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Peer Review Report

Feasibility Level

Pit Slope Design Criteria

Osisko Canadian Malartic Project

November 2008

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Osisko Exploration
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November 24, 2008

Attn: Paul Johnson, Ing.
Manager, Mining

Re — Peer Review – Canadian Malartic Project
Open Pit Slope Design and Stability

Dear Mr. Johnson.

1.0 Introduction

Further to your request I have reviewed reports re the open pit slopes, polishing pond and plan of tailings deposition, visited the site near Malartic, Quebec and met with you and Golder Associates staff in Montreal on October 25, 2008. Following is a summary of my comments. This report is intended to further the program towards licensing the project.

I have reviewed the following reports and communications:

A. Feasibility Level – Pit Slope Design Criteria, Osisko Canadian Malartic Project – August 2008;
B. Etude de Conception – Nouveau Bassin Polissage, Project Canadian Malartic, Malartic, Quebec, August 21, 2008;
C. Plan de Deposition, Parc à Résidus de la Canadian Malartic, Revision 1, August 27, 2008;

A review of the following reports will be submitted within 10 days.

D. Evaluation du Débit d'Exhaure et des Impacts Potentiels sur les Niveaux des Eaux Souterraines, Osisko Exploration Malartic, Quebec, Canada, July 2008;
E. Annual Water Balance Analysis for Osisko Malartic Mine, Revision 1, July 21, 2008;
F. Feasibility Study Report, Canadian Malartic Project September 2008;

The site was visited with Paul Johnson, Osisko, on October 27, 2008 and I met with Golder Associates staff and Osisko staff on October 28, 2008 in Montreal.

This report reviews the results of a pit slope evaluation for a feasibility study of pit slope design. The pit is being developed in an area of underground workings. The deposit is an Archean porphyry gold system hosted in diorite porphyry and metasedimentary rocks.

The scope of the work includes:

- Assessment of slope stability to design pit slope angles;
- Assessment of single and double bench and inter-ramp slope design;
- Assessment of mining through underground workings;
- Recommendations for a geotechnical program;
- Evaluation of the potential effect of the planned waste dump on pit wall stability;
- Recommendations for monitoring pit wall stability.

The pit proposed is about 2000m long and 750m wide. Wall height will be up to 380m. (Figure 1). Inter-ramp slope angles proposed are 46 to 55 degrees and bench face angles from 60 to 75 degrees. Catch bench width varies from 8 to 9 m. Bench height proposed is 10m with both single and double bench faces.

2.0 <u>Pit Slope Design</u>

The most important factor which usually influences pit slope stability is the structural geology – the dip and dip direction of discontinuities (bedding joints, foliation, shears and faults) single or in combination in the benches or final wall. Failure modes include planar, block, wedge, toppling, buckling and stepped planar of small to intermediate scale. In weaker, non-structurally controlled rock, circular failures may occur.

In all but the NE section, the rock in this project is classified as hard to very hard and, except for occasional single bench failure, pit wall failure is expected to be limited.

Golder performed structural geological mapping using core orientation, core mapping, borehole televiewing, mapping of underground openings and the development of stereo nets. There was very limited evidence of clay gouge in the core. Faults are generally thin.

In the NE slopes rock discontinuities are more prevalent. Flatter bench faces will be required and should be developed based on structural geological mapping of the pit slopes as the pit deepens. Preliminary design angles in this section are:

Bench face angle – 60° double bench
Bench height – 10 metres – single bench; 20 metres – double bench
Catch bench width – 8 metres
Overall slope angle – 46°

For the remainder of the pit slopes, circular failures, common in soils and weak rock are not expected due to the generally strong to very strong rock.

Pit slopes outside the NE section recommended are:

Good buffer blasting	**Effective pre-split blasting**
Bench face angle – 69°	75°
Bench height – 20m – double bench	20m – double bench
Bench width – 9 metres	8.5 metres
Overall slope – 50°	55°

I agree with proposed Golder designs. Field modification may be developed depending on the structural geological mapping and success of the blasting procedures.

Haul roads are generally preferred in hard rock. The bench face adjacent to the haul roads must be well scaled to limit rockfall.

Overburden is generally less than about 10m and primarily comprises glacio-lacustrine till with local soft sediment. The cut slope is proposed at 2H:1V with an 8-10m bench at the rock contact. Control of surface water and erosion will be required.

3.0 Blasting

The second most important factor to control bench face and overall slope angle is successful blasting. The blasting program and crews should be trained in the latest procedures of buffer and pre-split blasting design, explosives, powder factor, detonation, monitoring etc.
In the pre-split final line the hole diameter in inches should equal the hole spacing in feet. These holes should not exceed 5.5 inches diameter and 5.5 foot

spacing. Golder recommend maximum 8 holes per delay in the bench face line. Electronic blasting caps are recommended for accurate control of delays.

Because the north wall and pit area is very close to houses a maximum allowable peak particle velocity (ppv) and air pressure blast has been specified by the Quebec Mine Branch. This will require a blast quality monitoring program. Golder can design such a program.

Prior to mining, the program should include a pre-blast survey of all houses near the open pit. This should detail and photograph all cracks, differential settlement and other deficiencies agreed with each owner.

Definite times for blasting should be established and known to all. It is very disconcerting to be surprised by a blast. Also the blasting program and impact should be described to all townspeople in an information meeting. Graeme Major of Golder Associates should be at that meeting.

The blast monitoring, at least initially, should include high-speed photography of each blast. This can be reviewed and assessed with a slow-speed playback video.

The 60° bench face is best developed with an angle drill. However, loading such holes is often difficult and may require explosives placed in a tube.

Note that below the water table wet holes may be common requiring slurry or emulsion explosives.

If geologic structure is somewhat flatter than the final bench face the drill angle may require modification.

For the bench face angles specified and to limit rockfall a specific bench crest and face scaling program will be required, particularly in the NE section. One successful procedure is to modify a long-reach backhoe by replacing the bucket with a dozer ripper tooth. This was a major success for Cambior at pits at OMAI, British Guyana in allowing a steeper pit slope.

5m high confining dikes constructed of waste rock are tentatively proposed to confine the thickened tailings slope. The size of this rock should not exceed about 15cm. Selective separation to this size is difficult. One option would be to design blasts in selective areas in the waste rock in the pit to produce this maximum size.

The use of a specialist blasting engineer is recommended. One specialist I have worked with and recommend is Mr. Ron Woolf of Pacific Blasting and Demolition, Vancouver, B.C.

4.0 Groundwater

At many open pit mines, as the pit deepens, groundwater seepage will reduce stability.

As Golder have stated the rock is hard to very hard in all areas except the NE section. Where the rock is hard, groundwater pressures will not be sufficient to cause large-scale instability. There will, however, be local seepage which will cause icing and glaciation during extended periods of freezing.

In the NE section where the pit deepens below the water table, pit wall drainage may be desirable. Several piezometers should be installed to monitor the water table in this area.

The most common dewatering procedure of the slopes will be horizontal drains. They should be about 30 – 100m long depending on pit depth. Since the rock in this section is more fractured, perforated pipe will be required in the drain holes. They should be drilled with water – not air – at a plus 3 – 5° gradient. Several drains can be drilled in a fan pattern from one set-up for easier water collection.

Much of the underground mine is presently flooded. A program to dewater the underground in advance of deeper mining is required. Golder can advise on an appropriate program. Dewatering can potentially be developed through existing shafts.

Note that the climate will affect the north pit wall and south pit wall differently. The north wall will receive much more sun which will create drier conditions. The south pit wall will be more affected by freezing, frost-heave, wetter blast holes etc. by being potentially in the shade from the sun.

5.0 Engineering Characterization

Golder have developed a comprehensive program of characterizing the engineering properties. This has included developing the pit geology, geologic structure(s), material properties, rock mass properties and hydrologic conditions.

The pit geology includes geologic mapping, faulting, analysis and stability models. Material properties include unconfined compressive strength and point load tests and their correlation, elastic constants, unit weight and discontinuity shear strength and rock mass properties – (RQD, RMR and rock mass strength).

Osisko is evaluating the underground mine dewatering options.

6.0 Mining through Underground Workings

6.0 Mining through Underground Workings

The initial planning should be to develop N – S cross sections about every 100m showing all drifts and stopes. A major concern will be roof stability over the larger stopes. Larger vehicles could collapse the roof. Hence, the minimum safe roof thickness requires determination. To allow further mining, one option is to fill the stope.

At the Giant Mascot Mine in B.C. large stopes were filled by developing "drop-raises" and filling the stope with low-grade ore or waste depending on whether the stope walls were ore or waste. See attached paper describing the procedure.

Special mining sequences will also be required where the pit walls are near any drifts. Note that the blasted rock expands about 30 – 40% compared to intact rock.

Detailed planning will be required for mining in the area of existing openings. The water level must be lowered in advance.

Cemented tailings would be an option but will be very expensive.

7.0 Pit Sectors

While the total length of the pit wall is reasonably great the rock conditions can be developed into two major sectors – the poorer quality NE section and the remainder of the pit (see figure 1).

The remainder of the pit has been developed into 4 such sub-sections — See Golder Figure from page 33 — shown below. Design rock structure characterizations are shown in the Golder report pages 34 – 40.

CANADIAN MALARTIC PIT SECTORS				
Sector	**Wall Height (m)**	**Wall Orientation (Dip Direction, °)**	**Main Lithology**	**Large-Scale Structure**
Northwest	260-320	180	AGR	Sladen Fault, greywacke-porphyry contact, Northwest Conjugate Fault, syncline
Northeast	320-360	180-210	Porphyry, various alteration types	Sladen Fault, greywacke-porphyry contact
East	380	300-310	Greywacke, various alteration types	
South	340-380	330-025	AGR	Northwest Conjugate Fault
West	320	325-160	Greywacke, various alteration types	Northwest Conjugate Fault

Mapping of the geologic structure during pit development will assist in determining bench face angles which will influence the overall pit wall angle.

8.0 Stability Evaluation

Rock mass and kinematic stability analysis indicate the rock quality conditions appear favourable for the development of steep inter-ramp slopes in all sectors except the NE sector. Rock quality outside this sector is sufficiently good to preclude mass failure of overall slopes.

The NE sector has sufficient adverse local geologic structure that inter-bench slope failure control will require good blasting practice, flatter inter-bench slopes and potential slope dewatering for stability control.

It is considered that the Golder Slope Designs are realistic and can be justified based on a comprehensive rock mechanics investigation and evaluation. Table 1 summarizes the Golder design.

RECOMMENDED SLOPE DESIGNS IN BEDROCK					
Wall	**Operating Practices**	**Bench Configuration and Height (m)**	**Catch Bench Width (m)**	**Bench Face Angle (°)**	**Design Inter-Ramp Slope Angle (°)**
All sectors except Northeast sector	Buffer Blasting	Double Bench 2 x 10 m 20 m between catch benches	9	69	50
All sectors except Northeast sector	Pre-Split	Double Bench 2 x 10 m 20 m between catch benches	8.5	75	55
Northeast sector	Controlled Blasting to Break Cleanly Along Structure	Double Bench 2 x 10 m 20 m between catch benches	8	60	46

Table 1 – Feasibility Level Pit Slope Designs

In the west sector of the pit there are two concave areas where the overall slope can likely be steepened about 5° and one convex zone where the slope should be flattened about 5°. In the extreme east end the concave curvature will allow a 5° steeper slope.

The pit slope design rationale uses the average systematic discontinuity sets as a basis for evaluating structural control of bench stability.

Golder, on page 49, provide a review of general risks associated with steep pit slope designs. Pit slope behaviour should receive more attention during and following heavy precipitation and during spring thaw periods or following any earthquake.

During extended cold periods the pit walls will freeze. If they are saturated the wall becomes a barrier to seepage and the water table can rise in the slopes behind the face. The increased level of the water table decreases stability.

At the old Bethlehem Mine pit in B.C. a major pit-wall failure occurred following one week of very cold weather due to their condition.

9.0 Placement of the Waste Dump

Figure 2 shows an adjacent waste dump along the south side of the pit. The dump is proposed to be 85m high and 2300m long with 3H:1V side slopes. This slope angle can be reclaimed.

An analysis by Golder indicates the dump toe can be located 50 metres from the pit crest.
A safety factor of 2.1 was calculated.

The analysis used 2H:1V slopes. The use of the flatter slopes allows for fatigue stresses due to ongoing nearby blasting. The slope was assumed fully saturated. This is conservative.

10.0 Monitoring Geotechnical Behaviour

To provide safe and stable mining conditions, a program of geological, geotechnical, blasting and stability monitoring is essential. The Golder program is reasonable. Dedicated monitoring staff and responsibilities are desirable.

10.1 Pit Wall Structural Geological Mapping

An ongoing program of structural geological mapping and assessment of potential specific failure modes for single and multiple benches should be developed and maintained. Stereo net analysis should be used to assess potential instability.

10.2 Ongoing Geotechnical Evaluations

When variations in geologic structure, shear strength properties or groundwater conditions are encountered the potential for local instability should be reviewed. Pay particular attention to the south dipping structures in the NE sector.

If new faults or new fault orientations are encountered, stability should also be reassessed.

Wherever new mining is developed near underground drifts or stopes coordination of mine planning between mine engineering and geotechnical engineering is essential. The structural geology in the NE sector is the least favourable for stability. This area requires more detailed review and monitoring.

10.3 Blasting

The Quebec Mine Ministry has established blast criteria limitations for peak particle velocity and air blast over pressure. Canadian Malartic must establish blast monitoring stations in the town, acceptable to the Ministry. The establishment of set blasting times should be considered. The water level in accessible underground shafts should be monitored at least weekly or more frequently if rapid changes occur. Note that blast vibration extends farther in saturated than unsaturated rock.

Blasting in the mine should not be performed if very heavy cloud cover exists less than 500m above average town elevation. Blast vibration can be reflected from the cloud cover and break windows.

10.4 Groundwater

Groundwater is not expected to impact stability in most of the open pit walls except the NE sector. Several piezometers are recommended in the NE sector. If seepage or high water pressures develop which are of concern, consideration should be given to the installation of horizontal drains.

10.5 Slope Movement

A program of EDM (electronic distance monitoring or equal) should be considered for the NE sector slopes. Reflecting mirrors should be developed down the slope on lines about 200m apart and every 40m in elevation or where adverse structural geology exists initially.

One instrument station near the south-east crest should be adequate. A self-reading survey instrument located in a stationary shelter is recommended. Highland Valley Mine (Teck Corp.) British Columbia has an excellent slope monitoring program. They are amenable to external inspection.

Regular stability inspections should be made by geotechnical or geological staff and monthly reviews to management are recommended.

Sub-surface displacement monitoring should be considered if any indication of surface cracking develops, particularly in the NE segment. While several techniques are available the "slope indicator" system is efficient and well-proven.

10.6 Third Party Review

An annual third party review of geotechnical and stability conditions at Canadian Malartic is recommended.

10.7 Site Weather Station

Establishment of a site weather station is recommended if one does not already exist.

11.0 Conclusions

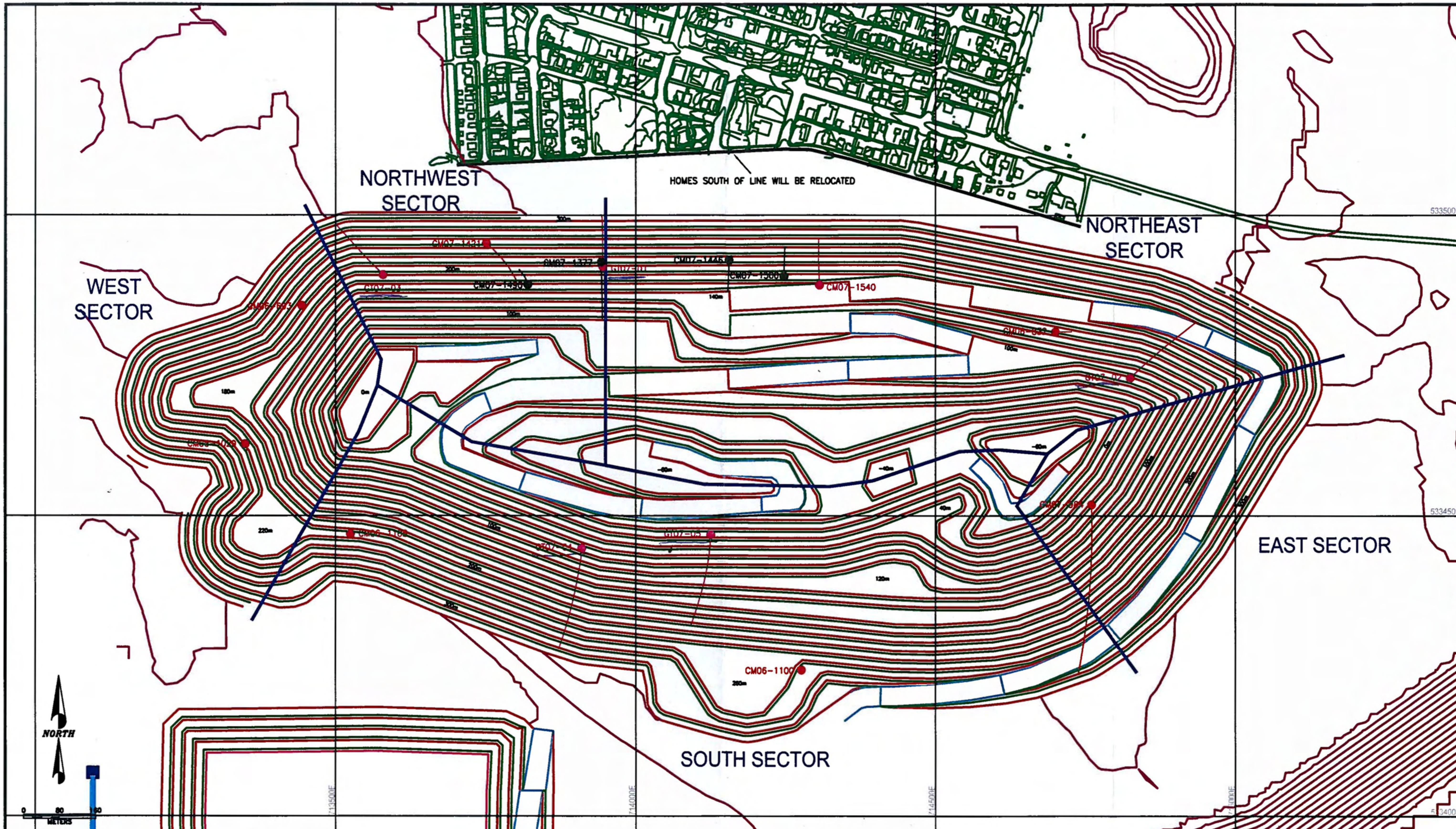
Golder Associates have provided a comprehensive site investigation and evaluation of pit slope design requirements for the new open pit mine proposed by the Osisko Canadian Malartic project. A number of specific comments and suggestions are included in this review.

The evaluation and design have dealt with site geology, rock mechanics conditions, groundwater, the impact of existing underground openings, blasting and blasting concerns, bench, berm and pit slope design and long-term monitoring of geotechnical stability.

This third party review has determined no "show-stopper" conditions and the Golder Associates report meets the standard of the industry.

C O Brawner

C. O. Brawner, P. Eng.
FCAE, FEIC, FCIM



Explanation

GT07-02 Geotechnical Corehole

CM07-1500 Golder Televiewer Corehole

CM06-832 Terratec Televiewer Drillhole

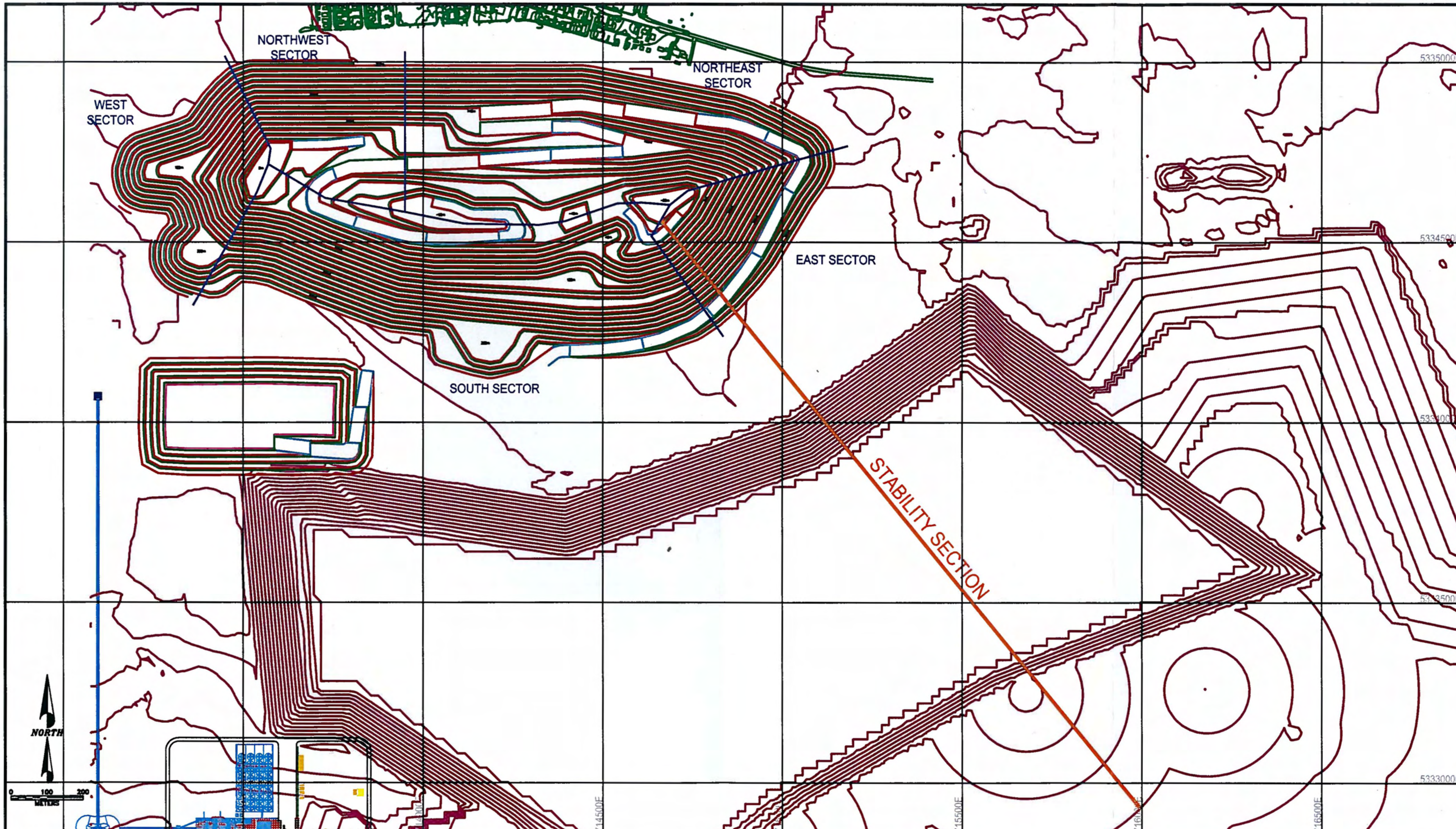
Sector Boundary

FIGURE 1

ULTIMATE PIT PLAN SHOWING DRILLHOLE AND SECTOR LOCATIONS

Golder Associates
RENO, NEVADA

PROJECT No. 07-1221-0028 DATE 06/30/08



Golder Associates
RENO, NEVADA

PROJECT No. 07-1221-0028 DATE 06/30/08

Sector Boundary

ULTIMATE PIT PLAN AND WASTE DUMP

FIGURE 2

Chapter 15

MINING THROUGH OLD UNDERGROUND STOPES AT MASCOT GOLD

by D.S. Cavers, C.O. Brawner and A. Bellamy

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Hardy BBT Limited

President
C.O. Brawner Engineering Ltd.

Mine Superintendent
Mascot Gold Mines, Nickel Plate Mine

ABSTRACT

Pit slope designs at Nickel Plate Mine have used relatively steep interim and final wall slopes of 60 degrees overall. The rock is unusually strong and tough with most discontinuity sets being steeply dipping. An integrated program of monitoring and drainage has been used. Stopes of large extent which are generally unsupported occur on the property. For the most part, no deterioration of the stability of the stopes has occurred over the last 30 or more years. Mining methods through these areas have used a combination of fills placed via VCR drop raises and crater blasting of the backs. One of the major challenges in mining through the stope areas is to maintain slope stability conditions and to reduce dilution of the higher value ore zones occurring immediately around the stope areas. The paper discusses geology, slope stability and mining methods at Nickel Plate Mine.

INTRODUCTION

The Nickel Plate Mine is a gold bearing skarn deposit located in the Southern Interior of British Columbia above the town of Hedley at an elevation of 5000 - 6000 feet. The mine is approximately 300 kilometres east of Vancouver. The history of the mine below is based in part on Millis (1983).

The Nickel Plate Mountain ore deposits were initially discovered in 1898. The Nickel Plate, Sunnyside, Bulldog, Horsefly, and Copperfield

claims were staked and by 1904 the Daly Reduction Mill had been erected and was in operation.

In 1909, the Nickel Plate Mine, under the name of Hedley Gold Mining, changed ownership from the Daly Estate to the U.S. Steel Corporation. In the same year, the Great Northern Railway linked Hedley to Oroville, Washington, and Princeton, B.C. The mine ran continuously, with one closure in 1920, until December 1930, because the then price of \$20.60/oz was too low to sustain mining. Production at that time was 200 TPD.

The Kelowna Exploration Company Limited (Kelowna) acquired the Hedley Gold Mining Co. in 1932 and developed sufficient underground reserves to commence mining again in 1935 at 325 TPD. From this period until its closure in 1955, a total of six separate underground ore zones were mined. These included the main Nickel Plate zone, four Sunnyside zones, and the Bulldog zone. The total published production for the Kelowna and Mascot mines up to this period was 1,556,749 oz (4.4×10^7 g) gold, 188,139 oz (5.3×10^6 g) silver, and 4,077,305 lbs (1.85×10^6 kg) copper.

In 1908, the Mascot Fraction Claim was staked by other interests on the west slope of Nickel Plate Mountain. A deal could not be struck with the owners of the Nickel Plate Mine and the Hedley Mascot Mine was formed. This mine went into production in 1936, producing over 250,000 oz (7×10^6 g) of gold from only one fractional claim which formed part of the main Nickel Plate ore zones.

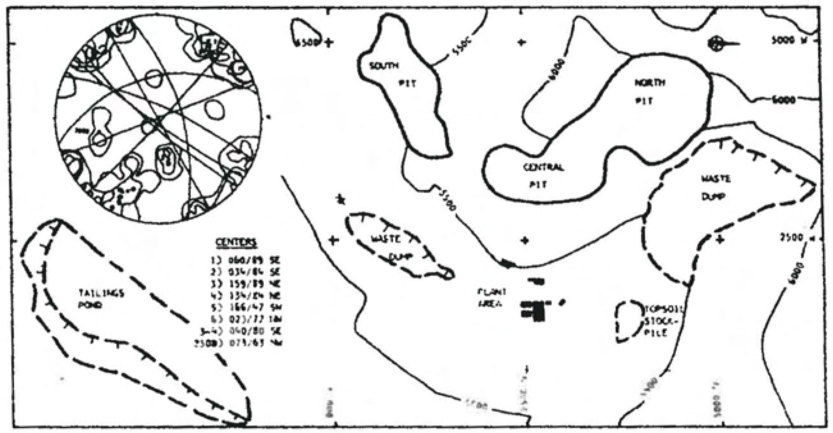


Figure 1. Schematic site plan. Stereonet is a lower hemisphere equal angle (Wulff) net. Contours are in 2% increments.

The Mascot-Nickel Plate Mine today encompasses the Kelowna Exploration and the Hedley Mascot properties. Earlier mining was predominantly from underground stopes which were generally unsupported due to the very competent ground. Figure 1 shows the overall layout of the three main open pit areas in the present mine. There are three main pit areas:

a) The Bulldog zone is the southernmost pit. The planned floor elevation of the pit is 5100 feet which will be accomplished using two pushbacks. There are no major stopes in the Bulldog area.
b) The Central Pit area comprises the North and South Central Pits with planned floor elevations of 5420 and 5440 feet, respectively. The South Central Pit, in which mining has recently been completed, was excavated through the Sunnyside 3 Stope area. The North Central Pit area will penetrate the 450 Stope area.
c) The Nickel Plate pit is the northernmost pit area and will be the major production area within a few years. The largest stopes occur in this area.

GEOLOGY - ORE RESERVES

The following discussion is based in part on Simpson and Ray (1986), McGonigle, Tremaine and Johnson (1948) and input from Nickel Plate geologists as noted in the acknowledgements. The Nickel Plate Mountain ore deposits are one of the largest metasedimentary gold bearing skarns known in the world. The Nickel Plate deposit lies within the upper Hedley Sequence where a large skarn zone developed peripherally to a stock of diorite and gabbro. The Hedley Sequence consists of thinly bedded siltstone, black argillites and minor wackes interbedded with impure limestone beds from 1 to 10 m thickness. The bedding generally dips 30 degrees to the west and is interrupted locally by small scale folds. The Nickel Plate skarns form a large west-dipping bowl centred around a stock of hornblende diorite, quartz diorite and gabbro, which in turn is part of a series of stocks termed the Hedley Intrusives.

The rock is unusually strong and tough. Measured rock strengths in endoskarn and skarn units range from 16,000 to 65,000 psi. Some weaker and more brittle limestones and intrusives occur locally.

Sills and dykes of diorite porphyry comprise about 40% of the skarned interval and commonly show skarn alteration. The remainder of the interval consists of varying amounts of iron rich pyroxene, garnet and calcite. Gold is associated with arsenopyrite and pyrrhotite and to a lesser extent chalcopyrite, pyritye and sphalerite. Gold occurs in native form as minute blebs (about 25 microns across) along the cracks and grain boundaries of the sulphides. Gold bearing zones often follow bedding and are usually localized by small scale folds and diorite porphyry sill/dyke contacts.

ROCK MECHANICS FOR OPEN PIT DESIGN

Structural geology

Prior to mine startup, an unusually large amount of structural geology data was available from mapping carried out during mining of the original stopes. Limited surface mapping was available from contemporary work. In addition, two oriented core holes were drilled and oriented using a clay orientor (Call et al, 1982).

Structural geology data from the different sources was all reasonably consistent, although small scale and local variations occurred. Figure 1 shows a typical summary stereonet for the Central Pit area. The main discontinuities which are present include joints and faults. Predominant joint sets are typically steeply dipping and, in many cases, are subparallel to the major trend of the walls. Continuity is typically greater than 10 m, although some smaller curved features occur locally. Bedding joints are less frequent but joints approximately parallel to bedding are important on some parts of the east wall of the Central Pit area where they are subparallel to the District Footwall Fault System.

Due to the prevailing discontinuity patterns, the predominant potential failure mode is toppling. In addition, wedges 1 to 2 benches high may occur in some areas on "random" discontinuities that do not fall into well defined sets. Local block ravelling may occur on some of the curved discontinuity surfaces.

It is noteworthy that the underground stopes show little evidence of failure, although they have been abandoned for at least 30 to 35 years. Except for the square set stope area, the stopes are unsupported and spans between pillars may, in some instances, be 30 to 60 m or more. Vertical openings range up to 45 m in the case of the 450N stope. The general lack of failure is due to the extremely competent rock, the generally steeply dipping discontinuities and to the shallow cover.

Blasting

In order to develop a blasting program unique to the conditions of the Nickel Plate Mine, three key factors had to be considered. These factors are grade control, wall control, and fragmentation. Each factor in itself has a major role in the successful, efficient development of the three Nickel Plate ore zones.

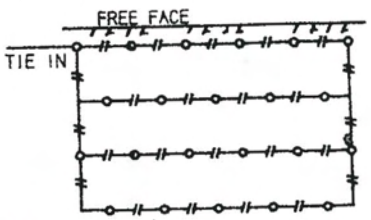
In a high waste-to-ore ratio mine as is the case for the Nickel Plate, grade control is essential to maximize ore production at minimum dilution. The maintenance of safe, stable pit walls requires controlled blasting, which is an inherent part of the Nickel Plate blasting program. Ore release is contingent upon the efficient removal of waste rock. Ultimately, good fragmentation governs the efficient loading of the broken muck.

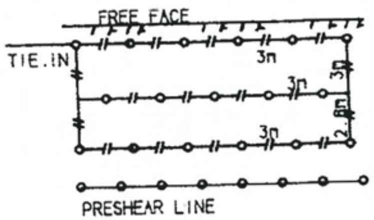
Production Blasting. The Nickel Plate skarn formations are both variable in hardness and toughness. To deal with the different rock types, different production pattern sizes have been developed based on the drillability and fragmentation characteristics of the rock. Waste pattern sizes, subgrade drilling depth and powder factors are listed below for the different rock types.

TABLE 1 - WASTE BLAST PATTERNS AT NICKEL PLATE

ROCK TYPE	BURDEN x SPACING Feet (m)	SUBGRADE Feet (m)	P.F. Lbs./S.Ton (kg/metric ton)
Gnt - Px. Skarn	12 x 14 (3.6 x 4.3)	2.5 (0.8)	0.77 (0.34)
Calc. Px. Skarn	13 x 15 (4.0 x 4.6)	3.0 (0.9)	0.69 (0.31)
Porphyry Sill	14 x 16 (4.39 x 4.9)	3.0 (0.9)	0.60 (0.27)
Limestone Skarn	15 x 17 (4.6 x 5.2)	3.5 (1.1)	0.55 (0.25)

In order to reduce subsequent pit wall damage, sequential blasting is incorporated in order to reduce the explosive charge per delay to tolerable levels. A typical waste initiation tie-in is shown in Figure 2.





Legend: 75 msec Surface Delay; 17 msec Surface Delay; O Hole with down the hole period 6 delay.

Figure 2. Tie in for production blast in waste.

Figure 3. Tie in for blast in ore.

Typical bench heights are 20 ft (6 m), but during pit development, waste areas are drilled to 40 ft (12 m) bench heights. Blast hole diameters are 6.5 in (16.5 cm).

All surface delays are double ended. Trunk lines are primacord. Down hole delays enable the surface to be ignited first to prevent cutoffs and thus misfires. Emulsion type bulk explosives are used with 12 oz primers. Sequencing can be varied depending on how it is intended to cast the muck. The powder factors have been designed to give optimal fragmentation relative to rock type. Ongoing experimentation is used to maintain blasting results at close to optimum under constantly changing conditions.

Control Blasting. Preshear blasting is generally used at the Nickel Plate Mine to form the final bench face surface. All holes are 6.5 inch diameter. The preshear holes are drilled at the design bench crest line to exact bench elevation at 40 ft (12 m) depth which represents a double bench height.

Preshearing two benches simultaneously has been found to improve stability conditions of the face due to elimination of the stepout area which would otherwise be required at 20 ft (6 m) depth (Cavers, 1987). This stepout area has been found to be a source of rock fall, to require additional scaling and can project rock fall farther out into the pit. Elimination of two of the three stepouts required over the quadruple bench interval between safety berms also potentially increases the width of the safety berm.

The final production blast in a bench sequence normally consists of three rows. The back row in front of the preshear row is the buffer row and is shot last. Sequential blasting is used. A typical blast pattern for ore is shown on Figure 3.

Groundwater

Groundwater pressures at Nickel Plate Mine are generally relatively small. Precipitation is low and, in many areas, the old stopes have provided bottom drainage for many of the near vertical discontinuities which are present.

Pneumatic piezometers are installed on an ongoing basis to provide data concerning the presence of permanent or sporadic water pressures which may develop in the discontinuities. Typically three piezometers are placed in a single 160 ft (50 m) hole drilled from a safety berm. Deeper holes are not efficient since the piezometers at depth would be too far behind the pit wall. The piezometer holes are drilled using the blast hole rig, which has been found to be fast and efficient.

To date, appreciable water pressures have only been observed in part of the South Central Pit area where there were no underlying stopes or drifts. In this case, maximum water heads of some 60 ft (20 m) were observed in local areas of the pit during the wet season.

Horizontal drains inclined upward at 5 to 10 degrees are used to drain water from the discontinuities. Drilling targets are established on the basis of geological observations and water pressure measurements. These drains occasionally tap water "pockets" but more frequently drip occasionally. In spite of the small volumes of water released, it should be noted that high water pressures can be generated on near vertical discontinuities by small volumes of water. The most important function of the drains is to reduce peak water pressures during heavy precipitation events or spring melt.

Slope Angle Recommendations

To date, the slope angles used at Nickel Plate Mine have been as follows:

- slope angle: 60 degrees overall
- interbench and intersafety berm heights: 20 ft and 80 ft (6 and 24 m).
- intersafety berm slope: 78 to 82 degrees.

Important considerations in using these angles include the following:

a) The rock is extremely competent, strong and tough.
b) Most of the discontinuities are relatively steep and would require some initiating event, such as high water pressures to precipitate failure of the overall wall.
c) More than one pushback is planned for many areas of the pit which allows operating experience to be obtained at the steep slope angles before the final slope is excavated. In areas of favourable conditions, mining at flatter angles would reduce cash flow and would not give a good indication of the conditions which would prevail for steeper walls. For example, the problem of drill stepout is not often realized for flatter walls.
d) Monitoring and drains are an integral part of the mine plan.
e) Unusually good structural data was available from the old mine plans.

INFLUENCE OF OLD STOPES ON SLOPE STABILITY

Three general configurations of old stopes are present at Nickel Plate Mine:

a) Relatively narrow stopes, such as the Sunnyside 3 stope which was approximately 60 ft (20 m) wide. Several narrower stopes also occur.
b) High, wide stopes. The 450N stope is the best example with heights and widths of approximately 120 to 150 ft (35 to 45 m).
c) Tabular stopes, such as the Nickel Plate stopes which may have widths of several hundred feet, spans between pillars of 100 to 300 ft (30 to 90 m) and variable heights. Some of the tabular stopes have additional tabular stopes underlying them.

The presence of the old stopes influences slope stability, stability of the pit floor and ore dilution. Wall conditions can be influenced by the presence of filled or unfilled stopes. Boundary element analysis of some of the larger stopes suggests that they are only stable as long as the roof arch is maintained in both directions. To date, detailed measurements of deformation above old stopes have not been made. Visual observations suggest that for the smaller stopes intersected to date, the deformation above the stope has been relatively small. In view of the potential problems with ore dilution, safety on the pit floor, and possible wall stability problems, detailed plans to deal with each significant stope intersected to date have been developed. It is stressed that much additional experience remains to be gained, particularly with mining methods through the large tabular Nickel Plate stopes.

CONSIDERATIONS FOR MINING THROUGH STOPE AREAS

Modelling Stopes

Underground plans and two directions of sections showing detailed mine development and sequencing dating back to the initial days of mining enabled the construction of plexiglass models for the Sunnyside 450N and Nickel Plate Glory Hole stopes. The models typically require only a few man days to construct.

Three dimensional CAD drafting equipment running on a VAX computer was used to develop a three dimensional (3D) model of the Sunnyside 3 stopes (see Photos 1 and 2). It was found that a large amount of memory was required and constructing a 3D surface around the perimeter of the sections was complex and only approximate, particularly in areas of overhangs or complex topography. Interpretation of the sections without the 3D surface construction proved to be difficult due to their complex shapes. The computer program used for the model was produced by Intergraph and was originally designed for modelling more regular shapes. Recent 3D modelling developments may offer increased possibilities with respect to easier modelling on less expensive computer equipment.

To date, the plexiglass model is considered the most cost effective. A major disadvantage is that it is not possible to incorporate detailed ore reserves and other data in the mine's computer data base into the model.

Dilution

Dilution of ore reserves around the old stopes forms an important constraint on the methods used for mining around the old stopes. In the underground stopes, pillars left behind contain high grade ore. Exact location of the pillars is critical to their recovery. Similarly, ore reserves in the back can be subject to high dilution for some mining methods.

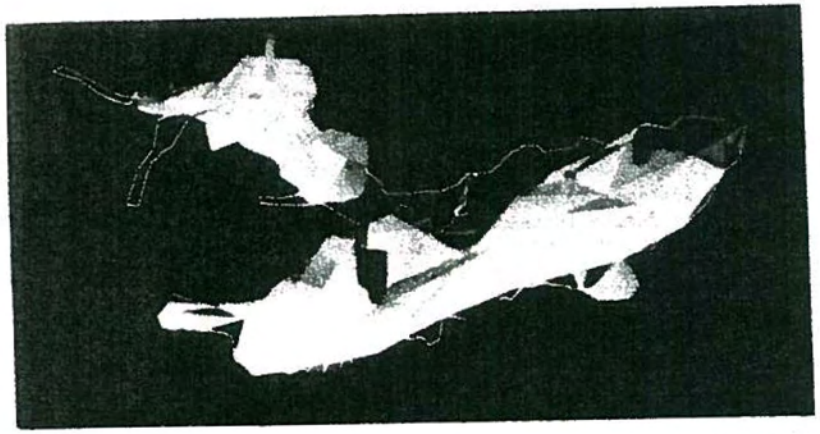


Photo 1. Sunnyside 3 computer generated stope model showing isometric view of outside of stope with back of stope in place in some areas. Existing pillars are dark vertical shapes (not trimmed to back line). Raises are shown as rectangular solids. Some cross sections are visible.

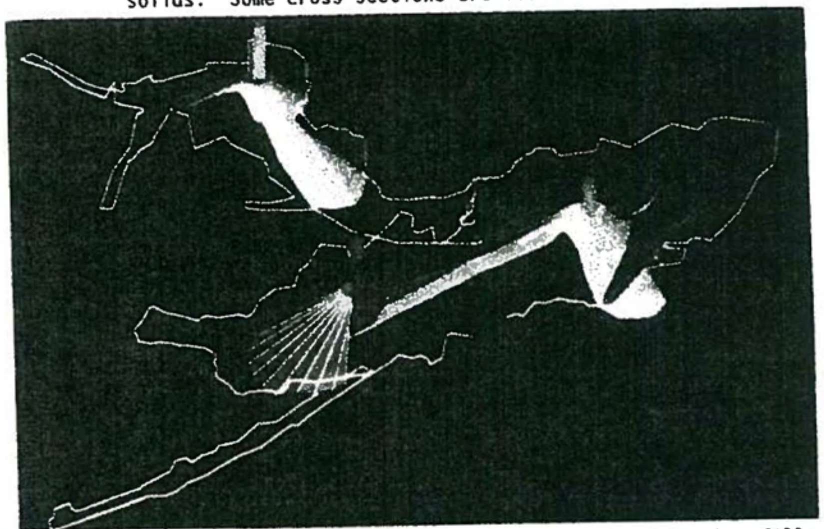


Photo 2. As Photo 1, but back of stope has been removed to show fills and existing pillars. Glory Hole fill is at right. "Cutouts" in conical fills are where fill hits back. Modelling for two subsequent stopes has used plexiglass sheets.

Safety

A major concern in working above old stopes is that the working bench floor does not suddenly drop or cave. Parameters that should be considered to decide what is a safe floor thickness before backfilling or blast caving a stope are faulting and slips, roof span, stope height and rock type.

After a stope has been backfilled, selected production blast holes are drilled deeper to search for possible voids. If small voids are found, another drop raise is not necessarily needed, however, the area may be marked off to keep heavy equipment out until mining has advanced through the stopes.

Square Set Area

As discussed above, most of the stopes at Nickel Plate Mine are unsupported. One exception is the square set area or Red Mud Stope which occupies some of the 450S stope area. This area is timbered with square sets using 12 in (30 cm) timbers on a 6 ft (1.8 m) cubic pattern. Drilling showed that some caving had occurred onto the top of the square set area, and the square set area itself appeared to be decked. It was not possible to achieve usable drop points for raises. Therefore, the area was blasted in, using heavy charges. Walls were blasted in first, followed by the back in-blasted down into the area using sequenced charges. Excavation occurred from the edges towards the centre.

Large Stopes vs Flat Lying Stopes vs Multiple Stopes

The most frequently used method to date at Nickel Plate Mine for dealing with the old stopes has been filling via drop raises. VCR blasting of the walls and back has also been used. The optimum stope for drop raise filling has a sloping floor and back and/or large vertical extent in order to allow significant spreading of fill from a single drop raise point. Flat lying stopes or stopes with low vertical extent require relatively large number of VCR drop raise points in order to achieve a reasonable percentage filling.

Multiple tabular stopes in which one comparatively flat-lying tabular stope overlies another are considered some of the most difficult to mine. Procedures for mining through these areas are still being developed at Nickel Plate Mine.

FILLING STOPES

A variety of methods were considered:

a) Filling with muck or low grade ore dropped through vertical crater retreat (VCR) raises or pushed into existing glory holes.
b) Pneumatic stowing of waste or low grade material into the stopes.

c) VCR blasting of the backs to fill the stopes with bulked material.

In addition, various slope geometries to reduce stress concentrations are presently under consideration for mining through the large tabular stopes.

Filling of the old stopes via drop raises or some method of stowing offers the best chance to reduce dilution. Operational experience shows that some dilution still occurs since it is impossible to completely fill the old stopes in all areas. In addition, it has frequently proven difficult for shovel operators and mine geologists to identify the transition between blasted high grade muck from the stope back and the lower grade ore placed as fill.

To date, pneumatic stowing has been viewed as being too expensive, particularly considering the high abrasiveness of the rock which would produce high equipment wear.

Drop Raises

Drop raises have formed one of the main methods for placing large volumes of fill into the underground stopes. The drop raises used at Nickel Plate Mine are 10 x 10 ft (3 x 3 m) and have been up to approximately 100 ft deep. The raises were blasted using VCR methods, using the drilling pattern shown on Figure 4. The holes were drilled using the blast hole rig with 6 1/2 in (16.5 cm) holes. It has been found possible to keep holes in line to depths of at least 100 ft (30 m).

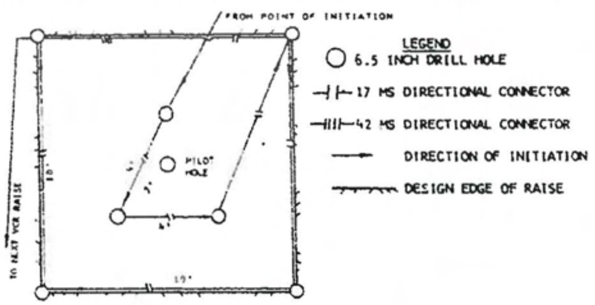


Figure 4. Drop raise blast layout.

The angle of repose of fill placed through the raises is variable. Measurements underground of fills placed using fills dozed into glory holes gave angles in the order of 38 degrees. This was also the angle for an upwardly unconfined fill placed through the drop raise points for the Sunnyside 3 opening. Where the fill approaches the back, however, the additional confinement may give an effective angle of repose of 45 degrees or more due to the fill tending to choke.

Note — Delays were placed every 10 feet detonating them sequentially from the bottom up. On completion of the blast a clean 10 foot by 10 foot shaft existed. A waste rock bumper pile was placed to stop trucks and allow rock dumping.

VCR Blasting

VCR blasting of the backs and/or walls of stopes relies on choking the opening with bulked material from the sides of the opening and back. VCR blasting has been routinely used on drifts and smaller stopes. In addition, the method is used where voids exist above fills in the larger stopes. VCR blasting suffers from a major disadvantage in that dilution is significantly higher and the distribution of the ore after blasting is much more difficult to predict.

Monitoring

Monitoring is considered to be an integral part of the overall program. Monitoring specifically related to stopes includes the following:

a) Checking of the distribution of stopes, back elevations relative to the old mine plans and fill configuration by deepening selected production blast holes.
b) Monitoring of the volume of fill placed through drop raises versus the volume which it should theoretically be possible to place. This helps to show areas in which premature choking of the fill may have occurred, raising the possibility that there are voids within the workings.

PRELIMINARY EXPERIENCE

To date, stability of pit slopes and backs of intersected underground openings has been good. Significant operational experience has been gained in blast design for production, wall control, stope filling and fill raises. Stope filling experience has been close to design. The following paragraphs briefly summarize experience to date.

During mining through the Sunnyside 3 stopes, the stability of the wall and the stope backs remained good. Photo 3 shows the north wall of the Central Pit which intersected the openings. There was no visual evidence on the pit floor that the opening existed below. There was no movement of the pit floor during mining through the filled Sunnyside 3 filled stopes. Both large and small scale structural instability have not been significant problems with the design and blasting practices used.

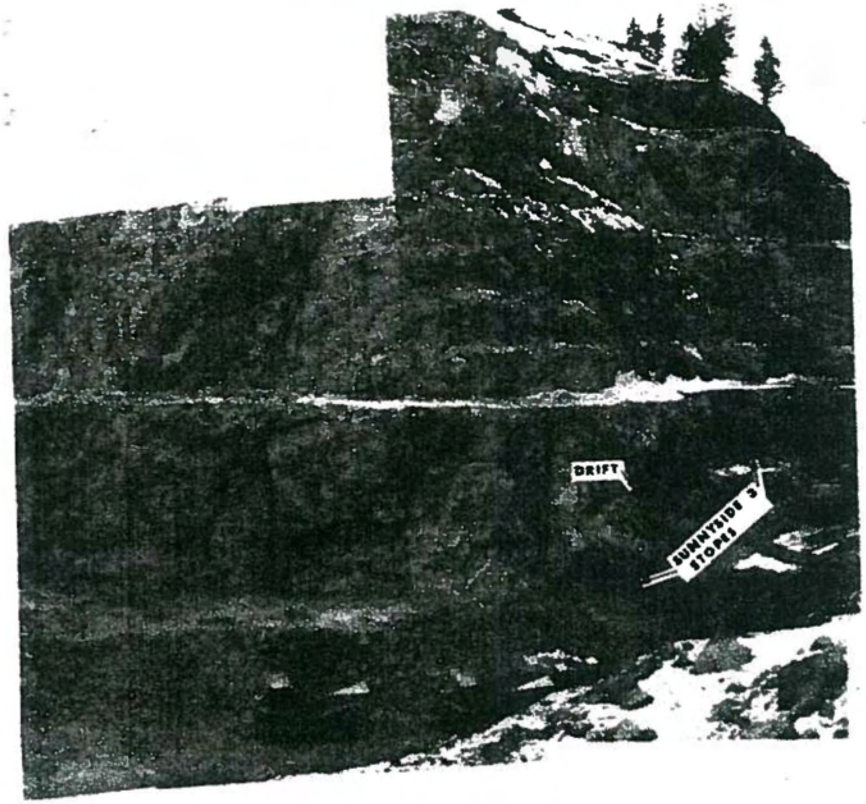


Photo 3. North side of Central Pit showing ends of Sunnyside 3 Stope (left side of Photos 1 and 2) intersecting final wall. Note edges of fills present in stope. Safety berms are at 80 ft. intervals.

Production and wall control blasts proved to be an important part in achieving safe, stable walls at the designed 60 degree slopes. VCR drop raises have been used for the Sunnyside 3 stopes and are planned for the 450N and 450S Stopes. The unreinforced raises were placed quickly with no problems over a short period of time. They stood up well during filling. VCR crater blasting of the back and walls has been used for the Sunnyside 4, Bulldog and Red Mud Stope areas. There were no significant mining problems in these areas; however, appreciable unavoidable dilution did occur.

The schematic outline of fill distribution in the Sunnyside 3 stope is shown in Figure 5. The stope locations were close to original design. The volumes of fills placed through the different drop raises were similar to those predicted. One additional raise was placed in order to place additional fill in the bottom end of the stope. The total volume of fill placed via the drop raises was 6700 cubic yards. From a production point of view, dilution occurred since it was not possible to completely fill the opening in all areas, and in many areas it was difficult to tell where the blasted ore/waste rock interface occurred.

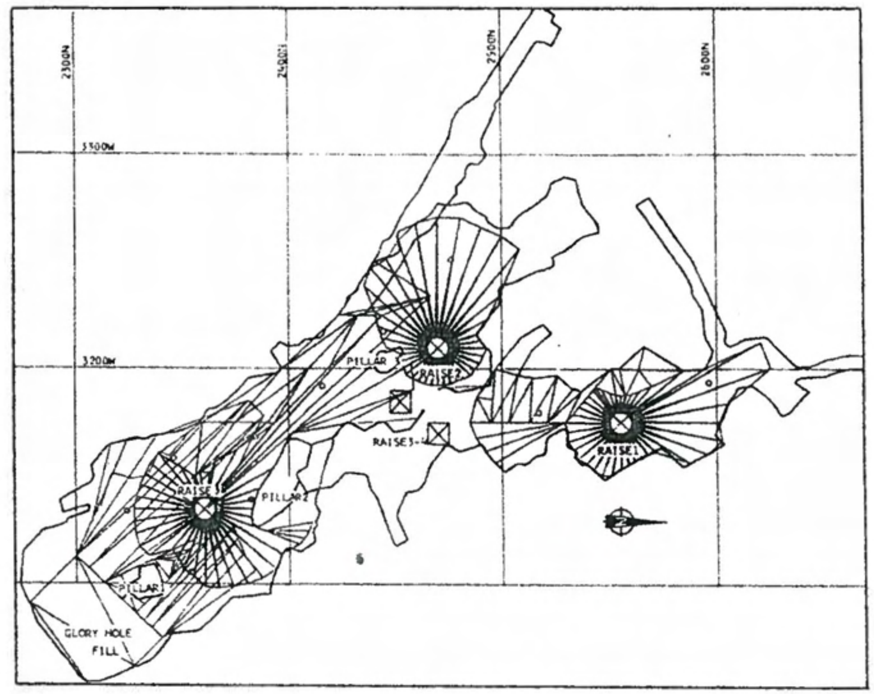


Figure 5. Plan of Sunnyside 3 Stope in South Central Pit showing distribution of raise fills. Generated from 3D computer model shown in Photos 1 and 2.

Major fills are planned via drop raises into the 450N and 450S Stopes (see Figure 6). These have not yet been placed but will have a total volume approximately ten times that of the Sunnyside 3 opening.

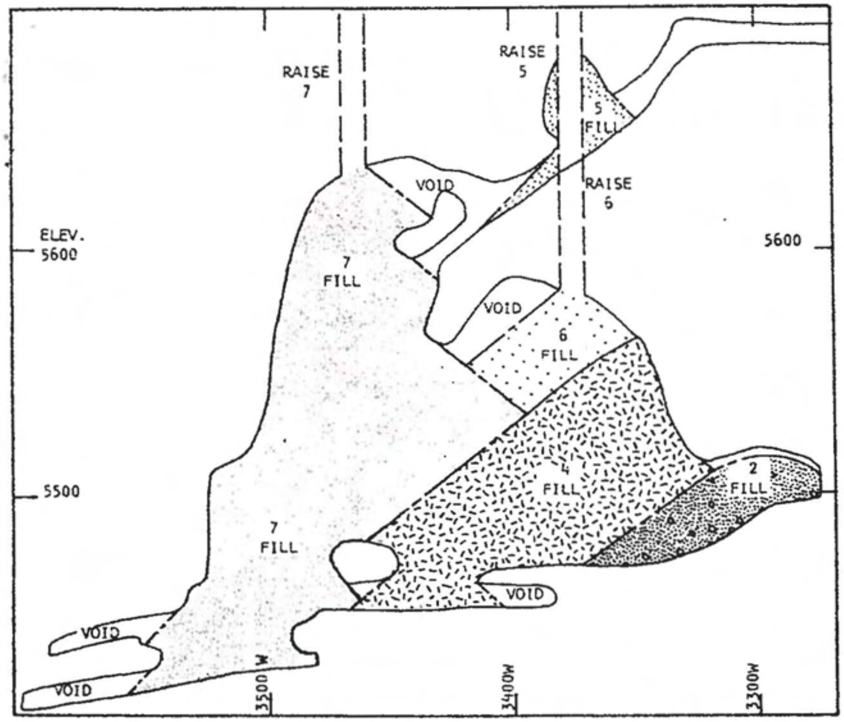


Figure 6. Composite longitudinal section showing 450N Stope fills in part of stope. Elevations and distances in feet.

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