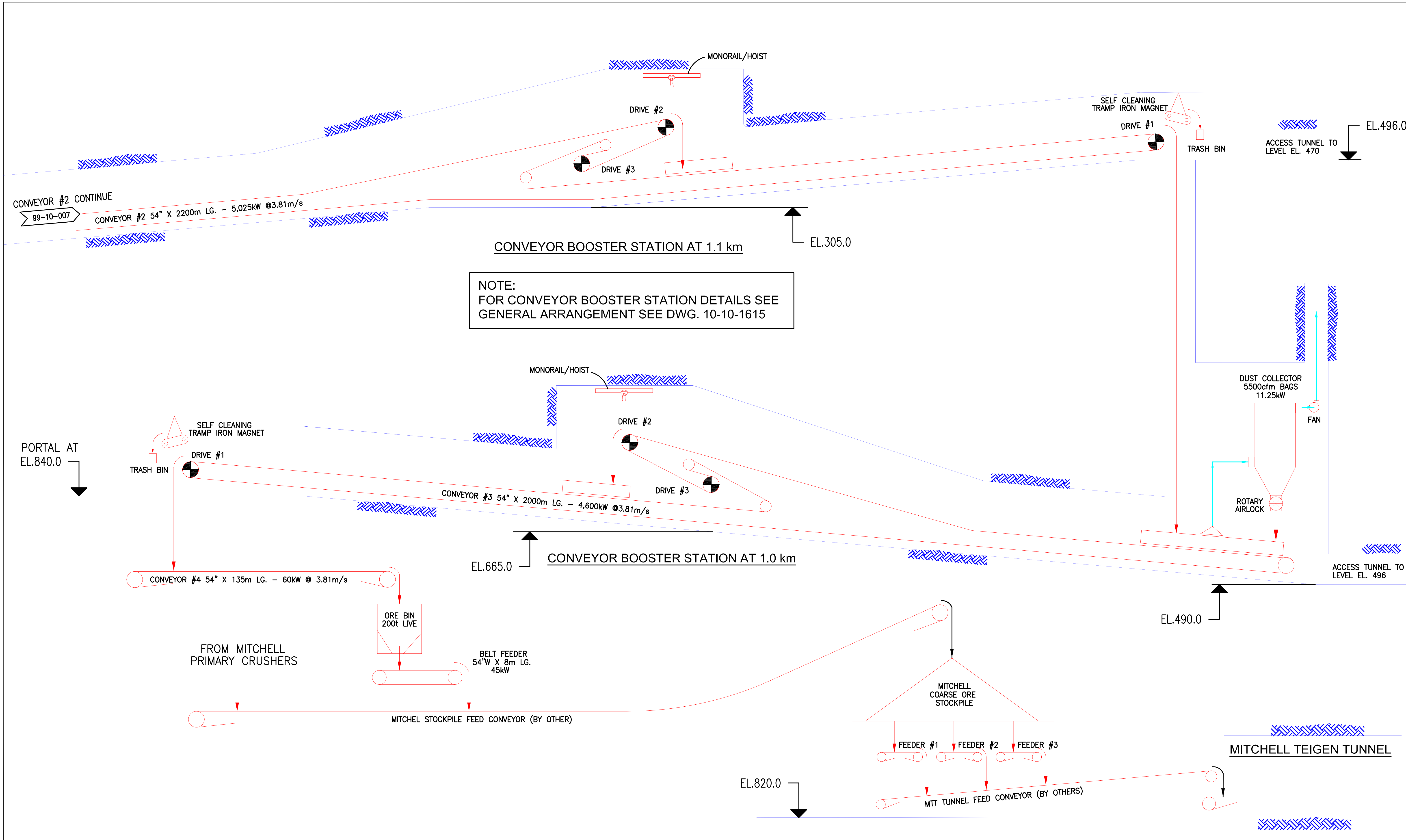
CLIENT: SEABRIDGE GOLD INC.

TITLE: MITCHELL MUCK FLOW 55,000t/d CRUSHING/CONVEYING FLOWSHEET #2

PROJECT No.:

DRAWING No.: 99-10-007

REV No.: D



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

CLIENT: SEABRIDGE GOLD INC.

TITLE: MITCHELL MUCK FLOW 55,000t/d CRUSHING/CONVEYING FLOWSHEET #3

PROJECT No.
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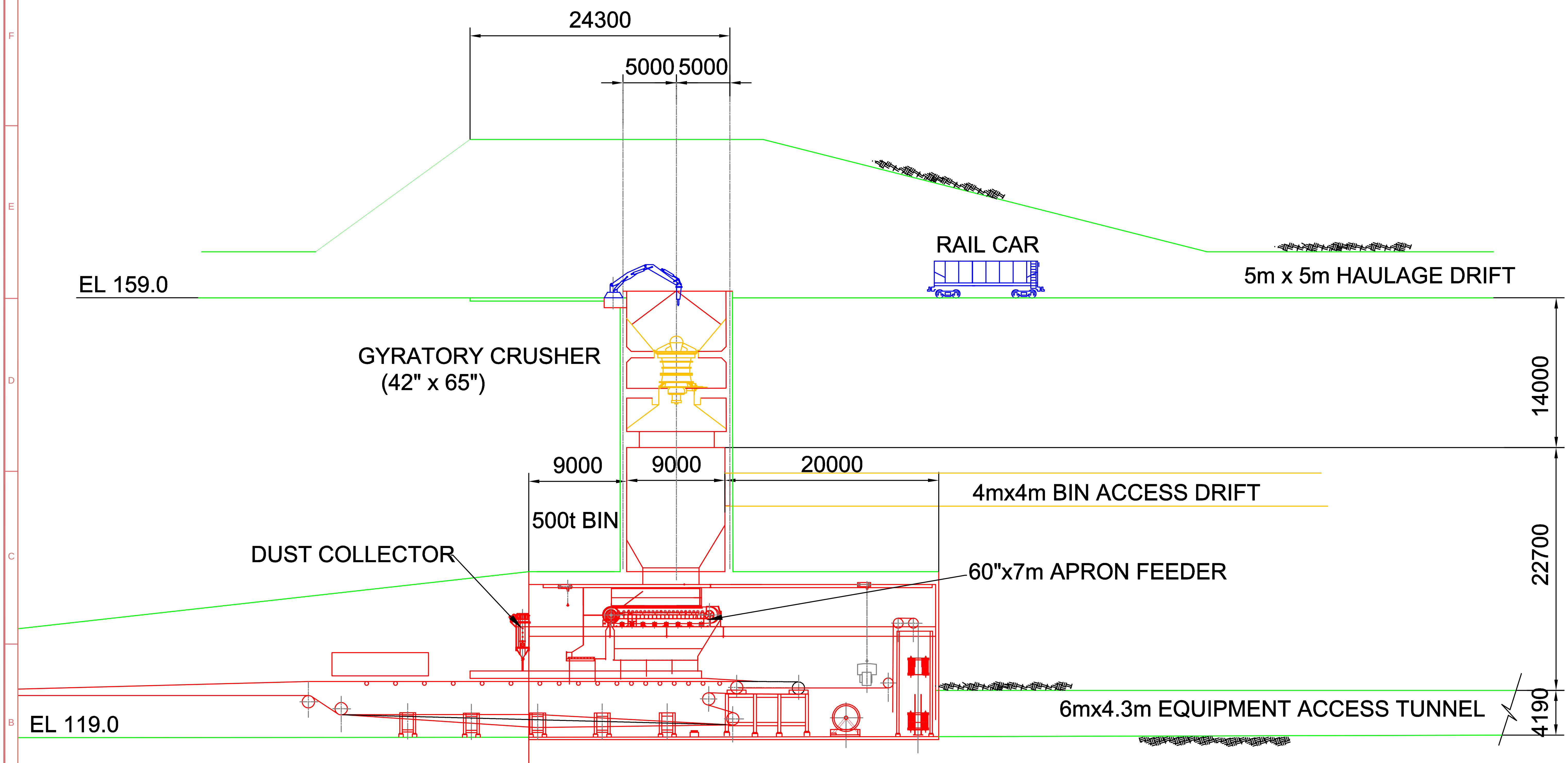

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REVISIONS

NUMBER	SUBJECT	REVISION
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SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

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

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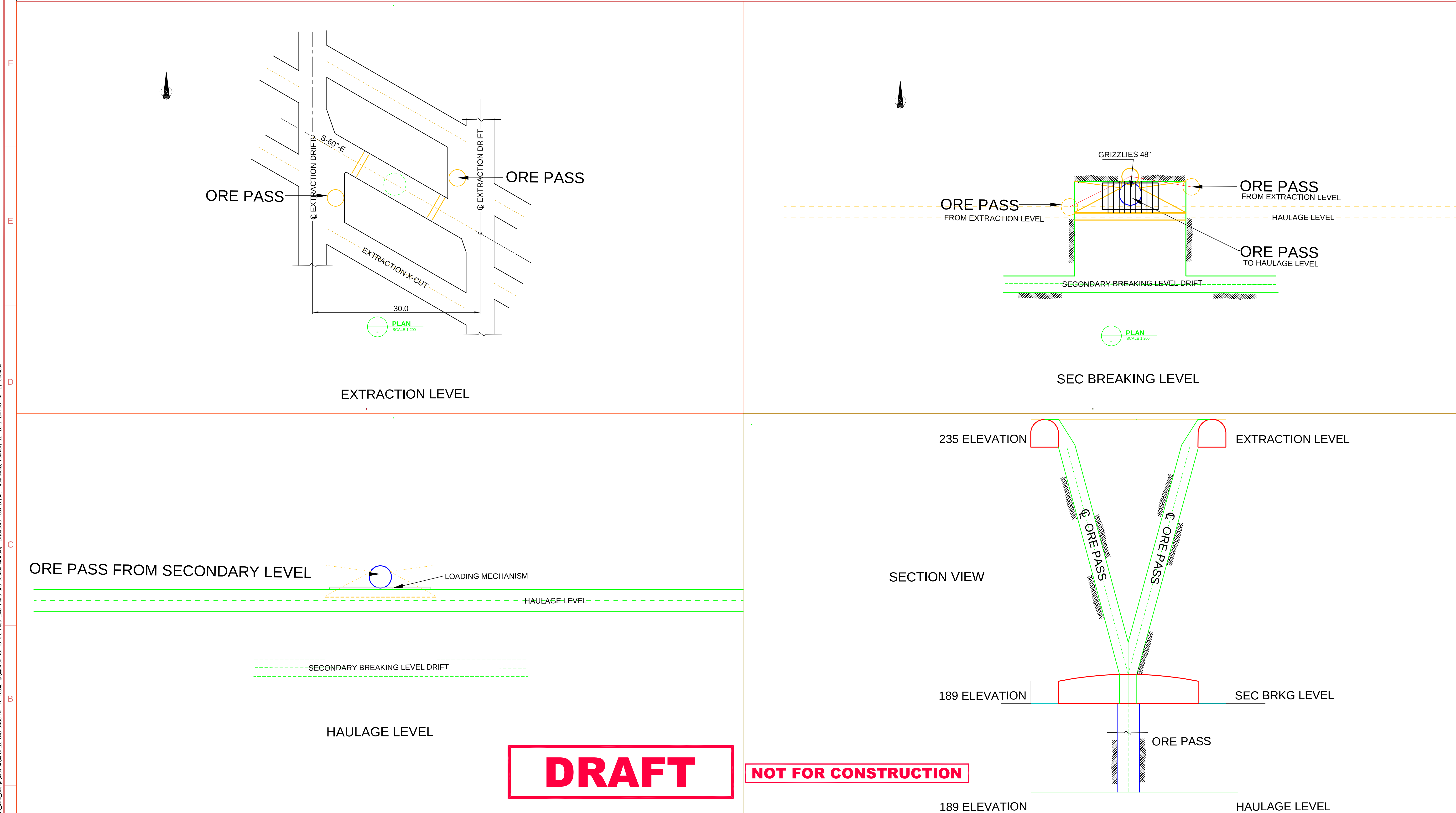

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	ORE PASS DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: **ORE PASS LEVEL PLANS AND SECTION**

PROJECT No.	PHASE No.	TASK No.
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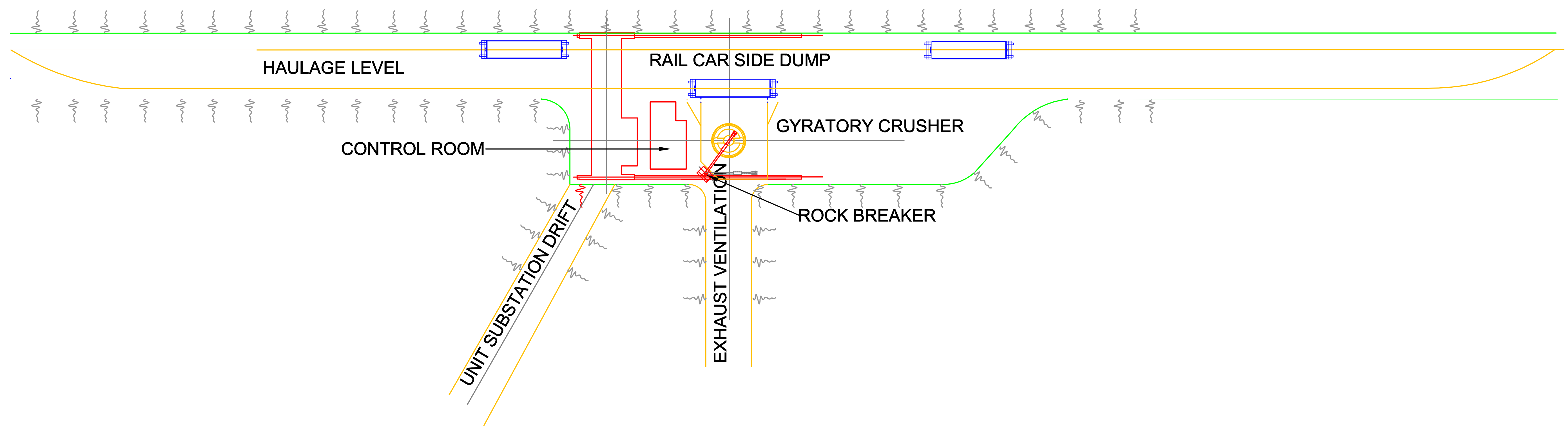

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REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	CAR SIDE DUMP DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

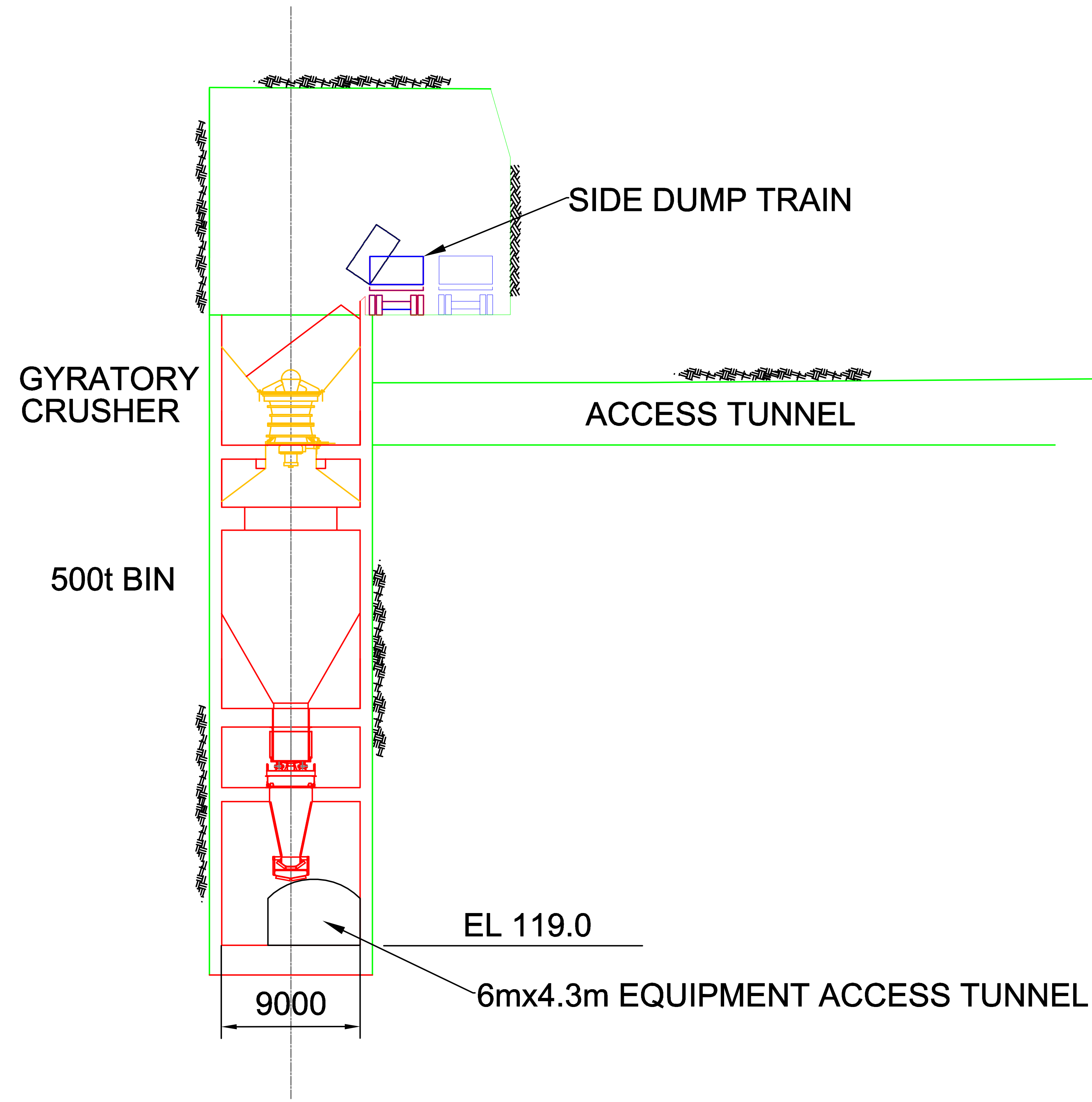
PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

CONSULTANT: Golder Associates

DRAWING TITLE: **MITCHELL CAR SIDE DUMP PLAN**

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-14	1



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REV	MM/DD/YY	REVISION	DES	CHK	DRW	CHK	RVW
1	12/09/11	0	####	INT	####	INT	####

REVISIONS

NUMBER	SUBJECT	REVISION
SP-1001-01	SIDE DUMP DIAGRAM	A

SPECIFICATIONS

REFERENCE TITLE

REFERENCES

STAMP/SEAL

CLIENT: SEABRIDGE GOLD INC

CONSULTANT: Golder Associates

PROJECT: MITCHELL BLOCK CAVE MINE DESIGN

DRAWING TITLE: **MITCHELL CAR SIDE DUMP SECTION**

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

ORIGINAL SHEET SIZE	DRAWING No.	REVISION
ANSI D	MITCHELL-15	1

APPENDIX H

Ventilation Design - Airflow Calculations for Each Level

1.0 VENTILATION DESIGN

The preconditioning (PC) level (305 Level) is ventilated by fresh air from one of the ramps that also ventilates the undercut (UC) level. The west undercut 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 m round ventilation raise to the extraction level (235 Level). The ventilation raise requires a fan installed to pull air up the preconditioning ramp, through the level, and down the raise to the extraction level.

Air from the FAD travels to the UC level (255 Level) via three ramps which are collared-off crosscuts between the FAD and the extraction level. 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 6.4 m/s, to access the undercut. The air flows through the UC level and down a 5.0 m round ventilation raise to the RAD.

The extraction level is ventilated through the ventilation raise from the PC level and thirteen crosscut drifts that connect the fresh air ring to the extraction level. In order to ensure that the air flowing through each drift on the extraction level is well-balanced to meet production needs, it is important that air inlets to the level are strategically located. The thirteen crosscuts are regulated to ensure the appropriate amount of air enters the extraction level. The quantity of airflow required for the extraction level was calculated according to Table H-3 (The calculations for all levels, including the workshop and conveyors, can be found between Table H-1 and H-9).

In the current ventilation model, 15% 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 by the time production begins. When production commences, the air that was being drawn up to the PC level will go directly to the extraction level through the thirteen crosscuts connecting the fresh air ring to the extraction level. The estimated recirculation on the extraction level will be approximately 7% due to air leakage through ore passes.

Air within the extraction level travels through the extraction drifts past the many draw points (which are not in the model) and then down 3.0 m ventilation raises to the RAD. Each extraction drift has an associated ventilation raise; there are currently 34 ventilation raises from the level to the RAD. It was assumed that a minimum of seven extraction drifts must have sufficient airflow for two LHDs per drift to meet production requirements (14 workplaces). The used air from the extraction level moves to the RAD and then to the Return Air Raises (RAR), which exhaust to surface.

Air movement through the extraction level will be controlled by regulators, located on the top of the 34 ventilation raises, which will have to be sealed very tightly when not open. To obtain the airflow necessary in the model, the resistance of the closed regulators was modelled with a resistance of 100 Ns^2/m^8, which is ten times the resistance value of one ventilation door.

Fresh air to the secondary breakage level (189 Level) is provided through a 3 m raise from the FAD. The air flows eastward across the level and exhausts to the RAD through a 2.4 m raise and the ramp to the haulage level on the east side of the level. Air speeds were maintained at a minimum of 1 m/s to dilute dust generated by the rockbreakers.

The haulage level (159 Level) is ventilated by three raises connecting to the FAD. Air flows eastward across the level and exhausts through three raises and one ramp down to the RAD. Air speeds were also maintained at a minimum of 1.0 m/s to help lower dust concentrations on the level generated by haulage level activities.

The underground mechanics shop is also located on the 159 Level. It is ventilated by a split of air from one of the three raises connecting the FAD to the 159 Level. The used air is exhausted to the conveyor ramp.

The crushing level (119 Level) is ventilated with air from the main and access ramps. The air from the ramps flows through the level and then exhausts out of the conveyor ramp and the RAD. Air speeds were maintained at a minimum of 1 m/s to dilute dust generated by the crushers. Also, each crusher will have its own split of air to ensure that the dust produced by one crusher will not affect the quality of air ventilating the second crusher.

1.1.1 Recommendations

Consideration should be given to widening the RAD to lower the overall mine resistance. Approximately 80% of the air exiting the mine will do so via the RAD and RAR. The dimensions of the RAD are 7.5 m by 7.5 m which, due to the high velocity of the air, is a bottleneck in the system. A detailed trade-off study between the cost of the additional development and the fan capital and operating costs by year may indicate that it would be more cost-effective to widen the RAD to a larger size with smaller fans on the surface.

The RAR and FAR collar locations will work well with the predominant southeasterly wind to ensure that return air does not enter back into the mine. However, the main ramp and conveyor ramp portal locations and proximity with regards to return air dispersion should be looked at in more detail. Southeasterly winds will pose a problem for the quality of the air entering the mine if some of the return air is recirculated back through the mine.

Staging of the ventilation system will be an important factor governing mine development priorities. For example, it will be necessary to use ore passes and crusher raise excavations as temporary ventilation raises to ensure development headings have appropriate airflow.

1.1.1.1 Future Work

There are a number of items in the Mitchell ventilation model which 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;
- a review of the current portal locations with regard to possible air recirculation; and
- a detailed trade-off study of RAD development costs vs. main fan costs.

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.

Airflow in ventilation shafts of a depth greater than 500 m should be modelled as compressible fluids. The Mitchell mine model is limited because it assumes air is an incompressible fluid with a constant density. Although compressible fluid modelling was not done, the current model is sufficient for pre-feasibility purposes. Further modelling using thermodynamic environment simulation for compressible air flows and for the effects of auto compression and heat transfer from the rock strata on the temperature underground is recommended.

Table H-1: 305 Preconditioning Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
Production drill	2	74	79%	25%	2
Face drill	1	120	71%	25%	1
Bolter	2	115	76%	25%	3
LHD	1	352	58%	75%	10
Truck	2	405	65%	75%	25
Anfo loader	1	111	55%	75%	3
Scissor lift	1	95	83%	50%	2
Toyota	1	96	75%	75%	3
Subtotal					50
Contingency					20%
Total required for equipment					**60**
Total modelled					**62**
Modelled leakage through cave					2
Recirculated air					-

Table H-2: 255 Undercut Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
Production drill	2	74	79%	25%	2
Face drill	1	120	71%	25%	1
Bolter	2	115	76%	25%	3
LHD	1	352	58%	75%	10
Truck	3	405	65%	75%	37
Anfo loader	1	111	55%	75%	3
Scissor lift	1	95	83%	50%	2
Crane truck	1	95	63%	50%	2
Toyota	2	96	75%	75%	7
Subtotal					67

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
Contingency					20%
Total required for equipment					**81**
Total modelled					**87**
Modelled leakage through cave					8
Recirculated air					-

Table H-3: 235 Extraction Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
Production drill	2	74	79%	25%	2
LHD	14	352	76%	75%	176
Face drill	1	120	71%	25%	1
Bolter	2	115	76%	25%	3
Truck	3	405	65%	75%	37
Anfo loader	1	111	55%	75%	3
Scissor lift	1	95	83%	50%	2
Crane truck	1	95	63%	50%	2
Mobile rockbreaker	2	95	63%	75%	6
Secondary rockbreaker	4	75	31%	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					273
Contingency					20%
Total required for equipment					**328**
Total modelled					**389**
Modelled leakage through cave					23
Recirculated air*					86

*60 m^3/s of this recirculated air is from the Preconditioning Level which will not be operating during production.

Table H-4: 189 Secondary Breakage Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
LHD	1	352	76%	75%	13
Scissor lift	1	95	83%	50%	2
Crane truck	1	95	63%	50%	2
Toyota	2	96	75%	75%	7
Subtotal					24
Contingency					20%
Total required for equipment					**29**
Total modelled*					**64**
Modelled leakage through cave					-
Recirculated air					4

*The modelled quantity for this level was dictated by minimum air speeds, not diesel equipment requirements.

Table H-5: 159 Haulage Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
LHD	1	352	76%	75%	13
Scissor lift	1	95	83%	50%	2
Crane truck	1	95	63%	50%	2
Toyota	2	96	75%	75%	7
Subtotal					24
Contingency					20%
Total required for equipment					**29**
Total modelled*					**165**
Modelled leakage through cave					-
Recirculated air					11

*The modelled quantity for this level was dictated by minimum air speeds, not diesel equipment requirements.

Table H-6: 119 Crushing Level Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
LHD	1	352	76%	75%	13
Truck	1	405	65%	75%	12
Toyota	1	96	75%	75%	3
Subtotal					29
Contingency					20%
Total required for equipment					**35**
Total modelled*					**75**
Modelled leakage through cave					-
Recirculated air					-

*The modelled quantity for this level was dictated by minimum air speeds, not diesel equipment requirements.

Table H-7: Main Ramp Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
LHD	2	352	76%	75%	25
Bolter	1	115	76%	25%	1
Truck	1	405	65%	75%	12
Scissor lift	1	95	83%	50%	2
Crane truck	1	95	63%	50%	2
Grader	1	114	63%	100%	4
Toyota	3	96	75%	75%	10
Personnel carrier	1	130	25%	50%	1
Subtotal					60
Contingency					20%
Total required for equipment					**72**
Total modelled*					**120**
Modelled leakage through mine portal ventilation doors					14
Recirculated air					-

*The modelled quantity for this ramp was dictated by overall vent design and minimum airspeeds.

Table H-8: Access Ramp Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m^3/s)
LHD	1	352	76%	75%	13
Truck	1	405	65%	75%	12
Toyota	1	96	75%	75%	3
Personnel carrier	1	130	25%	50%	1
Subtotal					30
Contingency					20%
Total required for equipment					**36**
Total modelled*					**53**
Modelled leakage through cave					-
Recirculated air					-

*The modelled quantity for this ramp was dictated by overall vent design and minimum airspeeds.

Table H-9: 159 Workshop Ventilation Requirements

Diesel Equipment	Quantity	Engine Size (kW)	Shift Utilization	Diesel Utilization	Total Airflow (m3/s)
LHD	1	352	76%	75%	13
Toyota	1	96	75%	75%	3
Subtotal					17
Contingency					20%
Total required for equipment					**21**
Total modelled					**21**
Modelled leakage through cave					-
Recirculated Air					-

APPENDIX I

Electrical Design - WN Brazier Associates Ltd.

WN BRAZIER ASSOCIATES INC

SEABRIDGE GOLD INC.
KSM PROJECT

MITCHELL BLOCK CAVE ELECTRICAL ESTIMATE - 2012 PFS

Rev. A - March, 2011
Rev. B – 11 April, 2011
Rev. 0 - March 2012
Rev. 1 – 28 March 2012

TABLE OF CONTENTS

1.0 INTRODUCTION

1.1 General

This pre-feasibility level report covers the electrical, communication and instrumentation capital costs for the proposed Mitchell Block cave installation and has been prepared for Golder Associates Ltd. as requested and directed by Seabridge Gold Inc.

The cost of the initial development stage power distribution system, to run mining equipment and vent fans for drift excavation, etc. is not included in this budget and is assumed to be included in mine development costs.

2.0 BASIS OF ESTIMATE

2.1 Golder Associates Ltd.

Golder Associates provided basic information on the proposed block-caving alternative. This included an electrical load list and basic mine plans.

Harold Bosche, of Bosche Ventures, provided drawings and additional information on the conveyor and crusher system.

All costs are in first quarter 2012 Canadian dollars. The costs associated with the 7% provincial sales tax (PST) that is being reinstated are included in costs where appropriate, but note this only applies to lighting and other non-process equipment and does not apply to construction labour, so the impact is small.

2.2 Items Included

The estimated costs include:

- Underground mine power supply cables.
- Ring main (circuit breaker) units.
- Underground mine switchgear and unit substations (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 supply for surface vent fans and air heaters.
- Communication system.
- Power supply to trolley system.
- Sustaining capital for Crusher Station 2.

2.3 Items Not Included

The cost estimate does not include:

- Electrical for the initial construction (drift excavation) phase.

- The equipment trailing cables, etc. as used in the actual operating phase mining (assumed to be in the mining cost).
- The per kilowatt hour cost of electric power as used in mining operations. (This is assumed to be factored into the mining costs.)
- The cost of the general site power distribution. (A 25 kV power line would be built to the area of the mine from the KSM Substation No. 2, at the Mitchell plant site, as part of the overall site development cost.)
- Surface facilities including any required emergency generators, etc. (These are being estimated by Wardrop).
- Any electrical costs associated with mine water, after its pumped from the mine.

3.0 POWER SYSTEM DESCRIPTION

3.1 General

The proposed block cave operation would be a large-scale operation, and as such a ring main style power distribution system is planned, rather than a simple radial distribution as one often sees in small underground mines.

Although the study has been based on a large amount of diesel equipment, the included power supply system 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 5 cents per kilowatt hour, additional electrically powered equipment may be cost effective compared to diesel powered machines. It is probable that future plans would include a higher electric power demand.

Figure 3.1 – 1 Block Cave General Layout (As Per Golder)

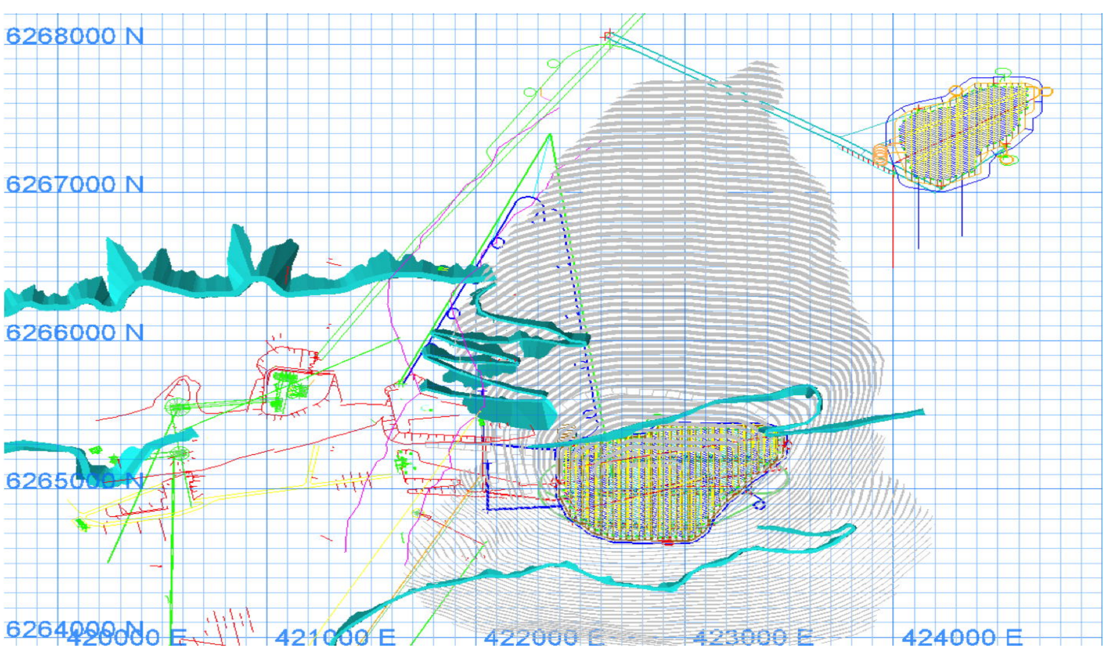


Figure 3.1 - 2 Overall View (From Golder)

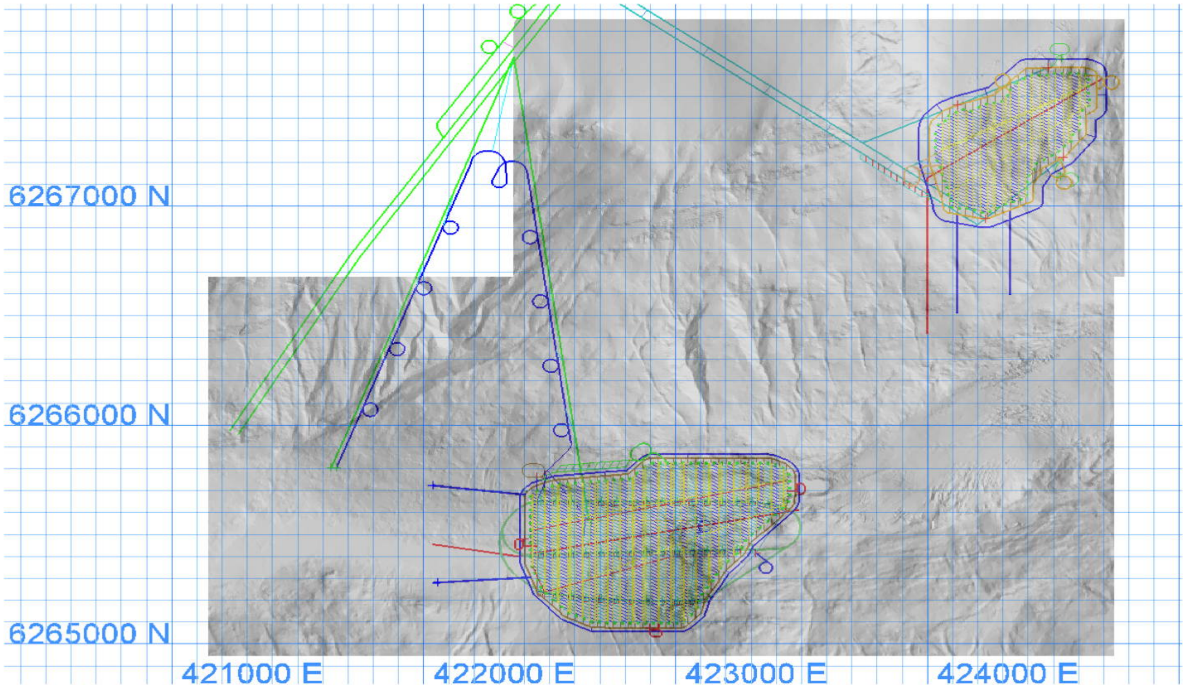


Figure 3.1 – 3 Mitchell Mine Design (From Golder)

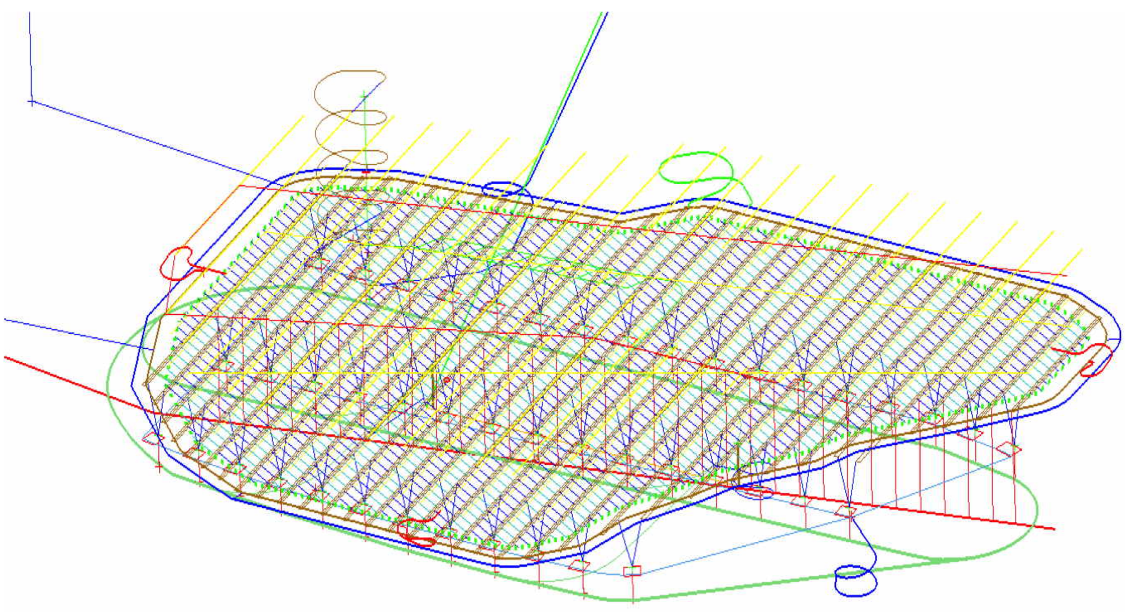


Figure 3.1 - 4 Mitchell Plan View (From Golder)

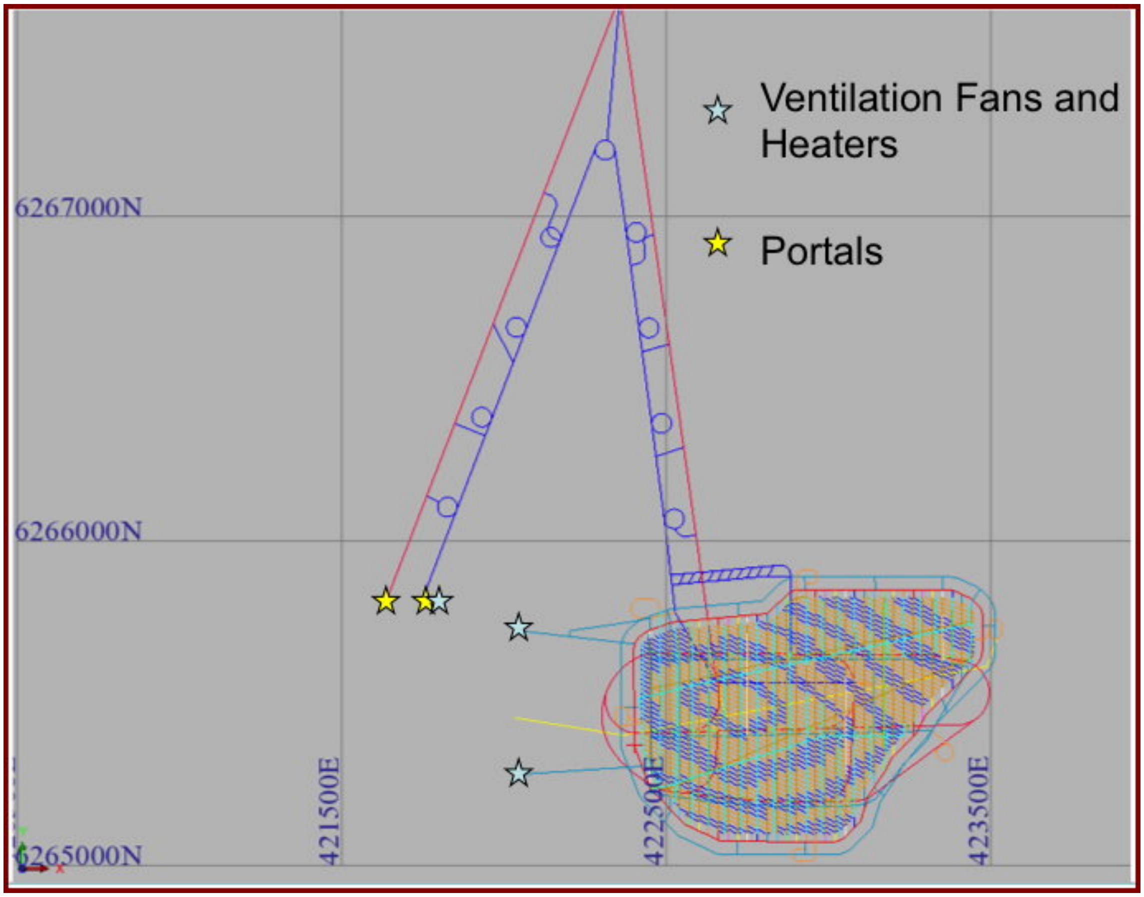


Figure 3.1 - 5 Mitchell Mine Cross Section (From Golder)

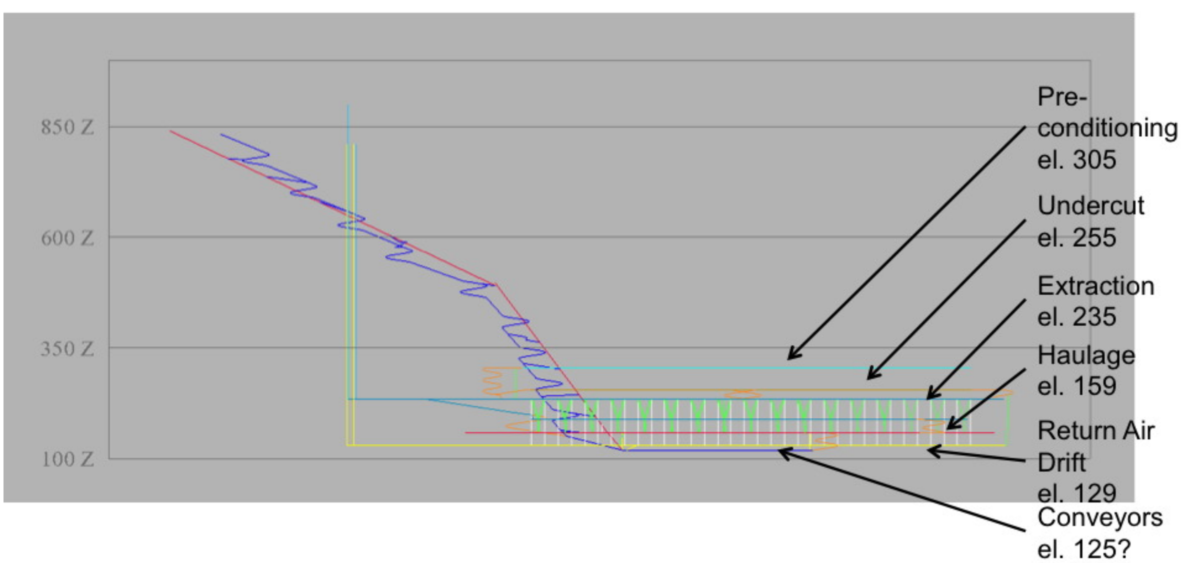


Figure 3.1 - 6 Mitchell Mine Design (From Golder)

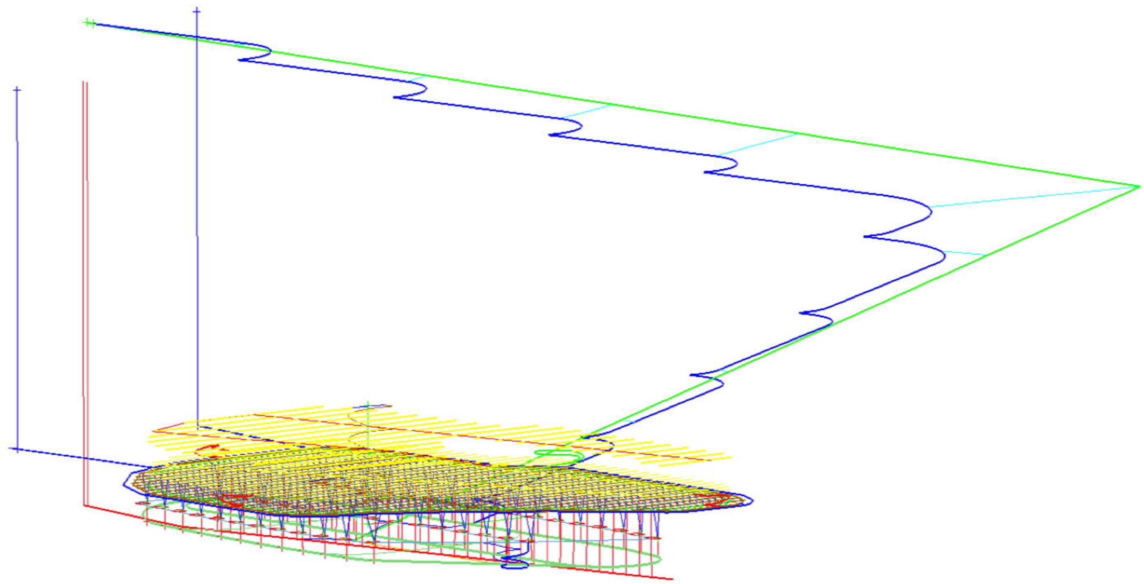


Figure 3.1 – 7 Distances

Item	Length, metres
Access ramp (surface), Length	6,342
Conveyor tunnels (surface)	4,600
Perimeter Drift, Length	3,100
Undercut	2,100
Secondary Breaking Level	2,300
Haulage Level	5,500
Preconditioning Level	2,400

3.2 Mine Main Power Distribution Cable

The main underground power distribution would be sourced from the mine site overhead 25 kV power lines (as included by Wardrop in the plant site area electrical cost estimates). The underground mine power supply would have circuit breaker and surge arrester protection at the entry points.

As the plant site distribution is 25 kV, the underground mine cables and switchgear will 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 a 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 and auxiliaries.

As a large-scale mining operation is planned, a ring main power distribution system is appropriate, rather than a simple radial supply. With a ring main system there are two main underground power feeders and all major loads can receive power from either, being supplied by ring main 3 breaker units. If a fault develops in any portion of the loop system, the fault is automatically isolated and there is no interruption of power supply to the rest of the mine loads. Essentially, all major loads are served by a loop and they can receive power from either side.

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. There will be a total of three separate 25 kV feeders to the Mitchell block cave operation. The conveyors and crushers will be fed from a separate radial feeder running down the conveyor drift. As the total mine running load is in the range of 20 MW, this exceeds the practical capacity of one ring main system so the mine electrical power including the vent fans will be on a ring main while the crushers and conveyors will be on a separate radial feed. However, in the event of power system problems alternate power connections could easily be made.

The two 25 kV ring main feeders for the supply to mining equipment and ventilation fans will be fed direct to the underground mine from an overhead line running to Mitchell Substation No. 2, which is much shorter than running cables down the access ramp or conveyor tunnel. Cables would be run down bore holes as necessary. Refer to Bosche Ventures drawing No. 10-10-1611.

600 volt systems would be resistance grounded with rapid tripping to eliminate equipment shock hazard. In addition, all trailing cables would have pilot check wires and tripping.

It is understood that current 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 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 of the Canadian Electrical Code.

For a detailed listing of included cable and power distribution equipment please refer to the detailed cost estimate spreadsheets.

3.3 Service Ramp

For the operations phase no power has been included for the service ramp. However, a ring main power cable does run down the adjacent conveyor tunnel.

3.4 Pre-Conditioning Level (El 305)

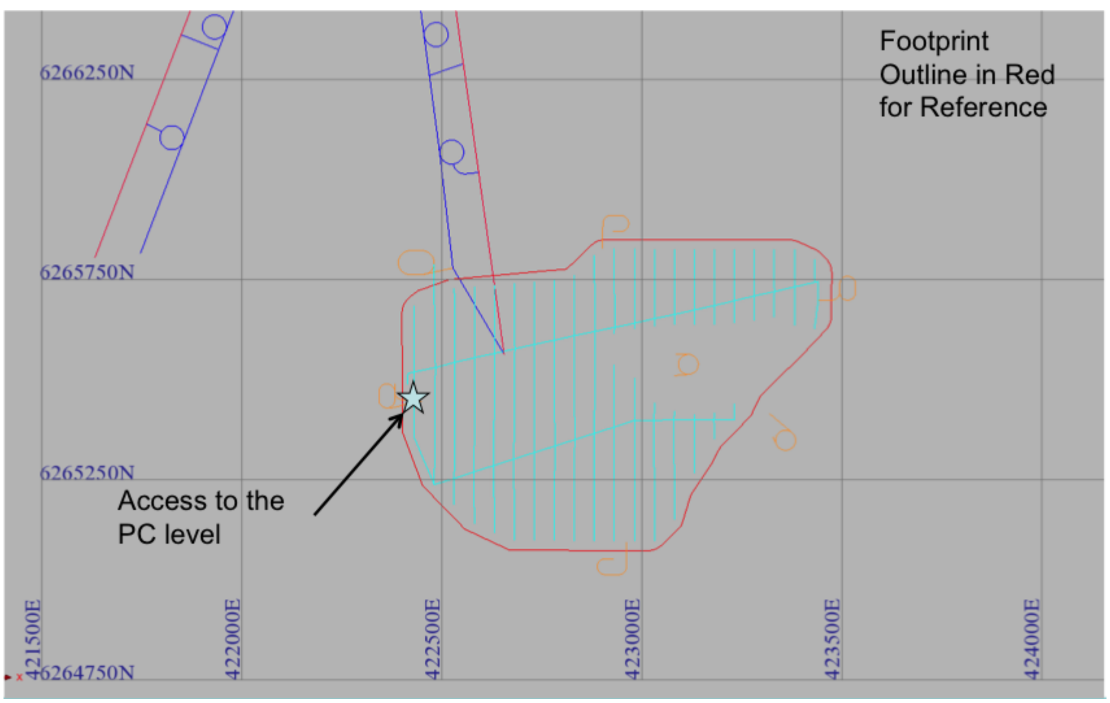
This level will require power during development. During this period it is understood that electro-hydraulic long-hole pre-conditioning drills that will be used to drill pre-conditioning bores. Portable substations and main 0.6/1 kV Teck power supply cables and end boxes are all included for use in this 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.

The electrical equipment used in pre-conditioning could very possibly be re-used after preconditioning is complete. However, due to the nature of mining, the re-use of this electrical equipment has not been considered in the estimates.

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

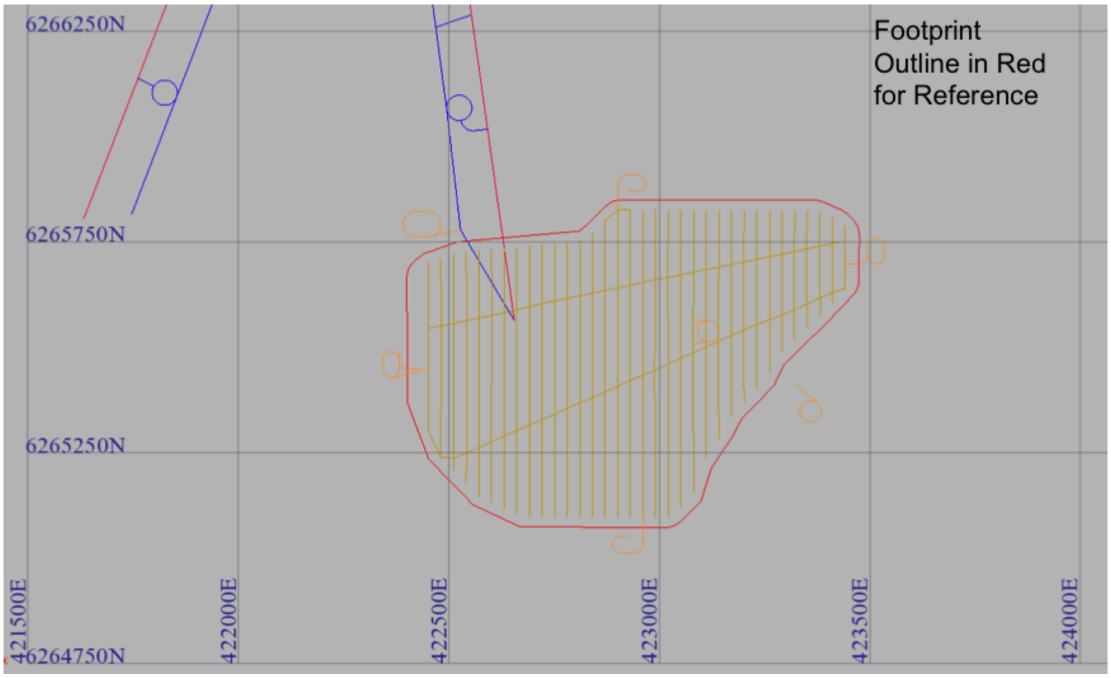
Figure 3.4 – 1 Preconditioning Level (From Golder)



3.5 Undercut Level (EL. 255)

Unit substations, 25 kV to 600 volts and end boxes are provided to power the drills on the undercut level.

Figure 3.5 – 1 Undercut Level (From Golder)



3.6 Extraction Level (EL. 235)

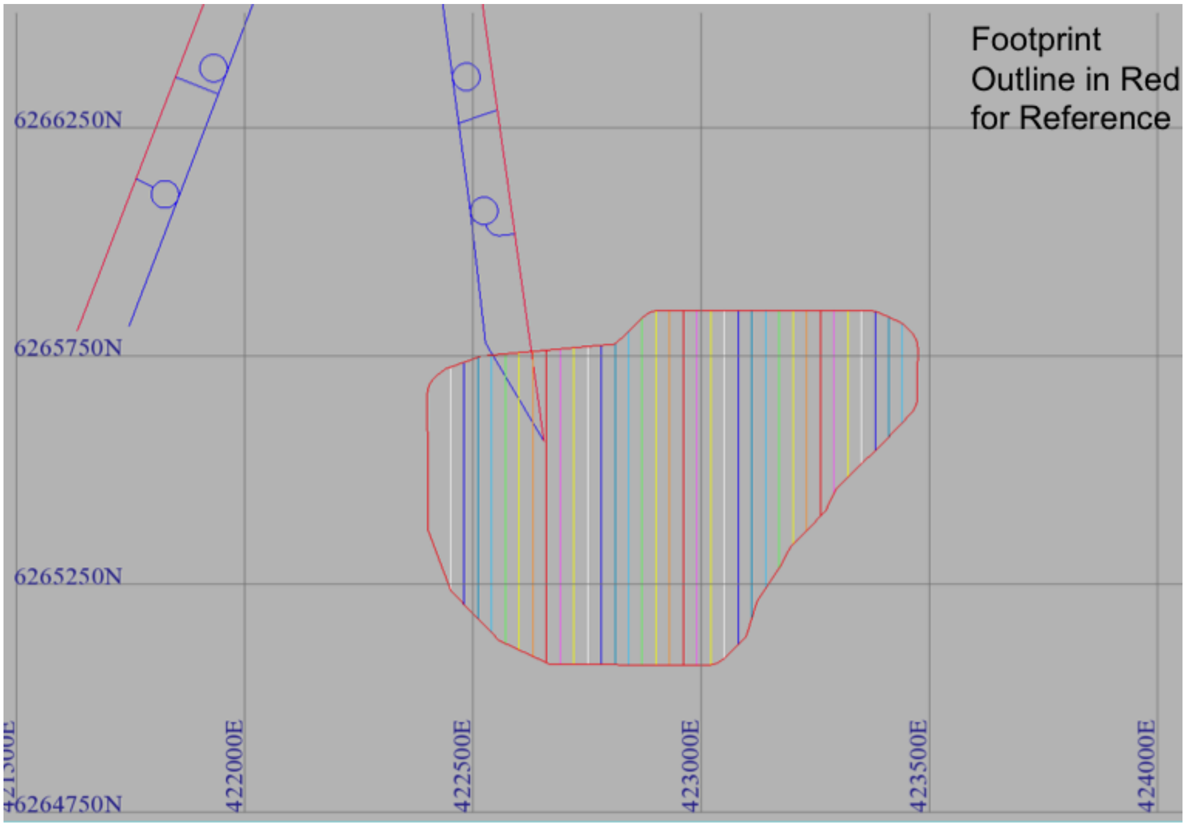
It is understood that for the extraction level that diesel LHD's will be used for dumping muck down the ore passes. (In Figure 3.7 -1 this is called the "Production Level.")

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. Refer to the detailed estimates.

A perimeter 25 kV ring main has been allowed for around extraction level, with ring main units feeding cables up/down boreholes to the other levels.

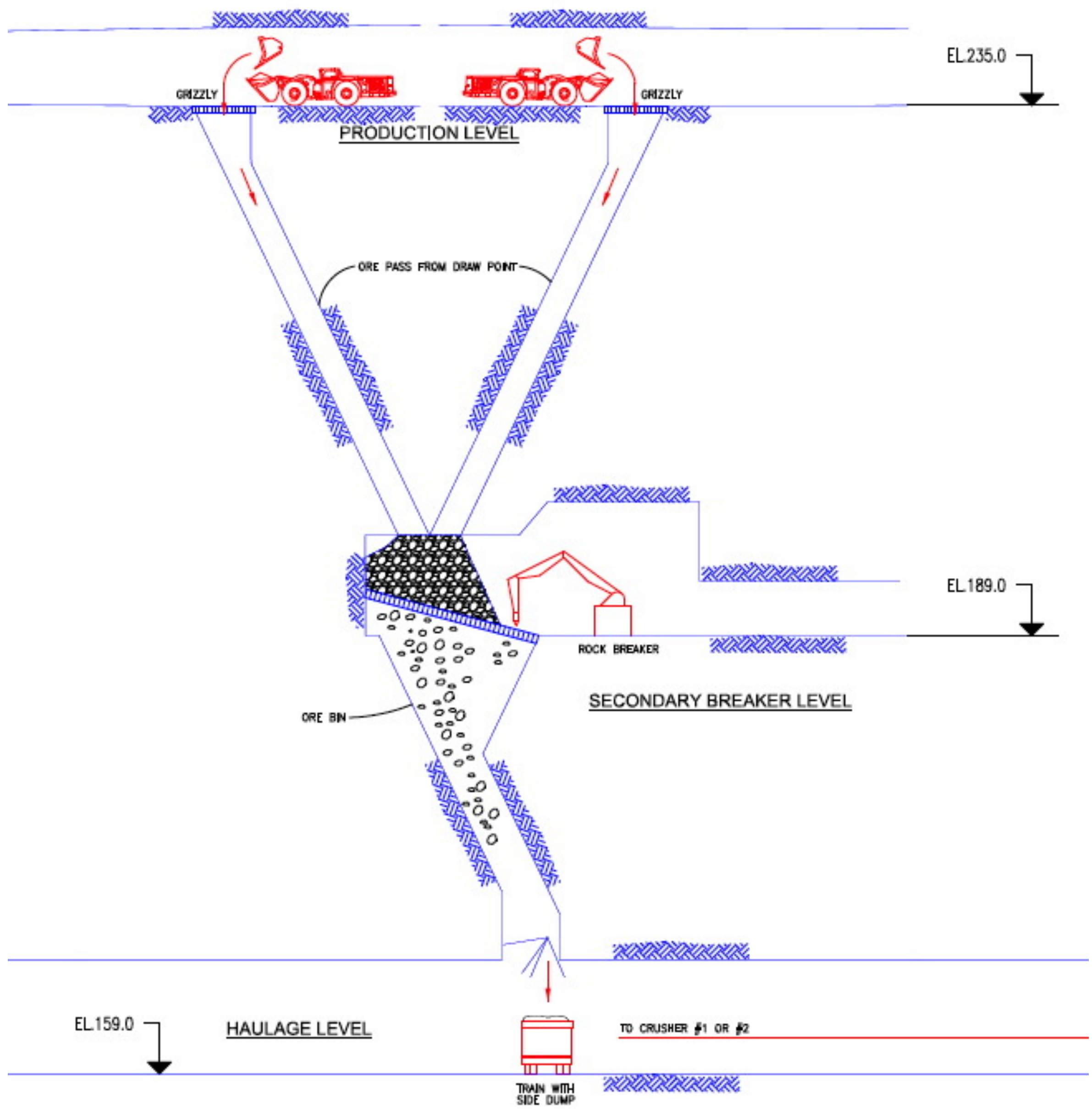
Mobile electrical equipment trailing cables are by others (part of mining cost).

Figure 3.6 – 1 Extraction Level (From Golder)



3.7 Secondary Extraction (Breaker) Level (EL 189)

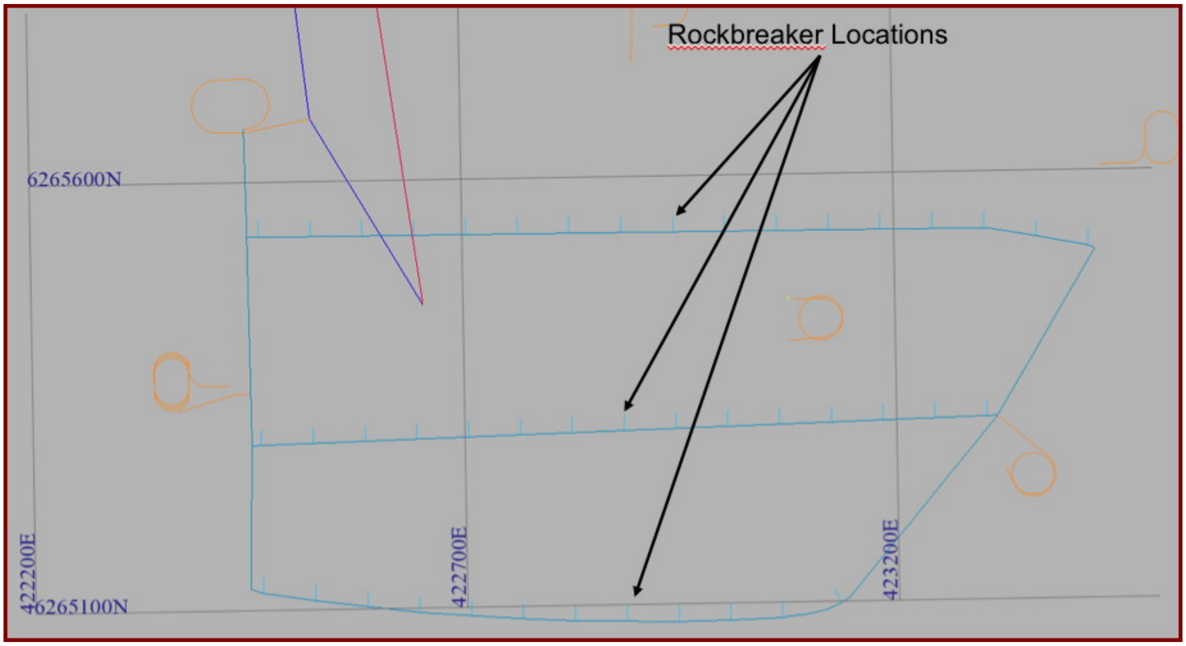
Figure 3.7 – 1 Ore Pass With Rock Breakers (From Bosche Ventures Drawing)



There will be 20 rock-breakers operational at once and these will be moved around. Each Rock-breaker is 37 kW. Six skid mount, 500 kVA, 25 kV to 600 volt unit substations have been allowed for area power supply.

As the mechanical equipment is installed over time, as the ore body is mined, the electrical equipment can be re-used.

Figure 3.7 – 2 Rock Breaker Locations (From Golder)



3.8 Haulage Level (EL 159)

Trolley type trains will be running on the haulage level.

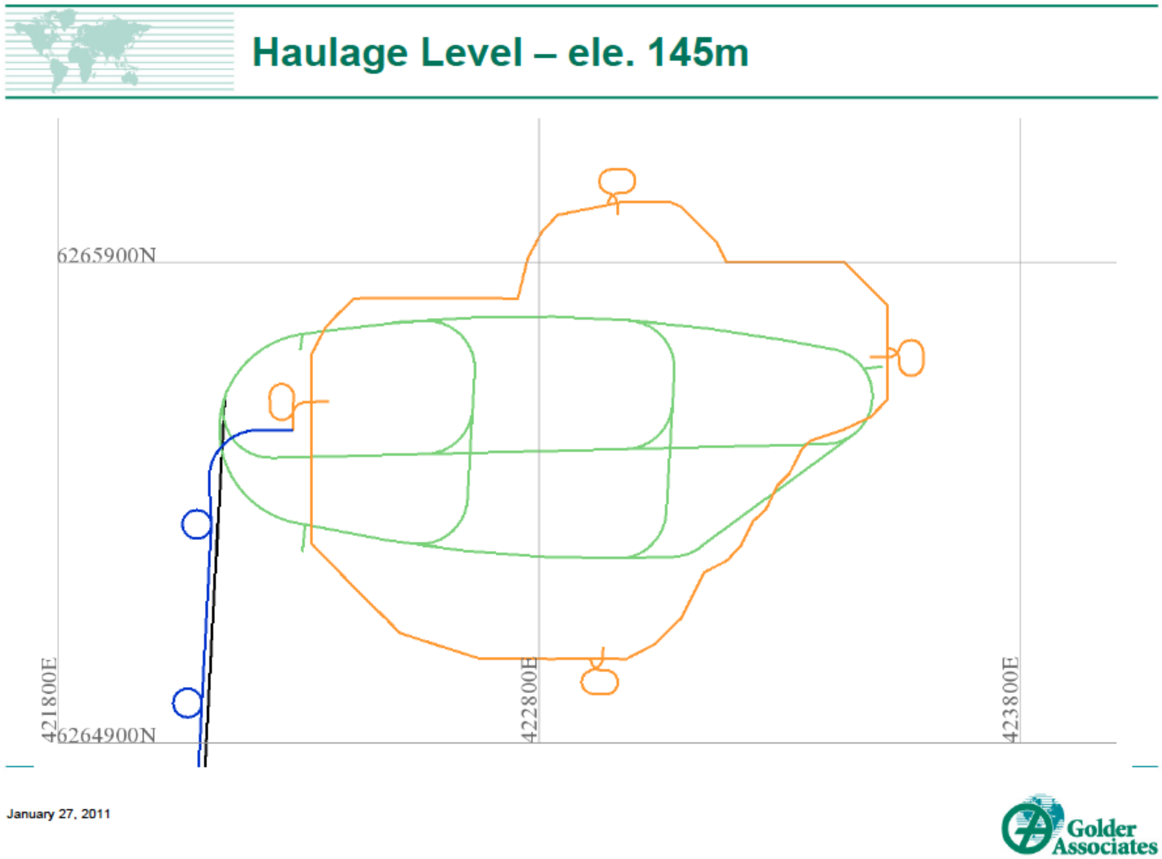
- Schalke trains – 1300 HP
- Haulage level is 5.5 km long

Power supply has been included in the electrical estimate for the trolley system as has the supply and installation of the trolley wire. The cost of the power rail has already been included with the cost of the locos, track, etc., by others.

The ore passes are designed with electric controls (rather than just manual air valves) in case the mine wants to move to an automated system in the future (this is an ideal case of an automated system).

Refer to mine cross section Figures 3.1 – 5, 3.7 -1, and 3.8 – 1.

Figure 3.8 - 1 Haulage Level



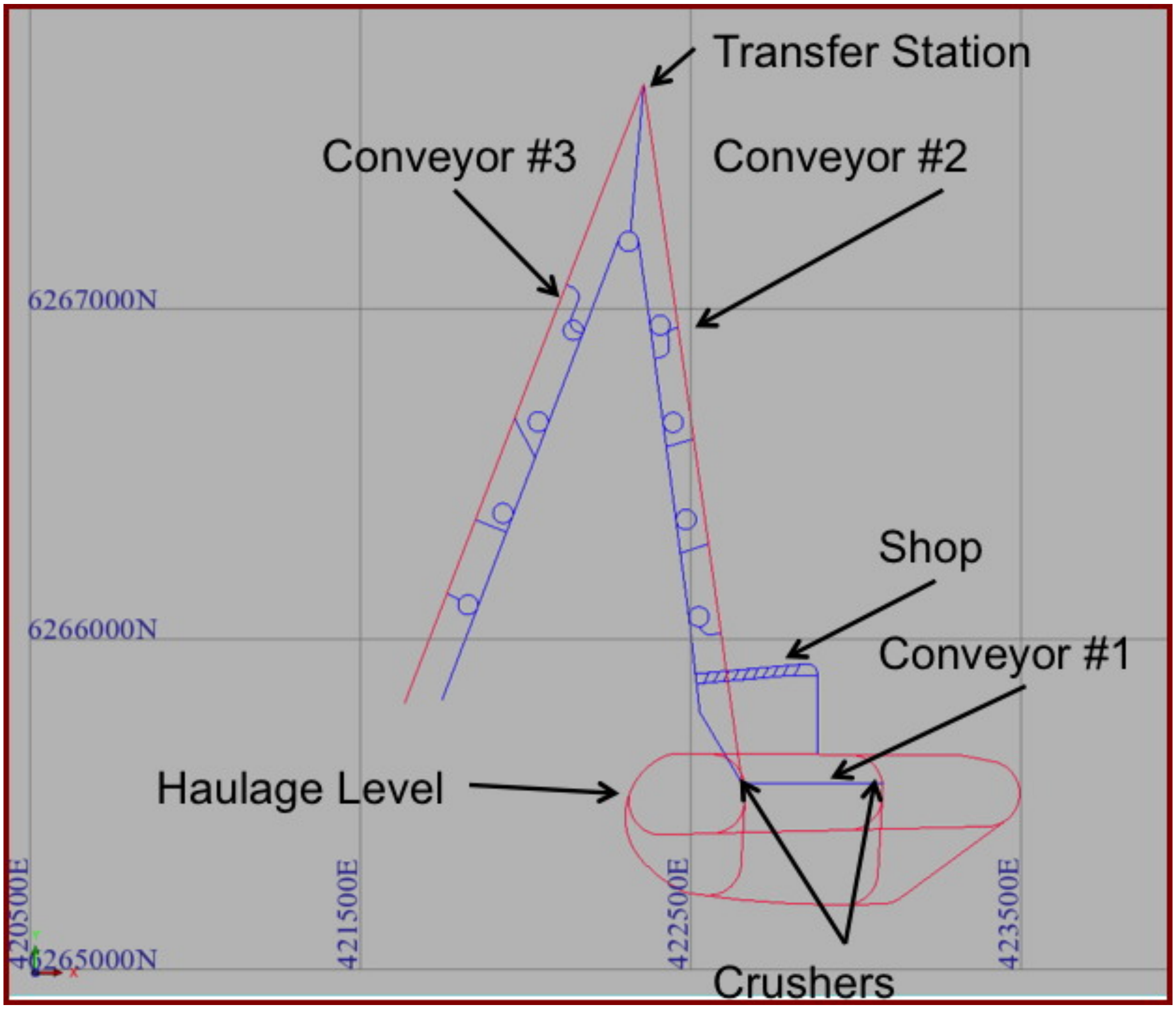
3.9 Conveyor Level (Starting At EL 119) Power And Control

Refer to Bosche Ventures drawings 10-10 -1611 “Overall Site Plan” and 99-10-007 and 007 “Flow Sheet 1” and “Flow Sheet 2” respectively.

Power Supply

Separate feeders and portable unit substations are provided for the conveying equipment that presents a large load. As noted, these cables would be separate from the mine ring power system, with the cables being run down the conveyor drifts.

Figure 3.9 – 1 Conveyors & Shop (From Golder)



Ancillary Loads

At each conveyor drive station a dry type step-down transformer and MCC, all underground mine type skid mounted, are provided for ancillary loads. In addition, lighting transformers and 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 and programming is included.

Conveyor Drives

Skid mounted unit substations and motor VFDs, auxiliary MCCs, etc. are included as detailed in the estimate.

Commissioning

The electrical costs associated with conveyor commissioning are included in the estimate.

3.10 Crusher Station Power And Control

General

There are two crushers. The two crusher stations will be separated approximately 400 metres. One will go in a few years later as production ramps up over about 5 years. The cost of the electrics for the second station is shown as sustaining capital in year 5.

Power Supply

The crusher stations would be fed off of the conveyor power radial system. A 1500 kVA, 25 kV to 4160 volt unit sub would be provided to run each of the crushers with a 1000 kVA, 600 volt unit sub to run the apron feeders, crusher auxiliaries, etc.

An electrical room is not required for the switchgear, as it's all underground mine type skid mount equipment.

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, PLC hardware and programming is included.

Crusher Drives

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

Control House

A pre-fabricated local control house, suitable for underground installation, is included in the estimates.

3.11 Mine Communications

The estimates include a leaky feeder system and a PED (through the earth) system.

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

3.12 Pumping

The Mitchell operation will be designed to have the mine water flow to a central collection point underground, then be pumped to surface.

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

3.13 Mine Ventilation Fans

There are three surface fans. Refer to Figure 3.1 - 4.

It is assumed that the fan motors are included with the fans by others, but this estimate includes the VFDs (variable speed drives) and a local substation (25 kV to 4160 volts) and E-house.

It is understood that mine air heating will likely be done with propane burners. Power supply from the vent fan E-house is included for this, but the remainder of this (substantial) installation is assumed to be by others.

The cost estimate includes bringing power from the mine ring main system to the vent fans and mine air heaters.

3.14 Engineering

The electrical estimate includes the cost of electrical design.

3.15 Construction Management, Purchasing, QA/QC and Commissioning

The electrical estimate includes the cost of construction management, purchasing, QA/QC and commissioning.

4.0 POWER CONSUMPTION

4.1 General

The plant load calculations are based on the load list provided by Golder.

4.2 Load Calculations

The below Excel spreadsheet includes an estimate of the total project running load. This includes all conveyors #1, 2, 3 and 4. It also includes the second crusher which comes on line 5 years after the start or production. Thus, for the first 5 years the load will be slightly less, by the amount of one crusher system. It is possible that the conveyor loads could also be different.

4.3 Annual Energy Consumption

The Load List (Figure 4.2 - 1) shows the total annual GW.h energy requirement for the mine. The cost of this power is to be included in the mine OPEX, using the per kilowatt hour power cost as per the published memo for the 2012 PFS update.

4.4 Mitchell Block Cave Load List

The estimated Mitchell electrical load is shown below.

Figure 4.4 - 1 Mitchell Load List And Calculations

LOAD NAME	HP	KW	VOLTS	AMPS X 1.25	CABLE "S"	CABLE "R"	QTY	TOTAL KW	EFFIC-IENCY %	DEMAND FACTOR	UTILIZATION %	RUNNING LOAD KW
					S = Spaced Fill R = Random Fill							
JUMBO		150					4	600	93%	0.8	75%	387
BOLTER		56					5	280	92%	0.8	70%	170
DRILLS		56					8	448	100%	0.8	70%	251
TRAIN - TROLLEY		1000					2	2000	100%	0.85	75%	1,275
RAISEBORE MACHINE		242					1	242	91%	0.75	75%	150
ROCKBREAKER		37					20	740	92%	0.8	65%	418
SHOTCRETER		56					2	112	91%	0.7	50%	43
PUMPS		56					6	336	92%	0.75	80%	219
SURFACE FANS	6903	5150	4160	1088	1/0		1	5150	94%	0.9	100%	4,931
U/G FANS	75	56	600	86.0	2		15	840	92%	0.9	75%	616
MAIN COMPRESS.	500	130	4160	27	2		2	260	100%	0.9	50%	117
MISC. HEATING		200	600				1	200	100%	1	50%	100
SURFACE BUILDINGS		60	600				1	60	100%	0.6	100%	36
APRON FEEDER (FROM CRUSH)	150	112	600				1	112	94%	0.75	90%	80
CONVEYOR 1		120	4160	24	2		1	120	94%	0.8	90%	92
CONVEYOR 1 DUST COLL. FAN	50	37.3	600				1	37	92%	0.95	90%	35
CONVEYOR 1 DUST COLL. SEAL	2	1.5	600				1	1	90%	0.9	90%	1
CV1 MAG	20	14.92	600	18.0		8	2	30	94%	0.8	90%	23
CV 1 SPILL FEEDERS	50	37.3	600	57.3	6	2	2	75	100%	0.7	10%	5
CONVEYOR 2		5025	4160	1097	2		1	5025	94%	0.8	90%	3,849
CV2 DUST COLLECOR	50	37.3	600	57.3	6	2	1	37	94%	0.9	90%	32
SEAL MOTOR	2	1.5	600				1	2	90%	0.9	90%	1
CV 2 SPILL FEEDERS	75	56	600	84.9	4		1	56	100%	0.7	10%	4
CV 2 BELT MAGNET	20	15	600	22.9	10	8	1	15	93%	0.9	90%	13
CV2 SPILL FEEDER	50	37	600				1	37	93%	0.85	10%	3
CONVEYOR 3		4600	4166	971	250		2	9200	100%	0.8	90%	6,624
CV3 DISCH FEEDER	150	111.9	600	166.2	2/0		1	112	100%	0.8	90%	81
CV3 DUST COLLECT.	50	37.3	600	57.3	6		1	37	100%	0.9	90%	30
CV 3 APRON FEEDER	200	152.8	600	221.5	250		1	153	100%	0.9	90%	124
CV3 AIR COMP	10	7.46	600	11.7		10	2	15	100%	0.9	90%	12
CV 3 BELT MAGNET	20	14.92	600	23.4		8	1	15	93%	0.9	90%	13
CV 3 SPILL FEDER	50	37.3	600	57.3	6		1	37	93%	0.9	70%	25
U/G CRUSHER NO. 1	550	410.3	4160	87	2		1	410	93%	0.9	70%	278
U/G CRUSH# 1 LUBE TOTAL	50	37.3	600	57.3	6		1	37	91%	0.86	70%	25
CRUSHER #1 CRANES	30	22.5	600	35.4		6	1	23	100%	0.5	10%	1
CRUSH#1 DUPLX COMP	15	11.19	600	17.6		10	2	22	93%	0.95	50%	11
CRUSH ROCK BREAKER	100	74.6	600	113.3	2		1	75	100%	0.7	5%	3
U/G CRUSHER NO. 2	550	410.3	600	630.0	6		1	410	100%	0.8	10%	33
U/G CRUSH# 2 LUBE TOTAL	50	37.3	600	57.3	6		1	37	100%	0.8	10%	3
CRUSHER #1 CRANES	30	22.38	600	34.4	6		1	22	100%	0.8	10%	2
CRUSH#2 DUPLX COMP	15	11.19	600	17.2	6		1	11	100%	0.8	10%	1
CRUSH #2 ROCK BREAKER	100	74.6	600	114.5	6		1	75	100%	0.8	10%	6
LTG & SMALL POWER		250					6	1500	100%	1	50%	750
HEAT TRACING		500					1	500	100%	1	35%	175
REFUGE STATIONS		30					7	210	100%	0.6	100%	126
U/G SHOP		150					1	150	100%	0.5	100%	75
MISC. MONO RAILS	5	3.7					6	22	91%	0.85	1%	0
MISC. SUMP PUMPS	25	18.7					6	112	91%	0.85	15%	16
									100%			
TOTALS								30,001 (connected)				**21,266** RUNNING LOAD, KW

ANNUAL GW.H = RUNNING LOAD X 8760 HRS/YR X PLANT AVAILABILITY (94%)

ENERGY USEAGE
ANNUAL GW.h = 175.11
MITCHELL BLOCK CAVE

5.0 ESTIMATE SUMMARY

5.1 General

A PFS level cost estimate was carried out and is summarized below. All sums are in first quarter 2012 Canadian dollars. Refer to the estimating spreadsheet for details including man-hours, etc.

The estimate sums below includes material, equipment and labour.

5.2 Main Power Distribution Into Mine

The main power supply estimate is:

Supply and install, labour, material and equipment: **$4,100,767**

5.3 Service Ramp

No operations phase electrical installation.

5.4 Pre-Conditioning Level Electrical Costs

Supply and install, labour, material and equipment: **$1,954,000**

5.5 Undercut Level

Supply and install, labour, material and equipment: **$1,499,400**

5.6 Extraction (Production) Level

Supply and install, labour, material and equipment: **$3,619,700**

5.7 Secondary Breaker Level

Supply and install, labour, material and equipment: **$1,889,425**

5.8 Haulage Level Electrical Costs

Supply and install, labour, material and equipment: **$1,801,550**

5.9 Conveyor Electrical Costs

Includes for supply and installation of power supply cables, switchgear, motor starters, grounding, lighting, instrumentation, and PLC controls, etc. for Conveyors 1 to 4 inclusive.

Supply and install, labour, material and equipment: **$10,711,041**

5.10 Crusher Station Electrical Costs

Includes for supply and installation of motor starters, switchgear, lighting, instrumentation, and controls, local pre-fab control room, etc.

Crusher Station 1 - Initial Crusher Installation

Supply and install, labour, material and equipment: **$2,554,879**

Crusher Station 2 - Sustaining Capital, Future Crusher Installation
(Not included in Total Initial capital cost)

Supply and install, labour, material and equipment: **$2,554,879**

5.11 Mine Communications Costs

Leaky Feeder and PED system.

Supply and install, labour, material and equipment: **$3,914,868**

5.12 Fire Pump Electrical

Supply and install, labour, material and equipment: **$220,000**

5.13 Mine Ventilation & Air Heating Electrical

Includes for supply and installation of motor starters, switchgear, lighting, instrumentation, and controls, E-house, etc.

Supply and install, labour, material and equipment: **$2,304,105**

5.14 Refuge Stations

Supply and install, labour, material and equipment:

Refuge Station No. 1: **$103,790**

Refuge Station No. 2: **$103,790**

Refuge Station No. 3: **$103,790**

Refuge Station No. 4: **$103,790**

Refuge Station No. 5: **$103,790**

Refuge Station No. 6: **$103,790**

Refuge Station No. 7: **$103,790**

5.15 Mine Air Compressor Power Supply

Supply and install, labour, material and equipment: **$513,035**

5.16 Engineering And Construction Management (Electrical)

Engineering, construction management and commissioning: **$7,588,500**

5.17 Grand Total (Electrical, instrumentation & Communications)

<u>Initial Direct Installation Cost</u>

Supply and install, labour, material and equipment: **$35,809,299**

<u>Initial Indirect Installation Cost</u>

Indirect costs, Electrical: **$7,588,500**
(From above.)

<u>Grand Total, Electrical, Instrumentation And Communications</u>

Total Electrical: **$43,397,799**

<u>Sustaining Capital Total Cost</u>

Crusher Station 2, sustaining capital, 5 years after startup: **$2,554,879**

W.N. Brazier

W.N. Brazier, P.Eng.

PROFESSIONAL PROVINCE OF W. N. BRAZIER BRITISH COLUMBIA ENGINEER

WNB

28 March 2012

6.0 APPENDIX 1 – PROJECT ESTIMATE SPREADSHEET

The following spreadsheet includes all details of the estimate.

KSM MITCHELL BLOCK CAVE 2012 PFS ELECTRICAL COST ESTIMATE

NOTES

1) COSTS IN 1ST QUATER 2012 CANADIAN DOLLARS.
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	$112						-		-	-	-	-	-	
COST CODE 1 - CONVEYORS CV1, CV2, CV3 & CV4					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
CONVEYOR POWER, CONTROL & INSTRUMENTATION					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
MAIN FEEDER CABLE FOR CONVEYORS, 3C, 25kV, 133%, 350 MCM (part of ring main)	5,500	M		0.52	1.25	$105	2.0	195.0				3,575	375,375	1,072,500	-	11,000	-	1,458,875	
												-		-	-	-	-	-	
(Ring main (circuit breaker) units, unit substations, etc. are included with each conveyor)												-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
HV Junction Boxes, 11T150 (PATTON AND COOKE)	5	lot		30	1.25	$112		500	6,410			188	21,000	2,500	32,050	-	-	55,550	
					1.25	$112						-		-	-	-	-	-	
Conv. 1 (Crusher 1 to Crusher 2, 430 m, 48" belt)					1.25	$112						-		-	-	-	-	-	
(Note, power supply is from Crusher)												-		-	-	-	-	-	
120 kW drive at Crusher 2, motor by others, terminations, etc.(200 HP @ 600V, 120 kW motors do not exist)	1			8.00	1.25	$112		500.0				10	1,120	500	-	-	-	1,620	
Assume this conveyor has a fluid coupling to start.					1.25	$112						-		-	-	-	-	-	
600 volt VFD fed from crusher switchgear, U/G type	1	ea		100.00	1.25	$112		1,500.0	30,000			125	14,000	1,500	30,000	-	-	45,500	
						$112						-		-	-	-	-	-	
Power cable, 3#250, 600V/1kV Teck	150	m		0.20	1.25	$112		160.0				38	4,200	24,000	-	-	-	28,200	
Control Cable		m			1.25	$112						-		-	-	-	-	-	
3c # 14 Teck	250	m		0.13	1.25	$112		2.85				41	4,550	713	-	-	-	5,263	
8c # 14 Teck	580	m		0.15	1.25	$112		9.20				109	12,180	5,336	-	-	-	17,516	
Cable Terminations	1	lot		50.00	1.25	$112		2,500.0				63	7,000	2,500	-	-	-	9,500	
Pilot Devices																			
Pull cord switches	17	ea		4.00	1.25	$112			500			86	9,632	-	8,600	-	-	18,232	
Belt Misc.	6			4.00	1.25	$112			500			30	3,360	-	3,000	-	-	6,360	
Speed Sw.	1	ea			1.25	$112			1,500			-		-	1,500	-	-	1,500	
Chute plugged	3	ea		4.00	1.25	$112		50.0	750			15	1,680	150	2,250	-	-	4,080	
Boxes, etc.	1	lot		20.00	1.25	$112		1,500.0				25	2,800	1,500	-	-	-	4,300	
Cable Tray, 6 inch	500	m		0.45	1.25	$112		75.0				281	31,500	37,500	-	-	-	69,000	
Cable Tray 12 inch	100	m		0.60	1.25	$112		82.0				75	8,400	8,200	-	-	-	16,600	
Cable Tray 18 inch	150	m		0.70	1.25	$112		95.0				131	14,700	14,250	-	-	-	28,950	
Belt rip detector	1	ea		16.00		$112	200.0	3,500.0				-		3,500	-	200	-	3,700	
Support material (channels, Unistrut, etc.)	1	lot		100.00	1.25	$112		7,500.0				125	14,000	7,500	-	-	-	21,500	
					1.25	$112						-		-	-	-	-	-	
2 Pair AIC (from speed switch)	550	m		0.15	1.25	$112		3.75				103	11,550	2,063	-	-	-	13,613	
Terminations	6	m		1	1.25	$112		15.00				8	840	90	-	-	-	930	
					1.25	$112						-		-	-	-	-	-	
Equipment rental	1	lot			1.25	$112		5,000.0				-		5,000	-	-	-	5,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Conv. 2 (Crusher 2 to Conveyor 3, 2200 m, 54" belt)					1.25	$112						-		-	-	-	-	-	
3 drives, 5025 kW total, motors by others.(Assume 3 * 2250 HP motors @ 4000V)					1.25	$112						-		-	-	-	-	-	
Note, motor supply & installation by Mechanical, all electrical labour for connection is in this budget.		ea			1.25	$112						-		-	-	-	-	-	
VFD, 25 KV input, c/w transformer down to 4160 volts, U/G mine type, skid mount, quick connect	3	ea		100.00	1.25	$112	2,500	41,700	417,000			375	42,000	125,100	1,251,000	7,500	-	1,425,600	
25 kV to 600 volt unit substations, 500 KVA each (at mid drive point and head end)	2	ea		100.00	1.25	$112		8,250.0	165,000			250	28,000	16,500	330,000	-	-	374,500	
Ring Main Switches	2	ea		50.00	1.25	$112	750.0		113,000			125	14,000	-	226,000	1,500	-	241,500	
					1.25	$112						-		-	-	-	-	-	
Power cable for moors, 500 MCM, 5 kV Teck	450	m		0.50	1.25	$112		260.0				281	31,500	117,000	-	-	-	148,500	

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
Control Cable		m			1.25	$112						-		-	-	-	-	-	
3c # 14 Teck	2500	m		0.13	1.25	$112		2.85				406	45,500	7,125	-	-	-	52,625	
8c # 14 Teck	2000	m		0.15	1.25	$112		9.20				375	42,000	18,400	-	-	-	60,400	
20c # 14 Teck	1000	m		0.2	1.25	$112		14.90				250	28,000	14,900	-	-	-	42,900	
Cable Terminations	1	lot		200.00	1.25	$112		5,000.0				250	28,000	5,000	-	-	-	33,000	
Pilot Devices																			
Pull cord switches	74	ea		4.00	1.25	$112			500			372	41,627	-	37,167	-	-	78,793	
Belt Misc.	12			4.00	1.25	$112			500			60	6,720	-	6,000	-	-	12,720	
Speed Sw.	3	ea		20.00	1.25	$112		100.0	1,500			75	8,400	300	4,500	-	-	13,200	
Chute plugged	3	ea		4.00	1.25	$112		50.0	750			15	1,680	150	2,250	-	-	4,080	
Boxes, etc.	1	lot		100.00	1.25	$112		5,000.0				125	14,000	5,000	-	-	-	19,000	
Cable Tray, 6 inch	500	m		0.45	1.25	$112		75.0				281	31,500	37,500	-	-	-	69,000	
Cable Tray 12 inch	2,500	m		0.60	1.25	$112		82.0				1,875	210,000	205,000	-	-	-	415,000	
Cable Tray 18 inch	150	m		0.70	1.25	$112		95.0				131	14,700	14,250	-	-	-	28,950	
(Note, ppull cord connections via Teck cable strapped to conveyor tables)						$112						-		-	-	-	-	-	
Support material (channels, Unistrut, etc.)	1	lot		200.00	1.25	$112		14,000.0				250	28,000	14,000	-	-	-	42,000	
Belt rip detector	2	ea		16.00	1.25	$112	200.0	3,500.0				40	4,480	7,000	-	400	-	11,880	
2 Pair AIC (from speed switches)	3000	m		0.15	1.25	$112		3.75				563	63,000	11,250	-	-	-	74,250	
Terminations	10	m		2	1.25	$112		15.00				25	2,800	150	-	-	-	2,950	
					1.25	$112						-		-	-	-	-	-	
Equipment rental	1	lot			1.25	$112		7,500.0				-		7,500	-	-	-	7,500	
					1.25	$112						-		-	-	-	-	-	
Conv. 2 Discharge Dust Collector					1.25	$112						-		-	-	-	-	-	
Fan, 50 HP, starter in 500 kVA unit sub					1.25	$112						-		-	-	-	-	-	
Power cable, 3c#6 Teck	200	m		0.15	1.25	$112		19.3				38	4,200	3,860	-	-	-	8,060	
Aux drive (air Lock)					1.25	$112						-		-	-	-	-	-	
Power cable 3C#12	250	m		0.10	1.25	$112		5.8				31	3,500	1,450	-	-	-	4,950	
Air Compressor cable 2* # 10	300	m		0.10	1.25	$112		7.8				38	4,200	2,340	-	-	-	6,540	
					1.25	$112						-		-	-	-	-	-	
Conv 2 Belt magnebt motor cable, 3#10	200	m		0.10		$112		7.8				-		1,560	-	-	-	1,560	
Belt magnet rectifier cable, 3c#10	200	m		0.10		$112		7.8				-		1,560	-	-	-	1,560	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Conv. 3 (Crusher 2 to Conveyor 4, 2000 m, 54" belt)					1.25	$112						-		-	-	-	-	-	
3 drives total 4600 kW total, motors by others. (Assume 3 * 2100 HP motors @ 4000V)	2,055				1.25	$112						-		-	-	-	-	-	
Note, motor installation by Mechanical, all electrical labour for connection is in this budget.					1.25	$112						-		-	-	-	-	-	
VFD, 25 KV input, c/w transformer down to 4160 volts	3	ea		100.00	1.25	$112	2,500	41,700	417,000			375	42,000	125,100	1,251,000	7,500	-	1,425,600	
25 kV to 600 volt, 500 kVA unit substations at mid drive point.	1	ea		100.00	1.25	$112		8,250.0	165,000			125	14,000	8,250	165,000	-	-	187,250	
25 kV to 600 volt, 1000 kVA unit substations at conv. 3 head end, also feeds conv. 4	1	ea		125.00	1.25	$112		9,250.0	185,000			156	17,500	9,250	185,000	-	-	211,750	
Ring Main Switches	2	ea		50.00	1.25	$112	750.0		113,000			125	14,000	-	226,000	1,500	-	241,500	
					1.25	$112						-		-	-	-	-	-	
Power cable, 500 MCM, 5 kV Teck	150	m		0.20	1.25	$112		45.0				38	4,200	6,750	-	-	-	10,950	
Control Cable		m			1.25	$112						-		-	-	-	-	-	
3c # 14 Teck	3500	m		0.13	1.25	$112		2.85				569	63,700	9,975	-	-	-	73,675	
8c # 14 Teck	2000	m		0.15	1.25	$112		9.20				375	42,000	18,400	-	-	-	60,400	
20c # 14 Teck	1000	m		0.2	1.25	$112		14.90				250	28,000	14,900	-	-	-	42,900	
Cable Terminations	1	lot		200.00	1.25	$112		5,000.0				250	28,000	5,000	-	-	-	33,000	
Pilot Devices																			
Pull cord switches	68	ea		4.00	1.25	$112			500			338	37,893	-	33,833	-	-	71,727	
Belt Misc.	10			4.00	1.25	$112			500			50	5,600	-	5,000	-	-	10,600	
Speed Sw.	3	ea		20.00	1.25	$112		100.0	1,500			75	8,400	300	4,500	-	-	13,200	
Chute plugged	3	ea		4.00	1.25	$112		50.0	750			15	1,680	150	2,250	-	-	4,080	
Boxes, etc.	1	lot		100.00	1.25	$112		5,000.0				125	14,000	5,000	-	-	-	19,000	
Cable Tray, 6 inch	500	m		0.45	1.25	$112		75.0				281	31,500	37,500	-	-	-	69,000	
Cable Tray 12 inch	1,500	m		0.60	1.25	$112		82.0				1,125	126,000	123,000	-	-	-	249,000	
Cable Tray 18 inch	150	m		0.70	1.25	$112		95.0				131	14,700	14,250	-	-	-	28,950	
Belt rip detector	2	ea		16.00	1.25	$112		200.0	3,500			40	4,480	400	7,000	-	-	11,880	
Support material (channels, Unistrut, etc.)	1	lot		200.00	1.25	$112		14,000.0				250	28,000	14,000	-	-	-	42,000	
Belt Scale	1	ea		150.00	1.25	$112		2,100.0	21,000			188	21,000	2,100	21,000	-	-	44,100	
Metal Detector	1	ea		100.00	1.25	$112		1,250.0	12,500			125	14,000	1,250	12,500	-	-	27,750	
					1.25	$112						-		-	-	-	-	-	
2 Pair AIC (from speed switch)	3500	m		0.15	1.25	$112		3.75				656	73,500	13,125	-	-	-	86,625	
Terminations	6	m		1	1.25	$112		15.00				8	840	90	-	-	-	930	
					1.25	$112						-		-	-	-	-	-	
Equipment rental	1	lot			1.25	$112	7500.0					-		-	-	7,500	-	7,500	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Conv. 4 (Conv. 1 to stockpile feed conveyor, 135 m, 54" belt)					1.25	$112						-		-	-	-	-	-	

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
60 kW (100 HP) drive, motor by others (100 HP @ 600 V, 60 kW motors do not exist) Power supply from 600 volt unit sub at conv. 3 head end	1			8.00	1.25	$112		500.0				10	1,120	500	-	-	-	1,620	
Feeder cable, 3c#2 AWG	200	m	0.19		1.25	$112		24.5			38	-		4,900	-	-	-	4,900	
600 V, 100 HP VFD	1	ea		75.00	1.25	$112		0.0				94	10,500	-	-	-	-	10,500	
Ring Main Switches	2	ea		100.00	1.25	$112	750.0		113,000			250	28,000	-	226,000	1,500	-	255,500	
Conv 3 Belt magnebt motor cable, 3#10	200	m		0.10		$112		7.8				-		1,560	-	-	-	1,560	
Belt magnet rectifier cable, 3c#10	200	m		0.10		$112		7.8				-		1,560	-	-	-	1,560	
		m		0.17		$112						-		-	-	-	-	-	
						$112						-		-	-	-	-	-	
Belt Feeder, 45 kW (60 HP), 3c#4 Teck cable	150	m		0.17		$112		17.0				-		2,550	-	-	-	2,550	
		m			1.25	$112						-		-	-	-	-	-	
Control Cable		m			1.25	$112						-		-	-	-	-	-	
3c # 14 Teck	350	m		0.13	1.25	$112		2.85				57	6,370	998	-	-	-	7,368	
8c # 14 Teck	580	m		0.15	1.25	$112		9.20				109	12,180	5,336	-	-	-	17,516	
Cable Terminations	1	lot		50.00	1.25	$112		2,500.0				63	7,000	2,500	-	-	-	9,500	
Pilot Devices																			
Pull cord switches	9	ea		4.00	1.25	$112						47	5,264	-	-	-	-	5,264	
Chute plugged	3	ea		4.00	1.25	$112		50.0	750			15	1,680	150	2,250	-	-	4,080	
Belt Misc.	4	ea		4.00	1.25	$112		500.0				20	2,240	2,000	-	-	-	4,240	
Speed Sw.	2	ea			1.25	$112		1,500.0				-		3,000	-	-	-	3,000	
Boxes, etc.	1	lot		50.00	1.25	$112		1,200.0				63	7,000	1,200	-	-	-	8,200	
Cable Tray, 6 inch	75	m		0.45	1.25	$112		75.0				42	4,725	5,625	-	-	-	10,350	
Cable Tray 12 inch	100	m		0.60	1.25	$112		82.0				75	8,400	8,200	-	-	-	16,600	
Cable Tray 18 inch	150	m		0.70	1.25	$112		95.0				131	14,700	14,250	-	-	-	28,950	
belt rip detector	1	ea		16.00	1.25	$112	200.0	3,500.0				20	2,240	3,500	-	200	-	5,940	
Support material (channels, Unistrut, etc.)	1	lot		75.00	1.25	$112		6,500.0				94	10,500	6,500	-	-	-	17,000	
					1.25	$112						-		-	-	-	-	-	
2 Pair AIC (from speed switch)	400	m		0.15	1.25	$112		3.75				75	8,400	1,500	-	-	-	9,900	
Terminations	6	m		1	1.25	$112		15.00				8	840	90	-	-	-	930	
					1.25	$112						-		-	-	-	-	-	
Equipment rental	1	lot			1.25	$112	3500.0					-		-	-	3,500	-	3,500	
						$112													
Instruments (bin level, chute plugged, etc.)	1	lot		100.00	1.25			9,500.0				125	14,000	9,500	-	-	-	23,500	
					1.25	$112						-		-	-	-	-	-	
Conveyor Drive Station and Crusher						$112													
Area Lighting					1.25							-		-	-	-	-	-	
(At drive stations and substations						$112													
only)					1.25							-		-	-	-	-	-	
Fixtures, 250 W HPS	120	ea		6	1.25	$112		400				900	100,800	48,000	-	-	-	148,800	
Lighting transformer and panel	6	lots		200	1.25	$112		5,500	12,000			1,500	168,000	33,000	72,000	-	-	273,000	
Lighting cable, 4c # 12	9000	m		0.05	1.25	$112		3.31				563	63,000	29,790	-	-	-	92,790	
Connectors, outlets and hardware	6	lot		100	1.25	$112		3,500				750	84,000	21,000	-	-	-	105,000	
(Note, lighting along the length of						$112													
the conveyor					1.25							-		-	-	-	-	-	
is not included.)					1.25	$112						-		-	-	-	-	-	
Grounding, Conveyors and Crusher						$112													
Areas					1.25							-		-	-	-	-	-	
Ground wire, 2/0 AWG	10000	m		0.05	1.25	$112	0.15	6.90				625	70,000	69,000	-	1,500	-	140,500	
Ground Connectors, compression	333	ea		0.5	1.25	$112		20.00				208	23,333	6,667	-	-	-	30,000	
Ground wire, 4/0	1000	m		0.15	1.25	$112	0.30	10.50				188	21,000	10,500	-	300	-	31,800	
					1.25	$112						-		-	-	-	-	-	
CCTV	1	lot		750	1.25	$112	5,000	8,000	65,000			938	105,000	8,000	65,000	5,000	-	183,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Fire System - Allowance for power to fire detection & alarm system (detection and alarms by others)	1	lot		300	1.25	$112	1,250	25,000				375	42,000	25,000	-	1,250	-	68,250	
This is to provide 120 volt power only.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
PLC system					1.25	$112						-		-	-	-	-	-	
PLC Hardware as per quote	1	lot		2000	1.25	$112	15,000	45,700	457,000			2,500	280,000	45,700	457,000	15,000	-	797,700	
					1.25	$112						-		-	-	-	-	-	
PLC programming	1	lot		1200	1.25	$140	10,000	10,000				1,500	210,000	10,000	-	10,000	-	230,000	
					1.25	$112						-		-	-	-	-	-	
Fibre Optic cable & messenger, etc.	12000	m		0.08	1.25	$112	0.3	4				1,200	134,400	48,000	-	3,600	-	186,000	
For mine communication system see					1.25	$112						-		-	-	-	-	-	
leaky feeder and PED estimate					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Allowance for control room						$112													
communication over fibre optic system (VoIP)					1.25							-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	

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
		lot		500.00	1.25	$112		6,500.0	25,000			-		-	-	-	-	-	
						$112						-		-			-	-	Area Subtotal
SUBTOTALS, COST CODE 1, CONVEYORS 1, 2, 3 & 4 ELECTRICAL AND INSTRUMENTATION											38	29,468	3,317,379	2,645,061	4,669,650	78,950	-		$10,711,041
COST CODE 2 - MAIN POWER DISTRIBUTION- MINE RING MAIN												-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
						$112													
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							-		-		-	-	-	
TWO RUNS OF RING MAIN FEEDER CABLESS ACROSS COUNTRY FROM MITCHELL NO. 2 SUBBSTATION TO THE UNDERGROUND MINE. , 25KV, 133%, 3c 350 MCM, TECK 90	2,400	m		0.45	1.25	$112	2.0	195.0				1,350	151,200	468,000	-	4,800	-	624,000	
CABLE DOWN BORE HOLES, 25KV, 133%, 3c 350 MCM, STEEL WIRE ARMOURED (Note, Mitchell plant elevation is at about 820 m while the mine extraction level is El. 235)	1,200	m		0.45	1.25	$112	2.0	210.0				675	75,600	252,000	-	2,400	-	330,000	
MESSENGER (For above cable, rock						$112	1.0												
bolts by others)	2,400	m		0.15	1.25			4.0				450	50,400	9,600	-	2,400	-	62,400	
					1.25	$112						-		-	-	-	-	-	
		m		0.45	1.25	$112	3.0	195.0				-		-	-	-	-	-	
						$112	3.0												
PERIMETER DRIFT, 2266 m, 25 kV, 3c 350 MCM	3,100	m		0.45	1.25			195.0				1,744	195,300	604,500	-	9,300	-	809,100	
						$112	3.0												
UNDERCUT LEVEL, 750 m, 25 kV, 3c 350 MCM		m		0.45	1.25			195.0				-		-	-	-	-	-	
						$112	3.0												
PRECONDITIONING LEVEL, 750 m, 25 kV, 3c 350 MCM		m		0.45	1.25			195.0				-		-	-	-	-	-	
2/0 Cable, 5 kV	0	m		0.45	1.25	$112	3.0	115.0				-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
						$112	5.0												
Messenger, ROCK BOLTS & clips for cables in drifts	3,100	M		0.15	1.25			10.0				581	65,100	31,000	-	15,500	-	111,600	
Terminations	1	lot		950.00	1.25	$112	3000.0	30,000.0				1,188	133,000	30,000	-	3,000	-	166,000	
HV Junction Boxes, etc.	1	lot		300.00	1.25	$112	1500.0	26,000.0				375	42,000	26,000	-	1,500	-	69,500	
(4783 m long drift)					1.25	$112						-		-	-	-	-	-	
RING MAIN (CIRCUIT BREAKER) UNITS (for mine equipment, ring main units for conveyors, etc. shown in other areas)	10	EA		100.00	1.25	$112	1000.0	2,000.0	113,000			1,250	140,000	20,000	1,130,000	10,000	-	1,300,000	
					1.25	$112						-		-	-	-	-	-	
UNIT SUB, 25 KV-600V, 1500 KVA (to supply mining equipment, fan, conveyor and crusher unit subs shown in other areas)		EA		180.00	1.25	$112	2500.0	9,450.0	189,000			-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Grounding					1.25	$112						-		-	-	-	-	-	
Ground wire, 2/0 AWG	5,000	m		0.07	1.25	$112	$2.0	7.00				438	49,000	35,000	-	10,000	-	94,000	
Ground Connectors, compression	167	ea		0.75	1.25	$112		30.00				156	17,500	5,000	-	-	-	22,500	
Ground wire, 4/0	5,500	m		0.1	1.25	$112	$2.0	11.00				688	77,000	60,500	-	11,000	-	148,500	
Ground Connectors, compression	183	ea		0.75	1.25	$112		35.0				172	19,250	6,417	-	-	-	25,667	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
HEAT TRACE ALLOWANCE FOR WATER PIPES IN MAIN DRIFTS (excludes pipe insulation)	1	LOT		1500.00	1.25	$112	2500.0	50,000.0	75,000			1,875	210,000	50,000	75,000	2,500	-	337,500	
					1.25	$112						-		-	-	-	-	-	
						$112								-			-	-	Area Subtotal
SUBTOTALS, COST CODE 2, GENERAL MINE POWER DISTRIBUTION												10,941	1,225,350	1,598,017	1,205,000	72,400	-		$4,100,767
						$112								-					
COST CODE 3 - FIRE PUMP POWER SUPPLY					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
ALLOWANCE FOR POWER					1.25	$112						-		-	-	-	-	-	
SUPPLY FOR FIRE PUMPS	1	lot		500.00	1.25	$112		50,000.0	100,000			625	70,000	50,000	100,000	-	-	220,000	
INCLUDING POWER SUPPLY					1.25	$112						-		-	-	-	-	-	
(No reliable details available)					1.25	$112						-		-	-	-	-	-	
THIS DOES NOT INCLUDE THE					1.25	$112						-		-	-	-	-	-	
PUMPS, MOTOR, FIRE					1.25	$112						-		-	-	-	-	-	
PUMP CONTROLLERS OR					1.25	$112						-		-	-	-	-	-	
POSSIBLE HEAT TRACE.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
						$112								-			-	-	Area Subtotal
SUBTOTAL COST CODE 3 - FIRE PUMP POWER SUPPLY											-	625	70,000	50,000	100,000	-	-		$220,000
COST CODE 4 PRECONDITIONING																			

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
POWER SUPPLY PRE-CONDITIONING					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
NOTE, 25 KV RING MAIN CABLE IS IN COST CODE 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	3,500	m		0.49	1.25	$112	2.0	120.0				2,144	240,100	420,000	-	7,000	-	667,100	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING SUPPLY	4	ea		75.00	1.25	$112	1000.0	2,000.0	113,000			375	42,000	8,000	452,000	4,000	-	506,000	
					1.25	$112						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600 VOLTS, 1500 kva	2	ea		100.00	1.25	$112	2500.0	9,450.0	189,000			250	28,000	18,900	378,000	5,000	-	429,900	
END (DISTRIBUTION) BOXES	10	ea		10.00	1.25	$112		950.0	9,500			125	14,000	9,500	95,000	-	-	118,500	
600 VOLT TECK CABLE 3c 250 MCM	1,000	m		0.20	1.25	$112	2.0	165.0				250	28,000	165,000	-	2,000	-	195,000	
and MESSENGER					1.25	$112						-		-	-	-	-	-	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
TERMINATIONS, ETC.	1	lot		150.00	1.25	$112	1500.0	15,000.0				188	21,000	15,000	-	1,500	-	37,500	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 4 PRE-CONDITIONING											-	3,331	373,100	636,400	925,000	19,500	-		$1,954,000
COST CODE 5 UNDERCUT LEVEL																			
POWER SUPPLY - UNDERCUT					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
NOTE, 25 KV RING MAIN CABLE IS IN COST CODE 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	2,500	m		0.49	1.25	$112	2.0	120.0				1,531	171,500	300,000	-	5,000	-	476,500	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING SUPPLY	2	ea		75.00	1.25	$112	1000.0	2,000.0	113,000			188	21,000	4,000	226,000	2,000	-	253,000	
					1.25	$112						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600 VOLTS, 1500 kva	2	ea		100.00	1.25	$112	2500.0	9,450.0	189,000			250	28,000	18,900	378,000	5,000	-	429,900	
END (DISTRIBUTION) BOXES	10	ea		10.00	1.25	$112		950.0	9,500			125	14,000	9,500	95,000	-	-	118,500	
600 VOLT TECK CABLE 3c 250 MCM	1,000	m		0.20	1.25	$112	2.0	165.0				250	28,000	165,000	-	2,000	-	195,000	
and MESSENGER					1.25	$112						-		-	-	-	-	-	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
TERMINATIONS, ETC.	1	lot		100.00	1.25	$112	1500.0	11,000.0				125	14,000	11,000	-	1,500	-	26,500	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 5 UNDERCUT LEVEL											-	2,469	276,500	508,400	699,000	15,500	-		$1,499,400
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
COST CODE 6 - EXTRACTION (PRODUCTION) LEVEL					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
POWER SUPPLY PRE-CONDITIONING					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
RING MAIN CABLE AND RING MAIN (BREAKER) UNITS ARE IN COST CODE 2					1.25	$112						-		-				-	
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	2,000	m		0.49	1.25	$112	2.0	120.0				1,225	137,200	240,000	-	4,000	-	381,200	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING SUPPLY	4	ea		75.00	1.25	$112	1000.0	2,000.0	113,000			375	42,000	8,000	452,000	4,000	-	506,000	
					1.25	$112						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600 VOLTS, 1000 kva	4	ea		100.00	1.25	$112	2500.0	8,750.0	175,000			500	56,000	35,000	700,000	10,000	-	801,000	
END BOXES	12	ea		10.00	1.25	$112		850.0	8,500			150	16,800	10,200	102,000	-	-	129,000	
MISC. 5 KV CABLE, ETC.	0	lot			1.25	$112						-		-	-	-	-	-	
600 VOLT TECK CABLE 3c 250 MCM					1.25	$112													
and MESSENGER	1,200	m		0.20				165.0				300	33,600	198,000	-	-	-	231,600	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
TERMINATIONS, ETC.	1	lot		125.00	1.25	$112	1500.0	15,000.0				156	17,500	15,000	-	1,500	-	34,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 6 EXTRACTION (PRODUCTION) LEVEL											-	11,163	1,250,200	2,174,400	3,577,000	71,500	-		$3,619,700

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 7 SECONDARY BREAKER LEVEL																			
POWER SUPPLY - SECONDARY BREAKER LEVEL					1.25	$112								-	-	-	-	-	
												-							
					1.25	$112						-		-	-	-	-	-	
NOTE, 25 KV RING MAIN CABLE IS IN COST CODE 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
LOCAL DRIFT CABLES, 25 KV, 3C#2/0	2,250	m		0.30	1.25	$112	2.0	120.0				844	94,500	270,000	-	4,500	-	369,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING					1.25	$112	1000.0	2,000.0	113,000										
SUPPLY	3	ea		75.00								281	31,500	6,000	339,000	3,000	-	379,500	
					1.25	$112						-		-	-	-	-	-	
UNIT SUBSTATIONS, 25 KV - 600					1.25	$112	2500.0	8,075.0	161,500										
VOLTS, 500 kva	3	ea		100.00								375	42,000	24,225	484,500	7,500	-	558,225	
END (DISTRIBUTION) BOXES	20	ea		10.00	1.25	$112		950.0	9,500			250	28,000	19,000	190,000	-	-	237,000	
600 VOLT TECK CABLE 3c 6	4,000	m		0.15	1.25	$112	2.0	19.3				750	84,000	77,200	-	8,000	-	169,200	
					1.25	$112						-		-	-	-	-	-	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$112						-		-	-	-	-	-	
Messenger c/w clips, rockbolts, etc.	6,250	m		0.10	1.25	$112		10.0				781	87,500	62,500	-	-	-	150,000	
TERMINATIONS, ETC.	1	lot		100.00	1.25	$112	1500.0	11,000.0				125	14,000	11,000	-	1,500	-	26,500	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 7 SECONDARY BREAKER LEVEL											-	3,406	381,500	469,925	1,013,500	24,500	-		$1,889,425
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
COST CODE 8 - HAULAGE LEVEL																			
POWER SUPPLY - EXTRACTION					1.25	$112						-		-	-	-	-	-	
POWER SUPPLY IS FOR LOCOMOTIVE AC TROLLEY SYSTEM					1.25	$112						-		-	-	-	-	-	
NOTE, 25 KV 350 MCM RING MAIN CABLE IS IN COST CODE 2					1.25	$112						-		-	-	-	-	-	
LOCAL DRIFT CABLES TO UNIT SUBS,					1.25	$112	2.0												
25 KV, 3C#2/0	500	m		0.49				120.0				306	34,300	60,000	-	1,000	-	95,300	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS FOR MINING					1.25	$112	1000.0	2,000.0	113,000										
SUPPLY	2	ea		75.00								188	21,000	4,000	226,000	2,000	-	253,000	
					1.25	$112						-		-	-	-	-	-	
UNIT SUBSTATIONS, TROLLEY					1.25	$112	2500.0	9,450.0	189,000										
SUPPLY, 25 KV - Stepdown, 1500 kva	2	ea		100.00								250	28,000	18,900	378,000	5,000	-	429,900	
END BOXES		ea		10.00	1.25	$112		850.0	8,500			-		-	-	-	-	-	
	0	lot			1.25	$112						-		-	-	-	-	-	
LOW VOLTAGE TECK CABLE					1.25	$112													
	500	m		0.25				165.0				156	17,500	82,500	-	-	-	100,000	
NOTE, 600 VOLT TRAILING CABLE TO EQUIPMENT ASSUMED TO BE IN MINING COSTS WITH EQUIPMENT.					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
TERMINATIONS, ETC.	1	lot		150.00	1.25	$112	1500.0	15,000.0				188	21,000	15,000	-	1,500	-	37,500	
Rock bolts, cement grouted, 6 ft	1,100	EA		0.20	1.25	$112	5.0	50.0				275	30,800	55,000	-	5,500	-	91,300	
Fibre optic communication cable with					1.25	$112	5.0												
messenger, rock bolts, clips, etc.	5,500							15.0				-		82,500	-	27,500	-	110,000	
Communication equipment, terminations,					1.25	$112	1500.0												
etc.	1	lot		300.00				2,500.0	25,000			375	42,000	2,500	25,000	1,500	-	71,000	
Local controls and terminations	1	lot		750.00	1.25	$112	1500.0	7,000.0	50,000			938	105,000	7,000	50,000	1,500	-	163,500	
Local control (Teck) cable	6,000	m			1.25	$112		7.8				-		46,800	-	-	-	46,800	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Trolley wire, AC system, alloy, supported					1.25	$112	2.0												
on mmessenger,	5,500			0.25				20.0				1,719	192,500	110,000	-	11,000	-	313,500	
Figure 8 or equal conductor, c/w messenger, clamps, insilulators etc. in unit price					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Cable Tray, 6 inch	200	m		0.45	1.25	$112		75.0				113	12,600	15,000	-	-	-	27,600	
Cable Tray 12 inch	200	m		0.60	1.25	$112		82.0				150	16,800	16,400	-	-	-	33,200	
Cable Tray 18 inch	150	m		0.70	1.25	$112		95.0				131	14,700	14,250	-	-	-	28,950	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 8 HAULAGE LEVEL											-	4,788	536,200	529,850	679,000	56,500	-		$1,801,550
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
COST CODE 9 - CRUSHER STATION #1																			
					1.25	$112						-		-	-	-	-	-	
54 x 75 crusher, 550 HP, 4000 V, (388 kW/520 HP motors are abnormal)					1.25	$112						-		-	-	-	-	-	
MOTORS					1.25	$112						-		-	-	-	-	-	

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
Assumed supplied with equipment					1.25	$112						-		-	-	-	-	-	
RING MAIN UNIT	1	ea		100.00	1.25	$112	2000.0	3,000.0	115,000			125	14,000	3,000	115,000	2,000	-	134,000	
UNIT SUBSTATION - 1500 KVA 25-4.16	1	ea		250.00	1.25	$112	2500.0	6,500.0	280,000			313	35,000	6,500		2,500	-	324,000	
kV with starters. This feeds crusher 4 kV																			
drive and conveyor head end drive															280,000				
UNIT SUB & AUX MCC FOR	1	ea		180.00	1.25	$112	2500.0	5,000.0	286,000			225	25,200	5,000	286,000	2,500	-	318,700	
600 VOLT DRIVES, 1000 KVA					1.25	$112						-		-	-	-	-	-	
(C/W motor starters)					1.25	$112						-		-	-	-	-	-	
POWER CABLE					1.25	$112						-		-	-	-	-	-	
(SP = Spaced, RF= Random Fill)					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
5 KV TECK					1.25	$112						-		-	-	-	-	-	
Crusher Motor 3#2 Teck	200	m		0.17	1.25	$112		65.0				43	4,760	13,000	-	-	-	17,760	
Terminations	4	ea		4.00	1.25	$112		250.0				20	2,240	1,000	-	-	-	3,240	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
600 /1KV CABLE					1.25	$112						-		-	-	-	-	-	
Lube system motors # 8 RF	300	m			1.25	$112		8.9				-		2,682	-	-	-	2,682	
Rock Breaker, 100 HP, 3c # 1 Teck	100	m		0.20	1.25	$112		45.0				25	2,800	4,500	-	-	-	7,300	
Rock Breaker, 1 HP Aux, 3c # 12 Teck	100	m		0.10	1.25	$112		5.8				13	1,400	580	-	-	-	1,980	
Apron feeder # 2/0 SP	150	m		0.10	1.25	$112		70.0				19	2,100	10,500	-	-	-	12,600	
Monorail hoist # 10	100	m		0.10	1.25	$112		6.7				13	1,400	670	-	-	-	2,070	
Bridge Crane #6	100	m		0.10	1.25	$112		14.6				13	1,400	1,460	-	-	-	2,860	
2 Air compressors #8	200	m		0.10	1.25	$112		8.9				25	2,800	1,788	-	-	-	4,588	
					1.25	$112						-		-	-	-	-	-	
Welding Outlets # 6	300	m		0.15	1.25	$112		14.6				56	6,300	4,380	-	-	-	10,680	
Belt splice station #6	75	m		0.15	1.25	$112		14.6				14	1,575	1,095	-	-	-	2,670	
spill feeder (see conveyor loads)					1.25	$112						-		-	-	-	-	-	
Dust Collector Fan, 3c# 4 Teck	200	m		0.17	1.25	$112		17.0				43	4,760	3,400	-	-	-	8,160	
Dust Collector Aux. 3c#12 Teck	200	m		0.10	1.25	$112		5.8				25	2,800	1,160	-	-	-	3,960	
Apron Feeder, 75 HP assumed, with					1.25	$112													
VFD	1	ea		20.00				725.0	14,500			25	2,800	725	14,500	-	-	18,025	
Apron feeder cable 3c#3 Teck	100	m		0.17	1.25	$112		33.0				21	2,380	3,300	-	-	-	5,680	
Tramp Magnet Motor cable, 3#10	150	m		0.10	1.25	$112		7.8				19	2,100	1,170	-	-	-	3,270	
Rectifier AC Cable 3#10	150	m		0.10	1.25	$112		7.8				19	2,100	1,170	-	-	-	3,270	
Lighting Transformer 45 kVA	2	ea		25.00	1.25	$112		200.0	2,100			63	7,000	400	4,200	-	-	11,600	
Control Transformer 30 kVA	1	ea		25.00	1.25	$112		200.0	1,800			31	3,500	200	1,800	-	-	5,500	
Unit Heaters 10 kW 600 V	5	ea		8.00	1.25	$112		200.0	1,950			50	5,600	1,000	9,750	-	-	16,350	
Jan 3/12 copper 3.43/lb.					1.25	$112						-		-	-	-	-	-	
						$112													
						$112													
Misc. 600V/1kV Teck Power & Control					1.25	$112													
Cable												-		-	-	-	-	-	
3#14	1,500	m		0.13	1.25	$112		4.76				244	27,300	7,140	-	-	-	34,440	
		m		0.13	1.25	$112						-		-	-	-	-	-	
6#14	1,500	m		0.13	1.25	$112		7.97				244	27,300	11,955	-	-	-	39,255	
	1,500	m		0.13	1.25	$112						244	27,300	-	-	-	-	27,300	
3#12	1,500	m		0.13	1.25	$112		5.86				244	27,300	8,790	-	-	-	36,090	
4#12	1,500	m		0.13	1.25	$112		6.95				244	27,300	10,425	-	-	-	37,725	
3#10	500	m		0.15	1.25	$112		7.87				94	10,500	3,935	-	-	-	14,435	
4#10	500	m		0.15	1.25	$112		10.27				94	10,500	5,135	-	-	-	15,635	
3#6	500	m		0.15	1.25	$112		19.30				94	10,500	9,650	-	-	-	20,150	
3#8	200	m		0.17	1.25	$112		14.60				43	4,760	2,920	-	-	-	7,680	
	0	m			1.25	$112						-		-	-	-	-	-	
Control Cable Terminations					1.25	$112						-		-	-	-	-	-	
3#14 Teck	100	ea		0.50	1.25	$112		100.0				63	7,000	10,000	-	-	-	17,000	
6#14 Teck	76	ea		0.75	1.25	$112		100.0				71	7,980	7,600	-	-	-	15,580	
12#14 teck	50	ea		1.00	1.25	$112		125.0				63	7,000	6,250	-	-	-	13,250	
	0	m			1.25	$112						-		-	-	-	-	-	
INSTRUMENT CABLE					1.25	$112						-		-	-	-	-	-	
INST Cable, 8TR-#16 300V ST-OS STC					1.25	$112													
AIA PVC	200	m		0.25				17.0				63	7,000	3,400	-	-	-	10,400	
Terminations	8	ea		2.00	1.25	$112		18.0				20	2,240	144	-	-	-	2,384	
4 Pair AIC	1,500	m		0.20	1.25	$112		4.2				375	42,000	6,300	-	-	-	48,300	
Terminations	32	ea		1.00	1.25	$112		15.0				40	4,480	480	-	-	-	4,960	
	0	m			1.25	$112						-		-	-	-	-	-	
CABLE TRAY					1.25	$112						-		-	-	-	-	-	
150	200			0.55	1.25	$112		75.0				138	15,400	15,000	-	-	-	30,400	
300	400			0.60	1.25	$112		82.0				300	33,600	32,800	-	-	-	66,400	
450	100			0.70	1.25	$112		95.0				88	9,800	9,500	-	-	-	19,300	
600	250			0.85	1.25	$112		110.0				266	29,750	27,500	-	-	-	57,250	
900	100			1.10	1.25	$112		120.0				138	15,400	12,000	-	-	-	27,400	
Fittings @ 15%	1	lot			1.25	$112						-		-	-	-	-	-	
Tray Support steel@ 10%	1	lot			1.25	$112						-		-	-	-	-	-	
	0	m			1.25	$112						-		-	-	-	-	-	
Boxes, fittings, local control stations, pilot					1.25	$112													
devices, welding outlets,disconnect																			
switches, etc.	1	lot		750.00				35,000.0	5,000			938	105,000	35,000	5,000	-	-	145,000	
	0	m			1.25	$112						-		-	-	-	-	-	
	0	m			1.25	$112						-		-	-	-	-	-	
	0	m			1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Grounding					1.25	$112						-		-	-	-	-	-	
Ground wire, 2/0 AWG	1,000	m		0.07	1.25	$112		7.00				88	9,800	7,000	-	-	-	16,800	

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 Connectors, compression	33	ea		0.75	1.25	$112		35.00				31	3,500	1,167	-	-	-	4,667	
Ground wire, 4/0	500	m		0.1	1.25	$112		11.00				63	7,000	5,500	-	-	-	12,500	
Ground Connectors, compression	17	ea		0.70	1.25	$112		30.0				15	1,633	500	-	-	-	2,133	
LOCAL LIGHTING AT CRUSHER	1	lot		550.00	1.25	$112	2500.0	40,000.0				688	77,000	40,000	-	2,500	-	119,500	
					1.25	$112						-		-	-	-	-	-	
CRUSHER E-HOUSE/CONTROL ROOM	1	ea		500.00	1.25	$112	2500.0	10,000.0	175,000			625	70,000	10,000	125,000	2,500	-	207,500	
					1.25	$112						-		-	-	-	-	-	
CONTROL SYSTEM INCLUDING PLC, PROGRAMMING, CABLE. PILOT DEVICES, ETC.	1	lot		1600.00	1.25	$112	2500.0	75,000.0	100,000			2,000	224,000	75,000	100,000	2,500	-	401,500	
					1.25	$112						-		-	-	-	-	-	
INSTRUMENTATION					1.25	$112						-		-	-	-	-	-	
General	1	lot		500.00	1.25	$112		2,500.0	50,000			625	70,000	2,500	50,000	-	-	122,500	
CCTV	1	LOT		200.00	1.25	$112		3,500.0	25,000			250	28,000	3,500	25,000	-	-	56,500	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 9 CRUSHER STATION 1											-	9,682	1,084,358	439,771	1,016,250	14,500	-		$2,554,879
																		-	
					1.25	$112						-		-	-	-	-	-	
COST CODE 10					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 10										-	-	-	-	-	-	-	-		$0
					1.25	$112						-		-	-	-	-	-	
COST CODE 11, REFUGE STATION 1 ELECTRICAL					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Transformer, breaker panel, etc.	1	lot		44.00	1.25	$112		1,000.0	6,500			55	6,160	1,000	6,500	-	-	13,660	
					1.25	$112						-		-	-	-	-	-	
Light fixtures, vapour tight fluorescent	10	ea		2.50	1.25	$112		20.0	225			31	3,500	200	2,250	-	-	5,950	
Emergency	2	ea		3.00	1.25	$112		20.0	400			8	840	40	800	-	-	1,680	
Door HID	1	ea		2.50	1.25	$112		20.0	325			3	350	20	325	-	-	695	
Heaters	3	ea		5.00	1.25	$112		25.0	1,500			19	2,100	75	4,500	-	-	6,675	
Communications	1	lot		30.00	1.25	$112	500.0	2,500.0	2,500			38	4,200	2,500	2,500	500	-	9,700	
					1.25	$112						-		-	-	-	-	-	
Allowance for air, safety equip. furniture, etc.	1	lot		100.00	1.25	$112	500.0	5,000.0	15,000			125	14,000	5,000	15,000	500	-	34,500	
					1.25	$112						-		-	-	-	-	-	
Outlets, boxes, etc.	1	lot		30.00	1.25	$112		2,500.0				38	4,200	2,500	-	-	-	6,700	
					1.25	$112						-		-	-	-	-	-	
Cable, 600 V Teck					1.25	$112						-		-	-	-	-	-	
3#12	250	m		0.13	1.25	$112		5.9				41	4,550	1,465	-	-	-	6,015	
4#12	100	m		0.13	1.25	$112		7.0				16	1,820	695	-	-	-	2,515	
					1.25	$112						-		-	-	-	-	-	
Terminations	1	lot		40.00	1.25	$112		1,500.0				50	5,600	1,500	-	-	-	7,100	
					1.25	$112						-		-	-	-	-	-	
Grounding	1	lot		40.00	1.25	$112	500.0	2,500.0				50	5,600	2,500	-	500	-	8,600	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 11, REFUGE STATION 1											-	473	52,920	17,495	31,875	1,500	-		$103,790
COST CODE 12 REFUGE STATION 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 12, REFUGE STATION 2											-	473	52,920	17,495	31,875	1,500	-		$103,790
					1.25	$112						-		-	-	-	-	-	
COST CODE 13 REFUGE STATION 3					1.25	$112						-		-	-	-	-	-	
					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 13, REFUGE STATION 3											-	473	52,920	17,495	31,875	1,500	-		$103,790
					1.25	$112						-		-	-	-	-	-	
COST CODE 14 REFUGE STATION 4					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 14, REFUGE STATION 4											-	473	52,920	17,495	31,875	1,500	-		$103,790
COST CODE 15 REFUGE STATION 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 15, REFUGE STATION 5											-	473	52,920	17,495	31,875	1,500	-		$103,790
COST CODE 16 REFUGE STATION 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	

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	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 16, REFUGE STATION 6											-	473	52,920	17,495	31,875	1,500	-		$103,790
COST CODE 17 REFUGE STATION 2					1.25	$112						-		-	-	-	-	-	
					1.25	$112						473	52,920	17,495	31,875	1,500	-	103,790	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 17, REFUGE STATION 7											-	473	52,920	17,495	31,875	1,500	-		$103,790
					1.25	$112						-		-	-	-	-	-	
COST CODE 18, MAIN VENT FANS (2) AND AIR HEATERS POWER (2)					1.25	$112						-		-	-	-	-	-	
MAIN VENT FANS						$112													
25 kV supply cable3c 2/0, Teck to VFD transformer	750	m		0.30	1.25	$112		120.0				281	31,500	90,000	-	-	-	121,500	
					1.25	$112						-		-	-	-	-	-	
MOTORS ASSUMED SUPPLIED WITH FANS					1.25	$112						-		-	-	-	-	-	
Ring Main Units, 25 KV	3	ea		75.00	1.25	$112	1500.0	5,650.0	113,000			281	31,500	16,950	339,000	4,500	-	391,950	
VFDs,2 x 3000 HP, DESIGNED FOR U/G MINE INSTALLATION, c/w step-down xformer from 25 kV	2	ea		175.00	1.25	$112	2500.0	22,500.0	450,000			438	49,000	45,000	900,000	5,000	-	999,000	
VFDs,2 x 900 HP, DESIGNED FOR U/G MINE INSTALLATION, c/w step-down xformer from 25 kV	1	ea		100.00	1.25	$112	1500.0	6,750.0	135,000			125	14,000	6,750	135,000	1,500	-	157,250	
LOCAL CABLE TO MOTORS, 2 x 3c 300 MCM, 5 kV	200	m		0.25	1.25	$112		215.0				63	7,000	43,000	-	-	-	50,000	
VFD, 900 HP		m			1.25	$112						-		-	-	-	-	-	
Cable, 1/0 AWG, 5 kV Teck	100	m		0.22	1.25	$112		60.0				28	3,080	6,000	-	-	-	9,080	
Messenger c/w rock bolts, hangers, etc.	1,050	m		0.15	1.25	$112		10.0				197	22,050	10,500	-	-	-	32,550	
Terminations, boxes, hardware, supports, etc.	1	lot		150.00	1.25	$112	2500.0	12,000.0				188	21,000	12,000	-	2,500	-	35,500	
					1.25	$112						-		-	-	-	-	-	
Stench gas system	3	lot		75.00	1.25	$112	1000.0	2,500.0	11,000			281	31,500	7,500	33,000	3,000	-	75,000	
					1.25	$112						-		-	-	-	-	-	
Control system fibre optic cable c/w messenger	1,000	m		0.50	1.25	$112	2.0	7.0				625	70,000	7,000	-	2,000	-	79,000	
Fibre optic terminations, panels, equip. etc.	1	lot		250.00	1.25	$112	2500.0	12,000.0	15,000			313	35,000	12,000	15,000	2,500	-	64,500	
Controls and Instruments	1	lot		200.00	1.25	$112	1500.0	6,000.0	7,000			250	28,000	6,000	7,000	1,500	-	42,500	
Misc. control & instrument cable, etc.	1	lot		250.00	1.25	$112	500.0	10,000.0	5,000			313	35,000	10,000	5,000	500	-	50,500	
					1.25	$112						-		-	-	-	-	-	
Boxes, etc.	1	lot		50.00	1.25	$112		3,000.0				63	7,000	3,000	-	-	-	10,000	
Cable Tray, 6 inch	75	m		0.45	1.25	$112		75.0				42	4,725	5,625	-	-	-	10,350	
Cable Tray 12 inch	150	m		0.60	1.25	$112		82.0				113	12,600	12,300	-	-	-	24,900	
Cable Tray 18 inch	0	m		0.70	1.25	$112		95.0				-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
HV JUNCTION BOX (FROM FANS)	0	EA		100.00	1.25	$112		5,500.0				-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
		M			1.25	$112						-		-	-	-	-	-	
	0	m		0.15	1.25	$112						-		-	-	-	-	-	
CONTROL (ALARM) CONNECTIONS	1	lot		200.00	1.25	$112	1000.0	10,000.0	10,000			250	28,000	10,000	10,000	1,000	-	49,000	
MISC. BOXES, CABLES, TERMINATIONS, ETC.	1	lot		350.00	1.25	$112	1500.0	15,000.0	5,000			438	49,000	15,000	5,000	1,500	-	70,500	
					1.25	$112						-		-	-	-	-	-	
Grounding					1.25	$112						-		-	-	-	-	-	
Ground wire, 2/0 AWG	750	m		0.07	1.25	$112	$2.0	7.00				66	7,350	5,250	-	1,500	-	14,100	
Ground Connectors, compression	15	ea		0.75	1.25	$112		30.00				14	1,575	450	-	-	-	2,025	
Ground wire, 4/0	500	m		0.1	1.25	$112	$2.0	11.00				63	7,000	5,500	-	1,000	-	13,500	
Ground Connectors, compression	10	ea		0.75	1.25	$112		35.0				9	1,050	350	-	-	-	1,400	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 18 MAIN VENT FANS AND AIR HEATER POWER (2)											-	4,437	496,930	330,175	1,449,000	28,000	-		$2,304,105
					1.25	$112						-		-	-	-	-	-	
COST CODE 19 MAIN AIR COMPRESSORS POWER & CONTROL					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Two x 200 HP					1.25	$112						-		-	-	-	-	-	
Note, power supply is from ring main system in the drift. See Cost Code 2)					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
RING MAIN UNITS	1	ea		100.00	1.25	$112	750.0		113,000			125	14,000	-	113,000	750	-	127,750	
					1.25	$112						-		-	-	-	-	-	
1500 KVA C/W MOTOR STARTERS	1	EA		200.00	1.25	$112	1500.0	9,450.0	189,000			250	28,000	9,450	189,000	1,500	-	227,950	
					1.25	$112						-		-	-	-	-	-	
LOCAL TECK POWER CABLE 3c #6	200	m		0.20	1.25	$112		22.0				50	5,600	4,400	-	-	-	10,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Boxes, etc.	1	lot		50.00	1.25	$112		3,000.0				63	7,000	3,000	-	-	-	10,000	

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
Cable Tray, 6 inch	50	m		0.45	1.25	$112		75.0				28	3,150	3,750	-	-	-	6,900	
Cable Tray 12 inch	100	m		0.60	1.25	$112		82.0				75	8,400	8,200	-	-	-	16,600	
Cable Tray 18 inch	0	m		0.70	1.25	$112		95.0				-		-	-	-	-	-	
Grounding					1.25	$112						-		-	-	-	-	-	
Ground wire, 2/0 AWG	250	m		0.07	1.25	$112	$2.0	7.00				22	2,450	1,750	-	500	-	4,700	
Ground Connectors, compression	5	ea		0.75	1.25	$112		30.00				5	525	150	-	-	-	675	
Ground wire, 4/0	200	m		0.1	1.25	$112	$2.0	11.00				25	2,800	2,200	-	400	-	5,400	
Ground Connectors, compression	4	ea		0.75	1.25	$112		35.0				4	420	140	-	-	-	560	
					1.25	$112						-		-	-	-	-	-	
CONTROL (ALARM) CONNECTIONS	1	LOT		200.00	1.25	$112	1000.0	10,000.0	10,000			250	28,000	10,000	10,000	1,000	-	49,000	
MISC. BOXES, CABLES, TERMINATIONS, ETC.	1	LOT		250.00	1.25	$112	1500.0	12,000.0	5,000			313	35,000	12,000	5,000	1,500	-	53,500	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
												-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 19, MAIN AIR COMPRESSORS POWER & CONTROL											-	1,208	135,345	55,040	317,000	5,650	-		$513,035
												-		-			-	-	
COST CODE 20 MINE COMMUNICATIONS SYSTEM (LEAKY FEEDER & PED)												-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Estimate based on quotations from"Mine Site Technologies"					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
LEAKY FEEDER SYSTEM		LOT			1.00	$112						-		-	-	-	-	-	
(BASIC INITIAL INSTALLATION)					1.25	$112						-		-	-	-	-	-	
Mine Leaky Feeder					1.25	$112						-		-	-	-	-	-	
LMR -600 Cable	1000	m		0.1	1.25	$112		8.66				125	14,000	8,660	-	-	-	22,660	
LMR -600 Terminations	4	ea		4	1.25	$112		31.00				20	2,240	124	-	-	-	2,364	
3/8 Inch Steel Messenger, Cable Hooks, Etc.	1000	m		0.05	1.25	$112		3.00				63	7,000	3,000	-	-	-	10,000	
Misc. cable, etc.	1	lot		100	1.25	$112		5,000.00				125	14,000	5,000	-	-	-	19,000	
					1.25	$112						-		-	-	-	-	-	
Conveyor Leaky Feeder					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Conveyor 1	0.43	km		150.00	1.25	$112		46,000.0				81	9,030	19,780	-	-	-	28,810	
					1.25	$112						-		-	-	-	-	-	
Conveyor 2	2.20	km		500.00	1.25	$112		46,000.0				1,375	154,000	101,200	-	-	-	255,200	
					1.25	$112						-		-	-	-	-	-	
Conveyor 3	2.00	km		500.00	1.25	$112		46,000.0				1,250	140,000	92,000	-	-	-	232,000	
					1.25	$112						-		-	-	-	-	-	
(Conveyor 4 included with Plant Communications)					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
771386					1.25	$112						-		-	-	-	-	-	
PED MATERIAL & EQUIPMENT	1	LOT		1500.00	1.25	$112	5000.0		556,386			1,875	210,000	-	556,386	5,000	-	771,386	
Leaky Feeder System					1.25	$112						-		-	-	-	-	-	
Leaky Feeder Headend (3 channel system) VC-HEADEND	1				1.25	$112			$46,375			-		-	46,375	-	-	46,375	
Leaky Feeder Antenna System VC-ANTENNA	1				1.25	$112			$11,842			-		-	11,842	-	-	11,842	
Leaky Feeder Cable - 350m roll, V-CA75-DS	20				1.25	$112			$1,090			-		-	21,800	-	-	21,800	
Infinity II Line Amplifiers, VC-1004-INF2	18				1.25	$112			$1,155			-		-	20,790	-	-	20,790	
Infinity Line Branch 3 Way, VC-1007-INF	5				1.25	$112			$821			-		-	4,105	-	-	4,105	
Pilot Tone Generator - Uplink, V-1007-INF2U	1				1.25	$112			$390			-		-	390	-	-	390	
KS-KS Rotational Adaptor, V-KSKSR	2				1.25	$112			$47			-		-	94	-	-	94	
KS - N Female Adaptor, V-KSNFA	2				1.25	$112			$45			-		-	90	-	-	90	
Line Termination Kit, VC-1011	1				1.25	$112			$111			-		-	111	-	-	111	
32V Leaky Feeder Power Supply w/Inserter, V-LF32V-PSKIT	2				1.25	$112			$3,045			-		-	6,090	-	-	6,090	
Stope Antenna (times this by number of drifts), STP-ANT	1				1.25	$112			$1,773			-		-	1,773	-	-	1,773	
Infinity Line Branch 3 Way (times this by number of drifts), VC-1007-INF	1				1.25	$112			$821			-		-	821	-	-	821	
Telephone Interconnect System (option), VC-1020	1				1.25	$112			$4,727			-		-	4,727	-	-	4,727	
VDV Amplifier Diagnostics Kit (option), VC-2000KIT	1				1.25	$112			$5,090			-		-	5,090	-	-	5,090	
UPS Backup System (option), UPS-BCKP	6				1.25	$112			$3,636			-		-	21,816	-	-	21,816	
Cabling					1.25	$112						-		-	-	-	-	-	
Customer Programming Software / Cloning Cables, CLN-SFT-CBL	1				1.25	$112		$408				-		408	-	-	-	408	
(The following cable lengths are summed and inserted into the cable costs below)																			
- Conveyor 1	430																		
- Conveyor 2	2,200																		
- Conveyor 3	2,000																		
(Conveyor 4 included with Plant Communications)																			
- Mine Drift																			

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
- Mine Drift																			
- Mine Drift																			
- Mine Drift																			
- Mine Drift																			
- Mine Drift																			
LMR-600 Low Loss Cabling and Connectors (500m roll), LMR-600	4,630	m		0.05	1.25	$112		8.66				289	32,410	40,105	-	-	-	72,515	
LMR-600 Connector, ANM-1610 (L-Com)	5	ea			1.25	$112		$31				-		144	-	-	-	144	
Messenger, 3/8" galv guystrand c/w hooks. Anchor bolts by others	4,630			0.10	1.25	$112		$3				579	64,820	13,890	-	-	-	78,710	
RG-213 Coaxial Cable (300m), RG-213					1.25	$112		$135				-		-	-	-	-	-	
RG58U coax (100 ft), RG-58U					1.25	$112		$133				-		-	-	-	-	-	
N Type Splice Connector, V-N-CONN					1.25	$112		$20				-		-	-	-	-	-	
N Type Male Connectors, V-N-M-CONN					1.25	$112		$18				-		-	-	-	-	-	
CAB Hooks, Per 1200, CB-934					1.25	$112		$2				-		-	-	-	-	-	
Lot, Consumables, MISC-CON					1.25	$112		$100				-		-	-	-	-	-	
Additional Services					1.25	$112						-		-	-	-	-	-	
Radio Licensing Administration Fee	1				1.25	$112				$250		-		-	-	-	250	250	
In-Shop Setup and Configuration/Custom Cabling/Radio programming	1				1.25	$112				$1,500		-		-	-	-	1,500	1,500	
On-site Commissioning	1				1.25	$112				$15,000		-		-	-	-	15,000	15,000	
Leaky Feeder Training	1				1.25	$112				$5,000		-		-	-	-	5,000	5,000	
Radios					1.25	$112						-		-	-	-	-	-	
FR5000 VHF Repeaters, FR-5000	2				1.25	$112			$2,291			-		-	4,582	-	-	4,582	
UR-FR5000 VHF Repeater Modules, UR-FR5000	1				1.25	$112			$1,745			-		-	1,745	-	-	1,745	
Base Stations					1.25	$112						-		-	-	-	-	-	
Icom IC-F5021 VHF 128ch Variable Power Analog Mobile, IC-F5021	3				1.25	$112			$513			-		-	1,539	-	-	1,539	
Samplex SEC-1223 Desktop Power Supply, SEC-1223	3				1.25	$112			$218			-		-	654	-	-	654	
Equipment Radios For Vehicles					1.25	$112						-		-	-	-	-	-	
Icom IC-F5021 VHF 128ch, IC-F5021 Variable Power Analog Mobile	66				1.25	$112			$513			-		-	33,858	-	-	33,858	
					1.25	$112						-		-	-	-	-	-	
Mobile Antenna Kit, WHP-ANT-KT	66				1.25	$112			$1,226			-		-	80,916	-	-	80,916	
Portable Radios (For 2 shifts)					1.25	$112						-		-	-	-	-	-	
Icom IC-F3021S 8 Channel Variable Power VHF Portable Radio (For 2 shifts), IC-F3021S	140				1.25	$112			$404			-		-	56,560	-	-	56,560	
3021 Tunable Antenna Kit, 3021-ANT-KT	140				1.25	$112			$28			-		-	3,920	-	-	3,920	
Spare 2000 mAh, 7.4V Li-Ion battery packs, BP-232N	14				1.25	$112			$87			-		-	1,218	-	-	1,218	
HM-159L Noise Cancelling Speaker Microphone, HM-159L	140				1.25	$112			$87			-		-	12,180	-	-	12,180	
6 unit multi charger with cups and power supply whole radio or batteries, BC-197 #22	24				1.25	$112			$544			-		-	13,056	-	-	13,056	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
PED Equipment Supply - Basic					1.25	$112						-		-	-	-	-	-	
Transmission System					1.25	$112						-		-	-	-	-	-	
PED 110 V Headend Fully Equipped (Long Loop), P-TX-CABC-000L	1	ea			1.25	$112			$43,391			-		-	43,391	-	-	43,391	
Smart External Modulator, P-SEMA-000	1	ea			1.25	$112			$4,364			-		-	4,364	-	-	4,364	
PEDcall Software Licence, P-S010	1	ea			1.25	$112			$7,249			-		-	7,249	-	-	7,249	
UPS	1	ea			1.25	$112			$4,500			-		-	4,500	-	-	4,500	
Antenna System					1.25	$112						-		-	-	-	-	-	
Single Core PED antenna (500m roll),P-XLP1X10	45	m			1.25	$112			10.56			-		-	475	-	-	475	
Antenna Splice Kits, P-SP2KIT					1.25	$112			$355			-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
PED Receivers					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
Caplamp, battery and PED ICCL with integrated Tracking Tag and LED Headpeice (lamp)., CL-92M-PC-T8-01L	850	ea			1.25	$112			$1,755			-		-	1,491,750	-	-	1,491,750	
Thirty unit charging rack with charging boards for LED Headpiece, CL-CHGR-30L	28	ea		8.00	1.25	$112		70.0	$7,000			283	31,733	1,983	198,333	-	-	232,050	
Vehicle PEDs, AutoPEDs	80	ea		4.00	1.25	$112		100.0	$2,590			400	44,800	8,000	207,200	-	-	260,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	

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	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 20, MINE COMMUNICATIONS:											-	6,465	724,033	294,294	2,869,790	5,000	21,750		$3,914,868
Direct costs include project contractor profit																			
												MH	C$	C$	C$	C$	C$	C$	C$
SUBTOTAL, DIRECT COSTS											38	91,289	10,241,336	9,853,797	18,743,315	402,500	21,750	34,272,399	$35,809,299
												-		-	-	-	-	-	(Checksum) 39,262,699
COST CODE 30 ENGINEERING, CONSTRUCTION MANAGEMENT AND SPARES												-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
SPARE PARTS	5.0%	OF EQUIPMENT			1.25	$112						-		-	625,000	-	-	625,000	
					1.25	$112						-		-	-	-	-	-	
CONSTRUCTION MANAGEMENT					1.25	$112						-		-	-	-	-	-	
(Note special (average) labour rates)					1.25	$112						-		-	-	-	-	-	
Construction Offices & Office Equipment, communications, vehicle rental etc.	24	mth			1.25	$112	12,500	2,500	4,000			-		60,000	96,000	300,000	-	456,000	
					1.25	$112						-		-	-	-	-	-	
Construction Management FOR electrical & inst.	20	mth		800	1.25	$150.0	7,500	10,000				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	$112						-		-	-	-	-	-	
COMMISSIONING					1.25	$112						-		-	-	-	-	-	
Vendor Reps (in addition to proposal)	1	lot			1.25	$112				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	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
ENGINEERING (POWER)					1.25	$112						-		-	-	-	-	-	
System electrical and instrument design	1	lot			1.25	$112				1,900,000		-		-	-	-	1,900,000	1,900,000	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-			-	-	Area Subtotal
SUBTOTAL COST CODE 30, 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							$41,860,899	$43,397,799
SUSTAINING CAPITAL																			
COST CODE 40 CRUSHER STATION 2 - SUSTAINING CAPITAL + 5 YEARS						$112						9,682	1,084,358	439,771	1,016,250	14,500	-	2,554,879	
					1.25	$112						-		-	-	-	-	-	
					1.25	$112						-		-	-	-	-	-	Area Subtotal
SUBTOTAL COST CODE 40 CRUSHER STATION 2											38	9,682	1,084,358	439,771	1,016,250	14,500	-		$2,554,879

APPENDIX J

Workforce Breakdown by Year

LABOUR SCHEDULE																						
						Project Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Labour Rate Accuracy	25%					Year	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
DEVELOPMENT LABOUR																						
Job Title	Peak Labour Quantity	Pay Scale	Contracted?	Annual Costs																		
Management																						
Mine Manager	0	A	No	$297,000		$0																
Underground Superintendent	1	Y	No	$297,000		$2,079,000	1	1	1	1	1	1	1									
Technical Superintendent	1	Y	No	$297,000		$5,940,000			1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Superintendent	0	Y	No	$297,000		$0																
Clerical/Admin	8	J	No	$152,100		$25,248,600	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Technical																						
Senior Engineer	8	B	No	$231,000		$15,246,000	2	2	6	6	6	6	8	2	2	2	2	2	2	2	2	2
Engineer	12	C	No	$189,750		$36,811,500	6	8	12	12	12	12	12	8	8	8	8	8	8	8	8	8
Technologist	10	F	No	$123,750		$25,740,000	3	5	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Trainers	8	F	No	$123,750		$12,375,000			8	8	8	8	8	4	4	4	4	4	4	4	4	4
Geologists	0	C	No	$189,750		$0																
Safety Technician	8	C	No	$189,750		$31,878,000	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Maintenance																						
Planner	1	F	No	$123,750		$742,500	1	1	1	1	1	1										
Shop Foreman	1	N	No	$181,500		$1,089,000	1	1	1	1	1	1										
Stores Person	8	J	No	$152,100		$5,475,600	4	4	4	8	8	8										
Electrician (Inhouse)	32	L	No	$193,050		$28,957,500	18	18	18	32	32	32										
Electrician (Cotnractor)	0	L	No	$193,050		$0																
Mechanics (Inhouse)	28	L	No	$193,050		$24,324,300	14	14	14	28	28	28										
Mechanics (Contractor)	0	L	No	$193,050		$0																
Development Crew																						
Shift Captain	4	R	No	$198,000		$15,840,000			4	4	4	4	4	4	4	4	4	4	4	4	4	4
Shift Boss	4	O	No	$181,500		$17,787,000	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Jumbo Operator	12	H	No	$163,800		$33,087,600	4	8	8	12	12	12	12	12	8	8	8	8	8	8	8	8
Bolter Operator	12	H	No	$163,800		$33,087,600	4	8	8	12	12	12	12	12	8	8	8	8	8	8	8	8
Loader Operator	12	H	No	$163,800		$31,449,600	4	8	8	12	12	12	12	12	8	8	8	8	8	8	8	8
Truck Operator	16	H	No	$163,800		$43,898,400	4	8	12	12	16	16	16	16	12	12	12	12	12	12	12	12
Crane Truck	8	J	No	$152,100		$27,073,800		8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Shotcrete Sprayer Operator	8	H	No	$163,800		$26,208,000			8	8	8	8	8	8	8	8	8	8	8	8	8	8
Agicar Operator	8	J	No	$152,100		$24,336,000			8	8	8	8	8	8	8	8	8	8	8	8	8	8
Grader	8	H	No	$163,800		$5,896,800				4	8	8	8	8								
Construction	32	I	No	$126,750		$16,224,000				32	32	32	32									
Labourer \Trainee	44	J	No	$152,100		$128,372,400	32	36	40	44	44	44	44	36	36	36	36	36	36	36	36	36
Development In-House Employees							110	142	200	284	292	292	224	169	145	145	145	145	145	145	145	145
Development Contractor Employees							0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal Cost (Inhouse)					25%	$619,168,200	$18,941,400	$24,014,400	$33,553,350	$46,852,350	$48,162,750	$48,162,750	$35,519,700	$27,309,900	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700
Subtotal Cost (Contractor)					0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal Contingency (Inhouse)						$154,792,050	$4,735,350	$6,003,600	$8,388,338	$11,713,088	$12,040,688	$12,040,688	$8,879,925	$6,827,475	$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,844,675
Subtotal Contingency (Contractor)						$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
PRODUCTION LABOUR																						
Job Title	Peak Labour Quantity	Pay Scale	Contracted?	Annual Costs																		
Management																						
Mine Manager	0	A	No	$297,000		$0																
Underground Superintendent	1	A	No	$297,000		$8,316,000							1	1	1	1	1	1	1	1	1	1
Technical Superintendent	1	A	No	$297,000		$3,564,000																
Maintenance Superintendent	1	A	No	$297,000		$8,316,000							1	1	1	1	1	1	1	1	1	1
Clerical/Admin	8	D	No	$90,750		$7,986,000																
Technical																						
Senior Engineer	2	B	No	$231,000		$12,993,750							2	2	2	2	2	2	2	2	2	2
Engineer	5	C	No	$189,750		$25,426,500							5	5	5	5	5	5	5	5	5	5
Technologist	10	F	No	$123,750		$32,917,500							10	10	10	10	10	10	10	10	10	10
Trainers	0	F	No	$123,750		$0																
Geologists	8	C	No	$189,750		$41,555,250							8	8	8	8	8	8	8	8	8	8
Safety Technician	0	C	No	$189,750		$0																
Maintenance																						
Planner	4	C	No	$189,750		$19,734,000							4	4	4	4	4	4	4	4	4	4
Shop Foreman	4	O	No	$181,500		$20,146,500							4	4	4	4	4	4	4	4	4	4
Stores Person	8	D	No	$90,750		$17,787,000							8	8	8	8	8	8	8	8	8	8
Electrician (Inhouse)	32	K	No	$179,400		$146,928,600							32	32	32	32	32	32	32	32	32	32
Electrician (Cotnractor)	0	k	Yes	$119,600		$0																
Mechanics (Inhouse)	28	J	No	$152,100		$110,310,525							28	28	28	28	28	28	28	28	28	28
Mechanics (Contractor)	0	J	Yes	$101,400		$0																
Production Crew																						
Shift Captain	12	R	No	$198,000		$64,944,000							12	12	12	12	12	12	12	12	12	12
Shift Boss	12	O	No	$181,500		$59,032,875							12	12	12	12	12	12	12	12	12	12
Loader Operator	40	H	No	$163,800		$154,995,750							16	16	24	32	40	40	40	40	40	40
Train Operator	8	H	No	$163,800		$34,602,750							8	8	8	8	8	8	8	8	8	8
Secondary Breaker Operator	8	H	No	$163,800		$30,794,400							8	8	8	8	8	8	8	8	8	8
Block Holer Operator	8	H	No	$163,800		$28,828,800							8	8	8	8	8	8	8	8	8	8
Grader Operator	8	H	No	$163,800		$32,432,400							8	8	8	8	8	8	8	8	8	8
Crusher/Conveyor Operator	20	H	No	$163,800		$80,139,150							16	16	16	16	16	16	16	16	20	20
Production Driller	24	H	No	$163,800		$70,761,600							24	24	24	24	24	24	24	24	24	24
Anfo Loader	8	H	No	$163,800		$31,449,600							8	8	8	8	8	8	8	8	8	8
Construction	32	I	No	$126,750		$57,798,000							32	32	32	32	32	32	32	32	32	16
Labourer \Trainee	10	J	No	$152,100		$38,785,500							10	10	10	10	10	10	10	10	10	10
Production In-House Employees						293	0	0	0	0	0	0	265	265	273	281	289	289	289	289	293	277
Production Contractor Employees							0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal Cost (Inhouse)					25%	$1,140,546,450	$0	$0	$0	$0	$0	$0	$42,826,650	$42,826,650	$44,137,050	$45,447,450	$46,757,850	$46,757,850	$46,757,850	$46,757,850	$47,413,050	$45,385,050
Subtotal Cost (Contractor)					0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal Contingency (Inhouse)						$285,136,613	$0	$0	$0	$0	$0	$0	$10,706,663	$10,706,663	$11,034,263	$11,361,863	$11,689,463	$11,689,463	$11,689,463	$11,689,463	$11,853,263	$11,346,263
Subtotal Contingency (Contractor)						$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN-HOUSE EMPLOYEES						489	110	142	200	284	292	292	489	434	418	426	434	434	434	434	438	422
TOTAL CONTRACTOR EMPLOYEES							0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL EMPLOYEES							110	142	200	284	292	292	489	434	418	426	434	434	434	434	438	422
TOTAL COST (INHOUSE)					25%	$1,759,714,650	$18,941,400	$24,014,400	$33,553,350	$46,852,350	$48,162,750	$48,162,750	$78,346,350	$70,136,550	$67,515,750	$68,826,150	$70,136,550	$70,136,550	$70,136,550	$70,136,550	$70,791,750	$68,763,750
TOTAL COST (CONTRACTOR)					0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL LABOUR COST					25%	$1,759,714,650	$18,941,400	$24,014,400	$33,553,350	$46,852,350	$48,162,750	$48,162,750	$78,346,350	$70,136,550	$67,515,750	$68,826,150	$70,136,550	$70,136,550	$70,136,550	$70,136,550	$70,791,750	$68,763,750
TOTAL CONTINGENCY (INHOUSE)						$439,928,663	$4,735,350	$6,003,600	$8,388,338	$11,713,088	$12,040,688	$12,040,688	$19,586,588	$17,534,138	$16,878,938	$17,206,538	$17,534,138	$17,534,138	$17,534,138	$17,534,138	$17,697,938	$17,190,938
TOTAL CONTINGENCY (CONTRACTOR)						$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY						$439,928,663	$4,735,350	$6,003,600	$8,388,338	$11,713,088	$12,040,688	$12,040,688	$19,586,588	$17,534,138	$16,878,938	$17,206,538	$17,534,138	$17,534,138	$17,534,138	$17,534,138	$17,697,938	$17,190,938

17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1	1	1	1	1	1														
8	8	8	8	8	6														
2	2	2	2	2	2														
8	8	8	8	8	8														
10	10	10	10	10	10														
4	4	4	4	4	4														
8	8	8	8	8	8														
4	4	4	4	4	4														
4	4	4	4	4	4	1	1	1	1	1	1	1	1	1	1				
8	8	8	8	8	8	1	1	1	1	1	1	1	1	1	1				
8	8	8	8	8	8	1	1	1	1	1	1	1	1	1	1				
8	8	8	8	8	8														
12	12	12	12	12	12														
8	8	8	8	8	8	1	1	1	1	1	1	1	1	1	1				
8	8	8	8	8	8														
8	8	8	8	8	8														
36	36	36	36	36	36	2	2	2	2	2	2	2	2	2	2				
145	145	145	145	145	143	6	6	6	6	6	6	6	6	6	6	0	0	0	0
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,074,500	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$0	$0	$0	$0
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,844,675	$5,768,625	$241,350	$241,350	$241,350	$241,350	$241,350	$241,350	$241,350	$241,350	$241,350	$241,350	$0	$0	$0	$0
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
						1	1	1	1	1	1	1	1	1	1	1	1		
1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
						8	8	8	8	8	8	8	8	8	8	8			
2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	0.25
5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	1		
10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	4	2		
8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	6	4	1	
4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2			
4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	1	
8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4				
32	32	32	32	32	32	32	32	32	32	32	32	32	26	24	16	12	4	1	
28	28	28	28	28	28	28	28	28	28	28	28	28	26	26	16	8	4	1	0.25
12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	8	8		
12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	8	4	1	0.25
40	40	40	40	40	40	40	40	40	40	40	40	28	32	32	24	12	8	2	0.25
8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	2	1	0.25
8	8	8	8	8	8	8	8	8	8	4	4	4	4	4	4	2	2		
8	8	8	8	8	8	8	8	8	8	4	4	4	4						
8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	2			
20	20	20	20	20	20	20	20	20	20	20	20	20	20	16	12	8	4	1	0.25
24	24	24	24	24	24	24	24												
8	8	8	8	8	8	8	8	8	8	8	8	8	8						
16	16	16	16	16	16	16	16	8	8	8									
10	10	10	10	10	10	10	10	10	10	10	10	10	10	5	5	5			
277	277	277	277	277	277	286	286	254	254	246	238	226	214	191	155	101	49	10	1.5
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
$45,385,050	$45,385,050	$45,385,050	$45,385,050	$45,385,050	$45,385,050	$46,408,050	$46,408,050	$41,462,850	$41,462,850	$40,152,450	$39,138,450	$37,172,850	$35,429,250	$31,689,150	$25,732,650	$17,139,750	$8,938,050	$1,770,450	$264,000
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
$11,346,263	$11,346,263	$11,346,263	$11,346,263	$11,346,263	$11,346,263	$11,602,013	$11,602,013	$10,365,713	$10,365,713	$10,038,113	$9,784,613	$9,293,213	$8,857,313	$7,922,288	$6,433,163	$4,284,938	$2,234,513	$442,613	$66,000
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
422	422	422	422	422	420	292	292	260	260	252	244	232	220	197	161	101	49	10	1.5
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
422	422	422	422	422	420	292	292	260	260	252	244	232	220	197	161	101	49	10	1.5
$68,763,750	$68,763,750	$68,763,750	$68,763,750	$68,763,750	$68,459,550	$47,373,450	$47,373,450	$42,428,250	$42,428,250	$41,117,850	$40,103,850	$38,138,250	$36,394,650	$32,654,550	$26,698,050	$17,139,750	$8,938,050	$1,770,450	$264,000
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
$68,763,750	$68,763,750	$68,763,750	$68,763,750	$68,763,750	$68,459,550	$47,373,450	$47,373,450	$42,428,250	$42,428,250	$41,117,850	$40,103,850	$38,138,250	$36,394,650	$32,654,550	$26,698,050	$17,139,750	$8,938,050	$1,770,450	$264,000
$17,190,938	$17,190,938	$17,190,938	$17,190,938	$17,190,938	$17,114,888	$11,843,363	$11,843,363	$10,607,063	$10,607,063	$10,279,463	$10,025,963	$9,534,563	$9,098,663	$8,163,638	$6,674,513	$4,284,938	$2,234,513	$442,613	$66,000
$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
$17,190,938	$17,190,938	$17,190,938	$17,190,938	$17,190,938	$17,114,888	$11,843,363	$11,843,363	$10,607,063	$10,607,063	$10,279,463	$10,025,963	$9,534,563	$9,098,663	$8,163,638	$6,674,513	$4,284,938	$2,234,513	$442,613	$66,000

APPENDIX K

Development and Production Schedules by Year

DEVELOPMENT SCHEDULE

	Project Year		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
	Year		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name																		
Conveyors																		
Conveyors		4,644	1,800.00	914.23	1,502.26	315.77	0.00	0.00	111.61								0.00	0.00
Subtotal (Inhouse)	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	25%	$28,675,842	$11,115,000	$5,645,346	$9,276,462	$1,949,849	$0	$0	$689,186	$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	$0	$0	$0
Capital Contingency (Contractor)		*$7,168,960*	$2,778,750	$1,411,336	$2,319,115	$487,462	$0	$0	$172,296	$0	$0	$0	$0	$0	$0	$0	$0	$0
Access																		
Access		6,342	1800	2687.662	1854.83													
Subtotal (Inhouse)	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	25%	$39,164,888	$11,115,000	$16,596,313	$11,453,575	$0	$0	$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	$0	$0	$0
Capital Contingency (Contractor)		*$9,791,222*	$2,778,750	$4,149,078	$2,863,394	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawpoints																		
Drawpoints		42,957	0	0	0	0	0	454.937	861.421	4485.93	2823.322	1734.288	3600	1585.854	3600	2053.003	3038.991	2946.115
Subtotal (Inhouse)	17%	$114,222,711	$0	$0	$0	$0	$0	$1,209,688	$2,290,538	$11,928,188	$7,507,276	$4,611,511	$9,572,480	$4,216,821	$9,572,480	$5,458,981	$8,080,745	$7,833,786
Subtotal (Contractor)	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		*$19,146,169*	$0	$0	$0	$0	$0	$202,770	$383,943	$1,999,419	$1,258,380	$772,988	$1,604,552	$706,829	$1,604,552	$915,042	$1,354,506	$1,313,110
Capital Contingency (Contractor)		*$0*	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Exctraction Drifts																		
Exctraction Drifts		20,363	0	0	0	211.344	0	6076.932	4300.354	1228.18	765.159	1875.712	0	2014.144	0	1556.997	561.007	653.883
Subtotal (Inhouse)	16%	$84,333,363	$0	$0	$0	$875,273	$0	$25,167,387	$17,809,756	$5,086,462	$3,168,877	$7,768,191	$0	$8,341,502	$0	$6,448,245	$2,323,390	$2,708,032
Subtotal (Contractor)	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		*$13,870,322*	$0	$0	$0	$143,956	$0	$4,139,284	$2,929,173	$836,571	$521,186	$1,277,636	$0	$1,371,928	$0	$1,060,544	$382,128	$445,390
Capital Contingency (Contractor)		*$0*	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Secondary Breakage Level																		
Level Access		3,584	0	0	0	3084.988	0	0	499.174	0	0	0	0	0	0	0	0	0
Rockbreaker Chambers		784	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	18%	$9,104,923	$0	$0	$0	$3,692,341	$0	$0	$597,448	$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	$0	$0	$0
Capital Contingency (Inhouse)		*$1,634,359*	$0	$0	$0	$697,922	$0	$0	$112,929	$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	$0	$0	$0
Ventilation Infrastructure																		
Exhaust Tunnels		1,717	0	0	63.854	1207.212	0	0	446.253	0				0	0	0	0	0
Fresh Air Tunnels		4,908	0	0	0	2793.81	203.64	0	1850.155	60.392	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	16%	$14,014,367	$0	$0	$213,665	$8,745,934	$343,050	$0	$4,609,983	$101,736	$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	$0	$0	$0
Capital Contingency (Inhouse)		*$2,214,662*	$0	$0	$36,233	$1,390,979	$51,458	$0	$720,733	$15,260	$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	$0	$0	$0
Undercut Excavation																		
Undercut Excavation		20,691	0	0	0	1169.299	5565.354	4872.773	2300.995	6782.972					0	0	0	0
Subtotal (Inhouse)	19%	$24,764,987	$0	$0	$0	$1,399,503	$6,661,027	$5,832,095	$2,754,001	$8,118,362	$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	$0	$0	$0
Capital Contingency (Inhouse)		*$4,681,045*	$0	$0	$0	$264,532	$1,259,059	$1,102,375	$520,558	$1,534,522	$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	$0	$0	$0
Haulage Level																		
Haulage Level		5,493	0	0	0	1785.444	0	0	1587.153	493.434	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	15%	$9,252,894	$0	$0	$0	$3,007,746	$0	$0	$2,673,706	$831,235	$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	$0	$0	$0
Capital Contingency (Inhouse)		*$1,387,934*	$0	$0	$0	$451,162	$0	$0	$401,056	$124,685	$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	$0	$0	$0
Maintenance Area (a.k.a. the shop)																		
Maintenance Area (a.k.a. the shop)		1,361	0	0	0	0	0	0	0	1349.091	11.517	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	$8,357,625	$0	$0	$0	$0	$0	$0	$0	$8,286,881	$70,744	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		*$1,429,363*	$0	$0	$0	$0	$0	$0	$0	$1,417,264	$12,099	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		*$0*	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Preconditioning Level																		
Preconditioning Level		11,754	0	0	0	298.349	8420.226	3035.36				0	0	0	0	0	0	0
Subtotal (Inhouse)	19%	$14,067,977	$0	$0	$0	$357,086	$10,077,948	$3,632,943	$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	$0	$0	$0
Capital Contingency (Inhouse)		*$2,659,111*	$0	$0	$0	$67,496	$1,904,921	$686,694	$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	$0	$0	$0
Perimeter																		

DEVELOPMENT SCHEDULE																		
		Project Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name																		
Perimeter		3,109	0	0	0	1567.358	0	0	1542.022						0	0	0	0
Subtotal (Inhouse)	16%	**$12,877,381**	$0	$0	$0	$6,491,155	$0	$0	$6,386,227	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,117,945***	$0	$0	$0	$1,067,601	$0	$0	$1,050,344	$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	$0	$0	$0
Internal Ramps																		
Internal Ramps		2,697	0	0	125	1459.878	210.781	0	900.865	0	0			0	0	0	0	0
Subtotal (Inhouse)	19%	**$3,227,399**	$0	$0	$149,609	$1,747,290	$252,278	$0	$1,078,222	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$610,039***	$0	$0	$28,279	$330,270	$47,685	$0	$203,804	$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	$0	$0	$0
Vertical Development																		
RAR Raises		1,364	0	0	0	1363.744	0	0	0	0	0	0	0	0	0	0	0	0
FAR Raises		1,241	0	0	0	83.182	1157.688											
Internal Return Air Raise		3,604	0	0	0	2438.084	0	0	1166.155					0	0	0	0	0
Internal Fresh Air Raise		619	0	0	0	31.623	0	0	0	0	0	0	0	0	0	0	0	0
Orepasses		5,417	0	0	0	181.061	0	0	513.154	241.352	0	0	0	0	0	0	0	0
Crusher Chambers		86	0	6.002	40	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	25%	**$72,892,683**	$0	$55,414	$369,301	$26,453,826	$10,501,095	$0	$8,444,218	$1,213,612	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$18,211,138***	$0	$13,853	$92,325	$6,609,067	$2,625,274	$0	$2,109,218	$303,139	$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	$0	$0	$0
Drawbell and Drawpoint Blasting																		
Drawbell and Drawpoint Blasting		1,080	0	0	0	0	0	60	60	120	120	120	120	120	120	120	120	
Subtotal (Inhouse)	23%	**$84,340,923**	$0	$0	$0	$0	$0	$4,685,607	$4,685,607	$9,371,214	$9,371,214	$9,371,214	$9,371,214	$9,371,214	$9,371,214	$9,371,214	$9,371,214	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$19,658,493***	$0	$0	$0	$0	$0	$1,092,138	$1,092,138	$2,184,277	$2,184,277	$2,184,277	$2,184,277	$2,184,277	$2,184,277	$2,184,277	$2,184,277	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Undercut Blasting																		
Undercut Blasting		20,691	0	0	0	1169.299	5565.354	4872.773	2300.995	6782.972	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	**$12,964,227**	$0	$0	$0	$732,626	$3,486,982	$3,053,044	$1,441,692	$4,249,882	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,146,356***	$0	$0	$0	$121,294	$577,304	$505,462	$238,686	$703,610	$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	$0	$0	$0
Rehabilition																		
Rehabilition		13,040	360	360	355	1,389	1,440	1,444	1,440	1,440	360	361	360	360	360	361	360	360
Subtotal (Inhouse)	17%	**$33,436,208**	$923,057	$923,541	$909,197	$3,562,344	$3,692,227	$3,702,483	$3,692,227	$3,692,227	$923,056	$925,621	$923,057	$923,056	$923,057	$925,621	$923,056	$923,056
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$5,708,130***	$157,581	$157,664	$155,215	$608,153	$630,326	$632,077	$630,326	$630,326	$157,581	$158,019	$157,581	$157,581	$157,581	$158,019	$157,581	$157,581
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL LATERAL DEVELOPMENT METERS		**130,404**	3,600	3,602	3,546	13,893	14,400	14,440	14,400	14,400	3,600	3,610	3,600	3,600	3,600	3,610	3,600	3,600
TOTAL REHABILITATION METERS (opex)	10%	**13,040**											360	360	355	1,389	1,440	1,444
TOTAL CONTIGENCY METERS (opex)	5%	**6,520**	180	180	177	695	720	722	720	720	180	181	180	180	180	181	180	180
TOTAL VERTICAL DEVELOPMENT METERS		**12,330**	0	6	40	4,098	1,158	0	1,679	241	0	0	0	0	0	0	0	0
TOTAL UNDERCUT METERS		**20,691**	0	0	0	1,169	5,565	4,873	2,301	6,783	0	0	0	0	0	0	0	0
TOTAL DRAWBELLS		**#REF!**	0	0	0	0	0	#REF!	#REF!	#REF!	60	60	120	120	120	120	120	120
TOTAL DEVELOPMENT ORE TONNES		**5,653,615**	0	1,273	8,482	231,086	604,177	771,170	553,715	1,138,843	176,288	200,361	153,090	203,393	153,090	192,401	167,101	169,421
TOTAL DEVELOPMENT INCREMENTAL TONNES		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT WASTE TONNES		**2,238,535**	234,684	236,687	234,141	768,856	87,750	0	360,818	4,001	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT TONNES		**7,892,150**	234,684	237,960	242,623	999,942	691,927	771,170	914,533	1,142,844	176,288	200,361	153,090	203,393	153,090	192,401	167,101	169,421
TOTAL CONTAINED AU		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL INHOUSE COST	19%	**$478,946,086**	$1,138,494	$1,194,505	$1,853,974	$51,196,473	$35,876,357	$48,147,390	$54,054,219	$44,623,431	$21,185,861	$22,892,572	$20,082,188	$23,068,030	$20,082,188	$22,420,096	$20,913,841	$11,680,311
TOTAL CONTRACTOR COST	25%	**$67,840,730**	$22,230,000	$22,241,658	$20,730,037	$1,949,849	$0	$0	$689,186	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	19%	**$546,786,816**	$23,368,494	$23,436,163	$22,584,011	$53,146,321	$35,876,357	$48,147,390	$54,743,405	$44,623,431	$21,185,861	$22,892,572	$20,082,188	$23,068,030	$20,082,188	$22,420,096	$20,913,841	$11,680,311
TOTAL INHOUSE CAPITAL CONTIGENCY		**$88,877,055**	$157,581	$171,518	$312,053	$10,482,054	$6,518,722	$7,855,338	$9,640,238	$7,503,515	$4,121,424	$4,392,920	$3,946,411	$4,420,616	$3,946,411	$4,317,882	$4,078,492	$1,916,082
TOTAL CONTRACTOR CAPITAL CONTIGENCY		**$16,960,182**	$5,557,500	$5,560,415	$5,182,509	$487,462	$0	$0	$172,296	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL CONTIGENCY		**$105,837,237**	$5,715,081	$5,731,932	$5,494,562	$10,969,517	$6,518,722	$7,855,338	$9,812,534	$7,503,515	$4,121,424	$4,392,920	$3,946,411	$4,420,616	$3,946,411	$4,317,882	$4,078,492	$1,916,082

DEVELOPMENT SCHEDULE																	
		Project Year	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name																	
Conveyors																	
Conveyors		4,644	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	$0	$0	$0
Subtotal (Contractor)	25%	**$28,675,842**	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Contractor)		***$7,168,960***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Access																	
Access		6,342																
Subtotal (Inhouse)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	25%	**$39,164,888**	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Contractor)		***$9,791,222***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawpoints																	
Drawpoints		42,957	2696.683	2148.2	1609.789	3149.101	3599.999	2569.024	0	0	0							
Subtotal (Inhouse)	17%	**$114,222,711**	$7,170,540	$5,712,112	$4,280,465	$8,373,530	$9,572,478	$6,831,092	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$19,146,169***	$1,201,936	$957,472	$717,497	$1,403,583	$1,604,552	$1,145,037	$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	$0	$0	$0
Exctraction Drifts																	
Exctraction Drifts		20,363	903.316	0	216.155	0	0											
Subtotal (Inhouse)	16%	**$84,333,363**	$3,741,050	$0	$895,198	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$13,870,322***	$615,291	$0	$147,233	$0	$0	$0	$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	$0	$0	$0
Secondary Breakage Level																	
Level Access		3,584	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Rockbreaker Chambers		784	0	328.68	455.216	0	0	0			0	0	0	0	0	0	0	0
Subtotal (Inhouse)	18%	**$9,104,923**	$0	$2,018,939	$2,796,195	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$1,634,359***	$0	$345,289	$478,219	$0	$0	$0	$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	$0	$0	$0
Ventilation Infrastructure																	
Exhaust Tunnels		1,717	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Fresh Air Tunnels		4,908	0	0	0	0	0	0	0	0	0	0	0					
Subtotal (Inhouse)	16%	**$14,014,367**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,214,662***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Undercut Excavation																	
Undercut Excavation		20,691	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	19%	**$24,764,987**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$4,681,045***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Haulage Level																	
Haulage Level		5,493	0	704.98	679.665	241.984	0	0					0	0	0	0	0	0
Subtotal (Inhouse)	15%	**$9,252,894**	$0	$1,187,604	$1,144,959	$407,644	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$1,387,934***	$0	$178,141	$171,744	$61,147	$0	$0	$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	$0	$0	$0
Maintenance Area (a.k.a. the shop)																	
Maintenance Area (a.k.a. the shop)		1,361	0											0	0	0	0	0
Subtotal (Inhouse)	17%	**$8,357,625**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$1,429,363***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Preconditioning Level																	
Preconditioning Level		11,754	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	19%	**$14,067,977**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,659,111***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Perimeter																	

DEVELOPMENT SCHEDULE																		
		Project Year	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name																		
Perimeter		3,109	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	16%	**$12,877,381**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,117,945***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Internal Ramps																		
Internal Ramps		2,697	0	0	0	0												
Subtotal (Inhouse)	19%	**$3,227,399**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$610,039***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Vertical Development																		
RAR Raises		1,364	0	0	0	0												
FAR Raises		1,241																
Internal Return Air Raise		3,604	0	0	0	0												
Internal Fresh Air Raise		619	0	48	138	92	0	0	22.793	78.829	207.654	0	0	0	0	0	0	
Orepasses		5,417	0	2109.521	2371.955	0	0	0	0	0	0	0	0					
Crusher Chambers		86	0	0	40							0	0	0	0	0	0	0
Subtotal (Inhouse)	25%	**$72,892,683**	$0	$10,849,181	$12,989,868	$461,008	$0	$0	$114,612	$396,383	$1,044,165	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Inhouse)		***$18,211,138***	$0	$2,709,936	$3,244,722	$115,152	$0	$0	$28,628	$99,010	$260,814	$0	$0	$0	$0	$0	$0	$0
Capital Contingency (Contractor)		***$0***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawbell and Drawpoint Blasting																		
Drawbell and Drawpoint Blasting		1,080																
Subtotal (Inhouse)	23%	**$84,340,923**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$19,658,493***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Undercut Blasting																		
Undercut Blasting		20,691	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	**$12,964,227**	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$2,146,356***	$0	$0	$0	$0	$0	$0	$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	$0	$0	$0
Rehabilition																		
Rehabilition		13,040	360	318	296	339	360	257	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	**$33,436,208**	$923,056	$815,844	$759,169	$869,490	$923,056	$658,710	$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	$0	$0	$0
Capital Contingency (Inhouse)		***$5,708,130***	$157,581	$139,278	$129,603	$148,437	$157,581	$112,453	$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	$0	$0	$0
TOTAL LATERAL DEVELOPMENT METERS		**130,404**	3,600	3,182	2,961	3,391	3,600	2,569	0	0	0	0	0	0	0	0	0	0
TOTAL REHABILITATION METERS (opex)	10%	**13,040**	1,440	1,440	360	361	360	360	360	361	360	360	360	318	296	339	360	257
TOTAL CONTIGENCY METERS (opex)	5%	**6,520**	180	159	148	170	180	128	0	0	0	0	0	0	0	0	0	0
TOTAL VERTICAL DEVELOPMENT METERS		**12,330**	0	2,158	2,550	92	0	0	23	79	208	0	0	0	0	0	0	0
TOTAL UNDERCUT METERS		**20,691**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DRAWBELLS		**#REF!**	120	120	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT ORE TONNES		**5,653,615**	175,650	166,653	169,822	151,162	153,090	109,248	302	1,045	2,752	0	0	0	0	0	0	0
TOTAL DEVELOPMENT INCREMENTAL TONNES		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT WASTE TONNES		**2,238,535**	0	130,650	180,948	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT TONNES		**7,892,150**	175,650	297,303	350,770	151,162	153,090	109,248	302	1,045	2,752	0	0	0	0	0	0	0
TOTAL CONTAINED AU		**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL INHOUSE COST	19%	**$478,946,086**	$12,050,084	$17,567,551	$19,101,887	$9,906,963	$10,710,971	$7,643,542	$114,612	$396,383	$1,044,165	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR COST	25%	**$67,840,730**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	19%	**$546,786,816**	$12,050,084	$17,567,551	$19,101,887	$9,906,963	$10,710,971	$7,643,542	$114,612	$396,383	$1,044,165	$0	$0	$0	$0	$0	$0	$0
TOTAL INHOUSE CAPITAL CONTIGENCY		**$88,877,055**	$1,974,808	$3,806,686	$4,239,056	$1,667,171	$1,762,133	$1,257,490	$28,628	$99,010	$260,814	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CAPITAL CONTIGENCY		**$16,960,182**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL CONTIGENCY		**$105,837,237**	$1,974,808	$3,806,686	$4,239,056	$1,667,171	$1,762,133	$1,257,490	$28,628	$99,010	$260,814	$0	$0	$0	$0	$0	$0	$0

DEVELOPMENT SCHEDULE															
		Project Year	33	34	35	36	37	38	39	40	41	42	43	44	45
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name															
Conveyors															
Conveyors		4,644													
Subtotal (Inhouse)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	25%	**$28,675,842**	$0	$0	$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)		***$7,168,960***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Access															
Access		6,342													
Subtotal (Inhouse)	0%	**$0**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Subtotal (Contractor)	25%	**$39,164,888**	$0	$0	$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)		***$9,791,222***	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Drawpoints															
Drawpoints		42,957													
Subtotal (Inhouse)	17%	**$114,222,711**	$0	$0	$0	$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)		***$19,146,169***	$0	$0	$0	$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
Exctraction Drifts															
Exctraction Drifts		20,363													
Subtotal (Inhouse)	16%	**$84,333,363**	$0	$0	$0	$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)		***$13,870,322***	$0	$0	$0	$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
Secondary Breakage Level															
Level Access		3,584													
Rockbreaker Chambers		784	0	0	0	0									
Subtotal (Inhouse)	18%	**$9,104,923**	$0	$0	$0	$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)		***$1,634,359***	$0	$0	$0	$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
Ventilation Infrastructure															
Exhaust Tunnels		1,717													
Fresh Air Tunnels		4,908													
Subtotal (Inhouse)	16%	**$14,014,367**	$0	$0	$0	$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)		***$2,214,662***	$0	$0	$0	$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
Undercut Excavation															
Undercut Excavation		20,691	0	0	0										
Subtotal (Inhouse)	19%	**$24,764,987**	$0	$0	$0	$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)		***$4,681,045***	$0	$0	$0	$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
Haulage Level															
Haulage Level		5,493	0	0	0										
Subtotal (Inhouse)	15%	**$9,252,894**	$0	$0	$0	$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)		***$1,387,934***	$0	$0	$0	$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
Maintenance Area (a.k.a. the shop)															
Maintenance Area (a.k.a. the shop)		1,361	0	0	0										
Subtotal (Inhouse)	17%	**$8,357,625**	$0	$0	$0	$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)		***$1,429,363***	$0	$0	$0	$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
Preconditioning Level															
Preconditioning Level		11,754	0	0	0										
Subtotal (Inhouse)	19%	**$14,067,977**	$0	$0	$0	$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)		***$2,659,111***	$0	$0	$0	$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
Perimeter															

DEVELOPMENT SCHEDULE

		Project Year	33	34	35	36	37	38	39	40	41	42	43	44	45
		Year	0	0	0	0	0	0	0	0	0	0	0	0	0
Development Name															
Perimeter		3,109	0	0	0										
Subtotal (Inhouse)	16%	$12,877,381	$0	$0	$0	$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)		*$2,117,945*	$0	$0	$0	$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															
Internal Ramps		2,697													
Subtotal (Inhouse)	19%	$3,227,399	$0	$0	$0	$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)		*$610,039*	$0	$0	$0	$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
Vertical Development															
RAR Raises		1,364													
FAR Raises		1,241													
Internal Return Air Raise		3,604													
Internal Fresh Air Raise		619													
Orepasses		5,417													
Crusher Chambers		86	0	0	0										
Subtotal (Inhouse)	25%	$72,892,683	$0	$0	$0	$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)		*$18,211,138*	$0	$0	$0	$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
Drawbell and Drawpoint Blasting															
Drawbell and Drawpoint Blasting		1,080													
Subtotal (Inhouse)	23%	$84,340,923	$0	$0	$0	$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)		*$19,658,493*	$0	$0	$0	$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
Undercut Blasting															
Undercut Blasting		20,691	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	$12,964,227	$0	$0	$0	$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)		*$2,146,356*	$0	$0	$0	$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
Rehabilition															
Rehabilition		13,040	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal (Inhouse)	17%	$33,436,208	$0	$0	$0	$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)		*$5,708,130*	$0	$0	$0	$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
TOTAL LATERAL DEVELOPMENT METERS		130,404	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL REHABILITATION METERS (opex)	10%	13,040	0	0	0	0	0	0							
TOTAL CONTIGENCY METERS (opex)	5%	6,520	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL VERTICAL DEVELOPMENT METERS		12,330	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL UNDERCUT METERS		20,691	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DRAWBELLS		#REF!	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT ORE TONNES		5,653,615	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT INCREMENTAL TONNES		0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT WASTE TONNES		2,238,535	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL DEVELOPMENT TONNES		7,892,150	0	0	0	0	0	0	0	0	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	19%	$478,946,086	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR COST	25%	$67,840,730	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	19%	$546,786,816	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL INHOUSE CAPITAL CONTIGENCY		$88,877,055	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTRACTOR CAPITAL CONTIGENCY		$16,960,182	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL CONTIGENCY		$105,837,237	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0

Mitchell Production Schedule - April 4, 2012

Period	Current tons	To date	Dil %	AUIDW	CUIDW	AGIDW	MOIDW	NSR	AUREC	AGREC	CUREC	MOREC	AUEQ
Yr1	909,991	909,991	0.0	1.00	0.32	6.18	16.54	52.5	78.74	75.53	86.86	1.96	1.831788
Yr2	3,056,765	3,966,757	0.0	0.81	0.26	6.35	15.57	42.4	77.43	73.88	85.13	1.77	1.470158
Yr3	5,825,580	9,792,336	0.0	0.68	0.22	5.88	14.99	35.6	76.26	70.82	83.73	1.54	1.241112
Yr4	9,408,546	19,200,882	0.1	0.63	0.20	5.26	15.67	32.5	75.44	67.90	82.75	2.03	1.145608
Yr5	13,992,188	33,193,070	1.7	0.62	0.20	4.64	16.62	31.4	73.76	64.85	80.86	2.76	1.11697
Yr6	18,827,016	52,020,088	7.2	0.58	0.18	4.09	16.53	29.2	68.95	59.96	75.51	3.00	1.047027
Yr7	20,000,000	72,020,088	9.7	0.54	0.18	3.83	16.53	27.6	66.56	57.54	72.74	3.00	0.992201
Yr8	20,000,000	92,020,088	9.5	0.53	0.17	3.73	17.04	26.7	65.70	56.61	71.68	3.32	0.964716
Yr9	20,000,000	112,020,088	9.3	0.52	0.17	3.61	17.79	26.0	64.50	55.46	70.28	3.89	0.940427
Yr10	20,000,000	132,020,080	9.4	0.51	0.16	3.45	19.07	25.5	63.49	54.40	69.07	4.63	0.920918
Yr11	20,000,000	152,020,080	8.6	0.50	0.16	3.29	21.58	25.1	63.15	53.89	68.64	5.82	0.908372
Yr12	20,000,000	172,020,080	8.5	0.50	0.16	3.15	24.66	25.1	63.69	53.94	69.23	7.16	0.911902
Yr13	20,000,000	192,020,080	8.4	0.51	0.16	3.07	27.43	25.2	64.42	54.24	69.98	8.56	0.917992
Yr14	20,000,000	212,020,080	9.0	0.50	0.16	3.03	28.87	25.0	64.81	54.30	70.36	9.66	0.914255
Yr15	20,000,000	232,020,080	8.2	0.50	0.16	3.09	31.35	25.2	65.76	55.12	71.35	11.29	0.918923
Yr16	20,000,000	252,020,080	9.4	0.51	0.16	3.15	33.03	25.3	65.58	55.16	71.18	12.43	0.921307
Yr17	20,000,000	272,020,064	10.0	0.50	0.16	3.29	37.22	25.4	65.08	55.50	70.65	14.45	0.918051
Yr18	20,000,000	292,020,064	11.2	0.50	0.16	3.40	41.98	25.2	64.28	55.51	69.86	16.36	0.907595
Yr19	20,000,000	312,020,064	10.4	0.50	0.16	3.53	49.62	25.3	64.82	56.41	70.47	19.21	0.905653
Yr20	19,999,998	332,020,064	11.0	0.49	0.16	3.56	53.24	25.0	64.39	55.99	70.04	20.83	0.893442
Yr21	20,000,000	352,020,064	8.8	0.51	0.16	3.83	56.17	26.0	65.90	57.64	71.83	22.53	0.922734
Yr22	20,000,000	372,020,064	7.6	0.53	0.16	3.78	55.39	26.5	66.15	57.64	72.33	23.67	0.939893
Yr23	19,029,696	391,049,760	8.2	0.54	0.16	3.38	49.80	26.5	64.91	55.41	71.25	23.33	0.946459
Yr24	16,919,538	407,969,312	10.0	0.54	0.16	2.88	45.02	26.0	64.44	53.20	70.55	22.46	0.942323
Yr25	13,517,387	421,486,688	12.4	0.54	0.15	2.49	45.63	25.3	63.41	50.63	68.78	21.86	0.921896
Yr26	8,936,915	430,423,616	14.4	0.55	0.13	2.15	52.09	24.5	62.30	47.97	66.55	22.80	0.895906
Yr27	4,734,489	435,158,112	16.0	0.55	0.12	1.91	60.89	23.5	61.11	45.78	64.26	25.52	0.86083
Yr28	2,067,905	437,226,016	19.3	0.55	0.10	1.64	66.12	22.2	58.43	42.89	60.50	27.50	0.815161
Yr29	648,561	437,874,560	26.5	0.50	0.09	1.45	63.94	20.0	53.25	38.96	54.58	26.57	0.735691
Yr30	91,702	437,966,272	33.5	0.46	0.08	1.40	58.27	17.8	48.23	36.10	48.64	24.22	0.654506

APPENDIX L

Capital Cost Schedule by Year and an Example of Development Cost Calculation

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		437,966,277	0	0	0	0	0	0	909,991	3,056,765	5,825,580	9,408,546	13,992,188
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		437,874,575	0	0	0	0	0	0	909,991	3,056,765	5,825,580	9,408,546	13,992,188
NSR	$/t			0.00	0.00	0.00	0.00	0.00	0.00	52.47	42.36	35.56	32.47	31.36
AUGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	0.00	1.00	0.81	0.68	0.63	0.62
CU%	%			0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	32.40%	25.70%	21.72%	20.09%	19.55%
AGGPT (gpt)	gpt			0.00	0.00	0.00	0.00	0.00	0.00	6.18	6.35	5.88	5.26	4.64
MOPPM (ppm)	ppm			0.00	0.00	0.00	0.00	0.00	0.00	16.54	15.57	14.99	15.67	16.62
Lateral Development	m		130,404	3,600	3,602	3,546	13,893	14,400	14,440	14,400	14,400	3,600	3,610	3,600
Vertical Development	m		12,330	0	6	40	4,098	1,158	0	1,679	241	0	0	0
Total Development	m		142,735	3,600	3,608	3,586	17,991	15,558	14,440	16,079	14,641	3,600	3,610	3,600
CAPITAL COSTS - CAPEX	**Accuracy (%)**													
In house														
Development Equipment		20%	$59,473,839	$4,369,403	$5,219,503	$4,100,000	$5,752,607	$1,650,393	$4,369,403	$5,219,503	$4,100,000	$3,966,314	$0	$784,987
Production Equipment		20%	$139,759,500	$0	$0	$0	$0	$0	$0	$0	$0	$2,989,500	$12,199,500	$8,702,500
Support Equipment		20%	$45,131,214	$0	$1,015,708	$2,495,934	$594,300	$414,650	$0	$1,015,708	$2,495,934	$594,300	$3,134,882	$2,747,872
Stationary Equipment		20%	$543,696,100	$1,080,000	$0	$20,000	$270,000	$65,051,100	$21,100,000	$21,000,000	$65,560,000	$21,000,000	$22,060,000	$34,527,000
Additional Equipment/Infrastructure		25%	$103,413,900	$1,096,600	$0	$0	$7,496,600	$47,425,007	$26,802,493	$2,400,000	$2,400,000	$2,400,000	$2,400,000	$896,600
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		21%	$891,474,553	$6,546,003	$6,235,211	$6,615,934	$14,113,507	$114,541,150	$52,271,896	$29,635,211	$74,555,934	$30,950,114	$39,794,382	$47,658,959
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		21%	$891,474,553	$6,546,003	$6,235,211	$6,615,934	$14,113,507	$114,541,150	$52,271,896	$29,635,211	$74,555,934	$30,950,114	$39,794,382	$47,658,959
DEVELOPMENT COSTS - CAPEX	**Accuracy (%)**													
Fixed Development Costs														
In house		22%	$39,401,628	$903,864	$1,712,106	$2,012,791	$2,834,649	$5,166,815	$5,369,594	$5,369,594	$5,369,594	$5,132,124	$5,530,498	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs														
In house		18%	$505,661,529	$1,138,494	$1,194,505	$1,853,974	$57,896,560	$35,876,357	$48,147,390	$57,325,373	$53,741,547	$21,256,605	$22,892,572	$20,082,188
Contractor		25%	$67,840,730	$22,230,000	$22,241,658	$20,730,037	$1,949,849	$0	$0	$689,186	$0	$0	$0	$0
Labour														
In house		25%	$619,168,200	$18,941,400	$24,014,400	$33,553,350	$46,852,350	$48,162,750	$48,162,750	$35,519,700	$27,309,900	$23,378,700	$23,378,700	$23,378,700
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		22%	$1,164,231,357	$20,983,758	$26,921,011	$37,420,115	$107,583,558	$89,205,922	$101,679,733	$98,214,667	$86,421,041	$49,767,428	$51,801,769	$43,460,888
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$67,840,730	$22,230,000	$22,241,658	$20,730,037	$1,949,849	$0	$0	$689,186	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		22%	$1,232,072,087	$43,213,758	$49,162,670	$58,150,152	$109,533,407	$89,205,922	$101,679,733	$98,903,853	$86,421,041	$49,767,428	$51,801,769	$43,460,888
PRODUCTION COSTS - OPEX	**Accuracy (%)**													
Fixed Production Costs														
In house		22%	$156,595,899	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$6,297,905
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			7.2%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	8%
Variable Production Costs														
In house		24%	$890,682,815	$0	$0	$0	$0	$0	$0	$1,818,582	$6,108,824	$11,642,191	$18,802,609	$28,393,711
Percentage			40.7%	0%	0%	0%	0%	0%	0%	4%	12%	21%	29%	35%
Labour														
In house		25%	$1,140,546,450	$0	$0	$0	$0	$0	$0	$42,826,650	$42,826,650	$44,137,050	$45,447,450	$46,757,850
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			52.1%	0%	0%	0%	0%	0%	0%	96%	88%	79%	71%	57%
TOTAL IN HOUSE PRODUCTION COSTS		24%	$2,187,825,164	$0	$0	$0	$0	$0	$0	$44,645,232	$48,935,474	$55,779,241	$64,250,059	$81,449,467
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$2,187,825,164	$0	$0	$0	$0	$0	$0	$44,645,232	$48,935,474	$55,779,241	$64,250,059	$81,449,467
TOTAL COSTS	**Accuracy (%)**													
TOTAL IN HOUSE COST		23%	**$4,243,531,074**	$27,529,761	$33,156,222	$44,036,049	$121,697,065	$203,747,072	$153,951,629	$172,495,110	$209,912,450	$136,496,784	$155,846,210	$172,569,314
TOTAL CONTRACTOR COST		25%	**$67,840,730**	$22,230,000	$22,241,658	$20,730,037	$1,949,849	$0	$0	$689,186	$0	$0	$0	$0
TOTAL COST		23%	**$4,311,371,804**	$49,759,761	$55,397,881	$64,766,086	$123,646,914	$203,747,072	$153,951,629	$173,184,296	$209,912,450	$136,496,784	$155,846,210	$172,569,314
TOTAL IN HOUSE CONTINGENCY			**$967,416,874**	$6,276,084	$7,558,773	$10,039,096	$27,743,828	$46,449,137	$35,097,046	$39,324,486	$47,854,686	$31,117,786	$35,528,962	$39,341,403
TOTAL CONTRACTOR CONTINGENCY			**$16,960,182**	$5,557,500	$5,560,415	$5,182,509	$487,462	$0	$0	$172,296	$0	$0	$0	$0
TOTAL CONTINGENCY			**$984,377,056**	$11,361,202	$12,648,504	$14,787,463	$28,231,197	$46,519,751	$35,150,402	$39,541,625	$47,927,437	$31,165,093	$35,582,975	$39,401,212
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$12.09**	-	-	-	-	-	-	$233.77	$84.35	$28.78	$20.35	$15.15
TOTAL COST PER TONNE (ORE) - Operating			**$5.00**	-	-	-	-	-	-	$49.06	$16.01	$9.57	$6.83	$5.82

COST SCHEDULE													
			Project Year	12	13	14	15	16	17	18	19	20	21
			Year	0	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**												
Production Ore	tonnes		437,966,277	18,827,016	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000
Production Incremental	tonnes		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
Production Total	tonnes		437,874,575	18,827,016	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000
NSR	$/t			29.23	27.56	26.74	26.05	25.47	25.06	25.07	25.18	25.03	25.16
AUGPT (gpt)	gpt			0.58	0.54	0.53	0.52	0.51	0.50	0.50	0.51	0.50	0.50
CU%	%			18.37%	17.54%	17.10%	16.59%	16.11%	15.84%	15.97%	16.09%	16.05%	16.18%
AGGPT (gpt)	gpt			4.09	3.83	3.73	3.61	3.45	3.29	3.15	3.07	3.03	3.09
MOPPM (ppm)	ppm			16.53	16.53	17.04	17.79	19.07	21.58	24.66	27.43	28.87	31.35
Lateral Development	m		130,404	3,600	3,600	3,610	3,600	3,600	3,600	3,182	2,961	3,391	3,600
Vertical Development	m		12,330	0	0	0	0	0	0	2,158	2,550	92	0
Total Development	m		142,735	3,600	3,600	3,610	3,600	3,600	3,600	5,339	5,511	3,483	3,600
CAPITAL COSTS - CAPEX		**Accuracy (%)**											
In house													
Development Equipment		20%	$59,473,839	$4,369,403	$1,650,393	$381,898	$0	$784,987	$4,369,403	$0	$381,898	$0	$403,089
Production Equipment		20%	$139,759,500	$6,709,500	$3,720,000	$3,720,000	$12,930,000	$9,209,500	$3,720,000	$3,720,000	$3,720,000	$6,920,000	$6,709,500
Support Equipment		20%	$45,131,214	$1,015,708	$2,495,934	$594,300	$3,430,350	$0	$1,015,708	$2,495,934	$359,300	$3,134,882	$530,468
Stationary Equipment		20%	$543,696,100	$21,000,000	$21,000,000	$21,000,000	$21,000,000	$21,100,000	$34,577,000	$0	$0	$1,060,000	$57,827,000
Additional Equipment/Infrastructure		25%	$103,413,900	$0	$0	$0	$0	$0	$96,600	$0	$0	$0	$0
Contractor													
Development Equipment		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
Support Equipment		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
TOTAL IN HOUSE CAPITAL COSTS		21%	$891,474,553	$33,094,611	$28,866,327	$25,696,198	$37,360,350	$31,094,487	$43,778,711	$6,215,934	$4,461,198	$11,114,882	$65,470,057
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		21%	$891,474,553	$33,094,611	$28,866,327	$25,696,198	$37,360,350	$31,094,487	$43,778,711	$6,215,934	$4,461,198	$11,114,882	$65,470,057
DEVELOPMENT COSTS - CAPEX		**Accuracy (%)**											
Fixed Development Costs													
In house		22%	$39,401,628	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Variable Development Costs													
In house		18%	$505,661,529	$23,068,030	$20,082,188	$22,420,096	$20,913,841	$11,680,311	$12,050,084	$20,774,094	$23,043,040	$10,314,607	$10,710,971
Contractor		25%	$67,840,730	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Labour													
In house		25%	$619,168,200	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700	$23,378,700
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		22%	$1,164,231,357	$46,446,730	$43,460,888	$45,798,796	$44,292,541	$35,059,011	$35,428,784	$44,152,794	$46,421,740	$33,693,307	$34,089,671
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$67,840,730	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		22%	$1,232,072,087	$46,446,730	$43,460,888	$45,798,796	$44,292,541	$35,059,011	$35,428,784	$44,152,794	$46,421,740	$33,693,307	$34,089,671
PRODUCTION COSTS - OPEX		**Accuracy (%)**											
Fixed Production Costs													
In house		22%	$156,595,899	$6,513,787	$6,729,669	$6,945,550	$6,935,625	$6,935,625	$6,903,674	$6,882,374	$6,882,374	$6,882,374	$6,882,374
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			7.2%	7%	7%	7%	7%	7%	7%	7%	7%	7%	7%
Variable Production Costs													
In house		24%	$890,682,815	$38,056,151	$40,393,617	$41,632,083	$41,692,711	$41,697,498	$41,692,711	$41,692,711	$40,400,087	$40,401,284	$40,400,087
Percentage			40.7%	42%	43%	44%	43%	44%	44%	44%	44%	44%	44%
Labour													
In house		25%	$1,140,546,450	$46,757,850	$46,757,850	$46,757,850	$47,413,050	$45,385,050	$45,385,050	$45,385,050	$45,385,050	$45,385,050	$45,385,050
Contractor		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage			52.1%	51%	50%	49%	49%	48%	48%	48%	49%	49%	49%
TOTAL IN HOUSE PRODUCTION COSTS		24%	$2,187,825,164	$91,327,788	$93,881,136	$95,335,483	$96,041,386	$94,018,173	$93,981,435	$93,960,135	$92,667,511	$92,668,708	$92,667,511
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$2,187,825,164	$91,327,788	$93,881,136	$95,335,483	$96,041,386	$94,018,173	$93,981,435	$93,960,135	$92,667,511	$92,668,708	$92,667,511
TOTAL COSTS		**Accuracy (%)**											
TOTAL IN HOUSE COST		23%	**$4,243,531,074**	$170,869,129	$166,208,351	$166,830,477	$177,694,277	$160,171,671	$173,188,930	$144,328,862	$143,550,449	$137,476,897	$192,227,239
TOTAL CONTRACTOR COST		25%	**$67,840,730**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST		23%	**$4,311,371,804**	$170,869,129	$166,208,351	$166,830,477	$177,694,277	$160,171,671	$173,188,930	$144,328,862	$143,550,449	$137,476,897	$192,227,239
TOTAL IN HOUSE CONTINGENCY			**$967,416,874**	$38,953,804	$37,891,266	$38,033,095	$40,509,764	$36,515,057	$39,482,660	$32,903,300	$32,725,842	$31,341,227	$43,822,909
TOTAL CONTRACTOR CONTINGENCY			**$16,960,182**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY			**$984,377,056**	$39,013,024	$37,948,870	$38,090,914	$40,571,349	$36,570,569	$39,542,683	$32,953,321	$32,775,593	$31,388,873	$43,889,530
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$12.09**	$11.15	$10.21	$10.25	$10.91	$9.84	$10.64	$8.86	$8.82	$8.44	$11.81
TOTAL COST PER TONNE (ORE) - Operating			**$5.00**	$4.85	$4.69	$4.77	$4.80	$4.70	$4.70	$4.70	$4.63	$4.63	$4.63

COST SCHEDULE													
		Project Year	22	23	24	25	26	27	28	29	30	31	32
		Year	0	0	0	0	0	0	0	0	0	0	0
PHYSICALS	**Unit**												
Production Ore	tonnes	437,966,277	20,000,000	20,000,000	20,000,000	20,000,000	19,999,998	20,000,000	20,000,000	19,029,696	16,919,538	13,517,387	8,936,915
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	437,874,575	20,000,000	20,000,000	20,000,000	20,000,000	19,999,998	20,000,000	20,000,000	19,029,696	16,919,538	13,517,387	8,936,915
NSR	$/t		25.30	25.37	25.25	25.32	25.04	26.03	26.50	26.48	26.04	25.30	24.52
AUGPT (gpt)	gpt		0.51	0.50	0.50	0.50	0.49	0.51	0.53	0.54	0.54	0.54	0.55
CU%	%		16.24%	16.18%	15.99%	16.02%	15.78%	16.06%	15.98%	15.81%	15.60%	14.75%	13.43%
AGGPT (gpt)	gpt		3.15	3.29	3.40	3.53	3.56	3.83	3.78	3.38	2.88	2.49	2.15
MOPPM (ppm)	ppm		33.03	37.22	41.98	49.62	53.24	56.17	55.39	49.80	45.02	45.63	52.09
Lateral Development	m	130,404	2,569	0	0	0	0	0	0	0	0	0	0
Vertical Development	m	12,330	0	23	79	208	0	0	0	0	0	0	0
Total Development	m	142,735	2,569	23	79	208	0	0	0	0	0	0	0
CAPITAL COSTS - CAPEX	**Accuracy (%)**												
In house													
Development Equipment	20%	$59,473,839	$784,987	$381,898	$0	$381,898	$403,089	$0	$381,898	$381,898	$0	$4,884,987	$0
Production Equipment	20%	$139,759,500	$3,720,000	$3,720,000	$3,720,000	$6,920,000	$9,209,500	$3,720,000	$3,720,000	$3,720,000	$9,420,000	$6,920,000	$0
Support Equipment	20%	$45,131,214	$1,015,708	$2,495,934	$359,300	$3,195,350	$235,000	$1,250,708	$1,311,176	$2,730,934	$235,000	$2,720,232	$0
Stationary Equipment	20%	$543,696,100	$0	$0	$0	$0	$100,000	$57,877,000	$0	$0	$1,060,000	$34,327,000	$100,000
Additional Equipment/Infrastructure	25%	$103,413,900	$0	$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
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	21%	$891,474,553	$5,520,695	$6,597,832	$4,079,300	$10,497,248	$9,947,589	$62,847,708	$5,413,074	$6,832,832	$10,715,000	$48,852,219	$100,000
TOTAL CONTRACTOR CAPITAL COSTS	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS	21%	$891,474,553	$5,520,695	$6,597,832	$4,079,300	$10,497,248	$9,947,589	$62,847,708	$5,413,074	$6,832,832	$10,715,000	$48,852,219	$100,000
DEVELOPMENT COSTS - CAPEX	**Accuracy (%)**												
Fixed Development Costs													
In house	22%	$39,401,628	$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	18%	$505,661,529	$7,643,542	$114,612	$396,383	$1,044,165	$0	$0	$0	$0	$0	$0	$0
Contractor	25%	$67,840,730	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Labour													
In house	25%	$619,168,200	$23,074,500	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS	22%	$1,164,231,357	$30,718,042	$1,080,012	$1,361,783	$2,009,565	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400
TOTAL CONTRACTOR DEVELOPMENT COSTS	25%	$67,840,730	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS	22%	$1,232,072,087	$30,718,042	$1,080,012	$1,361,783	$2,009,565	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400	$965,400
PRODUCTION COSTS - OPEX	**Accuracy (%)**												
Fixed Production Costs													
In house	22%	$156,595,899	$6,882,374	$6,624,467	$6,624,467	$6,624,467	$6,624,467	$6,624,467	$6,624,467	$6,327,399	$6,327,399	$5,815,507	$5,402,638
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage		7.2%	7%	7%	7%	7%	7%	8%	8%	8%	8%	9%	11%
Variable Production Costs													
In house	24%	$890,682,815	$40,400,087	$40,400,087	$40,401,284	$40,400,087	$40,400,083	$40,400,087	$40,350,041	$38,384,471	$34,218,900	$27,444,840	$18,167,552
Percentage		40.7%	44%	43%	43%	46%	46%	46%	47%	47%	45%	42%	37%
Labour													
In house	25%	$1,140,546,450	$45,385,050	$46,408,050	$46,408,050	$41,462,850	$41,462,850	$40,152,450	$39,138,450	$37,172,850	$35,429,250	$31,689,150	$25,732,650
Contractor	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Percentage		52.1%	49%	50%	50%	47%	47%	46%	45%	45%	47%	49%	52%
TOTAL IN HOUSE PRODUCTION COSTS	24%	$2,187,825,164	$92,667,511	$93,432,604	$93,433,801	$88,487,404	$88,487,400	$87,177,004	$86,112,958	$81,884,720	$75,975,549	$64,949,497	$49,302,840
TOTAL CONTRACTOR PRODUCTION COSTS	0%	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS	24%	$2,187,825,164	$92,667,511	$93,432,604	$93,433,801	$88,487,404	$88,487,400	$87,177,004	$86,112,958	$81,884,720	$75,975,549	$64,949,497	$49,302,840
TOTAL COSTS	**Accuracy (%)**												
TOTAL IN HOUSE COST	23%	**$4,243,531,074**	$128,906,248	$101,110,448	$98,874,884	$100,994,217	$99,400,389	$150,990,112	$92,491,432	$89,682,952	$87,655,949	$114,767,116	$50,368,240
TOTAL CONTRACTOR COST	25%	**$67,840,730**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL COST	23%	**$4,311,371,804**	$128,906,248	$101,110,448	$98,874,884	$100,994,217	$99,400,389	$150,990,112	$92,491,432	$89,682,952	$87,655,949	$114,767,116	$50,368,240
TOTAL IN HOUSE CONTINGENCY		**$967,416,874**	$29,387,337	$23,050,604	$22,540,952	$23,024,106	$22,660,754	$34,421,895	$21,085,688	$20,445,426	$19,983,321	$26,163,976	$11,482,674
TOTAL CONTRACTOR CONTINGENCY		**$16,960,182**	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TOTAL CONTINGENCY		**$984,377,056**	$29,432,013	$23,085,647	$22,575,220	$23,059,108	$22,695,204	$34,474,225	$21,117,743	$20,476,508	$20,013,701	$26,203,752	$11,500,131
TOTAL COST PER TONNE (ORE) - Capital & Operating		**$12.09**	$7.92	$6.21	$6.07	$6.20	$6.10	$9.27	$5.68	$5.79	$6.36	$10.43	$6.92
TOTAL COST PER TONNE (ORE) - Operating		**$5.00**	$4.63	$4.67	$4.67	$4.42	$4.42	$4.36	$4.31	$4.30	$4.49	$4.80	$5.52

COST SCHEDULE								
			Project Year	33	34	35	36	37
			Year	0	0	0	0	0
PHYSICALS	**Unit**							
Production Ore	tonnes		437,966,277	4,734,489	2,067,905	648,561	91,702	0
Production Incremental	tonnes		0	0	0	0	0	0
Production Waste	tonnes		0	0	0	0	0	0
Production Total	tonnes		437,874,575	4,734,489	2,067,905	648,561	0	0
NSR	$/t			23.51	22.22	20.04	17.84	0.00
AUGPT (gpt)	gpt			0.55	0.55	0.50	0.46	0.00
CU%	%			12.01%	10.49%	9.08%	7.68%	0.00%
AGGPT (gpt)	gpt			1.91	1.64	1.45	1.40	0.00
MOPPM (ppm)	ppm			60.89	66.12	63.94	58.27	0.00
Lateral Development	m		130,404	0	0	0	0	0
Vertical Development	m		12,330	0	0	0	0	0
Total Development	m		142,735	0	0	0	0	0
CAPITAL COSTS - CAPEX		**Accuracy (%)**						
In house								
Development Equipment		20%	$59,473,839	$0	$0	$0	$0	$0
Production Equipment		20%	$139,759,500	$0	$0	$0	$0	$0
Support Equipment		20%	$45,131,214	$0	$0	$0	$0	$0
Stationary Equipment		20%	$543,696,100	$0	$0	$0	$0	$0
Additional Equipment/Infrastructure		25%	$103,413,900	$0	$0	$0	$0	$10,000,000
Contractor								
Development Equipment		0%	$0	$0	$0	$0	$0	$0
Production Equipment		0%	$0	$0	$0	$0	$0	$0
Support Equipment		0%	$0	$0	$0	$0	$0	$0
Stationary Equipment		0%	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE CAPITAL COSTS		21%	$891,474,553	$0	$0	$0	$0	$10,000,000
TOTAL CONTRACTOR CAPITAL COSTS		0%	$0	$0	$0	$0	$0	$0
TOTAL CAPITAL COSTS		21%	$891,474,553	$0	$0	$0	$0	$10,000,000
DEVELOPMENT COSTS - CAPEX		**Accuracy (%)**						
Fixed Development Costs								
In house		22%	$39,401,628	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0
Variable Development Costs								
In house		18%	$505,661,529	$0	$0	$0	$0	$0
Contractor		25%	$67,840,730	$0	$0	$0	$0	$0
Labour								
In house		25%	$619,168,200	$0	$0	$0	$0	$0
Contractor		0%	$0	$0	$0	$0	$0	$0
TOTAL IN HOUSE DEVELOPMENT COSTS		22%	$1,164,231,357	$0	$0	$0	$0	$0
TOTAL CONTRACTOR DEVELOPMENT COSTS		25%	$67,840,730	$0	$0	$0	$0	$0
TOTAL DEVELOPMENT COSTS		22%	$1,232,072,087	$0	$0	$0	$0	$0
PRODUCTION COSTS - OPEX		**Accuracy (%)**						
Fixed Production Costs								
In house		22%	$156,595,899	$4,881,827	$3,316,055	$2,422,896	$681,667	$0
Contractor		0%	$0	$0	$0	$0	$0	$0
Percentage			7.2%	16%	20%	44%	72%	#DIV/0!
Variable Production Costs								
In house		24%	$890,682,815	$9,461,690	$4,132,627	$1,296,124	$0	$0
Percentage			40.7%	30%	25%	24%	0%	#DIV/0!
Labour								
In house		25%	$1,140,546,450	$17,139,750	$8,938,050	$1,770,450	$264,000	$0
Contractor		0%	$0	$0	$0	$0	$0	$0
Percentage			52.1%	54%	55%	32%	28%	#DIV/0!
TOTAL IN HOUSE PRODUCTION COSTS		24%	$2,187,825,164	$31,483,267	$16,386,731	$5,489,471	$945,667	$0
TOTAL CONTRACTOR PRODUCTION COSTS		0%	$0	$0	$0	$0	$0	$0
TOTAL PRODUCTION COSTS		24%	$2,187,825,164	$31,483,267	$16,386,731	$5,489,471	$945,667	$0
TOTAL COSTS		**Accuracy (%)**						
TOTAL IN HOUSE COST		23%	**$4,243,531,074**	$31,483,267	$16,386,731	$5,489,471	$945,667	$10,000,000
TOTAL CONTRACTOR COST		25%	**$67,840,730**	$0	$0	$0	$0	$0
TOTAL COST		23%	**$4,311,371,804**	$31,483,267	$16,386,731	$5,489,471	$945,667	$10,000,000
TOTAL IN HOUSE CONTINGENCY			**$967,416,874**	$7,177,382	$3,735,757	$1,251,459	$215,588	$2,279,745
TOTAL CONTRACTOR CONTINGENCY			**$16,960,182**	$0	$0	$0	$0	$0
TOTAL CONTINGENCY			**$984,377,056**	$7,188,293	$3,741,436	$1,253,362	$215,916	$2,283,211
TOTAL COST PER TONNE (ORE) - Capital & Operating			**$12.09**	$8.17	$9.73	$10.40	$12.67	-
TOTAL COST PER TONNE (ORE) - Operating			**$5.00**	$6.65	$7.92	$8.46	$10.31	-

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	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	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	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	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	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	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	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	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	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	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			Cost	Accuracy	Contingency
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	Cost	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	Accuracy	Contingency
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***

APPENDIX M

OPEX Calculation and Operating Cost by Year

NON LABOUR PRODUCTION OPERATING COSTS

PRODUCTION OPERATING COSTS SUMMARY

Equipmnent	Hourly Machine Capacity (tonnes)	Op Cost ($/tonne)	Accuracy (%)	$/hr
Production LHD	313	CAD 0.559	25%	219.02
Production Locomotive	3148	CAD 0.024	21%	91.58
Crusher	1556	CAD 0.267	25%	517.92
Conveyor 1 and 2	3111	CAD 0.268	23%	1023.55
Conveyor 3	3111	CAD 0.265	23%	1011.61
Block Holer	291	CAD 0.132	20%	45.91
Stationary Rockbreaker	0	CAD 0.293	25%	0.00
Mobile Rockbreaker	#REF!	CAD 0.191	25%	0.00

Production Equipment

LHD

Type:	Production LHD	Equipment:	Sandvik LH517			CAD$/t	Accuracy	Contingency
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	100	10	0.60					
Gradient Laden		5.8	0.00					
Flat/Down Gradient Empty	100	10	0.60					
Gradient Empty		7.5	0.00					
Total Tramming			1.20					
Total Cycle Time	mins	3.20						
Hourly Machine Capacity	tonnes/hour	313.47						
Number of Operating LHDs	each	10.00	80% mechanical availability					
Daily Machine Capacity	tonnes/day	56,000						
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
45	100%	45	CAD 45.00			CAD 0.14	25%	CAD 0.04
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 0.42	25%	CAD 0.10
Operating Cost per tonne						**CAD 0.56**	25%	**CAD 0.14**

Train Cost

Train Type	Production Locomotive	Equipment:	Trolley wire Loco - 75 T			CAD$/t	Accuracy	Contingency
Car Type	Production Rail Car							
Quantity of Cars	54							
Car Capacity	tonnes	35						
	m^3	14						
	Swell factor	1.25						
	Individual car tonnes at volume	30.24	Uses ore density					
Loading time	mins	3						
Tipping Time	mins	3						
Tramming Time								
Gradient	Tramming Distance (m)	Speed (km/h)	Duration (mins)					
Flat Laden	3140	7.5	25.12					
Gradient Laden		6	0.00					
Flat/Down Gradient Empty		20	0.00					
Gradient Empty		14	0.00					
Total Tramming			25.12					
Total Cycle Time	mins	31.12						
Hourly Capacity	tonnes/hour	3148						
Daily System Capacity	tonnes/day	56,671						
Production Rail Car - Maintenance Cost								
Maintenance Cost per hour								
CAD 12.00						CAD 0.00	25%	CAD 0.00
Production Locomotive - Maintenance Cost								
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	25%	CAD 0.00
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
950	92%	90%	80%	629	CAD 33.98	CAD 0.01	15%	CAD 0.00
Maintenance Cost per hour								
CAD 30.00						CAD 0.01	25%	CAD 0.00
Operating Cost per tonne						**CAD 0.02**	21%	**CAD 0.00**

NON LABOUR PRODUCTION OPERATING COSTS

PRODUCTION OPERATING COSTS SUMMARY

Equipmnent	Hourly Machine Capacity (tonnes)	Op Cost ($/tonne)	Accuracy (%)	$/hr
Production LHD	313	CAD 0.559	25%	219.02
Production Locomotive	3148	CAD 0.024	21%	91.58
Crusher	1556	CAD 0.267	25%	517.92
Conveyor 1 and 2	3111	CAD 0.268	23%	1023.55
Conveyor 3	3111	CAD 0.265	23%	1011.61
Block Holer	291	CAD 0.132	20%	45.91
Stationary Rockbreaker	0	CAD 0.293	25%	0.00
Mobile Rockbreaker	#REF!	CAD 0.191	25%	0.00

Stationary Equipment								
Crusher/Sizer								
Type:	Crusher	Equipment:	(42 x 65) includes acc. And installationa			CAD$/t	Accuracy	Contingency
Nominal Throughput	tonnes/hr	1,555.56	2 shifts per day, 2 crushers operating					
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	*25%*	*CAD 0.00*
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
450	75%	95%	90%	289	CAD 15.58	CAD 0.01	*15%*	*CAD 0.00*
Maintenance Cost per hour								
CAD 400.00						CAD 0.26	*25%*	*CAD 0.06*
Operating Cost per tonne						**CAD 0.27**	25%	***CAD 0.07***
Conveyor 1 and 2								
Type:	Conveyor 1 and 2	Equipment:	Mitchell Crusher station conveyors 1 and 2 - installed price			CAD$/t	Accuracy	Contingency
Nominal Throughput	tonnes/hr	3,111.11	2 shifts per day, 1 conveyor					
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	*25%*	*CAD 0.00*
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
5300	75%	95%	90%	3399	CAD 183.53	CAD 0.06	*15%*	*CAD 0.01*
Maintenance Cost per hour								
CAD 650.00						CAD 0.21	*25%*	*CAD 0.05*
Operating Cost per tonne						**CAD 0.27**	23%	***CAD 0.06***
Conveyor 3								
Type:	Conveyor 3	Equipment:	Mitchell Booster station conveyor - installed price			CAD$/t	Accuracy	Contingency
Nominal Throughput	tonnes/hr	3,111.11	2 shifts per day, 1 conveyors					
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	*25%*	*CAD 0.00*
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
5000	75%	95%	90%	3206	CAD 173.14	CAD 0.06	*15%*	*CAD 0.01*
Maintenance Cost per hour								
CAD 650.00						CAD 0.21	*25%*	*CAD 0.05*
Operating Cost per tonne						**CAD 0.26**	23%	***CAD 0.06***
Stationary Rockbreaker								
Type:	Stationary Rockbreaker	Equipment:	Wardrop including installation			CAD$/shift	Accuracy	Contingency
Quantity of rockbreakers	30							
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
0	0%	0	CAD 0.00			CAD 0.00	*25%*	*CAD 0.00*
Electrical power (kW)	Power Utilisation (%)	Power Factor (%)	Load Factor (%)	Effective Power (kW)	Power Cost per hour			
60	50%	95%	90%	26	CAD 1.39	CAD 12.47	*15%*	*CAD 1.87*
Maintenance Cost per hour								
CAD 29.00						CAD 261.00	*25%*	*CAD 65.25*
Operating Cost per shift						**CAD 8,203.98**	25%	***CAD 2,013.60***
Operating Cost per hour								
Operating Cost per tonne						**CAD 0.29**	25%	***CAD 0.07***
Secondary Breaking Equipment								
Block Holer								
Type:	Block Holer	Equipment:	Maclean BH3 Blockholes					
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 BH3 Blockholes					
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					
20	25%	5	CAD 5.00			CAD 45.00	*25%*	*CAD 11.25*
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 158.60						CAD 1,427.40	*25%*	*CAD 356.85*
Subtotal fuel/electricity cost per shift						**CAD 1,472.40**		***CAD 368.10***
TOTAL DRILLING COST per shift						**CAD 1,604.38**		***CAD 387.90***
TOTAL DRILLING COST per tonne						**CAD 0.06**	24%	***CAD 0.01***

NON LABOUR PRODUCTION OPERATING COSTS

PRODUCTION OPERATING COSTS SUMMARY

Equipmnent	Hourly Machine Capacity (tonnes)	Op Cost ($/tonne)	Accuracy (%)	$/hr
Production LHD	313	CAD 0.559	25%	219.02
Production Locomotive	3148	CAD 0.024	21%	91.58
Crusher	1556	CAD 0.267	25%	517.92
Conveyor 1 and 2	3111	CAD 0.268	23%	1023.55
Conveyor 3	3111	CAD 0.265	23%	1011.61
Block Holer	291	CAD 0.132	20%	45.91
Stationary Rockbreaker	0	CAD 0.293	25%	0.00
Mobile Rockbreaker	#REF!	CAD 0.191	25%	0.00

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	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	0.13		CAD 27.45	CAD 3.46	*15%*	*CAD 0.52*
Subtotal consumable cost for excavation per shift						**CAD 1,792.14**		***CAD 268.82***

Fuel/Electricity Costs

Type:	ANFO Loader	Equipment:	BTI ANFO Loader	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					
10	25%	2.5	CAD 2.50			CAD 22.50	*25%*	*CAD 5.63*
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 30.45						CAD 274.05	*25%*	*CAD 68.51*
Subtotal fuel/electricity cost per shift						**CAD 296.55**		***CAD 74.14***
TOTAL BLASTING COST per shift						**CAD 2,088.69**		***CAD 342.96***
TOTAL BLASTING COST per tonne						**CAD 0.07**		***CAD 342.96***
Operating Cost per shift						**CAD 3,693.07**	20%	***CAD 730.85***
Operating Cost per tonne						**CAD 0.13**	20%	***CAD 0.03***

Mobile Rockbreaker

Type:	Mobile Rockbreaker	Equipment:	Sandvik LH410			CAD$/shift	Accuracy	Contingency
Quantity of Rockbreakers	4							
Fuel Consmption (l/hr)	Diesel Utilisation (%)	Eff Consumption (l/hr)	Fuel Cost per hour					
45	100%	45	CAD 45.00			CAD 405.00	*25%*	*CAD 101.25*
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 103.41						CAD 930.69	*25%*	*CAD 232.67*
Operating Cost per shift						**CAD 5,342.76**	25%	***CAD 1,335.69***
Operating Cost per hour								
Operating Cost per tonne						**CAD 0.19**	25%	***CAD 0.05***

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Africa	+ 27 11 254 4800
Asia	+ 86 21 6258 5522
Australasia	+ 61 3 8862 3500
Europe	+ 356 21 42 30 20
North America	+ 1 800 275 3281
South America	+ 55 21 3095 9500

solutions@golder.com
www.golder.com

Golder Associates Ltd.
500 - 4260 Still Creek Drive
Burnaby, British Columbia, V5C 6C6
Canada
T: +1 (604) 296 4200

Golder Associates