Support Monitoring and Stope Stabilty at INCO's Manitoba Division

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Abstract

A new approach to underground excavation design at INCO's Manitoba Division was undertaken by incorporating ground and support monitoring, and numerical modelling into an empirical framework.

In the first phase of the study, a field measurement geomechanics guideline was developed for the Manitoba Division. Work in instrumentation led to the development of a new cable bolt monitoring device by INCO Ltd. and McGill University. This device has the ability to measure cable bolt performance reliably and at a low cost. As a result, a cost efficient and effective program is being implemented to monitor cable bolt layouts representatively.

The second phase of the project utilizes the computational power of numerical modelling to aid mine design of stope backs. Results from numerical modelling, presented in the form of stress signatures, suggest that there might be a correlation between back surface stability and calculated values of major and minor principal stresses.

By adding the tools of numerical modelling and instrumentation to traditional rock characterization data, a new data base for empirical modelling is established. The continual expansion of this database will lead to a more comprehensive approach to stope back design in the future.

Résumé

Une nouvelle approche pour l'excavation souteraine à la division Manitoba de INCO Ltée fut élaborée en incorporant la prise de donnees du roc et des supports et le modelage numerique, le tout étant reconstitue dans un cadre empirique.

Dans la priemière phase de l'étude, un guide pour les mesures geomécaniques sur le site a été élaboré pour la Division Manitoba. L'instrumentation nous a amené à developer conjointement avec L'Univeristé McGill et INCO Ltée un système pour la prise de donnees sur les câbles d'ancrages. Ce système nous donne la possibilité de verifier la performance des câbles avec une grande fiabilité et à un coût moindre. Le résultat est qu'un programme peu coûteux et efficace fût entrepris pour la prise de données sur les câbles en patrons représentatifs.

La deuxième phase du projet utilise le model numérique pour aider dans le design des plafonds de chantiers. Les resultats du model numérique présentés sous forme "d'indicatif de contrainte" (stress signatures) suggèrent qu'il peut y avoir une correlation entre la stabilité et les valeurs calculées des contraintes majeure et mineure principales.

En ajoutant les outils de modelage numérique à l'instrumentation et les données traditionnelles de caractérisation du roc, une nouvelle base de données pour le modelage empirique est établie. Cette nouvelle base de données n'est qu'à ses débuts, mais une approche plus aprofondie du design de plafonds de chantiers pourait être disponible prochainement.

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Chapter 1 - Introduction

1.0 Rock Mechanics In Mining

A chief objective of rock mechanics in mining is to optimize the design of excavations. An understanding of the influence of stress and structure on excavation stability as well as the knowledge of how various ground support systems operate in different ground conditions are crucial to good geotechnical design. These designs must be consistent with the principles of a safe workplace, accessibility to future mining sites, and minimal ground control costs[1]. In mining practices, rock mechanics principles are translated into design of stope sequencing, excavation size and shape, and ground control.

Optimization of excavation-support design requires the identification of the factors and conditions that affect stability. Fundamentally, the question could be, given an excavation with a certain purpose, size, shape, and lifespan in a particular rock mass and stress field, what's the most cost effective means of support? Responses to this question can be categorized into analytical, empirical, or numerical assessment. Of these, only the latter two methods are applied in this study as analytical tools are still under development.

1.1 Thesis Objectives

The fundamental objective of this study, which began in 1991, is to develop a continuous improvement approach towards stope back design by focusing on stability prediction. This is attempted in two steps. The first step is a survey of previous work in stability prediction. The second step is the development of a prediction tool into which new technologies are incorporated.

This paper is divided into five chapters. The remainder of chapter one gives an overview of INCO Ltd. - Manitoba Division as well as the

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[1] Brady and Brown, 1985.

general geology of the Thompson Nickel Belt. Chapter two discusses the role of monitoring within the framework of a geotechnical database. Details on the geotechnical instruments used during this investigation are also provided. In chapter three, numerical and empirical design methods are reviewed. A comprehensive 1990 CANMET empirical study at INCO - Manitoba Division is analyzed in the beginning of chapter four. The remainder of chapter four presents the results and findings from 13 case studies using a combination of monitoring, and empirical and numerical modelling. Conclusions and recommendations are found in the fifth and final chapter.

1.2 Overview of INCO Ltd.- Manitoba Division

The city of Thompson is located about 645 km (400 miles) north of the city of Winnipeg in the province of Manitoba (see figure 1). In the Thompson Nickel Belt, there are currently two active mine sites. These are Birchtree Mine and the Thompson Mine. Birchtree Mine is located five km south of Thompson. The Thompson Mine Complex is approximately 2 km east of the city.

Inco - Manitoba Division is a fully integrated mining and metallurgical complex of nickel since production commenced in 1961. Marketable amounts of copper, cobalt and precious metals are recovered with the nickel. At present, 50,000 tonnes (110 million pounds) of electrolytic nickel is produced annually with over 2,000 employees[2].

At present, ore is being extracted at 12,500 T.P.D. from two underground mines and one open pit operation. The Thompson Open Pit began mining in 1985 and was expected to produce until 1994. The Thompson Mine commenced production in 1961 and is serviced by two shafts (T-1 & T-3). A third shaft, called T-5, was scheduled to come into operation by 1996. Birchtree Mine began operation in 1967 and had closed in 1978. In 1988,

[2] INCO information pamphlet.

Birchtree Mine was reopened. Original production was overwhelmingly from cut and fill operations until the change to bulk mining began in 1981 (the open pit excepted). The conversion to bulk and highly mechanized mining methods facilitated safety and increased productivity. Bulk mining methods such as Vertical Crater Retreat (VCR) or open stoping accounted for over 87% of underground production in 1991.

1.3 Geology

1.3.1 Regional Geology

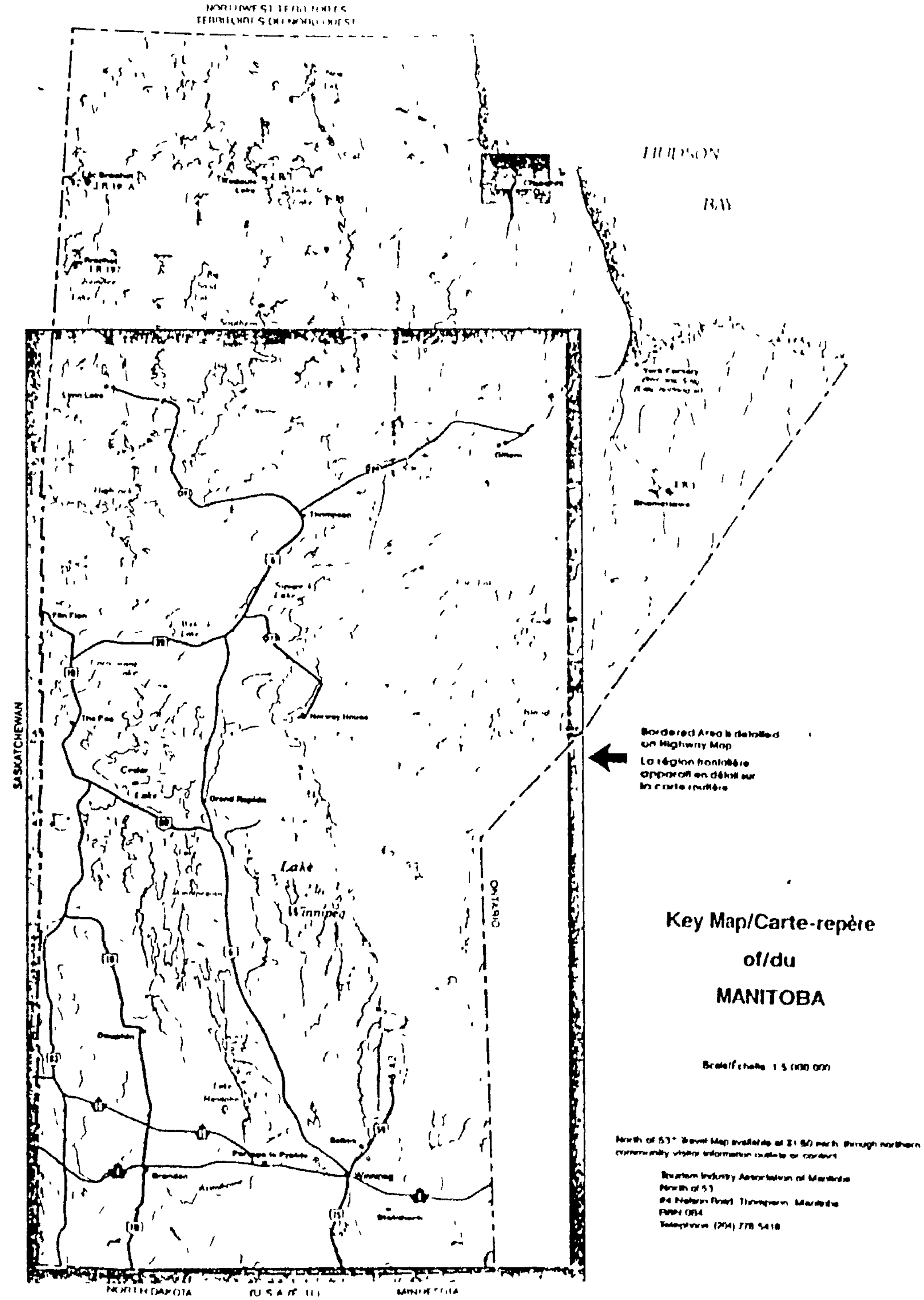
The Thompson Nickel Belt is approximately 130 km (75 miles) long and 16 km (10 miles) wide. The Thompson Nickel Belt is the largest known nickel deposit associated with ultramafic rocks in the Canadian Shield. The Nickel Belt trends in a northeasterly direction and is considered to be generally fault bounded on both sides. To the northwest, the Belt is bounded by the Churchill Province. To the southeast, the Belt is in contact with the Pikwitonei Region of the Superior Province.

1.3.2 Mine Geology

The Thompson ore body is a large and highly deformed stratabound deposit contained within a suite of proterozoi· metasediments. The ore hosting metasediments are near to and stratigraphically overlie an Archean Gneissic complex. In a typical stratigraphic column at Thompson Mine, from the hanging wall to footwall, the rock mass consists of archean gneisses, micaceous quartzite, skarn, schist, nickel mineralization, schist, chert (footwall quartzite), iron formation and core rocks (thick assemblage of quartzites, schists, and massive amphibolies).

The Birchtree ore body is a tabular stratabound structure. The dimensions are 1.5 km in length striking 30 degrees east of north. The ore body is generally four to seven metres wide and dips to the west at 70 degrees. Nickel mineralization is divided into massive sulphides, sulphide matrix, and stringer sulphides.

Figure 1: Map of Manitoba, Canada.[1]



[3] Reproduced from Map of Manitoba.

Chapter 2 - Monitoring and the Geotechnical Database

2.0 Introduction to Data Collection

"The development and maintenance of a well structured data collection system is of fundamental importance to a mining operation, regardless of the mining method being employed."[4] The basis of all sound design is data. All design models, be they empirical, numerical or analytical, require a database to calibrate their effectiveness and limitations. Geotechnical databases provide the only scientific and statistical basis of good design.

Data collection can be divided into three categories. These are characterization, testing, and monitoring.[5] Characterization is a one time measurement of a particular property. A scanline survey, the measurement of far field stresses, and the recording of the geology of an area, are all cases of data collection via characterization. In testing, a repeatable experiment is conducted to obtain data. Common examples of testing are the determination of the average uniaxial compressive strength (UCS) of a particular rock type, and the average pull out strength of a set of rockbolts. Lastly, monitoring is the intermittent recording of a particular behaviour. Measurements of changes in stress in a mine pillar, or the water table are examples of geotechnical monitoring. Further details about monitoring are described below.

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[4] 1990 CANMET report, Chapter 2.

[5] Mine Monitoring Manual, Franklin, 1990.

2.1 Monitoring

The basic purposes of geotechnical monitoring are to promote safety, to collect data for design, or to verify design assumptions. These reasons are elaborated upon in Table I.

Table I: Basic Purposes of Monitoring[6]

1- to protect miners and prevent accidents
2- to obtain data for design via back analysis of trial excavations.
3- to verify design data and assumptions so that adjustments can be made in the future.
4- to investigate failure and on going instability so that remedial work could be planned.

The costs and benefits of a monitoring program depend partly on whether it is conducted visually or with the aid of instrumentation. Visual monitoring is inexpensive, but only qualitative data is obtained. This data can be suspect (i.e. bias) as it relies entirely on the judgement and memory of personnel. The added costs associated with the use of instrumentation can aid decision making because quantitative and less or non-bias data (hopefully) can be collected.

An instrumentation guideline to limit potential problems, such as poorly installed equipment or poor site selection is proposed below. It is suggested that this guideline be updated periodically (i.e. twice a year) as new knowledge is obtained.

[6] ibid.

2.1.1 Suggested Monitoring and Instrumentation Guideline

The two most vital aspects of a monitoring program involving instrumentation are accuracy and completeness of information. Since decisions concerning safety and production could be greatly affected by instrumentation data, the potential costs of faulty information could be significant. It is also evident that it would be very frustrating to evaluate a particular case study with only a partial amount of the required information.

One simple method to control misinformation is to ensure that visual monitoring occurs in conjunction with instrumentation whenever possible. Visual monitoring acts as insurance against malfunctioned or poorly installed equipment. It also provides information at locations that might not be covered but are adjacent to the instrumented site. Visual monitoring that does not endanger personnel or equipment should always be conducted without exception.

Another convenient method to aid the visual monitoring aspect of an investigation is through the employment of a video camera. A video camera has certain advantages not offered by the eye or a conventional camera. The video and the camera are superior to the eye and memory because of its ability to visually document. The added advantage of the video over a conventional camera is that a video camera can record a superior perspective over an area since a camera may tend to capture only the areas deemed important at the time (i.e. usually failed ground or support members) and might not record future areas of interest. Therefore, the video camera can be a rock mass monitoring tool. Another feature is that an audio interpretation can be recorded along side the visual image. Moreover, film processing time and costs are virtually eliminated. Finally, a video image is a cost efficient ($2,500 start-up) presentation, communication, and managerial tool.

When implementing an instrumentation program, there are ten points to consider in order to maximize the information benefit. Firstly, ground behavioral trends must be predicted so that the proper inst uments, the number of instruments, and the cost of the program can be calculated. Instruments can then chosen according to the durability, and degree of sophistication required for the task at hand. Other concerns are the ease of installation, ease of reading and maintenance, and costs of the instrumentation. Since it is quite common that different areas within one excavation behave distinctly, the higher the extend of coverage the lower the chances an event would go undetected. This leads to the principle of redundancy, which provides insurance against faulty readings, and usually increases the extend of coverage. Finally, instrumentation diversity could account for the number of variables being examined in the investigation. These may include ground movement, generated support loads, or changes in stress. The above considerations are summarized below in Table II.

Table II: Ten Considerations of Any Geotechnical Program.

1- Extent of Coverage.
2- Redundancy.
3- Instrumentation Diversity.
4- Predictability of Ground Behaviour.
5- Costs.
6- Durability.
7- Sensitivity, accuracy, and precision.
8- Ease of installation of instrumentation.
9- Ease of maintenance and reading.
10- Negligible interference with mining operations.

2.2 Geotechnical Equipment

One of the objective of this investigation is to gain further understanding of support and ground behaviour. To accomplish this task, four types of instrumentation were used. These were rod extensometers to monitor ground movement or displacement, vibrating wire strain gages mounted on rockbolts, tensmeg 70 strain gages and the cable strain measuring device to monitor cable bolts' performances. These devices are briefly described below.

2.2.1 Vibrating Wire[7]

When tension is applied to a thin wire of length l and density d, the natural frequency of vibration f, is related to the tensile stress t on the wire by the equation,

f= (1/2l)*(t/d)exp0.5

The frequency can be measure by magnetic plucking. Then, the tension on the wire can be solved. The advantages of vibrating wire sensors are that they are very accurate and durable instruments.

2.2.2 Cable Bolt Monitoring Devices

At the start of this study, a market search into cable bolt monitoring devices indicated that the Tensmeg Gauges, supplied by Supportek Inc., mounted on cable bolts was the only means by which cable bolts could be monitored. Subsequently, INCO - Manitoba Division (Al Meston, Philip Ng, and Matt Sutcliffe) developed the Cable Strain Measuring Device (CSMD) for the sole purpose of monitoring cable bolts. Both of these devices are described in the following sections.

2.2.3 Tensmeg-70 Strain Gauges[8]

The Tensmeg gauge, developed by P. Choquet and F. Miller, functions on the principle of wire resistance for measurement of tension in 5/8" cable bolt grouted in a borehole. The three variables governing wire resistance are material type, temperature and wire stretch. To limit resistance variation due to temperature fluctuation a nickel-chromium alloy with a low temperature coefficient of resistance was selected for

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[7] Brady and Brown, 1985.

[8] Supportek information pamphlets, Choquet and Miller.

this instrumentation. The resistance wires are insulated from the grout by a rubber tube. The assemblage is wounded into the grooves between the wire strands of the cable bolt. Each gauge is 76 cm (25 in.) in length and has 70 ohm resistance. At the ends of the wire, it is moulded into a rubber anchor which is glued on to the cable bolt.

2.2.4 The Cable Strain Measuring Device (CSMD)

The Cable Strain Measuring Device was invented and developed at INCO -Manitoba Division by M. Sutcliffe, A. Meston (of INCO Ltd.) and P. Ng (McGill University). This device was invented to address the costs and availability issues. It is best described as a hybrid of a cable bolt and a wire extensometer. This is because two wires of different lengths are attached to different points of the cable bolt. From these anchors the wires are placed inside a small diameter pvc tube which runs parallel with the cable bolt. It operates strictly as a mechanical device where displacement is measured. During monitoring, the wires are tensioned using a hook on a spring to maintain reading consistency. The materials and tools used to manufacture this device are described in Table III and Table IV. The assemblage of the CSMD is described in Table V. The protruding end of the CSMD is shown in figure 2.

The Cable Strain Measuring Device has an accuracy of +/- 1 mm in each measurement when read manually with constant tension applied. The current procedure mandates that each set of measurements be taken three times to reduce errors. The average is then recorded. The three sets of measurements are:

1- The distance from a fix point on the cable to the barrel of the face plate (see figure 2).

2- The distance from the crimp of the short wire anchored 3 to 5 feet up the cable to a fix point on the cable bolt.

3- The distance from the crimp of the long wire anchored 20 feet up the cable to a fix point on the cable bolt.

Figure 2: Cable Strain Measuring Device. The set of measurements to be recorded are indicated by "PLATE" and "10 FT" on the right side.

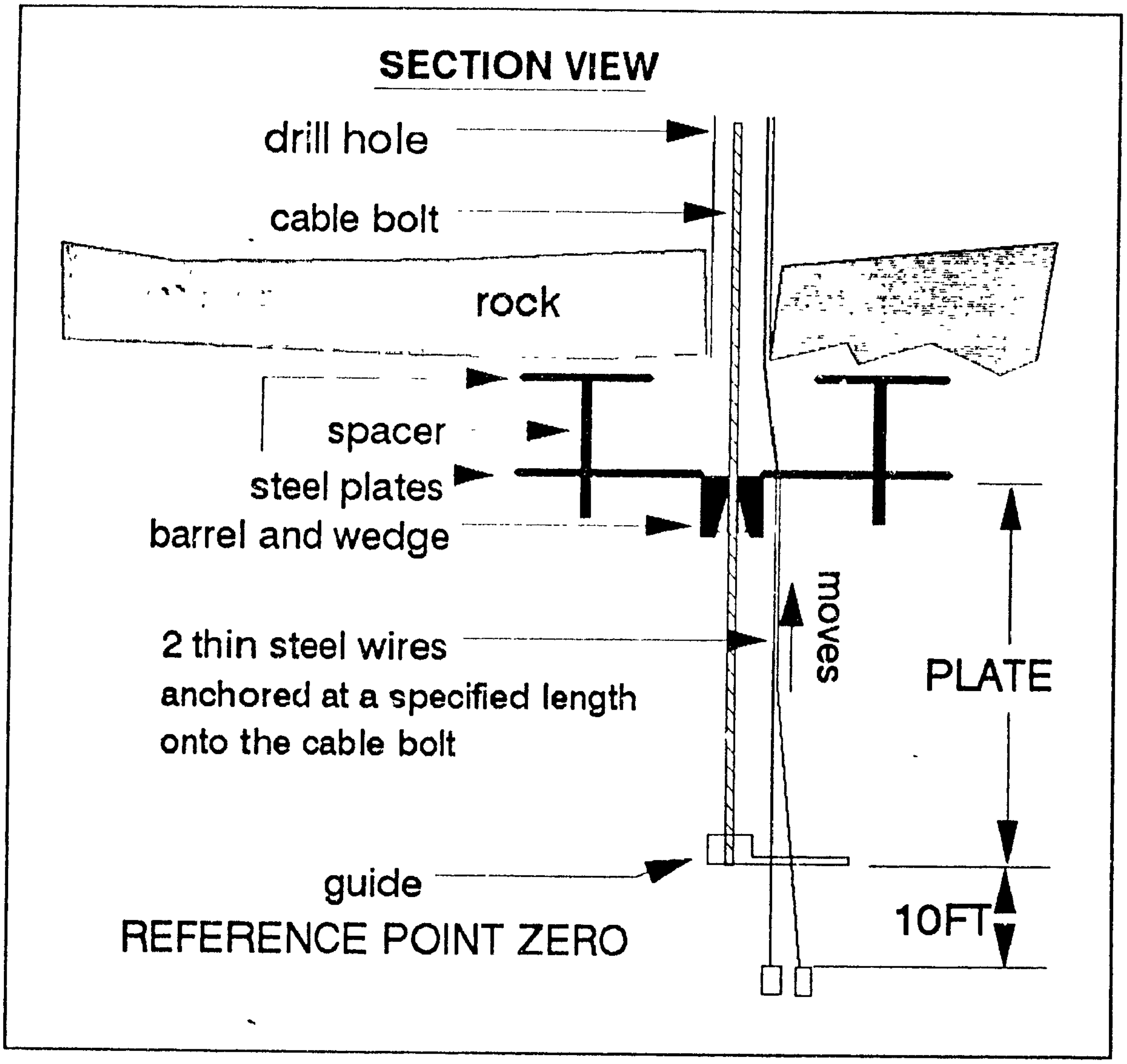


Table III: Materials for CSMD

1	5/8" cable bolt of length (L).
1	roll of 1.2 mm diameter electric welding wire.
2	#5 punch lock clamps.
1	1/4" outside diameter white polyethylene tube cut to length (L + 1.5 m).
5	#2 sh. marr wire connectors for 10-18 gage wire. electrical tape.

Table IV: Tools to Assemble CSMD.

Utility Knife
Pliers
Screw Driver
Punch Locks and Clamps

Table V: How to make your own CSMD.

Procedure:

1- Cut polyethylene tube to the required length and stretch out.

2- Slide one wire along the length of the tube. Do not bend wire. Cut wire from roll leaving approximately 0.7 m protruding from each end of the tube. Mark this wire and note its length so as to distinguish it from the other wire(s).

3- At the desired position, puncture the tube with the wire and slide through to the hole collar end of the tube. Mark this wire.

4- Using one marr connector, seal off the toe end of the tube, thereby, securing the position of the longest wire.

5- To the protruding toe end of each wire, secure two marr connectors head to head on each wire, five to eight cm from the seal.

6- Wrap electrical tape around the puncture, and the seal to prevent grout from entering the tube.

7- At the installation site, uncoil the cable bolt in a clean area. Place the CSMD along side the cable bolt. Place the clamps over the cable bolt and the head to head marr connectors at the desired position and fasten with the punch lock.

8- Tape grout tube to cable bolt if required and insert in the hole.

9- Grout and let set.

10- Place position indicators on the wires and the cable and measure base reading.

Table VI: Pull Test Results
Pullout strength of CSMD cable bolt vs single cable bolt with 0.4 w/c ratio. (left) No. 1-5 and 11-15 are CSMDs. (right) No. 6-10 and 16-20 are normal cable bolts.

No.	Length	Load	No.	Length	Load
1	30 cm	700 psi	6	30 cm	1000 psi
2	30	1000	7	30	1000
3	30	1100	8	30	1000
4	30	900	9	30	1000
5	30	1000	10	30	1000
Avg.	30	940	Avg.	30	1000

No.	Length	Load	No.	Length	Load
11	60 cm	3000 psi	16	60 cm	2900 psi
12	60	2950	17	60	3300
13	60	2600	18	60	2800
14	60	2950	19	60	2400
15	60	3400	20	60	3600
Avg.	60	2980	Avg.	60	3000

A cable bolt pull out test conducted in Thompson Open Pit suggests that the 1/4" diameter (6 cm) pvc tube did not significantly reduce the strength of the cable bolt. Four sets of five samples with embedment lengths of 30 cm and 60 cm of CSMDs and normal cable bolts were tested for their pull out strength. The results are given in Table VI. The average pullout strength of the 30 cm samples of CSMDs were 6% less than that of a normal cable bolt whereas it was only 1% for the 60 cm samples. Given that the reading accuracy of the pull test equipment was in the order of 100 psi (or 10% and 3% respectively), it can be said that this test did not indicate a significant pull out strength difference between a CSMD cable bolt and a normal cable bolt.

2.2.5 Rod Extensometers

The multi-point rod extensometers used in this investigation measures changes in displacement in the rock mass. This device is installed in boreholes where it is grouted. When the grout has set, a zero reading is taken as a reference point. Readouts are either performed manually with a calliper or intermittently with an electrical transducer. Further details about rod extensometers are widely available in numerous literatures.

2.3 Summary

The first part of chapter two discusses the objectives of a geotechnical database and the collection of data. The importance of visual monitoring are emphasized within the framework of a ground-support monitoring program. A guideline for monitoring with instrumentation is proposed to limit potential errors.

Different types of geotechnical instrumentation used in this investigation are reviewed in the latter part of the chapter. Brief reviews of vibrating wires, rod extensometers, and Tensmeg gauges, are provided. A detail description of the Cable Strain Measuring Device is also given.

Since a pull test indicated there is very little strength reduction of a CSMD cable bolt when compared to a regular cable bolt, its development is a significant step forward in the field of cable bolt support monitoring.

Chapter 3 - Design Methods

3.0 Introduction

The two current design approaches available and practised in geotechnical design are empirical and numerical methods. The most common numerical methods are the boundary element method and the finite element method; these are reviewed below. Rock mass classification systems have been the center of empirical design since their beginnings in the 1960's. The Rock Mass Rating system, and the Rock Mass Quality system are generally preferred in the Canadian mining industry and are described briefly. In addition, the more recent Stability Graph Method and the Modified Stability Graph Method are discussed as empirical methods of stope design.

3.1 Finite Element Method

In the finite element analysis, a domain around the opening(s) is defined into a mathematical network which consists of discrete interacting elements. This mathematical network is then solved via a five step process. These steps are discretization, problem approximation method, derivation of element equations, assembly of element equations to the global network, and computation of primary and secondary quantities.

Descretization involves the subdivision of the defined continuum into an equivalent system composed of the subdivided continua. These smaller continua are called finite elements. The adjoining corners that define the finite elements are known as nodal points.

Finite element assumes that transmission of internal forces between the edges of adjacent elements can be represented by interactions at the nodes of each element. This procedure analyses the problem as a continuum. The analysis then becomes one in which induced displacements and total stresses are calculated for each finite element. These induced strains and the elastic properties of the medium are then used to determine the induced stress in an element. The nodal displacement is often the primary quantity being calculated, whereas a secondary quantity such as stress is derived from displacement.

3.2 Boundary Element Method

The boundary element method computes a solution of a set of integral equations that connect the boundary tractions to boundary displacements. Whereas the finite element method involves differential equations, the boundary element method uses integral equations that consists of the discretization of the boundary of the body(ies) into a number of elements. Numerical solutions are obtained at the boundary first. Subsequently, solutions into the continuum are calculated based on previous solutions. As a result of the above approach, the boundary element method reduces the number of physical dimensions in the problem by a factor of 1.

There are benefits and drawbacks to the boundary element method when compare to the finite element method. Boundary element methods model far field stresses correctly, limit descretization errors to the problem boundary, and ensure continuous variation of stress and displacement throughout the continua. Unfortunately, the order of magnitude gained from the boundary element solution procedure is lost when modelling non-linear behaviour or multi-materials.

3.2.1 MAP3D

Numerical modelling is a very recent development in the mining industry. As with rock mass classification systems, there are a number of programs in the marketplace to estimate stress redistribution. A three dimensional elastic boundary element numerical model called MAP3D from Mine Modelling Ltd. was used in this report to estimate stress redistribution around underground excavations. An example of an input file is included in appendix A. A description of this program and the required input file are summarized in appendix B.

3.3 Rock Mass Classification Systems

Rock mass classification sytems came into prominence in the early 1970's. There was a general consensus that rock mass classification would provide a means to correlate the experience gained at previous sites for use in future excavation designs. Thus, the end use of rock mass classification systems was to aid the engineer in estimating the required support of an excavation.

Of the six major rock mass classification systems developed between 1969-1974, two are widely used in Canadian mines. These are the Rock Mass Quality (Q) system developed by Barton, Lein, and Lunde of the Norwegian Geotechnical Institute and the Geomechanics Classification (RMR) system developed by Bieniawski of the South African Council for Scientific and Industrial Research. Both of these methods seek to assign strength related values to particular rock mass characteristics. The parameters are then combined together to obtain an overall rating value for the rock mass. This value can then be use as a basis for comparison of mining experiences between different excavation sites. Below is a brief description of the Q and RMR rock mass classification systems.

The Q classification system was developed by Barton (NGI) with approximately 200 case studies. The value Q is a function of six parameters. These six parameters are:

1- The rock quality designation, RQD index (Deere, 1963).
2- The number of joint sets, Jn.
3- The roughness of the weakest joints, Jr.
4- The joint alteration, Ja.
5- The stress reduction factor, SRF.
6- The joint water reduction factor, Jw.

These parameters are then combined by the following equation,

$$Q = (RQD/Jn) * (Jr/Ja) * (Jw/SRF)$$

where,

- RQD/Jn represents joint spacing
- Jr/Ja represents adjustment factors for joint conditions
- Jw/SRF are adjustments due to water and stress factors

to obtain the value Q, which is plotted on a log scale. The higher the Q value of the rock mass the more competent the rock mass.

The Rock Mass Rating (RMR) system developed by Bieniawski is comparable to Barton's Q system as the main strength characteristics of the rock mass are recorded. Bieniawski uses six parameters to obtain his RMR value for a rock mass. These are:

1- The uniaxial compressive strength of the intact rock.
2- The Rock Quality Designation, RQD.
3- The spacing of joints.
4- The condition of the joints.
5- The ground water conditions.
6- The orientation of the joints.

Each of these parameters are assigned a value based on a point system. When the values of each parameter are added, an RMR value of up to 100 is obtained. RMR subdivides rock masses into five different classes.

3.4 Stability Graph Method

In 1981, K. Mathews, E. Hoek, E. Wyllie, and S.B.V. Stewart of Golder Associates presented a new empirical approach specifically for application in open stope design based on the NGI rock mass classification system. Along with a slight modification of the classification system, Mathews et al. advocated that the stability of each surface could be analyzed separately. In the Stability Graph Method, the stability number, N, is plotted against the shape factor, S of the surface. The stability number N is:

$$N = Q' \times (\text{factor A}) \times (\text{factor B}) \times (\text{factor C})$$

where,

- Q' is the Q of the ngi system with SRF = 1
- factor A is a function of the uniaxial compressive strength of the rock mass and the induced stress at the surface of the opening.
- factor B is a function of the angle of the critical joint set.
- factor C is the surface inclination factor.

The Shape factor S is simply the hydraulic radius of the surface. The hydraulic radius of a surface is its area divided by its perimeter.

Based on 55 case studies, Mathews et al. identified three zones in the Stability Graph. These are the Stable Zone, the Unstable Zone and the Caving Zone. Their definitions were:

- **Stable**: the excavation will stand unsupported with occasional localized ground support to control slabbing.
- **Unstable**: the excavation will experience some localized caving but will tend to form a stable arch. Open stoping is feasible if localized caving can be prevented by modifying extraction sequence, installing cable bolts, etc.
- **Caving**: the excavation will cave and will not stabilize until the void is full.

Further Mathews et al said that transition zones separated the three zones from each other. The Stability Graph is reproduced in Figure 3.

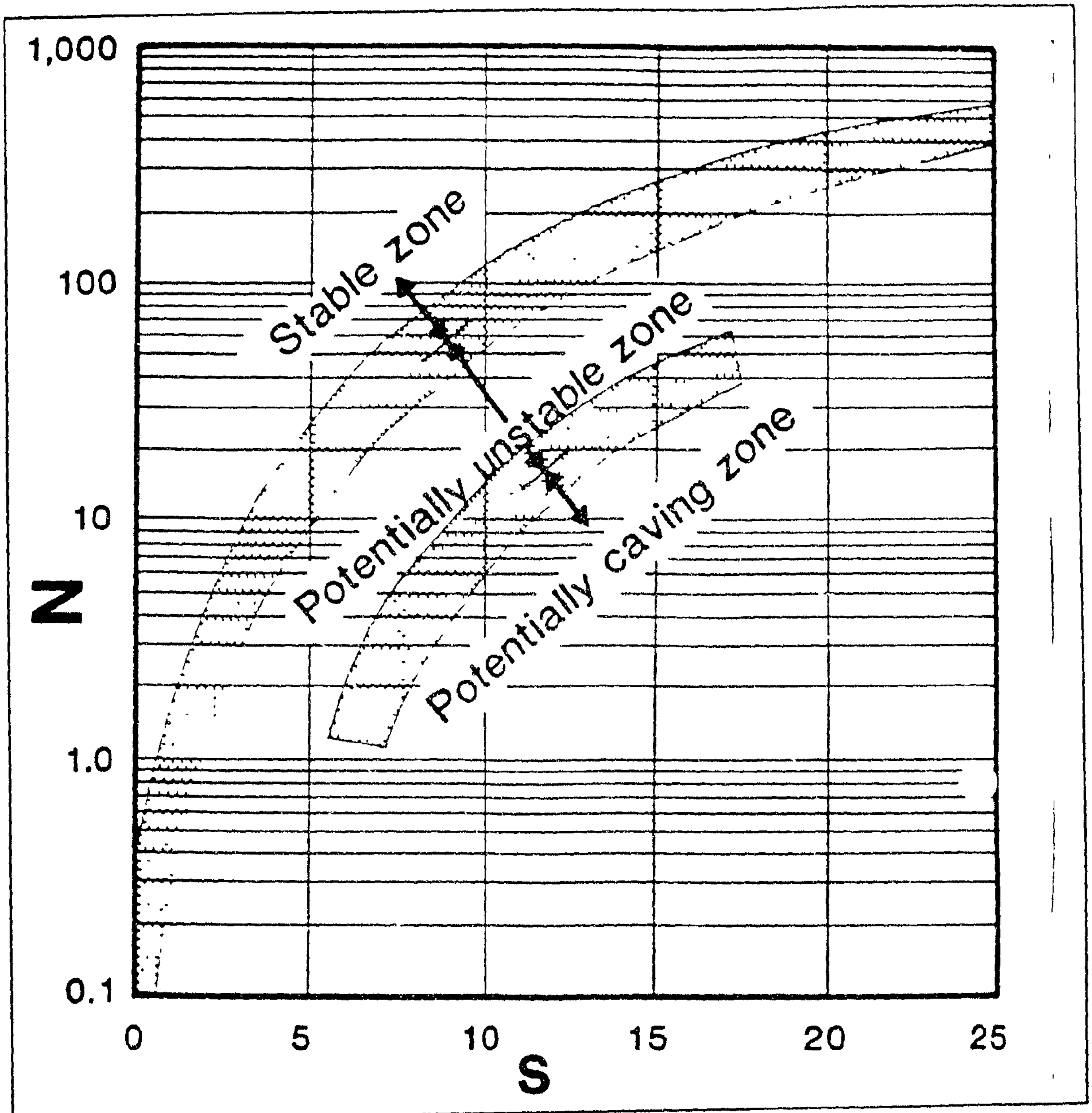
3.5 Modified Stability Graph Method

In 1985, Potvin (in a joint project between the University of British Columbia and Noranda Technology Centre) expanded the database of the Stability Graph to further his studies. From his work, he was able to reassess factors A, B, and C. In addition, he proposed the Modified Stability Graph as a simplification of the Stability Graph. In his latest version of the Modified Stability Graph (see figure 4), the two main zones of stability and instability are divided by a potentially unstable zone. Further, an unsupportable zone characterized by its large shape factor and very low modified stability number has been proposed (but this not shown in the figure).

3.6 Summary

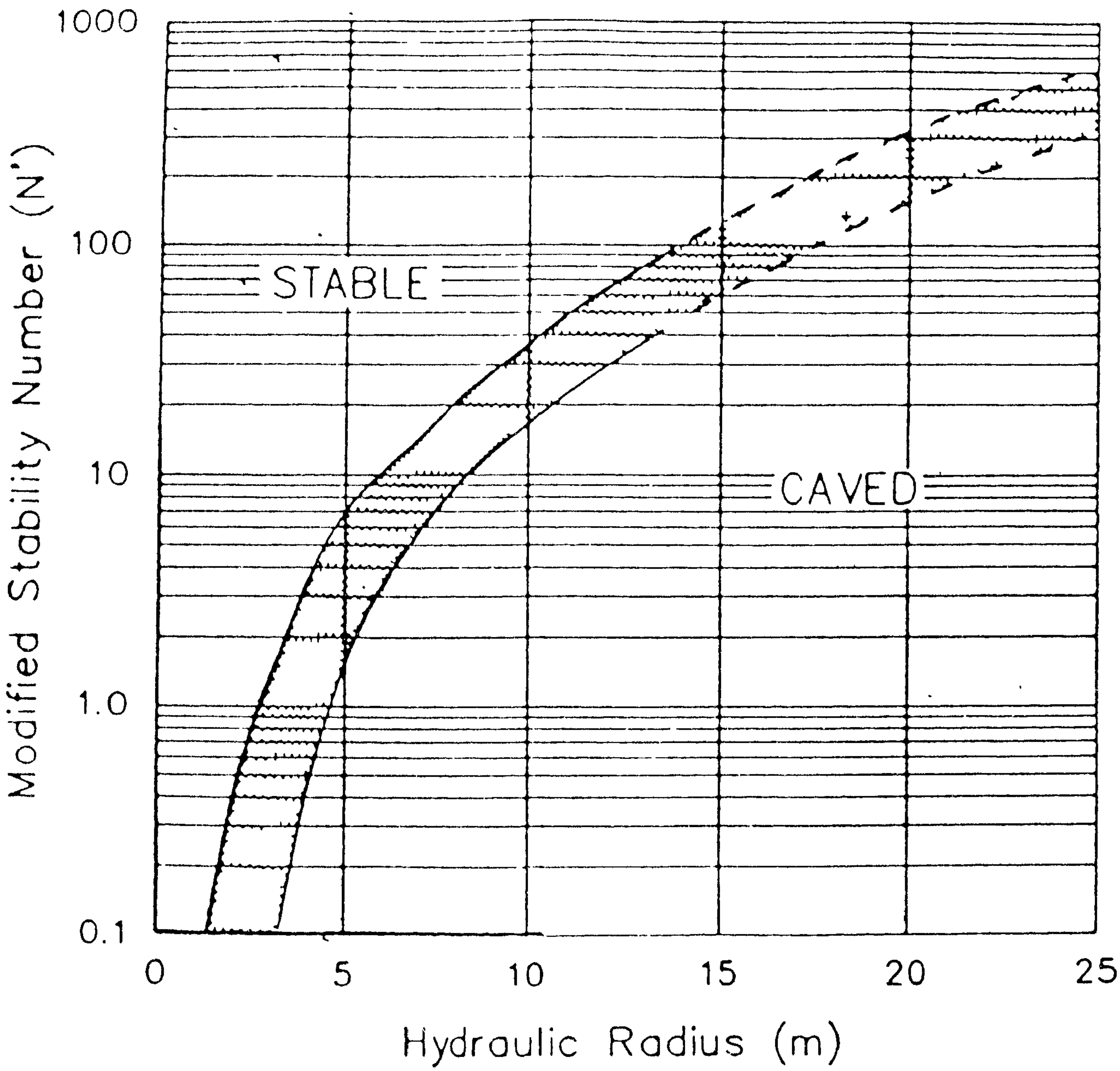
In this section a brief overview of numerical and empirical methods have been presented. Numerical techniques, represented here by the finite element, and boundary element approaches are summarized. In the empirical design methods, brief descriptions of the Q and RMR rock mass classification systems along with an overview of the two stability graph methods are provided.

Figure 3: The Stability Graph. (Mathews et al., 1980)



Relationship between N and S for a stope surface (after Mathews et al, 1980)

Figure 4: The Modified Stability Graph. (Potvin et al., 1988)



Chapter 4 - Case Histories

4.0 Introduction

This section begins with a brief review of the stope design chapter of a CANMET project conducted at Thompson Mine in 1990. After, a series of individual case studies in which a combination of instrumentation, numerical modelling and empirical modelling were used are presented.

4.1 The Existing Design Philosophy - CANMET 1990 PROJECT

As part of a CANMET project completed in 1990 at INCO - Manitoba Division, a philosophy of stope design was developed. This philosophy advocated the use of past experience to calibrate both numerical and empirical methods of stope design (The open stope design philosophy was summarized in a flow sheet and is shown in figure 5). These models were then to be used as predictive and design tools to assess practicable stope dimensions and ground support levels within an economic framework. At the time of the report, the Stability Graph and its derivative, the Modified Stability Graph methods were two empirical methods available to the stope design engineer. The use of numerical models were mostly limited to back analysis within a distinct mining domain.

Since the stability of stope surfaces are directly related to the economic success of open stoping, an extensive applications of the two Stability Graph methods was undertaken. From visual monitoring of 126 mined stopes, the study found that hangingwall slough occurred in three out of ten stopes, back instability in one out of ten stopes, and footwall slough in about one in fourteen stopes. The Stability Graph Method and the Modified Stability Graph Method were applied to these case studies for back analysis in order to evaluate their suitability for INCO's Manitoba Division. For the hangingwall, the CANMET study concluded that while the Stability Graph displayed some validity, the Modified Stability Graph was not within an acceptable correlation range (see figures 6 and 7). In the design of stope backs, the reverse was true (see figures 8 and 9). That is, the Modified Stability Graph was found acceptable but the Stability Graph was not. Finally, both graphs were found to be reasonable for rough design of footwall surfaces (figures 10 and 11). For further details, refer to CANMET project NO. 4-9147-3.

The apparent incompatibility of the two methods when they were so closely related was cause for further examination. In this thesis, only the stope back portion of the CANMET study was scrutinized. Upon closer analysis, a limitation to the CANMET study with respect to its application of the two Stability Graph methods beyond those mentioned by the study itself was discoverd. It was found that for several series of stopes mined within the same domain possessed the same (modified) stability number. This lack of variability could be the result of limited access to previously mined stopes. To obtain data from inaccessible stopes, the study did the best that was possible. That is, line surveys conducted nearby were probably averaged for the area if several surveys were conducted or extrapolated for a series of neighbouring stopes if only one line survey was conducted. This practice reduces the classification ability of both Stability Graph methods and should be avoided whenever possible. As both Stability Graph methods require the independent characterization of each stope surface, it is recommended that line surveys (or block surveys) or structural data be conducted for each surface. Further, a geological map of each surface should be kept for future references to expand the potential database.

Figure 5: CANMET Stope Design Flowsheet (1990).

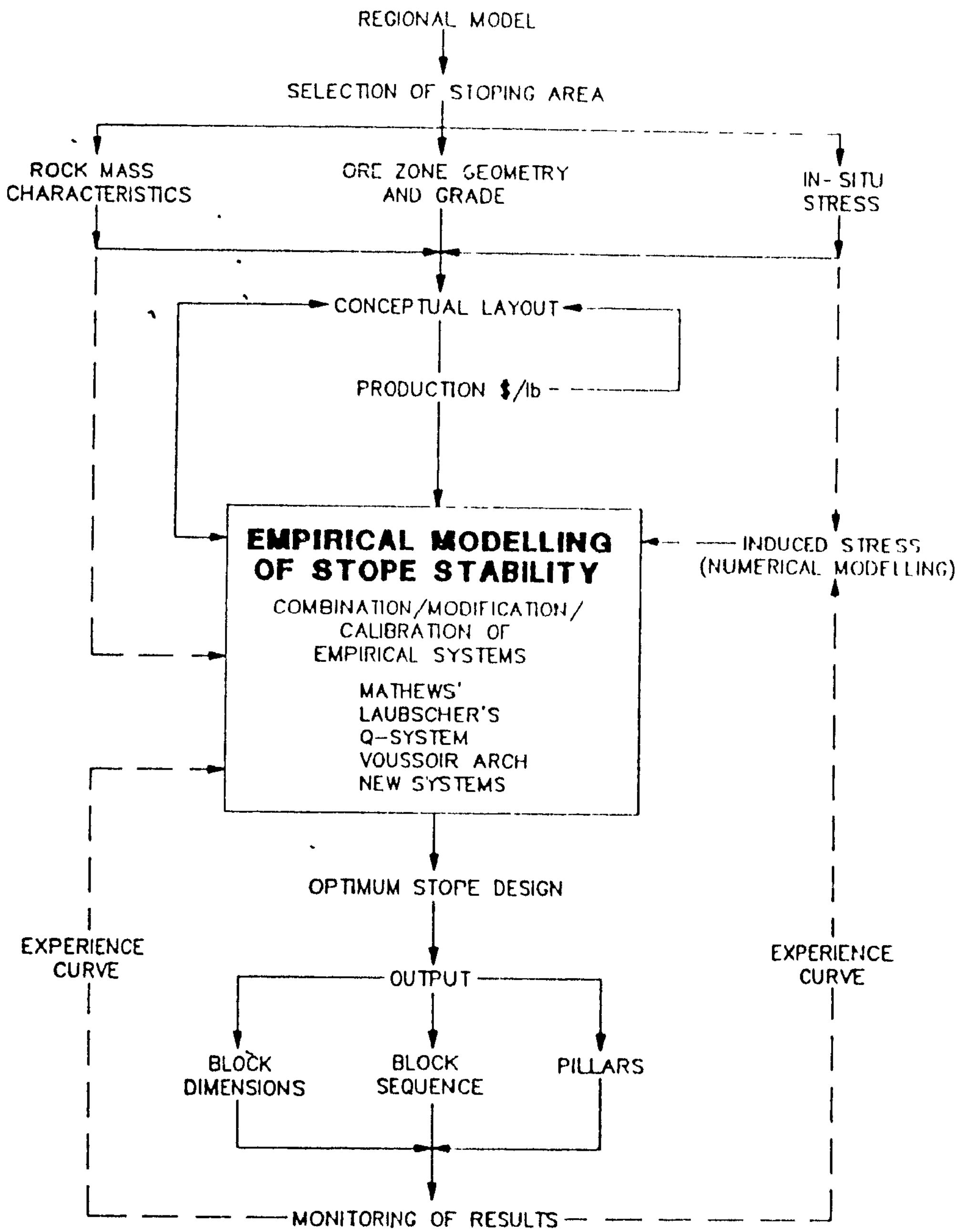


Figure 6: CANMET Stability Graph - Hangingwall (1990).

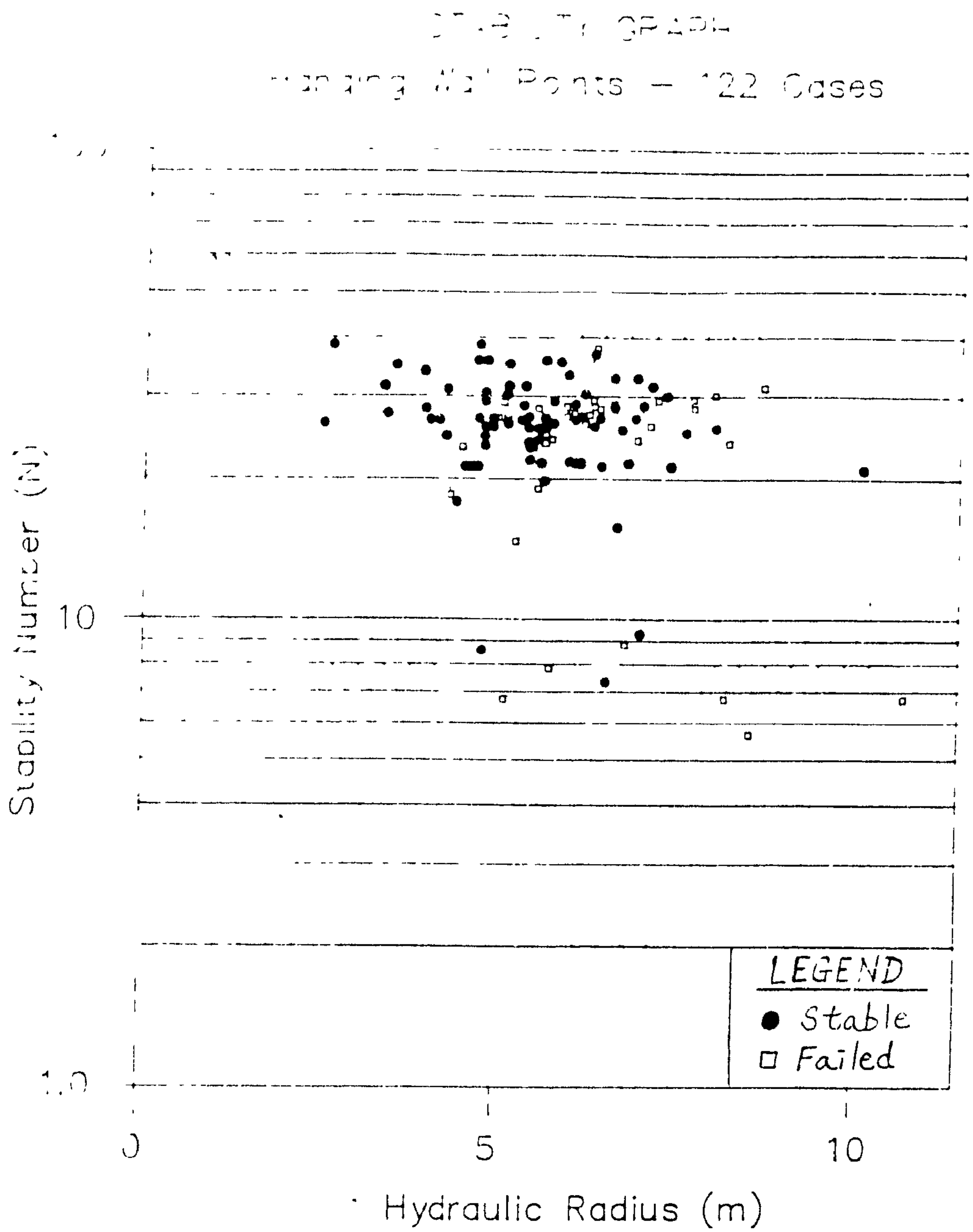


Figure 7: CANMET Modified Stability Graph - Hangingwall (1990).

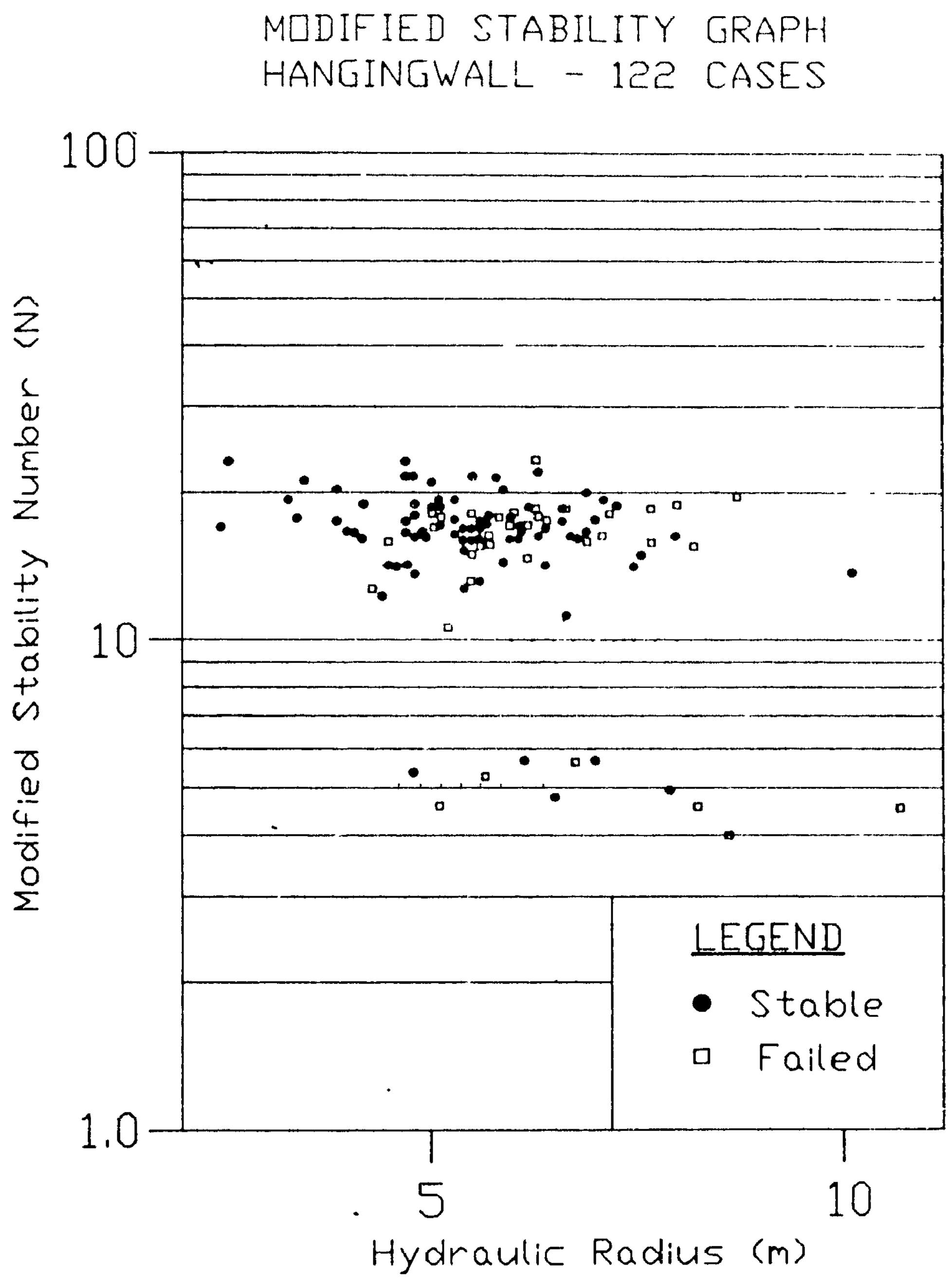


Figure 8: CANMET Stability Graph - Back (1990).

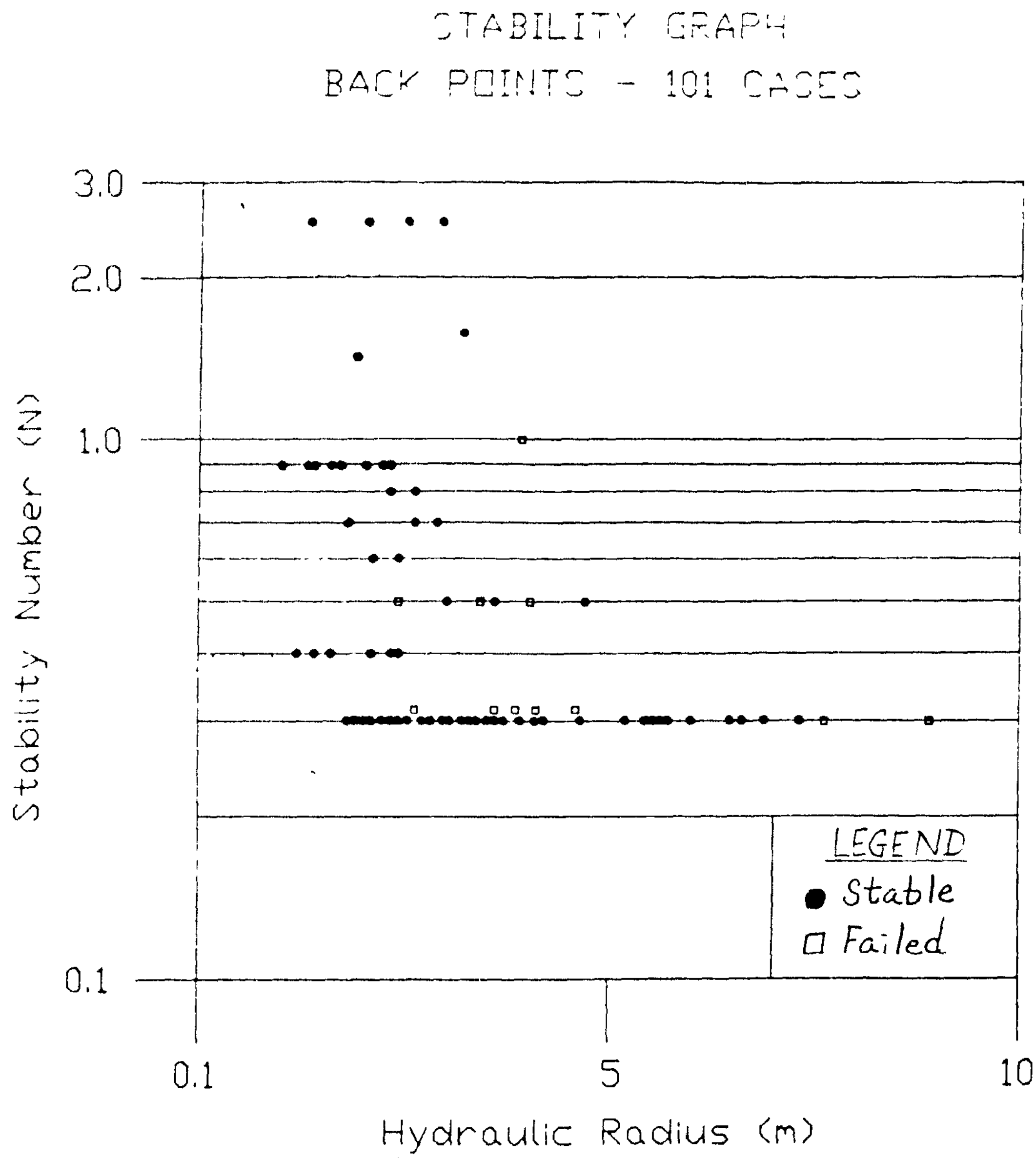


Figure 9: CANMET Modified Stability Graph - Back (1990).

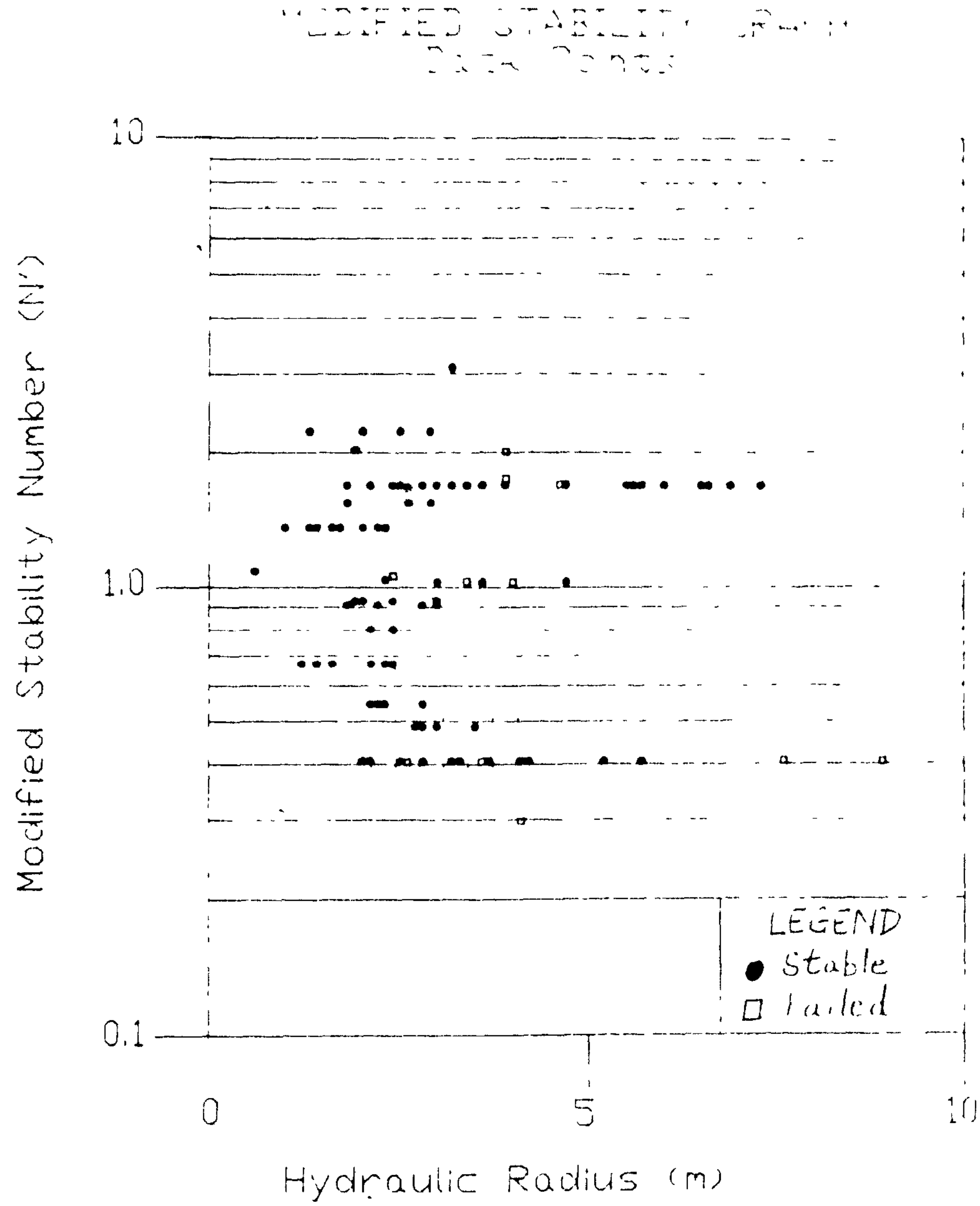


Figure 10: CANMET Stability Graph - Footwall (1990).

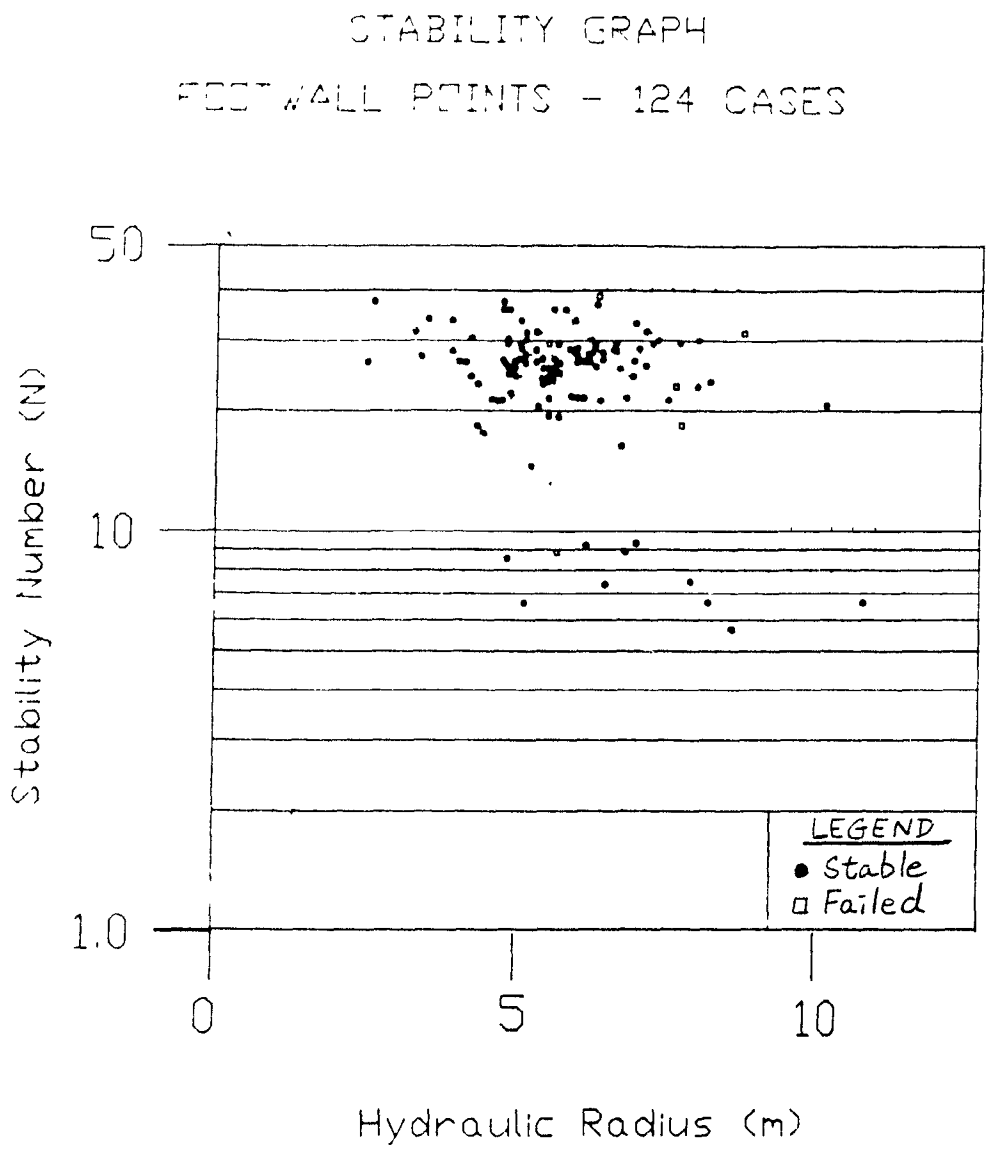
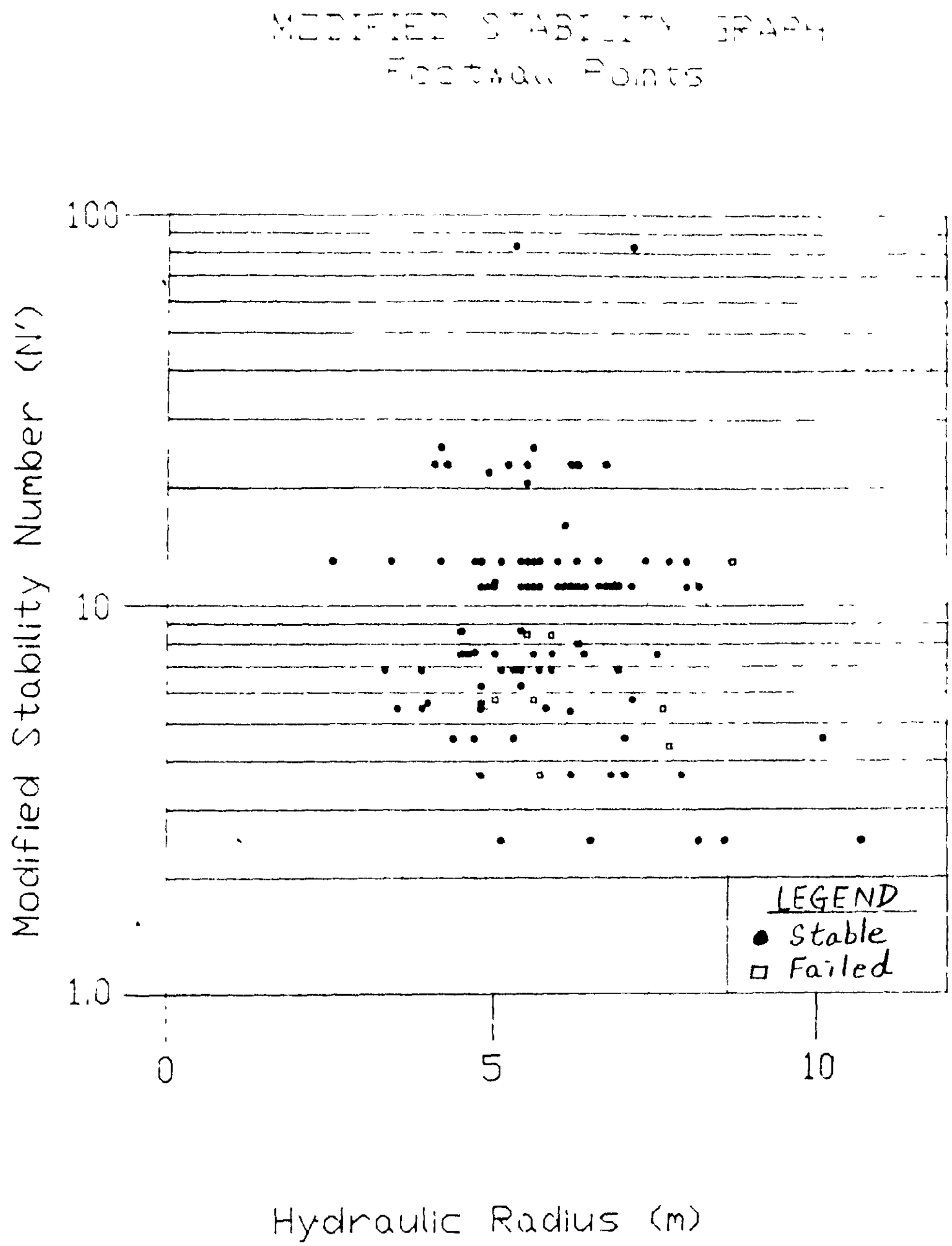


Figure 11: CANMET Modified Stability Graph - Footwall (1990).



4.2 Birchtree 1900 Level, 83 Footwall Drift

Background

The purpose of this investigation is to study the behaviour of mechanically anchored rock bolts and inferred the behaviour of the surrounding rock mass of a drift via the instrumented rock bolts and numerical simulation of mining.

The 83 footwall haulage drift on 1900 level of the Birchtree Mine is designed to operate over a long life span and its stability is of interest to the mining of the 83 ore body. The drift encircles the 83 orebody and its dimensions are 4.5 m (15 ft) wide by 3.6 m (12 ft) high. A line survey in the instrumented area was conducted. From the summary of the rock mass classification chart in Table VII, It can be seen that the footwall quartzites are fairly competent rocks with a Q′ value of 16.49. Primary back support in the drift was with 2.4 m (8′) long 18 mm (3/4") diameter mechanical anchor rock bolts in a staggered 1.2 m by 1.2 m (4′ by 4′) pattern. The walls were reinforced with 1.8 m (6′) rockbolts and split sets. Minor inflow of water in the area was noted.

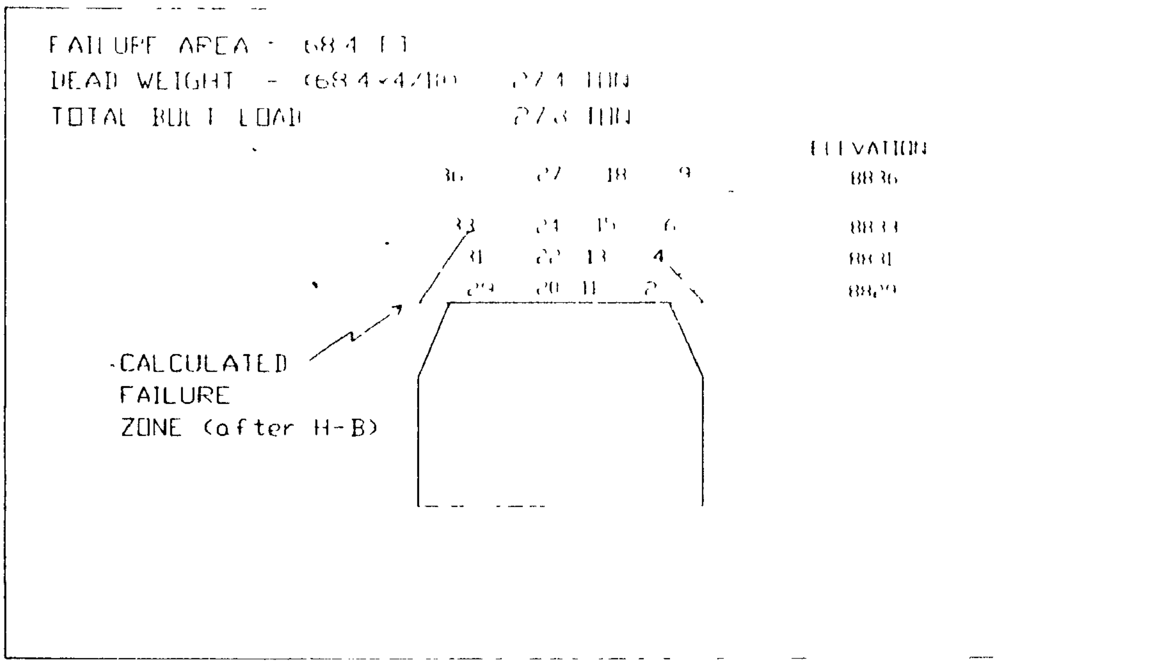
Instrumentation

Four vibrating wire mechanical anchor rockbolts were installed across the back of the drift within 3 m (10′) of the face on May 10th, 1991. The instrumented rock bolts were torqued to approximately 150 ft-lbs. From east to west are bolt number 1, 2, 3, and 4 (see figure 12 for cross section view of drift). A plan view of the site is given in figure 13. The dashed lines in figure 13 across the drift represent the mining advance when the strain on the rockbolts were measured.

Table VII: Rock Mass Classification - Birchtree Mine 83[9]

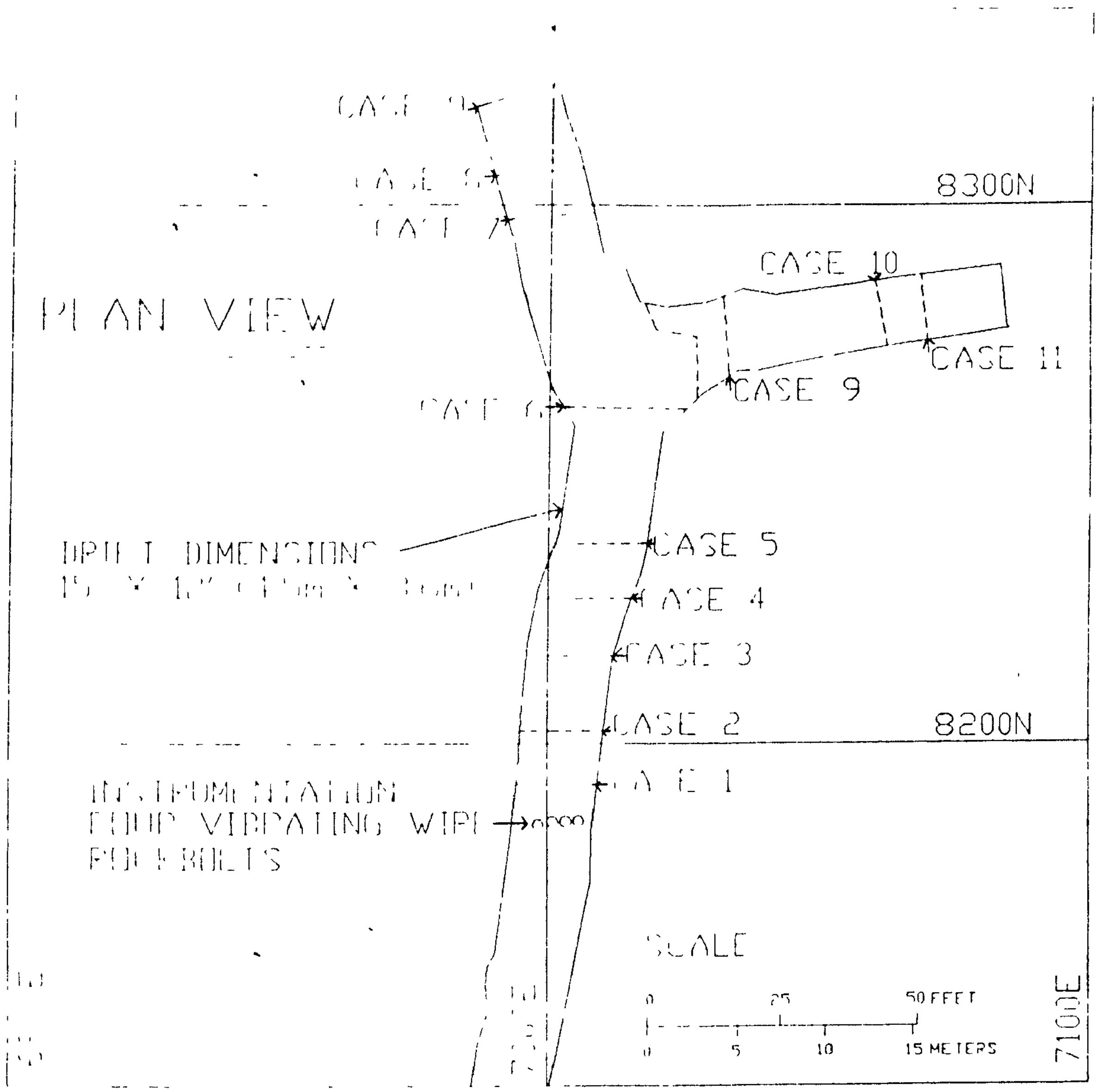
DOMAIN	JN	LN	#JT	JV	RQD	JR	JA	Q'	RMR
FW SCHIST	15	85.3	732	8.6	87	2.2	1.21	10.7	62
HW SCHIST	15	68.9	646	9.4	84	2.3	0.93	13.6	70
FW QUARTZITE	15	26.2	220	8.4	87	2.6	0.90	16.5	69
ORE	15	39.4	421	11	80	2.4	1.22	10.5	63

Figure 12: Cross Section 83 Footwall Drift Birchtree Mine 1900 Level. (Looking North) Calculated failure zone is plotted based on hoek-brown failure criteria. Note the serendipidy between the dead weight of failure zone and total load on four instrumented bolts.



[9] Mines Research, Geomechanics Group 1991.

Figure 13: Plan View 83 Footwall Drift Birchtree Mine 1900 Level. Instrumented bolts are represented by dotted circles. The dash lines represent the tunnel advance during which the strains of the rock bolts were read and a corresponding numerical model was constructed.



Results

Twenty five sets of measurements were recorded for the rockbolts through March 3rd, 1992. Monitoring frequency was a function of mining advance and the distance of this mining advance. Results are found in Table VIII, and figure 14. The strains of the four rockbolts ranged from 607 to 1525 microstrain (0.06%-0.15% strain), and were well within their operating capacities of 0.8% yield strain on March 3rd, 1992.

Throughout the monitoring period, small strain decreases in all four rockbolts were detected at least once. This anchor slippage (strain decrease) was usually followed by reanchoring and loading (strain increase) and was not a stability concern. The exception was bolt #3, which had slipped immediately following installation and had never established a solid anchor in the rock.

Numerical Modelling

A series of three dimensional elastic boundary element numerical models were constructed to estimate the rock mass response around the instrumentation due to the tunnel excavation sequence. Far field stresses were the average values found in the Canadian Shield, whereas the principle stress directions were derived from the set of stress measurements conducted in the Thompson Nickel Belt in previous studies.[10] From figure 15 and table IX, it can be seen that most of the calculated displacements (numerical model) for the back of the drift occurred by case #1 (i.e. before the bolts were installed eight feet from the face). This is contrary to the recorded behaviour as strains on the rockbolts (three of them anyways) increased progressively with time and drift advance. The behaviour of the rockbolts were consistent with viscous behaviour and is beyond the capabilities of current numerical simulation techniques. The model also indicated increases in major and minor principal stresses became smaller as the drift advanced away from the

[10] see appendix C for details, after Herget 1990

instrumented bolts (see figure 16, and Table X). Since the current model can not simulate non-elastic behaviour or time related behaviour of the rock mass, it should be expected that model behaviour did not correspond with what the instrumentation recorded in the field.

Table VIII: Monitoring Results of Four Instrumented Bolts - Birchtree Mine 83 Footwall Drift. (above) Total Microstrain, change in microstrain, and the corresponding load in tons are given for the corresponding period.

Figure 14: Plot of Microstrain vs. Time of Instrumented Rockbolts Birchtree Mine 83 Footwall Drift. (below)

				BOLT #1				BOLT #2				BOLT #3				BOLT #4			
Date	Case No	Days	Effective Dist. to Face (ft)	Read	MStrain	Delta MStrain	Load (tons)	Read	MStrain	Delta MStrain	Load (tons)	Read	MStrain	Delta MStrain	Load (tons)	Read	MStrain	Delta MStrain	Load (tons)
No Load Laboratory Calibration				2480	0		0	2520	0		0	2476	0		0	2497	0		0
10-May-91	1	0	6	4123	630		4.05	4052	593		3.81	4150	672		4.32	4292	708		4.55
10-May-91	2	0.5	18	4373	726	96	4.67	4278	661	66	4.36	4137	666	5	4.29	3930	565	-143	3.63
11-May-91	3	1	31	4663	837	111	5.38	4423	737	56	4.74	3650	471	-195	3.03	4216	678	113	4.38
11-May-91	4	1.5	42	4728	862	25	5.54	4478	758	21	4.88	3679	483	12	3.10	4269	699	21	4.49
13-May-91		3	42	4836	904	42	5.81	4590	802	43	5.15	3710	495	12	3.18	4372	739	41	4.75
14-May-91	5	4	52	4904	929	25	5.98	4640	821	19	5.28	3745	509	14	3.27	4391	747	7	4.80
15-May-91		5	63	4965	953	23	6.13	4697	843	22	5.42	3796	530	21	3.41	4418	757	11	4.87
16-May-91	6	6	75	5086	999	46	6.42	4782	878	33	5.63	3801	532	1	3.42	4454	771	14	4.96
17-May-91	7	7	113	5229	1054	55	6.78	4845	900	24	5.79	3716	498	34	3.20	4487	783	11	5.01
21-May-91	8	11	121	5586	1191	137	7.66	5031	972	72	6.25	3620	458	39	2.95	4526	801	18	5.15
24-May-91	9	14	136	5818	1280	89	8.23	5128	1010	38	6.49	3620	459	0	2.95	4545	807	7	5.19
28-May-91		18	141	5981	1342	62	8.63	5215	1044	34	6.71	3609	455	-4	2.92	4553	811	3	5.21
30-May-91		20	146	5972	1339	-3	8.61	5253	1058	15	6.81	3606	453	-1	2.92	4572	818	7	5.26
07-Jun-91	10	28	151	5918	1318	-21	8.47	5384	1109	51	7.13	3620	458	6	2.95	4612	834	16	5.36
13-Jun-91		34	156	6018	1356	38	8.72	5447	1131	22	7.28	3651	471	12	3.03	4643	846	12	5.44
14-Jun-91		35	161	6029	1360	4	8.75	5452	1135	4	7.30	3653	472	1	3.04	4653	850	4	5.47
25-Jun-91		46	166	6195	1424	64	9.16	5538	1169	33	7.51	3699	491	18	3.16	4687	863	13	5.55
28-Jun-91		49	171	6233	1439	15	9.25	5591	1189	21	7.65	3717	498	7	3.20	4663	854	9	5.49
09-Jul-91		60	176	6341	1480	41	9.52	5696	1230	41	7.91	3751	512	14	3.29	4740	884	30	5.69
22-Jul-91	11	73	181	6357	1486	6	9.56	5720	1239	9	7.97	3776	522	10	3.36	4778	899	15	5.78
13-Aug-91	12	95	186	6436	1516	30	9.75	5752	1251	12	8.05	3826	542	20	3.46	4859	931	32	5.99
27-Aug-91	13	109	191	6453	1525	8	9.81	5748	1250	-2	8.04	3856	554	12	3.56	4886	942	11	6.00
09-Oct-91		152	196	6270	1453	-72	9.34	5700	1235	-15	7.94	3906	574	20	3.69	4903	948	7	6.10
04-Dec-91		208	201	6339	1479	26	9.51	5738	1246	11	8.01	3972	600	26	3.86	4936	962	14	6.19
03-Mar-92		298	206	6437	1517	38	9.75	5707	1234	-12	7.94	3988	607	6	3.90	4954	969	6	6.23

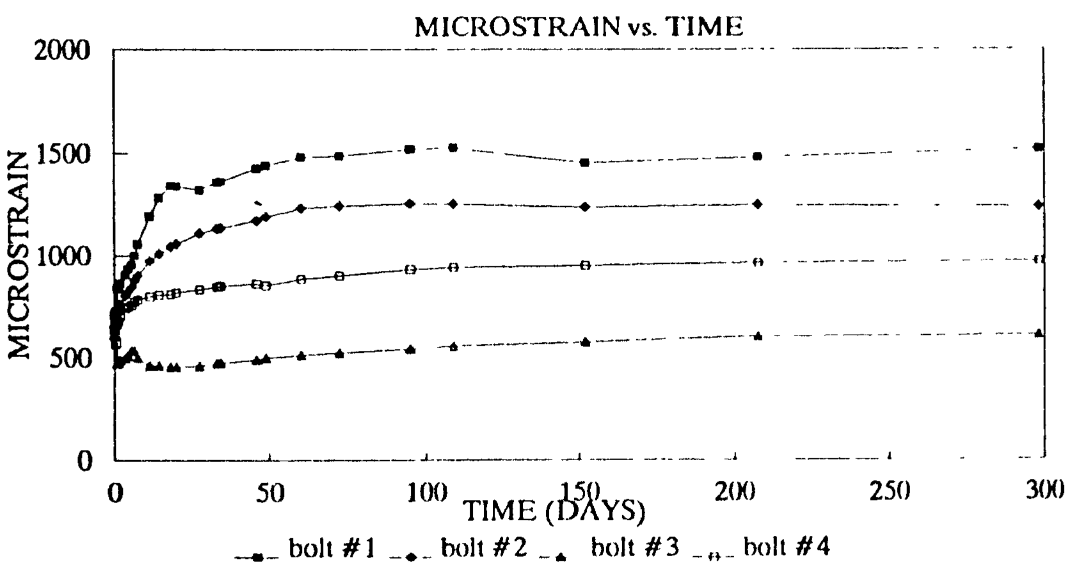


Table IX: Comparison of Displacement Between Numerical Model and Instrumentation - Birchtree Mine 83 Footwall Drift. Total displacements and changes in displacements are shown. (above)

Figure 15: Plot of Model Displacement vs. Measured Displacements Birchtree Mine 83 Footwall Drift. (below)

CASE No	Effective Dis to Face (FT)	BOLT #1 TOTAL BOLT Elong (mm)	BOLT #1 TOTAL MODEL Disp (mm)	BOLT #1 DELTA BOLT Elong (mm)	BOLT #1 DELTA MODEL Disp (mm)	BOLT #2 TOTAL BOLT Elong (mm)	BOLT #2 TOTAL MODEL Disp (mm)	BOLT #2 DELTA BOLT Elong (mm)	BOLT #2 DELTA MODEL Disp. (mm)	BOLT #3 TOTAL BOLT Elong (mm)	BOLT #3 TOTAL MODEL Disp (mm)	BOLT #3 DELTA BOLT Elong (mm)	BOLT #3 DELTA MODEL Disp (mm)	BOLT #4 TOTAL BOLT Elong (mm)	BOLT #4 TOTAL MODEL Disp (mm)	BOLT #4 DELTA BOLT Elong (mm)	BOLT #4 DELTA MODEL Disp (mm)
1	8	0.00	1.14			0.00	1.10			0.00	0.54			0.00	1.72		
2	18	0.23	1.21	0.23	0.07	0.21	1.26	0.21	0.17	-0.01	0.67	-0.01	0.12	-0.35	1.59	-0.35	-0.13
3	31	0.50	1.23	0.27	0.03	0.35	1.34	0.14	0.07	-0.49	0.93	-0.48	0.26	-0.07	1.60	0.27	0.01
4	42	0.57	1.26	0.06	0.03	0.40	1.41	0.05	0.07	-0.46	1.15	0.03	0.23	-0.02	1.63	0.05	0.03
	42	0.67	1.26	0.10	0.03	0.51	1.41	0.11	0.07	-0.43	1.15	0.03	0.23	0.08	1.63	0.10	0.03
5	52	0.73	1.27	0.06	0.00	0.56	1.43	0.05	0.01	-0.40	1.18	0.03	0.03	0.10	1.64	0.02	0.01
	63	0.79	1.27	0.06	0.00	0.61	1.43	0.05	0.01	-0.34	1.18	0.05	0.03	0.12	1.64	0.03	0.01
6	75	0.90	1.27	0.11	0.00	0.69	1.44	0.08	0.01	-0.34	1.22	0.00	0.03	0.16	1.69	0.[illegible]3	0.05
7	113	1.03	1.28	0.13	0.01	0.75	1.46	0.06	0.03	-0.42	1.33	-0.08	0.11	0.18	1.91	0.03	0.22
8	121	1.37	1.28	0.33	0.00	0.92	1.46	0.18	-0.00	-0.52	1.31	-0.09	-0.02	0.23	1.84	0.04	-0.08
9	136	1.58	1.29	0.22	0.01	1.02	1.50	0.09	0.04	-0.52	1.46	0.00	0.15	0.24	2.07	0.02	0.24
	141	1.74	1.29	0.15	0.01	1.10	1.50	0.08	0.04	-0.53	1.46	-0.01	0.15	0.25	2.07	0.01	0.24
	146	1.73	1.29	-0.01	0.01	1.13	1.50	0.04	0.04	-0.53	1.46	-0.00	0.15	0.27	2.07	0.02	0.24
10	151	1.68	1.25	-0.05	-0.04	1.26	1.38	0.12	-0.12	-0.52	1.09	0.01	-0.37	0.31	1.72	0.04	-0.36
	156	1.77	1.25	0.09	-0.04	1.31	1.38	0.05	-0.12	-0.49	1.09	0.03	-0.37	0.34	1.72	0.03	-0.36
	161	1.78	1.25	0.01	-0.04	1.32	1.38	0.01	-0.12	-0.49	1.09	0.00	-0.37	0.35	1.72	0.01	-0.36
	166	1.94	1.25	0.16	-0.04	1.40	1.38	0.08	-0.12	-0.44	1.09	0.05	-0.37	0.38	1.72	0.03	-0.36
	171	1.97	1.25	0.04	-0.04	1.45	1.38	0.05	-0.12	-0.42	1.09	0.02	-0.37	0.36	1.72	-0.02	-0.36
	176	2.07	1.25	0.10	-0.04	1.55	1.38	0.10	-0.12	-0.39	1.09	0.03	-0.37	0.43	1.72	0.07	-0.36
11	181	2.09	1.23	0.01	-0.02	1.57	1.32	0.02	-0.06	-0.37	0.79	0.02	-0.30	0.47	1.59	0.04	-0.12
12	186	2.16	1.23	0.07	0.00	1.61	1.31	0.03	-0.01	-0.32	0.73	0.05	-0.06	0.55	1.59	0.06	0.00
13	191	2.18	1.23	0.02	-0.00	1.60	1.32	-0.00	0.02	-0.29	0.84	0.03	0.11	0.57	1.61	0.03	0.01
	196	2.01		-0.18		1.56		-0.04		-0.24		0.05		0.59		0.02	
	201	2.07		0.06		1.59		0.03		-0.17		0.06		0.62		0.03	
	206	2.16		0.09		1.56		-0.03		-0.16		0.02		0.64		0.02	

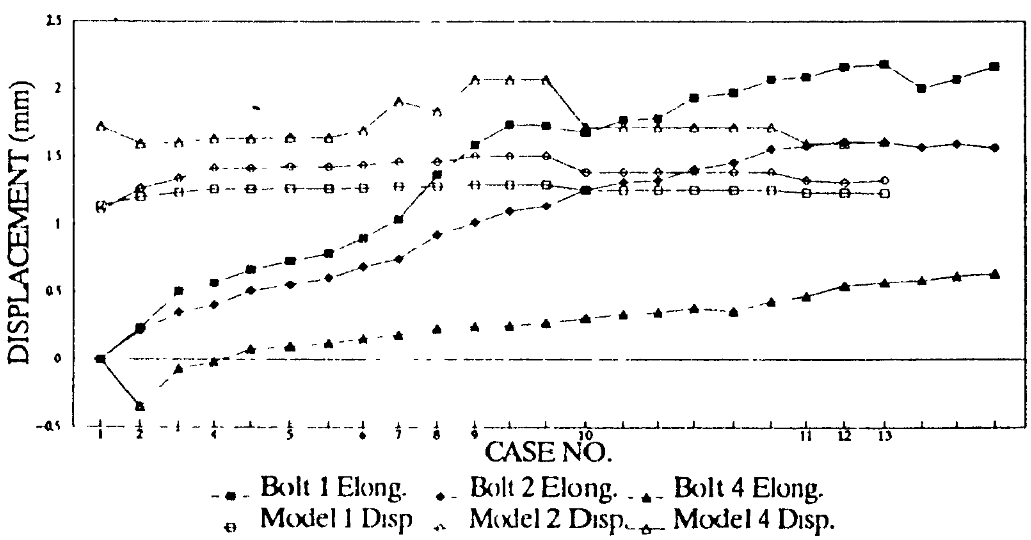


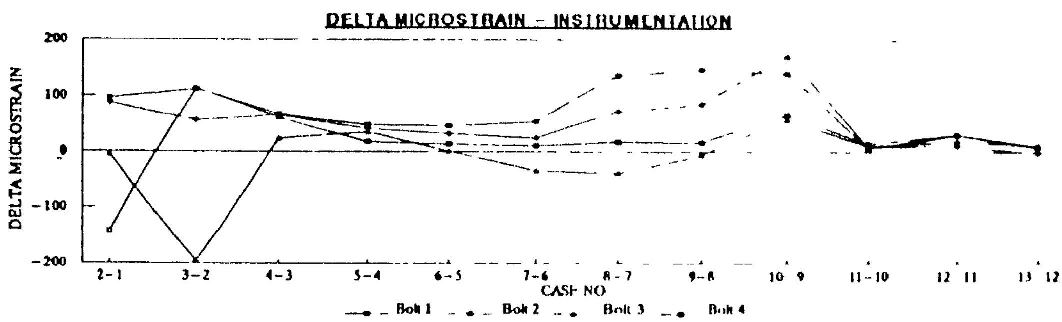
Table X: Change in Microstrain of Rockbolts vs. Change in Calculated Major and Minor Principal Stress - Birchtree Mine 83 Footwall Drift. Set numbers corresponds with the bolt numbers. (above)

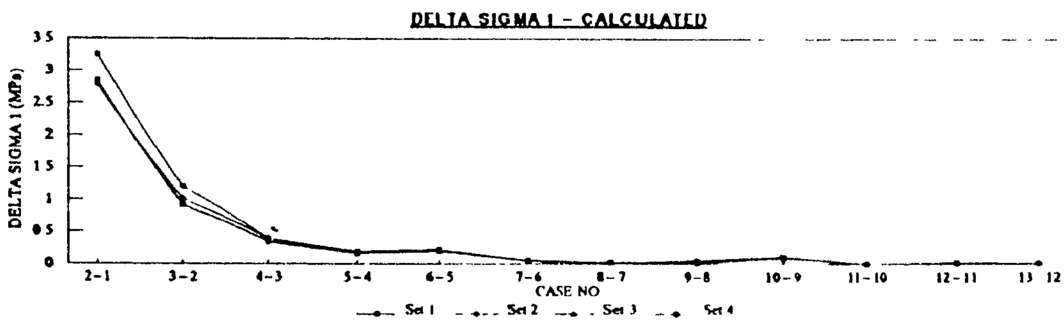
Figure 16: Plot of Changes in Microstrain of Rockbolts, and Calculated Major and Minor Principal Stresses Birchtree Mine 83 Footwall Drift. First graph plots changes in microstrain of each of the rockbolts. The second and third graph plots the changes in the Major and Minor principal stresses over time.

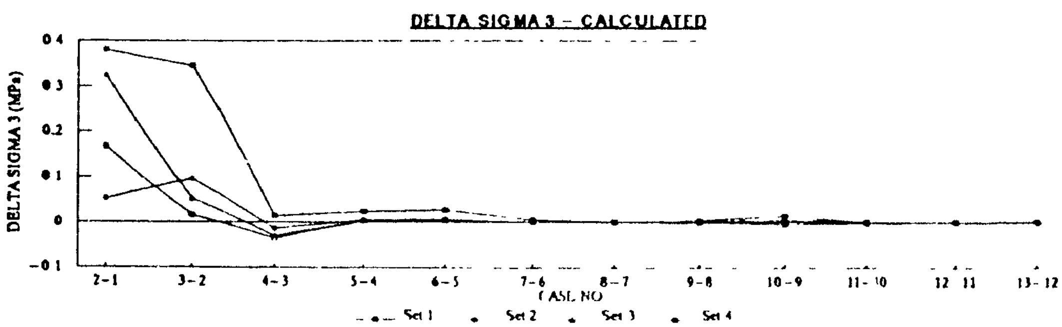
CHANGE IN MICROSTRAIN OF ROCKBOLTS, S1 AND S3 FOR SELECTED POINT GRIDS

CASE NO	DELTA MICROSTRAIN				DELTA SIGMA 1 (MPa)				DELTA SIGMA 3 (MPa)			
	Bolt 1	Bolt 2	Bolt 3	Bolt 4	Set 1	Set 2	Set 3	Set 4	Set 1	Set 2	Set 3	Set 4
2-1	96	88	-5	-143	3 25	2 78	2 85	2 84	0 38	0 05	0 32	0 17
3-2	111	56	-195	113	1 20	1 01	0 92	0 92	0 35	0 10	0 05	0 02
4-3	67	65	24	61	0 39	0 39	0 34	0 33	0 01	-0 01	-0 03	-0 04
5-4	49	41	35	18	0 19	0 17	0 16	0 15	0 02	0 01	0 00	0 00
6-5	46	33	1	14	0 22	0 21	0 19	0 19	0 03	0 01	0 00	0 01
7-6	55	24	-34	11	0 05	0 05	0 05	0 06	0 01	0 00	0 00	0 00
8-7	137	72	-39	18	0 02	0 02	0 02	0 03	0 00	0 00	0 00	0 00
9-8	148	86	-6	17	0 06	0 04	0 02	0 01	0 00	0 00	-0 00	0 00
10-9	141	172	58	66	0 12	0 09	0 08	0 09	0 01	0 00	0 00	-0 00
11-10	6	9	10	15	0 00	0 00	0 00	0 00	0 00	0 00	0 00	-0 00
12-11	30	12	20	32	0 02	0 02	0 01	0 01	0 00	0 00	-0 00	-0 00
13-12	8	-2	12	11	0 02	0 02	0 02	0 02	0 00	0 00	0 00	0 00

Note Set 1, 2, 3, and 4 represent point grids 3, 12, 21, and 30 at elevation 8830 (or 3 feet above the back)







4.3 Birchtree 1100 Level Sub 2, 124 Complex

Background

This test site was used to compare the performance between cable bolts mounted with Tensmeg gauges and the Cable Strain Measuring Device (CSMD). There was also an attempt to evaluate the adequacy of the support design.

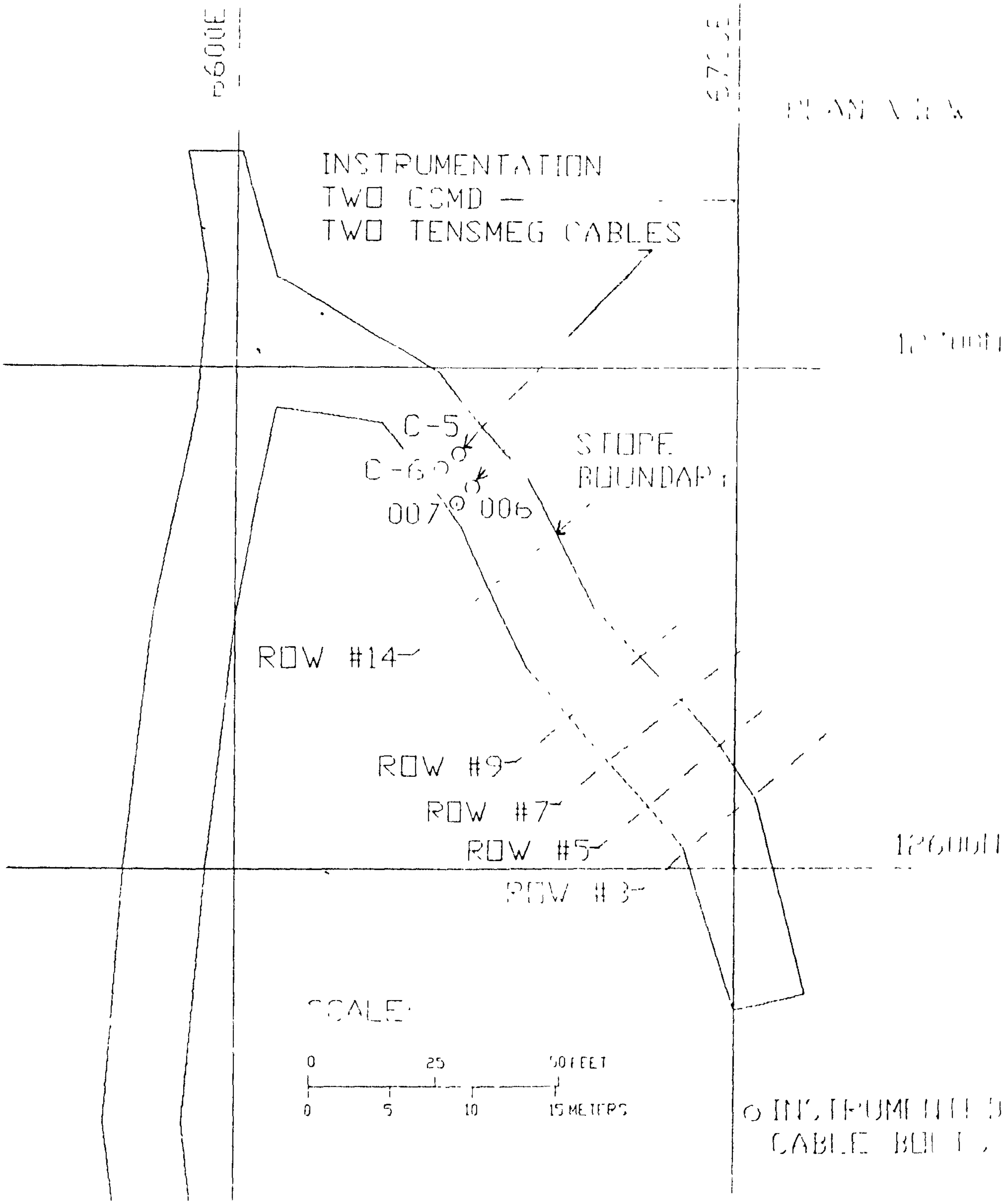
This area was cable bolted to provide additional reinforcement to the back, the drawpoint, and corner pillar (see figure 17). Cable bolts were fanned into the back and southwest wall at a density of 0.41 cable bolts per square meter (seven cables per row with a row spacing of 2.4 meters). The area below sublevel #2 had previously been mined and backfilled with sand and muck. Sixteen meter long up holes were drilled to remove the ore. As stoping progressed from row #1 to row #14, these cables were expected to load/strain.

Line surveys of the 124 complex indicates that there are four joint sets in this area. The rock mass in this area consists of peridotite interbedded within a sulphide matrix and is considered to range from fair to poor in quality (see table XI).

Table XI: Rock Mass Classification Birchtree Mine 124 Complex 1100 Level Subl (conducted by Simon Nickson, compiled by Philip Ng, 1992).

DOMAIN	JN	LN	#JT	JV	RQD	JR	JA	Q'	RMR
IRON FRM.	15	12.2	161	13.2	71	2.4	0.9	12	67
QTZE.	15	12.2	223	18.3	55	2.1	1.3	6	59
PRDT.	15	36.9	363	9.8	83	2.4	1.6	8	52
GNBIOT.	15	9.1	66	7.3	91	2.3	0.9	15	71
SCHBIOT.	15	15.2	129	8.5	87	2.3	1.8	7	52

Figure 17: Plan View of 124 Stope Birchtree Mine 1100 Subl. Dashed-Dot Lines marked the limits of the five step stoping sequence. Instrumented cable bolts are represented by dotted circles.



Instrumentation

Four instrumented cable bolts were used in this study. The cable bolts were placed as part of the support pattern for the area. Two cable strain measuring devices with three wires anchored to each of the two CSMD devices at 10', 20', and 40' were placed one row closer to the stope than the two Supportek's instrumented cable bolts. The Tensmeg gauges were mounted at 5', 10', 15' and 25' of their cable bolts.[11] All four cable bolts were hydraulically tensioned with a barrel wedge plate assembly (a common practice for the division) into place on November 4th, 1991. The locations of these cables are shown in figure 17. The C-6 cable bolt from supportek was angled 70 degrees into the shoulder of the drift whereas the C-5 cable bolt was positioned in the middle of the back. These tensmeg gauge mounted cable bolts were situated approximately 11 m (35 ft) from the open stope. The Tensmeg instrumented cable bolts required 63 mm diameter holes (2.5 in) to accommodate the mounted strain gauges. The CSMD were placed 2.4 m closer to the stope and inserted in regular 51 mm (2 in) diameter holes.

The cable bolted area acted as a drawpoint for this stope. Fourteen rows of blast holes were drilled to mine this stope. Mining proceeded in a slot and slash fashion from row #1. Rows #3 to #1 were blasted into the initial slot. The next blasts took rows #4 and #5. This was followed by a blast up to row #7, a blast up to row #9, and the final blast which took row #10 to row #14 (refer to figure 17).

Throughout this period, the strain of these cable bolts were recorded and a visual check of the area was conducted to assess local ground conditions. The area was determined to be stable during and well after the mining life of this stope. Monitoring was discontinued on February 17th, 1992. Subsequent stoping from sub #2 to sub #1 eliminated

[11] The distances such as 5', 10', 15' and 25' refer to the distance these gauges were mounted from the end protruding from the hole or the 0' point.

future access to the these instrumented cable bolts.

Results

The CSMDs did not detect any measurable strain response to mining activities even though they were located closer to the stope. Their results are not shown here because it is believed that these CSMDs malfunctioned due to excessive friction among the three wires within the pvc tube. It was determined that two wires is the maximum allowable number of wires within the current pvc tube size of 6 mm.

The Supportek instrumentation recorded small strain in most sections of the cable bolts. The tensmeg gauges performed adequately even though damaged had occurred to the 25' gauge on cable C-6. The results for cable bolts C-5 and C-6 are shown in Table XII and figure 18. Although some strain variability along the length of both cable bolts were noted, the gauges indicated that these two cable bolts did not reached half the yield strain of a 5/8" diameter cable bolt. Along with the visual monitoring of the area, the evidences suggest that the cable bolt pattern in this area was much more than adequate.

Table XII: Results of Tensmeg gages at Birchtree Mine 124 1100 Level Sub1.

BIRCHTREE MINE 1100 LEVEL 124 STOPE SUB #1

SUPPORTEK TENSMEG GAUGE MOUNTED CABLE BOLTS

C-5			5FT		10FT		15FT		25FT		
DATE	DAYS	CASE #	READING	MSTRAIN	READING	MSTRAIN	READING	MSTRAIN	READING	MSTRAIN	COMMENTS
04-Nov-91	0	1	3312	0	5070	0	1856	0	3560	0	AFTER TENSIONING
05-Nov-91	1	2	3530	218	5058	-12	2502	646	3090	-470	BLASTED ROW TO #3
07-Nov-91	3	3	3858	546	5138	68	2530	574	3070	-490	BLASTED TO ROW #5
08-Nov-91	4		4440	1128	5348	278	3176	1320	4546	986	BLASTED TO ROW #5
12-Nov-91	8	4	4548	1236	5538	468	3362	1506	5210	1650	BLASTED TO ROW #7
13-Nov-91	9		4658	1346	5354	284	3360	1504	4930	1370	BLASTED TO ROW #7
14-Nov-91	10	5	4717	1405	5450	380	3576	1720	5248	1688	BLASTED TO ROW #9
18-Nov-91	14	6	4970	1658	6082	1012	4122	2266	6073	2513	BLASTED TO ROW #14
21-Nov-91	17		5010	1698	6247	1177	4225	2369	6280	2720	BROW OPEN (REMOTE NEEDED)
25-Nov-91	21		5020	1708	6300	1230	4290	2434	6350	2790	
02-Dec-91	28		5004	1692	6294	1224	4316	2460	6362	2802	
04-Dec-91	30		5038	1726	6325	1255	4352	2496	6407	2847	CLOSE TO BEING MUCKED OUT
09-Dec-91	35		5036	1724	6326	1256	4380	2524	6410	2850	
16-Dec-91	42		5155	1843	6450	1380	4488	2632	6544	2984	
07-Jan-92	64		5046	1734	6335	1265	4443	2587	6432	2872	C-6 WET FROM DRILLING
21-Jan-92	78		5124	1812	6334	1264	4462	2606	6447	2887	
03-Feb-92	91		5142	1830	6328	1258	4480	2624	6457	2897	
17-Feb-92	105		5147	1835	6311	1241	4495	2639	6462	2902	

C-6			5FT		10FT		15FT		25FT		
DATE	DAYS	CASE #	READING	MSTRAIN	READING	MSTRAIN	READING	MSTRAIN	READING	MSTRAIN	COMMENTS
04-Nov-91	0	1	6782	0	3500	0	3448	0	3630	0	AFTER TENSIONING
05-Nov-91	1	2	6775	-7	3363	-137	3610	162	3734	104	BLASTED ROW TO #3
07-Nov-91	3	3	6800	18	4100	600	3290	-158	3530	-100	BLASTED TO ROW #5
08-Nov-91	4		6830	48	4840	1340	4110	662	3688	58	BLASTED TO ROW #5
12-Nov-91	8	4	6876	94	5740	2240	4951	1503	3800	170	BLASTED TO ROW #7
13-Nov-91	9		6862	80	5926	2426	5190	1742	3940	310	BLASTED TO ROW #7
14-Nov-91	10	5	6860	78	5974	2474	5290	1842	3880	250	BLASTED TO ROW #9
18-Nov-91	14	6	6881	99	6872	3372	6042	2594	4470	840	BLASTED TO ROW #14
21-Nov-91	17		6903	121	7245	3745	6166	2718	4585	955	BROW OPEN (REMOTE NEEDED)
25-Nov-91	21		6910	128	7504	4004	6435	2987	4848	1218	
02-Dec-91	28		6900	118	7748	4248	6569	3121	4924	1294	
04-Dec-91	30		6927	145	7768	4268	6595	3147	4945	1315	CLOSE TO BEING MUCKED OUT
09-Dec-91	35		6802	20	7690	4190	6537	3089	4850	1220	
16-Dec-91	42		7040	258	7850	4350	6705	3257	5055	1425	
07-Jan-92	64		4267	-2515	3845	345	2758	-690	N\R		C-6 WET FROM DRILLING
21-Jan-92	78		4130	-2652	5754	2254	4820	1372	N\R		
03-Feb-92	91		6578	-204	7036	3536	5388	1940	N\R		
17-Feb-92	105		6900	118	7614	4114	6544	3096	N\R		

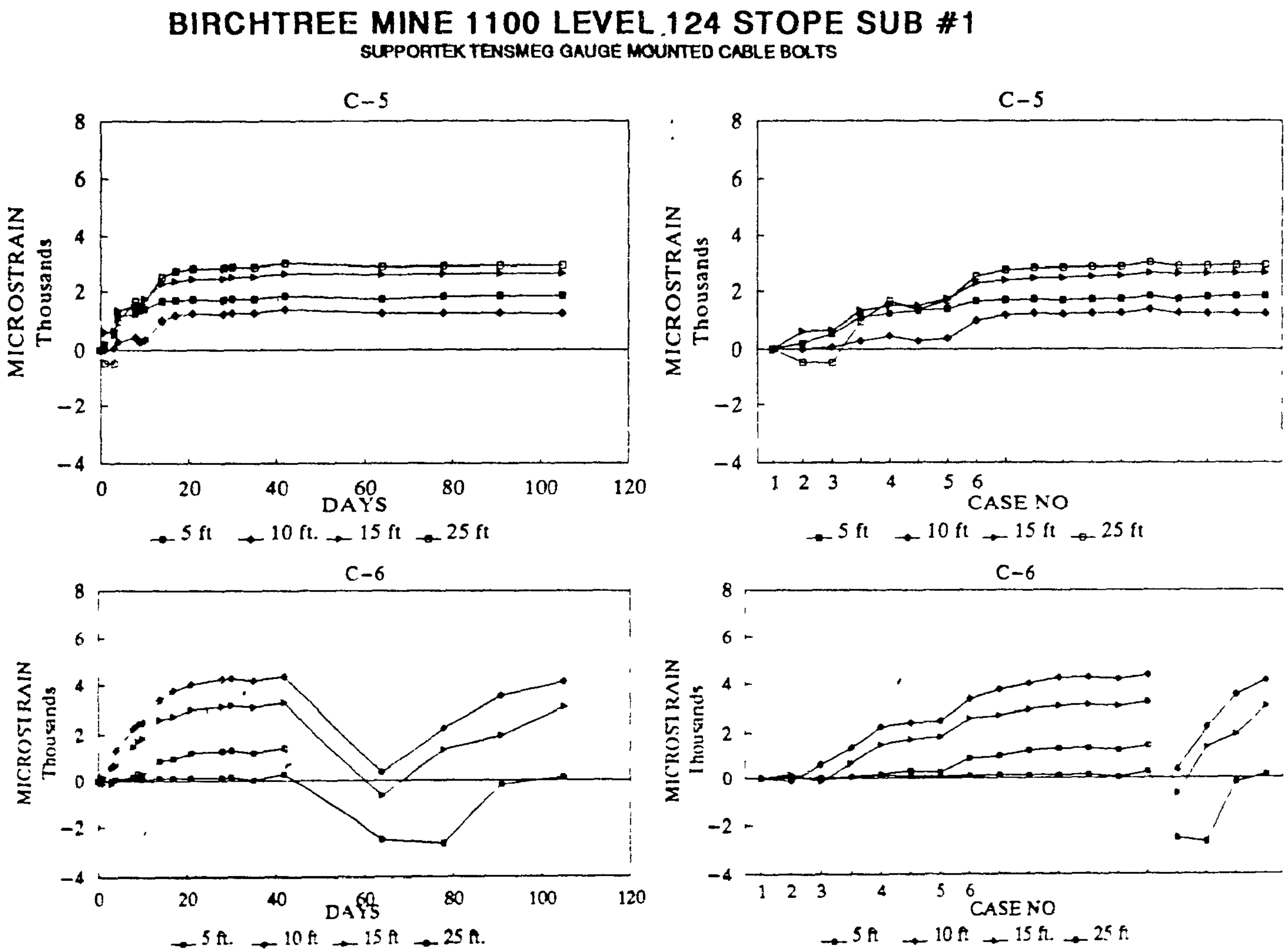


Figure 18: Results from Tensmeg Instrumented Cable Bolts at Birchtree Mine 124 1100 Level Sub1.

Numerical Modelling and Findings

Numerical modelling results simulating the extraction of 124 stope sub 1 indicated the area was under low stress throughout its mine life. This is mainly attributed to minimal extraction ratios in the surrounding area and the low stress levels common to relatively shallow depth. The stress changes attributed to each of the four mining steps were also minimal.

Even though it is difficult to determine if cable bolts were actually required in this particular case, it is clear that the full support capacities of each support member were never utilized. Also, visual inspection of the area showed no major signs of ground control problems following mining. Therefore, if cable bolts are used to support a similar situation in the future (i.e. a drawpoint at low stress levels), a decrease in the cable bolt pattern (such as 50%) is recommended to reduce costs.

4.4 Birchtree 1500 Level Sub 2 108 Complex Block 3

Background

This area was cable bolted to reinforce the crown pillar between the back of 1500 level sub #2 and 1300 level, and the by pass access drift to the stopes northwest of Block #3 (see figure 19). The purposes of this investigation were to assess the performance of the cable bolts, the adequacy of the design, and compare the two different types of cable bolt instrumentation.

Slot and slash stoping proceeded from the south end of the 108 orebody to the north. Blocks #1 and #2 (not shown), south of block #3, were mined and backfilled. Eight blasting sequences using a combination of slashing and decking were used to mine block #3. The dashed line indicating the limits of the last blast, case #8, seen in figure 19, represents the northern limits of block #3.

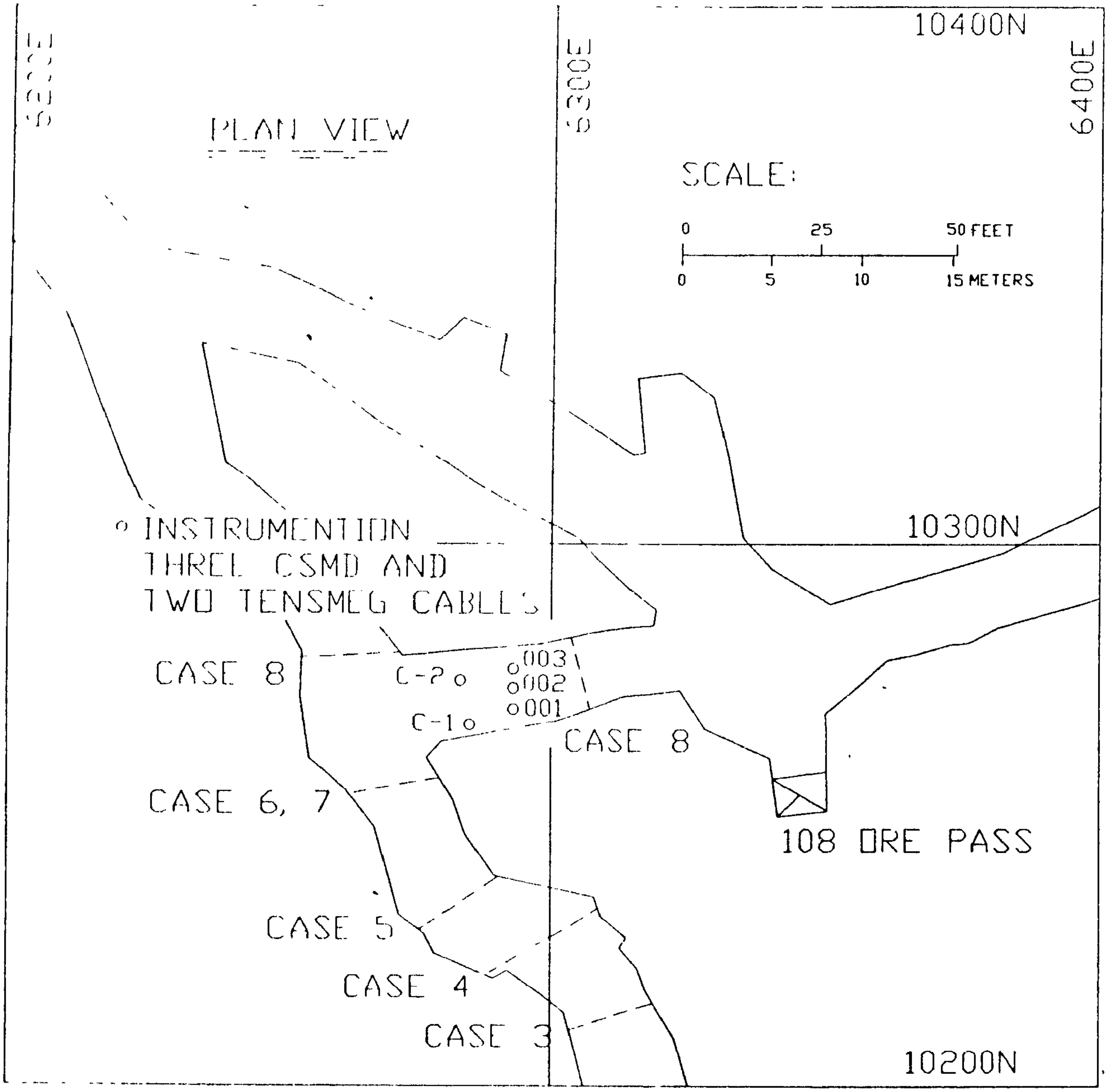
The back between the dash lines of case #6, and #8 was cable bolted. Seven 6 m (20 ft) long cable bolts per row spaced 1.8 m (6 ft) apart were fanned into the back and walls of the area.

Unfortunately, a line survey of the area was not conducted. However, an adjacent survey on the same sublevel produced an RMR rating of 61, and a Q' rating of 27.1 for the ore zone. The validity of these numbers for block #3 is questionable.

Instrumentation

Instrumentation in the form of Supportek's tensmeg gauges mounted on cable bolts and Cable Strain Measuring Devices were placed to monitor five cable bolts' performances (indicated by dotted circles in figure 19). These cables were part of the support pattern and required little additional effort or costs from operations. The three Cable Strain Measuring Devices (#001, #002, and #003) had the first wire anchored at 3 ft, 5 ft, 4 ft (1 ft = 0.3048 m) of their cable bolt and the second wire anchored at 20 ft. The two Supportek's instrumented cables (C-1, and C-2) had tensmeg Gauges mounted at the 5', 10', 15' and 25' of each cable

Figure 19: Plan View 108 Stope Birchtree Mine 1500 Sub2 Block 3.



BIRCHTREE MINE 1500 LEVEL SUB 2 108 ORE ZONE
MODEL POINT GRID AND INSTRUMENTATION LOCATIONS

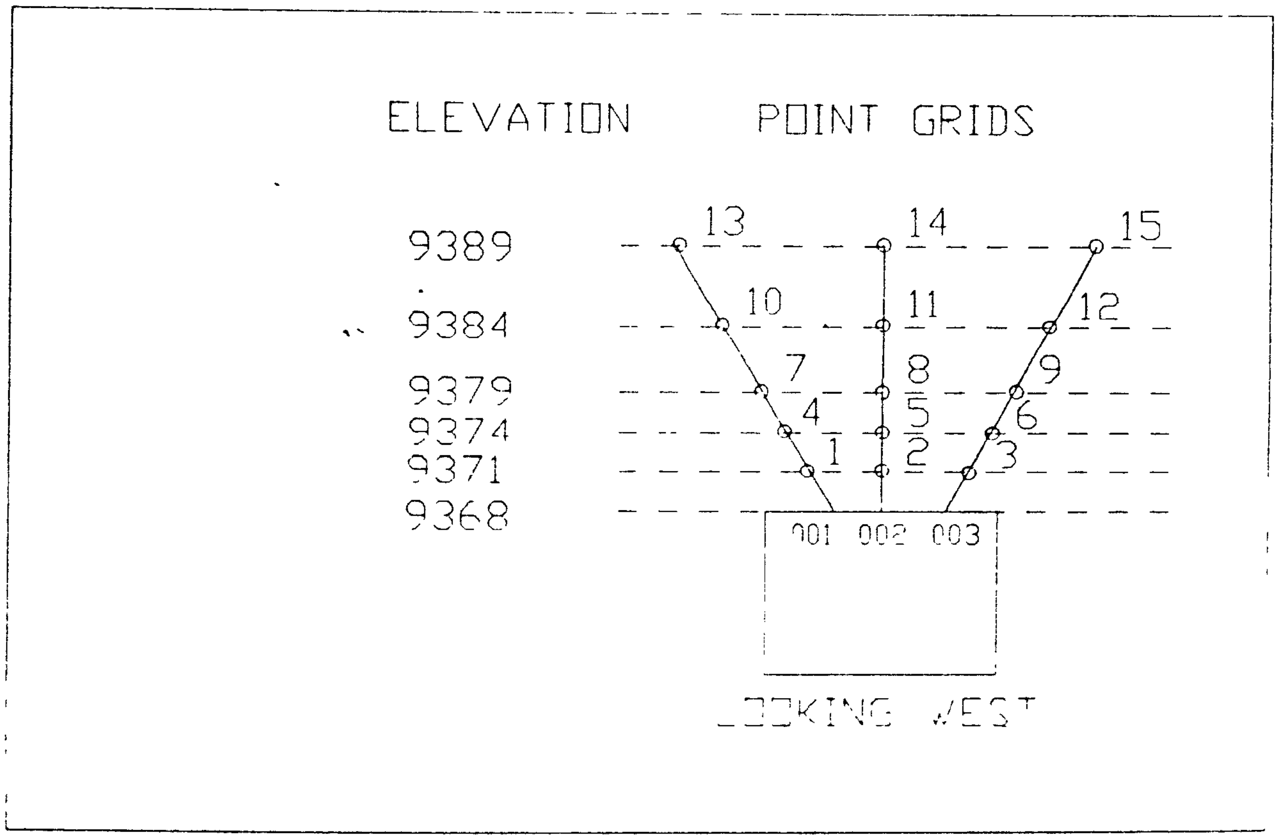


Figure 20: Cross Section Showing Orientation of Instrumented Cable Bolts Birchtree Mine 1500 Sub2 Block 3.

bolt. After the grout was set, all cables were tensioned into placed on October 3rd, 1991. The orientation of the three CSMD cable bolts are shown in figure 20.

Results

Results from the two different monitoring devices did not agree. After the seventh blast (case #7, decked), all three CSMDs began recording high strains whereas Supportek's tensmeg gauges showed small strains. The belief is that accurate results were obtained from the CSMDs and that the Supportek's Tensmeg gauges malfunctioned in this test site. An explanation was forwarded to support this theory. From visual monitoring, the area is known to be wet. This could have damaged the tensmeg gauges as they are very sensitive to water. For interest, a report from a study in Minnova indicated up to one third of tensmeg gauges used during a study eventually failed.[12]

All five instrumented cable bolts indicated little strain prior to the deck blast of row #23 (outlined by case #7, figure 19). Large strains were measured on all three 20' wires of the CSMDs after this deck blast. Subsequent measurements on following days indicated additional strain on each of the three CSMDs cable bolts. Visual monitoring noted that ground conditions continued to deteriorate during this time. The general consensus at the mine was that the back was destabilizing. It was decided a quick removal of just three more rows (up to row #26 or case #8) from this stope block was economically prudent. After this last blast, access to instrumentation was lost. Concern of an overbreak into 1300 level led to additional cable bolting of the crown pillar from the main level. Also, an extensometer based monitoring program was proposed and implemented at the top of this crown pillar. As the four extensometers detected no additional displacements, the area is thought to have stablized.

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[12] Hutchinson, 1991.

The reading on the short wires and the plate (see table XIII, and figure 21 for results), requires some explanation. On cable 001, the 3' foot wire to cable readings increased while the barrel to cable (under the heading "PLATE") readings decreased. For the PLATE reading, it is thought that ground movement induced tension on the cable bolt which then exceeded the gripping capacity of the barrel and wedge. Thus, the cable bolt slipped on the wedge and barrel up and into the hole. This would account for the decrease in distance between the end of the cable bolt and the barrel on the plate. As for the "0-3" reading, or the distance between the crimp attached to the wire anchored at the 3' mark of the cable bolt and the reference point on the cable bolt, two explanations are forwarded. One, the punch lock crimp anchored at 3' slipped. Two, the section of the cable bolt between the plate and this first anchored was stretched due to ground displacement. While Cable 003 (two wires anchored at 4', and 20') behaved very similarly to Cable 001, Cable 002 did not record a negative strain on its short (5') anchor. Perhaps this is due to its figher depth in the hole.

CABLE BOLT MONITORING RESULTS

DATE	DAYS	CASE #	CSMD 001 PLATE	0-3'	MSTRAIN	3-20'	MSTRAIN	CSMD 002 PLATE	0-5'	MSTRAIN	5-20'	MSTRAIN	CSMD 003 PLATE	0-4'	MSTRAIN	4-20'	MSTRAIN	COMMENTS
03-Oct-91	0			53		55			52		55			51		52		BEFORE TENSIONING
03-Oct-91	0	1		52	1094	52	386		52	0	55	0		53	-1640	52	410	AFTER TENSIONING
07-Oct-91	4	2	300	50	3281	50	38[illegible]	360	52	0	53	437	310	50	820	49	410	NO MINING NEAR INSTALATION
08-Oct-91	5		295	49	4374	48	579	360	50	1312	52	219	310	49	1640	48	410	NO MINING NEAR INSTALATION
11-Oct-91	8	3	300	51	2187	52	193	363	52	0	55	0	317	51	0	52	0	
16-Oct-91	13	4	302	51	2187	52	193	363	52	0	55	0	317	51	0	52	0	BLASTING ROW 20
21-Oct-91	18		297	51	2187	52	193	364	52	0	55	0	317	50	820	51	0	MINING 40 FT FROM CABLES
24-Oct-91	21	5	300	52	1094	51	579	364	52	0	53	437	317	51	0	51	205	ROW 20 SPALLING AT END
24-Oct-91	21		300	51	2187	49	772	363	52	0	52	656	318	51	0	49	615	DRILLING ROW 24
29-Oct-91	26	6	300	52	1094	49	965	363	52	0	51	875	318	51	0	49	615	BLASTED TO ROW 23
01-Nov-91	29		300	52	1094	50	772	363	52	0	52	656	316	52	-820	50	615	
04-Nov-91	32		300	52	1094	49	965	363	52	0	52	656	317	51	0	49	615	
05-Nov-91	33		300	52	1094	50	772	363	51	656	50	875	317	50	820	48	615	
06-Nov-91	34		300	51	2187	48	965	363	52	0	50	1094	317	51	0	48	820	
07-Nov-91	35	7	300	51	2187	47	1158	363	52	0	48	1531	317	52	-820	46	1435	OUT TO COLLAR ROW #23
08-Nov-91	36		295	51	2187	11	8106	359	50	1312	18	7655	309	58	-5741	22	7587	
12-Nov-91	40		295	55	-2187	-5	11965	356	48	2625	16	7655	305	60	-7382	9	10663	
13-Nov-91	41		294	56	-3281	-6	12351	358	48	2625	15	7874	304	61	-8202	5	11688	
14-Nov-91	42		294	58	-5468	-11	13702	356	47	3281	15	7655	305	61	-8202	5	11688	
18-Nov-91	46		293	57	-4374	-24	16018	357	45	4593	12	7874	333	62	-9022	0	12918	

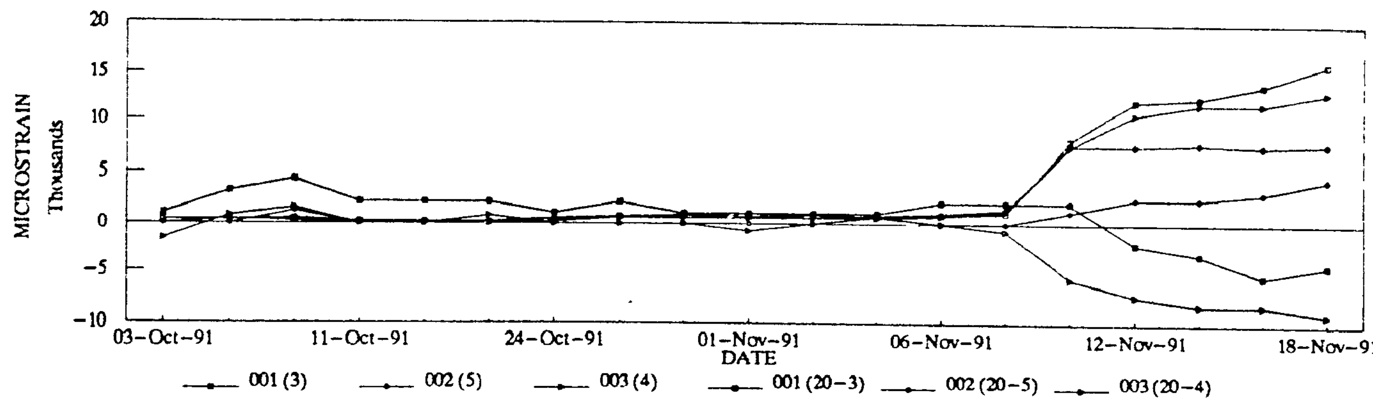


Table XIII: Results of Instrumented Cable Bolts at Birchtree Mine 1500 Level Sub 3 Block 3. (above)

Figure 21: Results of CSMD Instrumented Cable Bolts at Birchtree Mine 1500 Level Sub 3 Block 3. (below)

Numerical Modelling and Findings

Numerical modelling simulating the extraction sequence shows the back above block #3 was under relatively high stress levels. Point grids were constructed in the area where the CSMDs were placed. As stoping began, the estimated major principal stress was around 40 MPa. This is nearly twice the value of 25 MPa for the previous case study in 124 stope. After the last blast, values from 45 to 55 MPa were found in the back. Stress concentration associated with the crown pillar was the major cause of the high stress environment and was probably a cause for the noted ground control difficulties.

In this particular case, visual monitoring and instrumentation indicated the support capacities of these cable bolts were fully utilized. Prudence suggests that in a future situation, the cable lengths should be increased to 30′ (9 m). As for the support density, it is difficult to advocate if additional cable bolts would provide any net benefit.

4.5 Thompson Mine 2003 Level Sub #3, 78 Complex Block 53

Background

The purpose of this investigation was to monitor and relate the performance of cable bolts in the stope back, to compare the two different types of instrumentation for cable bolts, to assess ground conditions during mining of block #53 in 78 stope, and recommend ground support measures and strategies for future mining blocks in the area.

78 VBM stope is located between 1600 and 2000 levels in the 1B zone of Thompson Mine. The stope was mined via cut and fill to approximately 42 m (140 ft) above 2000 level before conversion to vertical block mining. The remainder of the stope is being mined in three lifts of 30 m (100 ft), 21 m (70 ft), and a crown of 15 m (50 ft). From 1600 level upwards, stoping had been via cut and fill.

The ore zone in the 78 complex is primary tabular dipping from 70 to 60 degrees east. The hangingwall consists of biotite schists and occasional peridotite inclusions. The ore zone is either massive sulphides or a sulphide matrix (Q' value of 12.4 and RMR of 72). The footwall is composed of biotite schist (Q' = 13.2, RMR = 70) for the first 6 m to 9 m from the sulphide contact. Foliation was extremely distinct in all three zones. Further, graphitic infilling was found near the hangingwall contact in some areas (Greer, Nyysola 1988). The rock mass in al' three zones are considered fair in the two rock mass classification systems.[13]

As site access would not be a barrier, block 53 provided an ideal site for cable bolt performance monitoring as severe ground conditions were anticipated. Due to the high extraction ratio in the area as ore have been mined from 1600 level up to surface, and 2000 level up to sub 2, high stress conditions prevailed throughout sub 3. It was decided to reinforce the back with 12 m (40 ft) cable bolts prior to stoping to

[13] Preliminary Report on 78 Stope, Greer.

control the 20 to 23 m (65-75 feet) thick crown pillar.

Cable Bolt Characteristics

The yield strain of a 5/8" diameter cable bolt is 8,000 Microstrain (0.8%) When the yield strain of a cable bolt is exceeded, the cable bolt has deemed to have failed and will not accommodate additional loading (behave like a plastic material). However, the full bond characteristic of a cable bolt allows sections of a cable bolt to fail while other sections remain functional.

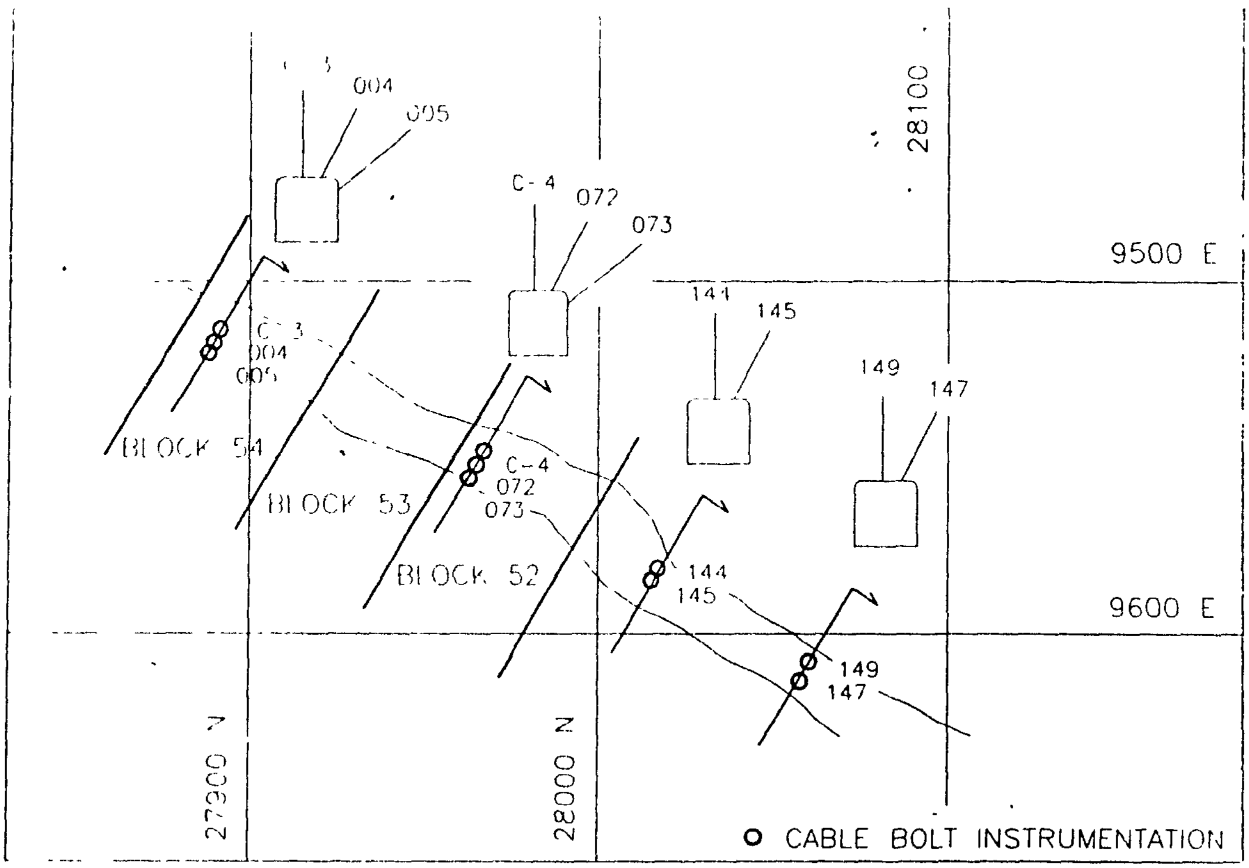
Instrumentation

There were ten instrumented cable bolts installed in this area. The location of these cable bolts are represented by the bold circles in figure 22. Cables C-3 and C-4 are instrumented with Tensmeg gauges. Cables 004, 005, 072, 073, 144, 145, 147, and 149 are Cable Strain Measuring Devices (CSMD) with anchors at 3 m (10') and 9 m (30') or 12 m (40').

Results

The monitoring results for the cable bolts in Microstrain are displayed in figures 23, 24, and 25 (see also Tables XIV, XV, and XVI). Thus far, the three cables (C-3, 004, and 005) south of block 53 have detected only small strain increases since the deck blast of block 53. This is consistent with visual observation of the stable ground condition in this area.

Figure 22: Plan View of 78 Stope 2000 Sub 3 in Thompson Mine. Instrumented Cable Bolts are represented by c rcles. Inserts show their orientation.

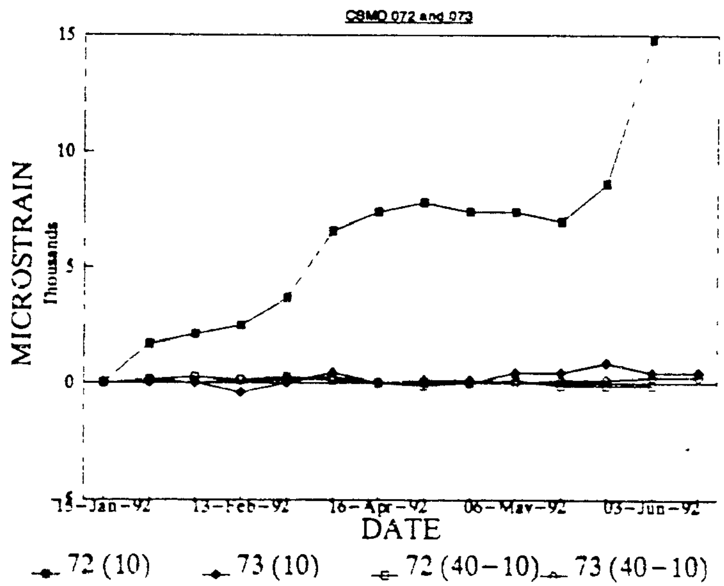


THOMPSON MINE 2002 LEVEL SUB 3 78 ORE ZONE

CABLE BOLT MONITORING RESULTS

DATE	DAYS	CASE NO.	CSMD 072 PLATE	CSMD 072 10'	CSMD 072 MSTRAIN	CSMD 072 40'	CSMD 072 40'-10' MSTRAIN	CSMD 073 PLATE	CSMD 073 10'	CSMD 073 MSTRAIN	CSMD 073 40'	CSMD 073 40'-10' MSTRAIN	SUPPO TEK'S C4 5FT READING	MSTRAIN	10FT READING	MSTRAIN	15FT READING	MSTRAIN	25FT READING	MSTRAIN	COMMENTS
24-Oct-91																					
22-Nov-91													7077		6152		5683		4298		
03-Jan-92													7093		6152		5680		4298		C4 AFTER TENSIONING (2,500 PSI)
15-Jan-92		1	700	58	0	54	0	549	55	0	57	0	7080	0	6152	0	5684	0	4300	0	C4 AFTER TENSIONING (5,000 PSI)
04-Feb-92			700	54	1640	49	117	549	55	0	56	117	7560	480	7291	1139	6656	972	4281	-19	
10-Feb-92			700	53	2051	47	234	550	55	0	57	0	7736	656	7590	1438	6810	1126	4291	-9	
13-Feb-92			700	52	2461	47	117	550	56	-410	57	117	7857	777	7583	1431	6900	1216	4280	-20	
28-Feb-92			697	49	3691	43	234	550	55	0	56	117	8490	1410	8164	2012	7314	1630	4278	-22	
07-Apr-92			695	42	6582	37	117	550	54	410	54	234	9200	2120	8722	2570	7996	2212	4258	-42	
16-Apr-92			694	40	7382	36	0	550	55	0	57	0	9322	2242	8425	2273	7998	2314	4252	-48	
01-May-92			693	39	7792	36	-117	550	55	0	56	117	9493	2413	8626	2474	8166	2482	4234	-66	BEFORE STOPING
04-May-92			694	40	7382	36	0	549	55	0	56	117	9576	2496	8709	2557	8225	2542	4242	-58	BLAST #1 MAY 1st
06-May-92			693	40	7382	35	117	549	54	410	56	0	9582	2502	8732	2580	8248	2564	4228	-72	
13-May-92		2	691	41	6972	38	-117	549	54	410	55	117	9766	2686	8890	2738	8378	2692	4220	-80	BLAST #2 MAY 8th
28-May-92			688	37	8612	34	-117	549	53	820	54	117	10610	3530	9718	3566	9300	3616	4216	-84	BLAST #1 & #4 MAY 19th & 21st
03-Jun-92		3	686	22	14764	19	-117	549	54	410	54	234	8650	1570	7792	1640	14326	8642	4344	44	BLAST #3 MAY 29th SLOT JUNE 2nd
04-Jun-92			657	48	4101	45	-117	549	54	410	54	234	broke	0	8602	2450	15410	9726	4760	460	DECK BLAST #? JUNE 3rd

Table XIV: Results from Instrumented Cable Bolts 072, 073, and C4 north of Block 53 Thompson Mine 78 Complex 2000 Sub3.



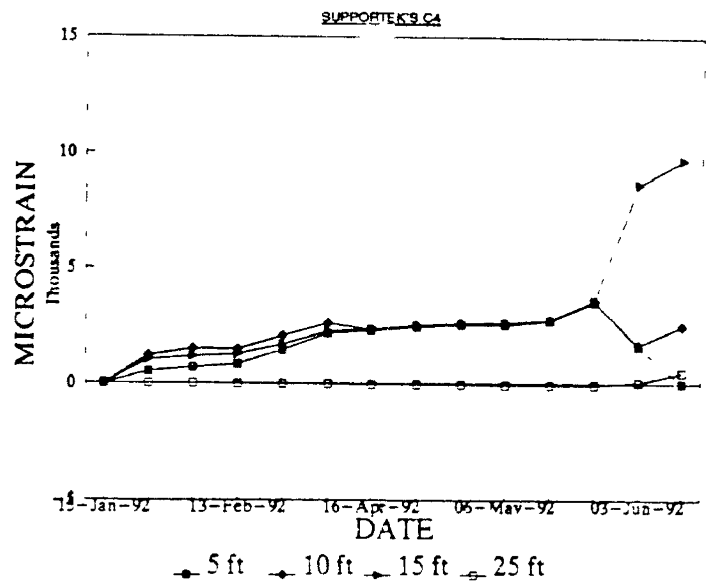


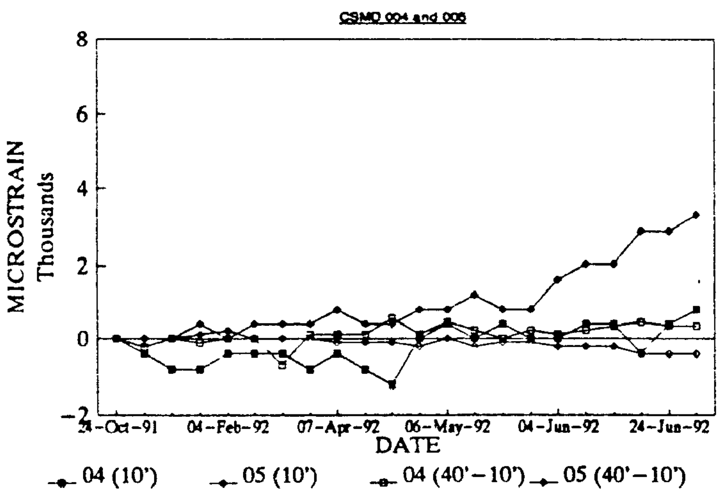
Figure 23: Plot of Results from Instrumented Cable Bolts 072, 073, and C4 north of Block 53 Thompson Mine 78 Complex 2000 Sub3.

Table XV: Results from Instrumented Cable Bolts 004, 005, and C3 south of Block 53 Thompson Mine 78 Complex 2000 Sub3.

THOMPSON MINE 2002 LEVEL SUB 3 78 ORE ZONE

CABLE BOLT MONITORING RESULTS

			CSMD 004					CSMD 005					SUPPORTEK'S C3								
DATE	DAYS	CASE NO	PLATE	10'	MSTRAIN	40'	40'-10' MSTRAIN	PLATE	10'	MSTRAIN	40'	40'-10' MSTRAIN	5FT READING	MSTRAIN	10FT READING	MSTRAIN	15FT READING	MSTRAIN	25FT READING	MSTRAIN	COMMENTS
24-Oct-91	0		604	56	0	59	0						2978		4085		5022		4238		C3 AFTER TENSIONING (2300 PSI)
22-Nov-91	29		604	57	-410	62	-234	298	58	0	54	0	3108	132	4138	53	5028	6	4277	39	
03-Jan-92	71		604	58	-820	61	0	299	58	0	54	0	3150	174	4232	147	5150	128	4278	40	
15-Jan-92	83	1	604	58	-820	62	-117	299	57	410	52	109	3187	211	4257	172	5256	234	4292	54	
04-Feb-92	103		605	57	-410	60	0	299	58	0	52	219	3230	254	4328	243	5354	332	4290	52	
10-Feb-92	109		605	57	-410	60	0	299	57	410	53	0	3248	272	4340	255	5362	340	4273	35	
13-Feb-92	112		604	57	-410	66	-703	299	57	410	53	0	3251	275	4347	262	5383	361	4278	40	
28-Feb-92	127		605	58	-820	60	117	299	57	410	53	0	2930	-46	3960	-125	5080	58	4382	144	
07-Apr-92	166		604	57	-410	59	117	298	56	820	53	-109	2910	-66	3912	-173	5066	44	4374	136	
18-Apr-92	175		604	58	-820	60	117	298	57	410	54	-109	2798	-178	3803	-282	4955	-67	4281	43	
01-May-92	190		603	59	-1230	57	586	298	57	410	54	-109	2942	-34	3920	-165	5052	30	4410	172	BEFORE STOPING
04-May-92	193		604	58	0	58	117	298	56	820	54	-219	2880	-96	3852	-233	5000	-22	4350	112	BLAST #1 MAY 1st
08-May-92	195		601	55	410	54	469	300	58	820	52	0	2910	-66	3900	-185	5040	18	4390	152	
13-May-92	202	2	604	58	0	57	234	298	55	1230	53	-219	2855	-121	3852	-233	4990	-32	4345	107	BLAST #2 MAY 8th
28-May-92	217		604	55	410	58	0	298	56	820	53	-109	2930	-46	3854	-231	4970	-52	4325	87	BLAST #3 & #4 MAY 19th & 21st
03-Jun-92	223	3	604	56	0	57	234	298	56	820	53	-109	2890	-86	3860	-225	5023	1	4340	102	BLAST #5 MAY 26th, SLOT JUNE 2nd
04-Jun-92	224		604	56	0	58	117	298	54	1640	52	-219	2723	-253	3706	-379	4968	-54	4215	-23	
08-Jun-92	228	4	602	55	410	56	234	296	53	2051	51	219	2876	-100	broke	0	broke	0	4288	48	DECK BLAST #7 JUNE 5th
09-Jun-92	229		604	55	410	55	352	298	53	2051	51	-219	2810	-166	3824	-261	6100	1078	4320	82	
17-Jun-92	237		604	57	-410	56	469	298	51	2871	51	-437	2800	-176	3810	-275	7006	1984	4280	42	
24-Jun-92	244		604	55	410	55	352	297	51	2871	51	-437	2900	-76	3820	-265	7314	2292	4366	128	
07-Jul-92	257		604	54	820	54	352	297	50	3281	50	-437	2560	-416	3390	-695	6662	1640	3968	-270	measurements taken before shutdown



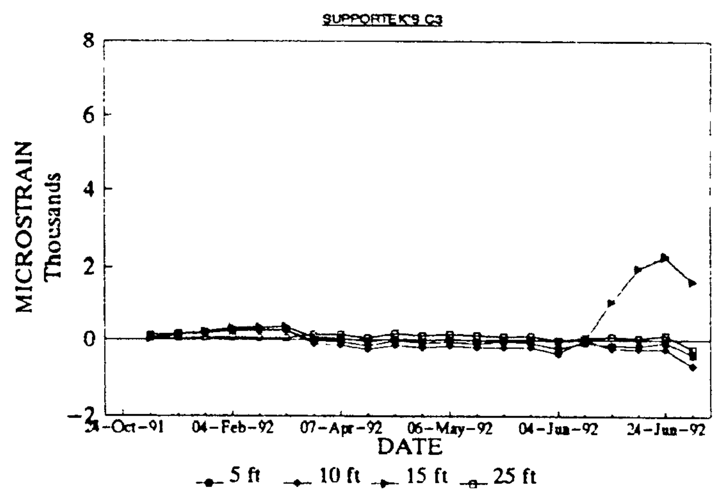


Figure 24: Plot of Results from Instrumented Cable Bolts 004, 005, and C3 south of Block 53 Thompson Mine 78 Complex 2000 Sub3.

Table XVI: Results from Instrumented Cable Bolts 144, 145, 147, and 149 in 78 Complex 2000 Sub3 Thompson Mine.

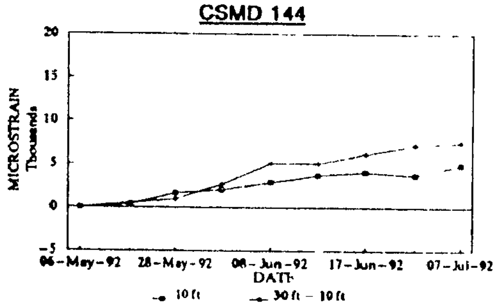
Figure 25: Plot of Results from Instrumented Cable Bolts 144, 145, 147, and 149 in 78 Complex 2000 Sub3 Thompson Mine.

THOMPSON MINE 2002 LEVEL SUB 3 78 ORE ZONE

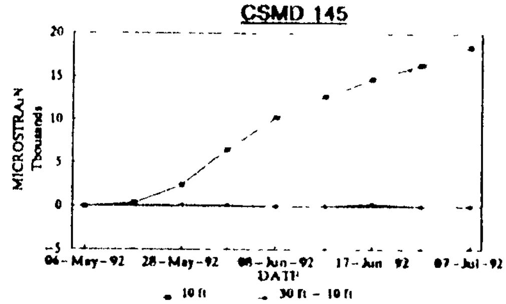
CABLE BOLT MONITORING RESULTS

			CSMD 144					CSMD 145				
DATE	DAYS	CASE NO	PLATE	10'	MSTRAIN	30'	30'-10' MSTRAIN	PLATE	10'	MSTRAIN	30'	30'-10' MSTRAIN
06-May-92	0		330	79	0	82	0	391	63	0	65	0
13-May-92	7	2	330	78	410	78	492	391	62	410	63	164
28-May-92	22		330	75	1640	72	984	391	57	2461	58	164
04-Jun-92	29	3	330	74	2051	61	2625	391	47	6562	48	164
08-Jun-92	33	4	326	72	2871	44	5085	389	38	10253	40	0
09-Jun-92	34		330	70	3691	42	5085	389	32	12713	34	0
17-Jun-92	42		330	69	4101	34	6234	388	27	14764	27	328
24-Jun-92	49		330	70	3691	29	7218	388	23	16404	25	0
07-Jul-92	62		329	67	4921	24	7546	388	18	18455	20	0

CSMD 144

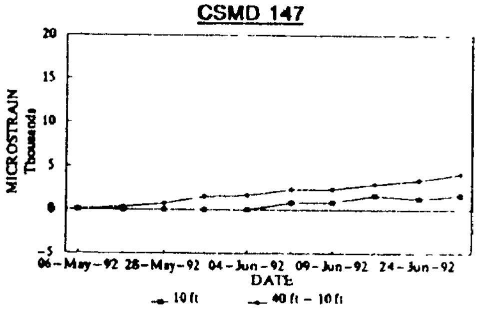


CSMD 145

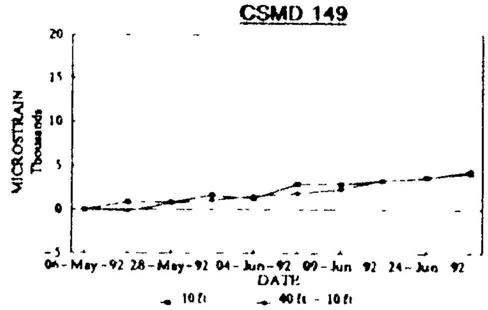


			CSMD 147					CSMD 149				
DATE	DAYS	CASE NO	PLATE	10'	MSTRAIN	30'	30'-10' MSTRAIN	PLATE	10'	MSTRAIN	30'	30'-10' MSTRAIN
06-May-92	0		464	84	0	86	0	404	71	0	72	0
13-May-92	7	2	464	84	0	84	328	404	69	820	71	-164
28-May-92	22		464	84	0	82	656	404	69	820	66	656
03-Jun-92	28	3	464	84	0	77	1476	404	67	1640	62	984
04-Jun-92	29		464	84	0	76	1640	404	68	1230	60	1476
08-Jun-92	33	4	465	82	820	70	2297	404	64	2871	54	1804
09-Jun-92	34		464	82	820	70	2297	404	64	2871	51	2297
17-Jun-92	42		464	80	1640	64	2953	404	63	3281	44	3281
24-Jun-92	49		463	81	1230	62	3445	404	62	3691	41	3609
07-Jul-92	62		463	80	1640	57	4101	404	61	4101	35	4429

CSMD 147



CSMD 149



The lower sections (first 3 m or 10') of cable bolts CSMD 072 and Tensmeg C-4, inserted just north of block 53, failed before the block was completely mined. CSMD 073, also just north of block 53, but inserted in the hangingwall, did not detect any significant strain. The hangingwall of this block was stable. During mining of this block, several cable bolts in the back of block 53 snapped or pulled through their strapping or barrel wedge plate assemblies. Large amount of loose was hanging on the screen before the final deck blast. Eventually, a significant ground fall occurred in the back, but this was over two months after the stope was mucked out. Currently cable bolts 072, 073, and C-4 are no longer accessible for monitoring.

As the back and hanging wall were still relatively intact after the deck blast, the ground support over block 53 is considered optimal as full strains were developed in the lower sections of the cables bolts and rock bolts. Thus, the support used in block 53 can be used as a good bases for support design for similar mining situations (i.e. the remainder of the 78 complex).

As the severity of the ground conditions became apparent, a complementary cable bolting and instrumentation program was implemented in the blocks north of block 53. Since large strains were not recorded between 10' and 40' of the previous six instrumented cable bolts, it was decided 9 m (30 ft) cable bolts would provide similar support to that of block #53 at a lower cost in these northern blocks.

CSMD cables 144, and 145 were placed over block 51. CSMD 145 indicated that the bottom ten feet of the cable bolt has failed while CSMD 144 has loaded to over half of its yield strength along the length of the cable bolt.

Cables 147, and 149 are located over block 50 and are farther away from the mined block and thus experienced less strain. Microstrains from 1,600 to 4,400 have been detected on these two CSMDs.

This area has been the only site where both types of instrumented cable bolts performed to satisfaction. It is clear that the results obtained by cable bolts C4 and 072 were as compatible to that of C3 and 004. This represents the final evidence that both types of instrumentaton can performed to their design functions.

From the experience in block 53, it is expected that stoping activities of these blocks will induce ground failures similar in scope to that of block 53. As time could be a critical factor, it was recommended that the north blocks be mined prior to the south blocks to minimize further ground deterioration.

Numerical Modelling

Numerical modelling results paints a high stress picture throughout the crown pillar over sub 3. This is not unexpected. Mining of Block #53 was simulated in three steps with the final step responsible for the largest calculated increase in principal stresses in the crown pillar. Point grids were constructed over block #53, and in the locations of the instrumentation. Modelling indicated relatively lower stress conditions existed to the south of block #53 than to the north. This was consistent with the observed ground conditions for the area. In addition, the model found that as the distance from block #53 increases, stress levels decreased. Again, this agreed with the observed ground condition and instrumentation. With these findings, an optimal stoping sequence in this area could be guided by numerical modelling although it will be quite a challenge to avoid these high stress levels as they already persists throughout the area.

4.6 Numerical Modelling and Support in Empirical Design Database

Background

Since one of the main objective of this study is to integrate the benefits of numerical modelling and instrumention of cable bolts into an empirical framework, a number of back analysis case studies were conducted in two large mining areas to build up the database. These are located in the 83 complex of Birchtree Mine, and the 1C zone of Thompson Mine. Two blocks were studied in the S3 complex and seven blocks for the 1C ore zone. Data from the four previous case studies were included in this effort of empirical design renewal.

Database

A database was collected to commence a new and broad effort to include the computational power of numerical modelling and the practical results of instrumentation of support within an empirical approach of design. Since data is the only viable foundation for good design, data that can be readily collected with minimal bias is most desirable. In reality, a complete and unbias data set is very difficult to achieve due to its prohibitve costs. However, it is recognized that information about geological structures, stress, support, size of opening, and life span of opening is required for design. With these considerations in mind, table XVII and XVIII, have been compiled to reflect the above variables associated with each of the thirteen case studies. These two tables combine traditional data types such as RQD, and Ja, with new data sets which documents support performance and stress levels as determined by numerical modelling.

The first table contains a more traditional data set. In table XVII, the first three columns give the location of the excavation. The next three data sets are H, W, and L and provide the excavation dimensions in feet. RQD, Jn, Jr, and Ja represent the geological and structural data set. In the last column, a numeric value supplies information on the stability of the back. A zero indicates that the back was observed to be stable while any other value indicates the amount of overbreak in feet that was observed to have occurred.

Table XVIII below consists of a support section and a numerical modelling section. In the former, the support type, pattern, coverage, and utilization are found. In the latter four columns, the calculated values of the major and minor principle stresses (in MPa) in the back of the excavation are given.

Under support type, the integer value gives the length of the support member in feet while the symbols represent the kind of support used (rb - rockbolts, rbe - rockbolts with extension bolts, Sc - shotcrete, and cb - cable bolts). The support pattern dictates the lateral distances between adjacent support members. Coverage is obtained by dividing the initial surface area that was supported by the final excavation surface area. Thus a bolted 15' by 50' drift used to mine a stope with final back dimensions of 30' by 50' would have a support coverage percentage of 50. Lastly, support utilization is defined as the amount of support capacity utilized. Here, support capacity is determined by the yield strength of the artificial support whereas the amount of this support capacity utilized is the median strain value as detected by instrumentation of these artificial support. Where no instrumentation was used, 'na - not applicable' is inserted. If the back was deemed to have been unstable, the support capacity value defaults to 100 (i.e. the inserted support was either fully utilized or insufficient). There is a word of caution for the support utilization number. That is, this number does not represent a safety factor since there is no known way to determine the grouping effect of neighbouring support members.

In the next four columns are two sets of Stress Signatures values of Sigma 1 and Sigma 3 at 10 and 20 feet over the centroid of each back surface. In the past, the calculated stress values from numerical modelling close to the surface of the excavation (at 0 to 2') was deem important (convenient). This was probably practiced to ease the comparison between different case studies. This practice could have been responsible for the low esteem of numerical modelling among designers as a practicable predictive and design tool. The underlying problem with this limited interpretation of numerical modelling where only one set of values was used to characterized behaviour for an entire stope was its self imposed restrictions. For instance, a model usually determines the stress levels at thousands of points. What makes one point more important and representative than the rest is not known. Also, computational limitations dictates that the margin of error in calculated stresses, and displacements increase as you approach the surface of the excavation. The stress signature set at ten away from the excavation was chosen to reduce the estimation errors of the numerical program. Further, to better represent the overall stress picture, a second stress signature set of twenty feet above the centroid was added to give a more dimensional perspective of the stress picture above the excavation. In fact, it would be beneficial to add to the number of stress signature sets in table XVIII.

Table XVII: Part I, Traditional Database for Back Design.

Level	Area	BK#	H	W	L	RQD	Jn	Jr	Ja	STABLE
3180	1C	17	140	25	50	86	12	1.9	0.8	0
3180	1C	19	140	105	50	86	12	1.9	0.8	0
3050	1C	17	125	120	50	86	12	1.9	0.8	0
3180	1C	15	140	50	50	86	12	1.9	0.8	0
3180	1C	21	140	100	50	86	12	1.9	0.8	10
3180	1C	14	140	50	100	86	12	1.9	0.8	20
2650	1C	2	185	50	120	86	12	1.9	0.8	40
2002	78	53	95	25	50	90	15	2.1	1	10
2000	83	800	115	22	60	73	15	2.5	1.4	30
2000	83	790	115	55	65	73	15	2.5	1.4	25
1500	108	3	95	25	180	97	12	1.6	0.9	5
1100	124	Dp	15	18	150	83	15	2.4	1.6	0
1900	83	Dr	13	15	150	87	15	2.6	0.9	0

Table XVIII: Part II, Complimentary Database for Design.

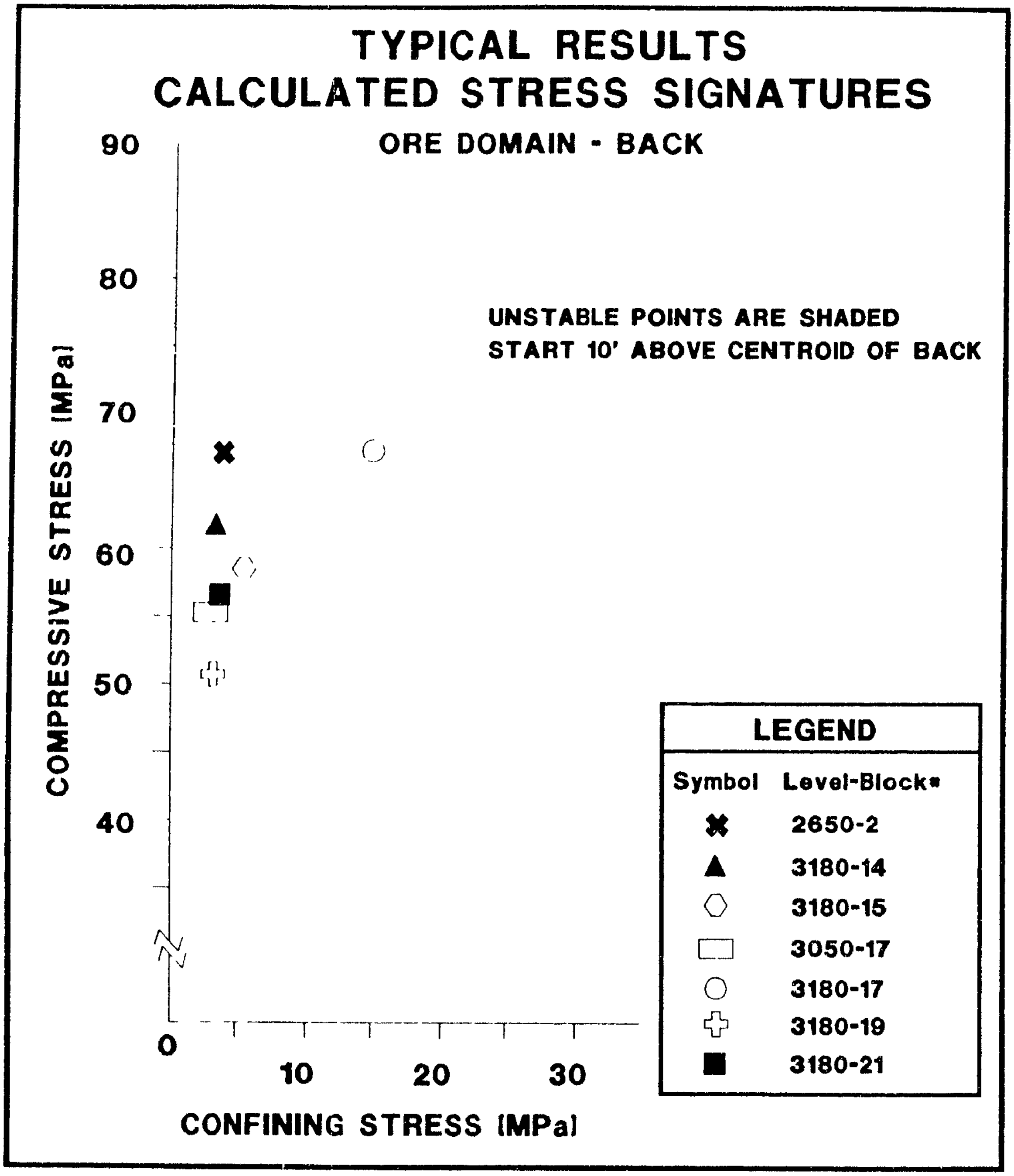
Level	Area	Bk #	Sup Type	Sup Pat	% Cov	Sup Util	S1 (10)	S3 (10)	S1 (20)	S3 (20)
3180	1C	17	8rbe	4x4	60	na	67	15	57	22
3180	1C	19	8rbe	4x4	30	na	51	3	52	8
3050	1C	17	8rbe	4x4	30	na	55	3	55	9
3180	1C	15	8rbe	4x4	30	na	58	6	55	14
3180	1C	21	8rbe	4x4	30	100	57	4	56	9
3180	1C	14	8rbe	4x4	30	100	61	3	60	10
2650	1C	2	8rbe	4x4	30	100	67	4	65	11
2002	78	53	40cb	8x10	100	100	94	30	93	36
2000	83	800	Scrb	4"	100	100	34	3	32	7
2000	83	790	30cb	9x10	50	100	44	9	39	14
1500	108	3	20cb	7x6	100	100	47	14	48	19
1100	124	Drp.	40cb	7x8	100	50	28	5	24	6
1900	83	Dr.	8rb	4x4	100	40	43	11	36	15

Findings and Sample Application

As the number of case studies in this database is limited, its best application would require the designer to find the closest equivalent case study as a reference standard. For example, if a stope in the 1C zone was to be mined, and the stability of the stope back was deemed important, a numerical model could estimate the stopes' Stress Signature. If it indicated either a relatively low major principal stress, sigma 1, or a relatively high minor principal stress, sigma 3, (see figure 26) and given that other factors such as rock mass characteristics and stope dimensions are similiar to the seven previous cases, then the stope would be predicted to be stable and no secondary support would be planned.

If no similiar comparison can be found in the tables above, the continual application of other empirical methods is still a reasonable design procedure. But the expansion of the above database vertically (in the number of case histories) and horizontally (in the number of variables) would become an integrated design tool of the future.

Figure 26: Stress Signatures (10) for seven 1C stopes.



4.7 Summary

In order to control the costs of over and under support of stope backs, an ability to predict their stability and instability is crucial. At present, Mathews' Stability Graph and its descendant, Potvin's Modified Stability Graph are considered the most advance empirical methods of surface stability prediction. An analysis of a past study using the above methods was conducted to scrutinize these methods, its strengths and weaknesses, and how it was applied. A summary presentation of these results are found in appendix D.

Thirteen case studies were assessed using a combination of ground-support monitoring, empirical modelling, and numerical modelling. With these investigations, a new geotechnical database has been compiled to further a new design philosophy. This philosophy stresses the importance of documenting past experiences for the purpose of future design. Through the inclusion of information concerning support type, density, coverage, utilization, and stress signatures, along with the more traditional data sets (from rock mass classifcation systems), a more comprehensive geotechnical database is constructed. It is conceivable that the vertical and horizontal expansion of this database would establish it as the leading predictive and design tool of choice in the near future.

Chapter 5 - Conclusions and Recommendations

5.0 Conclusions

The development of the Cable Bolt Strain Measuring Device by INCO Ltd. and McGill University has greatly facilitated economical field monitoring of cable bolt behaviour. This technological advance will improve cable bolt support designs as more knowledge about cable support performance is obtained. This device has led to an extensive monitoring program at INCO's Manitoba Division to gain knowledge about rock mass/cable bolt interaction over several stopes in different ore zones. To aid evaluation, a three dimensional elastic boundary element numerical model was employed to calculate the induced stresses in stope backs where the cables were placed for support.

The above knowledge was then inserted into a geotechnical database where rock mass characteristics, information on support, and results from numerical modelling are included. The premise of this geotechnical database is the belief that Stability prediction of excavation surfaces is improved by concentrating efforts on how to document and account for the three S'es of rock mechanics - Stress, Structure and Support. This is achieved via desegregation of stability variables, improvement in the measuring techniques of these variables, and the addition of relevant new stability variables. The basic premise of desegregation, and to a lesser extent, the input of new variables, is to facilitate a higher dimensional analysis that was lost when variables were combined together. Once enough case histories have been compiled, this higher dimensional analysis can take place (statistics you know). Meanwhile, beyond the addition of more case histories, continuous improvement of this database would require:

1- Inclusion of other relevent stability data, such as time of failure, adjacent ground conditions, and geological anormalities.

2- Improvement of current data collection, such as the replacement of RQD with the median block size or the shape of the block size distribution curve.

3- The long term commitment towards an instrumentation program to obtain unbias (at least less bias) data on ground and support behaviour.

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APPENDIX A

Sample Input File for MAP3D

```
* DEFINE PROJECT TITLE:
* --------------------

  78 MODEL - 2002 LEVEL sub3 block 53

* DEFINE PROGRAM PARAMETERS:
* -------------------------
*
* nld   nit   nps   rpar   stol    al    ag     dol   don   doc   doe   doq

  10    500   0     1.0    .03     5.0   5.0    1.5   1.0   2.0   2.0   2.0

* DEFINE 3D POLOGONS:
* ------------------

* n   i1   i2  i3   i4   i5   i6   i7   i8 type thic c1 c2 c3

*    sub 3
*
  2     1    2    3    4    5    6    7    8   1
  2     9    4   10   11   12    8   13   14   1
  2     4    3   15   10    8    7   16   13   1
  2     3   17   20   15    7   21   24   16   1
  2    25   26   18   17   27   28   22   21   1
  2    17   18   19   20   21   22   23   24   1
  2    20   19   30   29   24   23   34   33   1
  2    29   30   31   32   33   34   35   36   1
  2    32   31   38   37   36   35   42   41   1
  2    37   38   39   40   41   42   43   44   1
  2    45   37   40   46   47   41   44   48   1
  2    40   39   50   49   44   43   54   53   1
  2    49   50   51   52   53   54   55   56   1
  2    50   57   58   51   54   59   60   55   1
  2    52   51   62   61   56   55   66   65   1
  2    69   52   61   70   71   56   65   72   1
  2    61   62   63   64   65   66   67   68   1
  2    64   63   74   73   68   67   78   77   1
  2    73   74   75   76   77   78   79   80   1
  2    76   75   82   81   80   79   86   85   1
  2    81   82   83   84   85   86   87   88   1
* cross-cuts
  2    75   89   92   82   79   93   96   86   1
  2    89   90   91   92   93   94   95   96   1
  2    89   97   90   90   93   98   94   94   1
  2    90  177  180   91   94  181  184   95   1
  2    18  131  133   19   22  134  136   23   1
  2   131  118  132  133  134  122  135  136   1
  2   123  124  125  126  127  128  129  130   1
* drill horizon
  2    99  100  101  102  103  104  105  106   1
  2   102  101  108  107  106  105  112  111   1
  2   107  108  109  110  111  112  113  114   1
  2   110  109  116  115  114  113  120  119   1
  2   115  116  117  118  119  120  121  122   1
  2   118  117  123  123  122  121  127  127   1
  2   118  123  126  132  122  127  130  135   1
  2   132  126  138  137  135  130  142  141   1
  2   137  138  139  140  141  142  143  144   1
  2   140  139  146  145  144  143  150  149   1
```

```
  2    145   146   147   148   149   150   151   152   1
  2    148   147   154   153   152   151   158   157   1
  2    153   154   155   156   157   158   159   160   1
  2    156   155   162   161   160   159   166   165   1
  2    161   162   163   164   165   166   167   168   1
  2    164   163   170   169   168   167   174   173   1
  2    169   170   171   172   173   174   175   176   1
  2    172   171   178   177   176   175   182   181   1
  2    177   178   179   180   181   182   183   184   1
  2    180   179   186   185   184   183   190   189   1
  2    185   186   187   188   189   190   191   192   1

*  BLOCK 53 in three steps
* 2    157   158   159   160   193   194   195   196   1
* 2    193   194   195   196   197   198   199   200   1
* 2    197   198   199   200   205   206   207   208   1
*  1600 LEVEL TO SURFACE
* 2    269   270   271   272   273   274   275   276   1
* 2    272   271   282   281   276   275   286   285   1
  2    273   274   275   276   277   278   279   280   1
* 2    281   282   283   284   285   286   287   288   1
  2    285   286   287   288   289   290   291   292   1
*  1600 SUB 2 TO 2000 LEVEL
* 2    209   210   211   212   213   214   215   216   1
* 2    213   214   215   216   217   218   219   220   1
* 2    216   215   226   225   220   219   230   229   1
* 2    221   222   223   224   225   226   227   228   1
* 2    225   226   227   228   229   230   231   232   1
* 2    228   227   238   237   232   231   242   241   1
* 2    233   234   235   236   237   238   239   240   1
  2    237   238   239   240   241   242   243   244   1
  2    240   239   250   249   244   243   254   253   1
  2    245   246   206   205   249   250   250   249   1
  2    205   206   207   208   249   250   251   252   1
  2    208   207   247   248   252   251   251   252   1
  2    249   250   251   252   253   254   255   256   1
  2    252   251   262   261   256   255   266   265   1
  2    257   258   259   260   261   262   263   264   1
  2    261   262   263   264   265   266   267   268   1

* SURFACE                                              THICKNESS
* 4    300   301   302   303     0     0     0     0  2   0

  0

* DEFINE POINT COORDINATES:
* ------------------------
*
* n      x         y        z       t1      p1      t2      p2      t3      p3
*   sub 3 drifts
     1  9316      27670     9070
     2  9328      27663     9070
     3  9345      27685     9070
     4  9330      27690     9070
     5  9316      27670     9057
     6  9328      27663     9057
     7  9345      27685     9057
     8  9330      27690     9057
     9  9300      27691     9072
    10  9330      27707     9070
```

11	9300	27707	9072
12	9300	27691	9059
13	9330	27707	9057
14	9300	27707	9059
15	9345	27710	9069
16	9345	27710	9056
17	9355	27685	9068
18	9373	27689	9068
19	9383	27707	9068
20	9356	27716	9068
21	9355	27685	9055
22	9373	27689	9055
23	9383	27707	9055
24	9356	27716	9055
25	9359	27655	9068
26	9371	27655	9068
27	9359	27655	9055
28	9371	27655	9055
29	9368	27727	9065
30	9383	27721	9065
31	9391	27782	9060
32	9377	27782	9060
33	9368	27727	9052
34	9383	27721	9052
35	9391	27782	9047
36	9377	27782	9047
37	9425	27873	9053
38	9440	27866	9053
39	9445	27879	9053
40	9432	27886	9053
41	9425	27873	9042
42	9440	27866	9042
43	9445	27879	9042
44	9432	27886	9042
45	9404	27886	9053
46	9409	27897	9053
47	9404	27886	9042
48	9409	27897	9042
49	9462	27946	9062
50	9476	27939	9062
51	9487	27962	9062
52	9479	27978	9062
53	9462	27946	9052
54	9476	27939	9052
55	9487	27962	9052
56	9479	27978	9052
57	9503	27945	9065
58	9503	27953	9065
59	9503	27945	9052
60	9503	27953	9052
61	9487	27997	9062
62	9498	27985	9062
63	9527	28046	9065
64	9510	28046	9065
65	9487	27997	9052
66	9498	27985	9052
67	9527	28046	9055
68	9510	28046	9055
69	9460	28007	9062
70	9469	28018	9062

71	9460	28007	9053
72	9469	28018	9053
73	9553	28216	9074
74	9565	28211	9074
75	9588	28245	9072
76	9574	28250	9072
77	9553	28216	9062
78	9565	28211	9062
79	9588	28245	9060
80	9574	28250	9060
81	9585	28266	9068
82	9597	28257	9068
83	9684	28389	9046
84	9671	28398	9046
85	9585	28266	9056
86	9597	28257	9056
87	9684	28389	9036
88	9671	28398	9036
89	9598	28236	9069
90	9617	28217	9067
91	9634	28224	9067
92	9607	28248	9069
93	9598	28236	9058
94	9617	28217	9057
95	9634	28224	9057
96	9607	28248	9058
97	9601	28212	9068
98	9601	28212	9057
99	9332	27376	9075
100	9348	27376	9075
101	9360	27400	9073
102	9346	27400	9073
103	9332	27376	9061
104	9348	27376	9061
105	9360	27400	9060
106	9346	27400	9060
107	9341	27440	9076
108	9356	27440	9076
109	9393	27538	9075
110	9377	27542	9075
111	9341	27440	9064
112	9356	27440	9064
113	9393	27538	9062
114	9377	27542	9062
115	9379	27565	9076
116	9398	27572	9076
117	9430	27660	9070
118	9419	27671	9070
119	9379	27565	9061
120	9398	27572	9061
121	9430	27660	9058
122	9419	27671	9058
123	9440	27689	9071
124	9463	27694	9068
125	9463	27706	9068
126	9449	27714	9071
127	9440	27689	9055
128	9463	27694	9055
129	9463	27706	9055
130	9449	27714	9055

131	9400	27689	9067
132	9435	27720	9069
133	9400	27707	9067
134	9400	27689	9055
135	9435	27720	9054
136	9400	27707	9055
137	9440	27735	9069
138	9454	27728	9069
139	9474	27774	9069
140	9455	27783	9069
141	9440	27735	9055
142	9454	27728	9055
143	9474	27774	9055
144	9455	27783	9055
145	9485	27824	9071
146	9499	27817	9071
147	9512	27867	9072
148	9496	27874	9072
149	9485	27824	9058
150	9499	27817	9058
151	9512	27867	9059
152	9496	27874	9059
153	9516	27920	9074
154	9533	27912	9074
155	9554	27956	9075
156	9541	27962	9075
157	9516	27920	9060
158	9533	27912	9060
159	9554	27956	9061
160	9541	27962	9061
161	9562	28008	9075
162	9580	28000	9075
163	9609	28041	9071
164	9595	28048	9071
165	9562	28008	9060
166	9580	28000	9060
167	9609	28041	9056
168	9595	28048	9056
169	9639	28138	9068
170	9653	28130	9068
171	9677	28175	9069
172	9663	28182	9069
173	9639	28138	9055
174	9653	28130	9055
175	9677	28175	9055
176	9663	28182	9055
177	9669	28196	9068
178	9685	28189	9068
179	9687	28204	9068
180	9670	28210	9068
181	9669	28196	9056
182	9685	28189	9056
183	9687	28204	9056
184	9670	28210	9056
185	9675	28249	9071
186	9692	28241	9071
187	9812	28535	9080
188	9797	28541	9080
189	9675	28249	9057
190	9692	28241	9057

191	9812	28535	9066
192	9797	28541	9066
193	9517	27920	9030
194	9580	27890	9030
195	9576	27946	9030
196	9553	27957	9030
197	9526	27915	9012
198	9587	27887	9012
199	9586	27941	9012
200	9563	27952	9012
205	9543	27908	8995
206	9593	27884	8995
207	9602	27934	8995
208	9572	27947	8995
209	9013	26570	8995
210	9028	26570	8995
211	9072	26680	8995
212	9057	26680	8995
213	9066	26570	8910
214	9081	26570	8910
215	9125	26680	8910
216	9110	26680	8910
217	9188	26570	8725
218	9208	26570	8725
219	9252	26680	8725
220	9232	26680	8725
221	9148	26870	8995
222	9163	26870	8995
223	9222	27090	8995
224	9207	27090	8995
225	9201	26870	8910
226	9216	26870	8910
227	9275	27090	8910
228	9260	27090	8910
229	9308	26870	8725
230	9328	26870	8725
231	9396	27090	8725
232	9376	27090	8725
233	9280	27200	8995
234	9295	27200	8995
235	9319	27260	8995
236	9304	27260	8995
237	9333	27200	8910
238	9348	27200	8910
239	9372	27260	8910
240	9357	27260	8910
241	9440	27200	8725
242	9460	27200	8725
243	9484	27260	8725
244	9474	27260	8725
245	9475	27700	8995
246	9505	27700	8995
247	9605	28010	8995
248	9585	28010	8995
249	9560	27700	8870
250	9580	27700	8870
251	9685	27980	8870
252	9665	27980	8870
253	9640	27700	8725
254	9665	27700	8725

255	9770	27980	8725
256	9750	27980	8725
257	9672	28180	8995
258	9692	28180	8995
259	9832	28530	8995
260	9812	28530	8995
261	9752	28180	8870
262	9772	28180	8870
263	9912	28530	8870
264	9892	28530	8870
265	9875	28180	8725
266	9890	28180	8725
267	9950	28530	8725
268	9930	28530	8725
269	8090	26850	10300
270	8110	26850	10300
271	8378	27520	10300
272	8358	27520	10300
273	8805	26850	9520
274	8825	26850	9520
275	9093	27520	9520
276	9073	27520	9520
277	9070	26850	9120
278	9090	26850	9120
279	9358	27520	9120
280	9338	27520	9120
281	8458	27770	10300
282	8478	27770	10300
283	8882	28780	10300
284	8862	28780	10300
285	9173	27770	9520
286	9193	27770	9520
287	9597	28780	9520
288	9577	28780	9520
289	9438	27770	9120
290	9458	27770	9120
291	9862	28780	9120
292	9842	28780	9120
300	7500	26300	10300
301	10300	26300	10300
302	10300	29300	10300
303	7500	29300	10300

```
*   GRID POINTS - use 651 to 700 only
*        GRID 1 - BLOCK 53 - SECTION
    651  9494       27962       9100
    652  9585       27919       9100
    653  9585       27919       9050
    654  9494       27962       9050

*        GRID 2 - BLOCK 54 - SECTION
    655  9473       27921       9100
    656  9564       27878       9100
    657  9564       27878       9050
    658  9473       27921       9050
```

```
*       GRID 3  -  BLOCK 52  -  SECTION
   659  9511       27997       9100
   660  9601       27954       9100
   661  9601       27954       9050
   662  9511       27997       9050

*       GRID 4  -  BLOCK 51  -  SECTION
   663  9543       28034       9100
   664  9633       27993       9100
   665  9633       27993       9050
   666  9543       28034       9050

*       GRID 4  -  BLOCK 50  -  SECTION
   667  9565       28080       9100
   668  9655       28038       9100
   669  9655       28038       9050
   670  9565       28080       9050


 0

* DEFINE MATERIAL PROPERTIES:
* --------------------------
*
* n         s1,s2,s3          ds1,ds2,ds3          t1,p1,t3,surf
* type      E,B,or Kn(p/r)    v,G,or KS(p/r)       Gn,Gs,a,b
* type      1,To,Co,So,Phi(p/r)    2,To,Sc,m,s(p/r)
*
* Model Criteria used : - Mine wide model 2-1 stress ratio
*                       - start at 2200 level
*                       - Hoek Brown Failure To/Sc/m/s
*                         - for ore zone
*                       - s1 @ 296 deg. azimuth, 0 deg. plunge
*                       - ds1,ds2,ds3 in MPA/ft.
*                       - elasto-visco-plastic case

  1     39.10,26.00,16.50   -0.0123,-0.0089,-0.0069   296,0.0,0.0,8500
  1     15.5e3,15.5e3              0.25,0.25          1550.0,1550.0
  2     0.0,0.0        70.0,70.0       3.38,3.38       0.0094,0.0094


  0

* DEFINE GRID:
* -----------
*  n     g1   g2   g3   g4
   1     651  652  653  654
   2     655  656  657  658
   3     659  660  661  662
   4     663  664  665  666
   5     667  668  669  670

   0


* Boundary CONDITION:
* -----------------
* n    bc


  2      0
* 4      0

  0
```

APPENDIX B

Summary of MAP3D (from MAP3D manual - Mine Modelling Ltd.)

MAP3D is a copyrighted three dimensional elastic boundary element modelling program from Mine Modelling Limited. This software "simulates rock mass response and predicts displacements, stresses and safety factors around rock slopes, open pits, tunnels and underground excavations in rock. The rock mass can include several zones with different moduli (stiff dykes or soft ore zones). Excavations can be intersected by multiple discrete fault planes. Fault slip and crack opening are simulated. The faults may be non-planar and gouge filled, and are permitted to intersect other faults." A summary of the input file specification is given below:

1- Control Parameters:
NLD - maximum number of load steps required.
NIT - maximum number of iterations allowed for a load step.
NPS - number of planes of symmetry used in the analysis.
RPAR - maximum relaxation parameter. (1.5 for well condition problem)
STOL - stress tolerance factor. (0.1% of far field stress)
AL - minimum allowable element side length.
AG - minimum allowable grid side length
DOL - distance over length ratio for grid discretization.
DON - distance over length ratio for element discretization.
DOC - distance over length ratio for coefficiect lumping.
DOE - distance over length ratio for element lumping.
DOG - distance over length ratio for grid lumping.

2- Block Specification identifies the block name, and the 4 to 8 points which define the block.

3- Coordinate Specification identifies the coordinates of the block points.

4- Material Properties marks the stress state, material properties, and failure criterias of each material type.

5- Grid Specification identifies the corner points of the field grid to be calculated.

6- Material Code Specification links the appropriate material code to each block.

"MAP3D is a Mining Analysis Program in 3-Dimensions. MAP3D simulates rock mass response and predicts displacements, stresses and safety factors around rock slopes, open pits, tunnels and underground excavations in rock. The rock mass can include several zones with different moduli (stiff dykes or soft ore zones). Excavations can be intersected by multiple discrete fault planes. Fault slip and crack opening are simulated. The faults may be non-planar and gouge filled, and are permitted to intersect other faults."

"The program can also be used to analyze irregular tabular mining shapes which may be non-planar, and connected to bulky three-dimensional excavations, or open pit mining. For tabular mining shapes, yielding pillars and back-filled zones can also be simulated."

"MAP3D features a user friendly mouse driven WYSIWIG graphical user interface with context sensitive, on-line help. The graphics are highly optimized for EGA and VGA adapters and permit fast three-dimensional, colour shaded, perspective viewing of the mine geometry and results. Details of the displacement, stress and safety factor at any grid point can be selected with a click of the mouse. Viewing options are selected either through mouse driven menus, or function keys. The user interacts with the graphical display of both the geometry and results by use of a mouse."

"The model formulation is based on the Boundary Element Method, and incorporates simultaneous use of both fictitious force and displacement discontinuity elements. The elastic host rock mass can be non-homogeneous, and intersected by multiple fault planes. Elasto-visco-plasticity is currently under development."

"The geometry is constructed using solids modelling technology. A series of three-dimensional, six sided building blocks are used to construct excavations and accesses. These blocks can be any desired shape and size, thus permitting complex mining geometry to be constructed easily. True three-dimensional curvilinear coordinates are fully supported."

"Input for the MAP3D model is specified from a single data file which controls program execution, specifies material properties, and describes the geometry and mining sequence. Although the file is created and modified by use of a text editor, the geometric data can be easily generated from a CAD program. Many users make extensive use of AUTOCAD to create the mining geometry used for analysis."

"During model generation, MAP3D creates a surface description of the excavation shapes by joining all intersecting blocks at their common sides. These surfaces are then automatically meshed and discretized into smaller surface elements which are used in the analysis. The discretization process optimizes the use of these elements by concentrating elements only where results are requested."

"Results are generated on a series of user specified two-dimensional grids which may slice through the excavation geometry at any desired location and orientation. These grid planes are automatically discretized into a series of field points at which displacements and stresses are calculated. The automated discretization process minimizes the use of field points by concentrating them only near excavations."

"In addition to element and grid optimization, lumping techniques are also used to further reduce computer requirements. As a result, very large problem sizes can be accommodated. This permits the user to specify the existing mining geometry in detail, then add new mining as required. Alternative mining sequences can be compared with ease by simply re-running the same data file specifying grids at alternate locations. This greatly reduces the effort required to set up and run analysis. As a result, large problem sizes can be accommodated with modest hardware requirements."

A summary of the input file specification is given below:
Project Title: "Title"
Control Parameters:

NLD
NIT
NPS
RPAR
STOL
AL
AG
DOL
DON
DOC
DOE
DOG

Block Specification:

N
'Block Name'
I1
I2
I3
I4
I5
I6
I7

I8
TYPE
THIC
C1
C2
C3

Coordinate Specification:

N
X
Y
Z
T1
P1
T2
P2
T3
P3

Material Properties:

N
SIGMA 1
SIGMA 2
SIGMA 3
DIR SIGMA 1
DIR SIGMA 2
DIR SIGMA 3
T1
P1
T3
SURF

Material Properties:

MT, Ep, Er, PRp, PRr, Gn, Gs if MT=1
MT, Bp, Br, Sp, Sr, Gn, Gs if MT=2
MT, Kp, Kr, Ksp, Ksr, Gn, Gs if MT=3

Material Properties:

MF, Top, Tor, Cop, Cor, Sop, Sor, thetap, thetar if MF=1
MF, Top, Tor, Scp, Scr, mp, ms, sp, sr if MF=2

Grid Specification:

N
I1
I2
I3
I4

Material Code Specification:

N
MC
MA

APPENDIX C

COMPARISON OF THOMPSON STRESS MEASUREMENTS TO CANADIAN AVERAGE STRESS CONDITIONS

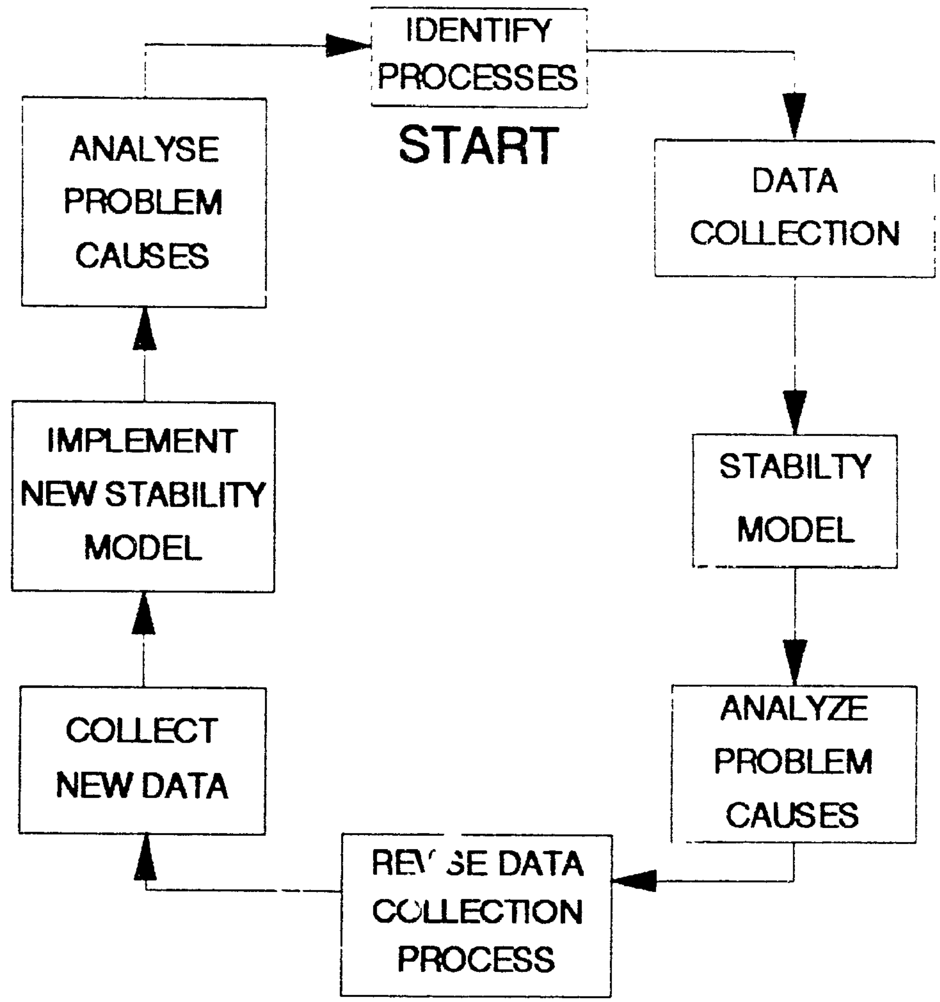
– after Herget 1990 10.91

sigma 1 = 12 10 + (0 0123) * depth
sigma 2 = 6 40 + (0 0089) * depth
sigma 3 – 1 40 + (0 0069) * depth

DEPTH (ft)	SIG. 1	SIG. 2	SIG. 3	THOMPSON MINE SIG. 1	THOMPSON MINE SIG. 2	THOMPSON MINE SIG. 3	BIRCHTREE MINE SIG. 1	BIRCHTREE MINE SIG. 2	BIRCHTREE MINE SIG. 3
0	12.10	6 40	1 40						
100	13 33	7 29	2 09						
200	14 56	8 18	2 78						
300	15.79	9 07	3 47						
400	17 02	9 96	4 16	20 30	10.30	3.10			
500	18 25	10 85	4 85						
600	19.48	11.74	5 54						
700	20 71	12.63	6 23						
800	21 94	13 52	6 92						
900	23.17	14 41	7 61						
1000	24.40	15.30	8.30	28.60	13.30	7.90			
1100	25.63	16 19	8 99						
1200	26 86	17 08	9 68						
1300	28 09	17 97	10 37						
1400	29.32	18 86	11 06						
1500	30.55	19 75	11.75				42 50	23.90	16.90
1600	31 78	20 64	12 44						
1700	33 01	21 53	13 13						
1800	34 24	22 42	13.82						
1900	35.47	23 31	14 51						
2000	36 70	24.20	15 20						
2100	37 93	25 09	15.89						
2200	39 16	25 98	16 58						
2300	40 39	26 87	17 27						
2400	41 62	27 76	17.96						
2500	42 85	28 65	18.65						
2600	44 08	29 54	19 34						
2700	45.31	30.43	20.03				31 70	26 80	15.00
2800	46 54	31 32	20 72						
2900	47 77	32 21	21 41						
3000	49 00	33 10	22 10						
3100	50 23	33 99	22 79						
3200	51.46	34 88	23 48						
3300	52 69	35 77	24 17						
3400	53 92	36 66	24 86						
3500	55 15	37 55	25 55						
3600	56 38	38 44	26 24						
3700	57 61	39 33	26 93						
3800	58 84	40 22	27 62						
3900	60.07	41.11	28 31						
4000	61 30	42 00	29 00		43 70	32 00			
4100	62 53	42 89	29 69						
4200	63.76	43 78	30 38						
4300	64.99	44 67	31 07						
4400	66 22	45 56	31 76						
4500	67 45	46 45	32 45						

APPENDIX D

CONTINUOUS IMPROVEMENT OF STABILITY PREDICTION

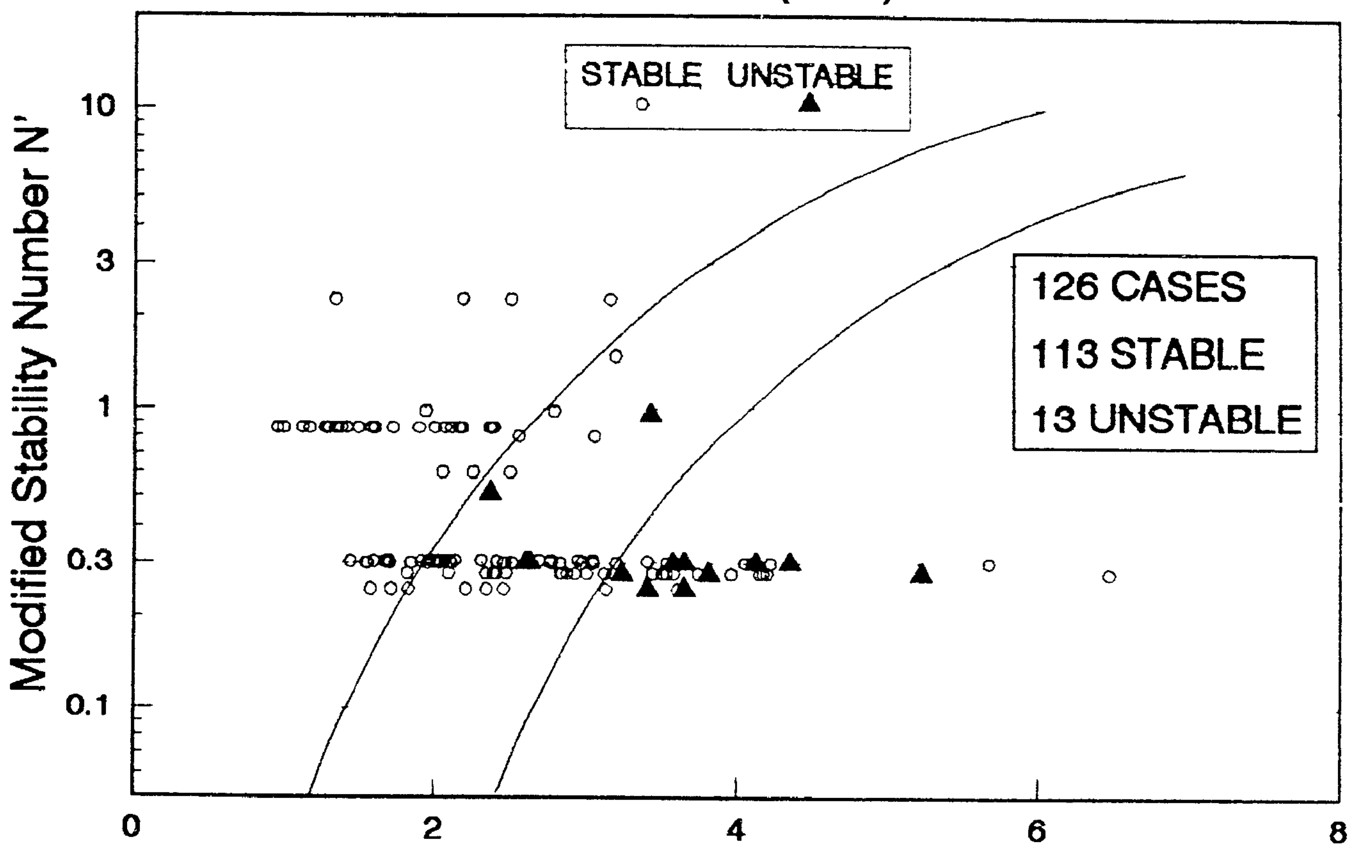


MAJOR FACTORS THAT EFFECT STABILITY

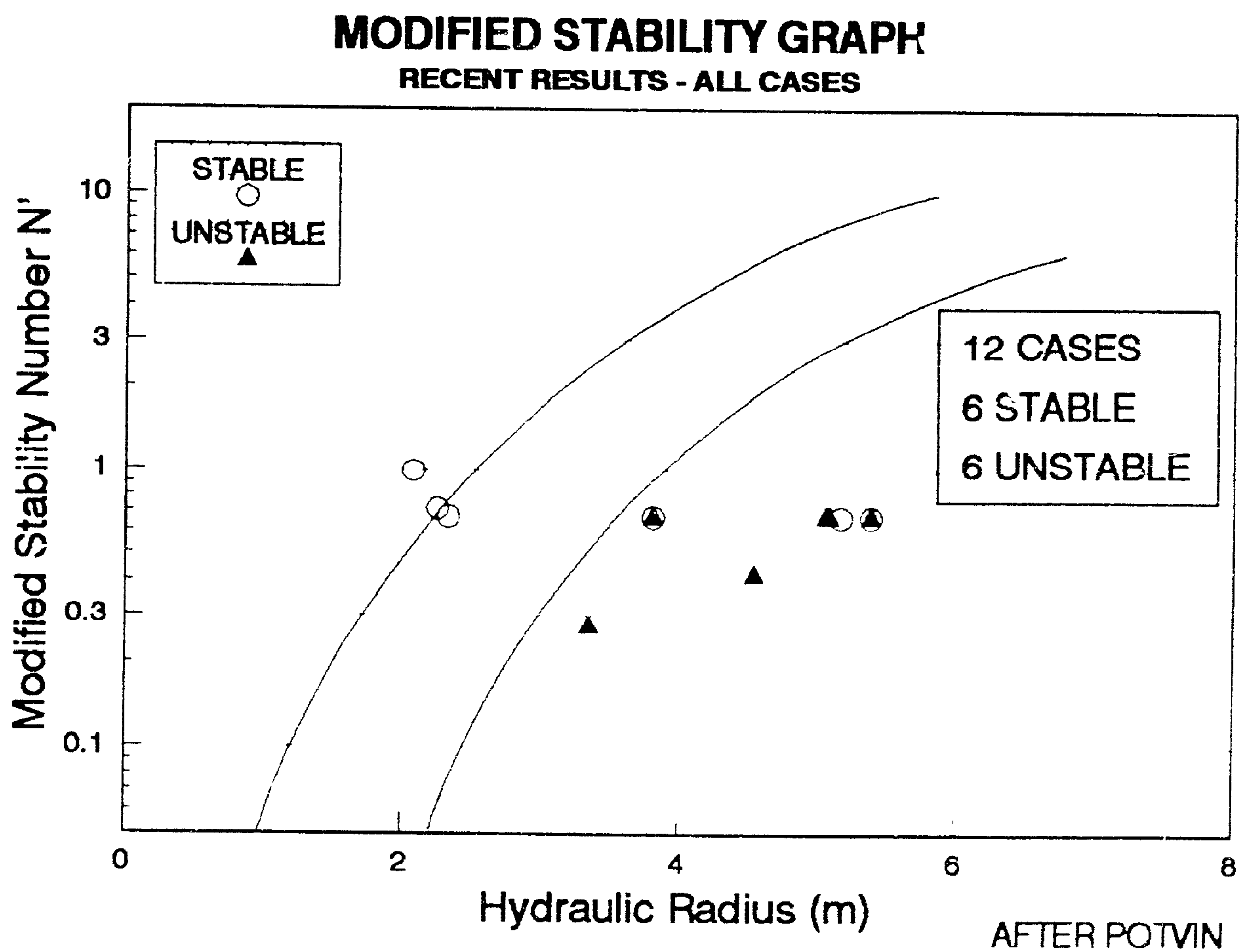
- STRUCTURE OF ROCK MASS
- STRESS
- SIZE & SHAPE
- SUPPORT
- BLASTING
- STRENGTH OF ROCK MASS
- TIME

MODIFIED STABILITY GRAPH

THOMPSON MINES (BACK)



INCO'S CANMET STUDY - MARCH 1990
AFTER POTVIN



MODIFIED STABILITY GRAPH
RECENT RESULTS - ALL CASES
STABLE
UNSTABLE
12 CASES
6 STABLE
6 UNSTABLE
Modified Stability Number N'
10
3
1
0.3
0.1
0
2
4
6
8
Hydraulic Radius (m)
AFTER POTVIN

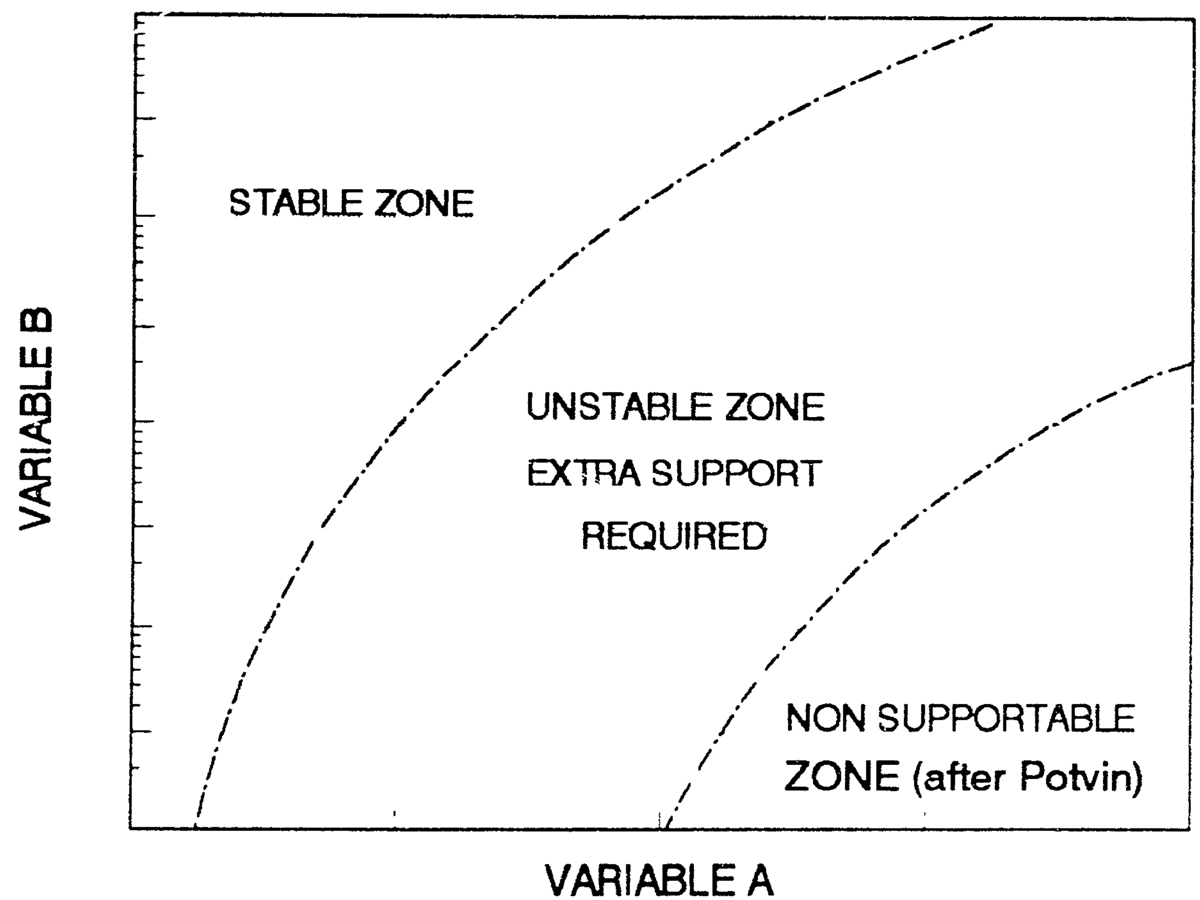
ALL RESULTS (CANMET & RECENT)

	DESCRIPTION	# STABLE STOPES	# UNSTABLE STOPES	TOTAL STOPES	CLASSIFICATION ACCURACY
1	STABLE ZONE	59	0	59	100%
2	TWILIGHT ZONE	39	4	43	NOT CLASSIFIED
3	UNSTABLE ZONE	21	15	36	42%
	TOTAL	119	19	138	54%

UN/MISCLASSIFICATION STABLE STOPES = 50%

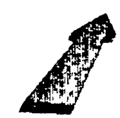
UN/MISCLASSIFICATION UNSTABLE STOPES = 21%

EXCAVATION STABILITY GRAPH



SUPPORT INTREPRETATION

ALL RESULTS

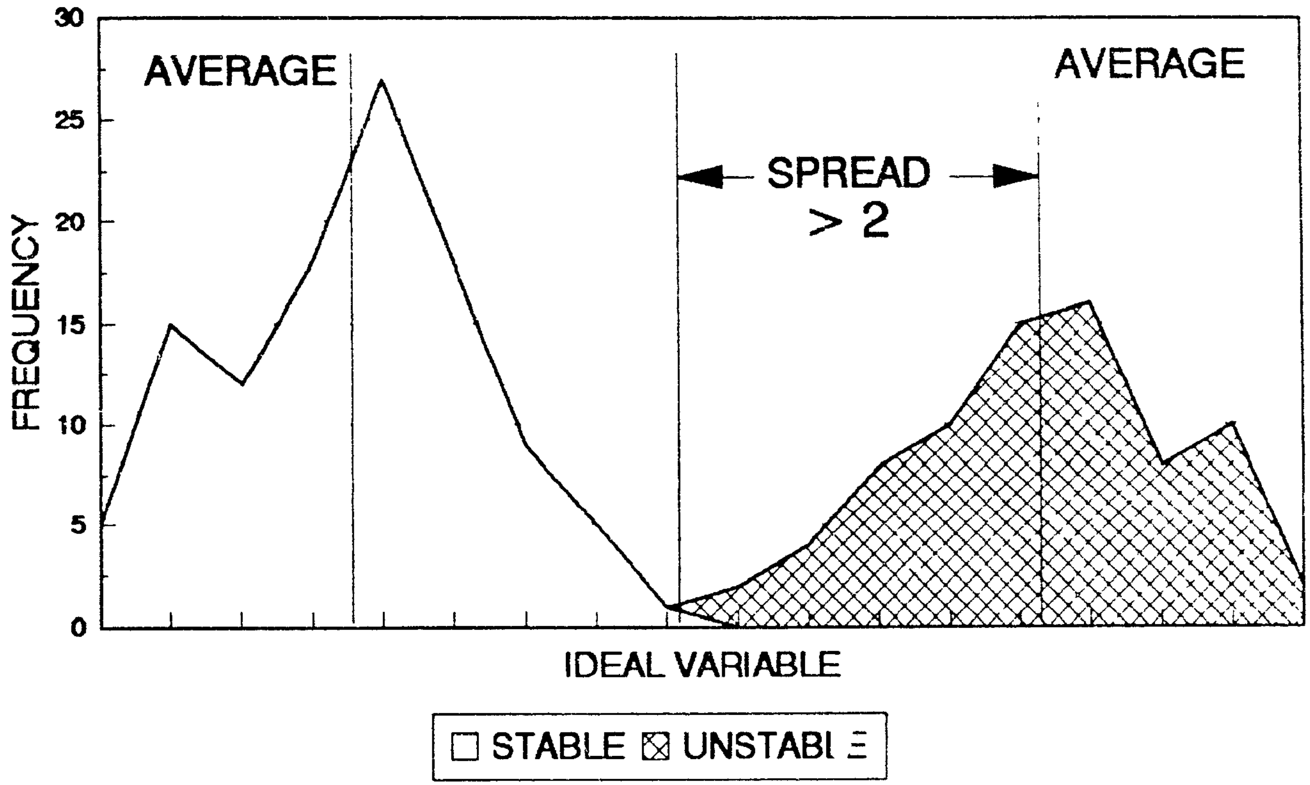
	DESCRIPTION	# STABLE STOPES	# UNSTABLE STOPES	TOTAL STOPES	CLASSIFICATION ACCURACY	INDICATOR
1	STABLE ZONE	98	4	102	96% (100)	
2	UNSTABLE ZONE	21	15	36	42% (42)	
	TOTAL	119	19	138	82% (54)	

MISCLASSIFICATION STABLE STOPES = 18% (50%)
MISCLASSIFICATION UNSTABLE STOPES = 21% (21)

SUCCESS RATES FOR VARIOUS ACTIVITES

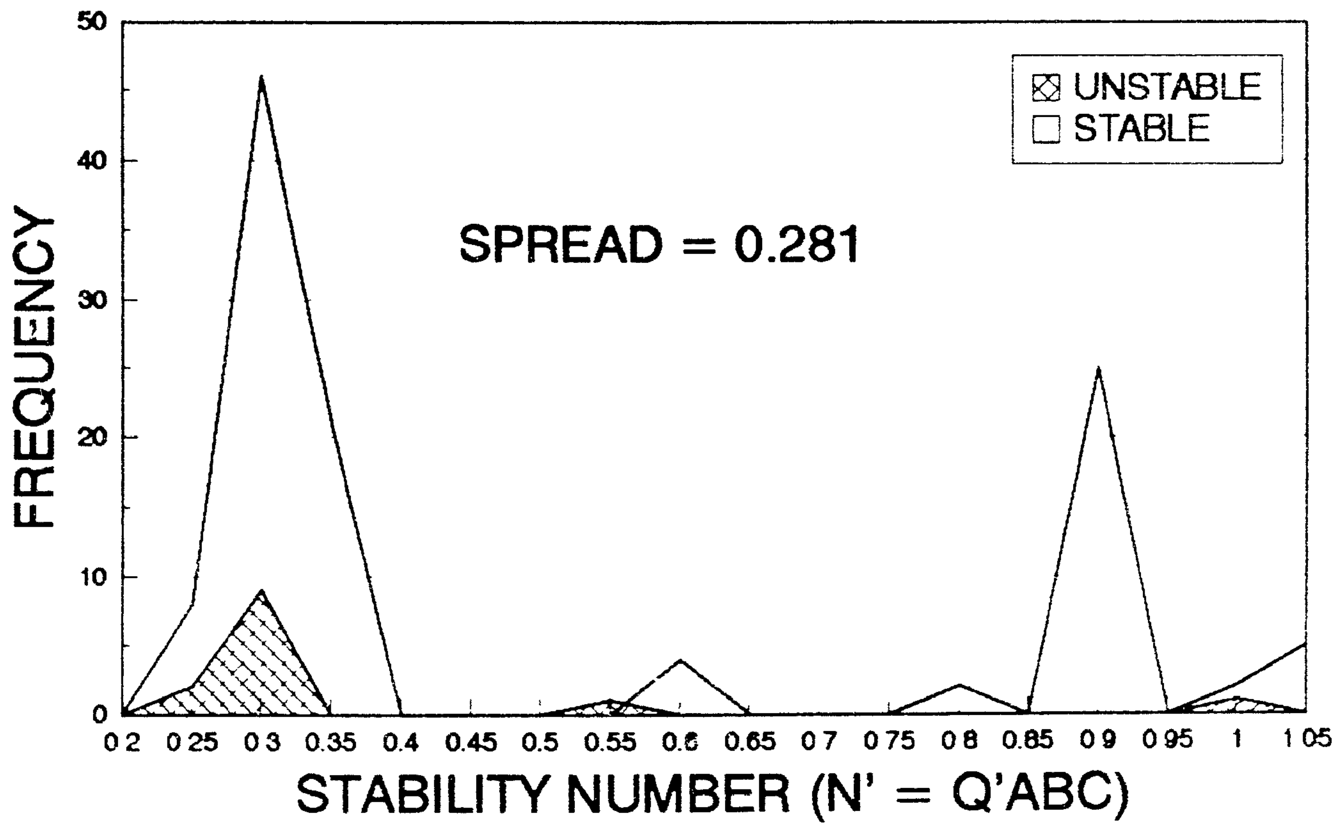
	VARIOUS ACTIVITIES	RATE OF SUCCESS OR RETURN
1	**STABLE ZONE**	96%
2	**DECISIVE MODEL**	83%
3	DOUG FLUTIE	65%
4	TAKE ON LOTTORIES	60%
5	DONALD TRUMP	59%
6	EXPOS	58%
7	**UNDECISIVE MODEL**	54%
8	FLIPPING A COIN	50%
9	MAPLE LEAFS	42% IN A GOOD SEASON
10	**UNSTABLE ZONE**	42%
11	BRIAN MULRONEY	15% OR LESS
12	GEOLOGIST IS RIGHT	10%

IDEALIZED VARIABLE



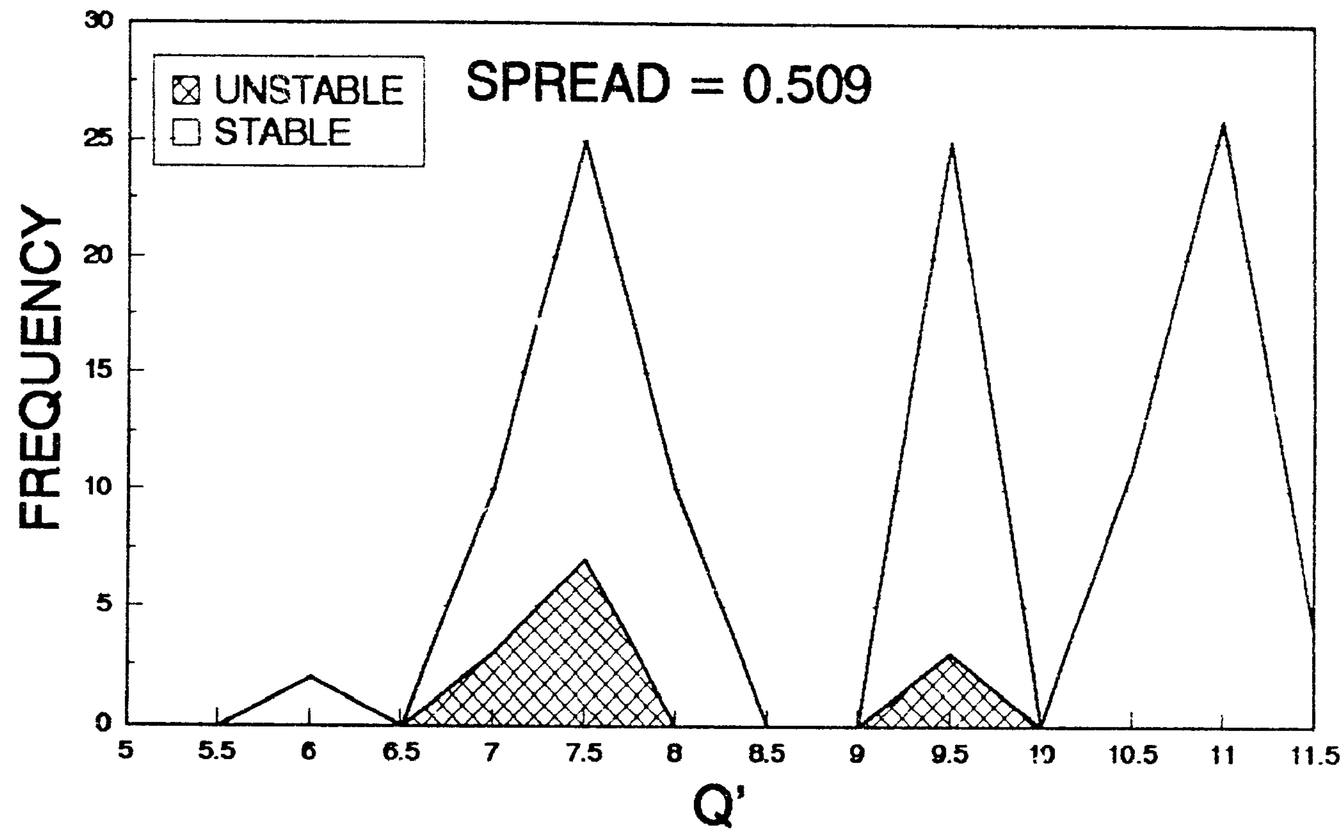
FREQUENCY DISTRIBUTION OF STABILITY NUMBER

CANMET RESULTS



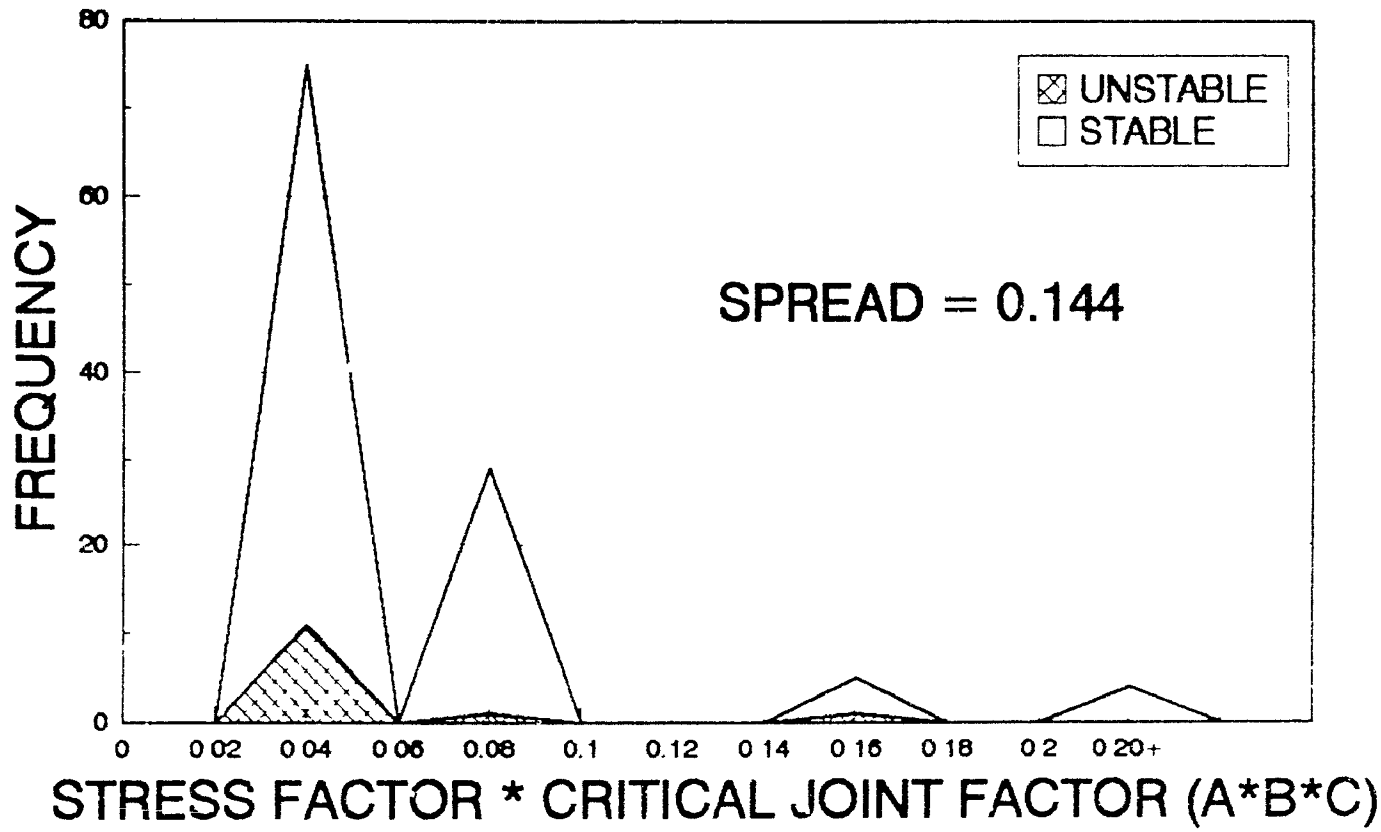
FREQUENCY DISTRIBUTION OF Q'

CANMET RESULTS



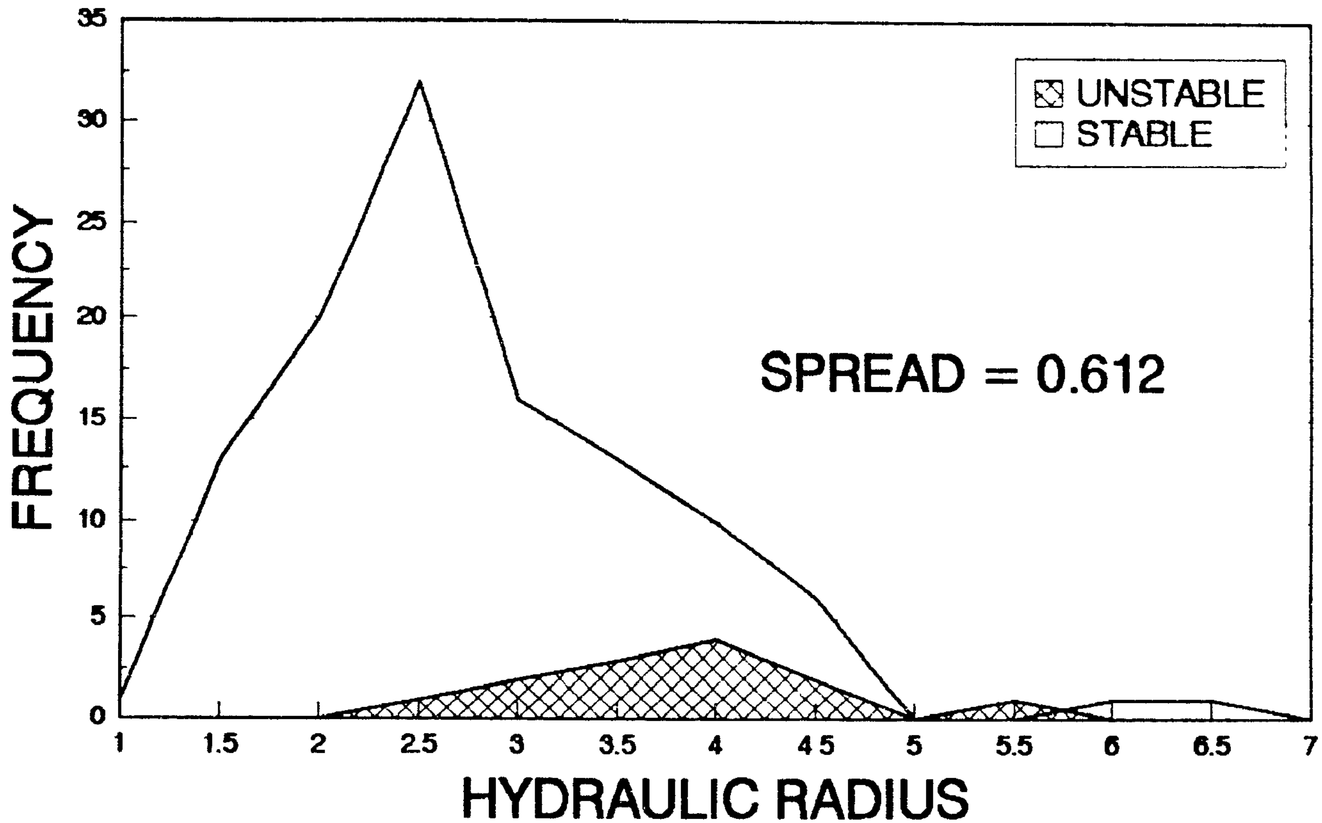
FREQUENCY DISTRIBUTION OF A*B*C

CANMET RESULTS



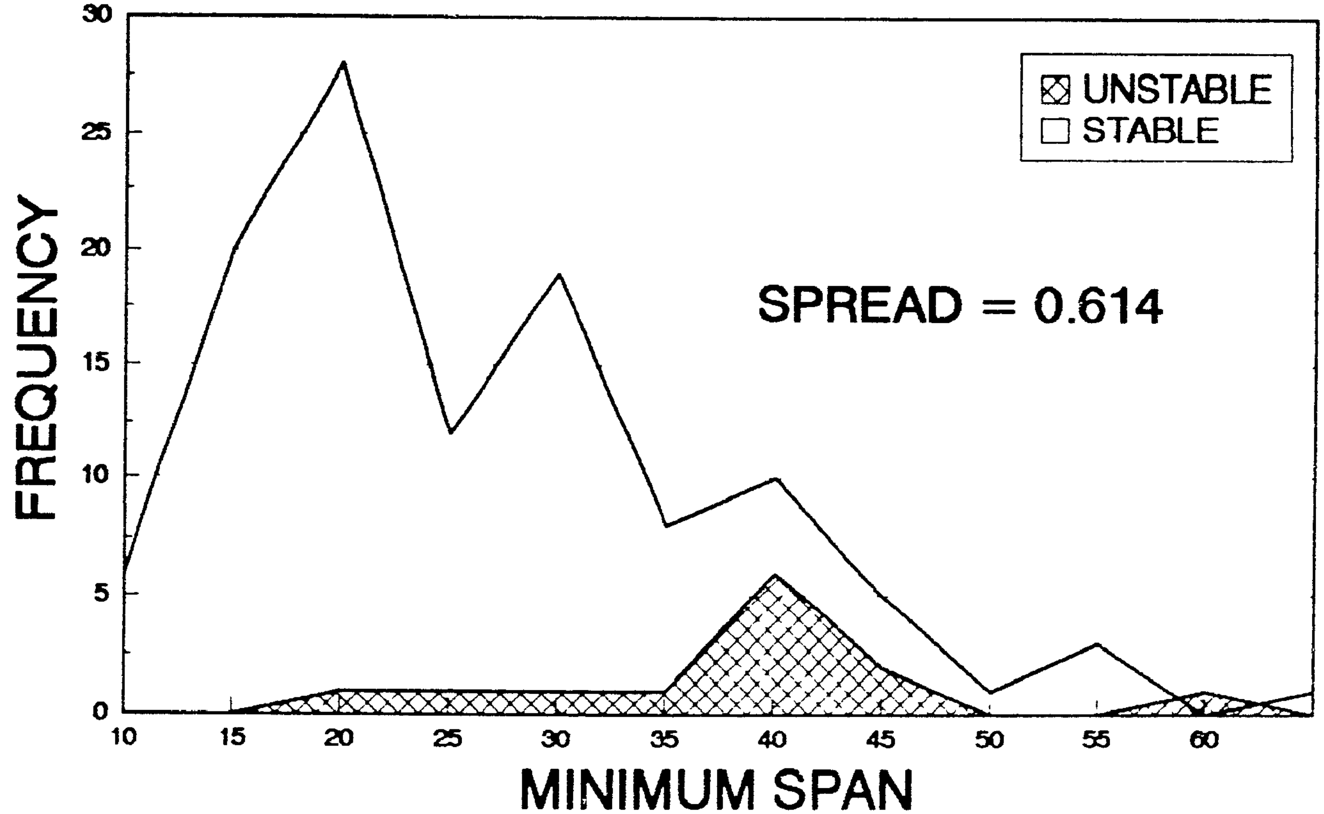
FREQUENCY DISTRIBUTION OF HYDRAULIC RADIUS

CANMET RESULTS



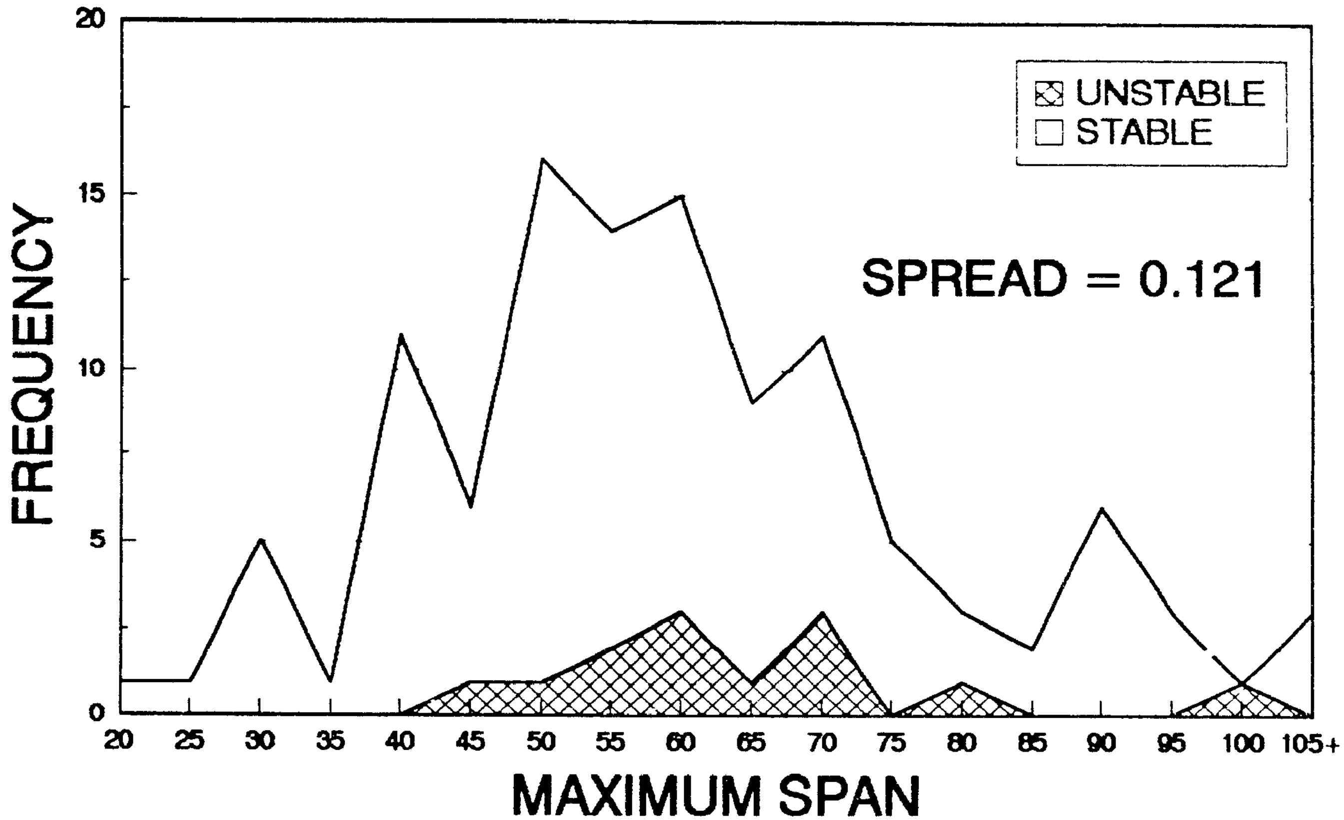
FREQUENCY DISTRIBUTION OF MINIMUM SPAN

CANMET RESULTS



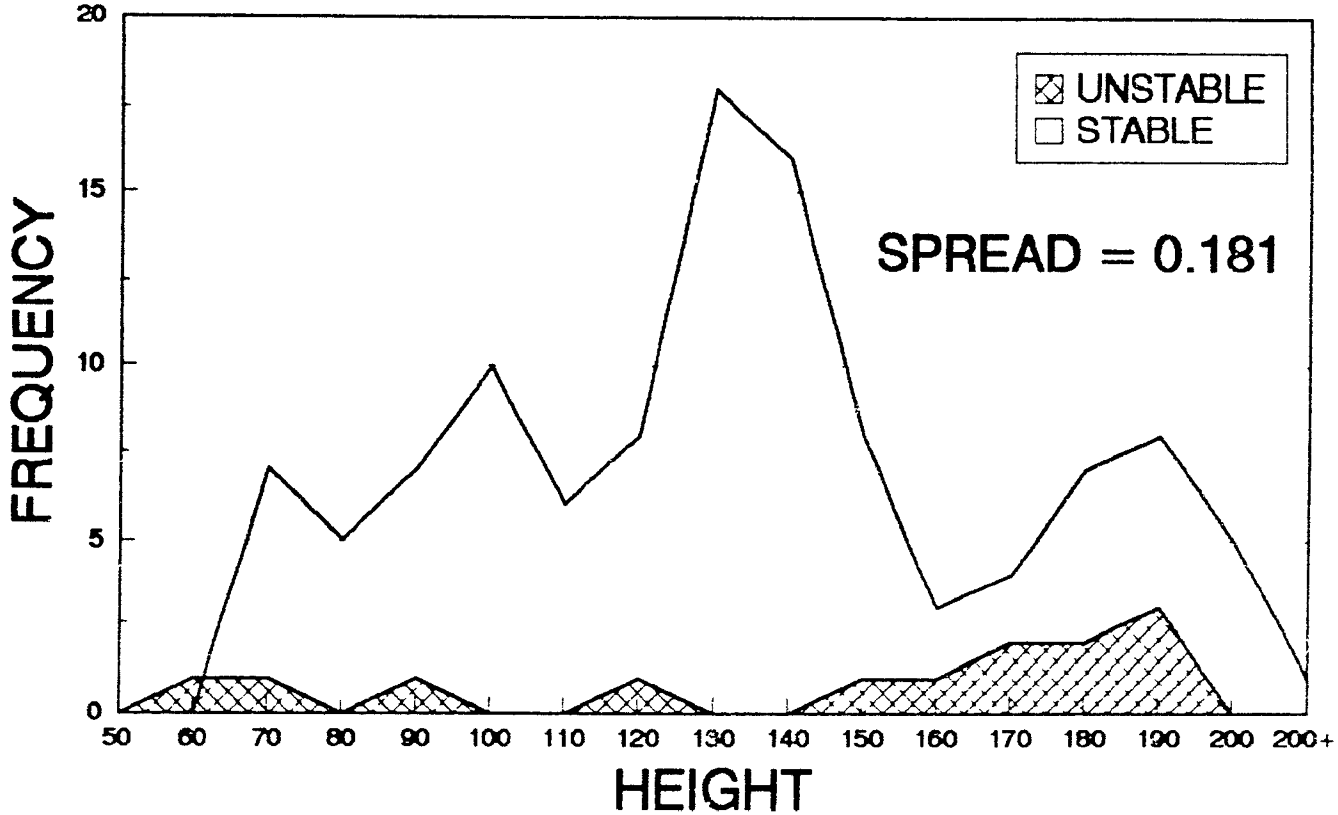
FREQUENCY DISTRIBUTION OF MAXIMUM SPAN

CANMET RESULTS



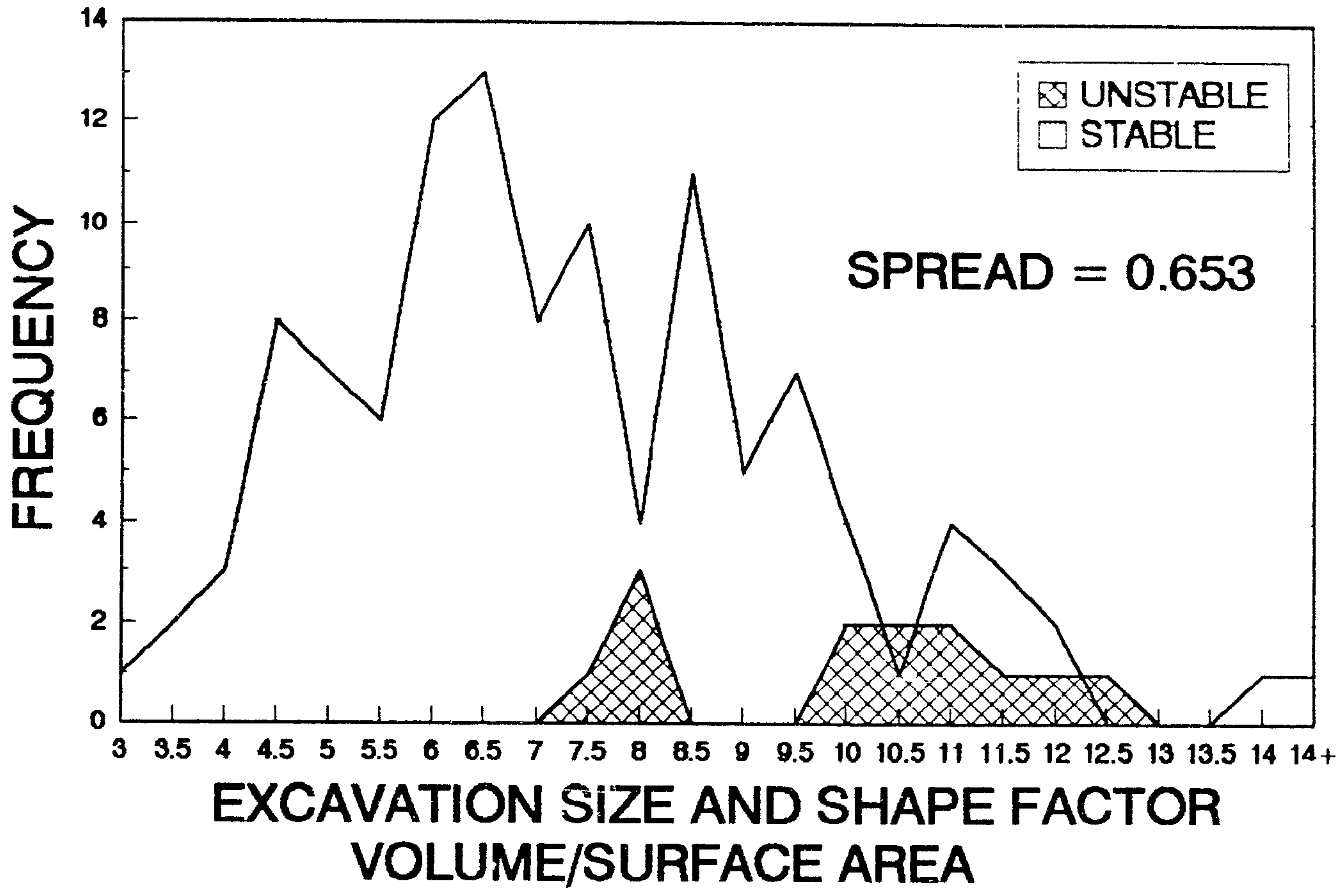
FREQUENCY DISTRIBUTION OF HEIGHT

CANMET RESULTS



FREQUENCY DISTRIBUTION OF EXCAVATION HYDRAULIC RADIUS

CANMET RESULTS



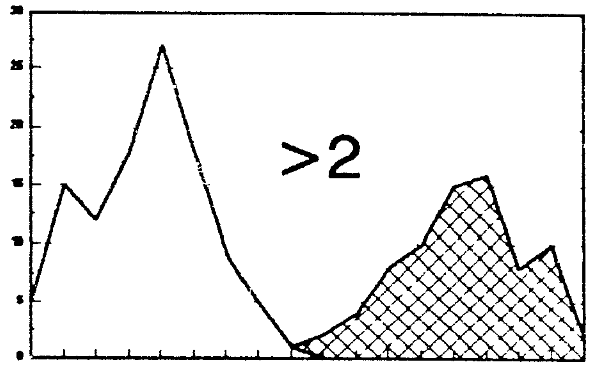
COMPARISON BETWEEN EXCAVATION HYDRAULIC RADIUS AND HYDRAULIC RADIUS

EXCAVATION HYDRAULIC RADIUS VOLUME/AREA	HYDRAULIC RADIUS AREA/PERIMETER
GOOD: - USES HEIGHT - SIMULATE MINING INADEQUATE: - HANGING WALLS - FOOTWALLS	GOOD: - HANGING WALLS - FOOTWALLS

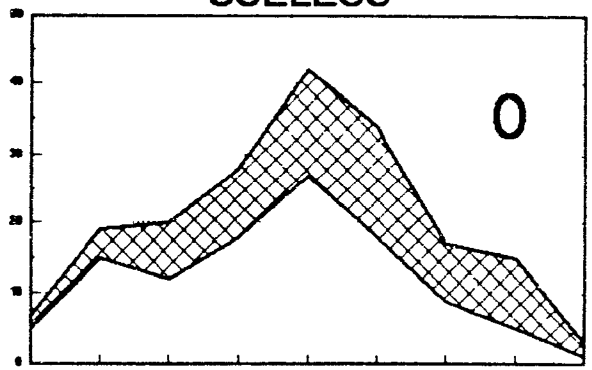
RANKING ACCORDING TO "SPREAD"

	VARIABLE(S)	"SPREAD"
1	EXCAVATION HyRad.	0.653
2	MINIMUM SPAN	0.614
3	SHAPE (HyRadius)	0.612
4	Q'	0.509
5	Q'* A * B	0.281
6	HEIGHT	0.181
7	A*B	0.144
8	A	0.122
9	MAXIMUM SPAN	0.121
10	B	0.010
11	C & CONSULTANTS	0.000

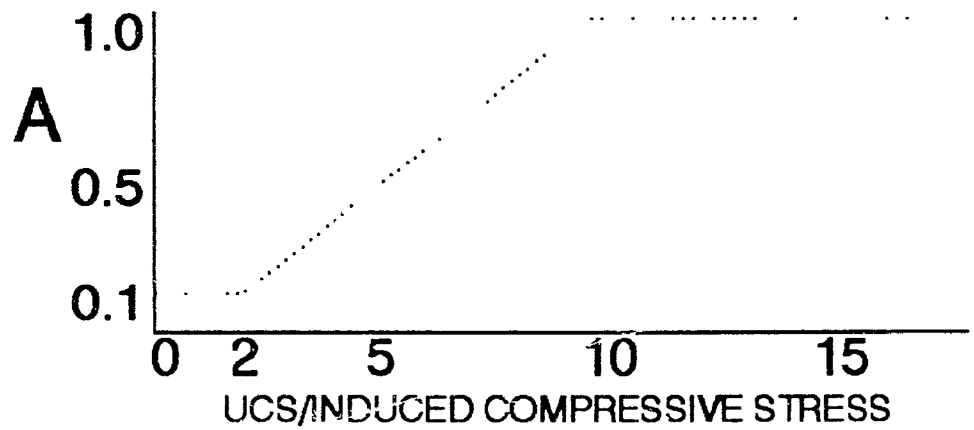
IDEAL



USELESS

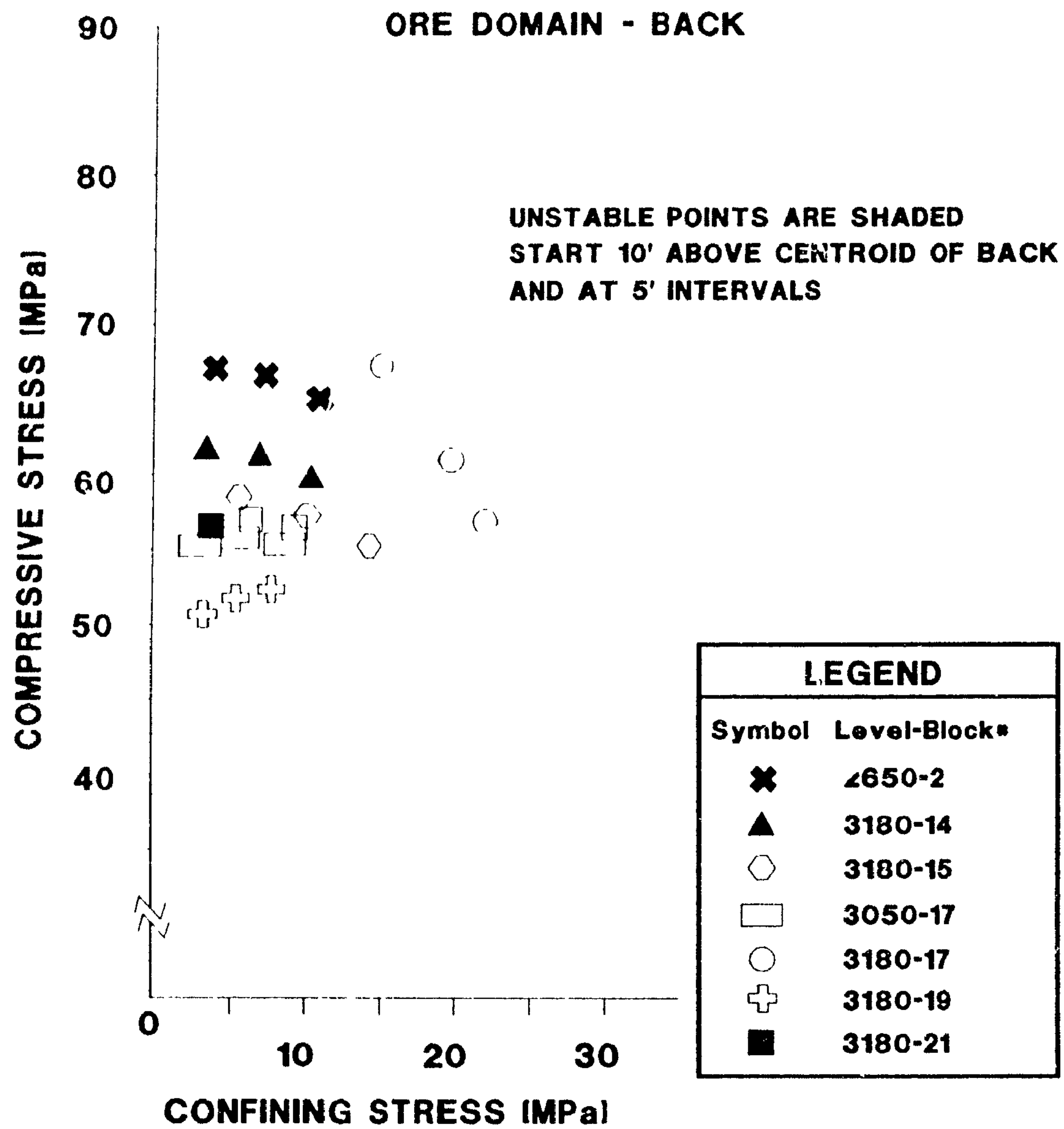


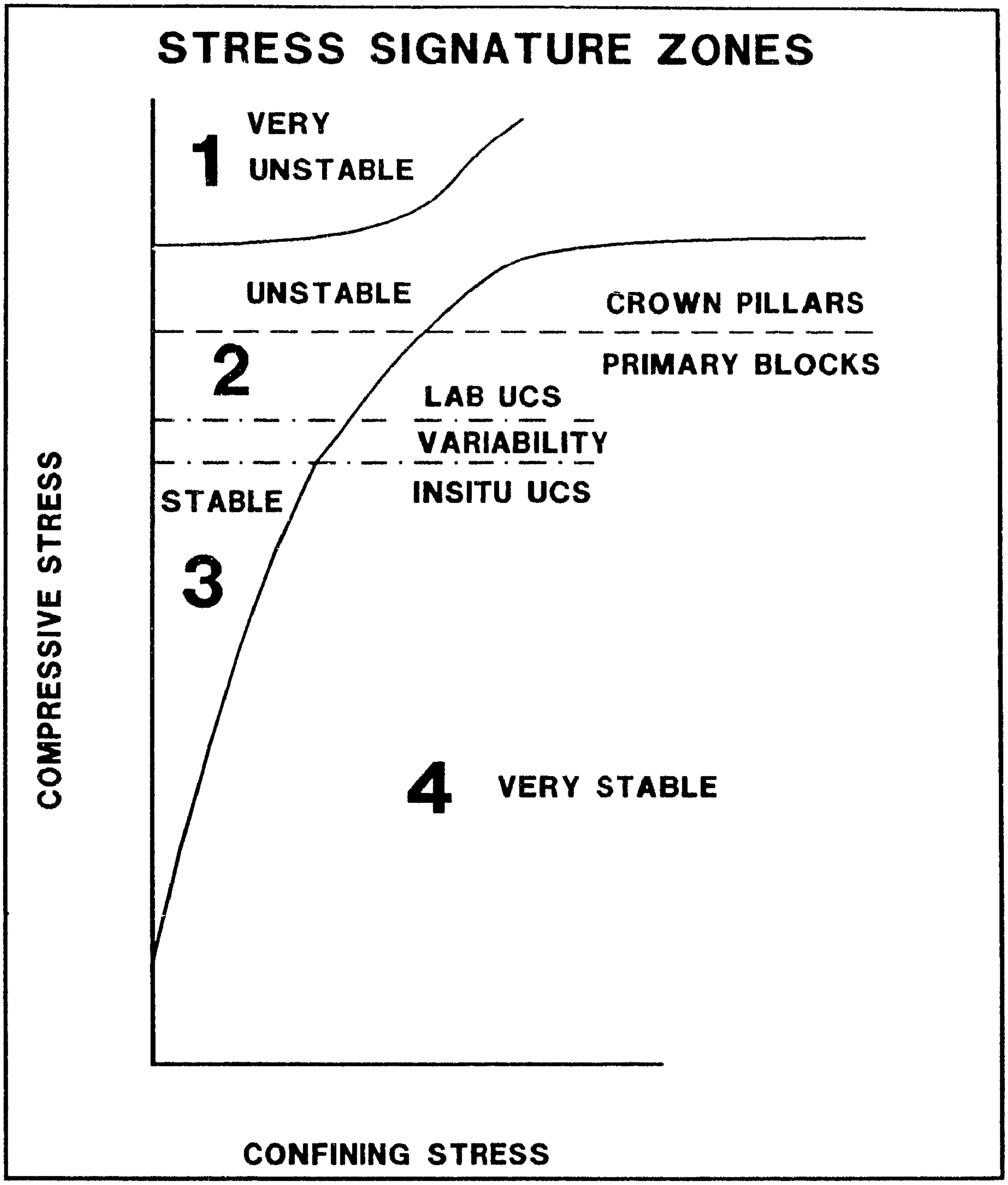
EXAMINATION OF STRESS FACTOR A



- SENSITIVE TO LAB. UCS VALUE
- CONSIDERS INDUCE STRESS AT ONLY ONE POINT
- IGNORES SIGMA 3
- POSSIBLY OVER/UNDER WEIGHTED

TYPICAL RESULTS
CALCULATED STRESS SIGNATURES




STRESS SIGNATURE ZONES
1
VERY
UNSTABLE
UNSTABLE
CROWN PILLARS
2
PRIMARY BLOCKS
LAB UCS
VARIABILITY
INSITU UCS
STABLE
3
4
VERY STABLE
COMPRESSIVE STRESS
CONFINING STRESS

PROGRESS OF EXCAVATION STABILITY MODEL