QUANTIFICATION AND PREDICTION OF WALL SLOUGH IN OPEN STOPE MINING METHODS

By

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We accept this thesis as conforming to the required standard.

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ABSTRACT

The objective of this thesis is to present a reliable estimate of dilution that can be readily applied for evaluating open stope stability. In the past, dilution has been a difficult term to quantify due to the inaccessibility of representative data. With the introduction of the Cavity Monitoring System (CMS), it is now practical for mine operators to collect realistic, three-dimensional stope profiles for the purpose of identifying unplanned dilution in the form of wall slough.

This thesis describes a methodology for interpreting CMS data. Analysis of CMS data enables one to identify and measure unplanned dilution. It is proposed that unplanned dilution be represented in the form of Equivalent Linear Over-break Slough (ELOS) measurements. Since dilution is considered an ideal measure of stope performance, it will be used to calibrate several commonly used stope design curves. In order to facilitate calibration, a CMS database (96 obs.) has been accumulated from various Canadian underground open stoping operations. The bulk of the database has been collected from the Detour Lake Mine (DLM), Placer Dome Canada over a period of two years.

At DLM, three baseline design curves have been developed and successfully calibrated for implementation:

- Pillar Failure Curve (3-D Numerical Modelling)
- Blast Damage Criteria (Vibration Monitoring - Scaled Distance Approach)
- Open Stope Design (Modified Stability Method).

It has been found that the CMS instrument provides the most reliable information for calculating dilution and calibrating design curves. The application of these design tools will enable the mine operator to anticipate and hence reduce dilution incurred from open stope mining methods.
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1. INTRODUCTION

The health of the mining industry has been, and always will be, driven by the fluctuations of the commodity market. For mine operations to remain profitable and productive, they must undergo a process of optimization, especially during periods when metal prices are low. Specifically in underground mines, there is a growing incentive to design larger stopes. The struggle to reduce development costs and increase production has lead to a rise in popularity of bulk mining methods.

Before changing to a new mining method, it is prudent to assess its advantages and disadvantages. Some key measures of design performance are recovery and dilution. Recovery relates to the effectiveness of a mining method and is measured by the proportion of the known orebody to what is extracted. Dilution is related to mining efficiency and is defined as the proportion of waste introduced during the mining process. Compared with dilution, recovery is considered a relatively easy term to calculate. The calculation of dilution is made difficult due to the inaccessibility of reliable data. The procedure for calculating dilution can range from visual estimates to equations that combine tonnage or grade variables. Unfortunately, this results in dilution figures that are not always representative.

During the last decade, advancements made in computer and laser technology have led to the development of the Cavity Monitoring System (CMS). This instrument is capable of remotely delineating three-dimensional profiles of underground openings. Utilization of this geometric data allows for the determination of dilution values as a function of wall slough. This thesis will attempt to quantify dilution with the CMS instrument for assessing the design of open stopes and delineating factors critical to estimating stope performance. This study is a culmination of a two year study conducted at the Detour Lake Mine, Placer Dome Canada focused on optimizing stope performance.
1.1. **Thesis Objective**

The primary objective of this thesis is to quantify the zones of stability, as defined in the *Modified Stability Method*, by superimposing dilution (ELOS) values onto the Stability Number vs. Shape Factor plot. Conducting a multivariate statistical analysis will affirm the significance and relevance of the relationships developed for quantifying the zones of stability.

In addition, several mechanisms that contribute to or influence dilution have been identified by this thesis study. These factors that are considered to be critical in open stope design include blast damage, drillhole deviation and pillar stability. The influence of blasting is addressed by conducting a vibration monitoring program. Reduction of this information has led to the development of a reliable blast damage criteria that is calibrated with CMS data. Also, drillhole deviation was measured and estimated as an equivalent ELOS value. Combining the effects of blasting and drillhole deviation defines one of the zones of stability on the *Modified Stability Method*. And finally, pillar stability was shown to be related to the overall stope stability at DLM. Utilizing an analytical approach enabled the development of a pillar failure curve that was calibrated with CMS data.

1.1.1. **Methodology**

This thesis will use the following guideline as a general approach:

1. Develop a Geotechnical Database
2. Collect a CMS Database
3. Construct a Numerical Model
4. Develop a Blast Damage Criterion
5. Conduct a Drillhole Deviation Study
6. Develop a Wall Slough Prediction Tool.

The foundation of this thesis study is built on the data collected with the CMS instrument. Compilation of CMS data has led to the co-development of a new term, Equivalent Linear Overbreak Slough (ELOS), used for measuring unplanned dilution as a function of wall slough.
Complementing mine design curves with ELOS enabled calibration of the *Modified Stability Method*, blast damage criteria and pillar failure curve.

The work presented herein is the accumulation of two years of findings gained through a joint project formed between the University of British Columbia (UBC), Canada Center for Mineral and Energy Technology, Energy, Mines and Resources Canada (CANMET) and the Detour Lake Mine (DLM), Placer Dome Canada.

### 1.2. Literature Search

To date, there is a limited amount of research in the area of dilution control (Scoble and Moss, 1994). An extensive literature search was conducted to find previous dilution studies. The following is a list of the computer databases reviewed:

- **UBC ON-LINE** (Includes: SFU Library, Vancouver Public Library)
- **UMI** (Archived University Thesis)
- **COMPENDEX** (Engineering Index)
- **NTIS** (Government Reports Announcements and Index)
- **GEOREF** (Geology Bibliography and Index)
- **METADEX** (Engineering Materials Abstracts)
- **CISTI** (Canadian Institute for Scientific and Technical Information)
- **Q & L** (CANMET MRL Database).

In addition to the above information, a collection of internal documents and conversations with mine personnel have been accumulated from various mine operations. The above coupled with an extensive field oriented program resulted in design tools presented in this study.
1.3. SUMMARY

The findings of the literature review directs this study to further develop the Modified Stability Method for open stope design. The focus of this thesis is to quantify the zones of stability with a measure of dilution. Unplanned dilution is estimated by measuring the equivalent linear over-break slough (ELOS) values as defined by the Cavity Monitoring System. The construction of a CMS database will enable statistical techniques, combined with engineering judgment, to develop an empirical relationship that divides the data into ELOS groupings (0.3m, 1.0m and 2.0m).
2. REVIEW OF OPEN STOPE DESIGN METHODS

Bulk mining methods are gaining in popularity with the increasing trend towards larger open stopes (Bawden, 1995). As new drilling, blasting and ground support technologies become available, the longhole open stoping method is quickly becoming the primary extraction method for many underground operations (Lizotte, 1990). The need to develop a quantifiable open stope design method was required to replace past methods based solely on 'trial and error' and precedent practice (Bawden et al., 1989). Existing methods for open stope design and dilution prediction are based on determining the critical design parameters for opening stability and relating them to the exposed wall surface area.

2.1. OBJECTIVE

The objective is to review the current and most applicable open stope design methods used by the mining industry. It has been found that empirical techniques are the most successful because of their ability to calibrate site conditions (Barclay et al., 1989). For this reason, the research conducted at the University of British Columbia is focused on refining the existing empirical design approaches. For discussion, two empirical methods are reviewed: the Dilution Approach and Stability Method.

2.2. THE DILUTION APPROACH

The Dilution Approach was developed from a comprehensive stope stability study conducted at the Ruttan Mine, Sherritt Gordon Mines Ltd (Pakalnis, 1986). The main objective was to determine ground stability guidelines for open stope mining methods measured in terms of dilution. Findings of the study have identified three key design variables that influence dilution: rockmass quality, extraction rate, exposed surface area. These variables are combined into empirical equations for various stope configurations (EQ. 2.2-1 - isolated stopes; EQ. 2.2-2 - echelon stopes; EQ. 2.2-3 - rib stopes).
Dilution = 5.9 - 0.08RMR - 0.01ER + 0.98HR  
EQ. 2.2-1

Dilution = 8.8 - 0.12RMR - 0.018ER + 0.8HR  
EQ. 2.2-2

Dilution = 16.1 - 0.22RMR - 0.011ER + 0.9HR  
EQ. 2.2-3

where: RMR - rockmass rating (CSIR Method) [%]
ER - exposure rate (volume mined/mth/stope width) [m²/mth]
HR - hydraulic radius [m].

In all cases, dilution values are estimated from observations. These equations are considered valid for predicting dilution at the Ruttan Mine. Design charts have been developed from the database that plot dilution values on a RMR vs. HR plot (Figure 2.3.1). Its applicability to other operations requires calibration in order to determine the site specific constants.

2.3. THE STABILITY METHOD

Similar in concept to the Dilution Approach is the Stability Method (K. Mathews, 1981). Both approaches plot stability observations based on an intuitive relationship between rockmass quality and opening geometry. The Stability Method\(^1\) differs in methodology by its assignment of a Stability Number (N) in place of a simple rockmass quality value. The Stability Number is designed to accommodate for stress, structure and stope orientation by applying correction factors A, B, and C.

\(^1\) In a study commissioned by CANMET, Mathews et al (1981) developed an empirical design methodology. The foundation of this work was based on a limited database for open stopes deeper than 1000 meters.
DILUTION APPROACH DESIGN EQUATIONS

ISOLATED STOPES (61 OBS)
\[
DIL(\%) = 5.9 \times 0.08(RMR) - 0.10(ER) + 0.98(HR)
\]
\[ r = \pm 0.78, s = \pm 2.3\%
\]

ECHELON STOPES (44 OBS)
\[
DIL(\%) = 8.8 \times 0.12(RMR) - 0.18(ER) + 0.90(HR)
\]
\[ r = \pm 0.85, s = \pm 2.3\%
\]

RIB STOPES (28 OBS)
\[
DIL(\%) = 16.1 \times 0.22(RMR) - 0.31(ER) + 0.90(HR)
\]
\[ r = \pm 0.81, s = \pm 3.0\%
\]

where:

- DIL(\%) - Stop Dilution(\%), le. 10%, DIL(\%) = 10
- RMR - CSIR Rock Mass Rating(\%), le. 60%, RMR = 60
- ER - Exposure Rate as Volume removed(m^3)/slope width(m)
- HR - Hydraulic Radius(m) of exposed stope wall

DATABASE
- Slope Dilution = 10% ± 6%
- Hydraulic Radius = 11m ± 3m
- Slope Width = 15m ± 8m
- Slope Height = 36m ± 23m
- ER = 180m^3/m^2 m/h
- Excavation Rate = 2700m^3/m^2 m/h
- Slope Angle = 31m ± 13m
- Height = 56m ± 22m
- Slope Depth = 36m ± 23m below surface
- Slope Inclination = 68° ± 5°

Joining Parallel to Hanging Wall
Hanging Wall in Relaxation

Figure 2.3.1 - "Dilution Approach to Open Stope Design (Pakalnis, 1986)"

In addition, the method defines three (3) zones of hangingwall and back stability on the Stability Number vs. Shape Factor plot:
Stable - Excavation stands unsupported with occasional localized ground support to control slapping.

Unstable - Excavation will experience localized caving (tendency to form a stable arch). Cable bolts and modification of the extraction sequence are suggested methods to control caving.

Caved - Excavation will not stabilize until the void is full.

2.3.1. Calculating the Stability Number

By definition, the Stability Number (N) is:

\[ N = (Q')(A)(B)(C) \]  

EQ. 2.3.1-1

where: 
Q' = rockmass classification
A = rock stress factor
B = rock defect orientation factor
C = design surface orientation factor.

Rockmass Classification: In this method, the rockmass classification scheme utilized is the Norwegian Geomechanics Institute (Barton et al, 1974) method (EQ. 2.3.1-2). This rockmass classification method was originally developed for civil tunneling projects and is otherwise known as the Rock Tunneling Quality Index (Q).

\[ Q = \left( \frac{RQD}{J_n} \right) \left( \frac{J_r}{J_o} \right) \left( \frac{J_w}{SRF} \right) \]  

EQ. 2.3.1-2

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.

Refer to reference #26 for the corresponding NGI design charts.
Its applicability to the underground mining environment requires that the method be modified by setting the SRF and $J_w$ factors to equal a numerical value of 1.0 (Hoek et al, 1995) to result in Eq. 2.3.1-3.

$$Q' = \left( \frac{RQD}{J_n} \right) \left( \frac{J_f}{J_a} \right)$$

EQ. 2.3.1-3

Figure 2.3.2 - "Mathews' Stability Method"
These modifications suggest that dry groundwater conditions are assumed to be the norm for many underground mines at depth. For now, the SRF is selected to represent moderate stress levels. It has been found that the stress reduction factor is too crudely defined to predict the stability of back and wall exposures at depth (Mathews, 1981).  

In theory, this rockmass classification scheme has numerical values that range from 0.001 to 1000. However, it is more realistic to measure and observe values in between 0.1 to 100.

'A' Factor: This correction factor incorporates the effects of induced stress on the exposed open stope surface. It is determined as a function of the ratio of the rockmass intact strength \((\sigma_c)\) to the induced compressive stress \((\sigma_i)\). Refer to Figure 2.3.3 for the established relationship.

'B' Factor: The 'B' factor represents the correction factor for rock defects that affect open stope wall stability (Figure 2.3.3). Proper application should utilize the joint set that governs the overall stability of the exposed surface. In the case of open stopes, the critical joint set is the one that lies closest to being parallel (i.e. in strike) to the stope surface. Structures perpendicular to the surface reflect the most favorable orientation and are given the highest rating.

'C' Factor: The 'C' factor accounts for the stope surface failure modes. Intuitively, backs are more unstable in comparison to vertical walls, primarily due to gravity. Mathews et al (1981) originally intended this factor to be applied strictly to hangingwalls and backs. The logic given is based on the understanding of failure mechanisms. For simplicity, rock surfaces can fail under two conditions: gravity or friction (i.e. sliding). Barton (1974) states that backs and hangingwalls generally fail by the force of gravity, whereas footwall failures are driven by gravity only after the forces of friction are overcome.

\[ \text{Note that the stress component of the analysis is better handled by numerical modeling techniques.} \]
This correction factor incorporates the effect that design surfaces will have on stability (Figure 2.3.3).

The possibility of applying this factor to footwalls was mentioned for steeply dipping and adversely structured cases (Mathews et al, 1981). Potvin (1988) has refined this factor to include an additional curve to account for the sliding mode of failure found in footwalls (Figure 2.3.4). However, insufficient data have been collected to confirm this amendment (Hadjigeorgiou et al, 1995). For this study, the treatment of footwall data will be analyzed separately from hangingwalls and backs.

2.3.2. Calculating HR

Hydraulic radius\(^4\), as used in the mining industry, provides a simple description of an exposed wall surface (EQ. 2.3.2-1). In general, the term hydraulic radius is considered a two-dimensional shape factor. Values of hydraulic radius are associated with degrees of stability. The logic is that large values of hydraulic radii describe large wall surfaces with greater instability. Conversely, smaller values of hydraulic radii imply smaller wall surfaces with much greater stability.

\[
HR(m) = \frac{Area(m^2)}{Perimeter(m)} \quad \text{EQ. 2.3.2-1}
\]

Hydraulic radius is insensitive to wall surface orientation. Walls that can be described as long and narrow in the longitudinal direction are expected to behave differently from walls that are long and narrow in the vertical direction (i.e. effects of gravity and/or geological structure are not accounted). From experience, its applicability is limited to simple square or rectangular shaped excavations.

---

\(^4\) The term hydraulic radius was originally derived from the civil industry. It is used to express the shape and size of a conduit.
Mathews Factors A, B, and C

Figure 2.3.3 - "Correction Factors for the Stability Method (Mathews et al, 1981)"

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
Potvin Factors A, B, and C

Figure 2.3.4 - "Modified Correction Factors (Potvin, 1988)"

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
2.4. MODIFIED STABILITY METHOD

To date, the Stability Method has been found to be the only practical tool for open stope design (Bawden et al, 1989). Improving the methodology has been the focus of much of the ongoing research at the University of British Columbia. Beginning with the work accomplished by Potvin (1988), the Stability Method has evolved to become the Modified Stability Method (Figure 2.4.1).

Some of the improvements included:

- Improved Reliability (Expanded Database - 175 case histories)
- Refinement of the Potentially Unstable Zone
- Modification to the ‘A’, ‘B’ and ‘C’ Correction Factors
- Inclusion of a Cable Supported Zone.

![MODIFIED STABILITY GRAPH](image)

Figure 2.4.1 - "Modified Stability Method with Cable Support Zone (Potvin, 1988)"
Nickson (1992) provided further refinements to the 'support required' zone using statistical techniques. The revised Modified Stability Method incorporates a more comprehensive approach used primarily for designing back support (Figure 2.4.2). The application of these new support guidelines is extended to the design of hangingwall cable support. This approach is known as the "Point Anchor Approach". With only a few case histories superimposed onto Figure 2.4.2, caution is advised when using the "Point Anchor Approach".

More recently, the zones of stability on the N' vs. HR plot have been further subdivided to include: Blast Damage, Minor Slough, Moderate Slough, Severe Slough, and Wall Collapse (Figure 2.4.3). Note that these zones of stability are largely based on their personal interpretation rather than quantified numbers.
Figure 2.4.2 - "Modified Stability Method with Hangingwall Cable Support (Nickson, 1992)"
In addition, Scoble and Moss (1994) suggest that there may be other critical design parameters to design - namely blast damage and drill deviation. Two other correction factors ('D' and 'E') are introduced into the Modified Stability Method to account for the effects of drilling and blasting. Further research is required to assess the adequacy of this proposed methodology.
2.5. OPEN STOPE DESIGN SUMMARY

In all the improvements made to the *Modified Stability Method*, much work is still required for defining stope stability in a quantitative manner. Previous attempts have utilized observational estimates of dilution to define stability. However, the unreliability of the dilution database has made its application questionable.

It is suggested that stability can be measured in terms of unplanned dilution obtained in the form of wall slough. Superimposing measures of wall slough onto the *Modified Stability Method* will provide a relationship for predicting wall stability.
3. NEW DILUTION ANALYSIS

The focus of the research presented in this thesis will be to further develop the Modified Stability Method. The original intent of the Stability Method was to develop a design tool capable of predicting the degree of wall stability for non-entry type stopes (Mathews et al, 1981). Mathew's study postulates that zones of stability can be defined in a descriptive manner. There is a perceived need to refine the zones of stability quantitatively.

3.1. OBJECTIVE

The objective of this thesis study is to complement the Modified Stability Method with some measure of stope performance; namely dilution values. Additional studies have been conducted to assess the degree that blasting, drillhole deviation and stress contribute to dilution.

3.2. DILUTION DEFINED

The importance of obtaining dilution values cannot be overstated since it provides a measure of mining efficiency. In general, dilution is defined as the reduction of grade (Planeta et al, 1990). Various studies have identified that dilution can occur at different stages of mining (ie: exploration, mining, processing). For simplicity, this study will consider only the dilution occurring at the mining stage - open stope method. Dilution can be thought to be comprised of two parts: planned and unplanned (Scoble and Moss, 1994).

Planned Dilution: Planned dilution (Figure 3.2.1) refers to the waste material that is inherent to the mining method selected. In fact, the degree of dilution is directly related to the selectivity of a mining method. Further to this, dilution is influenced by the complexity of the orebody. These factors, in general, are accepted as unchangeable consequences of mining.
Unplanned Dilution: In Figure 3.2.1, unplanned dilution is shown as the waste material that is obtained independently outside of the mine design. This waste material is produced by the mining activity and can be controlled. Dilution control is gained by varying design parameters, mine scheduling or combinations of both. For instance, blast induced overbreak, sloughing (due to rockmass, mine induced stress, time) and backfill are examples of stages in the mining process that offer potential for improvement.

Of the two components of total dilution, the focus of this study will be on determining unplanned dilution. In a survey conducted by Pakalnis (1986), twenty-two (22) Canadian mines were asked to define unplanned dilution. The findings of this survey recommend that the standard measure of unplanned dilution be represented by \( \text{EQ. 3.2-1} \) (Pakalnis et al, 1995).

\[
\text{Unplanned Dilution (\%)} = \frac{\text{Waste}}{\text{Ore}} \quad \text{EQ. 3.2-1}
\]

As per industry standard, there are no units for dilution since it is commonly presented as a percentage.
3.3. CMS INSTRUMENT

The foundation of this dilution study is built on Cavity Monitoring System (CMS) data. The CMS is an application of reflectorless laser technology used for delineating the profile of an underground excavation. During the past few years, the CMS has evolved and found early acceptance to the mining industry due to its potential to provide reliable dilution measurements. Historically, the determination of dilution was considered one of the major challenges for any
mining operation (Moss, 1994). Computing dilution is difficult due to the inaccessibility of reliable data. This thesis describes the application of the CMS instrument for determining dilution values. A short discussion is provided on the operational problems and physical limitations of the system. However, with recent advances made in the instrumentation field, the development of a new methodology for measuring dilution is finally possible.

The instrument was developed jointly by Noranda Technology Centre and Optech Systems (Miller et al, 1992) to delineate the size and shape of underground openings. This information is converted into a three-dimensional mesh image for analysis in AutoCAD.

**CMS Components:** The instrument is comprised of three primary components (Figure 3.3.1):

1. Laser Scanning Unit
2. Portable Controller and CPU

![Figure 3.3.2 - “CMS Components”](image)

The main component is the laser scanning unit (LSU) which utilizes a two beam laser system for distance measurements (Figure 3.3.2). It is housed in a motorized fork assembly that is capable of rotating the LSU a full 360° about the boom axis with the ability to incline up to...
135° about the pivot point. With these physical limitations, the CMS can provide approximately 80% coverage of a theoretical sphere surrounding the LSU.

The second component is the portable controller and CPU box (Figure 3.3.2). With the controller, the operator is able to remotely activate the LSU and prepare the data collector. The CPU box houses the data logger (2 Mb - capacity for four surveys) and 12 VDC power supply.

The third component is the support package which consists of a horizontal segmented extension boom (2-7.5 meters) and two adjustable vertical posts (2-5 m). The system is constructed to extend the CMS instrument into the open stope on the boom that is braced to the sill and back on the posts (Figure 3.3.2).
3.3.1. CMS Limitations

As with any laser-guided equipment, the CMS instrument is limited by visibility problems. In optimal conditions, the LSU is capable of measuring distances up to 250 m. This is, however, rare in an underground environment. Notably, problems of ‘line of sight’ are unique to each open stope surveyed (Mah et al, 1995). Stope cavities are characteristically irregular partly due to the method of excavating in rock (drilling and blasting) and the geometry of the ore body.

The effect is to produce bends and corners that create blind spots or ‘shadows’ in the survey (Figure 3.3.2). In order to minimize the effects of ‘shadows’, it is recommended that CMS surveys be taken from different vantage points on a frequent basis to provide a stope history.
Also, the effect of 'shadows' can be limited by adjusting the elevation step which varies the survey density. For example, a two degree elevation step implies that a distance reading is taken every two degree increment as the CMS laser is rotated (Figure 3.3.3).
Excerpts taken from the CMS manual highlight the sensitivity of the laser unit to the quality of underground atmospheric conditions. Suspended rock dust particles, gaseous by-products (blasting smoke) and water droplets in the form of fog can radically deflect the signals sent from the reflectorless EDM (electronic distance measurement) component. This results in a loss of continuity in the 3-dimensional mesh.
Also related to laser sensitivity is the influence that in situ wall surface conditions can have on computing distance readings\(^5\). Wet wall surfaces and or pools of water found on the floors of sills act as mirrors to reflect the laser signal to infinity. These distance measurements will appear as spikes in the 3-dimensional mesh.

From experience, the effective range of the CMS is a function of stope geometry and underground operating conditions. The combination of these physical limitations and human error produce CMS surveys that are not located correctly. Difficulty in locating the CMS mesh correctly is due to the inherent limitations of surveying equipment. Locating the CMS instrument in space is accomplished by using a contact EDM system and prism combination (i.e. typical diamond drillhole pick-up routine for calculating the azimuth and dip). As suggested by the manufacturer, two points on the CMS boom must be identified for orientating the CMS mesh. Errors associated with locating these two prism points are related to laser beam divergence (0.7 mrad). Depending on the distance and vantage point of the set-up, the contact EDM instrument may not be able to differentiate the two prism points.

Therefore, techniques of correcting the CMS mesh for location were developed.

- Slice the CMS mesh on a horizontal elevation that will provide the tope of a known sill contour for a 'best-fit' match.
- Identify the locations of known points on the mesh (i.e. sub-level locations, brows) and adjust accordingly.
- Manually collect reference points (i.e. muckpiles, isolated rib pillars) using the 'beach' command.
- Locate the two prism points between the first boom segment (on top of the LSU, end of first boom segment)\(^6\).

\(^5\) An average distance value is computed by taking multiple readings at a high frequency (~2 kHz). The non-contact concept involves sending and receiving multiple signals since the surface does not have retro-direct capabilities. The CMS unit is designed with a signal reflectivity of 20%.

\(^6\) This technique is recommended to minimize the effect of boom sag. It is believed that the in-house software provided does not adequately correct for the total boom sag effect. Only the material properties of the boom are considered which exclude the results of wear and tear on the boom connections.

- Set up the CMS boom on a sharp angle to provide the best vantage point for the surveyors to pick up the two points.

Applying these techniques will help to minimize the errors encountered when collecting CMS information.

3.4. PROPOSED IMPROVEMENTS TO THE DESIGN METHOD

The following changes are proposed for the Modified Stability Method.

3.4.1. Introduction of ELOS

The information gathered by the CMS instrument provides three-dimensional mesh profiles of underground openings. Initially, analysis can begin by comparing the actual cavity profile with the corresponding design dimensions. This simple test will give an indication of the in situ wall conditions. Areas of wall slough or under-break are easily identified for assessing the effects of blast damage or drill deviation.

However, the majority of the observations show that wall slough is a highly localized occurrence for 'good' rock (RMR = 70-80%). Isolated or small sections of the stope wall are vulnerable to degradation or failure due to influences of geological structure and/or rockmass quality.

It is suggested that a more comprehensive analysis be used to relate the total wall slough obtained from mining with the exposed wall surface. For this study, the total wall slough is converted into an equivalent linear overbreak/slough (ELOS) measurement averaged over the stope wall. By definition, ELOS\(^7\) is:

\(^7\) The original derivation of ELOS is from the preliminary M.A.Sc. research work conducted by L. Clark at the University of British Columbia (1996).
Effectively, ELOS converts a volumetric slough measurement into an average depth of slough redistributed over the entire exposed stope surface (Figure 3.4.1).

The attractiveness of using the ELOS term is that its meaning from the point of view of dilution is more readily apparent than a volumetric measurement.

\[
\text{Dilution} \, (\%) = \frac{\text{ELOS}}{\text{Width of Ore}} \quad \text{EQ. 3.4.1-2}
\]

Defining dilution in this manner (based on \textbf{EQ. 3.2-1}) incorporates the variability of ore widths thus making the method more universal and applicable to other mining operations (i.e. not limited to narrow vein open stoping mining methods).

In order to determine a representative ELOS value, a systematic approach is required for reducing CMS surveys of various underground openings. The advantage of utilizing a standardized procedure is to limit personal biases from interpretations of the CMS data.

Once the survey is corrected for its spatial location as referenced to existing mine workings, the following procedure is recommended for calculating ELOS:

1. Cut sections through the CMS survey along the stope rings
2. Calculate the area of slough from each section
3. Calculate a volume of slough by applying a thickness factor (i.e. longhole ring spacing)
4. Compute an ELOS value accordingly for each CMS survey.
Figure 3.4.1 - "ELOS Schematic (Clark, 1996)"

Step 1: Use a slicing routine to duplicate the cross sections generated for longhole drill layouts.

Step 2: Slough is defined exclusively in this study to be comprised of any unplanned material realized outside of the blasthole ring sections. Conversely, any material that is measured to be within the blasthole ring sections is considered as lost ore. Therefore, it is important to this analysis that both slough and lost ore be measured and reported separately. With the information provided by the CMS, it will be possible to delineate slough/lost ore from different wall surfaces (hangingwall, footwall and back).
Step 3: Apply a thickness factor (extrapolate one half burden spacing on both sides of the ring) to each blasthole section for calculating a volumetric equivalent slough. This value is selected to reflect the designed 'burden' as defined by the current blasting practice.

Step 4: Selectively, filter any slough values that are questionable and use only the values that are measured with confidence.

Once all slough measurements are confirmed, cumulatively sum these values over the entire survey and compute the ELOS value.

3.4.2. Introduction of Radius Factor

The geometry of an underground excavation is one of the key factors influencing stability and deformation (Milne et al, 1996). Traditionally, the term of hydraulic radius has been used to describe the geometry of an exposed wall surface.

\[
\text{Hydraulic Radius} = \frac{\text{Area}}{\text{Perimeter}} \quad \text{EQ. 3.4.2-1}
\]

The computation of the hydraulic radius can be expressed as a function of the distance to adjacent abutments for simple shaped excavations. However, in reality, the size and shape of open stopes are often irregular (isolated rib pillar, inverse raise and ore plunge) and cannot be accounted for by hydraulic radius.

Through a literature search, it was discovered that a new shape factor has been developed capable of handling more irregular stope geometries. It is referred to as the radius factor (RF)\(^8\).

\ -----------
\(^8\) A more rigorous discussion on RF can be found in Reference #42.
Equation 3.4.2-2 defines the 'effective radius factor' (ERF) which is the summation of the distances ($r$) from any point on the exposed surface to adjacent abutments for an angular increment ($\Theta$). It is proposed that the maximum effective radius factor value computed for any given surface be called the radius factor. Intuitively, the radius factor would be located in the geometric center of a planar surface, which is where instability is most likely to occur. Calculating RF in this manner assumes that hangingwalls will deform in a predictable manner excluding the effects of anomalous structure.

Unique to RF is its ability to handle complex stope surface geometries and the influence of items such as isolated rib pillars and raises. For this reason, it was selected to replace the traditional hydraulic radius term for application in the *Modified Stability Method*.

From experience, the hydraulic radius term tends to favor long and narrow shaped stopes over equidistant ones (also supported by Bawden, 1995). A comparison between hydraulic radius and radius factor demonstrates a difference in value as the longer dimension increases.
Figure 3.4.2 demonstrates that the radius factor tends to be larger in magnitude when compared to hydraulic radius for pseudo-square shaped openings. Conversely, radius factors tend to be smaller for long and narrow shaped excavations. This observation suggests that the radius factor takes into account the stabilizing effect of the abutments. Intuitively, it can be argued that for a square and rectangular shaped excavation with the same hydraulic radius, the square shaped excavation is considered more unstable as a result of the relatively large abutment distances. The radius factor appears to reflect this inherent instability by computing a larger value for the square shaped excavation (+10%) and a lower value for the rectangular shaped excavation (-18%).

Based on the above reasons, the use of the radius factor in place of the hydraulic radius is thought to be an improvement.
3.5. **Dilution Analysis Summary**

This study proposes to revise two aspects of the current *Modified Stability Method* by introducing two new terms; ELOS and RF. As defined, ELOS is a measure of unplanned wall slough. It is hypothesized that the degree of stope stability can be related to ELOS values.

Also, a new shape factor, RF, is selected in favor of the hydraulic radius for its ability to handle complex stope geometries.

In brief, the objective of this thesis can be restated:

"To develop a design tool capable of quantifying and eventually predicting stope stability in terms of total unplanned dilution (ELOS values)".

It is believed that the effects of drilling, blasting, stope geometry and stress are accounted for by using total unplanned dilution values. Through additional studies, this thesis will attempt to assess the degree that each factor contributes to the overall stope dilution.
4. DETOUR LAKE MINE

The bulk of the data collected for this study are derived from the 660-885 Captive stope. At Detour Lake Mine (DLM), the philosophy used for designing an optimal stope configuration is to maximize extraction while minimizing dilution. Consideration is given to two factors identified to affect the effectiveness of mining the trial stope: narrow orebody and captive mining method. At DLM, the term 'narrow' is used to describe orebody widths (2 - 10 m) measured relative to the mechanized cut and fill stope widths (10 - 20 m). Figure 4.1.1 demonstrates the sensitivity of narrow orebodies to wall slough (Hutchinson and Diederichs, 1995).

Figure 4.1.1 - "Sensitivity of Dilution to Ore Width"
The term 'captive' refers to a mining method that has limited access for equipment. In this case, access to the 660-885 Captive stope is from an alimak raise developed within the hangingwall. The primary advantage of a captive mining method is that it minimizes the amount of waste development required to bring the stope into production. Conversely, the disadvantage of a captive method is that it limits the size of equipment and flexibility for mining the stope.

It is recognized that the potential for dilution problems exists for the 660-885 Captive stope.

4.1. **OBJECTIVE**

As mentioned in Chapter 1, the steps required to construct an empirical database have been outlined. The objective of this section is to provide a description of the mine and some background information for the study.

4.2. **PROJECT BACKGROUND**

Detour Lake Mine (DLM) is 100% owned and operated by Placer Dome Canada. This remote gold operation is located 200 km north-east of Timmins, Ontario and is accessed by either land (all weather road) or air (~1.5 km air strip). The deposit was first discovered in 1974 by Amoco Canada Petroleum Company Ltd. using airborne geophysics. Further exploration conducted by a joint venture formed between Amoco and Campbell Red Lake Mines Ltd. led to the decision to start an open pit phase in 1981. By 1983, full production was achieved which lasted four years. At a final pit depth of 120 meters below surface, the operation switched to an underground mining method in 1987. Underground access is gained from a five compartment shaft sunk to 612 meters below surface and an adit (leading to a switch-back ramp system).

Today, the mine is operated at 3 750 tonnes per day mill capacity sourced from various development and stope headings. Currently, the underground stopes are mined by:
• Mechanized Cut and Fill
• Longhole (Conventional and Captive Methods) and
• Shrinkage.

Over a period of two years, data was obtained from the active longhole stopes (Figure 4.3.1):

1. 660-885 Captive Longhole
2. Sub-Level Retreat
3. 935 Q120 Longhole.

1 At the time of writing, the final 460 sill pillar recovery is to be completed during the first quarter of 1997.
4.3. **Mine Site Geology**

On a regional scale, DLM is located on the northwest rim of the Abitibi greenstone belt in the Superior Province of the Canadian Shield. This belt comprises a series of Archean felsic, mafic and ultramafic tuffs, flows and intrusions, as well as volcaniclastic and chemical sediments. The primary gold mineralization is hosted within a mafic volcanic complex close to the contact between iron rich and magnesium rich basalt. The structure that marks this contact is defined by a 'cherty' felsic rock unit. In addition, gold also occurs in both enclosing rock units and within narrow fracture systems splaying off from the contact. Identified within the mine property are six (6) gold bearing rocks:

1. Main Zone (MZ)
2. Quartz Zone (QZ)
3. Talc Zone (TZ)
4. Pillow Zone
5. Calcite Zone
6. QK Zone²

In general, the deposit lies on the north limb of an east-west striking anticline that plunges down from surface to the west and progressively flattens down to the 760 mL (Figure 4.3.1).

At this depth, the ore structure has leveled off and is heading west as a narrower, steeper dipping lode. The lithologies strike on azimuth 070° - 080° and dip 60° - 80° North (Figure 4.3.2). Shaft access is primarily on the hangingwall side of the deposit.

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² Current 1997 exploration target located approximately 1.5 km west of the present mine workings between the 500 and 860 mL.
Figure 4.3.1 - "Generalized Detour Lake Mine Longitudinal Section"
Figure 4.3.2 - "Typical DLM Cross-Section"
5. TRIAL STOPE (660-885 CAPTIVE LONGHOLE STOPE)

The trial stope is located between Levels 11 and 13. Hangingwall access is via an Alimak service raise (2.4 m x 4.3 m, dipping 70°) driven at mid-stope length and entirely within waste. The upper and lower limits of the stope occur between 545 mL and 660 mL (measured below surface). Between these levels, the stope is further sub-divided into six sub-levels (545, 565, 585, 610, 630 and 650) with a vertical interval spacing of 17-20 m. The combined vertical height is 140 m which extends along strike approximately 200 m.

Two ore zones, the MZ (25%) and QZ (75%), are found to parallel each other and hence will be mined together. Sub-levels are developed along the chert marker horizon along the hangingwall contact. Sub-parallel to the stope surface is a pervasive fault zone that occurs within the immediate footwall. Definition drilling indicates that the footwall fault approaches the open stope surface with depth. In fact, two areas on 660 mL have intercepted the footwall fault. Instability in the area has resulted in a larger rib pillar being left behind.

5.1. MINING METHOD AND DESIGN CONSTRAINTS

Longhole open stoping (also known as blasthole or sub-level stoping) is a moderately high production bulk mining method. This method is amenable to steeply dipping ore bodies (> 50°) since ore movement after blasting is by gravity flow. To facilitate the higher production rates, the ore body must be large and fairly regular in shape, thus enabling larger sub-level intervals (18-120 m) to be achieved (Haycocks et al, 1996).

Diamond drilling (BQTK on a 80 m x 40 m grid pattern) indicates that the orebody strikes E-W and dips 70° N on average with ore widths ranging between 6 - 10 m. Final ore width interpretation is confirmed with Bazooka drilling of 5 m holes on 5 m centers.
Design of the trial stope was focused on achieving high productivity. This was accomplished by dimensioning the hangingwall and footwall spans near the outer limits of the 'cable support required' zone as defined by the Modified Stability Method. Rib pillars were sized for a dual purpose: to maximize recovery and minimize span. Current empirical pillar design methods associate pillar strength to a width and height ratio plotted against induced stress values. In order to maximize recovery, the width to height ratio was selected to produce a 'yielding' pillar design¹. Past experience gained at DLM shows that a 'yielded' pillar will tend to fail gradually in relaxation in contrast to a rock-bursting mode of failure. Also, a reduction of the hangingwall and footwall spans was accomplished by using the rib pillars to divide the stope into panels approximately 45 m in strike length (Figure 5.2.1). The radius factor selected for design is 16.0.

5.2. DEVELOPMENT AND PRODUCTION CHARACTERISTICS

The reserves are estimated at 508 000 tonnes with an average grade of 5.3 grams/tonne (Moran et al, 1992). It is expected that this stope would supply one third of the total mine production for a period of three (3) years at a rate of 400 tpd. The recovery for this stope is estimated at 84%.

The sills were advanced as 3.6 mH x 5.0 mW drifts (Figure 5.2.2). The procedure of slashing the sub-levels to full ore width was not practiced due to time constraints. Ideally, slashing to full ore width minimizes the amount of production drilling that is required. With the expanded drift, the longhole drill is able to follow along the ore contact on either wall. Therefore, in areas where the ore outline has proved to be wider, a fan hole pattern is utilized and drilled to contact. It was found that in eliminating the slashing procedure, a substantial time and cost savings was gained despite the increase in production drilling.

¹ The term 'yielding' refers to the condition where stress is predicted to exceed the strength of the pillar (i.e. plots within the failed zone).
The designed standard hole burden for a vertical downhole is 1.52 m with a maximum spacing of 1.4 m. The upholes are dumped at 20° (from the vertical in the direction of retreat) to ensure a more stable brow with an effective burden of 1.43 m also spaced 1.4 m apart.

Mine sequencing is in a retreating fashion towards the access cross-cuts. The general blasting practice primarily uses a bulk blasting agent (ANFO) and emulsion primer combination. Both downholes and upholes are loaded pneumatically without tamping of the explosives. All holes
are double primed, one at mid and full borehole depth, with millisecond non-electric detonator caps. The blasts are tied in with a light (25 grain) detonating cord. Detonation is initiated by lighting a (~10 minute) tape fuse.

Figure 5.2.2 - "660-885 Captive Stope Typical Cross-Section"
Production aims to mine the stope one panel at a time (recovers the panels in the order of ‘A’, ‘B’, ‘D’ and ‘C’). Upon completion of the panel, a waste pass is driven through the sill pillar on the 525 mL. From this location, unconsolidated waste rock is then introduced as backfill. Fill fences are constructed at each sub-level located at each rib pillar to help prevent the waste material from entering the adjacent panel.

All the ore is mucked from one horizon (on 660 mL). Nine drawpoints were developed spaced 20 m apart to facilitate equal draw control of the muckpile.

5.3. CABLE BOLT SUPPORT

As indicated in Section 5.1, the trial stope requires cable bolting to maintain hangingwall stability. In total, ~14 000 m of cable bolts were installed (adjusted to exclude the sill pillar). The ‘Point-Anchor’ approach (Nickson, 1992) was utilized to design the cable lay-outs (Figure 5.3.1). Cable density included 3 bolts of 8 m lengths arranged in a fan coverage along every 3 m of strike length (Figure 5.3.2).

For each panel, a different cable configuration was installed and distributed as indicated in Figure 5.3.3. The cable geometries tested included:

- Nutcage (Panel A)
- Bulge (Panel B)
- Birdcage and Straight Combination (Panel C)
- Bulge and Straight Combination (Panel D).

Additional cable bolts were installed into each rib pillar. These bolts assisted in maintaining the overall integrity of the ‘yielding’ pillars.

---

2 A more detailed description can be found in Hutchison and Diederichs (1995).
Figure 5.3.1 - "Point Anchor Design Approach for Hangingwall Cable Bolts"
545-885 WEST

CABLE BOLT DRILLING - Blocks C&D

**H.W. HOLES**
- 3 Holes At 8.0 metres (26 feet) Long
- Flat, +20', +40'
- Drill only every 2nd ring (ODD # RINGS)
- 17 Rings of Drilling
- Total 408 metres (1339') of Drilling
- 51 Holes Total
- mark holes with white paint

**RING NUMBERS** - ODD NUMBER RINGS
- 63A TO 39A (13 RINGS)
- 31A TO 25A (4 RINGS)

**SECTION LOOKING WEST**

Figure 5.3.2 - "Typical Hangingwall Cable Bolt Configuration"
Figure 5.3.3 - "Cable Bolt Distribution by Panel"
5.4. **TRIAL STOPE SUMMARY**

The objective of this section was to describe the unique characteristics of the 660-885 Captive stope. It was found that the relatively ‘narrow’ ore widths encountered in the stope and the captive nature of the mining method to be the critical design and operating parameters. Stope performance will be gauged by the amount of wall slough that is incurred during mining. Ultimately, dilution values will be computed in accordance with the definition given in Chapter 3.

Design of the trial stope was focused on achieving high productivity. This was accomplished by dimensioning the hangingwall and footwall spans near the outer limits of the ‘cable support required’ zone as defined on the Modified Stability Method. Therefore, an extensive cable bolting program was implemented to help stabilize the hangingwall spans. Also, the stope was divided into panels by a series of vertical rib pillars which helped minimize the spans. These pillars were designed as ‘yielding’ pillars in an effort to maximize recovery.
A geotechnical program was devised for the trial stope to describe the distribution and properties of the rockmass. This information will aid in the design of safe and cost effective stope dimensions. In addition, it is hoped that the behavior of the rockmass can be predicted in the long and short term.

Among the factors that regulate the strength and deformational properties of a rockmass are the inherent planes of weakness. These structural discontinuities commonly occur as populations in a similar geologic environment. A population is defined as a group of discontinuities with comparable mechanical properties, the most important property being its orientation. It follows that a discontinuity population with the same orientation is expected to have other similar properties. Therefore, the design approach is based on structural and geological information. The general procedure for geological data collection is as follows:

1. Determination of Structural Domains
2. Determination of Design Sectors
3. Development of Rockmass Models
4. Kinematic Assessment of Failure Modes
5. Estimation of Strength Parameters.

6.1. GEOLOGICAL DATA COLLECTION

A detailed fabric analysis was carried out in the 660-885 Captive longhole stope. Systematic line mapping was conducted on all sub-levels to record structural features on a standard geotechnical mapping sheet: location, rock type, structure type, number of features having a similar orientation, joint spacing, rock and joint condition (roughness, planarity, infill characteristics), strike/dip, continuity and presence of groundwater. In total, approximately 2.4 km of line mapping yielded 596 poles for compilation on stereonets (Figure 6.1.1).
It is suggested (Stauffer, 1966) that at least 100 poles be plotted in a stereonet analysis. Using this guideline, only two sub-levels in the 660-885 Captive longhole stope have sufficient points to yield a reliable conclusion. For this study, it is assumed that even a few data points will give reasonable results given that they are accurate representations of the population. On this premise, interpolation and extrapolation techniques can be justified in areas that are deficient in data points.

**885 - All Line Mapping Data**

![Figure 6.1.1 - "Line Mapping Data Distribution"](image)

For convenience, the typical joint sets are defined with respect to the underground opening as follows:

- **Joint Set A (JSA)**: parallel (/) to the open stope configuration
- **Joint Set B (JSB)**: perpendicular (⊥) to the open stope configuration
- **Joint Set C (JSC)**: flat (< 30°)
- **Joint Set D (JSD)**: other cross cutting orientations.

Compilation of the large amount of structural data was accomplished using a computer program called DIPS version 3.1 (Hoek and Diederichs, 1989) and presented on stereonet plots. By

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1 All raw mapping data is collected in the right hand rule (RHR) format for recording strike and dip.
convention, the data is represented on lower hemisphere projections. The Lambert-Schmidt equal area plot (with a 1% area counting circle) will be utilized for density contouring because it offers a true random statistical evaluation of joint populations (Martin, 1991).

6.1.1. Determination of Structural Domains

Rational geotechnical and rock mechanics assessments require that the rockmass be divided into areas with similar engineering geology, geological structure and strength characteristics. The stope was divided into the following discrete zones:

- Vertical Depth
- Hangingwall vs. Footwall
- Longitudinally Along Strike.

It was found that the 660-885 Captive stope was defined by two principle joint populations: JSA and JSB with only minor variations in intensity (refer to Table 1). Mapping has indicated that there are large scale undulations in JSA. Both joint sets demonstrated a slight shift in strike characterized with a fairly consistent dip. The change in strike was located at approximately mid-stope height.

Noting this variation with depth, two random joint sets were noted. If 585 mL is selected arbitrarily as mid-stope height, JSC is more common on the lower sub-levels. Conversely, JSD appears to be a more controlling structure in the upper levels.

For design purposes, two primary plus one random structural domains have been identified\(^3\) (Figure 6.1.2).

\(^2\) All stereographic representations are given in the unweighted contour format as polar plots.
\(^3\) The predominant joint sets on each level are in highlighted in bold.
Quantification and Prediction of Wall Slough in Open Stope Mining Methods

Table 1 - "Distribution of Joint Sets in 660-885 Captive"

<table>
<thead>
<tr>
<th>Joint Set A</th>
<th>Joint Set B</th>
<th>Joint Set C</th>
<th>Joint Set D</th>
</tr>
</thead>
<tbody>
<tr>
<td>525 mL</td>
<td>092, 83</td>
<td>161, 75</td>
<td>N/A</td>
</tr>
<tr>
<td>545 mL</td>
<td>086, 88</td>
<td>173, 79</td>
<td>N/A</td>
</tr>
<tr>
<td>565 mL</td>
<td>092, 86</td>
<td>167, 87</td>
<td>169, 28</td>
</tr>
<tr>
<td>585 mL</td>
<td>271, 84</td>
<td>170, 80</td>
<td>041, 17</td>
</tr>
<tr>
<td>610 mL</td>
<td>272, 90</td>
<td>174, 82</td>
<td>N/A</td>
</tr>
<tr>
<td>630 mL</td>
<td>090, 87</td>
<td>170, 84</td>
<td>192, 00</td>
</tr>
<tr>
<td>650 mL</td>
<td>094, 83</td>
<td>173, 80</td>
<td>070, 12</td>
</tr>
<tr>
<td>660 mL</td>
<td>3 Poles*</td>
<td>172, 77</td>
<td>004, 02</td>
</tr>
</tbody>
</table>

Note: * A range of values exist for this joint set as follows:

JSA_1 (097, 80)
JSA_2 (073, 73)
JSA_3 (270, 72)

6.1.2. Determination of Design Sectors

Another design consideration that must be accounted for is the effect of wall orientation on stability. The 660-885 Captive stope strikes considerably different from all mining areas to the East. Most noteworthy is the degree of regularity in width and vertical extent. The orebody strikes in an east to west direction with an average dip of 70° to the north. Effectively, only one
wall orientation is possible: (270, 70 N). Therefore, two design sectors will be considered that separate the data into hangingwall and footwall groupings.

In the hangingwall data set, the stereonet (361 poles) plot indicates that JSB (26% peak concentration) and JSA are the predominant joint sets. A few scattered poles indicate that JSD was observed but not clearly defined (Figure 6.1.3).

Similarly, the footwall data set (239 poles) also exhibits both JSA and JSB (Figure 6.1.4).

**Figure 6.1.3 - "Hangingwall Data"**

**Figure 6.1.4 - "Footwall Data"**
For both walls, JSA is determined to be most critical joint set for design in terms of stope stability.

6.1.3. Development of Rockmass Models

Application of the DIPS software package enabled the peak pole population to be defined for each joint set (Table 2).

Table 2 - "Hangingwall and Footwall Data"

<table>
<thead>
<tr>
<th>Joint Set A</th>
<th>Joint Set B</th>
<th>Joint Set C</th>
<th>Joint Set D</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hangingwall</td>
<td>270, 90</td>
<td>172, 80</td>
<td>N/A</td>
</tr>
<tr>
<td>Footwall</td>
<td>090, 86</td>
<td>171, 81</td>
<td>N/A</td>
</tr>
</tbody>
</table>

With this information, a rockmass can be represented on a stereonet plot.

6.1.4. Kinematic Assessment of Failure Modes

Using the rockmass model defined in Section 6.1.3, the possibility of wall failure can be determined. In general, three (3) modes of failure will be analyzed:

1. Planar
2. Toppling
3. Wedge.

In each case, certain geometric conditions must exist for the physical possibility of failure. For this study, a simplistic approach for predicting failure will be used to initially assess the potential for the above mentioned failure modes.

For planar failure, a section of rock may slide along a continuous plane of weakness that lies sub-parallel to the slope face (i.e. strikes within +/- 20° of the slope). Conditions for

---

4 In cases of planar failure, there exists the potential for block failure. This is dependent on the joint condition and joint spacing.
imminent failure state that this plane of weakness must day-light the stope face and dip steeper than the friction angle.

In toppling failure, a set of steeply dipping, well developed and closely spaced discontinuities can lead to progressive deterioration if its orientation is sub-parallel to the slope surface. Toppling is dependent on the ratio of spacing to height.

The Markland test (Markland, 1972) will be used to assess the potential for wedge failures. This test predicts failure if the line of intersection formed by two intersecting planes lie within a defined friction circle.

In summary, the potential for wall failure or slough along JSA is present in both hangingwall and footwall. Depending on the orientation of the joint set, planar or toppling failure modes are expected. The potential for wedge failures is limited throughout the stope and is not as likely to occur.

6.1.5. Estimation of Strength Parameters

The host rocks found in the 660-885 Captive stope area are primarily mafic flows\textsuperscript{5}. The orebody is comprised of both QZ (75\%) and MZ (25\%) material. In situ samples have not been collected for appropriate laboratory testing due to the time constraints of this study. However, previous studies defining similar rock material in an adjacent stope are available from the following sources:

- CANMET (Cut & Fill Study\textsuperscript{6}) 1992
- CANMET (Stress Determination) 1985
- Terraprobe 1984

\textsuperscript{5} The hangingwall is comprised mainly of iron rich tholeiitic basalt flows. Differing slightly, the footwall tends to be more magnesium rich thus forming a komatiitic basalt.

\textsuperscript{6} "Ground Stability Guidelines for Cut & Fill Mining of Wide Ore Bodies" (Pakalnis et al, 1992)
It is anticipated that the material properties quoted will be comparable since the gold mineralization is similar.

The results of the tests for the mafic flows in the hangingwall, footwall and ore material are presented in Table 3 and Table 4 respectively.

**Table 3 - "Laboratory Testwork (HW and FW)"

<table>
<thead>
<tr>
<th></th>
<th>S.G.</th>
<th>UCS (MPa)</th>
<th>Tensile Str (MPa)</th>
<th>Elasticity (GPa)</th>
<th>Friction Angle</th>
<th>Cohesion (MPa)</th>
<th>Poisson's Ratio (#)</th>
</tr>
</thead>
<tbody>
<tr>
<td>UBC</td>
<td>2.9</td>
<td>230</td>
<td>...</td>
<td>88</td>
<td>50</td>
<td>60</td>
<td>0.25</td>
</tr>
<tr>
<td>CANMET (# of samples)</td>
<td>2.9 +/- 0.02 (12)</td>
<td>270 +/- 43 (5)</td>
<td>20 +/- 1 (8)</td>
<td>88 +/- 4 (6)</td>
<td>50 (4)</td>
<td>60</td>
<td>0.25 +/- 0.01 (6)</td>
</tr>
<tr>
<td>Terraprobe</td>
<td>...</td>
<td>91 +/- 17 (5)</td>
<td>...</td>
<td>26 +/- 5 (5)</td>
<td>...</td>
<td>...</td>
<td>0.29 +/- 0.07 (5)</td>
</tr>
<tr>
<td>JD Smith</td>
<td>...</td>
<td>147 +/- 31 (12)</td>
<td>19 +/- 2 (12)</td>
<td>57 +/- 13 (12)</td>
<td>...</td>
<td>...</td>
<td>0.1 +/- 0.02 (11)</td>
</tr>
</tbody>
</table>

Note: The numbers in () are the number of samples tested.

**Table 4 - "Laboratory Testwork (Ore)"

<table>
<thead>
<tr>
<th></th>
<th>S.G.</th>
<th>UCS (MPa)</th>
<th>Tensile Str (MPa)</th>
<th>Elasticity (GPa)</th>
<th>Friction Angle</th>
<th>Cohesion (MPa)</th>
<th>Poisson's Ratio (#)</th>
</tr>
</thead>
<tbody>
<tr>
<td>UBC</td>
<td>3.0</td>
<td>165</td>
<td>...</td>
<td>93</td>
<td>51</td>
<td>44</td>
<td>0.26</td>
</tr>
<tr>
<td>CANMET (# of samples)</td>
<td>2.9 +/- 0.01 (19)</td>
<td>169 +/- 26 (10)</td>
<td>19 +/- 1 (9)</td>
<td>89 +/- 6 (10)</td>
<td>43 (4)</td>
<td>40</td>
<td>0.26 +/- 0.02 (10)</td>
</tr>
<tr>
<td>Terraprobe</td>
<td>...</td>
<td>93 +/- 24 (5)</td>
<td>...</td>
<td>27 +/- 4 (4)</td>
<td>...</td>
<td>...</td>
<td>0.43 +/- 12 (4)</td>
</tr>
</tbody>
</table>

Note: The numbers in () are the number of samples tested.
6.2. ROCK MASS CHARACTERIZATION

Part of the detailed line mapping program includes the collection of data that describe the joint characteristics. The following factors are the significant physical and mechanical properties recorded:

1. Orientation (strike/dip by RHR)
2. Discontinuity Spacing
3. Continuity (length, ends visible)
4. Discontinuity Condition (roughness, waviness, infill)
5. Rock Material (strength)

Two empirical methods of rockmass classification were utilized: the South African Council for Scientific & Industrial Research (CSIR - Bieniawski, 1973) and Norwegian Geotechnical Institute (NGI - Barton et al, 1974) methods. Both methods have origins in civil projects such as underground tunneling and dam construction. Their popularity and applicability in the mining industry is the result of modifications made by Laubscher (1976), Mathews et al (1981), Kendorski (1983) and Hoek et al (1988).

6.2.1. CSIR Method

The CSIR method defines an index of rock competency in terms of a Rockmass Rating (RMR). This index is comprised of five key parameters (Table 5) that are considered to influence the behavior of a rockmass (EQ. 6.2.1-1): strength factor (R₁), RQD factor (R₂), joint spacing factor (R₃), joint condition factor (R₄) and groundwater condition (R₅).

\[
RMR(\%) = R_1 + R_2 + R_3 + R_4 + R_5 \tag{EQ. 6.2.1-1}
\]

For each parameter, a significance rating is assessed that can be algebraically summed to yield the RMR. The universality of this method is gained from the intuitive nature of the RMR scale.

(expressed as a percentage). For instance, an RMR of 100% represents an intact and homogeneous rockmass.

In total, 273 RMR observations were recorded throughout the 660-885 Captive stope and were distributed accordingly (Figure 6.2.1). All recordings represent an average rating for a given wall condition.

It is proposed that the line mapping data be combined to form contour plots. An assumption of rockmass continuity allows for interpolation and extrapolation of RMR values in the vertical extent.

![Figure 6.2.1 - "RMR Distribution Data"

Note: Numbers on bar charts are frequency values.
### Table 5 - "RMR Charts (Hoek and Brown, 1980)"

#### A. Classification Parameters and their Ratings

<table>
<thead>
<tr>
<th>PARAMETER</th>
<th>RANGES OF VALUES</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strength of intact rock material</td>
<td>Point load strength index</td>
</tr>
<tr>
<td>&gt; 8 MPa</td>
<td>4-8 MPa</td>
</tr>
<tr>
<td>&gt; 200 MPa</td>
<td>100-200 MPa</td>
</tr>
<tr>
<td>10-25 MPa</td>
<td>3-10 MPa</td>
</tr>
<tr>
<td>Rating</td>
<td>15</td>
</tr>
</tbody>
</table>

| Drill core quality RQD | > 25% | 75%-90% | 50%-75% | 25%-50% | < 25% |
| Rating | 20 | 17 | 13 | 8 | 3 |

| Spacing of joints | > 3 m | 1-3 m | 0.3-1 m | 50-300 mm | < 50 mm |
| Rating | 30 | 25 | 20 | 10 | 5 |

| Condition of joints | Very rough surfaces | Slightly rough surfaces | Slightly rough surfaces | Slightly rough surfaces | Smooth rock | Joint wall rock | Joint wall rock | Soft joint wall rock |
| Rating | 25 | 20 | 12 | 6 | 0 |

| Inflow per 10 m tunnel length | None | < 25 litres/min | 25-125 litres/min | > 125 litres/min |
| Rating | 25 | 20 | 12 | 6 |

| Ground water | Joint water pressure | None | 0.0-0.2 | 0.2-0.5 | > 0.5 |
| Rating | 25 | 20 | 12 | 6 |

| General conditions | Completely dry | Moist only (interstitial water) | Water under moderate pressure | Severe water problems |
| Rating | 10 | 7 | 4 | 0 |

#### B. Rating Adjustment for Joint Orientations

<table>
<thead>
<tr>
<th>Strike and dip orientations of joints</th>
<th>Very favourable</th>
<th>Favourable</th>
<th>Fair</th>
<th>Unfavourable</th>
<th>Very unfavourable</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tunnels</td>
<td>0</td>
<td>-2</td>
<td>-5</td>
<td>-10</td>
<td>-12</td>
</tr>
<tr>
<td>Foundations</td>
<td>0</td>
<td>-2</td>
<td>-7</td>
<td>-15</td>
<td>-25</td>
</tr>
<tr>
<td>Slopes</td>
<td>0</td>
<td>-5</td>
<td>-25</td>
<td>-50</td>
<td>-60</td>
</tr>
</tbody>
</table>

#### C. Rock Mass Classes Determined From Total Ratings

| Rating | 100-81 | 80-61 | 60-41 | 40-21 | < 20 |
| Class No. | I | II | III | IV | V |
| Description | Very good rock | Good rock | Fair rock | Poor rock | Very poor rock |

#### D. Meaning of Rock Mass Classes

| Class No. | I | II | III | IV | V |
| Average stand-up time | 10 years for 5 m span | 6 months - 4 m span | 1 week for 3 m span | 5 hours - 1.5 m span | 10 min. - 0.5 m span |
| Cohesion of rock mass | > 300 kPa | 200-300 kPa | 150-200 kPa | 100-150 kPa | < 100 kPa |
| Friction angle of rock mass | > 45° | 40°-45° | 35°-40° | 30°-35° | < 30° |

#### E. The Effect of Joint Strike and Dip Orientations in Tunneling

<table>
<thead>
<tr>
<th>Strike perpendicular to tunnel axis</th>
<th>Strike parallel to tunnel axis</th>
<th>Dip 0°-20° irrespective of strike</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drive with dip</td>
<td>Drive against dip</td>
<td>Very favorable</td>
</tr>
</tbody>
</table>

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
By trial and error, trends of consistent RMR values are connected with similar RMR values. The resulting plots are given on longitudinal section (Figure 6.2.2, Figure 6.2.3, Figure 6.2.4 and Figure 6.2.5). Projecting these contour lines to areas deficient in data allows for the rockmass quality to be predicted.

Figure 6.2.2 - "MZ Hangingwall RMR Contours"
It is noted that a sufficient number of RMR values were collected to enable adequate contour plots to be developed for the MZ hangingwall and footwall rockmass.
Figure 6.2.4 - "115 QZ HW RMR Contours"
Only a few RMR observations were collected in the QZ portion of the 660-885 Captive stope. Therefore, the contour plots developed are not considered to be as reliable (Figure 6.2.4 and Figure 6.2.5).

Overall, the average RMR value calculated for the trial stope is as follows:
Table 6 - "RMR Calculation"

<table>
<thead>
<tr>
<th>Description</th>
<th>Rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rock Strength</td>
<td>R4 (100-200 MPa)</td>
</tr>
<tr>
<td>RQD</td>
<td>75 - 90%</td>
</tr>
<tr>
<td>Joint Spacing</td>
<td>0.05 - 0.3 m</td>
</tr>
<tr>
<td>Joint Condition</td>
<td>Slightly rough surface</td>
</tr>
<tr>
<td></td>
<td>Hard joint wall rock, less than 1 mm separation</td>
</tr>
<tr>
<td>Groundwater</td>
<td>Dry</td>
</tr>
</tbody>
</table>

average RMR => 74%

6.2.2. NGI Method

In a similar fashion, the NGI method classifies a rockmass with a rock index (Q). A brief discussion of this methodology can be found in Section 2.3.1.

In total, 104 modified NGI observations were collected in the 660-885 Captive stope. The overall average Q' value is calculated as follows:

Table 7 - "NGI Calculation"

<table>
<thead>
<tr>
<th>Description</th>
<th>Rating</th>
</tr>
</thead>
<tbody>
<tr>
<td>RQD</td>
<td>75 - 90%</td>
</tr>
<tr>
<td>Joint Set Number</td>
<td>2+1 random</td>
</tr>
<tr>
<td>Joint Roughness Number</td>
<td>Roughness (smooth)</td>
</tr>
<tr>
<td></td>
<td>Waviness (planar)</td>
</tr>
<tr>
<td>Joint Alteration</td>
<td>Trace to minor chloritic infill</td>
</tr>
</tbody>
</table>

average Q' => 13.0 - 15.0
6.3. GEOTECHNICAL DATABASE SUMMARY

From the structural data collected, the 660-885 Captive longhole stope can be described to have two well-defined joint sets and one random joint set. Stereonet analyses identified planar and toppling type failure modes as the potential source of dilution. Wedge type failures are not as likely to occur within this open stope.

Rockmass characterization has been defined by both the CSIR and NGI methods. From the 273 observations collected by the CSIR method, the average RMR is 74% (standard deviation +/- 8%). Using the 104 observations as per the NGI method produced an average Q’ value of 14.0.

This information is to be used for plotting in the Modified Stability Method (Table 8).

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<td>‘C’ Factor</td>
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² The critical joint set, in terms of stability, is JSA (// with respect to the open stope). Calculation of the inter-plane angle consistently ranges between 15°-20°.
A comprehensive CMS database has been constructed for Detour Lake Mine. In total, data from 49 surveys have been collected. Supplemental information has been obtained for each of these surveys to include the following items:

1. Wall Dip
2. Blasting Information (# of blasts, tracing, powder factor, helper rings)
3. Muckpile Level
4. Cable Bolting
5. Under or Over-Cutting
6. Drilling Information (upholes/downholes, final wall definition).

The DLM database is compiled on Table 1 and Table 2. The supplemental information can be found in the Appendix II.

7.1. 660-885 Captive Stope Summary

Within a period of two years, a total of 18 CMS surveys were carried out in the 660-885 Captive longhole stope (Figure 7.1.1).

CA (January 17, 1995)
CB (January 17, 1995)
CC (April 25, 1995)
CD (April 25, 1995)
CE (April 26, 1995)
CF (May 11, 1995)
CG (June 13, 1995)
CH (July 15, 1995)
CI (July 29, 1995)
CJ (August 23, 1995)
CK (November 15, 1995)
CL (November 27, 1995)
CM (December 13, 1995)
CN (December 18, 1995)
CO (January 19, 1996)
CP (January 27, 1996)
CQ (January 29, 1996)

In every case, the ‘A’ factor equals 1.0 and the ‘B’ factor ranges between 0.2 and 0.3.
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In essence, each survey captures the state of wall stability for a given time frame. Reduction of this data has yielded ELOS values that are converted into dilution using Eq. 3.4.1-2.
### Table 3 - "Panel ‘A’ Dilution Results"

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<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CC</td>
<td>0.2</td>
<td>0.4</td>
<td>9.0</td>
<td>6.7</td>
</tr>
<tr>
<td>CF</td>
<td>0.3</td>
<td>0.5</td>
<td>9.0</td>
<td>8.9</td>
</tr>
</tbody>
</table>

### Table 4 - "Panel ‘B’ Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CA</td>
<td>0.3</td>
<td>0.2</td>
<td>10.0</td>
<td>5.0</td>
</tr>
<tr>
<td>CB</td>
<td>0.4</td>
<td>0.2</td>
<td>7.5</td>
<td>8.0</td>
</tr>
<tr>
<td>CD</td>
<td>0.1</td>
<td>0.3</td>
<td>11.5</td>
<td>3.5</td>
</tr>
<tr>
<td>CE</td>
<td>0.2</td>
<td>0.4</td>
<td>11.5</td>
<td>5.2</td>
</tr>
<tr>
<td>CG</td>
<td>0.4</td>
<td>0.2</td>
<td>7.5</td>
<td>8.0</td>
</tr>
<tr>
<td>CH</td>
<td>0.5</td>
<td>0.3</td>
<td>7.5</td>
<td>10.3</td>
</tr>
<tr>
<td>CI</td>
<td>0.5</td>
<td>0.5</td>
<td>7.5</td>
<td>13.3</td>
</tr>
<tr>
<td>C J</td>
<td>7.1</td>
<td>0.6</td>
<td>7.5</td>
<td>103.0</td>
</tr>
<tr>
<td>CO</td>
<td>8.5</td>
<td>0.6</td>
<td>7.5</td>
<td>121.3</td>
</tr>
</tbody>
</table>

### Table 5 - "Panel ‘D’ Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CK</td>
<td>0.1</td>
<td>0.2</td>
<td>7.0</td>
<td>4.3</td>
</tr>
<tr>
<td>CL</td>
<td>0.1</td>
<td>0.1</td>
<td>6.0</td>
<td>3.3</td>
</tr>
<tr>
<td>CP</td>
<td>...</td>
<td>...</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>CQ</td>
<td>0.3</td>
<td>0.2</td>
<td>7.0</td>
<td>7.1</td>
</tr>
</tbody>
</table>

### Table 6 - "Panel ‘C’ Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CM</td>
<td>0.0</td>
<td>0.2</td>
<td>15.0</td>
<td>1.3</td>
</tr>
<tr>
<td>CN</td>
<td>0.0</td>
<td>0.2</td>
<td>15.0</td>
<td>1.3</td>
</tr>
<tr>
<td>CR</td>
<td>0.2</td>
<td>0.5</td>
<td>13.0</td>
<td>5.4</td>
</tr>
<tr>
<td>CS</td>
<td>0.2</td>
<td>0.7</td>
<td>13.0</td>
<td>6.9</td>
</tr>
</tbody>
</table>

2 This CMS survey CJ captures the Fall of Ground Incident immediately after its occurrence. The subsequent CMS survey CO is taken some time after the hangingwall failure to assess the stability of the sill pillar. Both surveys are not included in the calculation of stope dilution since they represent ground failures due to structure.
A more detailed analysis can be found in Appendix I.

In general, 660-885 Captive stope produced an average of 5.5% dilution (16.1% including the two ground control incidents). By definition, dilution is dependent on ore width. Therefore, it is recommended that the panels be assessed primarily with ELOS measurements.

The average wall slough measured in Panel ‘A’ is 0.2 m HW and 0.4 m FW (7.8% dilution based on two surveys). From the CMS surveys, the primary sources of dilution in this panel are from drillhole deviation in the upholes and both under-cutting and over-cutting from sub-levels. The hangingwall was supported with nutcage cable bolts.

Panel ‘B’ averaged 0.3 m HW and 0.3 m FW ELOS values based on seven surveys (7.6% dilution). The primary sources of dilution are from blasting, under-cutting from sub-levels and discrete structural wedges in the hangingwall. In this panel, bulge cables were installed.

The three surveys collected in Panel ‘D’ yielded an average ELOS of 0.2 m for both HW and FW (dilution of 4.9%). In this panel, a ‘dog-leg’ feature was present on the lower levels. During the life of the stope, this adverse geometric feature did not fail. It was found that the primary source of wall slough was from blast damage and over-cutting. The combination of bulge and straight cable bolts appeared to have assisted in increasing the stability of the ‘dog-leg’.

Finally, Panel ‘C’ produced an average of 0.1 m HW and 0.4 m FW based on four CMS surveys (3.7% dilution). Much of the dilution observed was from under-cutting. In this panel, both birdcage and straight cable bolts were utilized.

The initial analysis of the trial stope indicates that the hangingwalls performed slightly better than the footwalls. This reduction of wall slough is likely a result of the cable support. An
average difference in wall slough is estimated at 0.3 m ELOS which implies a 3-5% savings of dilution (6-10 m wide). Employing a dilution cost of $30/tonne, it can be shown that 1% dilution for a 500 000 tonne stope would cost $150 000. Therefore, in order for a $450 000 cable bolting program to be justified, it must effectively reduce dilution by 3% (Dunne et al, 1996). By this criteria, the cable bolting program was a success.

Observe that the influence of the cable bolts appears to have a stabilizing effect on the hangingwall. It is interpreted that the collective effect of cable bolts contributes to maintaining low wall slough values (0.2 m ELOS). With only a few observations, it is difficult to specify the degree to which cable bolts increase wall stability. For this same reason, the individual performance of various cable bolt geometries cannot be evaluated.

For the 660-885 Captive stope, the 'Point-Anchor' approach proved to be a reliable method for designing hangingwall support.

In general, the primary source of hangingwall slough was from blast damage and discrete structural failures. The wall slough accredited to blast damage is limited to the parallel joint set (JSA) (Figure 7.1.2), whereas footwall dilution was observed to be from 'over-cutting' and 'under-cutting' the sub-level.
7.2. **SUB-LEVEL RETREAT (SLR) STOPE**

The SLR stope (Figure 7.2.2) is located between the 560 mL and the 660 mL. This stope consists of six sub-levels (575, 590, 610, 630, 650 and 660) spaced 15.0 m apart. Sub-

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Additional information can be found in the CANMET Final Report entitled *Design Guidelines for Sub-Level Retreat Mining Method* (Pakalnis et al, 1995).

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Quantification and Prediction of Wall Slough in Open Stope Mining Methods
levels (3.6 mH x 5.0 mW) are developed within the ore and driven along strike for 250 m along the footwall contact. In general, sub-levels are slashed to the hangingwall limit. Exceptions to this are isolated to the weaker rock material found in the east (maximum allowable HW to FW width of 5.0 m based on rockmass). Access to the stope is gained from the hangingwall ramp that connects the cross-cuts on each sub-level.

The orebody has a nominal thickness that ranges between 5-10 m and is considered to be fairly regular. The dip ranges from 60° - 85° which allows for ore extraction by gravity flow. Cable bolting (>37 000 m of birdcage configuration) was designed for the hangingwall using the ‘Point Anchor Approach’. Additional cables were installed in the footwall to confine the movements of the footwall fault. Dilution was also controlled by placing isolated rib pillars to minimize the wall span.

**Production Characteristics:** This bulk production mining method is termed sub-level retreat based on its mining sequence. Blasts are taken along strike in a retreating fashion towards the central cross-cut access from both directions (Figure 7.2.1). In theory, an echelon shaped mining face is desired which allows for optimal productivity. As the mining front retreats, the slough material incurred in the open stope can be left behind without diluting the active areas.

Both vertical downholes and dumped upholes were utilized in this stope. The dumped holes were designed to ensure that a more competent brow would be left behind after each blast. Also, dumping the holes was an attempt to confine the throw of the blasted material. Minimizing the distance that muck is thrown minimizes the opportunity of diluting the ore.
A total of nine (9) CMS surveys were collected in the SLR:

- AA (June 8, 1995)
- AB (November 30, 1993)
- AC (November 30, 1993)
- AD (January 17, 1995)
- AE (January 18, 1995)
- AF (January 18, 1995)
- AG (June 28, 1994)
- AH (June 28, 1994)
- AL (June 28, 1994).

Analysis of the SLR was divided into three zones: Eastern, Central and Western extents. A more complete description of the individual CMS surveys can be found in the Appendix III.

Table 7 - "SLR Eastern Division Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>AB</td>
<td>0.1</td>
<td>0.4</td>
<td>7.0</td>
<td>7.0</td>
</tr>
<tr>
<td>AG</td>
<td>0.0</td>
<td>1.8</td>
<td>6.0</td>
<td>30.0</td>
</tr>
<tr>
<td>AI</td>
<td>0.0</td>
<td>1.8</td>
<td>6.0</td>
<td>30.0</td>
</tr>
<tr>
<td>AF</td>
<td>0.0</td>
<td>1.8</td>
<td>6.0</td>
<td>30.0</td>
</tr>
</tbody>
</table>

Note: Severe wall sloughing was observed in CMS surveys AG, AI and AF. Difficulty in locating these surveys with respect to the mine excludes them from the analysis.

Table 8 - "SLR Central Division Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>AD</td>
<td>0.1</td>
<td>0.1</td>
<td>7.0</td>
<td>2.8</td>
</tr>
<tr>
<td>AE</td>
<td>0.3</td>
<td>0.4</td>
<td>7.0</td>
<td>10.0</td>
</tr>
</tbody>
</table>

Note that all surveys were collected using the 'automatic survey' feature of the instrument. Also, all surveys were collected utilizing the 'boom method'.

---

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
Table 9 - "SLR Western Division Dilution Results"

<table>
<thead>
<tr>
<th>Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>AC</td>
<td>0.1</td>
<td>0.4</td>
<td>7.0</td>
<td>7.0</td>
</tr>
<tr>
<td>AH</td>
<td>0.1</td>
<td>0.4</td>
<td>5.0</td>
<td>10.0</td>
</tr>
<tr>
<td>AA</td>
<td>0.9</td>
<td>1.9</td>
<td>5.0</td>
<td>56</td>
</tr>
</tbody>
</table>

Note: CMS survey AA captures a severe ground failure due to the footwall fault being daylighted (1.9 m ELOS). It was observed that both the back (~ 5 m ELOS) and hangingwall (0.9 m ELOS) were affected by the failure.

7.2.1. SLR CMS Summary

The results of the dilution analysis indicates that wall slough was problematic for the SLR stope. Gauging from the CMS data, the average hangingwall and footwall slough measured is 0.2 m and 0.3 m respectively. Note that correct placement of the CMS surveys was not achieved in some cases due to the effects of 'shadows' and deteriorating stope wall conditions. In effect, the sloughing of the stope walls eliminated one of the key reference points used for correcting the CMS survey (i.e. creating a horizontal slice at a known sill elevation). Regardless, dilution is defined to include slough from both walls to yield the total dilution. The resulting estimate of dilution is 7.4% for the SLR stope. Note that these dilution values exclude wall slough measurements due to structure. The design curve being developed is primarily focused on predicting wall slough as a result of rockmass degradation and not due to structure.

It was found that the placement of pillars was the most effective control method for containing wall slough in the weaker footwall material.
Figure 7.2.1 - "SLR Mining Front"
Figure 7.2.2 - "SLR Longitudinal Section"
Figure 7.2.3 - "Over-Sized Problems in the SLR"
The overall view on the performance of the SLR was that it did not achieve production expectations (Pakalnis et al, 1995). Throughout the stope life, mine production has recorded high occurrences of over-sized material that contribute to delaying the mining cycle (Figure 7.2.3). The remedial solution was to reduce the (15 m off set) echelon-shaped mining front to resemble a more vertical mining face. Additional flexibility could be gained if mining could retreat quickly enough to leave the over-sized pieces associated with wall slough behind in the stope. However, poor ground conditions were encountered that required the placement of additional isolated rib pillars, thus reducing stope recovery and increasing the mining cycle.

7.3. **COMPLEMENTARY CMS DATABASE**

In total, 46 observations were compiled to form the complementary CMS database. This complementary database includes data collected from two other Canadian underground mining operations that were visited by the author (15 obs.).

<table>
<thead>
<tr>
<th>Myra Falls Operation</th>
<th>Westmin Resources</th>
<th>(11)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Campbell Mine</td>
<td>Placer Dome Canada Ltd.</td>
<td>(4).</td>
</tr>
</tbody>
</table>

Also, a supplementary source of CMS data (31 obs.) has also been provided by L. Clark (M.A.Sc. candidate) of the University of British Columbia.

<table>
<thead>
<tr>
<th>Lupin Mine</th>
<th>Echo Bay Mines Ltd.</th>
<th>(18)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ruttan Mine</td>
<td>Hudson Bay Mining and Smelting Co. Ltd.</td>
<td>(7)</td>
</tr>
<tr>
<td>Trout Lake</td>
<td>Hudson Bay Mining and Smelting Co. Ltd.</td>
<td>(4)</td>
</tr>
<tr>
<td>Contact Lake</td>
<td>Cameco Corporation</td>
<td>(2).</td>
</tr>
</tbody>
</table>
Table 10 - "Complementary Database (FW)"

<table>
<thead>
<tr>
<th>Obs.</th>
<th>$Q'$</th>
<th>$N'$</th>
<th>ELOS (m)</th>
<th>RF (m)</th>
<th>Wall Dip (degree)</th>
<th>Cable</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>15.0</td>
<td>16.0</td>
<td>0.0</td>
<td>3.6</td>
<td>77</td>
<td>Yes</td>
</tr>
<tr>
<td>2</td>
<td>2.0</td>
<td>0.8</td>
<td>1.0</td>
<td>5.7</td>
<td>80</td>
<td>No</td>
</tr>
<tr>
<td>3</td>
<td>1.0</td>
<td>0.8</td>
<td>1.0</td>
<td>5.0</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>4</td>
<td>9.4</td>
<td>7.0</td>
<td>0.1</td>
<td>4.4</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>5</td>
<td>9.4</td>
<td>7.0</td>
<td>0.0</td>
<td>5.3</td>
<td>82</td>
<td>No</td>
</tr>
<tr>
<td>6</td>
<td>9.4</td>
<td>5.6</td>
<td>0.1</td>
<td>6.5</td>
<td>89</td>
<td>No</td>
</tr>
<tr>
<td>7</td>
<td>...</td>
<td>9.3</td>
<td>0.6</td>
<td>7.0</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>8</td>
<td>9.0</td>
<td>21.6</td>
<td>0.0</td>
<td>7.3</td>
<td>90</td>
<td>No</td>
</tr>
<tr>
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<td>9.0</td>
<td>21.6</td>
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<td>6.6</td>
<td>90</td>
<td>No</td>
</tr>
<tr>
<td>10</td>
<td>9.0</td>
<td>8.6</td>
<td>0.1</td>
<td>4.7</td>
<td>90</td>
<td>No</td>
</tr>
<tr>
<td>11</td>
<td>5.3</td>
<td>5.0</td>
<td>0.8</td>
<td>7.3</td>
<td>60</td>
<td>No</td>
</tr>
<tr>
<td>12</td>
<td>4.5</td>
<td>8.1</td>
<td>0.2</td>
<td>7.7</td>
<td>52</td>
<td>No</td>
</tr>
<tr>
<td>13</td>
<td>5.0</td>
<td>9.8</td>
<td>0.4</td>
<td>6.0</td>
<td>45</td>
<td>No</td>
</tr>
<tr>
<td>14</td>
<td>30.0</td>
<td>36.0</td>
<td>0.2</td>
<td>7.6</td>
<td>73</td>
<td>No</td>
</tr>
<tr>
<td>15</td>
<td>30.0</td>
<td>31.5</td>
<td>0.3</td>
<td>8.4</td>
<td>75</td>
<td>No</td>
</tr>
<tr>
<td>16</td>
<td>30.0</td>
<td>31.5</td>
<td>0.2</td>
<td>7.9</td>
<td>75</td>
<td>No</td>
</tr>
<tr>
<td>17</td>
<td>15.0</td>
<td>22.5</td>
<td>0.4</td>
<td>16.7</td>
<td>60</td>
<td>No</td>
</tr>
<tr>
<td>18</td>
<td>15.0</td>
<td>22.5</td>
<td>0.6</td>
<td>21.2</td>
<td>60</td>
<td>No</td>
</tr>
<tr>
<td>19</td>
<td>6.3</td>
<td>6.6</td>
<td>0.3</td>
<td>2.3</td>
<td>73</td>
<td>No</td>
</tr>
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<td>20</td>
<td>14.2</td>
<td>17.0</td>
<td>0.0</td>
<td>6.5</td>
<td>71</td>
<td>No</td>
</tr>
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<td>21</td>
<td>7.0</td>
<td>7.4</td>
<td>0.5</td>
<td>5.4</td>
<td>77</td>
<td>No</td>
</tr>
<tr>
<td>22</td>
<td>...</td>
<td>9.3</td>
<td>0.2</td>
<td>2.6</td>
<td>50</td>
<td>No</td>
</tr>
<tr>
<td>23</td>
<td>...</td>
<td>9.3</td>
<td>0.4</td>
<td>3.3</td>
<td>60</td>
<td>No</td>
</tr>
</tbody>
</table>

Note that the shaded observations in Table 10 were not used for the development of the ELOS design curves since the footwall dips are less than 70° ('C' factor = 6.0 using the HW curve). In Chapter 11, this topic is discussed in greater detail.
Table 11 - "Complementary Database (HW)"

<table>
<thead>
<tr>
<th>Obs.</th>
<th>Q'</th>
<th>N'</th>
<th>ELOS (m)</th>
<th>RF (m)</th>
<th>Wall Dip (degrees)</th>
<th>Cable</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>3.3</td>
<td>4.5</td>
<td>1.1</td>
<td>7.3</td>
<td>50</td>
<td>Yes</td>
</tr>
<tr>
<td>2</td>
<td>3.3</td>
<td>3.8</td>
<td>1.9</td>
<td>7.7</td>
<td>52</td>
<td>Yes</td>
</tr>
<tr>
<td>3</td>
<td>3.3</td>
<td>5.4</td>
<td>1.0</td>
<td>6.7</td>
<td>65</td>
<td>Yes</td>
</tr>
<tr>
<td>4</td>
<td>5.0</td>
<td>5.6</td>
<td>0.4</td>
<td>6.0</td>
<td>45</td>
<td>Yes</td>
</tr>
<tr>
<td>5</td>
<td>1.0</td>
<td>1.9</td>
<td>5.2</td>
<td>7.9</td>
<td>75</td>
<td>Yes</td>
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<td>6</td>
<td>14.2</td>
<td>100</td>
<td>0.0</td>
<td>10.4</td>
<td>90</td>
<td>Yes</td>
</tr>
<tr>
<td>7</td>
<td>1.0</td>
<td>2.0</td>
<td>2.3</td>
<td>3.6</td>
<td>77</td>
<td>Yes</td>
</tr>
<tr>
<td>8</td>
<td>...</td>
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<td>3.3</td>
<td>60</td>
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</tr>
<tr>
<td>9</td>
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<td>1.2</td>
<td>2.9</td>
<td>4.9</td>
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<td>Yes</td>
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<td>10</td>
<td>12.2</td>
<td>0.5</td>
<td>3.2</td>
<td>5.5</td>
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</tr>
<tr>
<td>11</td>
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<td>...</td>
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</tr>
<tr>
<td>12</td>
<td>...</td>
<td>5.0</td>
<td>4.5</td>
<td>9.3</td>
<td>...</td>
<td>Yes</td>
</tr>
<tr>
<td>13</td>
<td>...</td>
<td>0.8</td>
<td>3.1</td>
<td>4.7</td>
<td>40</td>
<td>No</td>
</tr>
<tr>
<td>14</td>
<td>9.0</td>
<td>21.6</td>
<td>0.1</td>
<td>7.3</td>
<td>90</td>
<td>No</td>
</tr>
<tr>
<td>15</td>
<td>9.0</td>
<td>21.6</td>
<td>0.0</td>
<td>6.8</td>
<td>90</td>
<td>No</td>
</tr>
<tr>
<td>16</td>
<td>9.0</td>
<td>7.6</td>
<td>0.2</td>
<td>4.7</td>
<td>78</td>
<td>No</td>
</tr>
<tr>
<td>17</td>
<td>...</td>
<td>0.8</td>
<td>4.8</td>
<td>3.9</td>
<td>...</td>
<td>No</td>
</tr>
<tr>
<td>18</td>
<td>1.0</td>
<td>1.7</td>
<td>2.8</td>
<td>7.6</td>
<td>66</td>
<td>No</td>
</tr>
<tr>
<td>19</td>
<td>1.0</td>
<td>1.9</td>
<td>4.3</td>
<td>8.4</td>
<td>75</td>
<td>No</td>
</tr>
<tr>
<td>20</td>
<td>15.0</td>
<td>22.5</td>
<td>1.7</td>
<td>16.7</td>
<td>60</td>
<td>No</td>
</tr>
<tr>
<td>21</td>
<td>15.0</td>
<td>22.5</td>
<td>1.2</td>
<td>21.2</td>
<td>60</td>
<td>No</td>
</tr>
<tr>
<td>22</td>
<td>...</td>
<td>19.3</td>
<td>0.2</td>
<td>7.0</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>23</td>
<td>15.0</td>
<td>32.0</td>
<td>0.0</td>
<td>5.7</td>
<td>80</td>
<td>No</td>
</tr>
<tr>
<td>24</td>
<td>10.0</td>
<td>23.0</td>
<td>0.0</td>
<td>5.0</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>25</td>
<td>6.3</td>
<td>12.0</td>
<td>0.1</td>
<td>2.3</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>26</td>
<td>10.0</td>
<td>22.0</td>
<td>0.0</td>
<td>4.4</td>
<td>85</td>
<td>No</td>
</tr>
<tr>
<td>27</td>
<td>15.0</td>
<td>32.0</td>
<td>0.0</td>
<td>5.3</td>
<td>82</td>
<td>No</td>
</tr>
<tr>
<td>28</td>
<td>15.0</td>
<td>35.0</td>
<td>0.1</td>
<td>6.5</td>
<td>89</td>
<td>No</td>
</tr>
<tr>
<td>29</td>
<td>14.2</td>
<td>26.0</td>
<td>0.0</td>
<td>6.5</td>
<td>71</td>
<td>No</td>
</tr>
<tr>
<td>30</td>
<td>14.2</td>
<td>28.0</td>
<td>0.1</td>
<td>5.4</td>
<td>77</td>
<td>No</td>
</tr>
</tbody>
</table>

In all cases, the ELOS terms were computed in accordance with a standard that was developed by the UBC Geomechanics Group.
7.4. CMS DATABASE SUMMARY

In total, 96 data points (Table 1, Table 2, Table 10 and Table 11) have been accumulated from various Canadian underground operations to include a diverse range of stope sizes and rockmass qualities (46 obs. from DLM). Compilation of this information will assist in the development of the ELOS design curves.
8. NUMERICAL MODELING

Rock mechanics modeling, in general, can be classified under the category of ‘data-limited problems’. This statement suggests that when designing in rock, it is difficult to collect sufficient data to define the rockmass as per the conventional engineering approach. Therefore, a different approach is taken which is outlined by the following five (5) steps (Starfield and Cundall, 1988):

1. Define the problem and propose a speculated solution
2. Collect the necessary amount of data required
3. Utilize appropriate simplifying assumptions
4. Confirm assumptions by modeling variations of the problem
5. Validate the model to in situ conditions with instrumentation.

This approach is particularly useful for application in numerical models.

8.1. OBJECTIVES

The objective is to describe the development and application of a numerical model for predicting the stresses induced from mining the 660-885 Captive stope. Analysis of the numerical model results will enable one to confirm the appropriate ‘A’ factor to be used in the Modified Stability Method. Also, a relationship between stope stability and sill and rib pillar condition can be demonstrated as these have bearing on exposed stope span (i.e. strike length increases).

Stope performance will be assessed by two methods: Stress Signature (Nicholls, 1994) and Mohr-Coulomb. It is anticipated that a $\sigma_1$ vs. $\sigma_3$ plot will define a relationship that is characteristic to a given pillar design. For confirmation, the Mohr-Coulomb failure envelope analysis will also be provided. The latter method applies to any solid material (i.e. an

---

1 It can be stated that the stress values are collected from pillar core locations and the mid-span location immediately within the hangingwall or footwall surface.
application in soil mechanics utilizes this approach for foundation design). This approach is derived from the basic laws of statics, therefore, fundamentally it can be applied to any material; including rock.

Validation of the numerical model will be accomplished through calibration with various types of instrumentation (i.e. extensometers, strain cells, CMS) data and visual observations.

8.2. NUMERICAL MODELING BACKGROUND

The application of numerical modeling codes is an accepted method for engineering design for many disciplines. Models are created to simulate processes or phenomena in order to provide an approximate solution to a design problem.

Within the mining industry, the four most commonly used numerical methods are (Mitri, 1991):

1. Finite Difference Method
2. Discrete Element Method
3. Finite Element Method

The Finite Difference and Finite Element formulations are well-developed for non-linear type problems such as plastic behavior and transient heat flow. The application of these codes to model the elastic behavior of a rockmass is not common since they are computationally intensive. This thesis will utilize a software program that incorporates the Boundary Element Method (BEM). Specifically, the Indirect BEM is selected because it is considerably more efficient than the Direct BEM. Through the use of lumping techniques, the construction and solution to the boundary element matrix is reduced (through simultaneous use of constant intensity fictitious force and displacement discontinuity type boundary elements).

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
Application of BEMs are best suited for the following conditions (E. Hoek/E.T.Brown, 1980):

1. Homogeneous, isotropic and linearly elastic material
2. Finite medium enclosed by a finite external boundary
3. Uniform loading field of stress
4. Increasing gravitational loading with depth.

Computational methods of stress analysis fall into two categories:

1. Differential methods
2. Integral methods.

In differential methods, the formulation of the solution is gained by dividing the problem domain into smaller sub-sets (i.e. finite elements). This process of division (i.e. discretization) allows for basic assumptions of equilibrium to exist. These appear in the mathematical form of differential equations that define the strain displacement and stress strain equations or the continuity of stress and displacements between elements.

The integral method (i.e. boundary integral) is a solution defined in terms of traction and displacement variables (Brady and Brown, 1993).

**8.2.1. Software Criteria**

Several criteria were used in making the selection of this computing program.

1. Ability to handle 3-dimensional geometry
2. Ease of application
3. Computational effort.

The influence of geometry and spatial location of underground openings is considered to be one of the critical factors affecting stope stability. It follows that realistic simulations of in situ conditions will provide the most accurate prediction of induced stress.
Numerical modeling codes have historically been very difficult to use. The attractiveness of using a modeling package becomes more viable if it is 'user friendly'. Features that make a program 'user friendly' are simple model construction routines and automated discretization functions.

In addition, numerical models are limited by memory constraints. The nature of the calculations require large memory capabilities for numerous iterations. Such a combination results in an enormously time consuming computational effort. With today's increasingly powerful computers, larger models can be simulated on smaller computers in less time.

Out of the many commercial modeling programs currently available, 'Map-3D' was chosen for this project because it satisfied the above criteria.

8.2.2. 'Map-3D' Software

'Map-3D' v3.5 is a comprehensive 3-dimensional rock stability analysis package. This program is capable of constructing 3-dimensional models, analyzing induced stress and displaying displacements, strains, stresses and strength factors. Currently, there are four versions of this program designed to handle very specific problems. For this study, the Standard Version (SV) will be used for its ability to model up to 64,000 elements, multiple-step mine sequence, multiple elastic zones of different moduli, and discrete fault planes.

'Map-3D' is a BEM type solution utilizing differential equations. Its principal advantage is in correctly defining the far-field boundary conditions. This implies that the inherent discretization errors are restricted to the boundaries. Stresses and displacements are assumed to be fully continuous throughout the medium.

---

3 Product of Mine Modeling Limited, 16 Park Street, P.O. Box #386, Copper Cliff, Ontario, P0M 1N0 Phone: (705) 682-1572 FAX: (705) 682-0087.
Figure 8.2.1 depicts the Detour Lake Mine as modeled in three-dimensions by the ‘*Map-3D*’ program\(^4\).

The analysis was designed to reflect the mine sequence in one month intervals. Figure 8.2.2 depicts the trial stope separated into these one month mined out blocks.

\(^{4}\) ‘*Map-3D*’ settings can be found in Appendix III.
Figure 8.2.2 - “Map-3D Model of 885 Captive Longhole Open Stope”
Table 1 - "Mine Sequence for the Trial Stope"

<table>
<thead>
<tr>
<th>Date</th>
<th>Panel</th>
<th>Mine Sequence</th>
</tr>
</thead>
<tbody>
<tr>
<td>October, November 1994</td>
<td>A (630-650 mL)</td>
<td>1</td>
</tr>
<tr>
<td>December 1994</td>
<td>A (610-630 mL)</td>
<td>2</td>
</tr>
<tr>
<td>January 1995</td>
<td>B (610-650 mL)</td>
<td>3</td>
</tr>
<tr>
<td>February 1995</td>
<td>A (585-610 mL)</td>
<td>4</td>
</tr>
<tr>
<td>March, April 1995</td>
<td>A (545-585 mL)</td>
<td>5</td>
</tr>
<tr>
<td>May 1995</td>
<td>B (565-585 mL)</td>
<td>6</td>
</tr>
<tr>
<td>June 1995</td>
<td>B (545-565 mL)</td>
<td>7</td>
</tr>
<tr>
<td>July 1995</td>
<td>FOG - No Mining</td>
<td></td>
</tr>
<tr>
<td>August, September 1995</td>
<td>D (630-660 mL)</td>
<td>8</td>
</tr>
<tr>
<td>October 1995</td>
<td>C (630-650 mL)</td>
<td>9</td>
</tr>
<tr>
<td></td>
<td>D (610-630 mL)</td>
<td></td>
</tr>
<tr>
<td>November 1995 (1st half)</td>
<td>C (565-630 mL)</td>
<td>10</td>
</tr>
<tr>
<td></td>
<td>D (565-585 mL)</td>
<td></td>
</tr>
<tr>
<td>November 1995 (2nd half)</td>
<td>C (585-610 mL)</td>
<td>11</td>
</tr>
<tr>
<td>December 1995 (1st half)</td>
<td>C (565-585 mL)</td>
<td>12</td>
</tr>
<tr>
<td></td>
<td>D (545-565 mL)</td>
<td></td>
</tr>
<tr>
<td>December 1995 (2nd half)</td>
<td>C (545-565 mL)</td>
<td>13</td>
</tr>
<tr>
<td>January 1996</td>
<td>C (545-565 mL)</td>
<td>14</td>
</tr>
</tbody>
</table>

8.3. NUMERICAL MODEL CALIBRATION

Model calibration is to be accomplished through correlation with visual stope inspections, extensometer, strain cell, and CMS data.

8.3.1. Extensometer

In this study, four extensometers (two multiple point and two point-snap ring) were installed in the trial stope (panels ‘C’ and ‘D’) to monitor the hangingwall performance (Figure 8.3.1 and Figure 8.3.2).

- 585 mL Extensometer #2 (2 point)
- 610 mL Extensometer #5 (2 point)
- 660 mL Extensometer #3 (4 point)
- 660 mL Extensometer #4 (4 point)
Measurements were collected four times daily with a CANMET datalogger pre-selected to coincide shortly after each shift blast.

8.3.1.1. Extensometer Analysis

In every case, the extensometer data indicated that the hangingwall was in a state of relaxation. The anchor points located near the wall surface measured discrete block movements. Dilation of the joints is evidence of a gravity induced block failure. This observation suggests that there is an absence of stress located immediately adjacent to the wall surface. It can be explained that the presence of blast induced fractures and natural jointing cannot be expected to carry stress since it is essentially forms a non-continuous medium. The depth of this zone is unknown.

Figure 8.3.1 - "Typical Extensometer Installation"
Additional extensometer information can be found in the Appendix IV.

8.3.2. Strain Cells

For this study, elastic strain recoveries are recorded with triaxial and uniaxial strain cells under the assumption that the principal stress directions could be predicted (i.e. relief method).

545 Sill Pillar (triaxial).
585 Rib Pillar (triaxial)
610 Rib Pillar (uniaxial).

The instruments used are CANMET strain cells (HNM - 76 mm diameter) which consist of a steel proving ring mounted with a wire (0.3 mm diameter piano wire, tensioned 20-40 N) perpendicular to the loading direction. It operates on a simple principle that relates strain with frequency. As the ring deforms under load, the wire frequency will increase due to compression or decrease due to relaxation. It is designed as a 'soft inclusion' type installation. This implies that the relative stiffness of the rock material to the instrument is much higher (~400 times) such that the instrument will deform easily with the rock. Selection of the strain cells is based on the following criteria:

1. High Precision and Resolution  
   (range of 0.2 mm with a resolution of 0.0002 mm)
2. Rugged Design  
   (frequency changes virtually unaffected by temperature and humidity)
3. Pre-Load or Seating Requirement  
   (~14 Mpa to maintain a solid contact)
4. Recoverability of Instrument
5. Easy Installation.

Strain data is also collected with a CANMET datalogger unit set for a six (6) hour recording interval.
Figure 8.3.2 - "Instrumentation Locations (660-885 Captive)"
Figure 8.3.3 - "CANMET Triaxial Strain Cell"

Figure 8.3.4 - "CANMET Uniaxial Strain Cell"
8.3.2.1. Strain Cell Analysis

The data collected for the triaxial strain cell located in the 545 sill pillar indicated that the stress measured is characterized by a decreasing trend. This strain cell was located in-between 'C' and 'D' panel. In total, the change in stress magnitude recorded did not exceed 10 MPa. It is noted that the numerical modeling predicted a change of stress of 20 MPa. One event that stands out in Figure 8.3.5 shows a sudden drop in stress. This coincides approximately two weeks after the opening up of the second slot which led to the floor heave incident (Section 8.3.3.1 - Case #3).

![545 Sill Pillar Triaxial Cell Stress Analysis](image)

Figure 8.3.5 - "545 Sill Pillar Strain Cell"

Similarly, the data collected from the triaxial cell located on 585 mL indicate a steady decline in stress magnitude (Figure 8.3.6). The event occurring in late September (i.e. sharp rise in stress) coincides shortly after the Fall of Ground (FOG) incident which will be described in Section 8.3.3.1.
And the results of the uniaxial cell appeared to be increasing for a one month duration (Figure 8.3.7). It was subsequently destroyed by stray flyrock from an adjacent blast.

In closing, the strain cells confirmed that the pillars were ‘yielding’. All three strain cells indicated a gradually decreasing trend in stress. It was observed that the total change in stress exceeded the measuring capabilities of the instrument (3-5 MPa). Since the pillars did not appear to be carrying load, the magnitude of stress could not be determined.
8.3.3. CMS Data

In the 660-885 Captive stope, nineteen (19) CMS surveys have been collected. Each survey provides in situ information on the condition of rib and sill pillars. The design of sill pillars is often for long term stability whereas rib pillars are for the short term (i.e. less than 1 year). At DLM, rib pillars were sized for a dual purpose: to maximize recovery and minimize span. Current empirical pillar design methods associate pillar strength to a width and height ratio plotted against induced stress values. In order to maximize recovery, the width to height ratio was selected to produce a 'yielding' pillar design. The term 'yielding' refers to the condition where stress is predicted to exceed the strength of the pillar (i.e. plots within the failed zone). Past experience gained at DLM observes that a 'yielded' pillar design will tend to fail gradually in relaxation in contrast to a rock-bursting mode of failure.
8.3.3.1. CMS Data Analysis

It is implied in a ‘yielding’ pillar design that the pillar condition would degrade as mining progressed. The primary purpose of leaving post rib pillars was to break up the exposed hangingwall and footwall surface area. This allowed for the open stope dimensions to be maximized.

![Diagram showing pillar stability classification]

The ‘yielding’ pillar design is computed using the ‘Confinement Formula’ (Lunder, 1994). A typical rib pillar in the 660-885 Captive stope has a width of 6.0-9.0 m (i.e. along strike) and a height of 6.0-10.0 m (i.e. HW to FW). Applying an induced core pillar stress load of 40-60 MPa plots the pillar design at the border of the failed zone (Figure 8.3.8). This
calculation confirms that the rib pillars are designed to fail by 'yielding' and hence to not carry any significant stress load.

Figure 8.3.9 - “Direction of Pillar Degradation”

It was observed that pillar integrity was reduced by gradual degradation in two dimensions: horizontal and vertical (Figure 8.3.9).

**Horizontal Slabbing:** During mining of this stope, apparent horizontal stress induced structures were observed. These tightly spaced structures generally unraveled (i.e. opened up) by the forces of gravity with each blast. An effort to improve pillar performance utilized vertically installed cable bolts as reinforcement.
Vertical Slough: Degradation of the rib pillar in the vertical dimension occurred as a sloughing mechanism. Pillar failure in this manner is attributed to the effects of blast damage and drill hole deviation based on the close proximity of the slot. Conditions of high confinement are often associated with blasting a drop-raise and developing a slot to full ore body width.

It is proposed that pillar degradation be represented by the cumulative summation of both horizontal and vertical pillar slough values.

\[
\text{Horizontal Slough + Vertical Slough = Pillar Degradation}
\]

EQ. 8.3.3.1-1

The following values of ELOS represent the pillar failure criteria (Table 2):

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Stability</th>
<th>Criteria</th>
</tr>
</thead>
<tbody>
<tr>
<td>(*)</td>
<td>Stable</td>
<td>(0.0 - 0.3 m ELOS)</td>
</tr>
<tr>
<td>(■)</td>
<td>Unstable - Minor</td>
<td>(0.3 - 1.0 m ELOS)</td>
</tr>
<tr>
<td>(★)</td>
<td>Unstable - Moderate</td>
<td>(1.0 - 2.0 m ELOS)</td>
</tr>
<tr>
<td></td>
<td>Yielded</td>
<td>(&gt; 2.0 m ELOS)</td>
</tr>
</tbody>
</table>

For every case, a ‘Map-3D’ model is required to predict the induced mine stress associated with a particular mining sequence. Numerous simulations are required, however, only 21 simulations were completed[^5]. Major (σ₁) and minor (σ₃) principal stress values are measured from the core location of each pillar. These values are plotted in accordance to Hoek’s failure criteria.

\[
\sigma_1 = \sigma_3 + \sqrt{m \sigma_c \sigma_3 + s \sigma_c^2}
\]

EQ. 8.3.3.1-2

where
- \( \sigma_1 \) - major principal stress [MPa]
- \( \sigma_3 \) - minor principal stress [MPa]
- \( \sigma_c \) - uniaxial compressive strength of intact rock [MPa]
- \( m, s \) - material constants [#].

[^5]: An average run requires 14-20 hours to compute.
This empirical relationship was first introduced in 1980 and has since been revised several times. As of 1988, the ‘m’ and ‘s’ constants have been defined as either peak or residual values. The peak material constant describes an undisturbed rockmass where excavations are carefully blasted or machined bored (EQ. 8.3.3.1-3 and 8.3.3.1-4).

\[
\frac{m_u}{m_i} = \exp\left(\frac{RMR - 100}{28}\right) \tag{EQ. 8.3.3.1-3}
\]

\[
s_u = \exp\left(\frac{RMR - 100}{9}\right) \tag{EQ. 8.3.3.1-4}
\]

where RMR - rockmass rating classification (as per Bieniawski)

\( m_i \) - determined from triaxial testing on intact lab specimens

\( m_u, s_u \) - undisturbed rockmass constants.

Conversely, the residual material constants represent the disturbed rockmass characterized by blast induced damage without confinement.

\[
\frac{m_{dis}}{m_i} = \exp\left(\frac{RMR - 100}{14}\right) \tag{EQ. 8.3.3.1-5}
\]

\[
s_{dis} = \exp\left(\frac{RMR - 100}{6}\right) \tag{EQ. 8.3.3.1-6}
\]

For this case study, the average RMR value is 75\% and the \( m_{dis} \) value is 2.8494 and the \( s_{dis} \) is 0.0155. These are the current m and s values employed at DLM.

Research conducted by INCO at the Thompson Division (Nicholls, 1994) observed several anomalies in their database. In all cases, the undisturbed line was an over-estimate of the in situ rockmass strength. Evidently, the disturbed line more accurately described the rock complexes...
found at the Thompson Division. Also, further refinement of the failure criterion resulted in a tapered lower limit approximating 45% of the laboratory UCS (Figure 8.3.10).

![INCO Failure Curves](image)

**Figure 8.3.10 - "Refined INCO Failure Curves"**

The following analysis will utilize this approach for the Detour Lake Mine data set.

**Case 1: Sill Pillar**

By definition, sill pillars are designed for long term stability. They are expected to remain stable throughout the stope life. As mining progressed in the 660-885 Captive, signs of instability were observed (Figure 8.3.11).

---

6 Note that the typical disturbed rockmass found at Thompson uses an ‘m’ and ‘s’ value of 1.23 and 0.0076 respectively.
### Table 3 - "Sill Pillar Stress Data"

<table>
<thead>
<tr>
<th>Location</th>
<th>Date</th>
<th>Condition</th>
<th>$\sigma_1$ (MPa)</th>
<th>$\sigma_3$ (MPa)</th>
<th>Horizontal Slabbing (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>B Panel Sill</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>38.5</td>
<td>14.0</td>
<td>...</td>
</tr>
<tr>
<td>C Panel Sill</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>39.0</td>
<td>14.6</td>
<td>...</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>40.0</td>
<td>15.0</td>
<td>...</td>
</tr>
<tr>
<td>Pillar (A/B)</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>42.0</td>
<td>16.0</td>
<td>...</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>Apr/May 95</td>
<td>Stable</td>
<td>...</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>B Panel Sill</td>
<td>June 95</td>
<td>Unstable</td>
<td>50.0</td>
<td>16.0</td>
<td>0.0</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>June 95</td>
<td>Unstable</td>
<td>51.0</td>
<td>23.0</td>
<td>...</td>
</tr>
<tr>
<td>C Panel Sill</td>
<td>Jan. 96</td>
<td>Unstable</td>
<td>56.0</td>
<td>25.0</td>
<td>...</td>
</tr>
<tr>
<td>B Panel Sill</td>
<td>July 95</td>
<td>Yielded</td>
<td>49.0</td>
<td>20.0</td>
<td>...</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>July 95</td>
<td>Yielded</td>
<td>58.0</td>
<td>21.0</td>
<td>...</td>
</tr>
<tr>
<td>B Panel Sill</td>
<td>Aug. 95</td>
<td>Yielded</td>
<td>...</td>
<td>...</td>
<td>0.0</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>Aug. 95</td>
<td>Yielded</td>
<td>...</td>
<td>...</td>
<td>3.5</td>
</tr>
<tr>
<td>B Panel Sill</td>
<td>Jan. 96</td>
<td>Yielded</td>
<td>54.0</td>
<td>21.0</td>
<td>1.0</td>
</tr>
<tr>
<td>A Panel Sill</td>
<td>Jan. 96</td>
<td>Yielded</td>
<td>61.0</td>
<td>25.0</td>
<td>4.0</td>
</tr>
</tbody>
</table>

Back analysis of the sill performance measured against the criteria designated in Section 8.3.3.1 has provided the baseline stress signature relationship to be used for future reference.
Superimposing the INCO disturbed failure curve (solid line), we observe that DLM data are significantly lower than the disturbed INCO failure curve (Figure 8.3.11).
Utilizing a 'least square' procedure with one constrained parameter, the slope of the failure curve is fitted to the DLM data by manipulating the $\sigma_c$ term ($\text{UCS}_{\text{fitted}} = 15 \text{ MPa}$) from EQ. 8.3.3.1-2 (Figure 8.3.12). This is observed to provide a reasonable fit to the in situ data.

Another format for representing the same data set is on the traditional Mohr-Coulomb failure envelope. In theory, yield or failure is not caused when a single normal or shear stress value reaches its maximum. Instead, failure is realized only for a critical combination of normal and shear stresses.
The Mohr-Coulomb failure envelope is defined by the following equation (Sowers, 1961):

$$\tau = c + \sigma \tan \theta$$

**EQ. 8.3.3.1-7**

where

- $\tau$ - shear [MPa]
- $c$ - cohesion [MPa]
- $\sigma$ - confinement ($\sigma_1$ and $\sigma_3$) [MPa]
- $\theta$ - friction angle [degrees].

A graphical representation of this equation plots shear vs. confinement on a 1:1 scale.

Plotting the major and minor principal stresses along the confinement axis yields half circles. The stress data collected for the sill pillar are plotted as shown in Figure 8.3.13. A line was constructed tangent to the Mohr circles representing the stress combinations that induced severe pillar slough. From this analysis, the failure envelope is defined by an in situ cohesion ($c$) value of **11.6 MPa** and friction angle ($\theta$) of approximately **10°**. These values represent the in situ rockmass incorporating the effects of various parameters (i.e. geological structure, blast damage).

Back analysis of two independent ground control incidents (Fall of Ground and Floor Heave Incident) will assist in calibrating the pillar failure curve.
Case #2: Fall Of Ground Incident

During the night shift on August 7, 1995, a large fall of ground event occurred in the upper portion of 660-885 Captive stope spanning Panels ‘A’ and ‘B’ (K. Dunne et al, 1996). An estimated ~70 000 tonnes of hangingwall material was displaced suddenly to create an air blast. Damage was minimal and confined to equipment located near the man-way landing. CMS survey CJ was taken on 565 mL to define the outer limits of this ground failure: ELOS values for the hangingwall (7.1 m) and footwall (0.6 m) are based on previous information (Figure 8.3.14).
Note that the failure surface extends beyond the cables’ supporting ability. It is speculated that the cables were successful in holding the rockmass together (Pakalnis, 1996). However, the driving force instigating the fall of ground was the continual deterioration of the rib pillar separating Panels ‘A’ and ‘B’. Earlier geotechnical mapping (Mah et al, 1995) had indicated that the area surrounding the rib pillar was assessed a lower rockmass quality (RMR = 60%).

Confined to production demands, the rib pillar (545-530) was not reinforced with cable bolts. It is also speculated that stress may have contributed to yielding of the rib pillars.

The effect of removing the rib pillars increased the exposed surface of hangingwall (designed RF = 16, actual RF = 25). Plotting the stope wall on the Modified Stability Quantification and Prediction of Wall Slough in Open Stope Mining Methods.
Method places the design well into the caved zone. The final stope dimension appears to have stabilized along the JSA structure (/j joint set).

Table 4 - "FOG Stress Data"

<table>
<thead>
<tr>
<th>Location</th>
<th>Date</th>
<th>Condition</th>
<th>$\sigma_1$ (MPa)</th>
<th>$\sigma_3$ (MPa)</th>
<th>Vertical (m)</th>
<th>Horizontal (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A/B Panel (HW)</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>42.0</td>
<td>16.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>A/B Panel (FW)</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>41.0</td>
<td>16.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>A/B Pillar (Core)</td>
<td>Oct/Nov 94</td>
<td>Stable</td>
<td>46.0</td>
<td>18.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td></td>
<td>Jan. 95</td>
<td>Stable</td>
<td>...</td>
<td>...</td>
<td>0.4</td>
<td>...</td>
</tr>
<tr>
<td></td>
<td>April 95</td>
<td>Unstable</td>
<td>...</td>
<td>...</td>
<td>...</td>
<td>1.4</td>
</tr>
<tr>
<td></td>
<td>May 95</td>
<td>Unstable</td>
<td>...</td>
<td>...</td>
<td>...</td>
<td>1.9</td>
</tr>
<tr>
<td></td>
<td>June 95</td>
<td>Yielded</td>
<td>...</td>
<td>...</td>
<td>0.2</td>
<td>2.9</td>
</tr>
<tr>
<td>A/B Panel (HW)</td>
<td>July 95</td>
<td>Yielded</td>
<td>33.0</td>
<td>14.0</td>
<td>1.5</td>
<td>2.9</td>
</tr>
<tr>
<td>A/B Panel (FW)</td>
<td>July 95</td>
<td>Yielded</td>
<td>39.0</td>
<td>17.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td></td>
<td>Aug/Sept 95</td>
<td>Yielded</td>
<td>Failed</td>
<td>Failed</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

It is interesting to note that the in situ stress is predicted to be approximately 50 MPa based on the stress determination studies conducted by CANMET (1985) at DLM. Application of the numerical model predicts the stress measured at mid hangingwall span to be 42 MPa as mining the Trial stope begins in October 1994. A subsequent numerical model run was conducted for July 1995 which shows a decline in stress magnitude. This is interpreted as an indication that the hangingwall is in a state of relaxation.
Comparison of the FOG data to the baseline analysis plots two yielded FOG points in the stable region (Figure 8.3.15). This suggests that the FOG incident was not a direct result of extreme stress conditions. This interpretation is made much more apparent by the Mohr-Coulomb approach.
From Figure 8.3.16, it is clear that 'yielding' pillar conditions exist only near the later stages of mining the panel.

**Case #3: Heave Incident**

During the final months of extraction, some ground problems began surfacing in the form of floor displacements. At this point, mining had progressed to the 585-610 block. For the majority of the 660-885 Captive, blast sequencing was primarily achieved with one active mining face for each panel. However, restricted by a central access system, two active mining faces were created in Panel 'C' to retreat towards the access cross-cut. The effect was to create a diminishing pillar situation which induced high levels of stress. Ground problems were experienced in the form of 'floor heave' and flat joints (Figure 8.3.18).
The buckling ground conditions resulted in large pieces of over-sized ore material and an unsafe working situation. Several CMS surveys (CM, CN) were conducted in Panel ‘C’ that confirmed the source of the over-sized. It revealed an arched shaped failure occurring within the ore block (565-585).

Table 5 summarizes the numerical modeling results obtained for the floor heave incident. It is highlighted by a sudden drop in $c_3$ values occurring in the month of November. The model suggests that the induced stress is being re-directed from the (565-585) ore block. Interpretation of the modeling results requires that the effects of mine sequencing be incorporated into the back analysis. It is noted that a diminishing pillar is created by opening up the second slot.

Associated with diminishing pillars is a high level of induced stress (i.e. flat jointing). Historically, pillars at DLM tend to fail in high stress environments by 'yielding'. Therefore, the sequencing of the second slot caused a sudden removal of stress as the ore block 'yielded'. Shortly after, a ground failure occurred that was characterized by large over-sized pieces. Mine personnel have also confirmed reports of audible cracking sounds.

Table 5 - "Floor Heave Stress Data"

<table>
<thead>
<tr>
<th>Location</th>
<th>Date</th>
<th>Condition</th>
<th>$\sigma_1$ (MPa)</th>
<th>$\sigma_3$ (MPa)</th>
<th>Vertical (m)</th>
<th>Horizontal (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>C Panel (HW)</td>
<td>Nov. 15, '95</td>
<td>Unstable</td>
<td>42.0</td>
<td>13.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C/D Pillar (Core)</td>
<td>Nov. 15, '95</td>
<td>Unstable</td>
<td>43.0</td>
<td>18.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C Panel (FW)</td>
<td>Nov. 15, '95</td>
<td>Unstable</td>
<td>43.0</td>
<td>20.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C Panel (FW)</td>
<td>Nov. 30, '95</td>
<td>Unstable</td>
<td>38.0</td>
<td>0.4</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C Panel (HW)</td>
<td>Nov. 30, '95</td>
<td>Unstable</td>
<td>37.0</td>
<td>1.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C Panel (HW)</td>
<td>Dec. 15, '95</td>
<td>Yielded</td>
<td>40.0</td>
<td>16.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C/D Pillar (Core)</td>
<td>Dec. 15, '95</td>
<td>Yielded</td>
<td>50.0</td>
<td>18.0</td>
<td>3.0</td>
<td>3.8</td>
</tr>
<tr>
<td>C Panel (FW)</td>
<td>Dec. 30, '95</td>
<td>Yielded</td>
<td>38.0</td>
<td>1.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>C Panel (HW)</td>
<td>Dec. 30, '95</td>
<td>Yielded</td>
<td>33.0</td>
<td>0.0</td>
<td>...</td>
<td>...</td>
</tr>
<tr>
<td>B Panel (HW)</td>
<td>Dec. 30, '95</td>
<td>Yielded</td>
<td>47.0</td>
<td>15.0</td>
<td>...</td>
<td>...</td>
</tr>
</tbody>
</table>

Plotting the corresponding stress data on the stress signature plot provides inconclusive results that neither indicate nor eliminate the influence of stress as a contributor to the floor heave incident (Figure 8.3.18). Several points taken from the floor heave data plot only on the y-axis. This is interpreted as the point where the Hoek failure curve becomes inappropriate for predicting failure. The research conducted at INCO, Thompson also support this observation.
Interestingly, the Mohr-Coulomb plot provides more definitive results (Figure 8.3.19).
From Figure 8.3.19, it is readily apparent that several points plot in a region that exceeds the pillar yield line. In retrospect, rearranging the sequencing could have minimized the incidents of over-sized and improved worker safety.

8.4. DISCUSSION

It was found that the extensometer and strain cell data did not adequately calibrate the numerical model. The extensometer data indicated minimal hangingwall movement throughout the life of the stope. Correlation of the movements to the observed ground control incidents showed an apparent delay in response between the rockmass and instrument. The occurrence of
the FOG and Floor Heave events did not coincide directly with the fluctuations in extensometer data.

The general trend of the functioning strain cells installed indicated a declining stress load occurring within the rib pillars. This implies that the rib pillars were ‘yielding’. The magnitude of stress induced from mining cannot be determined without any data demonstrating an increasing trend of stress. Therefore, it was found that the numerical model could not be calibrated with the instrumentation data collected.

Instead, two baseline pillar failure criterion were developed from the stress data. The calibration of these pillar failure criteria was accomplished with CMS data. Limited by a few case studies, the stress signature approach was found to be the least useful. Interestingly, the Mohr-Coulomb plots were more indicative of pillar ‘yielding’ conditions. This analysis proposes that the Mohr-Coulomb plots be interpreted in a slightly different manner from the traditional approach. This study suggests that the failure envelope line\(^7\) (F.O.S. = 1.0) be substituted with a ‘yielding’ pillar boundary. Note that the above analysis assumes the pillars to behave plastically.

The results of the numerical modeling suggest that the corresponding ‘A’ factors will range between 0.2 - 0.4. In fact, these values are expected at DLM since the measured major principal stress is oriented in the East-West direction. Keeping this in mind, the modeling produced relatively large stress values (i.e. 40-60 MPa measured at the wall contact) oriented parallel to the exposed stope surface\(^8\). However, the primary failure mechanism observed for producing wall slough is gravity. Evidence that supports this interpretation is provided from the extensometer data. In addition, observations of wall slough (i.e. failing on structure) made by mine personnel would also suggest that the hangingwall demonstrated non-elastic behavior. It can then be stated that the hangingwall is in a state of relaxation. Therefore, the ‘A’ factor as per the Modified Stability Method is set to equal 1.0.

\(^7\) The line constructed as a tangent to the failed circles on the Mohr-Coulomb plot.
\(^8\) The in situ stress has been determined to be ~ 50 Mpa (i.e. stress induced parallel to the stope wall is 2.6 times 0.029 MPa/m (vertical stress gradient).
8.5. **Numerical Modeling Summary**

The objective of this numerical modeling exercise was to construct a three-dimensional model to predict the induced stress regime from mining the 660-885 Captive stope. Utilizing the predicted stress values, in conjunction with CMS data, enables the development of a pillar failure curve and the determination of the ‘A’ factor in the *Modified Stability Method*.

It was found that the results of the numerical model could be applied to predict stress values for linear and elastic behaving rock material. This type of analysis was useful in the assessment of rib pillar stability. Conversely, the results of the numerical modeling for the hangingwall did not appear to match with the instrumentation data and visual observations. At this point, a non-linear numerical model may be more appropriate for predicting the induced stress values within the hangingwall.
9. BLAST DAMAGE CRITERIA

Blasting is a common and efficient method for excavating rock within the mining industry. It is a process that utilizes explosives for the purpose of breaking rock in a controlled manner. In a literal sense, ‘controlled blasting’ can be taken as an oxymoron since blasting is characterized by both constructive and destructive forces. At DLM, a study was conducted to ascertain the effects of blasting.

9.1. OBJECTIVES

This Chapter describes the blast monitoring program that was used to measure initiation and detonation sequences for longhole blasts. The objective is to relate the release of blast energy, in terms of vibration analysis, to incipient blast damage. Ultimately, a blast damage criteria would be developed using a traditional peak particle velocity approach. In calibrating this relationship with ELOS, the impact of blasting can then be related to dilution.

9.2. BLASTING THEORY

Rock is broken by the process of blasting as a result of rapid chemical energy release. The energy gained from converting a solid to gas is transformed into various forms: thermal (i.e. heat and light), and mechanical (i.e. high gas pressure induced impulse waves). It can be said that blasting impacts the stability of rock structures by the following fragmentation mechanisms:

1. Generating new fractures and cracks
2. Dilating in situ discontinuity surfaces
3. Inducing failure along major discontinuity surfaces.

New fractures are generated because blasting exploits the brittle nature in which rocks fail (i.e. strong in compression but relatively weak in tension). Detonating an explosive product in confinement produces a high compression wave that travels throughout the rockmass.
Reflection of this wave by various planar surfaces converts the compression wave into a tension wave. If the reflected wave is sufficiently greater than the strength of the rockmass, fractures are induced.

As mentioned above, one of the by-products of blasting is the creation of large volumes of gases. It follows that blocks of rock material are displaced from the original rockmass as a result of the injection of these high pressure blast gases into pre-existing discontinuity surfaces.

Finally, depending on the magnitude and frequency of the vibration signature, failures in a rockmass can be initiated along predominant slip surfaces.

In all cases, the effects of blasting can be related to the vibration signatures generated. Common to the industry are the terms 'near-field' effect and 'far-field' effect that describe the rock fragmentation mechanisms with respect to the proximity of the vibration source. The first two fragmentation mechanisms are attributed to 'near-field' effects (peak particle velocity) of blasting, whereas the latter is governed predominantly by 'far-field' effects (peak particle acceleration).

Of importance to this study is the influence of 'near-field' effects on underground workings and open stope walls. This effect is commonly measured in terms of peak particle velocity (PPV) which describes the process of inducing fragmentation by strain.

\[ \varepsilon = \frac{PPV}{V_p} \]

EQ. 9.2-1

where: \( \varepsilon \) - strain

\( PPV \) - peak particle velocity

\( V_p \) - compression wave velocity for a rockmass.
If a rock is assumed to fail in a brittle manner, Hooke's Law can be substituted to yield EQ. 9.2-2.

\[
PPV_{\text{max}} = \frac{(\sigma_T)(V_p)}{E}
\]

EQ. 9.2-2

where: 
- \(PPV_{\text{max}}\) - maximum peak particle velocity withstood by the rockmass before tensile failure
- \(\sigma_T\) - tensile strength
- \(V_p\) - P-wave velocity of propagation for a rockmass
- \(E\) - Young's Modulus.

With these basic equations, critical vibration levels can be estimated for specific rock types. However, there are other site specific characteristics (i.e. geological structure, quantity of explosive, distance from source, detonation sequence/direction, etc.) that have not been considered.

Blast vibration analysis is to be accomplished using an empirical approach to develop a site specific blast damage criterion.

9.2.1. Vibration Analysis

The most common form of vibration attenuation equation observed in the literature is the simple power curve. It is often referred to as the scaled distance relationship. This method states that a relationship exists between PPV and the amount of explosive initiated in a blasthole for a given distance. In a number of publications, scaled distance has been found to adequately describe blast vibration attenuation to be governed by an inverse square law (USBM, 1965)\(^1\) relating the amount of detonating explosive and the distance from the blast.

---

\(^1\) These curves defined will be considered site specific due to the variable nature of rockmasses.
Conducting a 'best fit' regression analysis for blast vibration data results in the following equation:

$$PPV = K \left( \frac{R}{\sqrt{Q}} \right)^{-b}$$  \hspace{1cm} \text{EQ. 9.2.1-1}

where:  
- **PPV** - maximum peak particle velocity [mm/s]  
- **Q** - maximum instantaneous explosive charge [kg]  
- **R** - distance from the blast to a given monitoring location [m]  
- **K, b** - site constants.

The site constants 'K' and 'b' are defined as the y-intercept at $R/(Q^{0.5}) = 1.0$ and the regressed slope value respectively (as plotted on a logarithmic-logarithmic relationship). Note that this type of analysis is indicative of the 'far field' effect (vibration events located ~60-90 m away) since the method considers only one blasthole at a time. At DLM, the 'far field' is defined as approximately **25.0 m** away from the vibration source. Scaled distance relationships are normally employed for spherical charges or short decked explosive columns.

For 'near field' effects, the rockmass is affected by the number of detonating blastholes and length of charge (Blair, 1990). The initial review of the DLM data suggests that a 'near field' analysis is required. To accommodate for a changing cylindrically shaped charge, Holmberg and Persson (1979) describe a more rigorous approach.

$$PPV = \left( K \gamma \right)^{\alpha} \int_{0}^{H} \frac{dx}{\left[ R_0^2 + (R_0 Tan \phi - x)^2 \right]^{\beta/2}}^{\alpha}$$  \hspace{1cm} \text{EQ. 9.2.1-2}

where:  
- **PPV** - peak particle velocity  
- **K, \alpha, \beta** - site constants  
- **\gamma** - linear charge loading [kg/m]

---

2 The 'far-field' range is the distance traveled by the vibration signature within the time it takes to consume the entire explosive column. It is computed as a function of the explosive VOD and in situ P-wave velocity. Details are included in the Appendix.

3 The very 'near-field' ranges from 10-20 m or two drillhole lengths away (B. Forsyth/A. Moss, 1990). Holmberg and Persson (1979) describe 'near-field' to be within one explosive column length.
Analysis of the DLM data will first consider the ‘far field’ approach. If a low correlation coefficient is obtained, then a ‘near field’ analysis will be required.

In addition, a classical analysis technique to determine the frequency content of a vibration signal will be used to assist in reducing and interpreting the data. An appropriate discrete Fourier transform (D.A. Anderson, 1995) is applied to the spectra.

\[
H_w = \sum_{k=0}^{N-1} h_k \left( \frac{2\pi i k w}{N} \right)
\]

EQ. 9.2.1-3

where: \(H_w\) - transform complex at frequency \(\omega\)
\(h_k\) - data points
\(N\) - number of data points.

The Fourier spectra analysis will provide an indication of seismic efficiency for a given vibration event. These calculations are computed by the software package provided by the BMX manufacturer.

9.3. Blast Vibration Test Site

The 935 Q120 longhole stope is located between the 460 mL and 525 mL (Figure 9.3.2). It is oriented parallel to the mined out Q100 stope. The orebody dips steeply (80° - 85°) and ranges in width between 3.5-5.0 m. Within the ore, sub-levels are spaced on 14.0 m intervals and slashed to full ore width to allow for the drilling of parallel downholes. Ultimately, the stope will have a height of 60.0 m and a strike length of 60.0 m (HR = 15). Rib pillars are left behind to break up the hangingwall span.
In general, the rockmass was also observed to be 'good' (RMR = 80-85%). Due to time constraints, only a few observations were made on 480 mL. Over-hangs were measured from engineering sections: HW averages 0.6 m, FW ranges 1.5-2.0 m (Figure 9.3.1).

**Production Characteristics:**

These panels were recovered as longhole retreat along strike. The overhand method of extraction is utilized. In an effort to control wall slough, only the swell material was removed (modified shrinkage approach). Due to production constraints, this is not always the case. The orebody is considered narrow, therefore, an extra hole is drilled inbetween the rings to form what is referred to as an 'A-rings' which is designed to assist in blasting efficiency. Mapping of the stope indicates that there were no major detrimental geologic structures present. Design of this stope, based on rockmass, excluded the need for cable bolt support.

**Figure 9.3.1 - "Typical Cross-Section (935 Q120)"**
LONGHOLE PRODUCTION BLASTING
935 Q120 EAST LONG-SECTION

Figure 9.3.2 "Longitudinal Section"

TRUE SECTION ON REFERENCE AZIMUTH 260°00'00"
All of the production blasts were monitored in panel ‘F’. In total eight (8) blasts were scheduled for the 480-510 ore block. Implementation of the blast monitoring program allowed for only seven (7) of the blasting events to be recorded.

Blast 1 Slot raise is opened between subs (June 2)
Blast 2 (F6) Slot is opened to full width (June 10)
Blast 3 (F7) Production blast (Rings # 160-166) (June 11)
Blast 4 (F8) Production blast (Rings # 167-172) (June 16)
Blast 5 (F9) Production blast (Rings # 173-178) (June 18)
Blast 6 (F10) Production blast (Rings # 179-185) (June 19)
Blast 7 (F11) Production blast (Rings # 186-190) (June 20)
Blast 8 (F12) Production blast (Rings # 191-196) (August 7).

Figure 9.3.3 - "Blast Monitoring Test Site"
9.3.1. Blast Monitor Equipment

A blast monitoring unit with sensors was purchased for this study. It was selected using some recommendations made by experts in the field (B. Forsyth, 1996 and ISRM, 1991). Specifications of the instrument are as follows:

1. Transducer Performance Characteristics
2. Time Zero Initiation
3. High Fidelity Sampling Rate
4. Multiple Channel Configurations
5. Sufficient Memory
6. User Friendly Software.

Transducer (Electromagnetic):

The transducer (alias Geophone) is a component designed to detect the magnitude of ground vibration in terms of either velocity or acceleration$^4$. The critical performance characteristic of a transducer is the linear response across a particular frequency range (1.0-1000 Hz)$^5$. Research completed by Noranda states that valuable blast information can be gained from transducers with a frequency response of up to 3 kHz and a sampling rate greater than 8 kHz (D. Sprott/B. Martin, 1990).

Data Collection Method:

Depending on the type of blast monitor, there may be several methods for collecting blast vibrations. The preferred method is via a wire triggering mechanism which allows for an exact time zero determination. A circuit is formed with the monitoring device and the blast pattern. Upon detonation of the first cap, the circuit is broken which starts the recording mechanism. This method allows for a true time zero reference point to aid in the waveform analysis.

$^4$ It is possible to obtain either displacement, velocity or acceleration by differentiation or integration. However, there are potential problems with these procedures. Therefore, it is recommended that the desired measurement be collected directly.

$^5$ The frequency range for ground vibration is typically 2-200 Hz (D.A. Anderson, 1995).
Multiple Channels:

If true peak particle velocity is desired, a triaxial transducer will require at least three channels; one for each direction.

Recording Device:

A built in memory device or a portable computer is required to record the blast vibration information.

Based on the above equipment selection criteria, the BLM Blastronic (BMX) was selected (Figure 9.3.4). It is capable of recording full waveform (64 kB) information on six (6) independent channels. The resolution capabilities of the channels are comprised of three 8 bit (1 MHz) and three 12 bit (200 kHz) cards.

Figure 9.3.4 - "BMX Monitor"
Collection of the blast data was facilitated by the installation of three (3) uniaxial and one (1) triaxial geophone grouted\(^7\) in 4" diameter holes on 480 mL sill at mid sub-level depth (approximately 7.5 m). Correct orientation of the geophones was enabled by cementing together segments of PVC piping. The geophones were linked to the monitor through lengths of shielded (18 gauge) cable. On site, both surface mounted (uniaxial) and grout installed (uniaxial and triaxial) geophones are available. All geophones were omni-directional 14 Hz velocity sensitive transducers (OYO 101 LT) that have a frequency range of up to 2 500 Hz\(^8\).

9.4. **Blast Damage Criteria Development**

The objective of the blast monitoring program was to optimize blast design by determining the following items and relating them to dilution:

- Determine the In Situ P-Wave
- Measure Duration of Effective Charge Separation
- Assess Detonator Performance (Degree of Cap Scatter)
- Measure and Identify Excessive PPV, Misfires, Cut-Offs, Sympathetic Detonations
- Develop a Basic Blast Damage Criteria and Calibrate.

**P-Wave Determination:** An estimate for the in situ P-Wave was computed with arrival time information.

**Duration of Effective Charge Separation:** Analysis of the complete vibration signature will allow for the time duration between two adjacent vibration events to be measured. The period of time that effectively allows one charge to detonate without interference from the subsequent charge is defined as the 'duration of effective charge separation'.

**Cap Scatter Determination:** The degree of cap scatter is often quoted from the supplier to range between 2-5% (ETI Explosives). Calculation of cap scatter is commonly defined as

\(^7\) Grouting is believed to be the best coupling method for connecting the geophone to rockmass (ISRM, 1991) and (DA Anderson, 1995). Coupling to the rockmass is most critical for the 'near-field' effect (R. Yang et al, 1993).

\(^8\) The frequency range for ground vibration is typically 2-200 Hz, therefore signal conditioning is not required.
the percentage difference between the nominal delay time and the mean measured time (A.R. Cameron/T. Kleine, 1992). At DLM, longhole blasts were initiated with non-electric detonators (short delay series). Misleading results were obtained from calculating the cap scatter as defined above because of varying nominal delay durations that separated each cap. For instance, detonator caps labeled from Period #1-14.5 are separated by a fixed delay duration (25 milliseconds). Additional caps added to the series labeled from Period #15-25 increase and vary in delay interval (from 100 to 600 milliseconds).

Intuitively, cap scatter is important when holes detonate out of sequence and for blast damage control. Therefore, cap scatter is redefined as the percentage error to cause 'overlap' or misfires measured relative to the next nominal delay time.

\[ \text{Cap Scatter} = \frac{\text{Actual} - \text{Nominal}}{\text{Cap Separation}} \times 100\% \]  
\text{Eq. 9.4-1}

where: Actual - detonation time measured by blast monitor [ms]  
Nominal - manufacturer's expected detonation time [ms]  
Cap Separation - nominal time interval separating two periods [ms].

Cap scatter results are critiqued relative to the 100% 'overlap' limit. Data points that plot above or below these limits imply potential incidents of misfires.

**Peak Particle Velocity:** Determination of PPV will be derived by combining the triaxial waveform signatures. A software package is provided by the blast monitor supplier to reduce the triaxial signals into a vector sum value.

Also, a direct comparison of individual vibration events to other events will provide an indication of the effectiveness of the detonating blasthole. Utilizing the provided Fourier Analysis allows for the effectiveness of a vibration event to be assessed in terms of the dominant frequencies.
9.4.1. Vibration Analysis (Scaled Distance)

The objective of a vibration analysis is to correlate the degree of damage to vibration levels. One common method is to define the critical particle velocity required for breaking solid rock. In the literature, there are many published vibration damage criterion. The following list demonstrates the wide variation in particle velocities observed in various rockmasses:

1. The critical particle velocity for hard igneous rocks is estimated between 700 and 1 000 mm/s (R. Holmberg/P. Persson, 1979).

2. Blast vibration levels at 300 mm/s have been observed to induce rock falls in unlined tunnels and is correlated to forming new fractures at approximately 600 mm/s (U. Langefors and B. Kihlstrom, 1978).

3. Excavation performance is correlated with peak threshold velocities for minor damage at 200 mm/s and substantial damage at 400 mm/s (B. Brady/E. Brown, 1993).

4. Blast damage thresholds in major underground single explosion tests estimate spalling at 900 mm/s and continuous wall damage at 1 800 mm/s (C. St. John/T. Zahrah, 1987).

5. A comprehensive study conducted at Kidd Creek Mine, Ontario (T.R. Yu, 1993) assigned some typical PPV values to subjective observations of blast induced damage (Table 1)\(^9\).

\(^9\) The blast damage is derived from blasts characterized by 200 mm diameter drillholes utilizing decked sequential blasting techniques.
Table 1 - "Typical PPV Damage Observations"

<table>
<thead>
<tr>
<th>Peak Particle Velocity (mm/s)</th>
<th>Damage Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>250</td>
<td>No noticeable damage to underground workings. Maximum tolerable vibration level for permanent workings.</td>
</tr>
<tr>
<td>500</td>
<td>Minor scabbing failure observed. Maximum tolerable vibration level for main drifts.</td>
</tr>
<tr>
<td>750</td>
<td>Development of cracks in weak ground, possibly requiring additional ground support. Maximum tolerable vibration level for temporary accesses.</td>
</tr>
<tr>
<td>1000</td>
<td>Possible formation of cracks 15 mm wide along plane of geological structures, potential failure.</td>
</tr>
<tr>
<td>1250</td>
<td>Major scabbing failure in an entire drift leading to failure of pillars associated with strong geological features.</td>
</tr>
<tr>
<td>&gt;1500</td>
<td>Possible formation of crack 15 mm wide in drifts of competent rock resulting in abandoned accesses.</td>
</tr>
</tbody>
</table>

This study will utilize a similar approach to define the amount of blast damage that is acceptable for the DLM in situ rockmass.

In theory, blast damage is a result of inducing dynamic stress during detonation. It is understood that the induced dynamic stress is a function of peak particle velocity and longitudinal wave velocity ($V_p$) for an elastic medium. During detonation, the magnitude of the dynamic stresses that develops around the blasthole induces primary cracking within the rockmass. Therefore, the critical particle velocity ($V_d$) at which the damage occurs can be defined as follows (S.P. Singh, 1992):

$$V_d = \frac{(V_p X T_e)}{E} 1000$$  

EQ. 9.4.1-1

where: $V_d$ - critical particle velocity [mm/s]
Utilizing this relationship, the critical particle velocity for the rockmass found at DLM is approximately $2800 \text{ mm/s}$.

Compilation of all the blasting data has yielded a PPV vs. Scaled Distance relationship shown in Figure 9.4.1. The equation that "best fits" this data is computed using the power fit inverse linear regression function provided by the Excel software package. Additional blasting information can be found in Appendix V.

---

**Figure 9.4.1 - "PPV vs. Scaled Distance"**

$10$ The R-squared value evaluates the reliability of the trend line. Note that the R-squared value is not an adjusted R-squared value. For logarithmic, power and exponential trend lines, Microsoft Excel uses a transformed regression model, in which the R-squared value refers to estimated rather than actual values.
Table 2 - "935 Q120 Blast Vibration Monitoring Results"

<table>
<thead>
<tr>
<th>Blast</th>
<th>ELOS (m)</th>
<th>Distance\textsuperscript{11} (m)</th>
<th>Ore. Width (m)</th>
<th>Explosive Scaled Distance (kg)</th>
<th>Ave. PPV (mm/s)</th>
<th>RF (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>F6</td>
<td>0.1</td>
<td>60.0</td>
<td>3.5</td>
<td>21.4</td>
<td>13.0</td>
<td>38.4</td>
</tr>
<tr>
<td>F7</td>
<td>0.3</td>
<td>54.0</td>
<td>3.5</td>
<td>28.7</td>
<td>10.1</td>
<td>43.4</td>
</tr>
<tr>
<td>F8</td>
<td>0.2</td>
<td>45.0</td>
<td>4.0</td>
<td>28.3</td>
<td>8.5</td>
<td>50.0</td>
</tr>
<tr>
<td>F9</td>
<td>0.4</td>
<td>34.0</td>
<td>4.0</td>
<td>28.8</td>
<td>6.3</td>
<td>56.2</td>
</tr>
<tr>
<td>F10</td>
<td>0.5</td>
<td>25.0</td>
<td>4.5</td>
<td>30.0</td>
<td>4.6</td>
<td>189.8</td>
</tr>
<tr>
<td>F11</td>
<td>0.7</td>
<td>16.0</td>
<td>4.0</td>
<td>22.1</td>
<td>3.4</td>
<td>293.0</td>
</tr>
<tr>
<td>F12</td>
<td>0.9</td>
<td>8.0</td>
<td>4.0</td>
<td>31.7</td>
<td>1.4</td>
<td>405.0</td>
</tr>
</tbody>
</table>

The distance value as defined for the scaled distance term is measured from each blasthole at mid-depth to the geophone location (\textit{Refer to Section 5.2}).

At DLM, the blast damage criteria is defined by the following equation\textsuperscript{12}:

\[
PPV_{50\%} = 855.3 \left( \frac{R_1}{\sqrt{Q}} \right)^{-1.32} \quad \text{EQ. 9.4.1-2}
\]

where:
- \( PPV_{50\%} \) - estimated average peak particle velocity [mm/s]
- \( R_1 \) - separation distance between detonation and monitoring location [m]
- \( Q \) - explosive charge [kg]
- Site constants - \( K = 855.3, b = 1.32 \).

Associated with this equation is a correlation coefficient of \textbf{0.81}. Based on a discussion held with B. Forsyth of Golder Associates Ltd.\textsuperscript{13}, correlation coefficient values > 0.7 obtained from

\textsuperscript{11} The distance value used for plotting is measured from the centroid of the blast to the geophone location.

\textsuperscript{12} The equation is derived for a ‘good’ rockmass (RMR = 70-80%) at Detour Lake Mine.

\textsuperscript{13} Golder Associates Ltd., Ste.# 500 - 4260 Still Creek Drive, Burnaby, B.C., V5C 6C6, Phone: (604) 298-6623 FAX: (604) 298-5253

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
performing regression analysis on field data can be considered to be exceptional in reliability. Observe that the data collected at DLM includes two blasts (F11 and F12) that are considered within the realm of 'near-field' analysis (i.e. the shaded figures in Table 2). Given that a strong correlation exists for the DLM data, a 'near-field' analysis as prescribed by Holmberg and Persson (1979) will not be required.

9.5. CALIBRATING THE BLAST DAMAGE CRITERIA

To complement the blasting monitoring program, CMS surveys and drill deviation data have been collected to provide the means for calibrating the blast damage criteria.

9.5.1. CMS Data

CMS surveys were collected after each blast recorded. A total of seven (7) CMS surveys were collected during the mining of the second lift\textsuperscript{14}.

- Blast 2 (F6) (June 11)
- Blast 3 (F7) (June 12)
- Blast 4 (F8) (June 15)
- Blast 5 (F9) (June 18)
- Blast 6 (F10) (June 19)
- Blast 7 (F11) (June 27)
- Blast 8 (F12) (August 8).

Reducing the above CMS surveys provides quantitative measures of stope performance in the 935 Q120. Dilution, measured in terms of ELOS, appears to increase as mining progresses along strike (Table 3).

\textsuperscript{14} All surveys were collected using the 'automatic survey' feature of the instrument. The full range of elevation (0-135°) is utilized with a one (1) degree elevation step. All surveys were surveyed utilizing the 'boom method' with the one exception of CMS survey F8. The 'mast method' was utilized since a survey crew was not available.
Table 3 - "ELOS Results for the 935 Q120 Stope (Unsupported)"

<table>
<thead>
<tr>
<th>CMS Survey</th>
<th>Hangingwall (m)</th>
<th>Footwall (m)</th>
<th>Ave. Width (m)</th>
<th>Dilution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>F6</td>
<td>0.0</td>
<td>0.1</td>
<td>3.5</td>
<td>3.0</td>
</tr>
<tr>
<td>F7</td>
<td>0.1</td>
<td>0.2</td>
<td>3.5</td>
<td>8.6</td>
</tr>
<tr>
<td>F8</td>
<td>0.0</td>
<td>0.2</td>
<td>4.0</td>
<td>5.0</td>
</tr>
<tr>
<td>F9</td>
<td>0.3</td>
<td>0.1</td>
<td>4.0</td>
<td>10.0</td>
</tr>
<tr>
<td>F10</td>
<td>0.4</td>
<td>0.1</td>
<td>4.5</td>
<td>11.0</td>
</tr>
<tr>
<td>F11</td>
<td>0.5</td>
<td>0.2</td>
<td>4.0</td>
<td>17.5</td>
</tr>
<tr>
<td>F12</td>
<td>0.5</td>
<td>0.4</td>
<td>4.0</td>
<td>22.5</td>
</tr>
</tbody>
</table>

Less than one meter of total slough was realized from both walls. Note that this stope was designed without cable bolts for either walls. Of interest, it is noted that the hangingwall (average 0.3 m ELOS) produced more wall slough than the footwall (average 0.2 m ELOS). This observation confirms the general assumption that hangingwalls are inherently less stable than the footwall primarily due to the effects of gravity.

The predominant failure mechanism for wall slough was observed to be related to the degree of 'under-cutting' and 'over-cutting' by sill development (Figure 9.3.1 and Figure 9.5.1).

Calibrating the CMS information to the blast damage criteria results in Figure 9.5.2.

For comparison, the typical blast damage criterion developed by the USBM\(^\text{15}\) (dashed line) is superimposed onto the DLM data (bold line). It was found that the rockmass found at DLM plots much higher and is considered a more competent. Calibration of the blast damage criterion with ELOS values was accomplished by making two assumptions:

---
\(^{15}\) The USBM blast damage criteria is based on a minimum 8 millisecond charge separation. Charge separation is the estimated time for the majority of the damped waveform energy to dissipate.
1. The blast sequences designed at DLM primarily retreat along strike in a V-pattern. It is assumed that significant wall damage is associated with the highest degree of confinement. Therefore, the lead blasthole (usually located near mid sill width) will contribute the most influence towards blast damage.\textsuperscript{16} The scaled distance is computed as follows:

$$ Scaled\ Distance = \frac{\left(\text{Ore Width}\right)}{2 \sqrt{W}} $$

EQ. 9.5.1-1

where: $W$ - weight of explosive from lead blasthole [kg].

2. It is understood that blast damage and exposed wall surface (radius factor) are important factors that contribute to the formation of wall slough. Differentiating the influence of either blast damage or radius factor separately is a difficult task. This analysis, for simplicity, will calibrate the blast damage criterion only for specific ranges of radius factor.

\textsuperscript{16} It is unclear which distance value should be used when predicting the damage incurred from the perimeter blastholes. Using distance values smaller than what is suggested by the above does not make intuitive sense.
Table 4 - "Calibrated Blast Damage Criteria"

<table>
<thead>
<tr>
<th>Blast</th>
<th>ELOS (m)</th>
<th>Distance (m)</th>
<th>Explosive (kg)</th>
<th>Scaled Distance</th>
<th>Predicted PPV</th>
<th>RF (m)</th>
<th>Cable Support</th>
</tr>
</thead>
<tbody>
<tr>
<td>CM</td>
<td>0.2</td>
<td>7.5</td>
<td>40.8</td>
<td>1.2</td>
<td>690</td>
<td>11.2</td>
<td>HW</td>
</tr>
<tr>
<td>CE</td>
<td>0.7</td>
<td>5.8</td>
<td>49.8</td>
<td>0.8</td>
<td>1120</td>
<td>12.0</td>
<td>HW</td>
</tr>
<tr>
<td>AD</td>
<td>0.2</td>
<td>3.5</td>
<td>36.2</td>
<td>0.6</td>
<td>1750</td>
<td>11.7</td>
<td>FW</td>
</tr>
<tr>
<td>F10</td>
<td>0.5</td>
<td>2.3</td>
<td>30.0</td>
<td>0.4</td>
<td>2760</td>
<td>11.1</td>
<td>Nil</td>
</tr>
<tr>
<td>F11</td>
<td>0.7</td>
<td>2.0</td>
<td>22.1</td>
<td>0.4</td>
<td>2640</td>
<td>11.0</td>
<td>Nil</td>
</tr>
<tr>
<td>F12</td>
<td>0.9</td>
<td>2.0</td>
<td>31.7</td>
<td>0.4</td>
<td>3350</td>
<td>11.0</td>
<td>Nil</td>
</tr>
</tbody>
</table>

17 Represents the modified Scaled Distance value as defined by EQ. 9.5.1-1.
Restricting the analysis to a narrow range of radius factors helps isolate the effect of blast damage. It is acknowledged that the effect of larger wall surfaces cannot be neglected for its contribution to increasing wall slough. By limiting the analysis to a specific range of radius factors, a trend of increasing wall slough is observed to be associated with increasing PPV. With only a few data points collected, the topic of calibrating scaled distance relationships warrants further research (Table 4).

Application of these assumptions allows the blast damage criteria to be extrapolated into the 'near-field' range with a reasonable degree of confidence.
9.6. BLAST MONITORING SUMMARY

A baseline study describing the effects of blast damage has been completed at the Detour Lake Mine. The results of this work has lead to the development of a calibrated scaled distance blast damage criteria. A trend has been established that implies blast damage can be controlled below a threshold value. The threshold value is identified as the critical particle velocity for the in situ rockmass. At DLM, it has been determined to be approximately 2 800 mm/s. Blast damage appears to be a significant contributor to dilution only if the scaled distance relationship is extrapolated beyond the ‘far-field’ range of influence and into what is considered the ‘near-field’ range. This extrapolation is possible since the regression analysis provided a high correlation coefficient to the in situ data. Also, two of the blasts (F11 and F12) monitored were located within the ‘near-field’ range.

Analysis of the CMS surveys has indicated the primary source of wall slough in the 935 Q120 stope to be from ‘over-cutting’ and ‘under-cutting’ the sub-level. It is observed that the hangingwall produced more wall slough which is contrary to the fact that the footwall was ‘under-cut’ more significantly (1.5-2.0 m).

This study also provided results to the objectives itemized in Section 9.4:

- The in situ P-Wave measured is approximately 7 km/s (~23 000 ft/s). This value is considered to be high which identifies the rockmass at DLM to be a competent rock medium for seismic travel. Minimal blast damage is expected.

- The minimum duration of effective charge separation has been estimated at 10 milliseconds. In the literature, it is often quoted as 8 milliseconds (H. Nicholls/V.

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All data is analysed based on the assumption that the compressional wave travels in a direct line. Sample calculation is included in the Appendix.

For example, granite is 12 000 - 20 000 ft/s (Atlas Powder Company, 1987).
Hooker, 1965). Therefore, the effect of cap scatter is minimized for the non-electric detonators utilized at DLM (separated by 25 to 600 milliseconds).

- A new cap scatter criterion has been determined for the blasting products. Analysis of arrival times suggests that incidents of misfires and cut-offs, resulting from cap scatter, are minimal. The occurrences of misfires and cut-offs are attributed to other factors such as drill deviation, inter-connecting geologic structure, over-drilling, etc.

- Seven blasts were monitored in the 935 Q120 longhole stope. Complete waveform analysis has identified trends in hole performance. Higher PPV levels have been observed to occur in holes located near 'helper-holes' and perimeter holes.

From this study, the effectiveness of 'tracing' and 'helper-holes' is still undetermined. It is likely that 'tracing' is not providing any significant benefit. In theory, the column of powder is thought to be side-initiated to reduce the impact of the explosive. This essentially reduces the powder factor for the same amount of burden. In doing so, the work required is unchanged with less energy available. This situation leads to high confinement blasting unless the charge is properly de-coupled. With the data collected, no correlation has been found between 'traced' holes and measured vibration levels (also supported by B. Forsyth, 1996).

At DLM, the use of 'helper-holes' refers to the practice of drilling an extra drillhole on the A-ring located near mid-sill width. The usefulness of this 'helper-hole' is debatable. Instead, it is suggested that two extra drillholes be placed along the blast perimeter. If the practice of 'tracing' is to continue, the effect of these two extra perimeter drillholes will effectively reduce the burden to better reflect the reduction in effective powder factor.
Without sufficient evidence, it is understandable why mine operators will often err on the side of over-blasting for difficult conditions. Therefore, it is recommended that additional vibration analyses is required to provide insight on the effectiveness of ‘tracing’ and ‘helper-holes’.

- Also, the analysis highlights the practice of reducing the amount of explosive loaded in the final ring. Essentially, holes with a reduced load of explosives produce higher than normal PPV levels.

- Incidents of sympathetic detonation have been consistently observed. The generic rule of thumb used at DLM states blast designs are limited to two (2) holes per delay for the purpose of blast damage control. Holes detonating with the same delay are usually located along the perimeter of the blast. Effectively, the holes are separated by a sufficient amount of spacing to allow each hole to detonate independently of the other. Excessive blast damage is not observed. Therefore, with the calibrated blast damage curve, the rule of thumb can be disregarded and replaced with a more realistic criteria.

Overall, at DLM, it is believed that blast damage can be confined to < 0.3 m ELOS if PPV levels are regulated below the threshold value.
10. DRILLHOLE DEVIATION STUDY

Drillhole deviation is recognized to be a contributing factor to stope dilution. Drilling inaccuracies produce situations where blastholes are either ‘over’ or ‘under’ burdened. In the case of under-burden, the drillhole is located closer to an adjacent drillhole than required. This results in the opportunity for blastholes to interact and sympathetically detonate and/or incidents misfires is clearly increased. Conversely, scenarios of over-burden refer to drillholes that are spaced too far apart to adequately fragment the rock. The increased fixation produces higher vibration levels that can ultimately damage the final wall perimeter. Of interest to this study is the drillhole deviation that occurs at the perimeter. At DLM, the drillholes located at the perimeter are designed along the ore and waste contact. Deviation from the design results in either lost ore or dilution.

By convention, total drillhole deviation is expressed as a percentage of the total drillhole length. It is reported, in an extensive survey conducted throughout various Canadian underground mine operations, that 3.5% is considered to be an obtainable drillhole deviation limit (J. Heilig/C. McKenzie, 1990).

10.1. OBJECTIVE

An investigative study was set up to assess the degree of drill deviation occurring in the longhole stopes. The study was restricted to the collection of collar and toe locations for break through holes. It is acknowledged that current technologies used in the exploration industry are available for monitoring drillhole trajectory\(^{20}\). However, these tests add considerable costs to the project and hence were not justified.

\(^{20}\) There are several quality control tests, such as Sperry sun and Trepani tests used in diamond drilling.
10.2. **DRILLHOLE DEVIATION TEST SITES**

Drillhole deviation data were collected from the two (2) longhole stopes:

- 935 Q120 (460-480) Tope Offset
- 935 Q120 (480-510) Tope Offset
- 935 Q120 (510-525) Tope Offset
- 865 Captive (630-650) Reference Line.

Both open stopes exhibit similar geological structures. There are two predominant joint sets mapped in this area (JSA - parallel to open stope, JSB - perpendicular to open stope). The orebodies trend in an east-west direction and dip approximately 70°-80°. Sublevels for the 935 Q120 and 865 Captive stope were developed within the ore at 14.0 m and 20.0 m interval respectively. All drillholes measured were drilled vertical.

In total, 39 drillholes (82 collar, 39 toe and 82 dip observations) were surveyed to assess the degree of drill deviation.

- 935 Q120 (29 obs.)
- 865 Captive (10 obs.).

### 10.2.1. Background

Improving drill accuracy represents an important quality control factor in minimizing dilution. According to Hendricks et al (1991), several parameters have been identified that are relevant to minimizing drillhole deviation.

- Drilling Equipment (drillhole diameter and length, type of drill steel, stiffness)
- Rockmass Properties
- Operating Procedures.

**Drilling Equipment:** At DLM, the longhole drilling is contracted out. Drilling is accomplished with either an air buggy, boom and arm set-up or electric hydraulic drill rig.
(Figure 10.2.1). The 2 1/8" diameter drillholes (Figure 10.2.2) normally range from **12.0 - 20.0 m** in length\(^{21}\). Drill strings consist of 4.0 ft length segments (R32 Hex rods with couplings). To control drillhole deviation, guide rods are used when collaring the hole. Also, current set-up procedures include the use of an inclinometer and laser line system to assist in positioning the drill rig.

**Rockmass Properties:** It is clear that the amount of drillhole deviation is dependent on the strength properties of the rockmass. For instance, properly matching the drill steel stiffness characteristics to rockmass hardness will affect drill performance. Even more so, drillholes that intercept surfaces that are at shallow angles with reference to the core axis will tend to bias the drillhole towards that direction of inherent weakness.

**Operating Procedures:** Like other Canadian mining operations, DLM provides an 'Incentive Program' that rewards underground workers based on production statistics. Historically, drilling performance is gauged upon the total footage completed per man-shift. This type of system encourages drill operators to maximize their time spent drilling which may lead to increasing human errors during the collaring process. In addition, drill accuracy is related to penetration rates [ft/min] as a function of the rockmass characteristics.

In addition to preventing blast damage, there are several other reasons for improving drilling accuracy that can have a direct economic impact on the operation. For instance, drillholes that exceed the limits can be rectified by re-drilling. In doing so, productivity suffers in part from the re-mobilization of drill equipment and disruption to the mine cycle. If drillholes are not inspected for re-drilling, the deviation will result in poorer fragmentation due to an unequal explosive distribution in the blast. The products of drillhole deviation are increased over-size and fine-size fragments in the muckpiles.

\(^{21}\) Rule of thumb limits used by Boart-Longyear (T. Kirkey, 1997): 2 1/8" diameter drill bits is 60 ft, 2 1/2" is 70 ft, 3.0" is 80 ft.
Figure 10.2.1 - "Boart-Longyear Electric Hydraulic Drill Rig"

Figure 10.2.2 - "2 1/8" Diameter Drill Bit and Guide Rod"
10.3. METHODOLOGY

A series of break-through drillholes were surveyed for collar and toe locations. Coordinates for the collar and toe locations were collected with an EDM-prism combination. Dip and azimuth calculations were computed using a standard diamond drillhole pick-up procedure.

The ensuing analysis will attempt to describe the drillhole deviation in terms of the operating procedures based on the premise that the drilling equipment is well matched to the in situ rockmass.

Consider now the following factors that govern the degree of drillhole deviation:

1. Collar Set-Up
2. Alignment
3. Drill Trajectory.

From the literature, an acceptable collar set-up error is one drillhole diameter (J. Heilig/C. McKenzie, 1990). At DLM, the collar set-up error is expected to be within one drillhole diameter (± 54.0 mm or 2 1/8”). With regards to alignment error, there are no current guidelines established. In general, the collar set-up and alignment procedures are within the control of the drill operator.

Drill trajectory is considered to be directly influenced by the material properties of both the drill equipment and the rockmass. Various studies have identified drill trajectory to be non-linear in behavior. The majority of the drillhole deviation occurs near the toe of the hole. From experience, the first 8.0 m of a 54 mm (2 1/8”) diameter drillhole can be accomplished with little to no measurable deviation.²²

²² Sandvik Rock Tools has published a brochure that supports this observed drilling accuracy for their Coromant Guide bits. Also, an internal report conducted at Campbell Mine, PDC observes similar results (C. Oakes/M. Desloges, 1996)
10.4. Data Analysis

Analysis of drillhole deviation data will comprise of plotting collar and toe locations on a Northing and Easting grid. Holes exhibiting zero drillhole deviation are placed at the origin (0,0). Performance guidelines, depicted as circles (collar - 0.1 m diameter and toe - 1.0 m diameter), are included for reference. Refer to Appendix VI for additional information.

Surveying errors are assumed to be negligible during the data collection process and ring mark-up procedures.

935 Q120 (460-480 mL): Collar errors are observed to represent 23% of the total drillhole deviation (+0.18 m). Figure 10.4.1 demonstrates a tight cluster pattern of collar locations near the origin.

Figure 10.4.1- "460-480 Collar and Toe Deviations (10 obs)"
The corresponding toe locations deviate an average of +0.59 m from design. Drillholes in this block appear to deflect preferentially in the north direction\(^{23}\). It appears that JSA (\(/\) to open stope) has the most effect on drilling.

The total deviation is calculated at 5.5\% (+0.8 m). Correcting for human set-up error results in an overall deviation error of 4.5\%.

Drillhole alignment is measured to occur within a +/- 5° range. No guidelines are currently in place.

935 Q120 (480-510 mL): Collar errors measured in this block average +0.75 m (represents 45\% of the total drillhole deviation). The increased amount of collar error is evident from the wider spaced cluster formed in the south-east quadrant. From Figure 10.4.2, the deviating trend is in both north-west and south-east directions.

Alignment error averages 4.4° and is estimated to be confined within a +/-5° envelope.

Toe deviation is measured to be +0.93 m off target. The combined total drillhole deviation in this block is +1.7 m (12.1\%). Correcting for human set-up error results in an 7.9\% deviation.

The drillholes appear to be governed by JSB (\(\perp\) to stope) and are trending in a north/west direction.

\(^{23}\) Only four holes demonstrated slight deviations in the South direction.
935 Q120 (510-525 mL): A tight cluster of collar locations plot in the north-east quadrant. The collar set-up error is determined to be +0.42 m off the mark (32% of the total drillhole deviation). Figure 10.4.3 shows the wide scatter pattern of the corresponding toe locations (+0.87 m off the mark). Holes tend to deviate in a south-west direction. It is speculated that the holes are deviating as they approach the bottom of the orebody plunge.

The total drillhole deviation in this area is 9.3% (1.3 m). Correcting for collaring error yields 7.8% which includes the influence of geological structure and equipment limitations.
Alignment error is measured to be within +/- 5.0° and averages 4.0°. The drillholes also appear to be controlled by JSB which has changed its orientation to a south/west direction.

865 Captive (630-650 mL): Note that the collar locations form another tight cluster in the south-west quadrant. Approximately 34% of the total error is attributed to an incorrect collar set-up. Toe deviations are measured to be +0.73 m and demonstrate no directional preference (Figure 10.4.4).

Alignment errors average 0.9° and are clearly within the +/- 5° range.

The total deviation is 1.1 m (5.5%) of which 0.4 m is attributed to uncontrollable parameters.
The drillholes appear to follow JSB and trend slightly towards the south/west direction.

Table 5 provides a summary of findings accumulated from the drillhole deviation program.

Table 5 - “Summary of Drilling Results”

<table>
<thead>
<tr>
<th>Collar (m)</th>
<th>Toe (m)</th>
<th>Dip (deg)</th>
<th>Total (m)</th>
<th>Total (%)</th>
<th>Drill Deviation (m)</th>
<th>Drill Deviation (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>935 Q120 - 460 mL</td>
<td>0.2</td>
<td>0.6</td>
<td>2.2</td>
<td>0.8</td>
<td>0.6</td>
<td>4.5</td>
</tr>
<tr>
<td>935 Q120 - 480 mL</td>
<td>0.7</td>
<td>0.9</td>
<td>4.4</td>
<td>1.7</td>
<td>1.1</td>
<td>7.8</td>
</tr>
<tr>
<td>935 Q120 - 510 mL</td>
<td>0.4</td>
<td>0.9</td>
<td>4.0</td>
<td>1.3</td>
<td>1.1</td>
<td>7.8</td>
</tr>
<tr>
<td>630-865 Captive</td>
<td>0.4</td>
<td>0.7</td>
<td>0.9</td>
<td>1.1</td>
<td>0.4</td>
<td>2.2</td>
</tr>
</tbody>
</table>

It is proposed that the following procedure be used to determine the proportion of wall slough attributed to drillhole deviation. In doing so, several assumptions are required:
• The first 8.0 m of the drillhole is free of drillhole deviation (based on literature review)
• Wall slough is approximated by triangulation (Figure 10.4.5)
• Blast damage is negligible.

Procedure for Converting Drillhole Deviation into ELOS

1. Plot the drillhole on a cross-section.
2. Calculate the area of wall slough contained between the actual and designed drillhole location. Estimate the area with a triangle.
3. Apply a tonnage factor/thickness to this area of wall slough.
4. Compute the ELOS by distributing the volume of wall slough over the entire length of the hole.

Figure 10.4.5 - "Drillhole Deviation Contribution to Wall Slough"
Utilizing this procedure, an estimate of the ELOS produced from drilling can be obtained.

Based on the drillhole deviation data presented in Table 5, the estimated wall slough is 0.2 - 0.4 m ELOS. This amount of wall slough represents a dilution of 4.0-9.0% (4 m wide stope) for the 935 Q120 stope.

10.5. DRILLHOLE DEVIATION SUMMARY

A complementary study has also been completed to assess the degree of drillhole deviation. The results collected indicate that the average total drillhole deviation is 8.0%. At DLM, this amount of drilling error is considered to be unacceptable. However, these values are still within industry norms. For the 935 Q120 stope, it is estimated that drilling will contribute 4.0-9.0% to dilution (0.2 - 0.4 m ELOS in a 4.0 m wide stope).

It is observed that collaring errors generally plot in clusters without any particular preference to spatial quadrants. These observations are attributed to repeatable survey errors incurred when painting up the rings and random set-up errors characteristic to each individual driller. Correcting for collar set-up error yields 5.6% drillhole deviation.

Toe errors were also observed. These errors, however, are dependent on rockmass, geological structures and drill equipment which are considered to be fixed design parameters.

The following are possible methods for improving drilling accuracy:

- Increase the drillhole diameter to >3.0" to enable ITH drilling. Within the literature, the use of larger diameter drill bits produce straighter holes as a result of a ‘stiffer’ drill string system (i.e. tube steel).
- Utilize one or two guide rods to stiffen up the drill string.
Correct and minimize the degree of collar error. In general, collar error is largely attributed to human set-up errors. A short list of techniques for reducing collar set-up error include ensuring accurate survey mark-ups for longhole rings, adequately prepared drill pads, and educating the drillers on the importance of quality control measures.

To date, there is no written standard governing drillhole alignment. It is proposed to introduce a new alignment error guideline of $\pm 2.5^\circ$.

Indirectly, dilution can be reduced if the management philosophy is changed to reflect more efficient mining practices. The current drive is to achieve production targets such as tonnes broken, meters drilled and ounces poured. Since mine design is defined as the process involving the compromise of inter-dependent variables (dilution, recovery, productivity and cost), it is suggested that an incentive program be designed with both rewards and penalties. Therefore, it is recommended to:

- Refine the longhole drilling program to include a mechanism for encouraging "quality" drilling instead of "quantity".

This study highlights the importance and sensitivity of controlled drilling to stope performance in narrow orebodies.

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24 An internal report published by Boart Longyear in cooperation with Placer Dome Canada suggests a recommended drillhole alignment of $10^\circ$ maximum (Yao and Desloges, 1996).
A database of slough observations has been collected for analysis with inferential statistical methods. Wherever possible, additional operating variables were collected to assist in interpretation. Appropriately, a multivariate analysis is utilized to determine if a relationship exists between these mine operating variables for the purpose of predicting ELOS values. The work presented was completed in conjunction with the University of British Columbia's Statistical Consulting and Research Lab (SCARL) group.

11.1. OBJECTIVE

The objective is to identify trends and produce logical conclusions with regard to dilution. By combining statistics with engineering judgment, it is anticipated that a priori classification boundaries can be defined by mathematical functions to separate between ELOS groups. It is assumed that the previous research conducted on the Modified Stability Method, based on hydraulic radius, can be superimposed onto a radius factor axis.

11.2. DESIGN PARAMETER DATABASE

In Chapter 7, the CMS database was compiled using data collected primarily from DLM (49 case histories). The analysis was enhanced with a supplemental source of 46 data points obtained from other Canadian underground operations\(^1\). This section will attempt to describe the characteristics of the database. Understanding the key parameters that comprise the database will enable the user to use the design curves with greater confidence (Table 1).

'A' Factor: In all the DLM case studies (49 obs.), it was observed that the 'A' factor was always equal to a value of 1.0. This implies that the hangingwall is in a state of relaxation.

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\(^1\) Note that this additional set of data points is provided by L. Clark of the University of British Columbia. Only the essential information (ELOS, N', and RF values) required for plotting on the Modified Stability Method were supplied. The mine operating variables as stated above were not available for these data points.
Under this stress state, wall slough is thought to be controlled primarily by gravitational forces. Evidence of this can be obtained from numerical modeling, collecting instrumentation data (i.e. extensometer, strain cells) or visual observations.

'B' Factor: A narrow range of values was obtained for the 'B' factor (0.2 - 0.3). In general, the joint set most critical to stability is often the one that is parallel to the stope walls. This observation is true for all the values collected at DLM (49 obs.)

'C' Factor: Intuition suggests that hangingwalls will behave differently from the footwall based on the effects of gravity. By this argument, footwalls are inherently more stable. The original work done by Mathews (1981) excluded the footwall from analysis. A friction curve was introduced in Potvin's (1988) study to account for the footwall stability.

To date, there are not enough data points to support the use of this friction curve (Figure 2.3.4) for design. Based on the data collected in this study, the friction curve does not appear to adequately correct for the footwall. It is suggested that footwall data be treated separately. In fact, it can be argued that as the footwall approaches vertical, it behaves more like a hangingwall. There is not much difference between a vertical hangingwall and a vertical footwall. This analysis uses the 'C' factor curve developed for the hangingwall indiscriminately for either hangingwall or footwall. A temporary guideline of C = 6.0 (i.e. 70° wall dip with a parallel joint set) has been determined as the cut-off for this analysis. Footwalls that dip less than 70° are not used in the analysis. This topic will be further discussed in Section 11.4.

Ore Width: The average ore width is 6.8 m with a standard deviation of +/- 3.0 m.

Stope Inclination: The average dip for the hangingwall and footwall is 74° +/- 13° and 79° +/- 6° respectively.
Rockmass: The database is comprised of a wide range of rockmasses collected from several Canadian operations. The values of Q' range between 1.0 to 34. The average RMR for the DLM database is 74% +/- 8%.

Modified Stability Number: Incorporating all the rockmass and correction factors together will yield the Modified Stability Number. The database is described with an average of 23 +/- 18. Again, it is useful to describe this term by the range covered in the database [0.45 - 100].

Radius Factor: The average radius factors encountered within the database is 9.5 m +/- 4.8. Also, the range of the data extends between 2.0 m and 25.5 m.

Cable Bolting: The database is sub-divided into cases with or without cable bolt support in either hangingwall or footwall. Of the total database (96 observations), 35% of the data is supported which leaves 65% unsupported. Note that the bulk of the data is from DLM (51%). The implication is that cable support must equal the standards utilized at DLM (i.e. design by the Point Anchor approach and quality control guidelines).

Hangingwall or Footwall: The database is further divided approximately into half by hangingwall (56%) and footwall (44%) data points. As expected, the majority of the hangingwall points are supported which contrast the 14% observed in the footwalls.

11.2.1. Discriminate Procedure
As suggested by the name of this statistical technique, a criteria must be defined in order to classify the database into groupings. Specific to the Modified Stability Method are five (5) zones of stability: blast induced damage, minor, moderate, and severe sloughing, and caved zone (Scoble and Moss, 1994). The formulation of these groups has been based on experience and
intuition. It is proposed that a new parameter, ELOS (Clark, 1996), be used to quantify these zones of stability.

Table 1 - "Characteristics of the Database"

<table>
<thead>
<tr>
<th>Design Parameter</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>'A' Factor</td>
<td>1.0 (state of relaxation)</td>
</tr>
<tr>
<td>'B' Factor</td>
<td>0.2 - 0.3 (parallel joint set)</td>
</tr>
<tr>
<td>'C' Factor</td>
<td>FW dip 70°-85° (use HW 'C' factor)</td>
</tr>
<tr>
<td></td>
<td>FW dip &gt; 85° (treat as HW)</td>
</tr>
<tr>
<td>Ore Width</td>
<td>μ = 6.8 m +/- 3.0 m</td>
</tr>
<tr>
<td>Stope Inclination</td>
<td>HW (μ = 74° +/- 13°)</td>
</tr>
<tr>
<td></td>
<td>FW (μ = 79° +/- 6°)</td>
</tr>
<tr>
<td>Rockmass</td>
<td>Q' = [1.0 - 34.0]</td>
</tr>
<tr>
<td></td>
<td>RMR μ = 74% +/- 8% (DLM)</td>
</tr>
<tr>
<td>Modified Stability Number</td>
<td>μ = 23 +/- 18</td>
</tr>
<tr>
<td></td>
<td>[0.45 - 100]</td>
</tr>
<tr>
<td>Radius Factor</td>
<td>μ = 9.5 m +/- 4.8</td>
</tr>
<tr>
<td></td>
<td>[2.0 - 25.5]</td>
</tr>
<tr>
<td>Cable Bolting</td>
<td>35% supported</td>
</tr>
<tr>
<td></td>
<td>65% unsupported</td>
</tr>
<tr>
<td>Hangingwall</td>
<td>56% of database</td>
</tr>
<tr>
<td>Footwall</td>
<td>44% of database</td>
</tr>
</tbody>
</table>

The selection of the appropriate ELOS values for discriminating is justified by reviewing the range of the ELOS data collected. As illustrated in Figure 11.2.1, approximately 86% of the
data is classified below 1.0 m ELOS value. This figure illustrates the skewed distribution of the ELOS values found in the database.

Initially, three (3) ELOS values are selected to define the dramatic rise in the distribution: 0.3 m, 0.5 m and 1.0 m. These values are considered to be important for narrow vein open stopes (i.e. forms the bulk of the database).

It is speculated that 0.3 m of external slough represents a reasonable level of blast damage and drillhole deviation combined.

Coincidentally, the selected ELOS value of 0.5 m parallels the acceptable dilution limit for the average narrow vein open stope found at DLM (~15% for a 4.0 m wide open stope).
And finally, the third discriminating value is selected arbitrarily at the 1.0 m ELOS value. Due to the lack of data, higher ELOS values could not be discriminated using the above statistical methods.

In summary, the ELOS values of 0.3 m, 0.5 m, and 1.0 m were selected for this analysis.

11.2.2. Database Assumptions

Plotting of the ELOS terms will utilize radius factor in place of the hydraulic radius. It is favored over the traditional shape factor for its ability to handle complicated stope geometries. A more complete explanation can be found in Section 3.4.2.

For this study, all CMS surveys are plotted with RF calculated based on the largest known survey dimension. This implies that in cases where muck is present within an open stope, the muckpile is treated as if it were providing some support to the hangingwall and footwall (i.e. exerts a passive force). The approach taken assumes that the failure mechanism responsible for generating slough is primarily due to gravity. This argument is particularly convincing for the slough material gained from the hangingwall and back. To date, the accepted mode of failure in the footwall is due to a complex relationship between the forces of gravity and friction. The physical presence of the muck prevents the wall slough from mixing with the ore until the muck level is drawn downwards. For this reason, the radius factor of the largest exposed surface is used for plotting ELOS values.

Inspection of the complete data set has identified that there were not enough data points to conduct a satisfactory multivariate analysis. With the additional data points provided, it was possible to conduct separate bivariate analyses between adjacent ELOS groupings. This non-standard procedure necessitated a unique comparative approach for interpretation. It is assumed that the discriminate functions delineated can be superimposed simultaneously.
Construction of the discriminate functions will utilize the traditional Stability Number vs. Shape Factor plot. Two intuitive assumptions are followed:

1. Slough increases for increasing radius factors
2. Slough decreases for improving rockmass.

Combining these assumptions, it can be stated that the *a priori* boundary conditions are expected to be smooth and continuous functions that cannot intersect one another. For instance, a 1.0 m ELOS slough zone cannot proceed or intersect the 0.3 m ELOS zone.

### 11.3. Discriminate Multivariate Technique

A discriminate multivariate analysis is selected for its ability to classify dependent variables into mutually exclusive groupings based on a set of independent variables. This study considers the modified stability number ($N'$) and radius factor (RF) variables to be dependent.
All the other mine operating variables collected are assumed to be independent. The proper application of this statistical analysis is based on two underlying assumptions:

1. The \( p \) independent variables must have a multivariate normal distribution.
2. The \( p \times p \) variance-covariance matrices of the independent variables in each of the groups are identical.

This method is capable of determining the probability of an observation belonging to a particular grouping through an \textit{a posteriori} transformation.

For a population \( G \), sub sets can be discriminated into \( G_n \) groupings (where \( n = 1, 2, \ldots \)) with \( X_m \) independent variables (where \( m = 1, 2, \ldots \)). It is expected that the scatter plot for each group population should resemble a hyper-elliptical shape if multivariate normality is satisfied. The objective is to define mathematically the equation of the line that separates these groupings.

\[
Y = b'X
\]

\textbf{EQ. 11.3.1-1}

where:
- \( Y \) \(- l \times n \) vector of discriminate scores
- \( b' \) \(- l \times p \) vector of discriminate weights (transposed)
- \( X \) \(- p \times n \) matrix containing the values for each of the \( n \) individuals on the \( p \) independent variables.

The discriminate score for each individual data point is obtained by multiplying the discriminate weight (\( b \)) associated with each independent variable by the data point's value on the independent variable which is then summed over the set of independent variables (Dillon and Goldstein, 1984).

In the case of the two group problem, Fisher's Approach (Fisher, 1936) is taken. This is based on finding a linear combination of \( X \) such that the ratio of the difference in the means of the linear combinations in \( G_1 \) and \( G_2 \) to its variance is maximized (Dillon and Goldstein, 1984). The method assumes that the true mean vector for \( G_i \) is \( \mu_i \) (where \( i = 1,2 \)) and the variance-
covariance matrices ($\Sigma_1$ and $\Sigma_2$) have a common value ($\Sigma$). The objective is to identify the vector of weight ($b$) for combinations of $Y = b'X$ such that the following criterion is maximized.

$$\Delta = \frac{(b'\mu_1 - b'\mu_2)^2}{b'\sum b}$$  \hspace{1cm} \text{EQ. 11.3-2}

For this study, it is assumed that the true mean vector $\mu_1$ can be replaced with an estimate mean vector $\bar{x}_1$ and $\Sigma$ can be replaced with the pooled sample variance-covariance matrix $S$.

$$\bar{x}_i' = (\bar{x}_{i1}, \bar{x}_{i2}, \ldots, \bar{x}_{ip})$$  \hspace{1cm} \text{EQ. 11.3-3}

$$S = \frac{1}{n_1 + n_2 - 2}(x'_1x_1 + x'_2x_2)$$  \hspace{1cm} \text{EQ. 11.3-4}

where $p$ - number of parameters
$n_1$ - number of observations in group 1
$n_2$ - number of observations in group 2.

Replacing these parameters with their respective sample-based estimates yields:

$$b = S^{-1}(\bar{x}_1 - \bar{x}_2)$$  \hspace{1cm} \text{EQ. 11.3-5}

where $S^{-1}$ - inverse of the pooled sample variance-covariance matrix.

To summarize, the discriminate function becomes a linear composite of the original measurements on which the sum of squared differences between group means is maximal relative to the within-groups variance.
Graphical representation of the boundary separating the groupings is defined as a linear function. In the case of a bivariate analysis, the discriminate function can be defined by the point at which the univariate distributions intersect. This point can be projected onto the scatter plot as a line which coincidentally intersects the two points where the two ellipses overlap. (Figure 11.3.1)

![Figure 11.3.1 - "Two-Group Discriminate Analysis"

11.3.1. Testing for Normality

To test the assumption of normality, several graphical methods are utilized.

**Scatter Plot:** The simple scatter plot demonstrates the overall shape of the database. For normally distributed data, it is expected that the scatter plot will resemble a hyper-elliptical shape (Figure 11.3.2).
Comparing Figure 11.3.2 to Figure 11.3.3 indicates that the database collected does not satisfy the initial test for normality.

It follows that more rigorous statistical test methods will have be conducted to verify normality.

**QQ Plots:** Under the recommendation of SCARL, QQ plots were used to test for univariate normality of the dependent variables $N'$ and RF individually. This method compares the quantiles of the standard normal distribution with the standardized values of a variable. Data is standardized by arranging the values of a particular variable into ascending order. The test
states that the condition of normality is satisfied if the plot of standardized values align with the standard normal distribution values (i.e. a straight line).

![Stability vs Radius Factor](image)

**Figure 11.3.3 - "Scatter Plot of Stability vs. Radius Factor"

It was found that both plots of $N'$ and RF indicate that normality was not easily obtained (Figure 11.3.4 and Figure 11.3.5) particularly at the extremities of the database.
Figure 11.3.4 - "QQ-Plot of the Stability Variable"

Figure 11.3.5 - "QQ-Plot of the Radius Factor Variable"
A mathematical transformation was required to modify the data set in order to appear 'more' normal.

Figure 11.3.6 - "Square Root Transformation of the QQ-Plot on the Stability Variable"

The QQ plot of the N' variable indicated that the square root transformation provided the best results.
Figure 11.3.7 - "Square Root Transformation of the QQ-Plot on the Radius Factor Variable"

Figure 11.3.8 - "Logarithmic Transformation of the QQ-Plot on the Radius Factor Variable"
Interestingly, the QQ plots for the RF variable showed no preference to either the square root or natural logarithmic transformation (Figure 11.3.7 and Figure 11.3.8). Both plots are similar in appearance.

Overall, the univariate test for normality was conditionally satisfied for the square root transformation.

**QQ Chi Square Plots:** To test for bivariate normality, the QQ Chi Square plot was utilized for the dependent variables $N^1$ and RF. Appropriately, the Chi Square test is used for its ability to handle multiple population proportions. In this case, the distance measured between an observation and its population mean is plotted against the corresponding Chi Square value. Standardizing the distance and Chi Square values in the same manner as the QQ plot method yields a similar straight line analysis.

It is evident from Figure 11.3.9 that a mathematical transformation is required to satisfy normality.
Interestingly, the QQ plots for the RF variable showed no preference to either the square root or natural logarithmic transformation (Figure 11.3.7 and Figure 11.3.8). Both plots are similar in appearance.

Overall, the univariate test for normality was conditionally satisfied for the square root transformation.

**QQ Chi Square Plots:** To test for bivariate normality, the QQ Chi Square plot was utilized for the dependent variables N' and RF. Appropriately, the Chi Square test is used for its ability to handle multiple population proportions. In this case, the distance measured between an observation and its population mean is plotted against the corresponding Chi Square value. Standardizing the distance and Chi Square values in the same manner as the QQ plot method yields a similar straight line analysis.

It is evident from Figure 11.3.9 that a mathematical transformation is required to satisfy normality.
Figure 11.3.9 - "QQ-Chi Square Plot of Bivariate Normality (Stability and Radius Factor)"

Figure 11.3.10 - "Square Root Transformation of the QQ-Chi Square Plot (Stability and Radius Factor)"
The bivariate test results indicate that the square root transformation demonstrates the closest approximation to normality (Figure 11.3.10). This finding is in general agreement with the univariate QQ test. In summary, the tests conducted indicate that the initial assumption of multivariate normality is sufficiently satisfied when the square root transformation is applied\(^2\).

11.3.2. Effectiveness of Discrimination

Essentially, all data points can be classified as either greater or smaller than the \textit{a priori} ELOS standards. In the bivariate case, there are two groups formed. The discriminate analysis is tested

\(^2\) Validation of normality is commonly accomplished through visual inspections. It is acknowledged that there are other methods for assessing normality. In particular, there are 'goodness of fit' tests that assess the degree of normality. However, its universal application is not well accepted in the statistical field since they are inherently biased. Therefore, the accepted statistical standard for interpretation utilizes primarily visual assessments of normality.
for efficiency by determining the distance that separates the centroids between groups. In this case, the centroids are the sample means of the multivariate normal distributions for each grouping. The software package provided by SCARL calculates the pair-wise generalized square distance (Mahalanobis's generalized distance) for each case considered.

$$\bar{Y}_1 - \bar{Y}_2 = D^2 = (\bar{x}_1 - \bar{x}_2)'S^{-1}(\bar{x}_1 - \bar{x}_2)$$  \hspace{1cm} \text{EQ. 11.3.2-1}

Intuitively, the greater the distance between centroids, the more pronounced is the effect of discriminating.

Further to this, a formal test can be constructed to determine if the Mahalanobis's generalized distance is statistically significant. Since the method utilizes the differences in sample means, the hypothesis statement can be stated as $H_0: \mu_1 = \mu_2$. The test consists of calculating the following test statistic.

$$Z = \frac{n_1n_2}{n_1 + n_2} \frac{n_1 + n_2 - p - 1}{(n_1 + n_2 - 2)} D^2$$  \hspace{1cm} \text{EQ. 11.3.2-2}

It is recognized that this test statistic $Z$ is distributed as an F-distribution with $p$ and $n_1 + n_2 - p - 1$ degrees of freedom. Using the above relationship, $H_0$ is rejected at the significance level ($\alpha$) if the $Z$ value is greater than the corresponding F-distribution value. This implies that the two populations formulated do not have a similar sample mean which suggests that the populations are sufficiently separated by the discriminate function.

Computing the Mahalanobis generalized distance for the \textit{a priori} using the whole database is summarized in Table 2. These values computed will formulate the baseline values for further comparisons.
Table 2 - "Baseline Case of Mahalanobis Generalized Distance (All Data)"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Generalized Distance</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>1.3</td>
</tr>
<tr>
<td>0.5 m</td>
<td>2.8</td>
</tr>
<tr>
<td>1.0 m</td>
<td>2.3</td>
</tr>
</tbody>
</table>

The database is now separated into groups of hangingwall and footwall or supported and unsupported subsets. These groups reflect the underlying assumption that hangingwalls are more unstable than footwalls due to the effects of gravity. For the supported case, the addition of cable bolts is expected to increase wall stability.

Table 3 - "Hangingwall Case of Mahalanobis Generalized Distance"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Generalized Distance</th>
<th>$\Delta$ in Generalized Distance wrt Baseline</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>1.5</td>
<td>+ 0.2</td>
</tr>
<tr>
<td>0.5 m</td>
<td>4.3</td>
<td>+ 1.5</td>
</tr>
<tr>
<td>1.0 m</td>
<td>4.0</td>
<td>+ 1.7</td>
</tr>
</tbody>
</table>

Table 4 - "Footwall Mahalanobis Generalized Distance"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Generalized Distance</th>
<th>$\Delta$ in Generalized Distance wrt Baseline</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>1.3</td>
<td>0.0</td>
</tr>
<tr>
<td>0.5 m</td>
<td>1.8</td>
<td>+ 1.0</td>
</tr>
<tr>
<td>1.0 m</td>
<td>1.3$^3$</td>
<td>- 1.0</td>
</tr>
</tbody>
</table>

By direct comparison with the baseline case, the hangingwall results indicate an improvement over the footwall case.

3 In all cases but one, the test confirmed that the Mahalanobis's generalized distances calculated were statistically significant. The one exception was the 1.0 m ELOS footwall case which failed because of a lack of data points.
<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Generalized Distance</th>
<th>Δ in Generalized Distance wrt Baseline</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>2.0</td>
<td>+ 0.7</td>
</tr>
<tr>
<td>0.5 m</td>
<td>4.1</td>
<td>+ 1.3</td>
</tr>
<tr>
<td>1.0 m</td>
<td>3.7</td>
<td>+ 1.4</td>
</tr>
</tbody>
</table>

Table 5 - "Supported Mahalanobis Generalized Distance"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Generalized Distance</th>
<th>Δ in Generalized Distance wrt Baseline</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>1.4</td>
<td>+ 0.1</td>
</tr>
<tr>
<td>0.5 m</td>
<td>2.7</td>
<td>- 0.1</td>
</tr>
<tr>
<td>1.0 m</td>
<td>1.9</td>
<td>- 0.4</td>
</tr>
</tbody>
</table>

Table 6 - "Unsupported Mahalanobis Generalized Distance"

In a similar fashion, the supported results were shown to be more effective than the baseline or unsupported case.

Combining these findings, a further analysis was conducted with only hangingwall and supported data points. The results were inconclusive due to the lack of data points in this subset.

Overall, this 'Z' significance test (EQ. 11.3.2-2) indicate that there is an improvement to the effectiveness of the discriminate analysis if the database is separated into groups of:

- Hangingwall or Footwall
- Supported or Unsupported.

11.3.3. Misclassification Rates

The most critical measure of discrimination is the estimated error rate expressed as the percentage of misclassification. Probabilities are assigned to an observation as to whether it has been properly classified through *a posteriori* transformation. Equal priors are utilized instead of proportional priors because they provide more reasonable results.
There are several methods described in the literature for calculating the estimated error rate. However, only two methods are applicable to this study: the re-substitution method and cross-validation method. It is well publicized in the statistical field that the re-substitution method optimistically biases the results. In an effort to counter this inherent flaw in methodology, the cross-validation method is designed to eliminate all biases. It would appear that the cross-validation method is the obvious method of choice. However, it does produce highly variable results. The advantages of reducing the bias is not enough to overcome the associated deviation. Therefore, the re-substitution method is recommended for interpreting the results.

A series of tests was completed to determine the optimal linear discriminate function for the database.

Case #1: Once again a baseline case is utilized for direct comparisons analysis. The results are as follows:

Table 7 -" All Data Discriminate Functions"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Equation</th>
<th>Misclassification Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>$\sqrt{RF} = 0.70 + 0.56\sqrt{N}$</td>
<td>23.4%</td>
</tr>
<tr>
<td>0.5 m</td>
<td>$\sqrt{RF} = -1.14 + 1.16\sqrt{N}$</td>
<td>14.5%</td>
</tr>
<tr>
<td>1.0 m</td>
<td>$\sqrt{RF} = -3.20 + 1.84\sqrt{N}$</td>
<td>23.1%</td>
</tr>
</tbody>
</table>

Observe that the construction of the discriminate functions were found to converge to a common interception point (located approximately at a Modified Stability Number of ten with a corresponding Radius Factor of six). Based on the assumptions stated in Section 11.2.2, the construction of ELOS lines must obey the following boundary conditions:

1. Smooth and continuous
2. Cannot intersect one another.

Therefore, the discriminate functions are corrected to 'better' reflect the data points (Figure 11.3.12).
Case #2: In Section 4, previous mapping has defined similar rockmass ratings and joint structures for both the hangingwall and footwall. Regardless of this observation, the database was separated into these two categories to determine if the ELOS measurements will identify an inherent difference.

It was also found that the construction of the discriminate functions converged at approximately the same point on the Modified Stability plot as in Case #1. The curves were corrected in accordance to the stated assumptions.

Figure 11.3.12 - "All Data Discriminate Functions"
Table 8 - "Hangingwall Discriminate Functions"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Equation</th>
<th>Misclassification Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>$\sqrt{RF} = -0.64 + 0.86N'$</td>
<td>22.0%</td>
</tr>
<tr>
<td>0.5 m</td>
<td>$\sqrt{RF} = -4.16 + 1.95N'$</td>
<td>12.4%</td>
</tr>
<tr>
<td>1.0 m</td>
<td>$\sqrt{RF} = -3.33 + 1.74N'$</td>
<td>13.8%</td>
</tr>
</tbody>
</table>

MODIFIED STABILITY GRAPH
Hangingwall Data (54 obs)

Figure 11.3.13 - "Hangingwall Dataset Discriminate Functions"

The 0.3 m and 0.5 m ELOS lines have shifted down with respect to the baseline curves (Figure 11.3.13). The hangingwall 1.0 m ELOS line does not appear to have changed position with respect to the baseline.

Table 9 - "Footwall Discriminate Functions"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Equation</th>
<th>Misclassification Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>$\sqrt{RF} = 1.51 + 0.39\sqrt{N}$</td>
<td>28.6%</td>
</tr>
<tr>
<td>0.5 m</td>
<td>$\sqrt{RF} = 0.68 + 0.60\sqrt{N}$</td>
<td>26.1%</td>
</tr>
<tr>
<td>1.0 m</td>
<td>$\sqrt{RF} = 2.63 + 0.20\sqrt{N}$</td>
<td>14.6%</td>
</tr>
</tbody>
</table>

MODIFIED STABILITY GRAPH

Footwall Data (42 obs)

Figure 11.3.14 - "Footwall Dataset Discriminate Functions"

4 Note: The FW results for the 1.0 m ELOS function is based on two data points that are greater or equal to 1.0 m ELOS.
Analysis of the footwall data set indicates that the 0.3 m and 0.5 m discriminate functions (Table 9) plot slightly higher than the baseline and hangingwall (Figure 11.3.15). The implication of this observation is that the hangingwall is more stable than the footwall. In all likelihood, this observation contradicts common sense. This discrepancy is interpreted as the inadequacy of the ‘C’ factor to account for footwall stability. Figure 11.3.14 depicts a large degree of scatter thus making it difficult to discern an observable pattern or trend in the ELOS values. It is concluded that the footwall points require re-plotting in accordance to the ‘C’ factor curve developed for the hangingwall. Re-plotting the data also provides no additional information.

![Modified Stability Graph](image)

Figure 11.3.15- "Comparison of Hangingwall and Footwall 0.3 m ELOS"

---

5 The footwall data collected at DLM are plotted using the ‘C’ factor based on friction (Y. Potvin, 1988).
6 Correcting the footwall data with the HW ‘C’ factor still produces a plot with a large degree of scatter. It is observed that the footwall slough ranges consistently between 0.0-0.6 m and averages 0.3 m ELOS.
It is concluded that the footwall results are disregarded for the following reasons:

1. Unequal distribution of footwall data (consists predominantly of unsupported points)
2. Lack of data points in the >1.0 m range
3. Higher misclassification rates than the hangingwall.

Treatment of the footwall slough must therefore be different from that of the hangingwall. For footwalls that dip between 70° and 85° and are characterized with a parallel joint set, it is recommended that the average value of 0.3 m ELOS be applied towards the total stope dilution. In the case where footwalls dip > 85°, essentially, there is no difference between either hangingwall or footwall. Prediction of footwall slough can be accomplished by using the ELOS design curves developed for the hangingwall.

As a temporary guideline, it is suggested that additional footwall data points (dip > 70°) be collected for plotting on the Modified Stability Method using the HW ‘C’ factor. Until sufficient evidence proves otherwise, all footwall data points that are < 70° are excluded from the analysis.

Case #3: The final case attempts to discriminate between the supported and unsupported data sets. It is anticipated that the ELOS curves will provide some evidence of support performance.

Table 10 - "Support Discriminate Functions"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Equation</th>
<th>Misclassification Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>$\sqrt{RF} = -1.99 + 134\sqrt{N}$</td>
<td>24.1%</td>
</tr>
<tr>
<td>0.5 m</td>
<td>$\sqrt{RF} = -7.43 + 3.16\sqrt{N}$</td>
<td>11.0%</td>
</tr>
<tr>
<td>1.0 m</td>
<td>$\sqrt{RF} = -4.60 + 2.35\sqrt{N}$</td>
<td>13.7%</td>
</tr>
</tbody>
</table>

7 The 70° has been selected as the transition point for footwalls that behave like a hangingwall based on the current CMS database.
For the respective *a priori* groups, the resultant curves of the supported data set demonstrate a downward shift as referenced to the baseline (Figure 11.3.16). The immediate interpretation suggests that stability is improved as the result of the installed ground support.

**MODIFIED STABILITY GRAPH**

In terms of the misclassification errors, the supported curves can be considered an improvement over the baseline.

Figure 11.3.16 - "Supported Dataset Discriminate Functions"
Table 11 - "Unsupported Discriminate Functions"

<table>
<thead>
<tr>
<th>ELOS Function</th>
<th>Equation</th>
<th>Misclassification Error</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.3 m</td>
<td>$\sqrt{RF} = 1.61 + 0.32\sqrt{N}$</td>
<td>23.9%</td>
</tr>
<tr>
<td>0.5 m</td>
<td>$\sqrt{RF} = 0.58 + 0.65\sqrt{N}$</td>
<td>12.7%</td>
</tr>
<tr>
<td>1.0 m</td>
<td>$\sqrt{RF} = -0.63 + 1.04\sqrt{N}$</td>
<td>21.8%</td>
</tr>
</tbody>
</table>

As expected, the unsupported data set provided a similar finding. In contrast, the unsupported curves plotted much higher than the baseline. This observation implies that the unsupported data is more unstable than the supported data which satisfies intuitive logic.

In general, the supported results are favored over the unsupported data set.
11.4. **DISCUSSION AND INTERPRETATION**

It is understood that the discriminate analysis is an exploratory method for determining a solution for complex problems. It is applied on a one time basis to investigate the interdependence of variables that influence the observations (Tabachnick and Fidell, 1987).

In conducting the normality tests, a degree of database variability has been observed primarily at the extremes. This observation corresponds with open stopes that plot in the upper right and lower left quadrant of the *Modified Stability* plot. This suggests that the stopes designed as either small openings in weak ground conditions or large openings in good ground conditions.
behave differently from the majority of the data collected thus far. Extrapolation of the discriminate functions into these zones for design is not recommended.

Confined by the database, a bivariate discriminate analysis is used to determine the ELOS boundaries. Interpretation requires a unique comparative approach that superimposes the discriminate functions regardless of the fact that a 'true' multivariate solution was not computed.

Of the discriminate functions delineated, two criteria were used to select the optimal solution. The first criteria provided a measure of the effectiveness of discrimination: Mahalanobis's generalized distance values. Through direct comparisons and tests for significance, it was found that the hangingwall and supported data set were significantly better than the footwall and unsupported data set respectively.

The marked difference in hangingwall and footwall results highlights the difference in failure mechanisms. In the Modified Stability Method, the calculation of the modified stability number includes three (3) correction factors; namely A, B, and C. For all stopes observed in the database, the 'A' factor was set at 1.0 which represents a relaxed stope wall condition. Similarly, the 'B' factor did not deviate much from a value of 0.2 or 0.3. However, it was found that the 'C' factor is more likely to change for each design.

The original intent of the design method excluded the importance of the footwall. Recent amendments to the methodology has introduced a new set of 'C' factor values specifically for footwalls (Potvin, 1988). It is questionable whether the 'C' correction factors adequately represent the footwall. Current research conducted at the University of British Columbia describes the inadequacy of the 'C' factor (Clark and Pakalnis, 1997). The argument suggests that hangingwalls and footwalls are expected to behave differently in terms of stability. Initial inspection of this statement makes such strong intuitive sense that it is often assumed to be without exception. However, an inherent flaw has been revealed in the
weighting of the two sets of correction factors. To illustrate the error, determine the 'C' factor for a hangingwall and footwall dipping at 90°. Intuitively, both walls are vertical and hence should be considered equal. However, the footwall curve assigns a weighting of C = 2.0 which contradicts the hangingwall curve weighting of C = 8.0. At this point, a flaw has been identified. However, further refinement as suggested by Hadjigeorgiou et al (1995) is considered beyond the scope of this study and hence not pursued. It is recommended that the analysis be continued with the assumption that if the footwall dips between 70° and 85°, the footwall can be treated as a hangingwall ('C' factor > 6.0) if parallel structure exists.

It can be stated that the use of support provides a means of separating the database to provide a better discriminate analysis. Figure 11.4.1 depicts the positive trend that cable bolts have on improving wall stability. However, this analysis does not comment on the actual performance of the support itself. Instead, the interpretation is that the installation of cable bolts help the wall to behave in a more predictable manner. In a sense, the cable bolts assist to control sloughing by making it more predictable.

The second criterion uses misclassification rates to assess discriminate performance. By definition, low misclassification rates are allocated a higher degree of confidence. Application of this criteria allowed for the elimination of the footwall based on the direct comparison with the baseline and hangingwall results. Similarly, the supported is favored over the unsupported.

Of the data collected, it is apparent that for stopes designed below a RF of 6, performance based on ELOS varied throughout a wide range of Stability Numbers. The few data points that exist in this lower range of RF do not justify extrapolation into this region for design.

---

8 Assume the same parallel dipping structure for both hangingwall and footwall.
11.5. **Development of a Design Curve**

The foundation of the work completed is to be based on the premise that statistics are to be used as a tool to aid in the development of a practical design curve for the underground hardrock mining industry. One of the prime objectives is to develop a design curve that can be easily integrated with the existing Modified Stability Method. Therefore, the findings of this statistical exercise are to be combined with engineering judgment in order to formulate the final interpreted design curve.
Contrary to the recommendations made in Section 11.4, the unsupported discriminate functions will be utilized instead of the supported. The unsupported case is preferred for the initial phase of stope design since it provides the most economical and more conservative approach to design.

As stated in Section 11.2.1, the a priori discriminated consisted of three zones: 0.3 m, 0.5 m and 1.0 m ELOS. These values were selected to reflect the distribution of the database. However, in the author’s opinion, these zones need to be adjusted to represent more practical scalar values of wall slough.

In particular, the usefulness of the 0.5 m ELOS line is debatable. The relevance of differentiating between 0.3 m and 0.5 m is questionable in the overall computation of stope dilution. From the ELOS curves computed for the hangingwall data set, observe that all the discriminate functions exhibit a common shape and slope. From Figure 11.3.13 we observe that the zone that separates the 0.5 m and 1.0 m ELOS curves is barely discernible. Intuitively, we expect a more gradual transition to occur between zones. Therefore, based on this observed discrepancy, it is recommended that the 0.5 m ELOS line be removed from this analysis.

To replace the loss of an ELOS boundary, it is recommended that a 2.0 m ELOS line be included in the analysis. Due to a lack of data points in this range, it was derived based on a visual estimate. In summary, the refined ELOS zones are:

- **< 0.3 m ELOS**: Blast Induced and Drillhole Deviation
- **0.3 - 1.0 m ELOS**: Minor Slough
- **1.0 - 2.0 m ELOS**: Moderate Slough
- **> 2.0 m ELOS**: Severe Slough.

The 'Blast Damage and Drillhole Deviation' zone represents the amount of slough (< 0.3 m ELOS) attributed from drilling and blasting. Realistically, mining in this manner will produce a certain amount of slough. This guideline is deemed an acceptable level of wall slough as attributed by these factors. Note that the separation of the two effects was not
possible given the information collected in this study. Therefore, these two effects were combined into one zone.

The 'Minor Slough' zone represents the area between 0.3 m - 1.0 m ELOS. It is speculated that the slough occurring in this range can be controlled through adjusting mine operating factors (i.e. control blasting, drilling accuracy, etc.).

The 'Moderate Slough' zone is defined between the 1.0 m - 2.0 m ELOS lines.

And finally, the 'Severe Slough' zone defines a wide range of slough values that are > 2.0 m ELOS. Ideally, it is hoped that this zone can be further refined to include additional increments of 1.0 m ELOS to ultimately define the 'Caved' zone. The current database is limited by the lack of data points that fall within this category.

Figure 11.5.2 is the final interpreted unsupported, hangingwall design curves. The portion of the ELOS contours that are of the highest degree of confidence are depicted as solid lines (i.e. 0.3 m ELOS line). Extrapolation or adjustment of these ELOS curves are represented by a dashed line. It follows that with additional data in the 1.0 m and 2.0 m ELOS range, the existing ELOS lines may be altered accordingly.

Also, for certain cases, the treatment of footwalls must be designed separately.
MODIFIED STABILITY GRAPH

Final Design Curve (54 obs)

Stability Number ($N'$)

Radius Factor (m)

- HW Cable Supported
- HW Unsupported


Figure 11.5.2 - “Interpreted Unsupported, HW Design Curves”
11.6. **Design Curve Summary**

Various tests were performed to verify the normality of the database. It was found that the CMS database collected is sufficiently normal if the square root transformation is applied to the data.

Use of the Mahalanobis's generalized distance and misclassification rates proved to be a suitable method for determining the significance of each discriminate function. With these indicators, it was identified that two data sub-sets provided better results: hangingwall vs. footwall and supported vs. unsupported. The hangingwall and supported data sets yielded more significant results than the footwall and unsupported data sets respectively.

By combining statistical techniques with engineering judgment, discriminate functions were determined for the hangingwall and unsupported case (0.3 m, 1.0 m and 2.0 m ELOS). Design curves are created by superimposing these discriminate functions onto the *Modified Stability Method*. The incorporation of both ELOS boundaries and Nickson's cable bolt curve (1992) provides a new approach to open stope design (Figure 11.6.1 and Figure 11.6.2).
MODIFIED STABILITY GRAPH
All Data (54 obs)

Radius Factor (m)

Stability Number (N')

Figure 11.6.1 - “Superimposition of S. Nickson’s Curves”

MODIFIED STABILITY GRAPH

Final Design Curve (Based on 54 obs)

Drillhole Deviation
Blast Damage
Minor Slough

0.3 m ELOS

0.3 m

1.0 m ELOS

1.0 m

2.0 m ELOS

2.0 m

Unsupported
Stable
Support
Transition Zone
Severe Slough

DATABASE:
A Factor = 1.0 (HW in 'relaxation')
B Factor = 0.2-0.3 (Critical Joint Set - Parallel wrt Slope)
HW C Factor ONLY
FW treated as HW if dip > 85°
Rackmass Q' ranges between [1.0 - 34.0]
Critical Threshold: Blasting Vibration Below
Drillhole Deviation < 7%


Figure 11.6.2 - “Final Design Curves”
12. CONCLUSIONS

The objective of this thesis was to present a reliable estimate of dilution that can be readily used for evaluating open stope stability. A methodology was co-developed for determining unplanned dilution incurred from open stope mining methods. This was achieved through the use of a new laser instrument called the Cavity Monitoring System (CMS). The CMS is designed to measure the profiles of underground openings. Analysis of this information allows the user to identify several sources of unplanned dilution in the form of wall slough. It is proposed that unplanned dilution be estimated with a term Equivalent Linear Over-break Slough (ELOS). Superimposition of these ELOS values onto the Modified Stability Method enables one to quantify the zones of stability. An empirical database (96 observations) was collected primarily at the Detour Lake Mine, Placer Dome Canada. Other contributions to the CMS database were made from various Canadian underground operations.

It was found that the CMS data provides evidence of several wall slough mechanisms (Table 1).

Table 1 - "Sources of Dilution"

<table>
<thead>
<tr>
<th>INHERENT</th>
<th>INDUCED</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rockmass quality</td>
<td>Exposed Wall Surface</td>
</tr>
<tr>
<td>Geologic Structure (i.e. wedges)</td>
<td>Blast Damage</td>
</tr>
<tr>
<td>‘Over-hangs’ (‘under-cutting’ and ‘over-cutting’)</td>
<td>Drillhole Deviation (i.e. collaring)</td>
</tr>
<tr>
<td>Stope Geometry (i.e. ‘dog-leg’ )</td>
<td>Time (i.e. excavation rate, idle)</td>
</tr>
<tr>
<td>Drillhole Deviation (i.e. joint sets)</td>
<td></td>
</tr>
<tr>
<td>Stress.</td>
<td></td>
</tr>
</tbody>
</table>

Intuitively, the advantage gained from identifying the primary source of stope dilution is the ability to assign the proper remedial measure.
12.1. **Trial Stope (660-885 Captive)**

Analysis of the CMS database collected at DLM has enabled stope performance to be evaluated. The trial stope, in terms of design, was a success for several reasons. The average dilution measured in this stope was 5.5%. In part, the implementation of a cable support program contributed towards controlling the wall slough (reduced dilution by 3-5%)\(^1\). Also, the placement of 'yielding' pillars allowed the hangingwall span dimension to be optimized for DLM. The consequence of exceeding the designed span resulted in the Fall of Ground incident that occurred at the final extraction of Panels ‘A’ and ‘B’. Analysis of the CMS sections indicated that the cable bolts maintained the integrity of the hangingwall since the failure surface extended past the cable bolt length. Two isolated rib pillars were left behind to prevent the same type of ground failure from occurring in Panels ‘C’ and ‘D’. In doing so, the recovery for this stope was reduced to 80% (from the original design of 84%). Reconciling the mucked tonnage records indicates that the trial stope was over-drawn. If the excess tonnage is assumed to be comprised entirely of dilution, then the dilution can be estimated at approximately 9.0%\(^2\).

In summary, the trial stope provided valuable information regarding stope design. Several different cable bolt geometries were tested in the trial stope. The assessment indicated no particular preference to cable configuration in terms of stope performance. Overall, it can be said that cable bolt support enhances hangingwall stability.

For the trial stope, dilution and the captive nature of the mining method were considered the critical design and operating parameters. Albeit the fact that dilution was minimal, two major ground control incidents were encountered which resulted in lost ore and over-sized pieces. It was found through back analysis of these incidents that design curves could be developed and

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\(^1\) Refer to Section 7.1.

\(^2\) The reliability of reconciled mucked tonnage figures is questionable due to the inherent flaw in the present incentive system which promotes 'paper tonnes'.

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
calibrated with CMS data. Application of these design curves will assist the mine planner in regulating unplanned dilution.

12.1.1. Modified Stability Method

This thesis proposes two revisions to the Modified Stability Graph that are intended to enhance the method's ability to predict stope stability in terms of unplanned dilution values. Utilization of the radius factor in place of the hydraulic radius provides a more accurate description of the exposed wall surface. Also, the zones of stability are quantified with unplanned dilution terms (ELOS). Three ELOS contour lines have been delineated using statistical techniques in conjunction with engineering judgment for the unsupported, hangingwall database. These lines are defined as follows:

<table>
<thead>
<tr>
<th>Blast Damage and Drillhole Deviation</th>
<th>(&lt;0.3) m ELOS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Minor Slough</td>
<td>0.3 - 1.0 m ELOS</td>
</tr>
<tr>
<td>Moderate Slough</td>
<td>1.0 - 2.0 m ELOS</td>
</tr>
<tr>
<td>Severe Slough</td>
<td>&gt;2.0 m ELOS</td>
</tr>
</tbody>
</table>

It was found that the effects of the drilling and blasting could not be separated. A procedure was developed for converting drillhole deviation into an equivalent value of ELOS. The drilling study estimated an average error of 0.3 m ELOS for DLM. Further discussion on the effects of blast damage is found in Section 12.1.3.

As discussed in Section 11.4, the 'C' factor has been identified as an unresolved issue that is recommended for future research. This study finds the 'C' factor, based on sliding (Potvin, 1988), to be inadequate for correcting footwall stability. To date, there are not enough data points to support the use of this curve for design. Instead, the footwall is treated as a hangingwall if the dip is greater than 70° (HW 'C' factor > 6.0). This guideline was selected to reflect the majority of the data points collected in the CMS database (Figure 12.1.1).
Re-plotting the footwall data in this manner still did not provide a definitive trend in ELOS values (i.e. a large scatter). Inspection of the CMS database has identified the following distribution of footwall ELOS values.

Figure 12.1.2 - "FW ELOS Distribution"
From Figure 12.1.2, it is clear that the footwall slough values ranged between 0.0 - 0.6 m ELOS with an average value of 0.3 m ELOS. This simplifying assumption is considered to be valid for the DLM operation. Until the discrepancies of the 'C' factor have been properly addressed, this study proposes to compute footwall dilution by adding the average amount of footwall slough towards total stope dilution (for cases where wall dip is between 70° and 85° and with a parallel joint set). It can be argued that hangingwalls and footwalls that dip greater than 85° are essentially vertical walls. Therefore, the footwall slough is predicted as per the ELOS relationship developed for the hangingwall.

12.1.2. Development of a Pillar Failure Curve

A numerical model (3-D) was constructed to simulate the mining sequence for the trial stope. Excavations were constructed on a monthly time interval. Several extensometer and strain cell instruments were installed to assist in calibrating the numerical model. It was found that this data did not provide the means for successful calibration. Therefore, the magnitude of stress could not be determined from this study.

Instead, another approach was utilized to develop a pillar failure curve. It was found that the change in stress as determined by the numerical model could be calibrated against the observed pillar degradation. This approach simulates a standard triaxial compression test. By this analogy, pillar failure can be predicted for a critical combination of stresses. It is hypothesized that open stope stability is related to the condition of the sill and rib pillars. At DLM, pillar stability is defined as follows:

- **Stable** (0.0-0.3 m ELOS)
- **Unstable - Minor** (0.3-1.0 m ELOS)
- **Unstable - Moderate** (1.0-2.0 m ELOS)
- **Yielding** (>2.0 m ELOS)

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
The pillar failure curves are presented in two formats: 'Stress Signature' and standard Mohr-Coulomb plots. Complementing these curves with CMS data provide the initial calibration. To date, only two ground control incidents have been back analyzed. It is acknowledged that the numerical codes are limited to modeling homogeneous, isotropic and linearly elastic materials. By calibrating the model to enough CMS data, the in situ conditions (i.e. effect of jointing and blast damage) can be reasonably accounted for in the development of a reliable design tool. The findings so far suggest that the Mohr-Coulomb design curve is capable of predicting inevitable pillar failure.

12.1.3. Development of a Blast Damage Criteria

A blast monitoring program was conducted to collect (PPV) vibration data from several longhole blasts. The scaled distance approach was utilized to formulate a standard regression line. Limitations associated with the scaled distance approach are well documented in the literature. For example, the analysis assumes that only one charge per delay can be considered at a time. This implies that the effect of cap scatter is negligible which is not always the case. However, since the monitoring data is comprised of small blasts\(^3\), the effect of cap scatter was observed to be insignificant. Also, this study finds the scaled distance relationship is unable to account for the effect of drillhole proximity. It can be argued that for blastholes with the same delay located in close proximity of each other will inherently induce more blast damage than blastholes that are separated by greater spacing. Again, this effect is limited since only small blasts were monitored.

Regardless of these inherent limitations, the scaled distance analysis of the DLM data has provided a high correlation coefficient \((r = 0.81)\). The analysis of this 'far-field' model has

\(^3\) A small blast is defined when: the number of holes < the number of delay detonators available and/or drillholes re spaced wide enough for each blast event to behave independently of the other (Blair, 1990).
indicated a potential for extrapolating into the 'near-field' range for design. Calibration of the blast damage criteria was enabled with the CMS data. The ELOS values superimposed onto the blast damage criterion included both hangingwall and footwall wall slough.

At DLM, the critical particle velocity computed for the in situ rockmass is estimated at 2 800 mm/s. This value appears to support extrapolating the scaled distance relationship into the 'near-field' range since wall slough increases with increasing PPV. Interpretation and extrapolation into the 'near-field' range is enabled by assuming that blast damage to the stope walls can be predicted by using the distance measured from the lead blasthole to the stope wall. This assumption is justified since all blasts designed at DLM utilize a 'v-cut' formation. Therefore, the highest degree of confinement experienced by the stope wall is attributed to the blasthole initiated at the center of the blast. Intuitively, the perimeter blastholes are expected to produce minimal blast damage since they are usually 'traced' and perform more of a 'slashing' function.

Application of this design curve will enable the user to predict the amount of blast damage associated for a number of holes per delay. At DLM, blast damage is controlled (<0.3 m ELOS) by designing below the critical particle threshold.

Limited by the number of available data points, the Blast Damage Criteria is restricted to radius factors of 11.0 +/- 1.0 in order to demonstrate an intuitive trend. With only six surveys, there is an apparent trend that indicates increasing blast damage with increasing PPV. It follows that additional case studies are required to improve upon calibrating the design curve.

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4 Slashing is when the blasthole is detonated into a relatively larger opening in comparison to the initial blasthole. Hence, the bulk of the energy is released quicker as a result of the reduced burden.
12.2. DESIGN EXAMPLE

In order to demonstrate the applicability of the design curves, an example stope is provided.

An ore block has been identified in the upper portion of the Detour Lake Mine. It is to be mined as a captive longhole. The objective is to design an economical and feasible stope dimension with the predicted dilution. The following are the design constraints:

1. Design for no cable bolts
2. Accelerated Development
3. Captive Equipment (2 yd scoop, Alimak service raise)
4. 260 mL mucking horizon (8 yd scoop).

12.2.1. Stope Geometry

The 260 Q100/4 (Figure 12.2.1) extends from 120 mL to 260 mL (~140 m in height). A hangingwall Alimak raise has been established to provide central stope access. Six sub-levels have been developed (140, 160, 185, 200, 220 and 240) over an average strike length of 190 m (E-W). The orebody is steeply dipping (85°) quartz zone with an average mineable width of 3.0 m. The host rock is comprised of mafic flows (estimated UCS = 165 MPa).

Geological mapping has identified two major joint sets. To the west of the access, the hangingwall and footwall is defined by the typical parallel (JSA) and perpendicular (JSB) joint set found in other parts of the mine. However, to the east of the access two joint sets that form small wedge features are predominant (JSC and JSD).
12.2.2. Yielding Pillar Design

Historically, stress in this area has not been a problem. Refer to Chapter 8 and the Appendix III for additional information on the DLM stress regime. Sill pillar design has determined a thickness of 4.0 m to separate the 260 Q100/4 from the above open stope (Carter, 1990). Rib pillar dimensions are estimated from the ‘Confinement Formula’ (Lunder, 1994) to be 4.5 m (on strike) and 3.0 m (HW to FW).

The placement of these pillars will assist in breaking up the hangingwall and footwall span.
12.2.3. Span Design

Computing a rockmass rating yields the following results:

<table>
<thead>
<tr>
<th>Rock Strength Factor</th>
<th>R4</th>
<th>12</th>
</tr>
</thead>
<tbody>
<tr>
<td>RQD Factor</td>
<td>80-90%</td>
<td>18</td>
</tr>
<tr>
<td>Joint Spacing Factor</td>
<td>1-2 m</td>
<td>22</td>
</tr>
<tr>
<td>Joint Condition Factor</td>
<td>hard, tight, smooth</td>
<td>20</td>
</tr>
<tr>
<td>Groundwater Factor</td>
<td>dry</td>
<td>10</td>
</tr>
</tbody>
</table>

**average RMR = 80-85%**

| RQD Factor | 80-90% | 80-90 |
| Joint Set Number | 2 joint sets | 4 |
| Joint Roughness Number | Roughness smooth | 1 |
|                     | Waviness planar | 1 |
| Joint Alteration Number | none | 0.75 |

**average Q' = 25-30**

The resulting Stability Number ranges from 30-40. Based on these rockmass values, the optimal radius factor was determine to be 12.0. However, historical experience gained in mining adjacent stopes in this area proved that radius factors of 18.0 could be feasible. Without this previous experience, the design proposed by the Modified Stability Method would have been far too conservative.

Numerous iterations were computed for various sill and rib pillar configurations. Consideration was given to the cost associated with leaving ore behind, slot-raise development and equipment productivity. The advantages and disadvantages were reviewed to produce the final stope design (Figure 12.2.2).
12.2.4. Dilution Estimate

From Figure 11.6.2, the estimated hangingwall slough plots between the 0.3 and 1.0 m ELOS contour line. Interpolation between the ELOS contour lines produces a low estimate of dilution (0.3-0.4 m ELOS) since the open stope configuration plots closer to the 0.3 m ELOS contour line. By combining the geological mapping information and past experience, a value of 0.6 m ELOS is produced for the hangingwall design. Application of this design curve is valid if drilling is kept within 7% drillhole deviation and blasting PPV is restricted below the critical particle velocity threshold.

For footwalls that dip between 70° - 85°, an average footwall wall slough value of 0.3 m ELOS is applied towards calculating stope dilution. However, the stope walls found in the 260 Q100/4...
dip steeply (> 85°). Essentially, the footwall and hangingwall can be treated equally. Therefore, the total wall slough predicted is ~1.2 m ELOS. In a stope that is only 3.0 m, this amount of wall slough equates to 40% dilution. The relatively high value of dilution emphasizes the sensitivity of wall slough to narrow ore bodies.

Given the higher grade associated with this ore block and historical precedence, the amount of wall slough predicted from mining the 260 Q100/4 stope is thought to be reasonable and acceptable at DLM.

This design example highlights the fact that the design curve is to be used in conjunction with engineering judgment. Design based solely upon the curves provides a ‘rough’ estimate of dilution. However, if application of the design curve is coupled with engineering judgment, a more reliable dilution estimate can be produced. Design of open stope dimensions are restricted to the characteristics of the database used to derive the ELOS relationships.

12.3. FUTURE WORK

The findings of this thesis include the co-development of a methodology to estimate dilution based on CMS data and the derivation and calibration of three design curves: Modified Stability Method, Blast Damage Criteria and Pillar Failure Curve.

Open Stope Design: Two revisions have been proposed for the Modified Stability Method that enable the user to predict hangingwall slough with a higher degree of confidence (i.e. based on low misclassification rates). The design curves are valid for open stopes that are characterized by relaxed stress conditions ('A' factor equal to 1.0) and a parallel joint set ('B' factor ranging between 0.2 - 0.3).
Determination of footwall slough, however, is much more complicated than as described in the literature. The footwall observations that comprise the CMS database, in general, do not deviate much from the average value of 0.3 m ELOS. Further refinement of the ‘C’ factor is required to ‘better’ reflect the observations collected in the database.

The derivation of the 0.3 m and the 1.0 m ELOS design curves are based on the hangingwall, unsupported data set. Minor adjustments were required to modify the trends at the end points to match the surrounding data points. In a similar manner, the 2.0 m ELOS line was placed onto the Modified Stability Method with a ‘best-fit’ approach. The findings of this thesis indicate that wall slough is predictable within the confines of the methodology presented. It follows that there is merit in collecting additional CMS data points to further refine these ELOS contours. Ultimately, additional ELOS contours (i.e. 3.0 m, 4.0 m, 5.0 m and so on) can be developed given enough data points.

Application of these curves for designing supported stopes is considered conservative for stopes with Stability Numbers greater than 10. For Stability Numbers below 10, the stable with support zone does not appear to be as valid. Design in this range is not dependent on the size of exposed wall surface. Instead, the rockmass quality is the key design parameter. Therefore, it is recommended that mine operators determine, on an individual stope basis, if a certain amount of wall slough is economically acceptable.

In addition, a blast damage criteria and pillar failure curve were developed and calibrated with CMS data.

**Blast Damage Criteria:** Blast damage is observed to be minimal (< 0.3 m ELOS)\(^5\) for the DLM operation. The initial calibration of the scaled distance curve with CMS data is limited to only seven (7) blasts monitored. The effects of exposed wall surface and drillhole

\(^5\) It is suggested that the zone of predicted blast damage ranges between 0.3 - 0.7 m of wall slough (Singh, 1992).
deviation could not be separated from blast damage exclusively given the few data points collected. In order to demonstrate a trend, the blast damage criteria was restricted to a finite range of radius factors (11 +/- 1). This thesis study found that wall slough increased with increasing PPV. Intuitively, this observation is considered valid for all ranges of radius factors. However, it is suspect that the magnitude of wall slough might differ for individual stope sizes. Therefore, it is recommended that additional case histories be back analyzed to confirm this statement.

Also, a complementary study was conducted to measure the degree of drilling error. The findings of this study estimates drillhole deviation with an equivalent 0.3 m ELOS over the entire length of drillhole. It is assumed that the effects of drilling and blast damage can be grouped together in this analysis.

Future work is recommended to further develop the blast damage criteria to include a wider range of radius factors and to quantify drillhole deviation apart from blasting. Also, efforts to control and regulate drillhole deviation is possible with the current technology. Justification must weigh the cost required to improve drillhole deviation to the benefits gained.

**Pillar Failure Curve:** The pillar failure curve was developed to determine the critical stress combination that will induce a ‘yielding’ pillar condition. At DLM, the anticipation of a ‘yielding’ pillar allows for appropriate ground control measures to be installed (i.e. steel straps in combination with cable bolts). Maintaining the integrity of the pillar affects the overall stope stability by preventing the hangingwall span from exceeding the strength of the rockmass. The difficulty remains in predicting the useful life of a ‘yielding’ pillar design. As more case histories are collected, the effect of time can be assessed for pillar degradation.
Equipped with these calibrated design curves, the mine planner is better able to predict the behaviour of the rockmass. It follows that optimisation studies can be conducted to improve safety, reduce costs and increase production.

12.4. Final Comments

It has been found that the CMS instrument provides the most useful information for assessing stope performance. This information is key for calculating the amount of unplanned dilution incurred from mining. Dilution impacts negatively on a mining operation by reducing the overall milling grade and delaying the mine cycle. In the extreme case, over-sized muck has the potential for prematurely ending the stope life by plugging up the drawpoints. As reported in various studies, many operations today continue to base their mine designs on experience and intuition. This thesis suggests that a certain degree of human error can be eliminated from the design process if engineering judgment is coupled with reliable CMS data. By nature of empirical approaches to design, the process of calibration is an on-going procedure. Therefore, the collection of additional CMS surveys is recommended to provide further validity to the current design curves.

In the author’s opinion, the acquisition of a CMS instrument for use at a Canadian hardrock underground mine is the first step towards optimizing various aspects of mine design. It provides three-dimensional data that represents a ‘snap-shot’ image of the in situ stope wall condition. Collecting enough CMS surveys to document a complete stope history provides the valuable information required to help take out the ‘guess-work’ in mine design and thereby enabling one to ‘engineer’ the workplace.
REFERENCES


34. Lizotte, Y. 1990. "Blasthole Stoping For Narrow Vein Mining". 92nd CIM-AGM, Ottawa


Quantification and Prediction of Wall Slough in Open Stope Mining Methods


APPENDIX I

I. CMS SURVEYS (DLM)

The CMS database is comprised of data collected primarily at DLM. In total, 49 observations were collected from three longhole stopes. This portion of the database is partitioned in the following manner:

- 660-885 Captive (I-a)
- SLR (I-b)
- 935 Q120 (I-c).

The remaining data has been obtained from various Canadian underground operations. A limited description of these data points can be found in Appendix II.
APPENDIX I (a)

660-885 CAPTIVE STOPE

Panel ‘A’

In October 1994, the first production blast was taken in Panel ‘A’. The two CMS surveys\(^1\) were collected shortly after completion in April of 1995.

<table>
<thead>
<tr>
<th>Survey</th>
<th>Date</th>
</tr>
</thead>
<tbody>
<tr>
<td>CC</td>
<td>April 25, 1995</td>
</tr>
<tr>
<td>CF</td>
<td>May 11, 1995</td>
</tr>
</tbody>
</table>

CC (April 25, 1995): CMS survey CC was taken to assess the condition of the stope. In particular, the sill and dividing stope pillars were of concern since they were designed to be yielding pillars.

At this time, the panel is half full of broken ore material. From the longitudinal section, the condition of the sill and dividing (SLR and 660-885 Captive) stope pillars can be described as stable and intact. The upper portion of the survey confirms that the dumped upholes are breaking to the toes.

Overall the ELOS values for the hangingwall and footwall are 0.2 m and 0.4 m respectively. The lower slough value measured on the hangingwall is misleading. Instead, a lost ore measurement (ELOS = 1.5 m) is indicative of fan drillhole performance.

CF (May 11, 1995): To complement the previous survey (CC), CMS survey CF was collected shortly afterwards to assess the effect of drawing down the ore level.

As mining progressed, the stability of the dividing rib pillars was in doubt. Failure of the dividing rib pillar could result in excessive amounts of dilution (unconsolidated backfill in the adjacent SLR stope). Therefore an extra ring was taken off the blast to minimize the effects of blast damage. From survey CC and CF, signs of degradation were observed.

Computing the ELOS values for the stope walls showed an incremental increase in wall slough: HW = 0.3 m and FW = 0.5 m.

\(^1\) All surveys were collected using the ‘automatic survey’ feature of the instrument. The full range of elevation (0-135°) is utilized with a one (1) degree elevation step. Surveying in the instrument is always accomplished by the ‘boom method’ (CMS Manual, 1995).
Figure 0.1 - "CMS Survey CC"
Figure 0.2 - "CMS Survey CF"

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
Panel ‘B’

Mining in this panel began in December 1994 and was completed in June 1995. In total, nine (9) CMS surveys were collected:

- CA (January 17, 1995): CMS survey CA captures the slot blast, taken to full orebody width, on 610 mL. In this case, the drop-raise was drilled along the hangingwall contact.
  
  Similar ELOS values were computed for both walls: 0.3 m (HW) and 0.2 m (FW). It was expected that the hangingwall would yield a larger amount of wall slough as the result of high confinement blasting based on the location of the drop-raise.

- CB (June 17, 1995): After completion of the 630-650 block, CMS survey CB was taken on 630 mL. It was collected early in the production stages of Panel ‘B’ to provide a reference for assessing the performance of the stope.
  
  Analysis indicated that the hangingwall was controlled by local structural failures (wedges) occurring near the collar of the block (Figure 0.3). The corresponding ELOS values are 0.4 m (HW) and 0.2 m (FW).

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All surveys were collected using the ‘automatic survey’ feature of the instrument. The full range of elevation (0-135°) is utilized with a one (1) degree elevation step. Surveying in the instrument is always accomplished by the ‘boom method’ (CMS Manual, 1995).
CD (April 25, 1995): CMS survey CD was collected on 585 mL to confirm reports from mine operations of short or missing drillholes for subsequent blasts. Initial speculation assumed excessive blast damage as the problem. However, this area has a span of ~17.0 m with a corresponding RMR rating of 65-70%. Utilizing the "Stability Graph for Entry Type Excavations", we find that this sub-level plots in the potentially unstable zone (B. Lang, 1991). Re-assessment of the rockmass observed the presence of flat joints. Correcting the RMR for flat joints lowers it to 55-60% which plots the sub-level in the unstable region.
It was confirmed that drillholes located in drift center were broken back to a maximum arch height of **10.0 m**. Figure 0.4 depicts a typical section taken two rings back from the brow. The cause of this ground failure is speculated to be due to higher blast damage associated with initiating the lead blastholes (to form the V-shape) in combination with a weaker rockmass. Therefore, the critical span was exceeded for that rockmass strength (R. Pakalnis, 1995).

Wall slough was measured at 0.1 m and 0.3 m for the hangingwall and footwall respectively.

**CE (April 26, 1995):** Taken from the sub-level below (610 mL), CMS survey CE was designed to complement the information gained by CD.

```
Ring 81

565 mL

565 mL

565 mL

Lost ore

610 mL

Muck-Pile

0.0 5.0 m
```

**Figure 0.5 - “CMS Survey CE”**

Combining the data from CD yields an ELOS value of 0.1 m (HW), 0.6 m (FW). The increase in footwall slough is due to better visibility from the lower vantage point. A higher degree of confidence is given to this CMS survey.

**CG (June 17, 1995):** As mining progressed to the 545 mL, CMS survey CG is taken to assess the performance of Panel ‘B’. The dumped up-holes are measured to be breaking to full
depth (sill pillar is intact). The rib pillars separating Panel ‘A’ and ‘B’ are observed to be stable and intact with minor wear on the brow of the 565-585 pillar.

Minor wall slough is observed in this panel: ELOS values of 0.4 m (HW) and 0.2 m (FW).

CH (July 15, 1995): CMS survey CH was taken on 565 mL after firing the last production blast scheduled for this panel. Based on the previous CMS survey collected, the unplanned dilution has been consistently measured to be minor in this panel: ELOS value of 0.5 m (HW) and 0.3 m (FW).

At this point, the condition of the sill and rib pillars located in the upper portion of the stope are beginning to show signs of degradation (Figure 0.6).

Figure 0.6 - "Initial Degradation of the Sill and Rib Pillars (CH)"
CI (July 29, 1995): To further complement CMS surveys CG and CH, CI was taken on 565 mL to measure the effects of time. The sill and rib pillar have sloughed by ~3.0 m (Figure 0.7).

During this time period, the hangingwall appears to be stable (ELOS = 0.5 m). However, the footwall shows a slight increase (ELOS = 0.5 m) in wall slough.

CJ (August 23, 1995): During the night shift on August 7, 1995, a large fall of ground event occurred in the upper portion of 660-885 Captive stope spanning Panels ‘A’ and ‘B’ (K. 224

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
Dunne et al, 1996). An estimated ~70,000 tonnes of hangingwall material was displaced suddenly to create an air blast. Damage was minimal and confined to equipment located near the man-way landing. CMS survey CJ was taken on 565 mL to define the outer limits of this ground failure: ELOS values for the hangingwall (7.1 m) and footwall (0.6 m) are based on previous information (Figure 0.8).

Note that the failure surface extends beyond the cables' supporting ability. It is speculated that the cables were successful in holding the rockmass together (R. Pakalnis, 1996). However, the driving force instigating the fall of ground was the continual deterioration of the rib pillar separating Panels ‘A’ and ‘B’. Earlier geotechnical mapping (S. Mah, 1995) had indicated that the area surrounding the rib pillar was assessed a lower rockmass quality (RMR = 60%). Confined to production demands, the rib pillar (545-530) was not reinforced with cable bolts.

The effect of removing the rib pillars increased the exposed surface of hangingwall (designed RF = 16, actual RF = 25). Plotting the stope wall on the Modified Stability Method places the design well into the caved zone. The final stope dimension appears to have stabilized along the JSA structure (/\ joint set).
CO (January 19, 1996): To monitor the stability of the FOG, CMS survey CO was collected on 545 mL. Additional wall sloughing was minimized by introducing unconsolidated waste material from an above drop-raise. Analysis of this survey indicates that the hangingwall is ‘working’ towards an equilibrium state (ELOS = 8.5 m). The footwall has stabilized at an ELOS value of 0.6 m.

Figure 0.9 - "CMS Survey CA"
Figure 0.10 - "CMS Survey CB"
Figure 0.11 - "CMS Survey CE"
Figure 0.12 - "CMS Survey CG"
Figure 0.13 - "CMS Survey CJ"
Production blasting started in August 1995 and was completed by December. In this panel, four CMS surveys were collected:

- CK (November 15, 1995)
- CL (November 27, 1995)
- CP (January 27, 1996)
- CQ (January 29, 1996).

**CK (November 15, 1995):** As a result of the FOG incident, an isolated rib pillar was left in mid panel span to minimize the exposed hangingwall surface. CMS survey CK was collected on 610 mL to assess the bottom portion of the stope.

On this level, a 'dog-leg' stope geometry is evident (Figure 0.14). Associated with this type of stope geometry is the potential for a substantial FOG failing on parallel geologic structures. However, in this case, the absence of detrimentally oriented structures results in only minor amounts of wall slough: HW ELOS = 0.1 m, FW ELOS = 0.2 m.

![Figure 0.14 - "Dog-Leg Feature in CMS Survey CK (Panel 'D')"](image)

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3 All surveys were collected using the 'automatic survey' feature of the instrument. The full range of elevation (0-135°) is utilized with a one (1) degree elevation step. Surveying in the instrument is always accomplished by the 'boom method' (CMS Manual, 1995).
CL  (November 27, 1995):  CMS survey CL was taken on 565 mL located just above the isolated rib pillar. At this point, only a small portion of the upper panel has been blasted. Wall slough is observed to be negligible: HW and FW = 0.1 m ELOS.

Figure 0.15 - "CMS Survey CK"
CP (January 27, 1996): Analysis of CMS survey CP was not performed because the data is incomplete. The survey routine was interrupted as a result of an inadequate set up location.

CQ (January 29, 1995): Upon completion of this panel, CMS survey CQ was collected to assess the performance of the upper panel. The effect of the isolated rib pillar is to reduce the RF from 16 to 14. It is expected that for this relatively smaller wall exposure will yield a minimal amount of wall slough. Reducing the survey results in a hangingwall ELOS of 0.3 m and a footwall ELOS of 0.2 m.

Figure 0.16 - "CMS Survey CL"
Figure 0.17 - "CMS Survey CQ"
Panel 'C'

This final mining sequence consisted of blasting Panel ‘C’ which commenced in September 1995 and was terminated in January of 1996. Four (4) CMS surveys were collected to describe this area:

<table>
<thead>
<tr>
<th>Survey</th>
<th>Date</th>
</tr>
</thead>
<tbody>
<tr>
<td>CM</td>
<td>(December 13, 1995)</td>
</tr>
<tr>
<td>CN</td>
<td>(December 18, 1995)</td>
</tr>
<tr>
<td>CR</td>
<td>(January 29, 1996)</td>
</tr>
<tr>
<td>CS</td>
<td>(February 28, 1996)</td>
</tr>
</tbody>
</table>

**CM (December 13, 1995):** At this point, mining had progressed to the 585-610 block. For the majority of the 660-885 Captive, blast sequencing was primarily achieved with one active mining face for each panel. However, restricted by a central access system, two active mining faces were created in Panel ‘C’ to retreat towards the access cross-cut. The effect was to create a diminishing pillar situation which induced high levels of stress. Ground problems were experienced in the form of 'floor heave' and flat joints. The buckling ground conditions resulted in large pieces of over-sized ore material and an unsafe working situation.

Reduction of this survey provided low ELOS values for the hangingwall (0.0 m) and footwall (0.2 m).

**CN (December 18, 1995):** Understandably, concerns for worker safety required that CMS survey CN be taken to assess the stability of the stope. The results indicated that the stope was stable since neither walls exhibited any observable increase in slough.

**CR (January 29, 1996):** Shortly after the final blast, this survey was collected from 585 mL. The resulting ELOS values are 0.2 m and 0.5 m for the hangingwall and footwall respectively.

In general, the hangingwall showed equal amounts of slough and lost ore measurements. The larger footwall ELOS is primarily from an ‘under-cutting’ mechanism. Sub-level development is ideally designed to full ore widths. However, in actuality, sill development can often slash further than is required to form ‘under-cuts’ to the above sub-level. From experience, the rockmass succumbs to the forces of gravity and tends to slough along structure.

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4 All surveys were collected using the ‘automatic survey’ feature of the instrument. The full range of elevation (0-135°) is utilized with a one (1) degree elevation step. Surveying in the instrument is always accomplished by the ‘boom method’ (CMS Manual, 1995).
Figure 0.18 - "CMS Survey CM"
CS (February 28, 1996): This is the final survey taken for this stope. Analysis of this survey has yielded an ELOS value of 0.2 m and 0.7 m for the hangingwall and footwall respectively. In general, the stope is stable.

Figure 0.19 - "CMS Survey CS"
APPENDIX I (b)

SUB-LEVEL RETREAT STOPE

For simplicity the SLR is divided into three sections: east, central and west.

Eastern Division

AB (Nov. 30, 1993): CMS survey AB is the first of four taken on the east side of the SLR. It captures the profile of a ‘benched’ blast estimated to represent $9 500 in lost tonnes (200 tonnes @ SG = 2.9, 4.7 g/t, gold @ $350 US/oz) Figure 0.1.

ELOS values for the hangingwall and footwall yield 0.1 m and 0.4 m respectively. Minimal hangingwall slough is attributed to good blasting practice where parallel holes are utilized. However, sections defined by fan drillholes left ore behind (0.8 m ELOS). From the survey, the footwall slough is related to structure and ‘over-cutting and under-cutting’ from the above sub-level.

Figure 0.1 - "CMS Survey AB (SLR)"
AG (June 28, 1994): CMS survey AG was collected on 590 mL to assess stope performance (Figure 0.3). It identifies poor ground conditions in the Talc material and possible blasting problems.

Low ELOS values (0.0 m) are measured in the hangingwall are not indicative of dilution problems. Instead, they highlight blasting problems (cut-offs or misfires) resulting in lost ore (~2.0 - 3.0 m). Larger ELOS values associated with the footwall (1.8 m) are attributed to the footwall fault.

Figure 0.3 - "CMS Survey AG (SLR)"
Figure 0.4 - "Over-Sized Material Due to Footwall Fault"
Al (June 28, 1994): CMS survey Al was taken from the sub-level below to complement the data collected in AG. The values for wall slough are in agreement with the previous survey.

AF (January 18, 1995): This section of the stope has now been inactive for approximately six months. The void is currently being filled with unconsolidated waste material deposited from a drop raise connecting 560 mL to 575 mL.

ELOS measurements show no additional slough material gained on both walls.

Central Division

AD (January 17, 1995): This survey was conducted on 630 mL to capture a benched blast. The lost ore was estimated at $41 000 (920 tonnes, SG=2.9, 4.5 g/t, gold @ $350 US/oz). The wall slough observed is negligible: both HW and FW = 0.1 m.
AE (January 18, 1995): CMS survey AE was collected from the above 610 mL to complement AD. A 'benched' blast was also observed and estimated to represent $25 000 (550 tonnes, SG=2.9, gold @ $350 US/oz) in lost ore. In this upper region of the stope, much greater wall slough is measured: HW = 0.3 m and FW = 0.4 m.

Figure 0.6 - "CMS Survey AE"

Western Division

AC (Nov. 30, 1993): CMS survey AC was taken on 610 mL to provide a baseline for dilution analysis. At the time, the sill pillar showed signs of degradation near the brow. Wall conditions were stable: ELOS values for the hangingwall and footwall are 0.1 m and 0.4 m.
respectively. The larger wall slough measured on the footwall is due to a localized wedge failure near the 610 mL.
AH (June 28, 1994): CMS survey AH was taken to assess the effect of changing the mining sequence. Increasing incidents of ‘over-sized’ material were delaying mine production. Rescheduling the blasting sequence to form a more vertical mining face (from a staggered 15 m echelon shaped mining front) was hoped to provide greater flexibility in handling the ore. If mining could retreat quickly enough, the over-sized pieces associated with wall slough could be left behind in the stope.

All the sections indicate minimal wall slough: ELOS values of 0.1 m in the hangingwall and 0.4 m in the footwall. The higher dilution observed in the footwall is accredited to a localized wall failure.

It is concluded that the change in mine sequence did not negatively affect wall stability.

AA (June 8, 1995): CMS survey AA was taken approximately one year after AH to assess the condition of the sill pillar and to measure the effect of time on wall stability. The sill pillar showed signs of progressive failure. Wall conditions were degrading: HW = 0.9 m and FW = 1.9 m.
Figure 0.8 - "CMS Survey AH"
Figure 0.9 - "CMS Survey AA"
APPENDIX I (c)

935 Q120 STOPE

Blast F6 (June 11): CMS survey F6 is the first of a series of seven taken on 480 mL. It captures the slot blast taken to full orebody width.

The ELOS values for the hangingwall and footwall are 0.0 m and 0.1 m respectively. Note that the drop raise was driven on the footwall side resulting in higher confinement blasting which may explain the slightly higher ELOS value.

Figure 0.1 - "CMS Survey F6"

Blast F7 (June 12): This CMS survey was collected shortly after the subsequent production blast. Analysis of the survey provided an ELOS value of 0.1 m (HW) and 0.2 m (FW).
Blast F8 (June 15): Due to manpower constraints, CMS survey F8 was surveyed using the 'mast method'. The ELOS values for the hangingwall and footwall are 0.0 m and 0.2 m respectively. Sections indicate good blasting practice that resulted in lowering the overall ELOS value for the hangingwall side.

A moderate degree of confidence is given to this CMS survey since the surface mesh becomes irregular. Using the information gained from previous surveys and the fact that the stope is regular in geometry, linear interpolation was used to complete the mesh where possible.

Blast F9 (June 18): CMS survey F9 was taken shortly after the next production blast. Analysis of the survey provided an ELOS value of 0.3 m (HW) and 0.1 m (FW). Linear interpolation was used again to complete the mesh since it was fragmented.
Blast F10 (June 19): Blast F10 yielded ELOS values of 0.4 m and 0.1 m for the hangingwall and footwall respectively.

Blast F11 (June 27): The next blast was taken a week later. The corresponding hangingwall and footwall ELOS values are 0.5 m and 0.2 m. Ventilation conditions improved to provide an adequate mesh continuity.

Blast F12 (August 8): Mine production was delayed on the second lift for a period of 30 days while mining began on the third lift. Only three blasts were taken during this time since both upheoles and downholes were blasted resulting in a larger tonnage per blast. The final blast on the second lift is captured by CMS survey F12.

The calculated ELOS values for the hangingwall and footwall are 0.5 m and 0.4 m respectively.
TRUE SECTION ON REFERENCE AZIMUTH 260°00'00"
June 27, 1995

TRUE SECTION ON REFERENCE AZIMUTH 260°00'00"
August 8, 1995
APPENDIX II

CMS DATABASE

In total, 96 data points has been accumulated from various Canadian underground operations to include a diverse range of stope sizes and rockmass qualities. This Appendix II summarizes the information associated for each data point.
## Detour Lake Mine DATABASE

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Quantification and Prediction of Wall Slough in Open Stope Mining Methods
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Quantification and Prediction of Wall Slough in Open Stope Mining Methods

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**Quantification and Prediction of Wall Slough in Open Stoping Mining Methods**


256
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## COMPLEMENTARY DATABASE

### Complementary Database (FW)

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<th>RF</th>
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</tr>
</tbody>
</table>
APPENDIX III

NUMERICAL MODEL INPUT PARAMETERS

Input Parameters

The application of numerical modeling is anticipated to resolve the stress redistribution within the host rockmass as a result of mining. At this stage, only the elastic response is of interest. Predictions of rockmass deformation will be cross referenced with extensometer data for calibration.

In building the numerical model, certain constraint parameters need to be defined for the program. These parameters are defined in an input file (*.inp) written in ASCII format or a *.DXF (AutoCAD v12) file. The program requirements consist of the following components:

1. Project Title
2. Control Parameters
3. Building Blocks
4. Coordinate Specification
5. Material Characteristics
6. Grid Definition

Project Title

The project title is included to provide a description of the file and its purpose. It is limited to a maximum of 70 characters.

Control Parameters

The very basic control parameters utilized by the software program consists of:

Nld (Number of Load Steps)

Nld defines the maximum number of load steps allowed to obtain a converged elasto-visco-plastic solution. It has a default value of 10 000 steps. If convergence is not obtained within the number specified, the program will fail.

Nit (Number of Iterations)

Nit is defined as the maximum number of Gauss-Seidel iterations allowed to obtain a converged solution for a load step. It has the same default value as the Nld parameter. Similarly, if the iterations exceed this number, the program will fail.
Nps (Number of Planes of Symmetry)
Nps enables the program to make some simplifications to the calculations for situations where planes of symmetry exist. The default assumes no symmetry.

Rpar (Relaxation Parameter)
Rpar is a parameter that enables the program to obtain convergence in as few iterations as possible. This convergence is dependent on the 'conditioning' of the problem. A well conditioned problem is defined by bulky shapes or surfaces that are spaced apart and are distinct. For this type of problem, the maximum relaxation parameter is set to 1.2. Conversely, 0.8 is used for a poorly conditioned problem.

Stol (Stress Tolerance)
During Gauss-Seidel iterations, convergence is assumed to be complete when the largest error is less than this value. It is recommended that 0.1% of the far field stress magnitude be selected for accuracy.

Al (Allowable Element Side Length)
The automated discretization routine requires a numerical value for the minimum allowable element side length. It is suggested to reflect the smallest stope or pillar width used in the model.

Ag (Allowable Grid Side Length)
Ag is the minimum allowable grid side length to be used in the discretization routine. As a rule of thumb, the Al parameter should be less than or equal to the Ag parameter.

In general, the following settings were used for this case study.

Table 1 - “DLM Control Parameter Settings”

<table>
<thead>
<tr>
<th>Nld</th>
<th>Nit</th>
<th>Nps</th>
<th>Rpar</th>
<th>Stol</th>
<th>Al</th>
<th>Ag</th>
</tr>
</thead>
<tbody>
<tr>
<td>100</td>
<td>200</td>
<td>0</td>
<td>0.8</td>
<td>1.0</td>
<td>1.5</td>
<td>1.5</td>
</tr>
</tbody>
</table>

Nld and Nit are set to their recommended values. Nps is made zero since there are no planes of symmetry. The Rpar is selected at 0.8 to represent a poorly conditioned model. The recommended Stol values are quite small which increases the computational effort. Therefore, a decision was made to increase this value thus improving the run times. Al is determined by the smallest stope dimension. In this case, 5.0 m is the minimum width of the ore. However, a greater degree of accuracy was required and the Al was set to 1.5 m. Ag is selected to equal the Al value as recommended.

Model accuracy is varied by a simplification process referred to as the ‘lumping’ process. It allows the user to solve larger problems that otherwise would have been restricted by memory and computational effort. The basis for this simplification assumes the effects of entities, as distance increases with respect to the point of interest, to have less and less of an influence with
respect to the overall stress regime. Therefore, the model calculates with greater accuracy near the point of interest. Accuracy is controlled by six optional parameters that are stated after the control parameters:

**Dol (Distance over Grid Length)**
Dol controls the way in which grid and block surfaces are subdivided. Large numerical values of Dol imply a finer discretization. It follows that the degree of accuracy required is related to both Dol and Ag. Ag is selected to reflect the distance from the point of interest to the grid.

**Don (Distance over Element Length)**
Don controls the way block surfaces are discretized into boundary elements. The higher the Don value, the finer the boundary elements produced. In conjunction with the Al parameter, these two parameters ensure that a solvable coefficient matrix will be created for problems that are not well conditioned.

**Doc (Distance over Coefficient Lumping Length)**
Doc controls the way in which elements are lumped during matrix assembly. For calculation purposes, the combined side length of the discretized elements located a distance from the point of interest are lumped if they are less than the Doc ratio. Therefore, increasing the Doc ratio will increase accuracy.

**Doe (Distance over Element Lumping Length)**
Doe controls the way in which elements are lumped during field point calculations. Selection of this parameter requires a compromise between accuracy and smooth stress distributions to be considered. For example, the stress regime near an excavation surface is more erratic. Increasing the value of Doe helps to smooth out the stress distribution calculation while maintaining accuracy.

**Dog (Distance over Grid Lumping Length)**
Dog controls the way in which field points are lumped during field point calculations. Similar to the Doe parameter, larger Dog values ensure a smooth stress distribution.

**Dor (Maximum Allowable Element Aspect Ratio)**
The automated discretization routine uses the Dor ratio to decide whether an element requires subdividing. For instance, if the side length of an element is larger than Dor, it is subdivided. Increased accuracy can be obtained by increasing Dor.

**Table 2 - “DLM Accuracy Settings”**

<table>
<thead>
<tr>
<th>Dol</th>
<th>Don</th>
<th>Doc</th>
<th>Doe</th>
<th>Dog</th>
<th>Dor</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.0</td>
<td>1.0</td>
<td>2.0</td>
<td>2.0</td>
<td>2.0</td>
<td>5.0</td>
</tr>
</tbody>
</table>
The overall level of accuracy desired is set for the ‘detailed analysis’ result. Default values for the six accuracy settings will be used. The only change is to the Don factor set (at 1.0) to account for a poorly conditioned model.

**Building Blocks**

The program requires that the geometry of the mine be described in terms of simple blocks (maximum of 8,000 blocks). These blocks are entered in a counter-clockwise fashion from top to bottom as eight points or incidents (\(I_1, I_2, I_3, I_4, I_5, I_6, I_7, I_8\)). The six surfaces are described as either boundaries for excavations or displacement discontinuities.

**Co-ordinate Specification**

For each block incident there is a corresponding Easting, Northing and elevation (maximum of 8,000 incidents). These must be specified in the order given.

**Material Characteristics**

A material number is assigned to each material type (maximum of 100) encountered. The number 1 is reserved for the host rockmass. Each material is defined by three properties:

1. Stress State
2. Constitutive Properties

**Stress State:** It is well documented that a rockmass is subject to an in situ stress state incurred from genesis. Mining activity disturbs and re-distributes this stress regime. An estimate of the pre-mining stress conditions is required for modeling.

‘Map-3D’ inputs the magnitude of principal stresses in the order of major, intermediate and minor (\(\sigma_1, \sigma_2, \sigma_3\)) taken at an arbitrary depth (Surf). At depth, these stresses increase and by convention are represented as negative values (\(d\sigma_1, d\sigma_2, d\sigma_3\)).

From previous studies, the following stress magnitude and direction has been determined by an over-coring strain relief technique utilizing triaxial cells (Arjang, 1991).

**Table 3 - “DLM Host Rockmass Gradient - Stress State”**

<table>
<thead>
<tr>
<th>(\Delta) Major (\sigma) (MPa/m)</th>
<th>(\Delta) Intermediate (\sigma) (MPa/m)</th>
<th>(\Delta) Minor (\sigma) (MPa/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>-0.0754 (0° - E-W)</td>
<td>-0.0377 (90° - N-S)</td>
<td>-0.0290 (0° - vert.)</td>
</tr>
</tbody>
</table>

This tensor measurement was taken in the (Abitibi Belt) northern parts of the Superior Tectonic Province of the Canadian Shield. Characteristic features of this large expanse of...
rock are the easterly trending, alternating volcanic-plutonic and metasedimentary-gneiss belts (Figure 0.1). A long-standing stress pattern has been observed that trends north-south in compression and east-west in tension horizontally (Goodwin and Ridler, 1970). However, for this case study the tensor value quoted in Table 3 opposes this well documented stress regime. The major principal direction of compression is, in fact, trending in the east-west direction. Since the orebody also trends in this general direction, the major principal horizontal stress is considered to be favorably oriented in terms of stope stability.

This change in stress regime could be a result of the formation of a localized stress domain. The presence of a major fault or anisotropic rock properties can act as a barrier to stress. Other stress observations made in northeastern Ontario and Quebec were found to be comparable in direction and magnitude (Herget, 1990).

**Constitutive Properties:** The behavior of the rock material is entered in terms of several quantifiable measures. For the non-linear response, the peak value is used for the first load step. Once yielding occurs, the residual value becomes more appropriate.

Laboratory and in-situ testing of the intact rock was not conducted in this project. Estimates of rock strength and elasticity were obtained from previous testwork discussed in Section 6.1.5.

From this data, we observe the difficulty in obtaining consistent laboratory results. Applying the admissibility limits for conventional isotropic analysis (EQ 1 and 2), we find that the host and ore material essentially behave as one material (Worotnicki, 1995).

\[
\frac{E_{\text{max}}}{E_{\text{min}}} < 1.3 \quad \text{(admissible)} \quad \text{EQ. 1}
\]

\[
\frac{E_{\text{max}}}{E_{\text{min}}} > 1.5 \quad \text{(inadmissible)} \quad \text{EQ. 2}
\]

---

1 The results of this work is assumed to resemble the rock material currently being mined. Hence, the rock properties can be extrapolated based on continuity of ore genesis.

2 Note that a wide variation in range exists for the unconfined compressive strength (UCS) parameter whereas Young's Modulus and Poisson's ratio show relatively consistent results. Terraprobe and J.D. Smith values appear to differ significantly from the UBC and CANMET testwork and hence were not considered.
In addition, cohesion and the tensile strength values are omitted from the calculation. It is assumed that they do not contribute significantly to the overall stability of the open stope.

Selection of the required design parameters is intended to error on the conservative side. Therefore, the weaker rock material, in this case the ore, is used for numerical modeling input (Table 4). This assumption satisfies model homogeneity.

3 The footwall material has been slightly altered and is made considerably weak by the presence of a prominent fault. Mapping has confirmed that much of the fault is filled with a talc gouge material in areas where the fault has been ‘day-lighted’. Diamond drilling indicates that the fault diverges into the footwall away from the open stope starting from the bottom (660 mL) to the top (545 mL).
Table 4 - “Design Rockmass Material Properties”

<table>
<thead>
<tr>
<th>Young’s Modulus (MPa)</th>
<th>Poisson’s Ratio (#)</th>
<th>UCS (MPa)</th>
<th>Tensile (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>93 000</td>
<td>0.26</td>
<td>165.0</td>
<td>35-40</td>
</tr>
</tbody>
</table>

Strength Characteristics: In determining the displacements associated with mining, a material failure criteria must be selected. Depending on which criteria is chosen, appropriate variables must be specified. For this study, the Hoek and Brown failure criterion (EQ. 3) has been selected.

\[
\sigma_1 = \sigma_3 + \sqrt{m \sigma_c \sigma_3 + s \sigma_c^2}
\]  

EQ. 3

where

- \(\sigma_1\) - major principal stress (MPa)
- \(\sigma_3\) - minor principal stress (MPa)
- \(\sigma_c\) - uniaxial compressive strength of intact rock (MPa)
- \(m, s\) - material constants (#)

This empirical relationship was first introduced in 1980 and has since been revised several times. The peak material constants describe an undisturbed rockmass where excavations are carefully blasted or machined bored (EQ. 4 and 5).

\[
\frac{m_p}{m_i} = \exp \left( \frac{RMR - 100}{28} \right)
\]  

EQ. 4

\[
s_u = \exp \left( \frac{RMR - 100}{9} \right)
\]  

EQ. 5

where

- RMR - rockmass rating classification (as per Bieniawski)
- \(m_i\) - determined from triaxial testing on intact lab specimens.

Conversely, the residual material constants represent the disturbed rockmass characterized by blast induced damage without confinement (EQ. 6 and 7).

\[
\frac{m_{dis}}{m_i} = \exp \left( \frac{RMR - 100}{14} \right)
\]  

EQ. 6

\[
s_{dis} = \exp \left( \frac{RMR - 100}{6} \right)
\]  

EQ. 7

Quantification and Prediction of Wall Slough in Open Stope Mining Methods
For this case study, the average RMR value is 75% and the \( m_{ds} \) value is 2.8494 and the \( s_{ds} \) is 0.0155\(^4\). Table 5 is provided to show the most recent peak and residual 'm' and 's' values as per the "Hoek and Brown Method".

**Grid Specifications**

The results of this program are presented on grids. Grid locations are described in the same manner as the model blocks.

Stress values will be taken from mid-pillar core locations for design purposes. The placement of these grids are designed to assess the stability of the sill (separating the above open stope) and rib pillars (dividing the adjacent panels).

**Mining Step Specification**

Mine sequencing can be simulated in terms of mining steps. Each step will consist of the mining blocks identified for excavation by specifying a surface stress state of zero. Figure 0.2 is a longitudinal view of the Detour Lake Mine layout.

---

\(^4\) Values currently employed at Detour Lake Mine.
Table 5 - "Hoek and Brown Failure Criteria (1988)"

<table>
<thead>
<tr>
<th>Disturbed rock mass m and s values</th>
<th>Undisturbed rock mass m and s values</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>EMPIRICAL FAILURE CRITERION</strong></td>
<td><strong>CARBONATE ROCKS WITH WELL DEVELOPED CRISTALS AND NODULES</strong></td>
</tr>
<tr>
<td></td>
<td><strong>LITHIFIED ARGILLACEOUS ROCKS</strong></td>
</tr>
<tr>
<td></td>
<td><strong>ARFIFICIOUS ROCKS WITH POORLY DEVELOPED CRYSTALS AND QUARTZITE</strong></td>
</tr>
<tr>
<td></td>
<td><strong>FINE GRAINED POLYMINERALIC IGNEOUS CRYSTALLINE ROCKS</strong></td>
</tr>
<tr>
<td></td>
<td><strong>COARSE GRAINED POLYMINERALIC IGNEOUS &amp; METAMORPHIC CRYSTALINE ROCKS</strong></td>
</tr>
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</table>

**INTACT ROCK SAMPLES**

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<th>Laboratory size specimens free</th>
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<th>10.00</th>
<th>15.00</th>
<th>17.00</th>
<th>25.00</th>
</tr>
</thead>
<tbody>
<tr>
<td>m from discontinuities</td>
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<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
<tr>
<td>s</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
<tr>
<td>CSIR rating: RMR = 100</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating: Q = 500</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**VERY GOOD QUALITY ROCK MASS**

<table>
<thead>
<tr>
<th>Tightly interlocking undisturbed rock</th>
<th>2.40</th>
<th>3.43</th>
<th>5.14</th>
<th>5.02</th>
<th>8.56</th>
</tr>
</thead>
<tbody>
<tr>
<td>m with unweathered joints at 1 to 3m.</td>
<td>0.082</td>
<td>0.082</td>
<td>0.082</td>
<td>0.082</td>
<td>0.082</td>
</tr>
<tr>
<td>s</td>
<td>4.10</td>
<td>5.85</td>
<td>8.78</td>
<td>9.95</td>
<td>14.63</td>
</tr>
<tr>
<td>CSIR rating: RMR = 85</td>
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<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating: Q = 100</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**GOOD QUALITY ROCK MASS**

<table>
<thead>
<tr>
<th>Fresh to slightly weathered rock, slightly disturbed with joints at 1 to 3m.</th>
<th>0.575</th>
<th>0.821</th>
<th>1.231</th>
<th>1.395</th>
<th>2.052</th>
</tr>
</thead>
<tbody>
<tr>
<td>m</td>
<td>0.00293</td>
<td>0.00293</td>
<td>0.00293</td>
<td>0.00293</td>
<td>0.00293</td>
</tr>
<tr>
<td>s</td>
<td>2.006</td>
<td>2.865</td>
<td>4.298</td>
<td>4.871</td>
<td>7.163</td>
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**FAIR QUALITY ROCK MASS**

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<th>Several sets of moderately weathered joints spaced at 0.3 to 1m.</th>
<th>0.128</th>
<th>0.183</th>
<th>0.275</th>
<th>0.311</th>
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**POOR QUALITY ROCK MASS**

<table>
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<th>Numerous weathered joints at 30-500 mm, some gouge. Clean compacted waste rock</th>
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**VERY POOR QUALITY ROCK MASS**

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<th>Numerous heavily weathered joints spaced &lt;50mm with gouge. Waste rock with fines</th>
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The model created will attempt to reflect the mine sequencing in one month intervals (Figure 0.3).
Stress Background

In materials science, the behavior of a material is often defined by a stress and strain relationship (Figure 0.4).

![Typical Stress vs. Strain Curve](image)

**Figure 0.4 - "Typical Stress vs. Strain Curve"**
Mathematically, this relationship is more commonly known as Hooke’s Law (EQ. 8).

\[
\sigma = (E) (\varepsilon) \quad \text{EQ. 8}
\]

where

- \( \sigma \) - stress [N/m\(^2\)]
- \( \varepsilon \) - strain [mm/mm]
- \( E \) - Young’s Modulus [N/m\(^2\)]

Stress is defined as force over area whereas strain is the deformation (change in length) over original length.

The dotted line represents the ideal material response termed elastic behavior. It shows a proportional increase in deformation with increasing stress. Calculating the slope of this line will yield the elastic modulus ‘E’ of the material (Young’s Modulus) as defined in Hooke’s Law.

The solid line represents the stress-strain curve for a rock material. As stress is increased, strain also increases. It will eventually reach a peak value that defines the yield strength of that particular rock material. However, deformation may continue with time and equilibrate at a residual strength. For hard rock material, failure will generally be brittle in nature. Therefore, this report will confine its analysis to fall within this linear elastic region.

The stress values determined are installed to assume approximate principal stress directions. Since the model is constructed to resemble an intact material, the stresses induced are intuitively expected to be interdependent of the principal directions. Consider a cube of material being compressed in the vertical direction (Figure 0.5).

![Figure 0.5 - "Infinitesimal Cube of Rock"](image-url)
By Newton's first law (For every applied force there is an equal and opposite reactive force), the 
induced stress is distributed to an x-y plane perpendicular to the applied vertical force. Therefore, the strains associated with this vertical stress are given by the following equations (Hoek and Brown, 1982):

\[ \varepsilon_z = \frac{1}{E} [\sigma_z - \nu(\sigma_x + \sigma_y)] \] \hspace{1cm} \text{EQ. 9}

\[ \varepsilon_x = \frac{1}{E} [\sigma_x - \nu(\sigma_y + \sigma_z)] \] \hspace{1cm} \text{EQ. 10}

\[ \varepsilon_y = \frac{1}{E} [\sigma_y - \nu(\sigma_x + \sigma_z)] \] \hspace{1cm} \text{EQ. 11}

where \( \varepsilon_{x,y,z} \) - strain in the respective directions
\( \sigma_{x,y,z} \) - stress in the respective directions
\( E \) - Young's Modulus
\( \nu \) - Poisson's ratio.

This closed form solution can be solved since there are only three unknowns with three equations. Isolate \( \sigma_{x,y,z} \) for each equation respectively.

\[ \sigma_z = E \varepsilon_z + \nu(\sigma_x + \sigma_y) \] \hspace{1cm} \text{EQ. 12}

\[ \sigma_x = E \varepsilon_x + \nu(\sigma_y + \sigma_z) \] \hspace{1cm} \text{EQ. 13}

\[ \sigma_y = E \varepsilon_y + \nu(\sigma_x + \sigma_z) \] \hspace{1cm} \text{EQ. 14}

Solve in terms of strain only (Herget, 1991).

\[ \sigma_y = \frac{E [\varepsilon_y + \nu (\varepsilon_z + \varepsilon_x - \varepsilon_y)]}{(1 - 2\nu)(1 + \nu)} \] \hspace{1cm} \text{EQ. 15}

\[ \sigma_x = \sigma_y + \frac{E (\varepsilon_x - \varepsilon_y)}{(1 + \nu)} \] \hspace{1cm} \text{EQ. 16}

\[ \sigma_z = \sigma_y - \frac{E (\varepsilon_y - \varepsilon_z)}{(1 + \nu)} \] \hspace{1cm} \text{EQ. 17}

\textbf{Equations 15, 16, and 17} are applicable if the borehole is drilled in a principal stress direction where strains are measured in the axial direction and in the plane at right angles to the borehole.
APPENDIX IV

EXTENSOMETER RESULTS

Extensometer #2 showed a steady increase in hangingwall displacement that finally settled upon completion of the stope. This instrument is a two-point extensometer with the anchors located at 1.0 m and 2.0 m from the hangingwall. An overall cumulative displacement of approximately 23 mm was recorded.

Figure 0.1 - "Extensometer #2"
Extensometer #5 was also a two-point extensometer that indicated a total displacement of approximately 2.0 mm.

Figure 0.2 - "Extensometer #5"
Extensometer #3 (multiple point) demonstrated minimal displacement. The cumulative amount of wall movement was approximately 2.5 mm.

Figure 0.3 - "Extensometer #3"
The final extensometer was #4. One of the four anchors produced unrealistic results. In total, the measured displacements never exceeded 1.5 mm.

In summary, the overall extensometer data confirmed that the hangingwall was in a state of relaxation. Discrete block movements were recorded by the four mechanical extensometers.
APPENDIX V

BLAST MONITORING PROGRAM

In order to develop an in situ blast damage criteria for DLM, seven blasts have been reduced and are summarized as follows:

**Blast 2 (F6):** This blast facilitated the opening of the slot to full ore width. Two rings were omitted from the original design to avoid damaging the adjacent pillar. Fourteen holes were expected, however; only 12 vibration events were recorded (Figure 0.1). Review of the waveform indicated the original capping order had been altered during the loading procedure.

Cap scatter was observed to be within tolerance levels. Of the caps that detonated, there were no misfires observed. The average PPV levels did not exceed 50 mm/s.

![Cap Scatter Analysis](image)

**Figure 0.1 - “Blast F6 - Cap Scatter Analysis”**

**Blast 3 (F7):** Analysis of the waveform indicated that all holes were recorded (Figure 0.2). Cap scatter analysis indicates that one hole (Cap #14) demonstrated the potential for a misfire. In this blast design, seven pairs of caps with the same delay period were initiated. Four of these pairs behave as individual vibration events (simultaneous detonation).

Application of the Fourier Analysis function describes a single detonating hole with an average frequency of 1.1 kHz which represents 10-15% of the total energy. Using this as a reference, four holes (Cap #15, #20, #21, #23) are observed to have been interfered with by adjacent holes.
The average PPV readings range from 35-65 mm/s. Only one hole detonated at 140 mm/s (Cap #23). It is speculated that the ‘helper hole’ ahead did not perform its function.

![Cap Scatter Analysis](image)

**Figure 0.2 - “Blast F7 - Cap Scatter Analysis”**

**Blast 4 (F8):** In this blast, a total of 25 vibration events were recorded. Cap scatter was observed to be within the accepted tolerances (Figure 0.3). Analysis of the waveform indicates that there were four holes missing (Cap #3, #10.5, #14.5, and #23). Interestingly, these missing holes occur along the perimeter of the blast. Vibration control in this blast design was limited to two holes per delay. Of the three sets of pairs, the vibration signatures for two of them (Caps #16 and #20) were easily distinguishable (at least 10 milliseconds separated each double cap). Cap #24 was the only pair that experienced destructive interference.

Overall, the blast damage incurred from this blast was minimal. The PPV levels were never in excess of 60 mm/s. It was observed that the vibration levels from the ‘helper holes’ were slightly reduced in comparison to the regular blastholes.
Blast 5 (F9): Vibration analysis of this blast identified 28 detonating holes out of a total of 36. The five missed holes are located along the perimeter of the blast and adjacent to the 'helper holes'. Cap scatter was observed to be within tolerance (Figure 0.4).

Overall, the blast is characterized by PPV levels measuring below 75 mm/s. However, there are four events that recorded significantly higher vibration levels (Cap #12, #19, #22, #23). These holes produced PPV levels in the range of 100-300 mm/s.

Blast 6 (F10): In this blast, vibration control was expanded to three holes per delay. Of the 11 pairs of caps that detonated, Caps #12.5, #15 and #19 were subject to constructive interference. These caps produced higher than normal PPV vibrations in the range of 130 mm/s (average PPV = 80-90 mm/s).
A total of four holes were unaccounted. These missing holes were either 'helper holes' or holes located adjacent to 'helper holes'. Three simultaneous events occurred for Caps # 15, 19 and 15. Misfires were not probable since cap scatter was observed to be within the accepted tolerances (Figure 0.5).

![Cap Scatter Analysis](image)

**Figure 0.5 - “Blast F10 - Cap Scatter Analysis”**

**Blast 7 (F11):** Twenty eight holes were recorded in this longhole blast. Only one hole was missed in Ring 187 located along the perimeter. The frequency analysis suggests that this hole was disrupted by an adjacent hole (average hole exhibits 15-50% at 150 Hz). Cap scatter is noted to be within the accepted tolerances.

Overall, the recorded PPV averaged around 160 - 200 mm/s. However, several holes produced >500 mm/s PPV values which may lead to some incipient blast damage (near Ring 189).

![Cap Scatter Analysis](image)

**Figure 0.6 - “Blast F11 - Cap Scatter Analysis”**
Blast 8 (F12): The final blast in this block consists of 22 recorded events with no missing holes (Figure 0.7). One hole (Cap #14) demonstrated the potential for misfire. Three holes exhibit signs of interference based on vibration traces (average reading of 15-20% at 2.0 kHz). This observation is attributed to near simultaneous detonation of multiple caps (Cap # 12, 16 and 22).

The average PPV readings for this blast ranged from 300 - 500 mm/s. Seven holes produced PPV levels >500 mm/s. The location of these damaging holes plot along the hangingwall and footwall.

Figure 0.7 - “Blast F12 - Cap Scatter Analysis”
P-WAVE CALCULATION

Known Data:
- Distance from Geophone 1 to 2 = 9.10 m
- Distance from Geophone 1 to 3 = 18.30 m
- Distance from Geophone 1 to 4 = 26.70 m
- Distance from Geophone 2 to 3 = 9.20 m

Fact:
\[ D = V \times T \]

where
- \( D \) = distance (m)
- \( V \) = velocity (m/s)
- \( T \) = time (s)

Geophone 1 to 2:
- \( 9.1 \text{ m} / (37.3 \text{ ms} - 35.9 \text{ ms}) = 6500 \text{ m/s} \)
- \( 9.1 \text{ m} / (129.3 \text{ ms} - 127.9 \text{ ms}) = 6500 \text{ m/s} \)
- \( 9.1 \text{ m} / (152.3 \text{ ms} - 153.8 \text{ ms}) = 6067 \text{ m/s} \)

Geophone 1 to 3:
- \( 18.3 \text{ m} / (38.2 \text{ ms} - 35.9 \text{ ms}) = 7956 \text{ m/s} \)
- \( 18.3 \text{ m} / (130.6 \text{ ms} - 127.9 \text{ ms}) = 7778 \text{ m/s} \)
- \( 18.3 \text{ m} / (154.9 \text{ ms} - 152.3 \text{ ms}) = 7038 \text{ m/s} \)

Geophone 2 to 3:
- \( 9.2 \text{ m} / (38.2 \text{ ms} - 37.3 \text{ ms}) = 10222 \text{ m/s} \)
- \( 9.2 \text{ m} / (130.6 \text{ ms} - 129.3 \text{ ms}) = 7077 \text{ m/s} \)
- \( 9.2 \text{ m} / (154.9 \text{ ms} - 153.8 \text{ ms}) = 8364 \text{ m/s} \)

Note: Times taken from Blast F7

The average for all the times is ~7 400 m/s. Removing the outlier 10 222 m/s yields an average of 7 035 m/s.

For subsequent calculations, use 7 000 m/s (23 000 ft/s) for estimating the P-wave.
PLACER DOME CANADA  
Detour Lake Mine  
935 Q120 Longhole Stope

**Predicted critical particle velocity**

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<th>Time Parameter</th>
<th>Value</th>
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- for 2 1/2" hole (Atlas Powder)

@ file: c:sambo/thesis/done/blast/wk4/ppv.wk4
### Quantification and Prediction of Wall Slough in Open Stope Mining Methods

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**Note:**
Distance measurements are taken from the geophone location to the toe initiation point.
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Explosives Used: 864.2

Note: Distance measurements are taken from the geophone location to the toe initiation point.
APPENDIX VI

DRILLHOLE DEVIATION DATA

This section presents the drillhole deviation data collected from four longhole stope blocks:

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Quantification and Prediction of Wall Slough in Open Stope Mining Methods

DETOUR LAKE MINE
PLACER DOME CANADA

Drillhole Deviation Test

Date: June 10, 1995
Stope Survey: Topo Offset
Location: 460-480
Stope: 935 Q120

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## PLACER DOME CANADA
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**# of obs:** 9.0  9.0  9.0
## Quantification and Prediction of Wall Slough in Open Stoping Methods

### PLACER DOME CANADA

#### DETOUR LAKE MINE

**Drillhole Deviation Test**

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### PLACER DOME CANADA
#### DETOUR LAKE MINE

**Drillhole Deviation Test**

**Date:** June 6, 1995  
**Stop Survey:**  
**Reference Line:**  
**Location:** 630-650  
**Stope:** 865 Capping

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**Remarks:**
- **Survey Error:**
- **Missing Info:**

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**Notes:**
- **# of obs:** 53.6
- **Mean:** 10.0
- **95% CI:** 54.0