MINIMIZING DILUTION IN OPEN STOPE MINING
WITH A FOCUS ON
STOPE DESIGN AND NARROW VEIN LONGHOLE BLASTING

By

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ABSTRACT

This thesis presents the results of three years of research which focused on minimizing dilution in open stope mining. The research encompassed both stope design and narrow vein longhole blasting.

A new empirical design approach has been developed for estimating unplanned dilution from open stope hangingwalls and footwalls. The resulting design charts are based on quantifiable measurements of overbreak/slough made with the *Cavity Monitoring System (CMS),* and were developed from a comprehensive database of stoping histories compiled from six (6) Canadian underground open stoping operations. A new parameter termed ELOS (equivalent linear overbreak/slough) has been introduced and incorporated into the design charts as a measure of unplanned dilution. Theoretical justification for the design methodology has been demonstrated through a numerical modelling study examining the zone of relaxation around open stopes. Statistical methods, neural networks, and additional case histories have been used to validate the proposed design zones. This new approach to stope design is an improvement over existing methods in that it allows stope sizes to be determined based on an "acceptable" level of dilution rather than qualitative descriptions of stability such as: "stable"; "transition zone"; or "potentially unstable".

In narrow vein open stope mining, even if stopes are sized to be inherently stable (i.e. good stope design), blast induced overbreak can result in high levels of unplanned dilution. This study assessed the performance of three narrow vein blast patterns: the 3:2 pattern; the 2:1 (dice-five) pattern; and the 1:1 (stagger) pattern. The study was carried out at the Lupin Mine (NWT). The patterns were evaluated on the basis of: cost; blast damage potential; charge interaction; fragmentation; and tolerance to lapses in quality of drilling and loading practice. Guidelines have been developed regarding the choice and implementation of narrow vein patterns.

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CHAPTER 1

INTRODUCTION

1.1 GENERAL

This thesis concludes a three year study which focused on minimizing dilution in open stope mining. Funding for the research was provided by the Natural Sciences and Engineering Research Council of Canada (NSERC) and the Department of Natural Resources Canada - CANMET.

1.2 OBJECTIVES OF RESEARCH

The overall objective of this research was to improve our understanding of factors which control dilution in open stope mining and to provide guidelines for minimizing dilution during the mining process. More specifically, the objectives of this research were twofold:

1. To develop an empirical method of designing open stopes based on quantifiable measurements of overbreak/slough from open stope hangingwalls and footwalls. A design tool such as this can be used to size stopes based on a certain acceptable level of dilution. A move away from qualitative estimates of stability to quantitative estimates of stability represents an improvement over existing stope design methods.

2. To evaluate the performance of various narrow vein longhole blast patterns with regard to: costs; blast damage potential; charge interaction; fragmentation; and impact of drilling and loading quality, with the objective of developing guidelines for narrow vein blast pattern selection. Open stope narrow vein mining represents a significant challenge to the mining industry since there is very little tolerance for dilution. The research was conducted at the Lupin Mine, NWT. Three (3) narrow vein patterns were assessed: the 3:2 pattern; 2:1 (dice-five) pattern; and the 1:1 (stagger) pattern. With regard to this study, narrow vein refers to orebodies with widths ≤ 2.5m.

The two objectives stated above are inter-related with regard to minimizing dilution in open stope mining, since even if stopes are sized to be inherently stable, blast induced overbreak can still result in high levels of dilution for narrow vein orebodies. For example, consider a 2m wide orebody, if 0.5m of blast induced overbreak is realized from both the hangingwall and footwall, the volumetric dilution due to overbreak is 50%. It can be appreciated from this example that minimizing blast induced overbreak is extremely important in narrow vein open stope mining.

1.3 THESIS OVERVIEW

A brief overview of the contents of each chapter is presented below.

1.3.1 Development of an Empirical Design Approach for Predicting Unplanned Dilution From Open Stope Hangingwalls and Footwalls - Chapters 1 to 8

Chapter 1: Defines dilution and recovery and discusses the economic significance of dilution to open stope mining. A detailed examination of factors which influence dilution and recovery in open stope mining is presented.

Chapter 2: Evaluates existing empirical stope design methods with regard to estimating dilution from open stope surfaces.

Chapter 3: Discusses the use of the Cavity Monitoring System (CMS) for quantifying dilution from open stopes. Two new parameters, ELOS (equivalent linear overbreak/slough), and ELLO (equivalent linear lost ore), are introduced which are measures of hangingwall and footwall performance.

Chapter 4: Describes the development of the CMS database which is comprised of stope performance data collected from six (6) underground open stoping operations. A detailed description of the data contained in the database is presented.

Chapter 5: Details the incorporation of the ELOS parameter into the Modified Stability Graph Method (Potvin, 1988) and the development of a new design graph based on RMR. The validity of the proposed design charts is examined through the use of statistics, neural networks, and additional case histories.

Chapter 6 : Theoretical justification for the ELOS design approach is examined through a numerical modelling study of the zone of relaxation around open stope hangingwalls and footwalls. Model ELOS values and actual ELOS values from the CMS database are compared.

Chapter 7 : Relationships between ELOS and other CMS database parameters are examined through the use of scatter plots and neural networks. The analyses presented in this chapter attempt to quantitatively evaluate what factors, other than those accounted for in the stability graph method, influence hangingwall and footwall stability.

Chapter 8: Finalized versions of the design charts are presented and formal definitions for the ELOS design zones are given. Limitations of the design approach are discussed.

1.3.2 Assessment of Narrow Vein Longhole Blast Patterns at the Lupin Mine Chapters 9 to 14

Chapter 9 : Presents a general description of the Lupin Mine and the geotechnical characteristics of the ore and wall rocks are discussed. A historical review of the narrow vein blast patterns used at the mine is given.

Chapter 10: The methods used to assess blast performance in this study are discussed.

Chapter 11: 3:2 pattern blast monitoring results are presented. Field trials with 100/3800 dual delay detonators and short period non-electric delays are discussed.

Chapter 12 : Cost savings associated with implementing the more optimized 2:1 (dice-five) and 1:1 (stagger) patterns are presented.

Chapter 13: Design and implementation of the 2:1 (dice-five) and 1:1 (stagger) patterns is discussed. Blast monitoring results from field trials are presented.

Chapter 14: Presents a comparison of blast monitoring results for the three narrow vein patterns evaluated. Blast damage potential, charge interaction, fragmentation, and impact of drilling and loading practices are discussed. The main findings from the blast monitoring are summarized.

1.3.3 Thesis Conclusions - Chapter 15

Chapter 15: The main findings from Chapters 1 to 14 are summarized and guidelines for stope design and narrow vein pattern selection, design, and implementation are presented.

1.4 DEFINING DILUTION

The dilution and recovery realized for a particular open stope are a measure of the quality of the design and mining practice. A good design is one that maximizes recovery and minimizes dilution, bearing in mind that the two measures are inter-dependent (i.e. achieving a certain recovery may only be possible at the expense of accepting a certain level of dilution).

Dilution can be defined as the contamination of ore by non-ore material during the mining process (Wright, 1983). The consequences of this contamination are as follows:

- the actual amount of material extracted will be larger than what is necessary to obtain the same equivalent metal content.
- the grade of the run-of-mine ore will be lower than the estimated in-situ grade.

Scoble and Moss (1994) define *Total Dilution* as the sum of the *Planned Dilution* and the *Unplanned Dilution,* where:

Planned Dilution is the non-ore material (below cutoff grade) that lies within the designed stope boundaries (mining lines) as determined by: the selectivity of the mining method; the continuity of the orebody along strike and along dip; and the complexity of the orebody shape.

Unplanned Dilution is additional non-ore material (below cutoff grade) which is derived from rock or backfill outside the stope boundaries (mining lines). Incorporation of this material is due to: blast induced overbreak; and/or sloughing of unstable wall rock or backfill.

Figure 1.1 is a schematic illustrating the above.

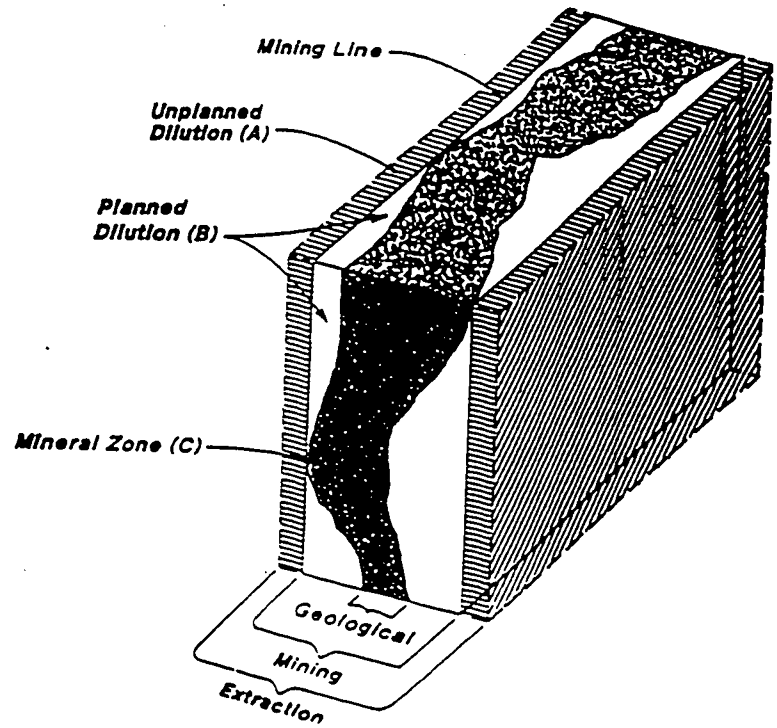


Figure 1.1 Planned and unplanned dilution (From Scoble and Moss, 1994)

Various methods for calculating dilution exist. For this reason one should be very cautious when comparing values from different operations. Scoble and Moss (1994) state that the two most common methods are based on tonnage as follows:

Dilution = Tonnes Waste / Tonnes Ore (Eq. 1.1)

Dilution = Tonnes Waste / (Tonnes Ore + Tonnes Waste) (Eq. 1.2)

Pakalnis et. al. (1995) recommends using equation 1.1 as a standard measure of dilution since equation 1.2 is much less sensitive to increases in waste. This is depicted graphically in Figure 1.2. With regard to this study, equation 1.1 was used.

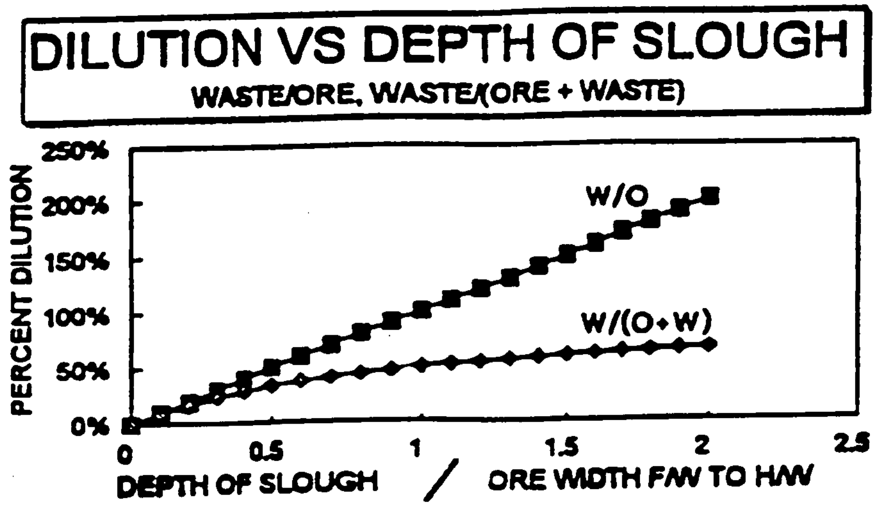


Figure 1.2 Comparison of dilution equations (From Pakalnis et.al., 1995)

1.5 THE COST OF DILUTION IN OPEN STOPE MINING

Dilution directly increases the cost of production (i.e. cost per unit weight of metal mined), however, assigning an accurate cost to dilution is difficult since it is comprised of both direct and indirect cost components.

The direct costs are relatively straight forward since dilution must bear the costs of: mucking; tramming; crushing; conveying; hoisting; milling; backfilling; and disposal to a tailings facility. From experience and literature (Anderson et. al, 1995; Dunne et. al., 1996), typical costs for mining, milling, and administration, to handle the waste material, range between approximately $30-40/tonne. This represents a very significant cost.

For example, Anderson et. al. (1995) states that the cost of dilution at the Golden Giant Mine (Hemlo Gold Mines Inc.) is approximately $5.4 million per year (mining rate = 3000 tonnes/day; avg. 14% dilution). Another factor to consider is that if excess dilution prevents an operation from meeting production quotas, it can be argued that the dilution should also bear some of the operations infrastructure costs (i.e. fixed costs: power; buildings; camp costs; transportation; etc..). Infrastructure costs can be quite high, especially for mines in remote locations. For example, the Lupin mine located in the Northwest Territories has infrastructure costs of approximately $30-35/tonne. If infrastructure costs are included with mining, milling, and administration costs, the direct cost of dilution could be as high as $60-75/tonne.

It is more difficult to assign values to the indirect costs, since they represent those factors in the mining and milling system which contribute to inefficiency in the operation. A number of examples are given below.

- Dilution often enters the stope as oversize slough material. If drawpoints become plugged, significant production delays and secondary blasting costs may be incurred.

- Depending on the location of the oversize slough material within the stope, secondary blasting may not be possible. This may result in lost ore if the oversize has fallen on, or, is blocking access to broken ore.

- Dealing with oversize and additional tonnage from dilution lowers productivity and lengthens the time to complete mucking of a stope. This effects the mining schedule and may necessitate changes to the mine plan. Subsequently, this may require that additional time be spent on planning and re-directing resources to address the problem. In some instances, new areas are brought into production sooner than anticipated, occasionally at the expense of certain information. For example, additional diamond drilling to better determine ore contacts.

- In bottom-up mining sequences, significant wall sloughing can undercut the walls of planned overlying stopes, potentially resulting in increased dilution when these stopes are mined.

- Slough from stope walls can damage remote scoops and occasionally results in the loss of a scoop. Repairs and/or replacement are costly. Production may be affected in the short term.

- Mills are designed to operate at a certain mill feed grade, lower grade material can unbalance the system resulting in decreased mill recoveries.

Possibly the most serious cost of dilution is the lost opportunity cost resulting from ore being displaced by waste within the mine/mill circuit. In other words, the cost of the deferred earnings from mining and milling waste instead of ore. The economic ramifications of displacing ore with waste are largely dependent on an operations mining and milling capacity. The worst scenario with regard to profitability is an operation which is running at its peak mining and milling capacity. If production quotas are not being met due to excess dilution, it may not be possible to make up shortfalls in production until the end of the mine life. The net effect is an increase in the planned mine life. Subsequently, yearly earnings are reduced and additional costs are incurred due to the extended life, resulting in an overall decrease in net present value for the project. The other scenario to consider is an operation with excess milling capacity. In this case, if excess dilution is being incurred, additional tonnes can be processed to ensure that production quotas are met. Although a lost opportunity cost still exists, there is no increase to the planned mine life, and thus the impact on the profitability of the project is not as serious. It does suggest, however, that the planned mine life may be longer than what is necessary, and that profitability could be improved by minimizing dilution and subsequently reducing the mine life by increasing the production rate, resulting in an increase in net present value for the project. Pakalnis et.al. (1995) presented an interesting third scenario, where cutoff grade is increased in reaction to excessive dilution in an attempt to maintain the head grade of the ore feeding the mill. In this case, the total quantity of metal produced during the life of the mine would be reduced and the opportunity cost would then apply to the unmined portion of the deposit.

From the above discussion it can be appreciated that assigning a concrete cost to dilution is very difficult, however, it cannot be argued that the cost is very significant and can seriously effect the profitability of a mining operation. Unfortunately, a certain amount of dilution must be expected given the many factors involved with delineating, developing, and ultimately extracting ore. However, minimizing dilution to the lowest practical levels, should be a goal for all mining operations. This requires that more concern be placed on the quality of tonnes of ore hoisted rather than the quantity of tonnes. Correspondingly, less emphasis should be placed on minimizing the cost/tonne and more emphasis placed on minimizing the cost/unit wt. of metal mined. As pointed out by Ingler (1975), sometimes increasing the cost/tonne (i.e. through improvements in drilling and blasting or perhaps even a higher cost mining method) to reduce dilution, can turn out to be a profitable alternative.

1.6 WHAT IS ACCEPTABLE DILUTION?

Ideally, an acceptable level of dilution is one that does not lower the average mill feed grade below a value which represents an acceptable return on investment for the operation. At the worst, the level of dilution should not be such that the average mill feed grade is lowered below the break-even cut-off grade for the operation. Pakalnis et.al. (1995) state that what is considered acceptable dilution is a function of: ore grade; grade of dilution material; costs; and metal prices. Consequently, the level of acceptable dilution varies from one mine operation to another.

An example is presented below which demonstrates an approach that could be used to calculate the maximum level of acceptable dilution for an open stope in a gold mine.

Example

Stope Dimensions:	5m wide x 15m strike x 17m height
S.G. of Ore:	3.48 tonnes/m^3
Undiluted Grade of Ore:	0.35 oz/tonne
Grade of Dilution Material:	0.0 oz/tonne
Total Cash Costs:	$68 US/tonne
Recovery of Ore in Stope:	100%
Mill Recovery:	92%
Gold Price:	$345 US/oz
Minimum Profit:	10%

1) Determine Minimum Acceptable Mill Feed Grade:

i) $$\frac{\text{Mining Cost (\$/tonne)}}{\text{(Ore Grade (oz/tonne)) x (Mill Recovery)}} = \text{\$Cost/oz}$$

ii) $$\frac{\text{(Gold Price (\$/oz)) - (\$Cost/oz)}}{\text{\$Cost/oz}} = \text{\%Profit}$$

Solving for $Cost/oz in (ii), substituting into (i), and solving for Ore Grade (oz/tonne) gives:

iii) $$\frac{\text{Mining Cost (\$/tonne)}}{\text{(Mill Recovery) x (Gold Price (\$/oz) / (Profit + 1))}} = \text{Ore Grade (oz/tonne)}$$

Solving:

$68 US/tonne / ((0.92) x (($345 US/oz)/ (0.1 + 1))) = 0.236 oz/tonne

Minimum Acceptable Mill Feed Grade = 0.236 oz/tonne

2) Determine Maximum Acceptable Level of Dilution

i) $$\text{Diluted Grade} = \frac{((\text{Tonnes Was.})(\text{Was.Grade})) + ((\text{Tonnes Ore})(\text{Undil. Ore Grade}))}{\text{Tonnes Waste} + \text{Tonnes Ore}}$$

Assuming the waste has negligible grade, solving for tonnes of waste gives:

ii) $$\text{Tonnes Waste} = \frac{(\text{Tonnes Ore})(\text{Undil. Ore Grade})}{\text{Diluted Grade}} - \text{Tonnes Ore}$$

Solving:

(((4437 tonnes)(0.35 oz/tonnes))/ (0.236 oz/tonne)) - 4437 tonnes = 2143 tonnes waste

iii) $$\%\ \text{Dilution} = \frac{\text{Tonnes Waste}}{\text{Tonnes Ore}} \times 100$$

Solving:

(2143 tonnes / 4437 tonnes) x 100 = 48%

The Maximum Acceptable Dilution = 48%

It should be appreciated that small changes in the values of the input variables can dramatically effect the calculated maximum acceptable dilution. For example, changing the undiluted grade to 0.3 oz/tonne in the above example, reduces the maximum acceptable dilution to 27%. If in addition, the price of gold dropped to $320 US/oz, the maximum acceptable dilution would be 18%.

It should be noted that for a given mine operation, mining costs may vary considerably for different parts of the deposit. For example: cost change with depth; cost difference with ore width; and cost difference by mining method. In addition, mining costs may be broken up into categories such as: total cash costs (i.e. applies to main stoping areas); post development costs (i.e. may apply to pillar recovery); and post development and infrastructure costs (i.e. may apply to low grade tonnage mined over and above the daily required tonnage - incremental ore).

The decision on what value to use as minimum profit will vary for different operations since it will be related to the projects economics as a whole and corporate philosophy pertaining to what is an acceptable return on investment.

1.7 QUANTIFYING DILUTION AND RECOVERY IN OPEN STOPE MINING

If serious efforts are to be made with regard to controlling dilution and maximizing recovery, accurate methods of quantifying these parameters are required.

Quantifying dilution and recovery in open stope mining has traditionally been very difficult due to:

- less accurate ore delineation prior to mining, as compared to more selective mining methods such as cut and fill, thus less understanding with regard to ore loss and planned dilution;
- the non-entry nature of the mining method which, until recently, limited the accurate quantification of recovery and unplanned dilution.

Due to these difficulties, dilution and recovery were/are often evaluated based on visual observation and/or reconciled from data such as: design tonnes vs. mucked tonnes; design stope grade vs. mill head grade; and design stope grade vs. muck sample grades. Due to inherent inaccuracies, evaluations of dilution and recovery with these methods are often global estimates at best.

Quantification of unplanned dilution has been made possible in recent years through the application of non-contact laser rangefinders (Miller et.al., 1992). Actual surveying of open stopes is now possible and the dimensions of excavated stopes can be accurately determined, thus permitting quantifiable values for recovery and unplanned dilution to be calculated. This technology represents a major breakthrough with regard to understanding the processes influencing unplanned dilution.

Quantification of planned dilution is more difficult, and still represents a challenge, since it is dependent on the accuracy of ore delineation, which is related to the density of available information (i.e. diamond drillhole data; blasthole sludge or dust samples; geophysical logging data; distance between mapped sub-levels). Stone (1985) surmises that poorly defined ore contacts are a major reason why many mines have to use some arbitrary factor to make "as mined" grades correlate with the ore reserve grades. A study carried out by Braun (1991) showed how ore loss and planned dilution may vary according to information density, refer to Figure 1.3. His study showed that by reducing drill hole spacing from 100ft. to 25ft., planned dilution was reduced by 8.53% and the quantity of mineable ore was increased by 6.39%. A similar study can be found in Puhakka (1990). The information density required to accurately define ore contacts is dependent on the orebody shape and complexity, and therefore varies between different operations. Studies akin to the one carried out by Braun (1991) should be done at all operations to quantify the *cost/benefit* of reducing planned dilution and ore loss through more accurate ore contact interpretations (i.e. through tighter diamond drill hole spacing; smaller sub-level interval; etc.). Note that stope surveys when coupled with the reconciled grade for a stope can provide some information regarding the accuracy of ore delineation. For example, if stope surveys consistently show minimal dilution, but the reconciled stope grades are always low, one possible explanation may be ore delineation problems.

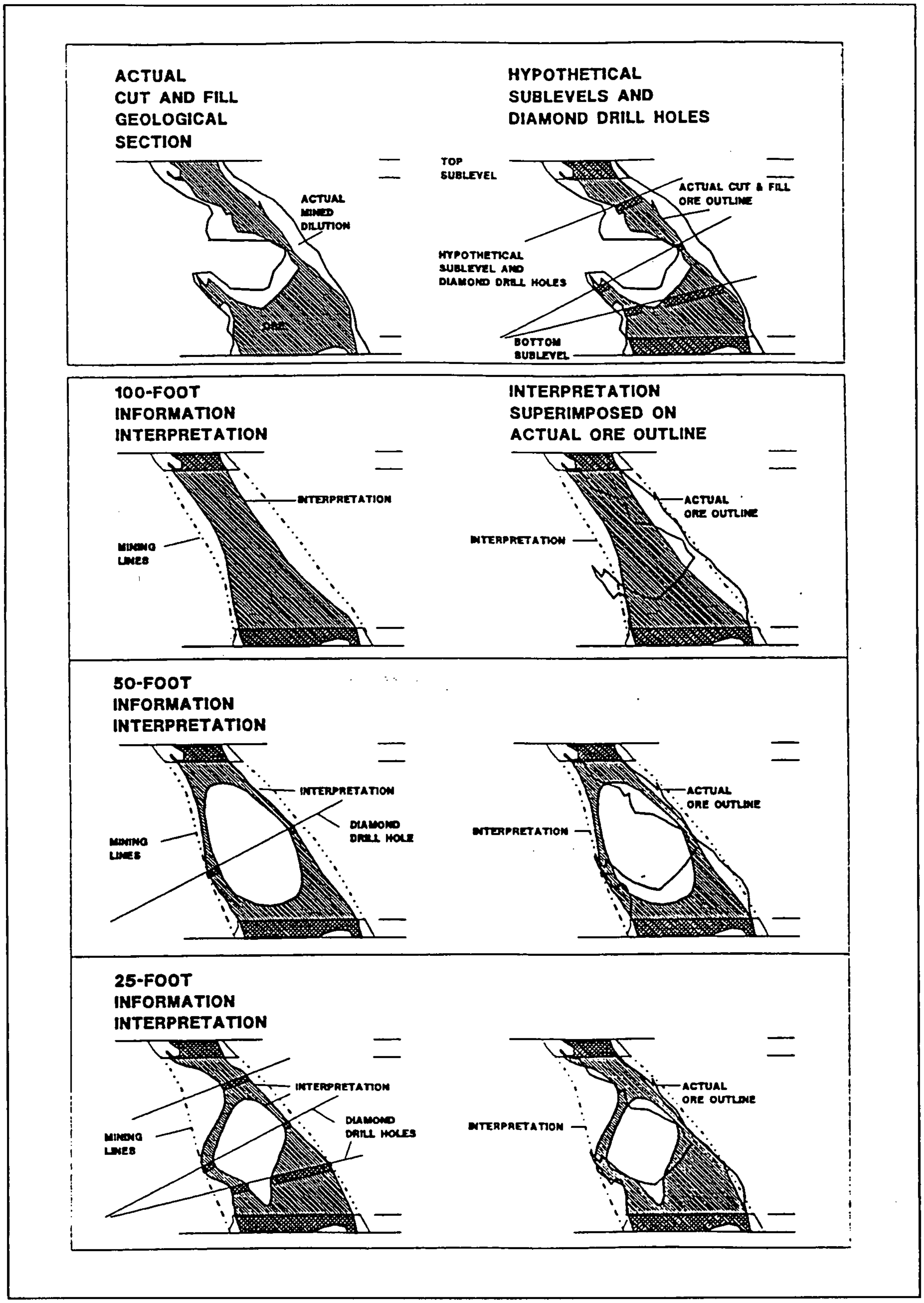


Figure 1.3 Ore delineation and information density (After Braun, 1991)

In recent years, some emphasis has been placed on developing borehole geophysical logging tools which can be used to locate ore boundaries in blastholes (Ashcroft, 1991; Lulea University, 1991; Killeen, 1991), example results are shown in Figure 1.4. If proven to be practical, these tools may provide an effective means of helping to quantify planned dilution and may also prove effective in controlling some aspects of unplanned dilution (i.e. identify perimeter blastholes which have been unexpectedly collared in waste rock or that have deviated into waste rock).

Scoble and Moss (1994) state that further potential for geophysics lies in development of ground penetrating radar borehole tools, and seismic and radiowave tomography. They also state that another supplement to exploration data could be data obtained through monitoring and interpretation of blasthole drilling performance parameters. Some research on this topic has been carried out by Schunnesson (1990).

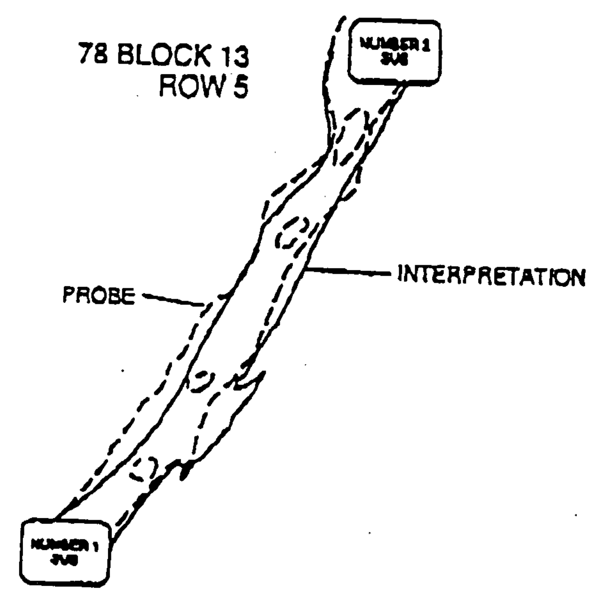


Figure 1.4 Effect of borehole probe information on ore outline (From Ashcroft, 1991)

1.8 FACTORS INFLUENCING DILUTION AND RECOVERY IN OPEN STOPE MINING

1.8.1 General

Figure 1.5 is a generalized flowsheet showing the general progression for delineating and ultimately excavating an open stope and the various factors which may impact dilution and recovery along the way. Referring to the figure it can be seen that the entire process can be divided into four main phases of activity:

1. Exploration and Ore Delineation
2. General Mine Design and Detailed Stope Design
3. Mining: Drill; Blast; Muck; and Fill
4. Quantifying Stope Performance and Improving the Design

Factors are present within each phase of the open stoping process which can impact the resulting dilution and recovery for a particular stope. Furthermore, the quality of work conducted in a particular phase directly influences the success of the following phase. Also note that the majority of factors influencing dilution and recovery are human factors. The only factors that are not within human control are: the orebody characteristics; rock structure; rock quality; and to some degree stress conditions. However, for the most part, these factors should be accounted for in the design. The following sections briefly discuss the four phases of stoping activity and the impact on dilution and recovery.

1.8.2 Exploration and Ore Delineation

A crucial aspect during preparation of a stoping block is ensuring a high quality of in-ore development (i.e. drill drifts). Keeping ore development in the ore and out of the hangingwall and footwall rocks is very important. Undercutting of stope walls during ore-drift development can be a major contributor to stope wall instability in the future. To help minimize undercutting it is very important to ensure that the ore has been delineated sufficiently in the exploration stage prior to drifting, thus, giving the mine geologists sufficient geologic control to keep the drift in the orebody. This is especially critical in

FACTORS WHICH INFLUENCE DILUTION AND RECOVERY IN OPEN STOPE MINING

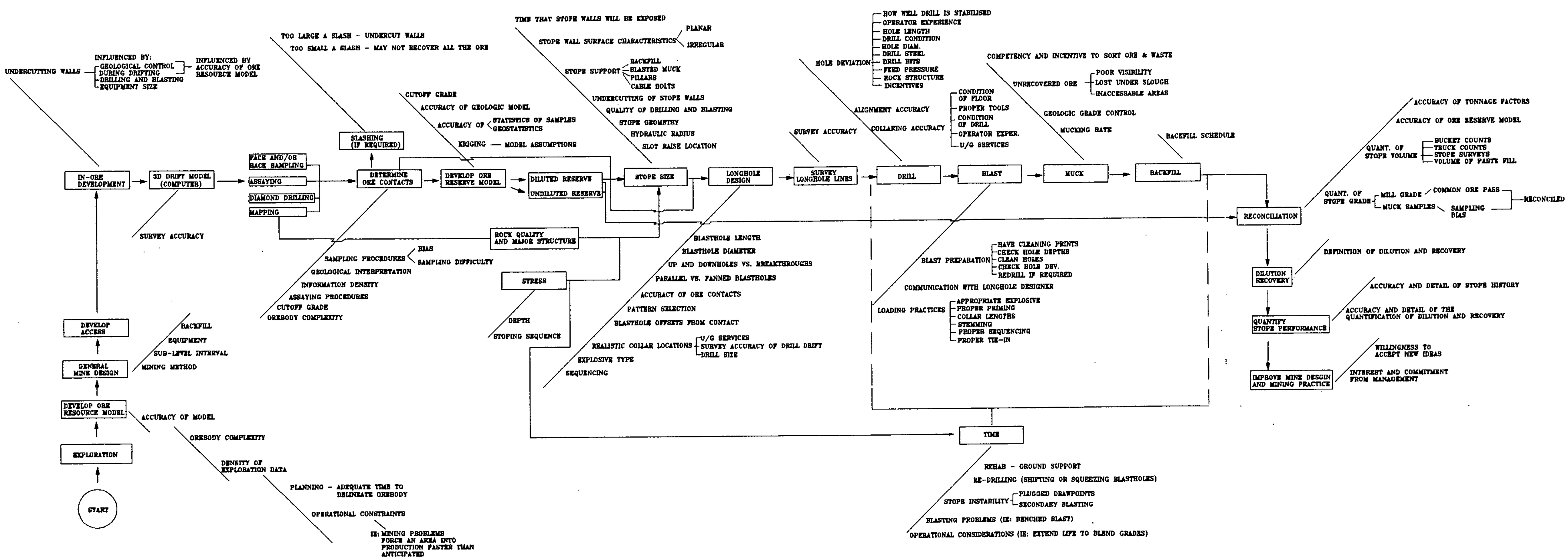


Figure 1.5 Factors influencing dilution and recovery in open stope mining

new deposits and in areas where there is a limited experience base. If later, slashing to the ore limits is required, which is generally preferred to enable more detailed ore delineation and increased control over longhole blast damage, it should be done under strict geological/engineering control (i.e. use diamond drilling to define slashing limits and issue slashing layouts).

In narrow orebodies some undercutting may be unavoidable. The amount of undercut should be minimized by using the smallest practical equipment size. Strong efforts should also be made to carry the majority of the undercut in whichever wall rock is less susceptible to instability. This generally requires some compromise in order to maintain proper clearance for the longhole drill, such that it can later drill parallel blastholes to both the hangingwall and footwall contacts.

Once the in-ore development is complete, the orebody morphology, grade distribution, and ore limits are established through detailed mapping, sampling, and possibly additional drilling. These activities all need to be of high quality to accurately characterize the orebody contacts and ultimately the ore reserve.

The impact of information density on the accuracy of ore delineation and subsequently the potential for ore losses and planned dilution has been discussed previously in Section 1.7.

1.8.3 General Mine Design and Detailed Stope Design

1.8.3.1 General Mine Design

Early in the mine design process decisions must be made regarding: equipment; sub-level interval; stoping method; and backfill. All of these factors either directly or indirectly impact dilution. For example, large highly productive stopes utilizing: large equipment; increased sub-level intervals; and large diameter blastholes, may achieve low costs per tonne but possibly at the expense of unacceptable amounts of dilution due to: poor ore delineation; unstable stope spans; and blast damage resulting from high powder factors and hole deviation. Furthermore, if reliance is placed on a small number of highly productive stopes, disruptions to ore flow (i.e. due to stability problems) can have severe economic consequences (Moss et. al., 1995). These design factors must be weighed against mining smaller but more selective stopes utilizing: smaller sub-level intervals; smaller equipment; and smaller diameter blastholes, and hence potentially less dilution but at a higher cost per tonne (but not necessarily a higher cost per unit weight of metal mined). The appropriate choice will be largely dependent on grade; orebody morphology; and quality of the wall rocks.

Incorporating high strength backfill into the mining cycle enables the use of pillarless stoping sequences and is an effective means of reducing stope wall exposures which aids wall stability and helps control unplanned dilution. It can also be a significant source of unplanned dilution if backfill exposures are too large and/or if blast damage is excessive.

Paste backfill, which is relatively new to the mining industry, has significant potential with regard to minimizing dilution. For example at the Lupin Mine (NWT), due to paste backfill's fast curing time (7 to 10 days) and good strength (0.5 MPa), mining small stopes with fast turn around times are possible. Slots raises are excavated in the paste which further decreases stope cycle times (Sandhu, 1996). This mining approach has the benefit of smaller more stable stope walls and limited impact from the de-stabilizing effects of time.

1.8.3.2 Detailed Stope Design

Design issues that are addressed during the detailed stope design phase include: stope sizing and support; slot raise locations; longhole design; and stope sequencing.

Stope Sizing

The stability of open stope surfaces is dependent on a number of factors, principally: rock quality; stress; orientation of structure relative to the opening; surface size; surface shape; surface dip; surface character (i.e. planar; undulating); amount and quality of ground support; undercutting of the surface; quality of drilling and blasting; and the amount of time that the stope surfaces are exposed. Currently no stope design method exists which incorporates all of these factors.

At present, the sizing of open stopes is largely based on experience and empirical design methods (Mathews et.al., 1981; Potvin, 1988; Pakalnis, 1986; Laubscher, 1990; Villaescusa, 1995). The most commonly used methods (Mathews et. al., 1981; and Potvin, 1988) correlate qualitative measures of stope stability (i.e. stable; unstable; caved) with: rock quality; stress; orientation of structure relative to the opening; surface dip; surface size; and surface shape. Scoble and Moss (1994) have proposed very preliminary factors, which with more research, could possibly be incorporated to account for: drillhole deviation; orebody knowledge; and blast damage.

Support of stope surfaces is primarily accomplished through the use of cablebolts, although, minimizing stope spans through the use of pillars and/or backfill can also be considered a form of stope support. Empirical cablebolt design charts have been developed by: Potvin and Milne (1992); Nickson (1992); and Hadjigeorgiou (1995). A very thorough treatise on the subject of cablebolt support can be found in Hutchinson and Diederichs (1995). Another potential method of providing stope support, if production constraints permit; is to muck only the swell during the longhole blasting process . The blasted muck left in the stope will provide additional wall support. Once a block is fully blasted, it should be mucked out as quickly as possible to minimize wall exposure times. Although conversely, Morrison (1991) states that there is no conclusive evidence that this is an effective means of controlling dilution since the dilution ultimately occurs when the final draw is made. Waste material rarely just falls on top of the ore but inevitably gets mixed with ore and is often drawn preferentially into the drawpoint at the expense of ore.

Existing empirical methods of stope design are useful for getting a general estimate of stability for a particular design. A major limitation, however, is that they are largely derived from qualitative measurements and estimate stability using qualitative terms. For example, what does "stable" mean in terms of a volume or tonnage of waste material? In terms of unplanned dilution, what is considered "stable" for a 10m wide stope might not be considered "stable" for a 1.5m wide stope. Design methods are needed which are derived from quantifiable measurements of stope and backfill wall stability and which can be used to estimate the amount of overbreak/slough for a particular stope design. Stopes can then be designed to achieve a certain acceptable level of dilution.

Regardless of the current limitations of stope design methods, the existing methods are still valuable engineering tools. For example, if at a particular operation the drilling and blasting is of marginal quality, and/or the walls are commonly undercut during development of the ore drift, stopes should not be sized to a marginal level of stability. Rather, they should be designed such that they fall well within the "stable" zones of the existing design methods. In all instances, if dilution is to be minimized, stope sizes should be determined using considerable engineering input. Furthermore, detailed records of stope performance should be kept so that the design process can be continually improved.

Slot Design

Once the stope limits are designed, the location of the slot raise must be decided upon. This is an important aspect of stope design since damage at the slot can induce stope wall instability. If slot raises are to be driven along the hangingwall or footwall, the raise should be driven against the more competent of the two. If possible, it is preferable to keep the raise away from both walls, especially if drop-raising techniques are being used, due to greater potential for blast damage. Another factor to consider is that the process of opening up the slot is often prone to significant blast damage due to greater confinement and subsequently higher powder factors. For this reason it is good practice to open up the slot perpendicular to the strike of the orebody. Opening the slot in this manner allows the hangingwall and footwall to be exposed sequentially with the subsequent longhole blasts as opposed to fully exposing one of the walls early in the blasting process. This is shown schematically in Figure 1.6 (From Villaescusa, 1995).

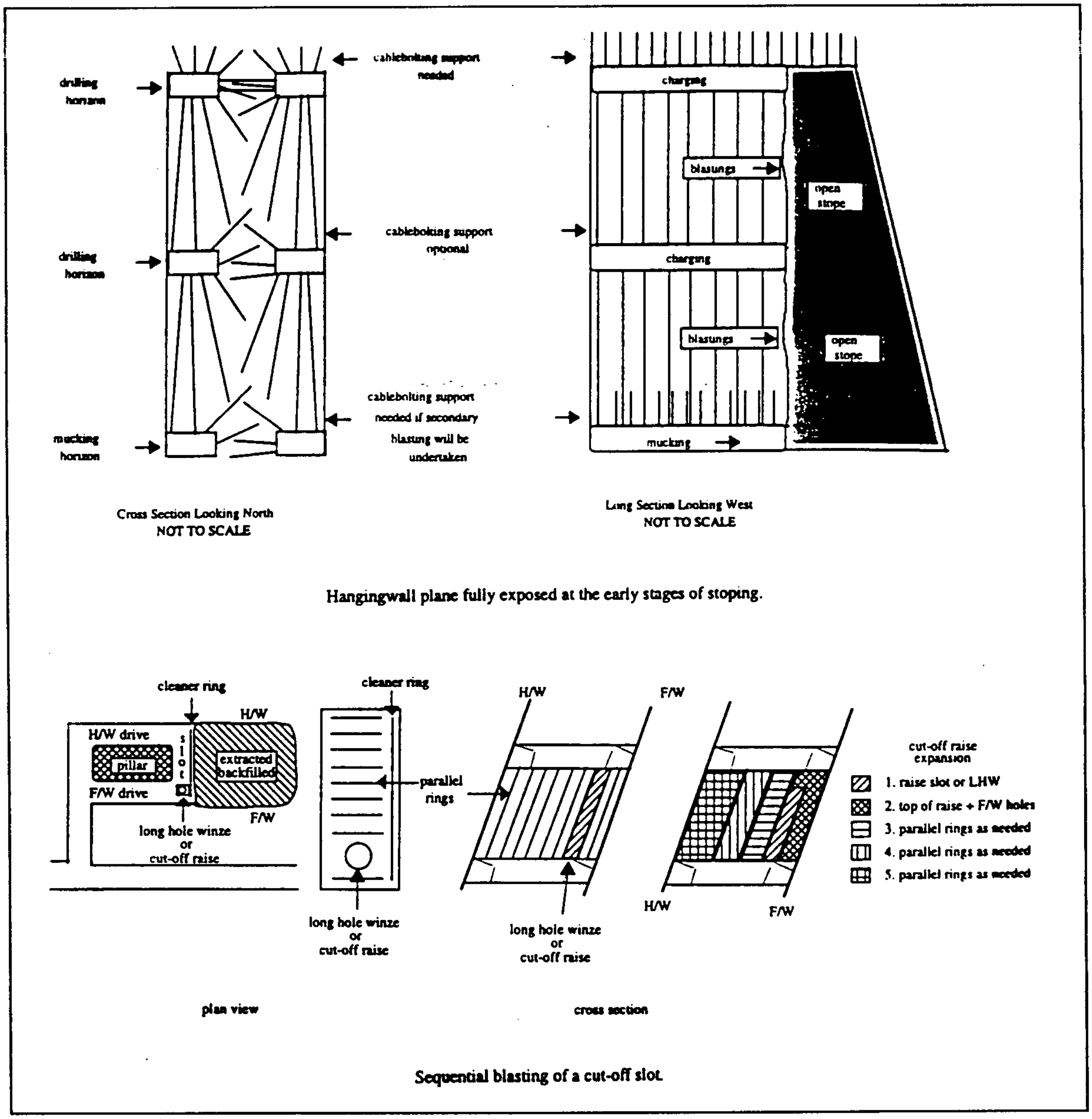


Figure 1.6 Slot design (From Villaescusa, 1995)

Longhole Blast Design

At the detailed design stage some factors pertaining to longhole blast design may have already been determined (i.e. hole diameter; hole length; parallel vs. fanned blastholes). A note pertaining more to general mine design, is that if possible, drilling horizons should be designed such that blastholes can be drilled parallel to the orebody contacts, thus making possible the application of controlled blasting techniques to minimize wall damage. As stated by Morrison (1995), blastholes which are not parallel to the desired wall will generate excessive damage and make controlled blasting techniques impossible. A further note pertains to drilling, if possible, blastholes should be designed to breakthrough, as opposed to drilling shorter length up and downholes. The benefit of being able to check breakthrough locations for drilling accuracy should not be underestimated. It is very difficult to generate the confidence to shoot good size blasts if drilling accuracy cannot be reliably determined. Furthermore, if drilling accuracy cannot be checked, efforts to minimize wall damage may prove frustrating.

With regard to actual blast design, spacings and burdens should be designed such that the distribution of explosive energy provides adequate fragmentation with minimal to no charge interaction between blastholes, thus ensuring that each blasthole is doing its assigned work. Blasting sequences should be engineered to ensure good free face geometry and efforts should be made to minimize the number of holes assigned to a particular delay period. It is good engineering practice to periodically check blast performance using blast monitoring techniques. Blast monitoring is a very useful tool for: optimizing blast patterns; evaluating explosive and detonator performance; examining blast damage potential; and evaluating methods of wall control blasting. Examples of blast monitoring applications can be found in: LeJuge et.al. (1993); Golder Associates Ltd. (1995); Forsyth et.al. (1997).

To help minimize blast damage, perimeter holes are often offset a certain distance from the final stope walls (this distance is sometimes referred to as the stand-off distance). Careful consideration should be given to the offset distance since the amount of blast damage will be related to factors such as: hole diameter; rock quality; burden; and explosive type. Improper offset distances can result in either excessive overbreak (unplanned dilution) or underbreak (lost ore). Stope surveying coupled with blast monitoring is an effective means of determining appropriate offset distances.

Other factors which can influence dilution and recovery in the longhole design process include: the accuracy of the ore contacts on the longhole ring sections; the designed blasthole lengths (i.e. must not exceed the capacity of the drill); the practicality of actually collaring a blasthole at the designed location; and the accuracy of the ring mark-up underground. Longhole ring-sections should be constructed using all available information regarding ore contacts and should be updated with information gained during the actual drilling phase (i.e. driller may intersect an unexpected pocket of waste; or an unmapped fault). Collar locations should be designed taking into account the size and maneuverability of the longhole drill (this is especially critical in narrow drifts) and the location of underground services. Ring mark-ups underground should be done under strict survey control. These factors are especially critical for narrow orebodies where tight blast patterns with relatively few blastholes per ring are used.

Stope Sequencing

Some typical stoping sequences include: primary/secondary sequences; pillarless centre-out sequences; and bottom-up or top-down sub-level retreat with or without pillars. Ultimately, the sequence decided upon will be dependent on factors such as: orebody width; mining rate; availability of backfill; whether or not the ore can carry the cost of haulage drifts and drawpoints and if so at what vertical interval; mining depth; and rockburst potential.

The stoping sequence is critical in that it significantly influences stress conditions. Stress related ground control problems can seriously delay production (i.e. rehabilitation or loss of accesses and ore drifts; redrilling of blastholes due to squeezing and shifting ground; uncontrolled falls of ground; rockbursts) and subsequently result in reduced recoveries and possibly increased levels of dilution due to the increased time that a stope may have to remain open.

Maintaining adequate control of the ground during the stoping process, through proper ground support design and application of rock mechanics principals is a key factor in maintaining required levels of production. Numerical modelling can be a very valuable tool for helping to determine stoping sequences which minimize mining induced stress levels. Furthermore, when coupled with underground observations and previous stoping histories numerical modelling can be integrated into long range planning to help forecast future ground support requirements and identify areas where production delays due to adverse ground conditions may occur. An example of how rock mechanics can be integrated into planning can be found in Connors et.al. (1997).

1.8.4 Mining: Drill; Blast; Muck; and Fill

Longhole Drilling

To achieve effective control over dilution, recovery, and fragmentation, accurate drilling is essential. Consequences of inaccurate longhole drilling include:

- dilution and oversize material from wall rocks if blastholes deviate into or too close to the stope walls;
- dilution and oversize from overbreak if hole deviation results in overburdened wall holes. Excessive burdens cause increased vibration levels and increased potential for blast gasses to penetrate and damage wall rocks;
- ore oversize resulting from poor distribution of explosive energy;
- lost ore from rock that was designed to be drilled but wasn't due to hole deviation;
- reduced drilling productivity resulting from having to redrill holes.

Figure 1.7 is a schematic illustrating many of the above points (after Almgren et.al., 1981).

Three sources of drilling inaccuracy exist (Hendricks et.al., 1994): set up collar location; set up alignment; and trajectory deviation. The resulting hole deviation represents the sum total of all the sources of error and is governed by factors such as: rockmass characteristics; operator expertise and motivation; mechanical condition of the drill, drill rods and bits; hole length and diameter; type and stiffness of drill rods; drill performance parameters (i.e. thrust, rpm, etc..); availability and adequacy of set up and alignment tools (i.e. angle indicators; lasers); physical condition of the drill level (i.e. floor is clean; back height is such that the "stinger" can reach the back and stabilize the drill; services are out of the way); and accuracy of the ring mark-up. Note that the only factor which is not within human control is the rockmass characteristics. For more detailed discussions on drillhole deviation refer to: Hendricks et.al. (1994); Hamrin (1995); Almgren (1981); Klein et.al. (1992); Paley (1993); and Sinkala (1985).

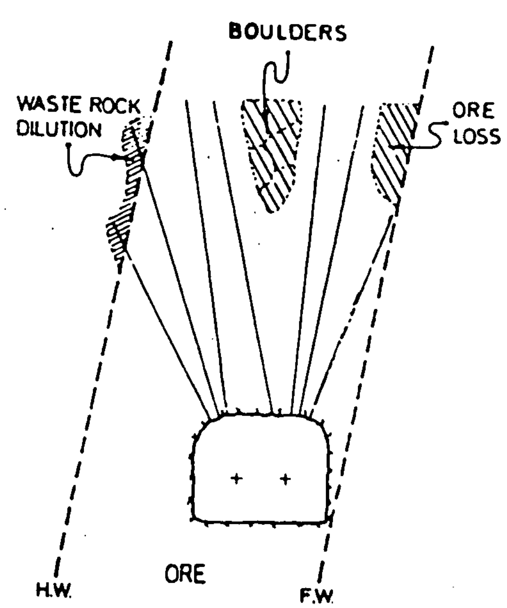


Figure 1.7 Consequences of inaccurate longhole drilling (After Almgren et.al., 1981)

To take advantage of information that can be gained during the drilling process there should be well established lines of communication between the drillers and the longhole designer. For example, drillers should be encouraged to write comments on the drill prints (i.e. no breakthrough; hit waste at a depth of X; intersected a fault at a depth of X). The drill prints should be returned to the longhole designer as drilling progresses. If warranted adjustments to the design can be made. The longhole designer should diligently update his prints with the additional information so that the information is passed on to the longhole blasters.

Drilling accuracy should routinely be checked by surveying the collars and breakthrough locations of blastholes. Blastholes with unacceptable deviation should be redrilled. To improve the likelihood of getting holes redrilled, the pick-up of collars and breakthroughs should preferably be done before the drill moves out of an area, and at the very least before the longhole blasters are sent in to the area to load.

<u>Longhole Blasting Practices</u>

The responsibilities of the longhole blasters general include: preparing the area to be blasted; and loading and sequencing the blastholes as per the design.

The blast preparation stage represents the last chance to identify and correct potential problems with the longhole design and drilling. Blast preparation involves: locating the collars of all the blastholes; cleaning the blastholes; measuring the length of the holes and noting whether they are as designed; if it has not already been done, breakthrough toe locations should be checked for unacceptable deviation. Conditions such as: presence of water; and squeezing and shifting of blastholes should also be noted. It is essential that the blasters have up to date copies of the longhole layout and ring sections during the blast preparation process. There should be well established lines of communication between the blasters and longhole designer, so that if necessary, holes can be redrilled or other changes be made to the design.

Quality workmanship during the loading and sequencing of blastholes is essential to the success of the blast. This involves: ensuring the right explosive products are being used for the existing conditions (i.e. do not load Anfo in wet holes); ensuring the loading equipment is in good mechanical condition (i.e. Anfo loader; cartridge loader; loading hose); ensuring the explosives are of good quality; following proper priming practices; sequencing the blast as per the design; and ensuring the blast is tied in well (i.e. more than one path of initiation).

Poor blasting practices can result in: blast damage; poor fragmentation; lost ore from inadequate breakage; and costly production delays if areas have to be redrilled and reblasted (i.e. "benched" blasts). Fixing "benched" blasts often requires creative blasting techniques which often cause additional blast damage. Furthermore, time delays caused by blasting problems will increase the time a stope has to remain open, increasing the probability of wall instability.

Mucking

Once a stope has entered production, the stope should be blasted and mucked out as fast as is practically possible. This will minimize the time that stope walls are exposed and will reduce the probability of instability. If production constraints permit, consideration should be given to only mucking the swell during the longhole blasting process thus allowing the blasted ore to provide some support to the stope walls. Although as mentioned previously, some practitioners feel there is not conclusive evidence that this practice is an effective means of minimizing dilution.

During the mucking process, some factors which can influence dilution and recovery include: the ability of the scoop operator to sort ore and waste; whether or not the geologists aid with grade control at mucking levels; the rate at which the stope is mucked;

wall slough may fall on blasted ore making it unrecoverable; and depending on the size and shape of the stope, visibility may be poor when remote mucking, resulting in lost ore.

Backfilling

If backfill is part of the mining cycle, once a stope is mucked out it should be backfilled as quick as possible. At this stage of the mining cycle, if the stope walls collapse before backfill is placed it will not directly dilute any ore, however, it may jeopardize the stability of overlying stopes and/or adjacent stopes along strike, refer to Figure 1.8.
Scoble and Moss (1994) state that the stoping schedule should be based upon the backfilling capability to avoid excessive exposure times for cavities.

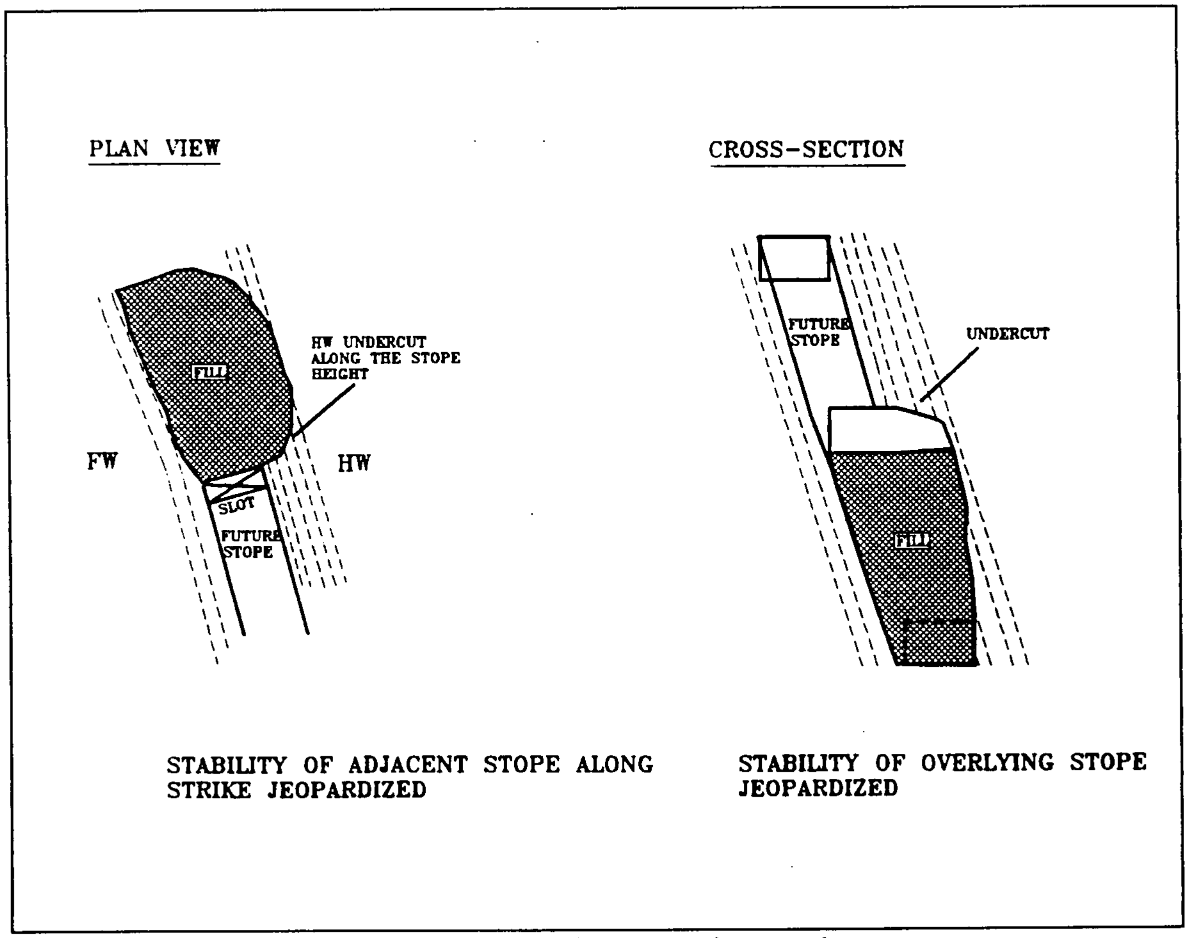


Figure 1.8 Effect of wall collapse on adjacent stoping panels

1.8.5 <u>Quantifying Stope Performance and Improving the Design</u>

Dilution and recovery are essential measures of stope performance although many other factors should also be considered. For example: amount of redrilling; fragmentation; effectiveness of stope support; production delays; sources of dilution; causes of lost ore; time required to fully blast and muck the stope; number of blasts required to fully excavate stope; and availability of backfill. To fully understand the mechanisms which impact dilution and recovery , considerable time should be spent underground observing and documenting the various activities during the mining process. This is the only way to identify which of the many possible factors are having significant impact. The key to bringing all this information together is the accurate quantification of dilution and recovery. As was mentioned in Section 1.6, stope surveying with the use of non-contact laser rangefinders represents a major breakthrough with regard to understanding the processes influencing unplanned dilution and recovery. Through accurate quantification of stope performance on a stope by stope basis, methods of improving different aspects of mine design and mining practice will become apparent. Stope surveying provides a method for quantifying the cost/benefit of possible improvements to the open stoping process and for validating the effectiveness of any recommended changes.

Ultimately, the success of efforts to minimize dilution and maximize recovery will be dependent on management's interest and commitment to the cause, their willingness to try new ideas, and the realization that sometimes you have to "spend money to save money".

CHAPTER 2

ESTIMATING DILUTION WITH EXISTING EMPIRICAL DESIGN METHODS

2.1 GENERAL

This chapter reviews existing empirical stope design methods and evaluates the different methods with regard to potential for estimating dilution. With the exception of Section 2.2 (O'Hara, 1980), all of the stope design methods reviewed relate surface stability to rockmass conditions and the hydraulic radius of the surface. Since hydraulic radius is such a commonly used parameter in stope design a brief description is warranted.

Description of Hydraulic Radius Parameter

Hydraulic radius (area/perimeter) is a parameter borrowed from fluid dynamics where it was used to relate fluid flow in square pipes to that in circular pipes. In rock mechanics, it is sometimes referred to as the shape factor and is used to allow for the comparison of various excavation shapes, since it accounts for the distance to supporting abutments from the center of an opening (Milne, 1997). This is important in stope design since the stope ends can provide significant support, in other words, two way spanning of open stope surfaces needs to be considered. For example, with regard to tunnel stability generally just one way spanning is considered (tunnel width) because the ends of the tunnel are far enough away that they do not provide support to most of the length of the tunnel. However, when the length of a rectangular opening is less than three times the width (which is often the case with open stope surfaces) the support provided by the opening ends is significant (Milne, 1997).

2.2 ESTIMATION OF WASTE ROCK DILUTION FROM STOPE WALLS O'Hara (1980)

O'Hara (1980) states that dilution from waste rock off of the stope walls varies with: mining method; the width of the stope; the angle at which the stope is dipping; and the competence of the stope wall rock. Based on data from Canadian and foreign mining projects, the following relationship is presented:

$$\%D = K / (W^{0.5} * \text{Sin}(A)) \qquad \text{(Eq. 2.1)}$$

Where:

%D = Percent Dilution
K = Factor which varies with stoping method (blasthole = 100; shrinkage = 60; cut and fill = 45; room and pillar = 70)
W = Stope width in feet.
A = Dip of Orebody

This relationship is shown graphically in Figure 2.1.

When the stope walls are regular and competent, dilution may be only 0.7 of the above. If the stope walls are incompetent dilution may be as high as 1.5 times the above.

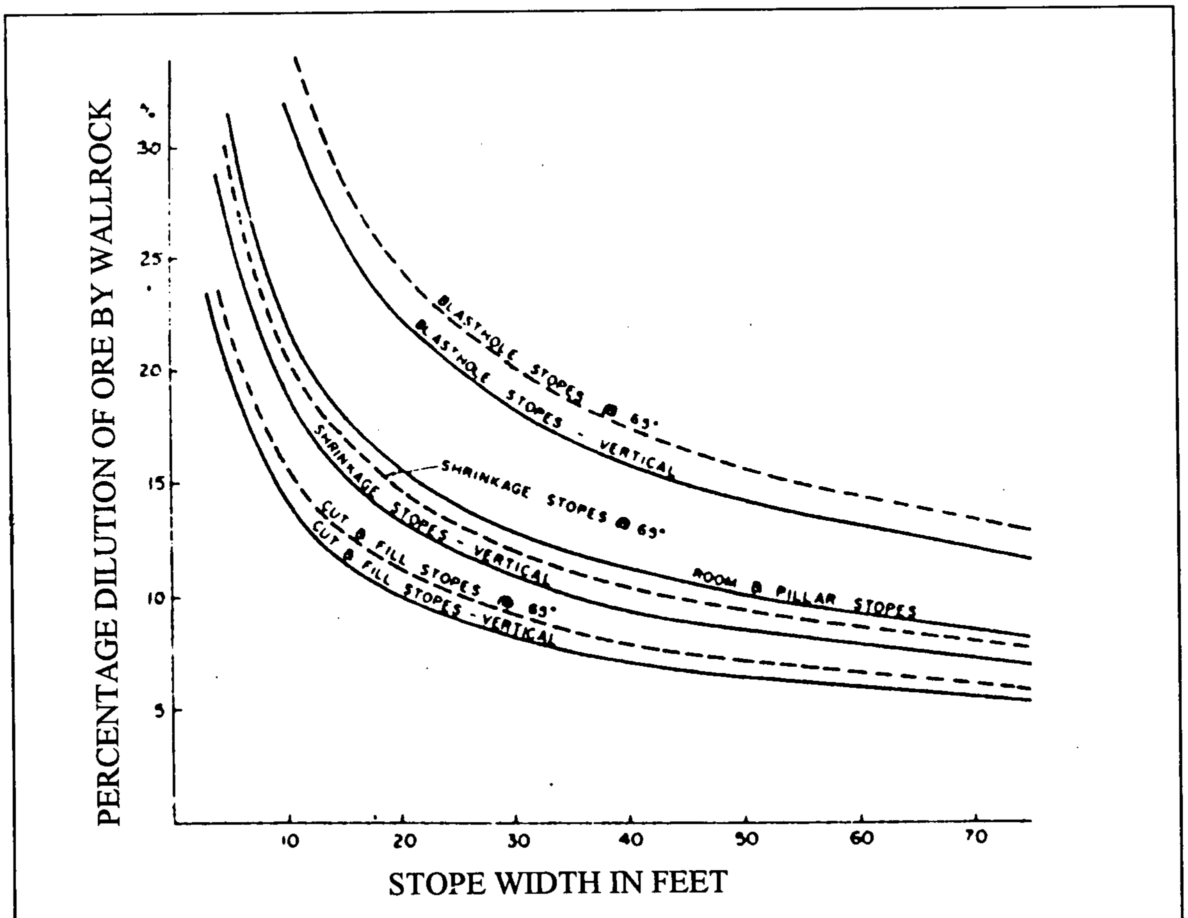


Figure 2.1 Estimation of waste rock dilution from stope walls (From O'Hara, 1980)

Limitations

With regard to estimating dilution from open stopes, the following are considered to be limitations of the method:

- it is not clear how the relationship was developed;
- the method reflects past practice;
- subjective terms are used to describe rock quality (ie:"competent" and "incompetent");
- no account is made for: stope size; stope shape; stress; structure; wall support; drilling; blasting; etc.;
- the method cannot be used for dimensioning open stopes.

2.3 THE DILUTION APPROACH - Pakalnis (1986,1993)

The Dilution Approach, used for sizing open stopes, was developed from a 5 year joint study between the Ruttan Mine of Hudson Bay Mining & Smelting Inc., CANMET, and the Department of Mines of Manitoba (Pakalnis, 1993). The objective of the study was to develop ground stability guidelines for the mining of large open stopes.

A database was compiled which included 43 stopes at various stages of excavation, thereby yielding 133 observations of:

- Rock Mass Rating (RMR - Bieniawski, 1976);
- stope dimensions;
- dilution - based on: observation; assays; tonnes of ore blasted; and mucked tonnage;
- rate of excavation;
- stope configuration (isolated, rib, echelon);
- mining sequence/method.

The observations were supplemented with: in-situ stress measurements; structural mapping; stress and deformation monitoring; and historical observations.

Through both statistics and observation it was determined that for a particular stope the resultant dilution (tonnes of external slough / tonnes of ore blasted) was largely a function of:

- the Rock Mass Rating (RMR - Bieniawski, 1976) of the hangingwall
- the hydraulic radius of the hangingwall
- the rate of extraction
- the stope configuration (isolated, rib, or echelon)

Figure 2.2 shows the derived relationships and design charts.

Limitations

The following are considered limitations of the method:

- the dilution values collected during the study likely contain a significant margin of error due to inherent inaccuracies in the methods used to quantify dilution;

- the database is limited to one mine and is thus biased by the operational parameters (i.e.: development practice, drilling, blasting, etc..) and orebody characteristics (width, dip, contact characteristics, etc..) particular to that mine;

- since dilution values are related to stope width, the method is only applicable for stopes with similar widths to those in the database (8m - 15m, Pakalnis, 1993);

- does not distinguish between the different stope surfaces, assumes all the dilution comes from the hangingwall;

- does not address factors such as: stope support; undercutting of stope walls; drilling; and blasting.

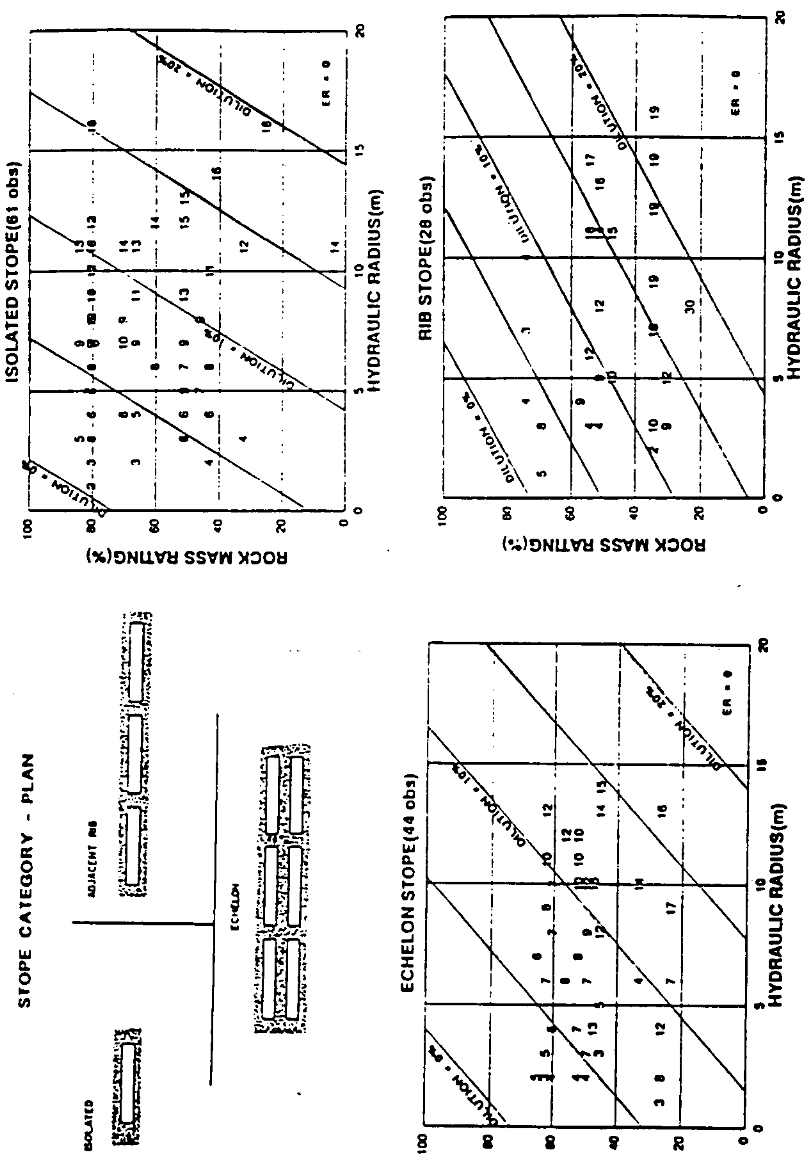


Figure 2.2 The Dilution Approach (From Pakalnis et.al., 1993)

2.4 STABILITY GRAPH METHOD

The Stability Graph Method for open stope design was initially proposed by Mathews et. al. (1981). It has since become known as Mathews Method for Open Stope Design. Stewart & Forsyth (1993) updated the method with additional case histories and redefined the original zones of stability.

Potvin (1988) modified Mathews Method and calibrated it using 242 case histories (176 unsupported, 66 supported). Potvin's modified method has since become known as the Modified Stability Graph Method. The influence of cable bolt support was re-examined by Potvin & Milne (1992) and by Nickson (1992) who added an additional 59 case histories to the database (14 unsupported, 45 supported) and based on statistics introduced two new cable bolt design zones. Hadjigeorgiou (1995) further augmented the database with 90 case studies (38 unsupported, 52 supported) and statistically redefined the zone where cable bolting can be successfully used for back support.

Figures 2.3 and 2.4 show the evolution of Mathews Method and the Modified Stability Graph Method, respectively.

Referring to Figures 2.3 and 2.4, it can be seen that both methods plot a stability number N (Mathews et al., 1981) or N' (Potvin, 1988) versus the hydraulic radius of the surface being analyzed. The factors N and N' are similar except that the weighting factors used in their calculation differ. Refer to the equations presented below:

$$N \text{ or } N' = Q' \times A \times B \times C \qquad \text{(Eq. 2.2)}$$

$$HR = \text{Wall Surface Area} \,/\, \text{Wall Perimeter} \qquad \text{(Eq. 2.3)}$$

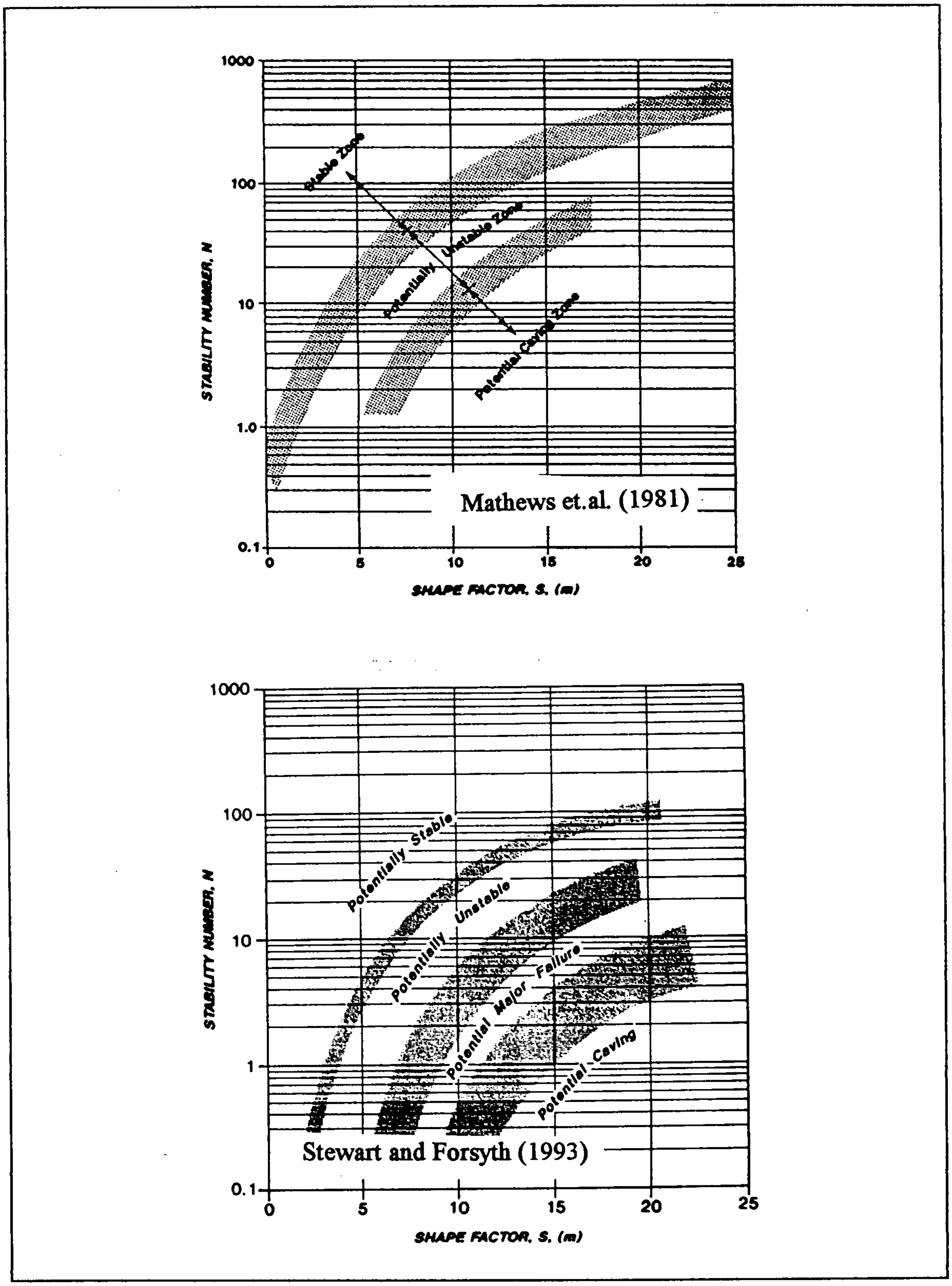


Figure 2.3 Evolution of Mathews Method of open stope design

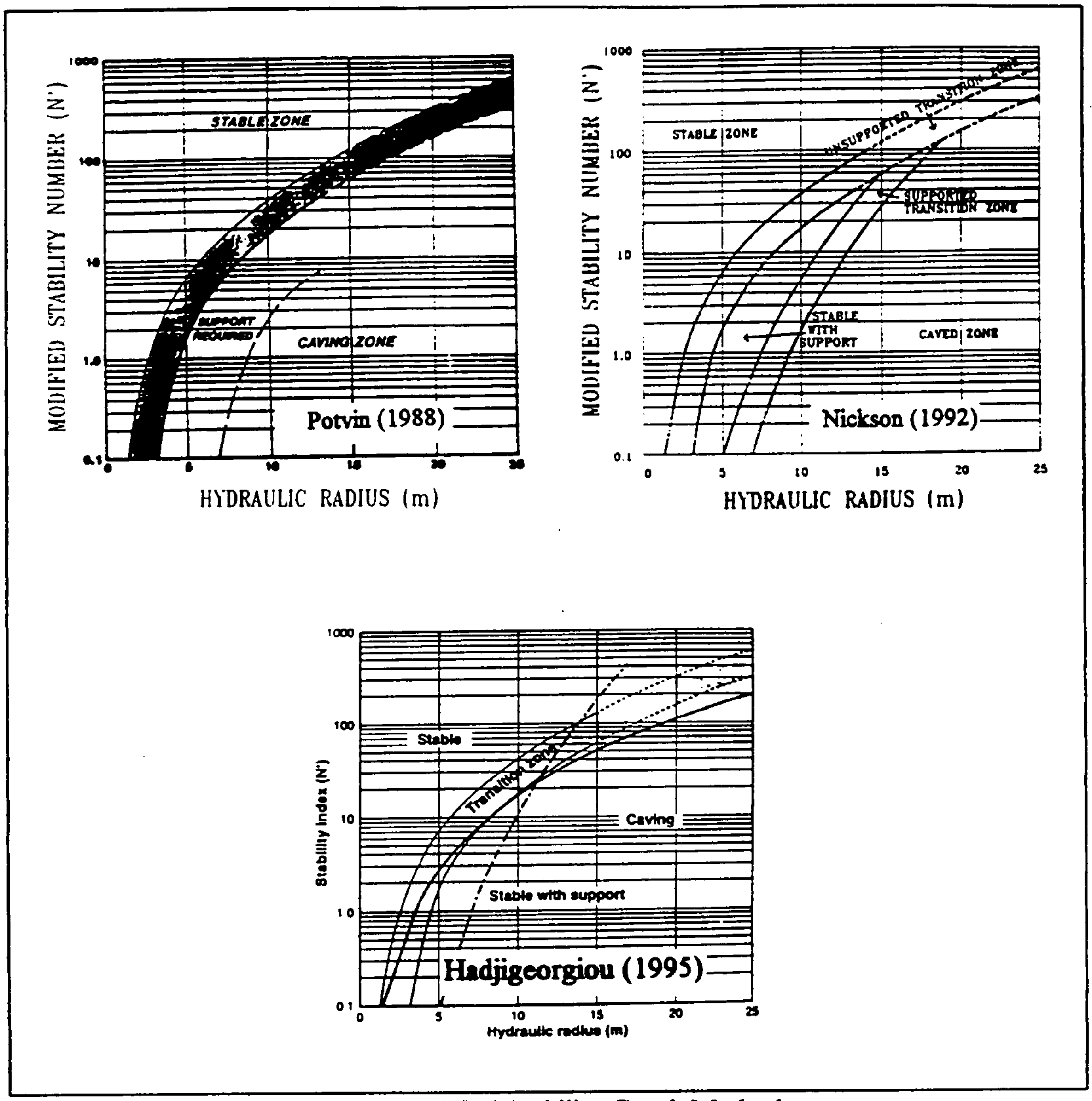


Figure 2.4 Evolution of the Modified Stability Graph Method

Where:

N or N' = Stability Number

Q' = Modified Tunneling Quality Index (NGI) with Jw/SRF set to one. (after Barton, 1974)

A = Stress Factor

B = Joint Orientation Factor

C = Gravity Factor

HR = Hydraulic Radius

As far as determining which method to use, Hutchinson and Diederichs (1995) state that Canadian mines use Potvin's (N') and mines in Australia use Mathews (N). In reality, the modifications introduced by Potvin (1988) do not greatly effect the results. Both methods have proven to be reliable tools for dimensioning open stopes.

A drawback of both methods is the use of qualitative terms such as *stable* and *unstable* to describe the various design zones. These terms have little quantifiable meaning with regard to dilution. However, the respective authors do provide reasonable definitions of the design zones and it is accepted that as a general rule the amount of unplanned dilution will increase as designs plot closer to the caving zone.

With regard to Mathews Method, Stewart and Forsyth (1993) provide definitions for the various design zones. Abbreviated versions of the definitions are provided below:

> *Potentially Stable* - Surface is essentially self supporting. Dilution should be minimal, estimated at less than 10%.
>
> *Potentially Unstable* - If pattern support cannot be installed, some failure with associated dilution should be anticipated. A stable unsupported configuration should eventually be reached. Dilution is estimated to be between 10% to 30%.
>
> *Potential Major Collapse* - If pattern support cannot be installed, a stable configuration may be reached only after relatively large and probably unacceptable failure with associated excessive dilution and/or ore loss. Dilution is estimated to be greater than 30%.
>
> *Potential Caving* - Surface under consideration is probably unsupportable and will fail and continue to fail until the void is completely filled or surface breakthrough occurs (i.e.: a true caving situation).

With regard to the Modified Stability Graph Method, definitions for the various design zones have been assembled based on Potvin (1988) and Nickson (1992):

> *Stable* - Low dilution.

Unsupported Transition Zone- Many case histories plotting in this zone experienced dilution and groundfalls causing operational problems. Stope surfaces plotting in this zone are sensitive to blast vibrations and the effect of time.

Stable With Support - Stope surfaces plotting below the Transition Zone require some form of artificial support. Reasonable confidence in maintaining stability can be achieved by installing pattern support. If the stope surface is not supported severe ground control problems can be expected.

Supported Transition Zone - Reduced confidence in maintaining stability through installation of pattern support. If stope surface is not supported severe ground control problems can be expected.

Caved Zone - Stope surface is likely unsupportable. Severe ground control problems can be expected.

Limitations

With regard to using the Stability Graph methods as a tool for predicting dilution the following are considered limitations:

- the qualitative descriptions used to describe stope stability only provide very rough estimates of dilution;

- Stope width is not taken into account. For example, "stable" for a 10m wide stope may be unacceptable from a dilution perspective for a 2m wide stope, since small amounts of slough can significantly dilute the ore;

- does not address factors such as: undercutting of stope walls; drilling; and blasting;

It should be noted that Scoble and Moss (1994) suggest additional factors (i.e.: D and E factors) be introduced into the calculation of the Stability Number to account for factors such as: orebody knowledge; drillhole deviation; and blast damage. They provide very tentative ratings for these factors and present a stability graph which can be used to derive a qualitative indication of the amount of unplanned dilution associated with a particular design, refer to Figure 2.5. As with the other stability graphs the qualitative descriptions have little quantifiable meaning with regard to dilution.

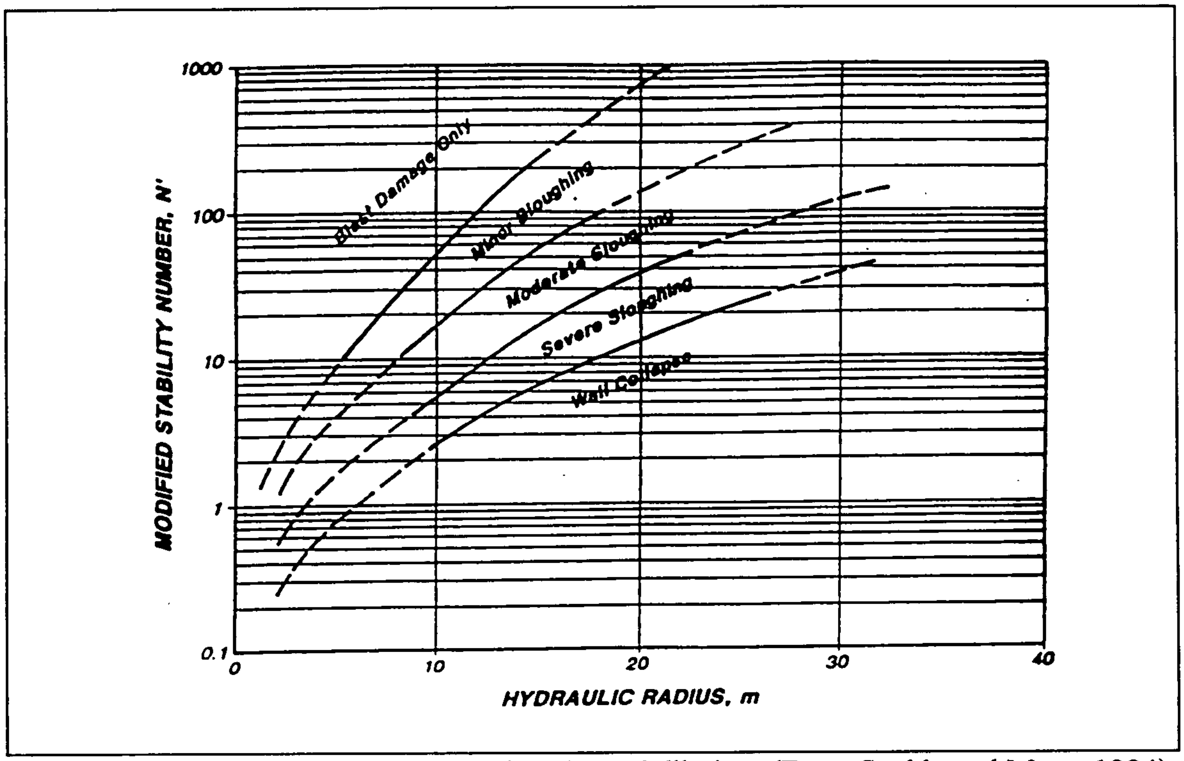


Figure 2.5 Empirical Estimation of unplanned dilution (From Scoble and Moss, 1994)

2.5 OTHER STABILITY GRAPH METHODS Villaescusa (1995), Laubscher (1990)

There exists two other stability graph methods for open stope design but neither have found wide spread application. Both methods follow the same premise of relating qualitative measures of stability to rockmass conditions and hydraulic radius. Only brief descriptions of the methods follow.

Laubscher Design Method - Laubscher (1990)

Laubscher and Taylor (1976) developed a rockmass classification system based partially on Bieniawski's RMR system. In this system, now referred to as the Mining Rockmass Rating System (MRMR), an *in situ* MRMR is determined based on: RQD; joint spacing; intact UCS of the rock; and joint condition. The *in situ* MRMR is then further adjusted using factors which account for: weathering; stress; joint orientation; the effects of blasting; and the presence of shear zones. The adjusted rating has been incorporated into design methods for predicting: support requirements for development openings; the required undercut area to induce caving; the angle of cave and extent of the surrounding failure zone; open pit slope angles; and applicability of open stope mining. Much of the data used in the development of these methods has come from operations using caving methods. This is apparent in the design graph presented in Figure 2.6 (Laubscher, 1990) which plots hydraulic radius values as high as 60. The bias towards caving methods is likely why the method has found little application in open stoping environments.

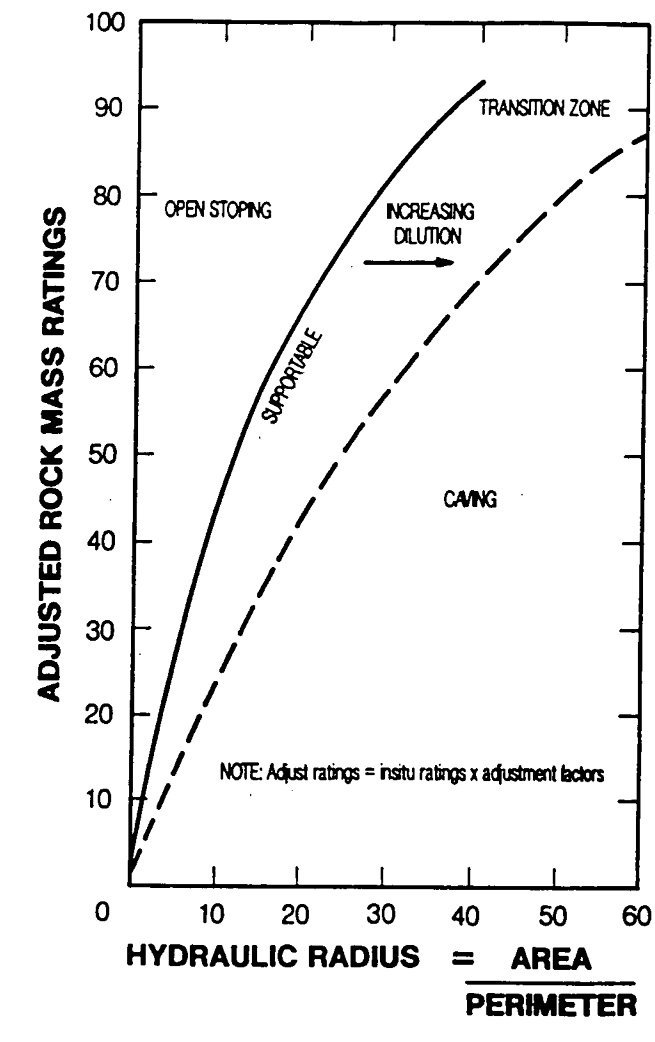


Figure 2.6 Laubscher design graph (After Laubscher, 1990)

HSR Bench/Stope Stability Method - (Villaescusa, 1995)

This method is a site specific method developed for Mount Isa Mines. The method incorporates a hangingwall stability rating (HSR) for characterizing hangingwall ground conditions. The HSR is based on three factors which have historically controlled hangingwall stability at Mount Isa Mines:

- *Geological Discontinuities (weighting 70%)* - accounts for: bedding plane breaks per meter (weighting 50%); number and continuity of joint sets (weighting 20%);
- *Mining Induced Stresses (weighting 20%)* - accounts for magnitude of stress normal to the orebody;
- *Blast Damage (10%)* - accounts for hangingwall blasthole orientation relative to bedding orientation (i.e. fanned vs. parallel blastholes).

The HSR bench/stope stability chart is presented in Figure 2.7. Brief definitions of the design zones are presented below (after Villaescusa, 1995):

> *Stable Zone* - unsupported bench hangingwalls remain stable with little or no dilution;
>
> *Dilution Onset Zone* - the outer layers of the hangingwall start to deteriorate and minor failures up to 1m deep are observed. The failures do not propagate along the strike length of the bench;
>
> *Failure Zone* - substantial hangingwall failures up to 3m deep are experienced along the entire hangingwall plane. Failures are usually arrested updip by provision of appropriate cable support at each of the bench-sill intervals.

Although this design method has proven successful at Mount Isa Mines, it has not found use at other open stoping operations due to the site specific nature of the HSR rating system.

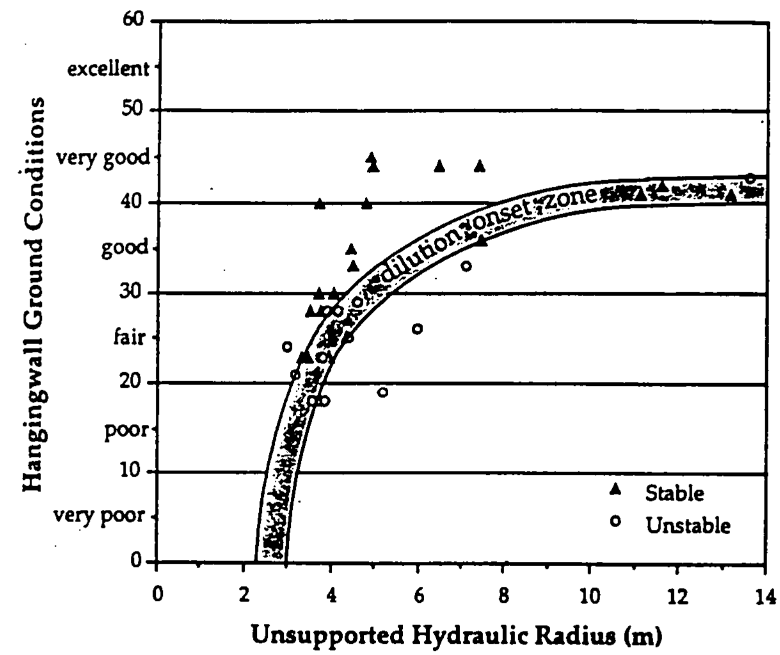


Figure 2.7 HSR bench/stope stability chart (From Villaescusa, 1995)

2.6 SUMMARY

Based on the empirical design techniques reviewed, there is currently no general design method which can be used to size stopes based on some acceptable level of dilution. The existing methods either, provide only qualitative measures of stability, or, are too site specific to be of general use. The stability graph methods (Mathews et.al, 1981, Potvin, 1988) have however, proven to be useful tools for dimensioning open stopes and have been well accepted by industry. For this reason, quantification of the design zones associated with either of the stability graph methods in terms of unplanned dilution, is considered a good starting point for the development of an empirical method for estimating unplanned dilution. For this study, the Modified Stability Graph Method (Potvin, 1988) will be used as a starting point, mainly because of the larger database of information associated with it.

CHAPTER 3

QUANTIFYING DILUTION WITH THE CAVITY MONITORING SYSTEM (CMS)

3.1 GENERAL

The first step towards the development of an empirical method to predict unplanned dilution was to build a database of quantifiable measurements of overbreak/slough from open stope surfaces. This chapter describes the actual measurement of overbreak/slough. A description of the resulting database is given in Chapter 4.

As was discussed in Chapter 1, actual surveying of open stopes has been made possible through the application of non-contact laser rangefinders. For this study, stope surveys were conducted using the "Cavity Monitoring System (CMS)" developed jointly by the Noranda Technology Centre (NTC) and Optech Systems. Examples of applications of this system can be found in papers by: Potvin (1991); Miller et al. (1992); Pakalnis et al. (1995); and Anderson and Grebenc (1995).

3.2 CAVITY MONITORING SYSTEM (CMS)

3.2.1 General Set-Up

The CMS is comprised of three main components, which are listed below:

- laser rangefinder (scanning head);
- portable controller and controller case;
- support package.

A typical set-up is shown in Figure 3.1.

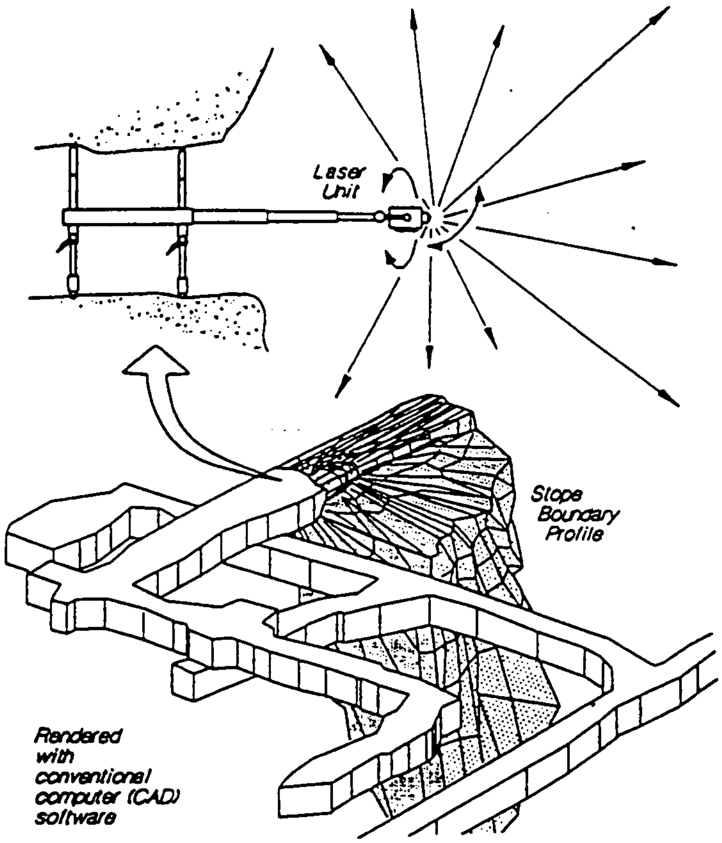


Figure 3.1 Typical Cavity Monitoring System (CMS) set-up
(From Hutchinson and Diederichs, 1995)

The laser rangefinder utilizes reflectorless laser technology and is capable of measuring distances of up to 100 m. The rangefinder is housed in a motorized fork assembly that is capable of rotating 360° about the boom axis and up to 135° about the pivot axis. This defines a sphere of 270° which is generally adequate to define a stopes dimensions, refer to Figure 3.2.

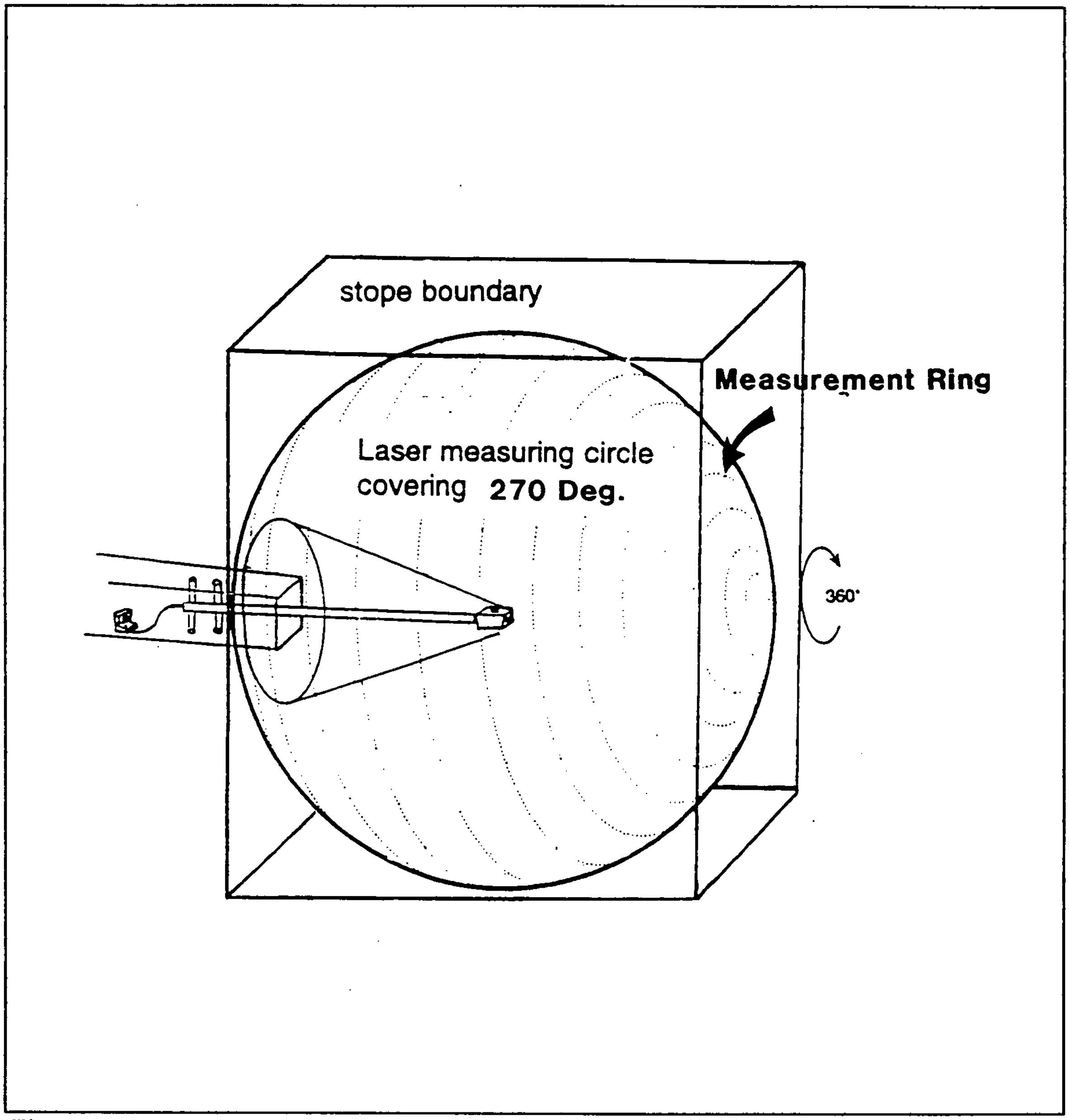


Figure 3.2 CMS measurement details (After Miller, 1991)

The portable controller enables the operator to program the survey and remotely activate the laser rangefinder. The controller case houses the data logger, CPU, and battery. The data logger has the capacity to store 2 Megabytes of information, which is four complete surveys.

The support package consists of a 10 m segmented boom and two adjustable posts (2 m to 5 m) that brace the system to the sill and back. The boom and posts are constructed of light weight composite fibers and high density polyethylene for strength and light weight.

3.2.2 Data Collection

A typical CMS survey is conducted as follows:

1. construct the support system (posts and boom) - the cable which connects the scanning head to the controller case should be running through the boom;
2. attach cable to scanning head and then attach scanning head to boom;
3. attach survey targets to boom (the scanning head location and boom azimuth are required to reduce the survey data into mine coordinates);
4. extend scanning head into cavity;
5. using the hand-held controller, program and initiate the survey;
6. survey in the two survey targets located on the boom;
7. following completion of the survey perform data verification to identify if any errors occurred;
8. if satisfied with the survey, retrieve scanning head and pack-up support system;
9. once on surface download survey information to a personal computer for processing.

Total set-up and survey time (not including transportation time) is approximately 1.0 to 1.5 hours depending on the detail of the survey (which is controlled by the operator).

3.2.3 Data Reduction

Software is provided to reduce the CMS survey results into either: XYZ coordinates; ASCII, or DXF format (triangulated 3D-mesh, or, polylines). For this study, surveys were reduced to a DXF file format (triangulated 3D-mesh) and imported into AutoCAD for further processing.

Following conversion into the appropriate format it is very important to overlay the results onto the existing 3D mine model. This allows verification of the mesh location. Significant errors in mesh location can occur depending on the method used to determine the scanning head location and boom azimuth. If errors are identified, a software package such as AutoCAD can be used to correct the mesh location. In this study, it was found to be very useful to set up the CMS such that a portion of the stope access was surveyed along with the stope. This aided in verifying and correcting the mesh location (if required).

Following verification of the mesh location, cross sections can be generated using software supplied with the system or with other commercially available packages.

The following lists the general data reduction procedures used during this study:

1. Conduct CMS survey.
2. Download the survey results to a PC.
3. Reduce survey data into DXF file format (triangulated 3D-mesh).
4. Overlay survey on existing 3D mine model to verify location. Correct if necessary.
5. Cut cross-sections along the longhole rings. The sections should contain the following information: ore drift(s); ore contacts or mining lines; CMS survey.
6. Plot blastholes on sections

7. On each section, calculate the area (m^2) of overbreak/slough and underbreak for both the hangingwall and footwall. Overbreak/slough and underbreak is measured relative to the ore contacts or mining lines. Dilution from ore drift development is not included when determining the area of overbreak/slough.

8. Based on the areas calculated on each section, calculate a volume of overbreak/slough and a volume of underbreak for both the hangingwall and footwall. This will require that a thickness be assigned to each section.

9. Convert the volumetric measurements into parameters termed ELOS (Equivalent Linear Overbreak/Slough) and ELLO (Equivalent Linear Lost Ore) which are both expressed in meters (m).

Descriptions of the ELOS and ELLO parameters are given in Section 3.3.

3.2.4 Limitations of the CMS System

The main factor to remember when setting up a CMS survey, is that the scanning head can only measure distances to what it can "see", thus, portions of the stope can be "shadowed" and the resulting mesh erroneous at these locations. Some common underground conditions which can impair the line of sight of the scanning head are listed below:

- muck laying on sill and/or resting against walls;
- mesh hanging off walls or back;
- cables protruding from walls or back;
- extreme fog or dust; and
- irregular stope surfaces.

The accuracy of the resulting CMS mesh is also dependent on the density of data used to construct the mesh. This is controlled by both the operator and the set-up location. The operator has control over the angle at which the scanning head increments upwards about the pivot point before taking another set of measurements (total range of motion is 135°). A higher increment (i.e. 10° vs. 1°) produces significantly fewer data points. The operator also has some control when the data is being reduced. A choice can be made between using all the data points on a particular measurement ring (refer to Figure 3.2) or a portion of the data (i.e. data point at every 5° interval). With regard to set-up location, the greatest point density (and therefore greatest accuracy) is achieved close to the scanning head, as the measurement distance increases the point density decreases. The rate at which the point density decreases is related to the position of the scanning head relative to a particular surface. For example, the closer and more parallel the scanning head is to a particular surface, the quicker the point density decreases along that surface. This results from the shallow angle formed between the laser and the surface.

Based on the above discussion, it can be appreciated that the set-up location is very important with regards to obtaining quality CMS information. The chosen set-up location should provide a clear line of sight to the stope surfaces of interest and be located such that adequate point density is achieved. For very irregular and/or large stope walls it may be necessary to perform more than one survey to obtain adequate coverage. Furthermore, the operator should program the survey for maximum point density if irregular and/or large stopes are being surveyed.

Figures 3.3 and 3.4 are schematics which try to illustrate many of the factors discussed above. It can be seen from the figures that it is unlikely the CMS will over-estimate overbreak. However, it may over-estimate underbreak (lost ore) depending on "shadow" effects. Correspondingly, it is more likely measured stope volumes will be slightly under-estimated rather than over-estimated.

Based on the above discussion, it can be appreciated that it is possible to draw erroneous conclusions from a particular survey (i.e. too much ore loss; measured overbreak/slough less than actual), therefore, it is very important to make detailed notes and sketches during the survey. Areas which may give erroneous results due to shadows should be flagged and the survey results not used.

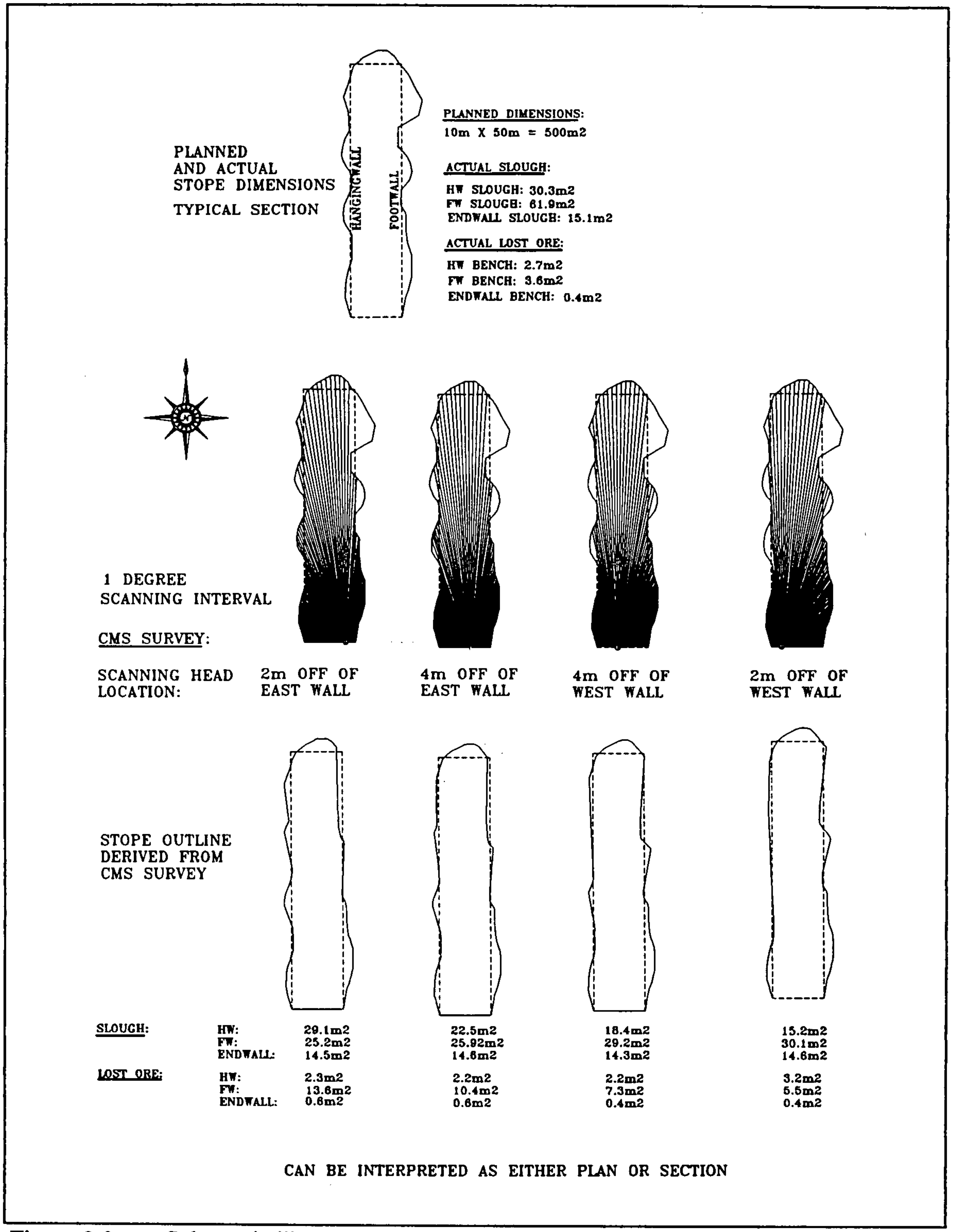


Figure 3.3 Schematic illustrating CMS limitations
Dense survey data (1° scanning interval)

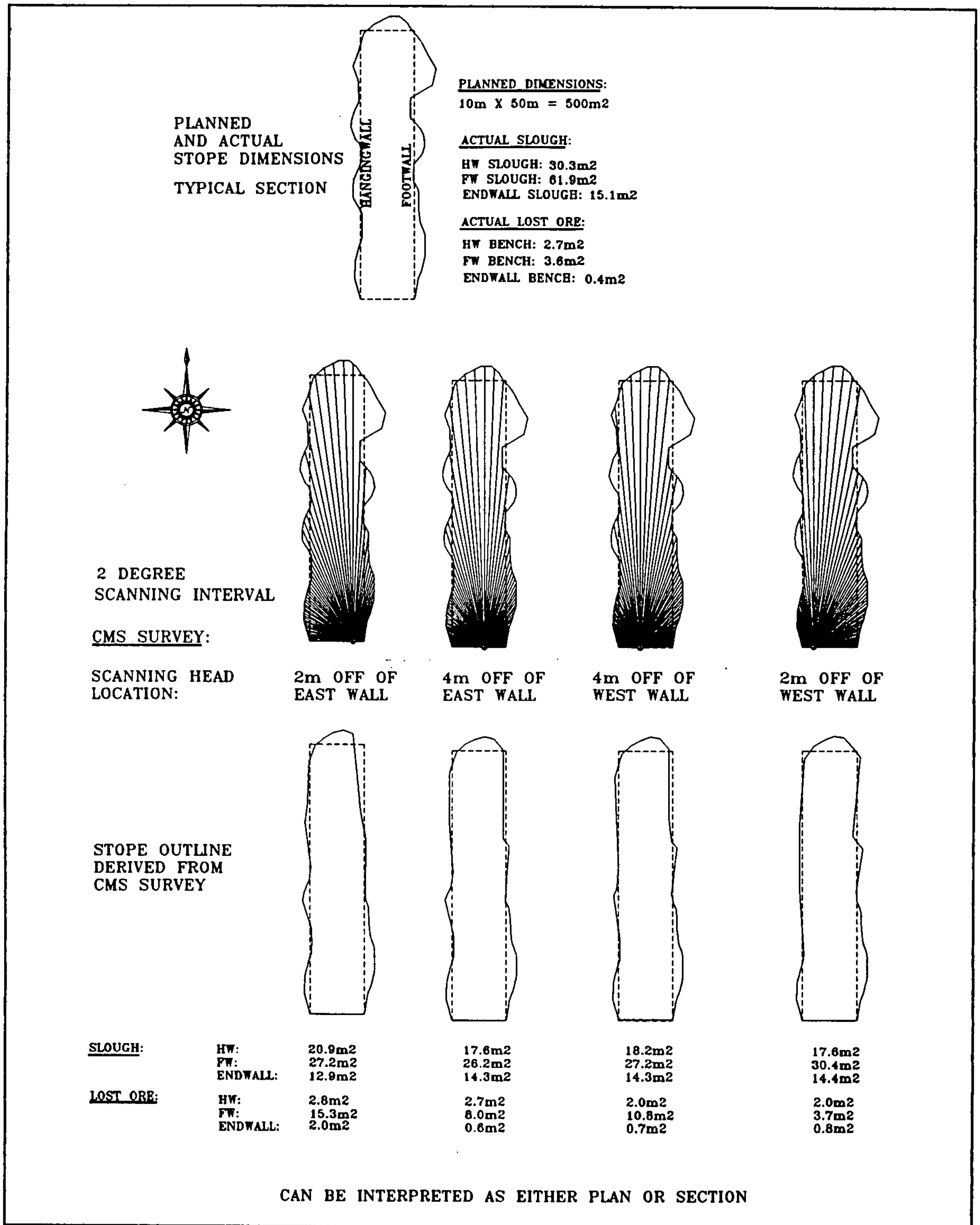


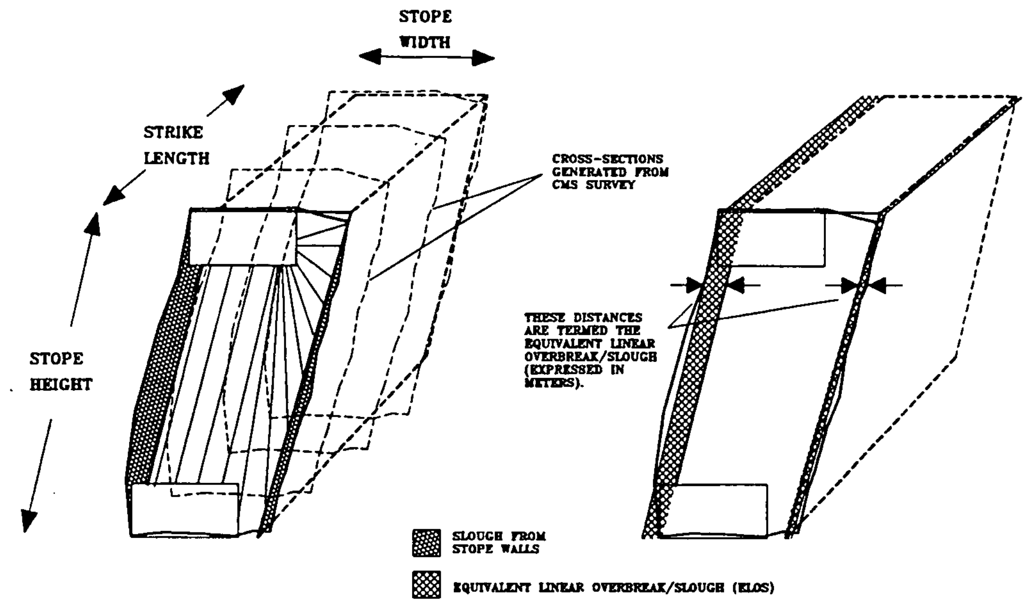
Figure 3.4 Schematic illustrating CMS limitations
Less dense survey data (2° scanning interval)

3.3 ELOS & ELLO - NEW PARAMETERS TO RELATE HANGINGWALL AND FOOTWALL PERFORMANCE

ELOS (Equivalent Linear Overbreak/Slough) and ELLO (Equivalent Linear Lost Ore) are alternate ways of expressing the volumetric measurements (m^3) of overbreak/slough and underbreak. They represent conversions of the true volumetric measurements into an average depth (ELOS) or thickness (ELLO) over the entire stope surface. A schematic describing ELOS and the method of calculation is shown in Figure 3.5. ELLO is calculated the same way except "volume of overbreak/slough" is replaced with "volume of underbreak". The attractiveness of these terms is that their meaning from a dilution or recovery point of view is more readily apparent than a volumetric measurement. For example, if 10m wide stopes are being designed and the ELOS is estimated to be 1.0m, approximately 10% unplanned dilution (by volume) can be expected. Conversely, an ELLO of 1.0m would represent a recovery of approximately 90% (i.e. Recovery = Actual Ore Mined / Planned Ore).

A benefit of using the ELOS parameter for empirical design is that it allows comparisons with other mining operations. This is not possible if dilution values are used since the values determined are a function of: stope width; grade of wall rock; and the associated tonnage factors.

Since the focus of this study is on estimating unplanned dilution, the ELOS parameter will be examined in much greater detail than the ELLO parameter.



PROCEDURE FOR CALCULATING EQUIVALENT LINEAR OVERBREAK/SLOUGH (ELOS)

1) CUT SECTIONS THROUGH THE CMS SURVEY ALONG THE LONGHOLE SECTION LINES.

2) ON EACH SECTION CALCULATE THE AREA (m^2) OF OVERBREAK/SLOUGH FROM EACH OF THE STOPE SURFACES. THE OVERBREAK/SLOUGH SHOULD BE MEASURED RELATIVE TO THE ORE CONTACTS OR MINING LINES. UNPLANNED DILUTION TAKEN DURING DEVELOPMENT OF THE ORE DRIFTS SHOULD NOT BE INCLUDED.

3) BASED ON THE AREAS CALCULATED FROM THE SECTIONS, CALCULATE A VOLUME OF SLOUGH (m^3) FOR EACH STOPE SURFACE. THIS WILL REQUIRE THAT A THICKNESS BE ASSIGNED TO EACH SECTION.

4) THE EQUIVALENT LINEAR OVERBREAK/SLOUGH (m) FOR A GIVEN SURFACE CAN BE CALCULATED USING THE EQUATION GIVEN BELOW:

$$\text{EQUIVALENT LINEAR OVERBREAK/SLOUGH} = \frac{\text{VOLUME OF SLOUGH FROM STOPE SURFACE}}{\text{STOPE HEIGHT X WALL STRIKE LENGTH}}$$

Figure 3.5 Schematic describing ELOS (Equivalent Linear Overbreak/Slough)

CHAPTER 4

CAVITY MONITORING SYSTEM (CMS) DATABASE

4.1 GENERAL

To date, the cavity monitoring system (CMS) database consists of 47 stope surveys from 6 different mines. The contributing mines are listed below:

> *Trout Lake Mine* (Hudson Bay Mining & Smelting Co. Ltd) - Flin Flon, Manitoba Contributed 4 stope surveys.
>
> *Ruttan Mine* (Hudson Bay Mining & Smelting Co. Ltd.) - Leaf Rapids, Manitoba. Contributed 4 stope surveys.
>
> *H-W Mine - Myra Falls Operation* (Westmin Resources Ltd.) - Buttle Lake, 90km west of Campbell River, Vancouver Island, British Columbia. Contributed 6 stope surveys.
>
> *Contact Lake Mine* - (Cameco Corporation) - Contact Lake, 45km northeast of La Ronge, Saskatchewan. Contributed 2 stope surveys.
>
> *Lupin Mine* - (Echo Bay Mines Ltd.) - Contwoyto Lake, Northwest Territories (Nunavut). Contributed 21 stope surveys.
>
> *Detour Lake Mine* - (Placer Dome Canada Ltd.) - Detour Lake, 210km northwest of Timmins, Ontario. Contributed 10 stope surveys.

All of the mines in the database utilize open stoping as the main mining method. Of the 47 stopes surveyed, 30 are considered primary stopes (i.e. all surfaces of the stope comprised of solid rock) and 17 were mined adjacent to backfill (longitudinal retreat stopes, therefore, far endwall comprised of fill). Mining depths ranged from approximately 75m to 1100m. The majority of stopes surveyed were located between depths of 600m to 1100m.

Volumetric measurements of overbreak/slough and underbreak, and the corresponding ELOS and ELLO values, have been recorded for: 31 unsupported hangingwalls; 16 cable bolted hangingwalls; 39 unsupported footwalls; and 2 cablebolted footwalls, resulting in a total of 88 measurements. This study did not analyze overbreak/slough from open stope backs or surfaces comprised of fill.

Referring back to Section 1.8 of Chapter 1, it can be appreciated that there are a number of factors that can influence the dilution and recovery for a particular stope. To help gain a better understanding for which factors are most critical with regard to stope wall stability, it was deemed necessary to collect extensive data complementary to each stope survey. The complementary data consists of details on:

- stope geometry;
- rockmass classification;
- undercutting of stope walls;
- drilling;
- blasting;
- stope support;
- time.

Figure 4.1 is a schematic showing the database structure.

4.2 DETAILED DESCRIPTION OF DATA

4.2.1 General

The CMS database provides the raw data for the development of an empirical design approach for estimating unplanned dilution. Note that empirical design methods are only applicable for design situations similar to those from which the method was derived. Therefore, when using any empirical design method it is imperative to have a good understanding of the data set used to develop the method.

A complete copy of the CMS database is presented in Appendix I.

A detailed description of the database is given in the following sections. Note that all the histograms presented for the remainder of this chapter are derived from 88 data points.

This is because in the CMS database each measurement of hangingwall or footwall overbreak/slough (88 measurements in total) is treated as an individual entity with 44 other pieces of associated information (i.e. Q', RMR; depth; HR; avg. blasthole length; etc.). Thus for many of the histograms there is a lot of duplicate information since only 47 stopes were surveyed. This is being stated so as to avoid any potential confusion when reviewing the following histograms and charts.

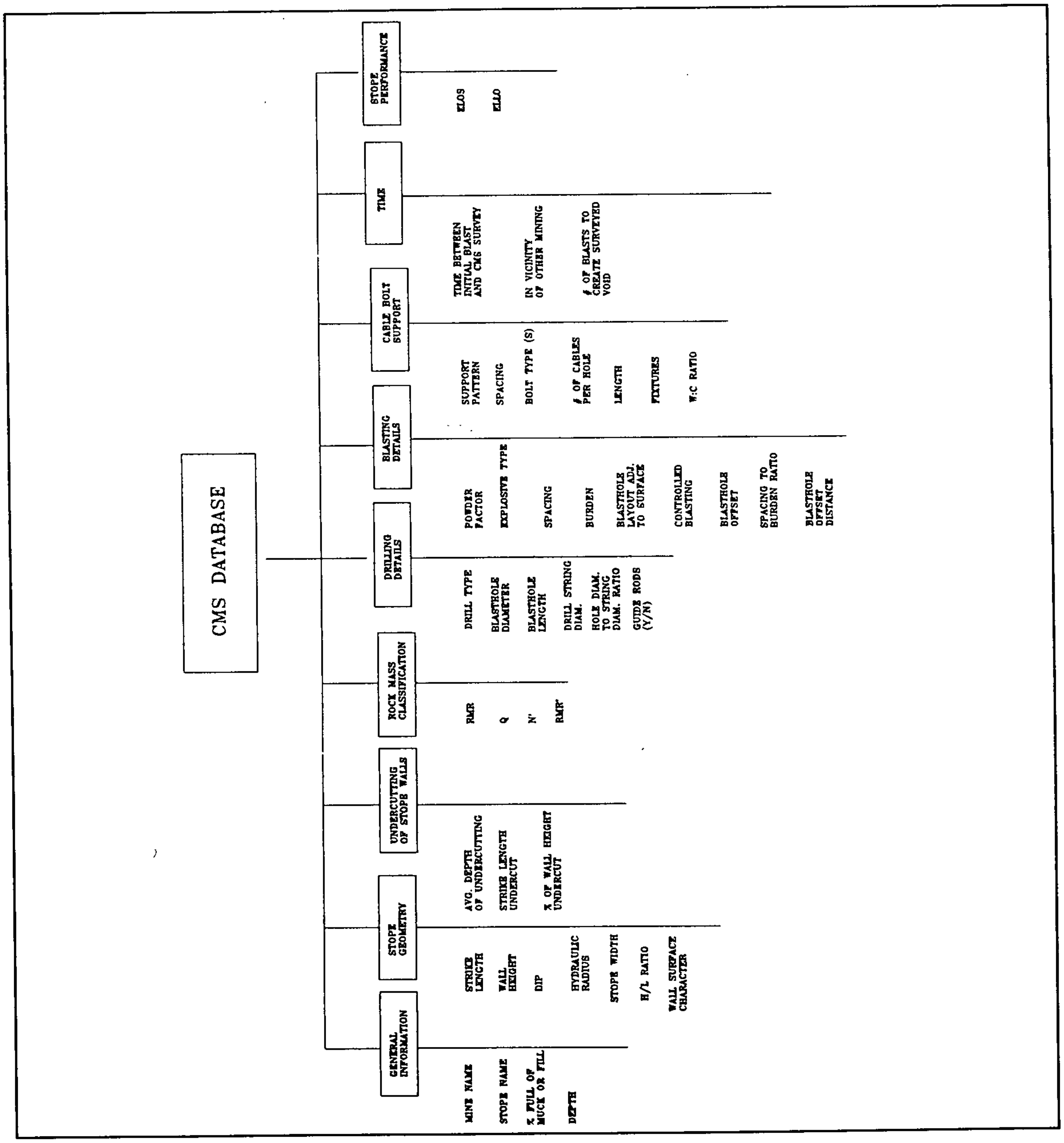


Figure 4.1 CMS database structure

4.2.2 Stope Geometry

Histograms describing the geometry of the stopes are presented in Figures 4.2 and 4.3.

In general, the majority of the stopes are: steeply dipping (>80°); have strike lengths of 15m to 30m; heights (in dip) of 20m to 70m; and height to length ratios of between 0.5 and 3.5. Stope widths varied between 1.4m and 37m, with approximately half of the stopes having widths less than 5m. The majority of wall surfaces had a regular surface character (i.e. relatively planar) and hydraulic radius values between 5m and 8m.

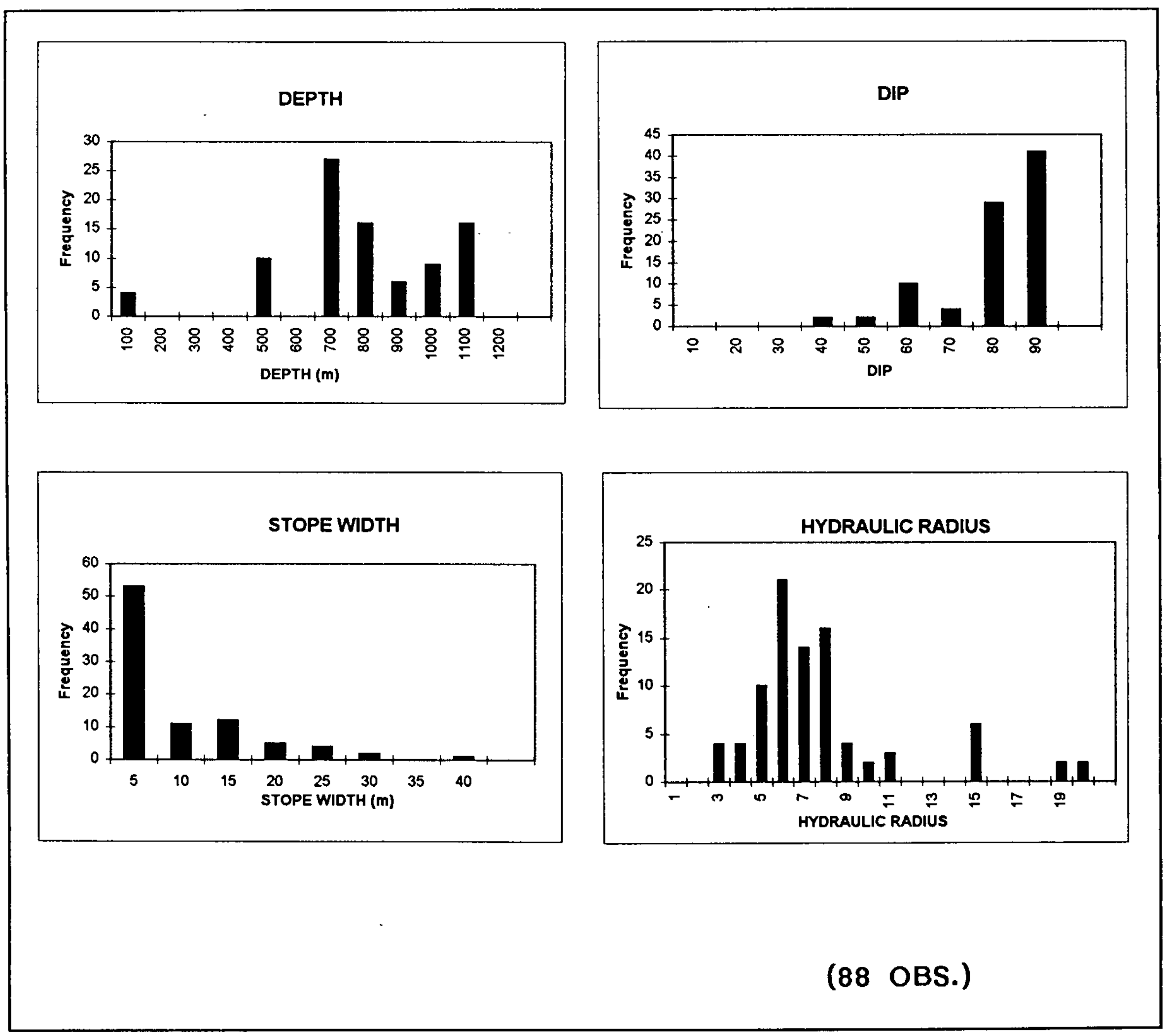


Figure 4.2 Histograms relating stope geometry information

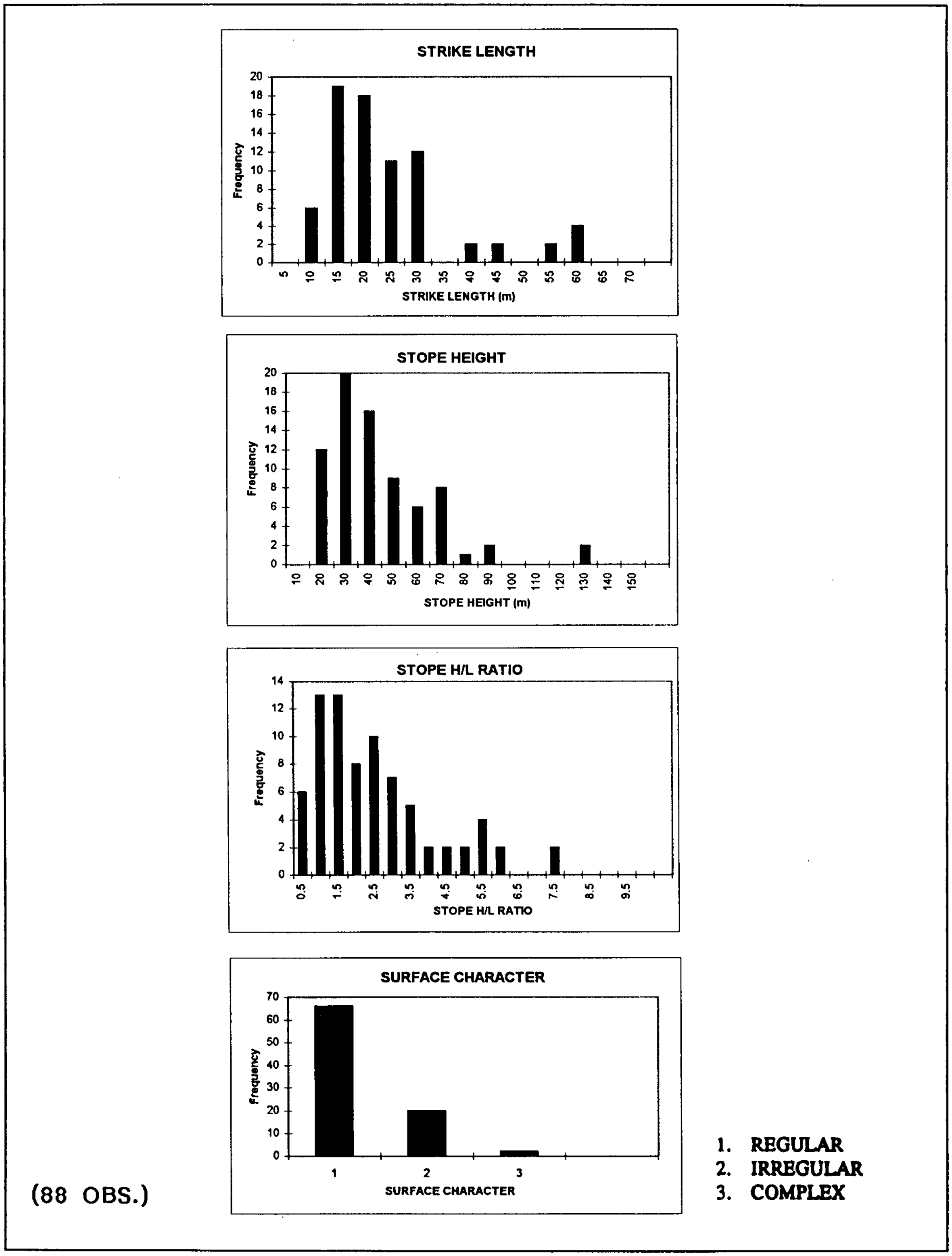


Figure 4.3 Histograms relating stope geometry information (cont'd)

4.2.3 Rockmass Classification

For each stope surveyed, the hangingwall and footwall rocks were classified according to the RMR (Bieniawski, 1976) and NGI "Q" (Barton, 1974) rockmass classification systems. With regard to the NGI system, Q' was determined (Mathews et.al., 1981) which involves arbitrarily assigning values of 1 to the stress reduction and joint water factors. Very little use was made of the relationship, $RMR = 9 \ln Q + 44$, the majority of rockmass classification values were determined based on the guidelines for the particular system. Histograms showing the distribution of RMR and Q' for the hangingwall and footwall rocks are shown in Figure 4.4. The relationship between RMR and Q' based on the rockmass classification data is shown in Figure 4.5.

Referring to Figure 4.4, both the hangingwall and footwall rocks varied between an RMR of approximately 45 to 80 (Q' between 1 and 30) with the majority having a RMR between 60 and 80 (Q' between 5 and 15).

4.2.4 Undercutting of Stope Walls

Undercuts in the database can be classified according to the system shown below:

1) No Undercut
2) Minor: < 0.5m
3) Moderate: 0.5 - 1.0m
4) Large: 1.0 - 2.0m
5) Severe: >2m

A histogram showing the undercutting of stope walls based on the above classification is presented in Figure 4.6. Referring to the figure, it can be seen that the majority of walls in the database were undercut to some degree.

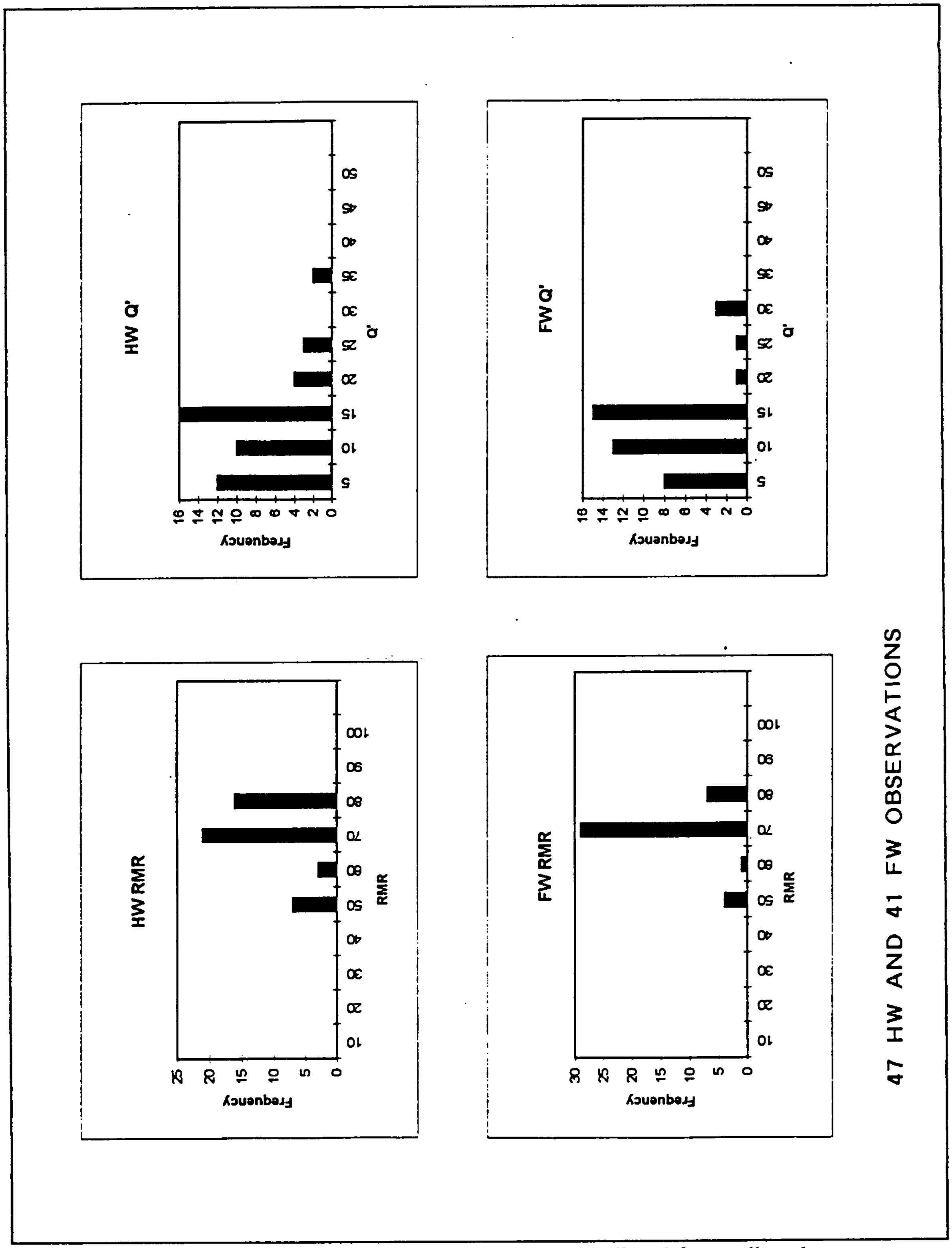


Figure 4.4 Distribution of RMR and Q' for hangingwall and footwall rocks

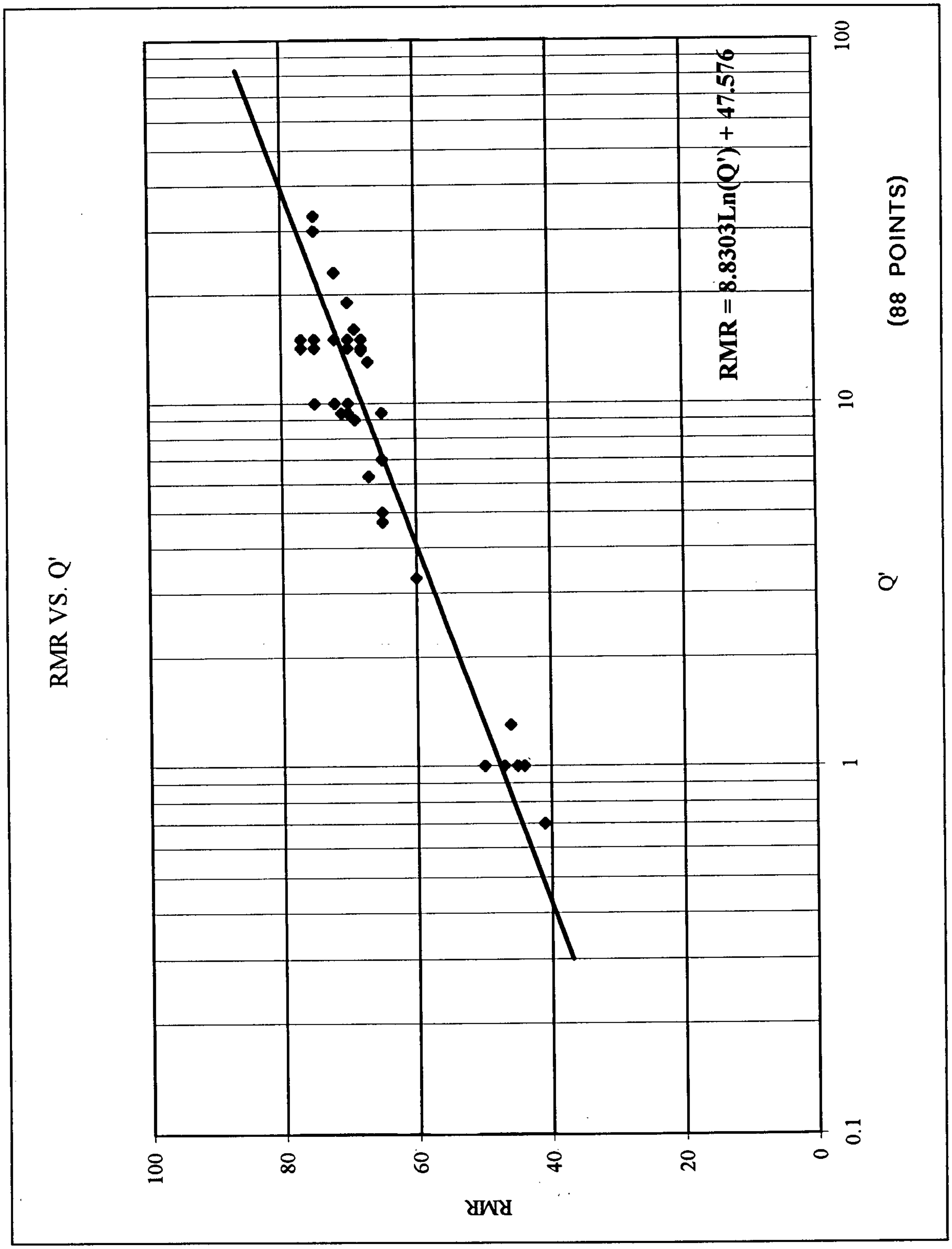
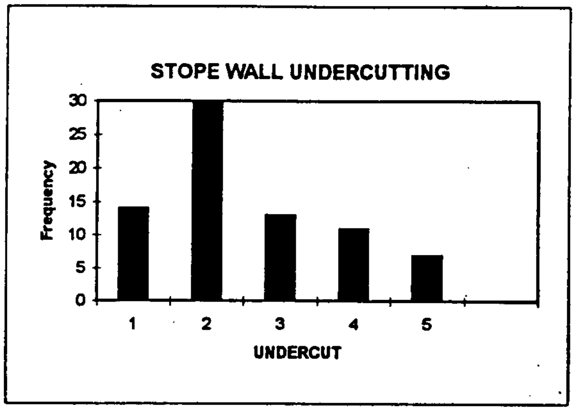


Figure 4.5 Relationship between RMR and Q'



1. **No Undercut**
2. **Minor Undercut: 0 - 0.5m**
3. **Moderate Undercut: 0.5 - 1.0m**
4. **Large Undercut: 1.0 - 2.0m**
5. **Severe Undercut: >2m**

(88 OBS.)

Figure 4.6 Histogram showing the classification of stope wall undercuts

4.2.5 Drilling

With regard to longhole drilling, 4 of the 6 mines in the database used electric-hydraulic top hammer drills, the other two utilized ITH (in-the-hole) drills. The table below lists the blasthole diameter/drill string combinations utilized by the mines in the database.

Table 4.1 - Blasthole Diameter / Drill String Combinations

Blasthole Diameter (mm)	**Drill String**
50 -65	R32
65-70	T38
76	T45
89	T51
100	76mm (ITH)
114-120	89mm (ITH)

A histogram showing the blasthole diameters recorded in the database is presented in Figure 4.7. It can be seen from the figure that the majority of stopes in the database were excavated using relatively small diameter blastholes (50 - 65mm). Figures 4.8 and 4.9 show relationships between: blasthole diameter and average blasthole length; and blasthole diameter and stope width, respectively. As would be expected a good correlation exists between blasthole diameter and average blasthole length. A weak correlation exists between blasthole diameter and stope width, although the plot does indicate a preference for smaller diameter blastholes for stopes less than 5m wide (i.e. largest diameter blasthole is 76mm).

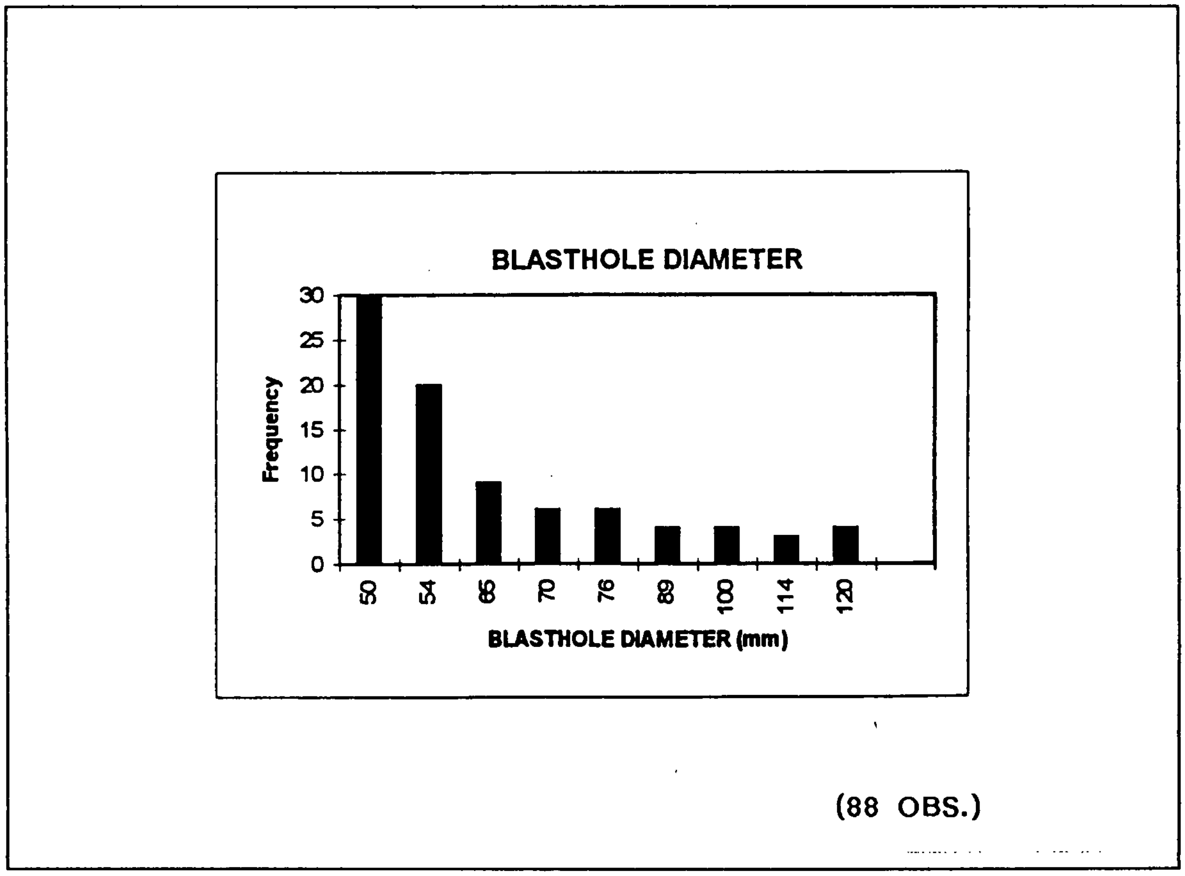


Figure 4.7 Histogram of blasthole diameters

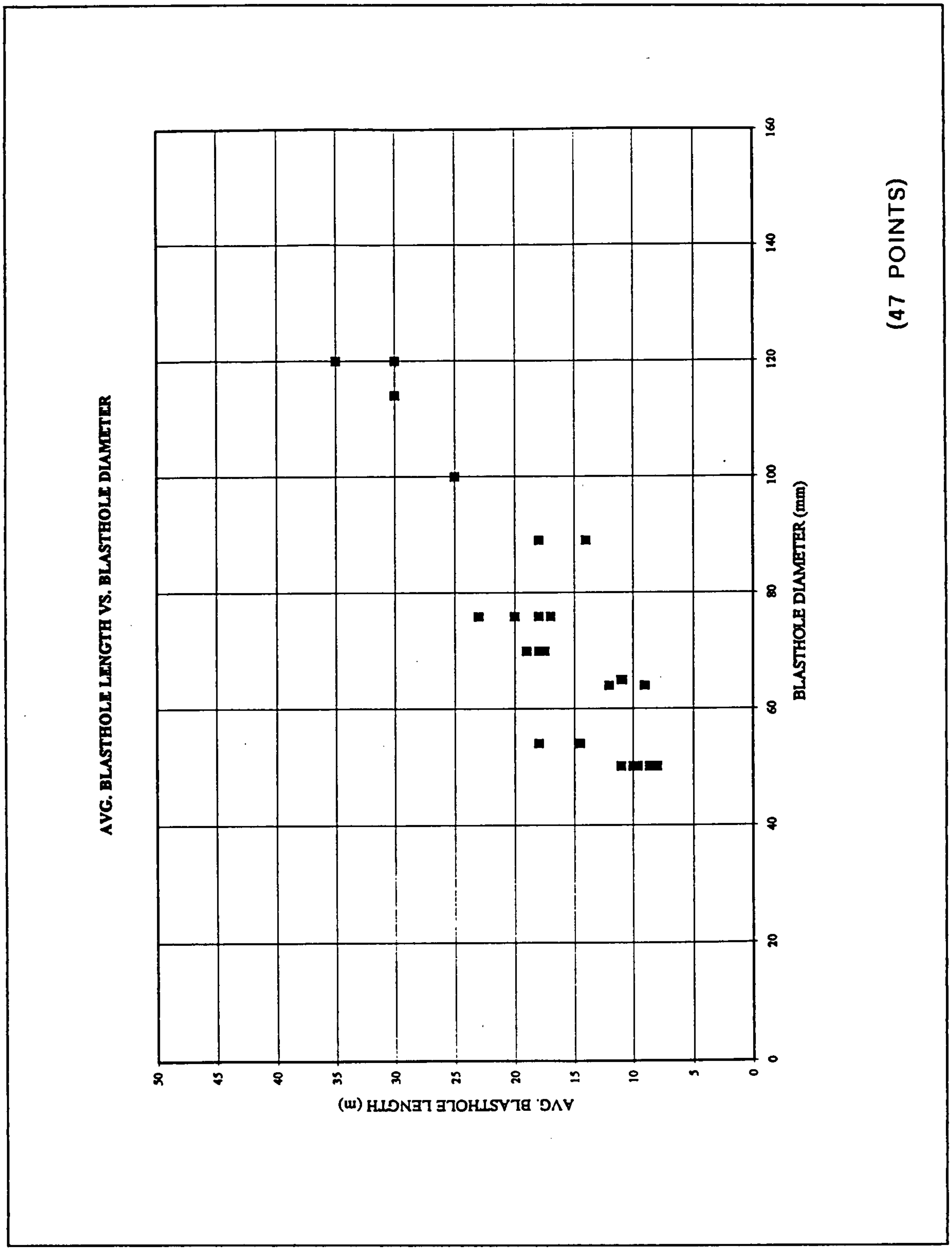


Figure 4.8 Relationship between blasthole diameter and blasthole length

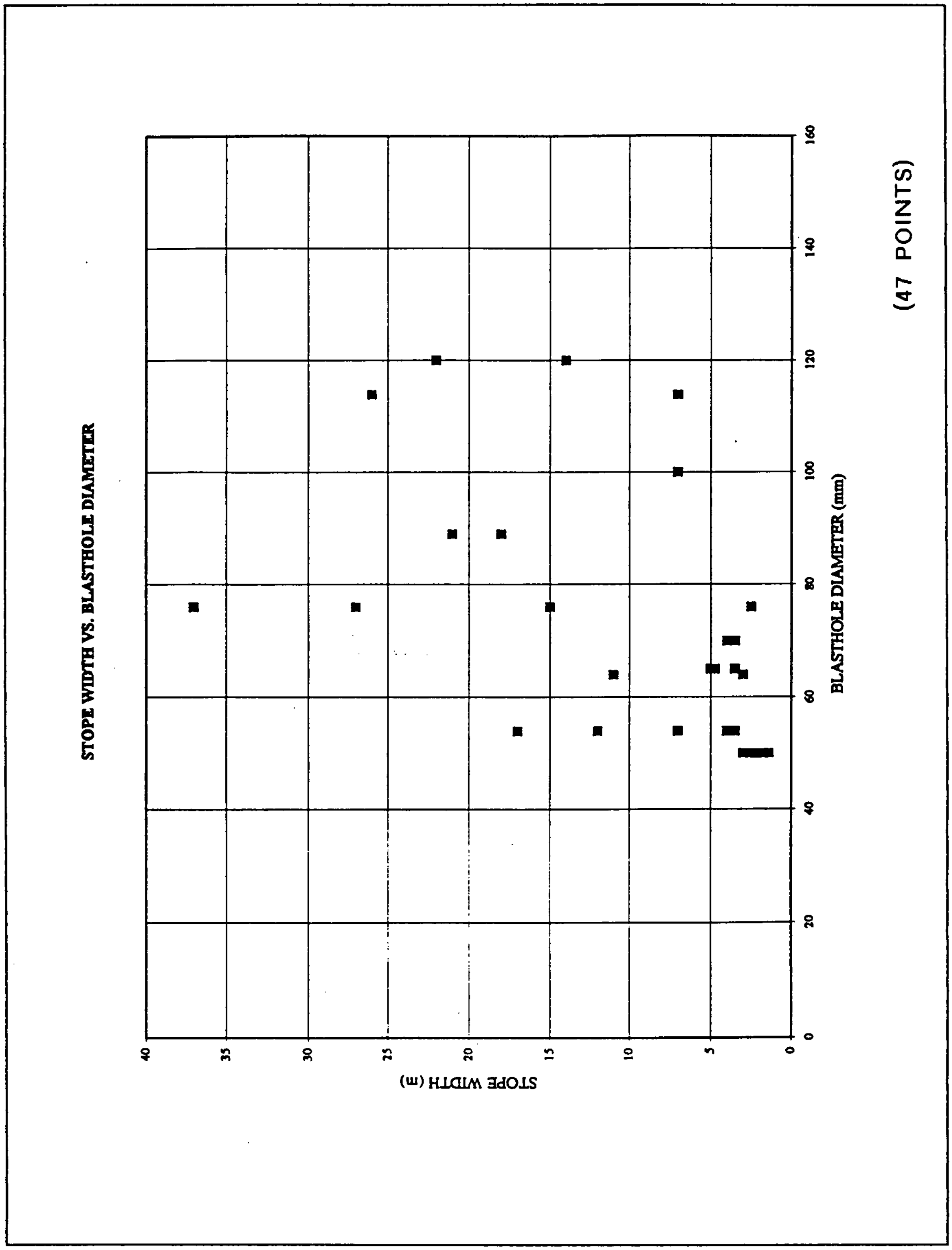


Figure 4.9 Relationship between blasthole diameter and stope width

4.2.6 Blasting

A plot showing blasthole burden versus blasthole diameter for the stopes in the database is shown in Figure 4.10. A rule of thumb in bench blasting is that the burden should equal approximately 20 to 35 times the blasthole diameter (Golder Associates Ltd., 1996), this is represented by the solid lines on Figure 4.10. The majority of the burdens fell in this range. The points falling outside the recommended range for burdens are associated with narrow stopes and indicate a tendency to drill tighter blast patterns.

Figure 4.11 shows a plot of spacing to burden ratio versus stope width. Another rule of thumb in bench blasting is that the spacing to burden ratio should equal approximately 1.2 (Golder Associates Ltd., 1996). Referring to the figure, the spacing to burden ratios roughly correlate with the rule of thumb except for narrow stopes (<5m). The scatter in the narrow stope region indicates that there is a tendency for both over-drilling (hole spacing too tight relative to burden) and under-drilling (spacing too large relative to burden). Over-drilling is the more common of the two.

Figure 4.12 shows histograms of: typical offset distances for perimeter blastholes; powder factors; and perimeter blasthole orientation relative to the stope wall (i.e. 1 - parallel; 2 - fanned ; 3 - parallel/fanned). Referring to the figure, blasthole offsets varied between 0 to 1.1m with the majority being between 0 to 0.5m. The large majority of perimeter blastholes were drilled parallel to the stope wall and powder factors for stope blasts varied between 0.3 kg/tonne to 1.2 kg/tonne. The higher powder factors in the database reflect the large number of narrow stopes surveyed. This is demonstrated quite clearly in Figure 4.13 which shows a plot of powder factor versus stope width.

Anfo was the main explosive used for 42 of the 47 stopes surveyed. Emulsion explosives were used for the remaining five stopes.

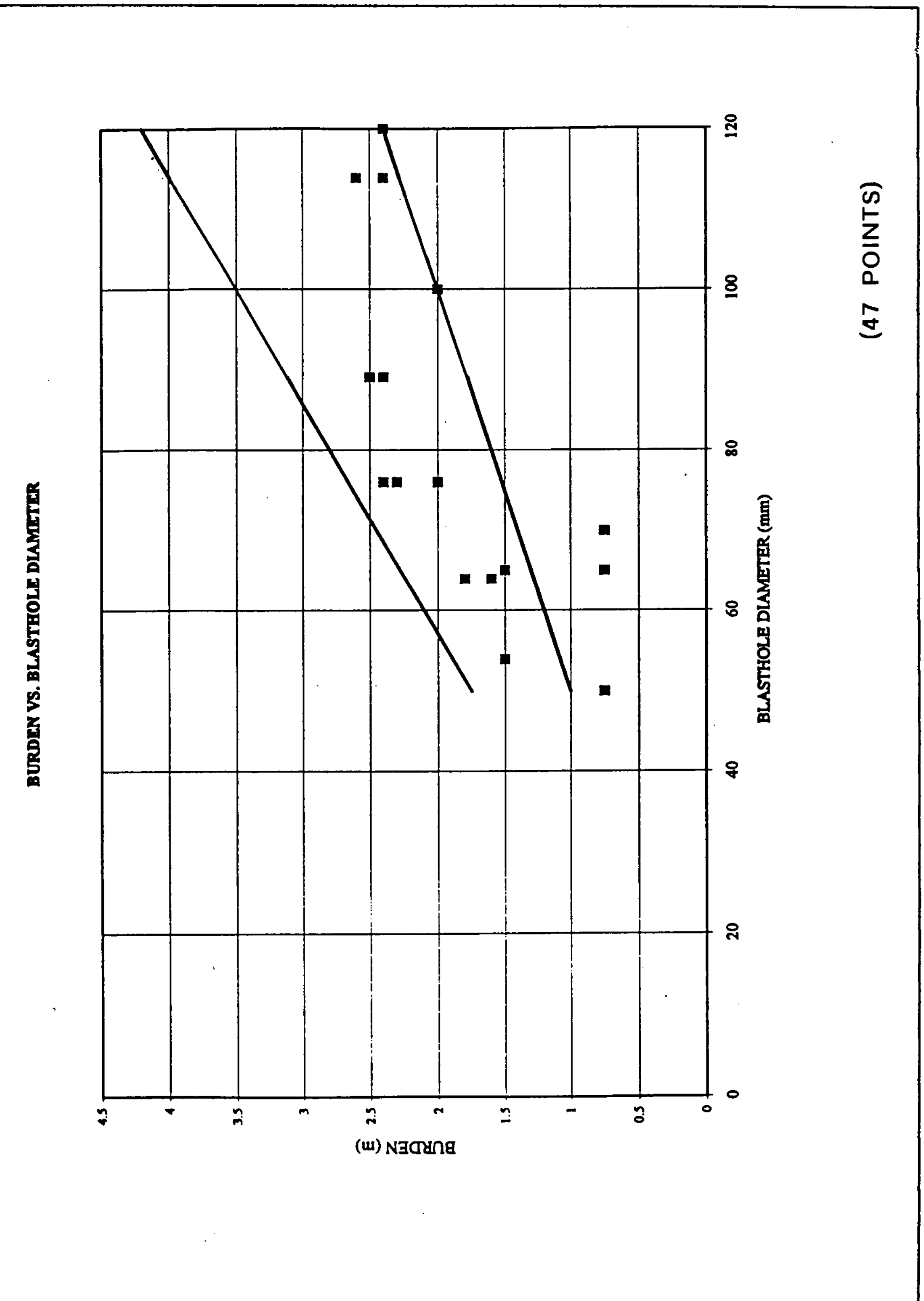


Figure 4.10 Relationship between burden and blasthole diameter

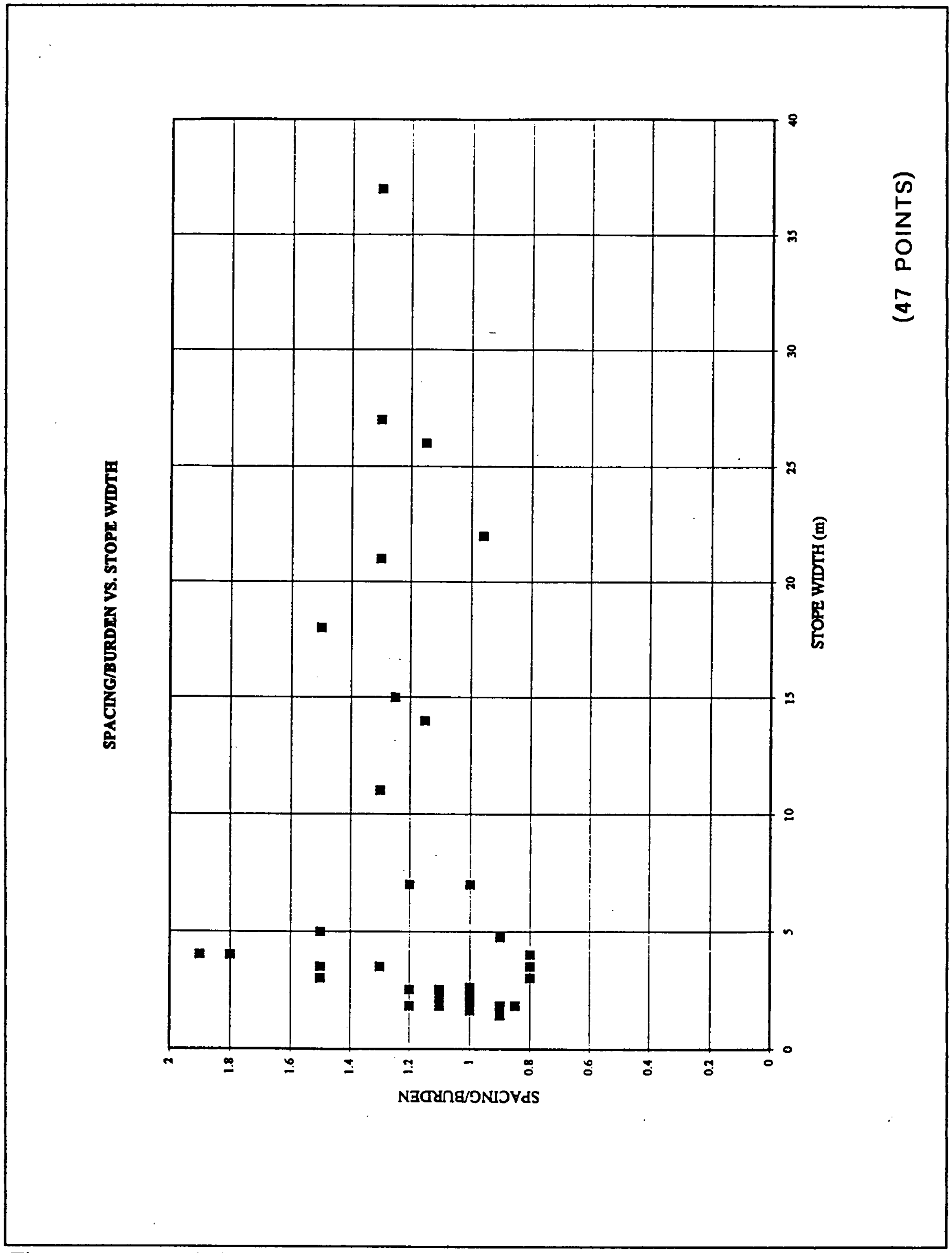


Figure 4.11 Relationship between spacing/burden ratio and stope width

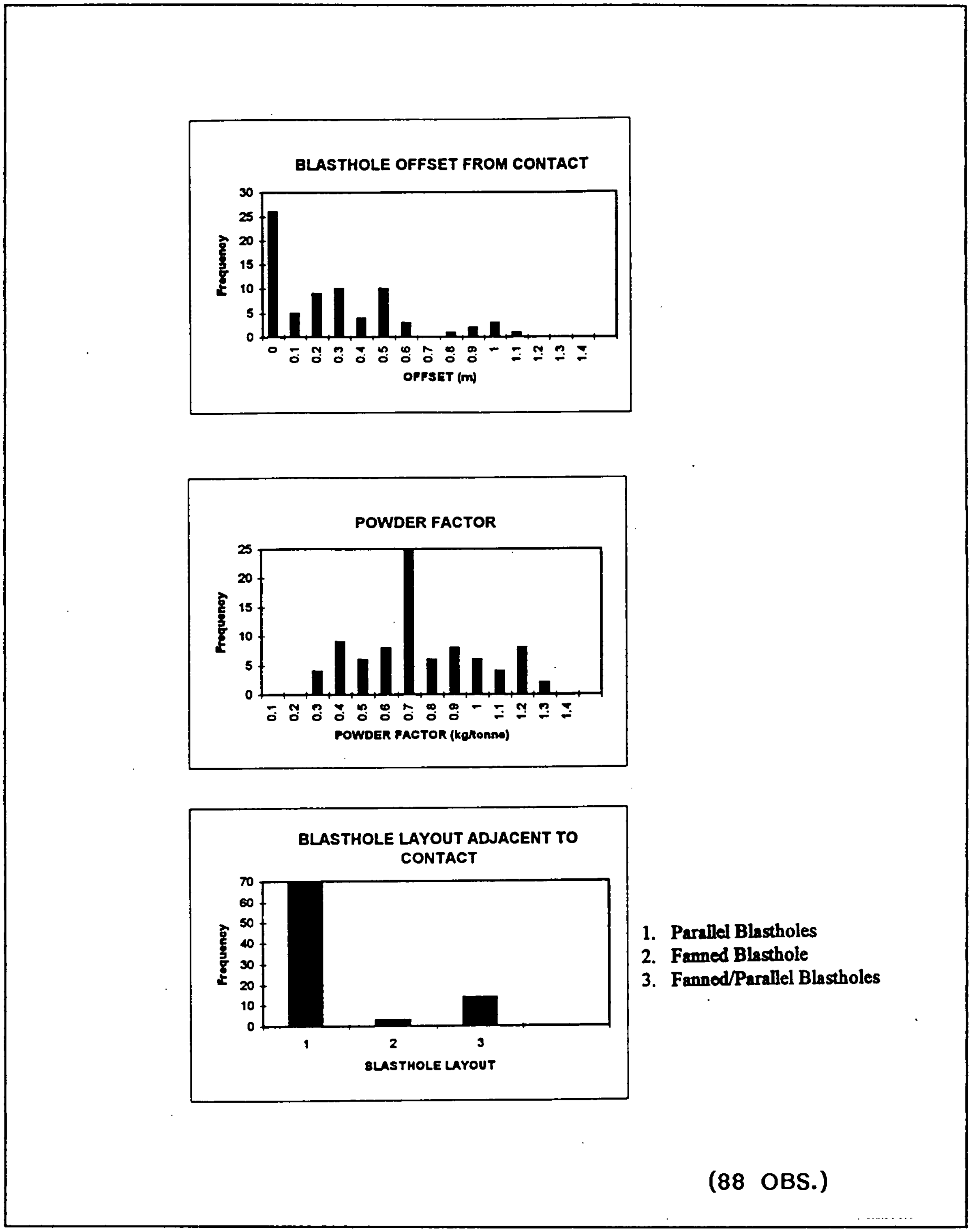


Figure 4.12 Histograms showing: offset distances; powder factors; and perimeter blasthole layout

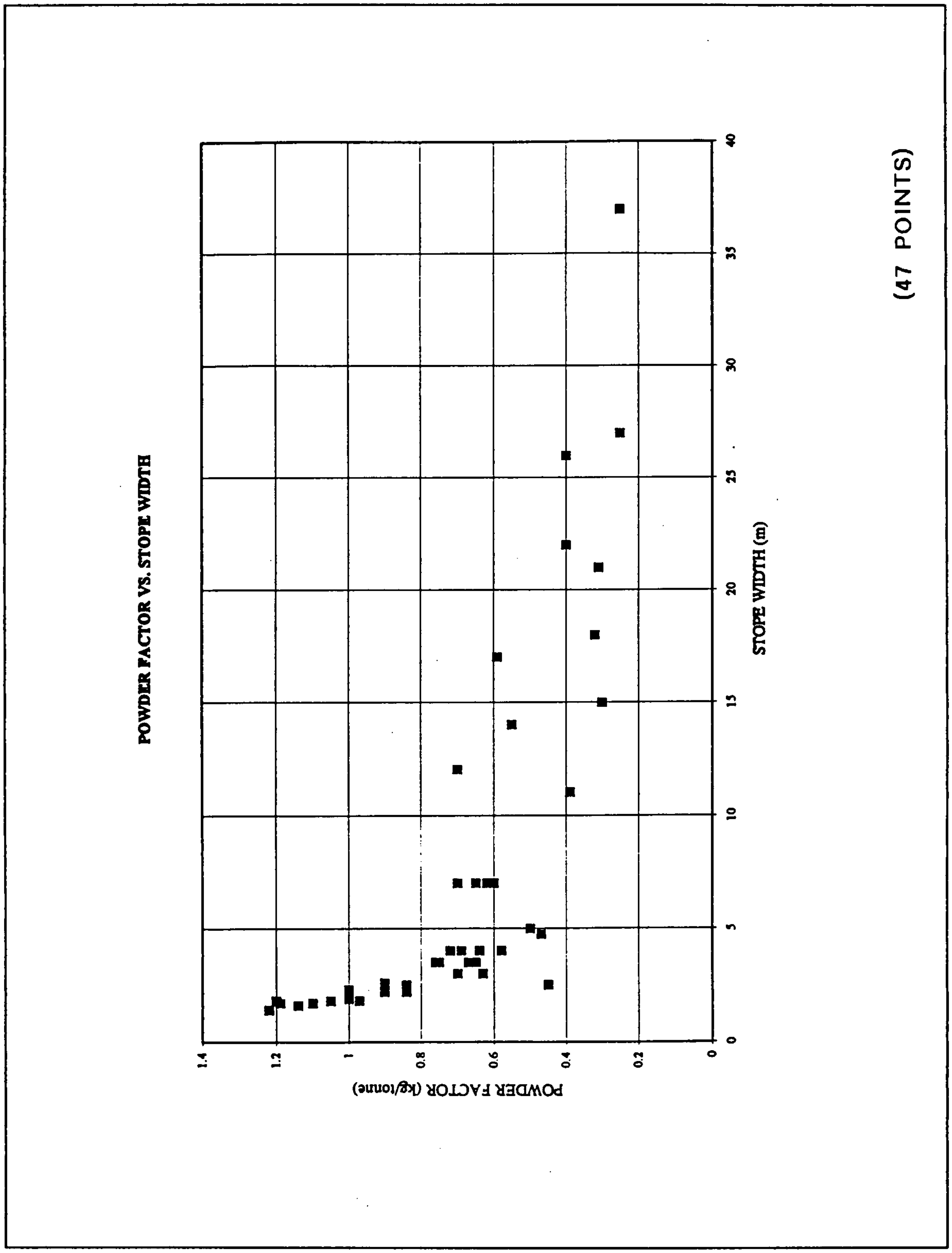


Figure 4.13 Relationship between powder factor and stope width

Controlled blasting adjacent to the hangingwall or footwall was only attempted in 6 of the 47 stopes surveyed and generally involved using Lomex or decoupled cartridge explosives in the perimeter holes. No modifications to the regular blasthole patterns were made for the controlled blasting attempts (i.e. burden on perimeter holes was not reduced).

For the majority of stopes in the database, between 2 to 9 longhole blasts had occurred prior to the stope survey, refer to the histogram presented in Figure 4.14.

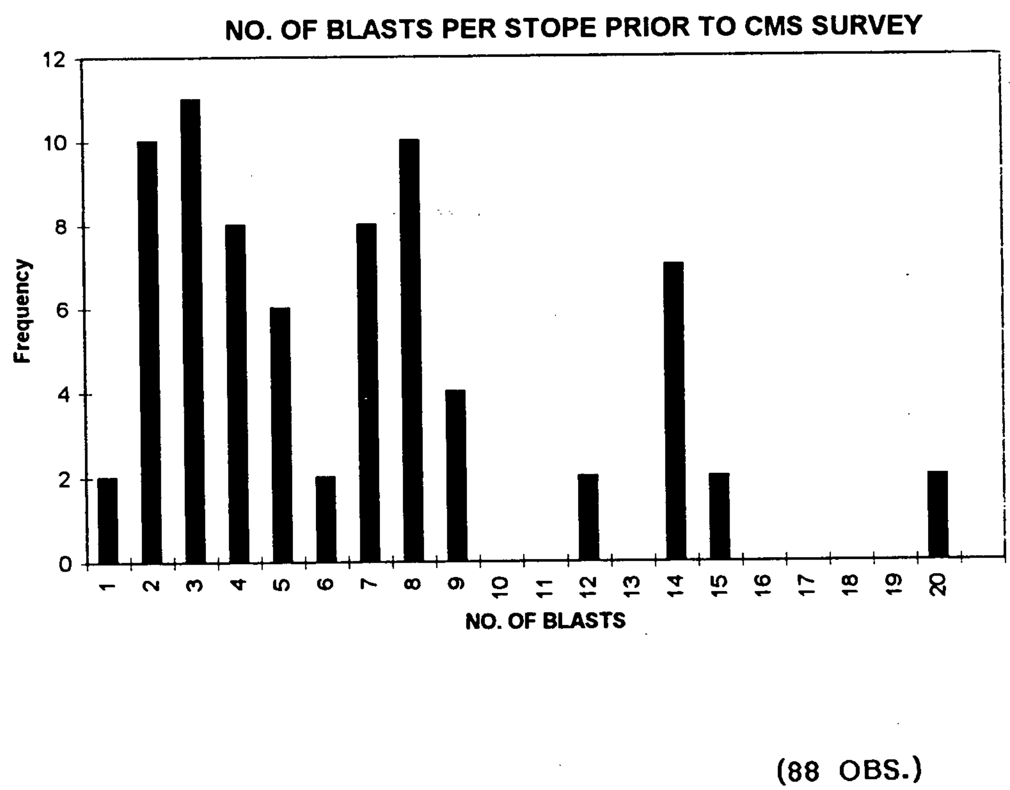


Figure 4.14 Histogram showing number of longhole blasts per stope prior to being surveyed

4.2.7 Stope Support

Of the 47 hangingwalls in the database, 16 were supported with cablebolts. Of the 41 footwalls, 2 were supported with cablebolts. In all cases, cablebolt support consisted of cables installed at the sub-levels (point anchor approach), no pattern bolted wall surfaces were surveyed. For 14 of the cablebolted hangingwalls and both of the cablebolted footwalls, the sub-level spacing was approximately 20m. For the remaining 2 cablebolted hangingwalls, the sub-level spacing was approximately 60m (Ruttan Mine).

The cablebolt support pattern was generally a fan of 3 cable holes every 2 to 3m along strike. Double cables (2 per hole) were used in the majority of the cases. Cablebolt length varied from 4.5 to 15.5m. Face plates were installed on all the cables.

4.2.8 Time

A histogram showing stope life (time between first stope blast and CMS survey) for each of the stopes in the database is presented in Figure 4.15. Referring to the figure, stope life ranged from 2 to 300 days with the majority of stopes having a life of between 15 to 50 days. Figure 4.16 shows a plot of stope life versus stope volume. Referring to the figure, the majority of stopes in the database are small volume (2000 to 3000 m^3) with a relatively quick cycle time.

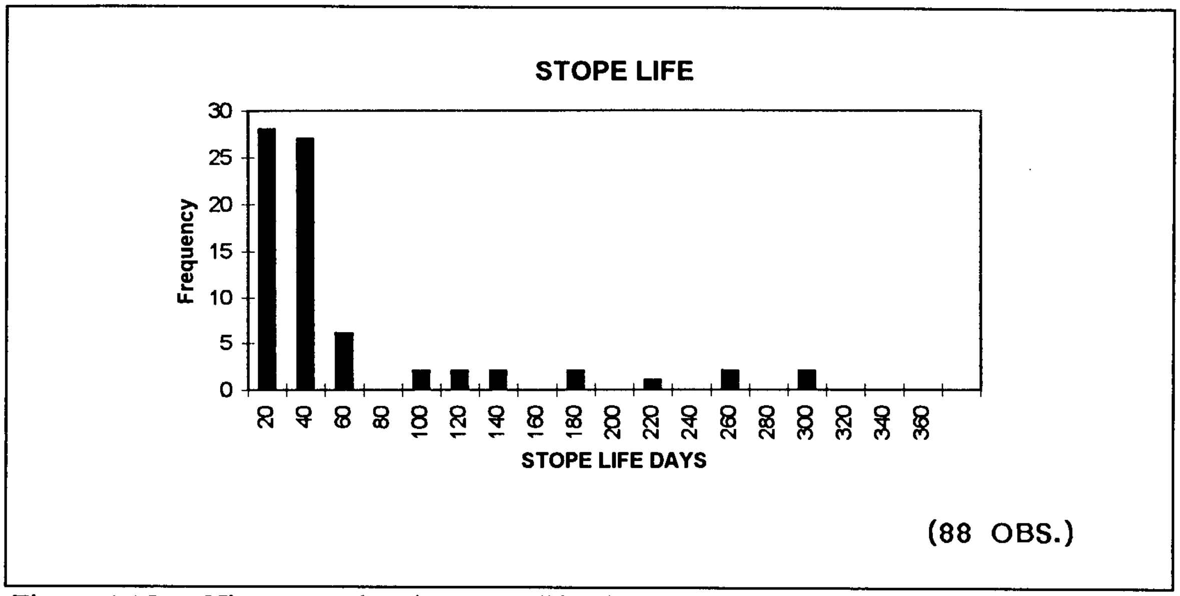


Figure 4.15 Histogram showing stope life (time between first longhole blast and CMS survey)

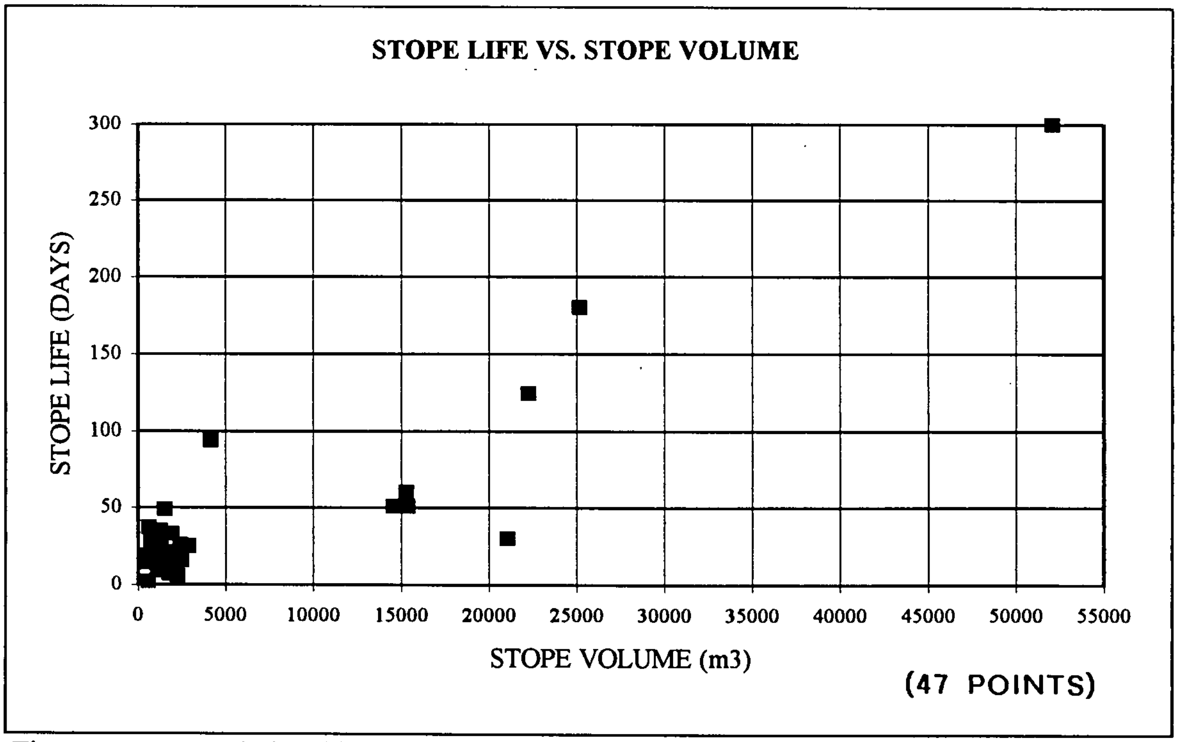


Figure 4.16 Relationship between stope volume and stope life

4.3 SUMMARY

Average values for the main parameters in the CMS database are listed in Table 4.2 below.

Table 4.2 - Average Values for Main Parameters in CMS Database

CMS Database Parameter	Average Value	Remarks
Depth	740 m	
Stope Dip	76 degrees	
Stope Width	7 m	53% of stopes have widths ≤ 4 m.
Stope Strike Length	27 m	
Stope Height	39 m	
Stope Height/Length	2.1	
Hydraulic Radius	7.3	
Wall Surface Character	Regular	
Stope Wall Undercutting	Minor/Moderate	Typical undercut 0.5 - 1 m.
RMR	67	
Q'	12	
Blasthole Diameter	65 mm	63% of stopes mined with blastholes ≤ 65 mm diam.
Blasthole Length	15 m	
Blasthole Offset Distance	0.3 m	
Explosive Type	Anfo	
Powder Factor	0.7 kg/tonne	
Perimeter Blasthole Orientation	Parallel	
Cablebolt Support	34% of HW's supported; 5% of FW's supported	Cables installed on sub-levels; no pattern support
Time Between First Stope Blast and CMS Survey	51 days	53% of stopes mined in ≤ 30 days.
Number of Longhole Blasts to Excavate Stope	7	

Important points to consider with regard to the CMS database are as follows:

- there are only 88 data points;

- there is a bias towards narrow stopes mined with small diameter parallel blastholes (i.e. 50mm - 65mm diameter) in fair to good quality rock;

- more data is needed from mines that utilize larger diameter blastholes;

- the database needs more large stopes (i.e. hydraulic radius values >10) and more stopes in poor quality rock;

- there is limited data with regard to supported stope surfaces;

- the database is limited to stope hangingwalls and footwalls.

CHAPTER 5

DEVELOPMENT OF AN EMPIRICAL DESIGN APPROACH FOR ESTIMATING UNPLANNED DILUTION FROM OPEN STOPE HANGINGWALLS AND FOOTWALLS

5.1 INCORPORATING ELOS INTO THE MODIFIED STABILITY GRAPH

5.1.1 General

As discussed previously, the Modified Stability Graph Method (Potvin, 1988) has generally been well accepted by industry for open stope design. For this reason, it was chosen as the starting point for developing an empirical method for estimating unplanned dilution.

The approach taken to incorporate ELOS into the Modified Stability Graph Method was to simply plot the location of the hangingwall and footwall data points on the stability graph and at each location plot the associated ELOS value. This approach enables the following:

- quantification of the different design zones on the stability graph in terms of volume of overbreak/slough;
- validation of the methodology associated with the Modified Stability Graph with measured quantifiable results.

A potential drawback with this approach is plotting ELOS with hydraulic radius. The complicating factor is that stope walls with the same hydraulic radius can have different surface areas. For example, tall stope walls with a short strike length, or, short stope walls with a long strike strength, will have a greater surface area than a square shaped stope wall, given the same hydraulic radius. Since ELOS is calculated by dividing the volume of overbreak/slough by the wall surface area, there is potential to get some scatter in the ELOS values associated with a particular hydraulic radius and rock quality. To disregard the possible scatter effect it must be shown that, for a given rock quality, stope walls with the same hydraulic radius but different surface areas will produce similar ELOS values. In other words, given two stope walls with the same hydraulic radius but different surface areas, the stope wall with the greater surface area has to produce a greater volume of overbreak/slough than the stope wall with the smaller surface area in order to arrive at a similar ELOS value. Intuitively this seems reasonable. Numerical modelling provides some theoretical justification for this line of reasoning and is shown schematically in Figure 5.1. The numerical modelling is explained in detail in Chapter 6. Field measurements taken from the CMS database also provide some support, refer to Table 5.1 below.

Table 5.1
Influence of Hydraulic Radius (HR) and Height/ Length Ratio (H/L) on ELOS

Stope	N'	H/L	HR	Surface Area (m^2)	Volume of Slough (m^3)	ELOS (m)
WZN1010	2.4	5.1	5.4	858	858	1.0
CZ-990	2.0	1.4	5.2	450	360	0.8
WZS1010	2.4	5.2	5.4	871	871	1.0

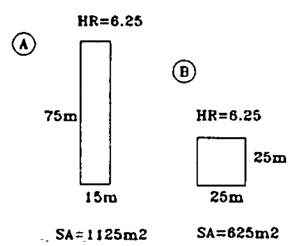
Comparisons such as this are complicated by many other factors (i.e. time stope was open; undercutting; blasting; etc..) therefore more data is needed before definite conclusions can be drawn.

An alternative would be to plot, *N' vs. Wall Surface Area*, instead of, *N' vs. Hydraulic Radius*. This would eliminate the potential scatter problem discussed above, however, from a rock mechanics perspective, surface area is not as meaningful as hydraulic radius. Surface area does not account for the distance to supporting abutments whereas hydraulic radius does. This is likely not that significant when dealing with smaller stopes (i.e. HR < 5 or 6) in good quality rock, however, in poor quality rocks or in big stopes the distance to the supporting abutments plays a critical role in the stability of the surface. The latter are also the most critical with regard to design. Based on this reasoning hydraulic radius was considered a more appropriate design parameter and surface area was not used.

Given all the factors that can influence the stability of a stope wall a significant amount of scatter in the data must be expected. At this stage only broad design zones can realistically be identified. Much more data, than is currently in the database, is required to refine the approach.

THEORETICAL BACK-UP FOR PLOTTING
ELOS WITH HYDRAULIC RADIUS (HR)

BASED ON 3D NUMERICAL MODELLING OF VERTICAL STOPES



NUMERICAL MODELLING SHOWS THAT FOR A GIVEN HR THE MAXIMUM DEPTH OF RELAXATION IS APPROXIMATELY CONSTANT REGARDLESS OF THE STOPE HEIGHT/LENGTH RATIO.

ASSUMING K=2 AND INSITU STRESSES PERPENDICULAR TO THE SURFACE, THE MAXIMUM DEPTH OF RELAXATION IS APPROXIMATELY 1.7m FOR BOTH GEOMETRIES.

ASSUMING THE VOLUME OF THE RELAXED ZONE CAN BE APPROXIMATED BY HALF OF AN ELLIPSOID:

VOLUME A = 0.5 $((4/3\pi)(1.7m)(37.5m)(7.5m))$ = 1001m3

VOLUME B = 0.5 $((4/3\pi)(1.7m)(12.5m)(12.5m))$ = 556m3

AND

ASSUMING THE VOLUME OF THE RELAXED ZONE IS CORRELATABLE TO THE VOLUME OF SLOUGH REALIZED FOR A PARTICULAR STOPE, THE RESULTING ELOS WOULD BE:

ELOS A = 1001m3/1125m3 = 0.9m

ELOS B = 556m3/625m3 = 0.9m

THIS INDICATES THAT VOLUMETRICALLY YOU MAY GET MORE SLOUGH WITH A STOPE WITH A LARGER H/L RATIO BUT BECAUSE THE STOPE ITSELF IS BIGGER THE ELOS WILL STAY APPROXIMATELY THE SAME AS A STOPE WITH A LOWER H/L RATIO AND THE SAME HR.

Figure 5.1 Theoretical justification for plotting ELOS vs. HR

5.1.2 Details on the Calculation of the Modified Stability Number (N')

With regard to the calculation of the Modified Stability Number (N') the following criteria were consistently used:

- the RQD was generally based on the spacing of the joint set parallel to the stope walls;
- Jr and Ja were based on the critical joint set with regard to stability which was always the joint set parallel to the stope walls;
- the A factor was always set to a value of 1 (stope wall in relaxed or low stress state);
- the B factor was always set to a value of 0.3 (critical joint set with regard to stability parallels the stope wall);
- the C factor equaled: 8-6cos∅ for hangingwalls (where ∅ equals the hangingwall dip); and 8 for all footwalls.

The following paragraphs discuss the assumptions made regarding the A, B, and C factors.

Stress Factor - A

The A factor accounts for the induced stress in the particular stope surface being analyzed. The factor is determined using the chart presented in Figure 5.2.

In this study, the A factor was always assumed to be equal to 1 (stope wall in a relaxed or low stress state). The reasoning behind this assumption is presented below:

- Arjang (1991) shows that in hardrock mines with near vertical orebodies the maximum horizontal stress is commonly oriented approximately perpendicular to the strike of the orebody and has a magnitude of approximately twice the vertical stress. The minimum horizontal stress is generally aligned along strike. It can be shown through numerical modelling that in this stress regime hangingwalls and footwalls are generally under low stress or in relaxation.
- No evidence of stress induced hangingwall or footwall failure was observed or noted during the collection of the CMS data contained in the database.

- There is very little precedence for using an A value other than 1 for hangingwalls and footwalls. In the original Potvin (1988) database, all but one of the hangingwall and footwall data points (106 observations) were assigned an A value of 1, the other was assigned an A value of 0.9.

Having said the above, this author has observed both hangingwall and footwall failures which were suspected to have been caused, at least partially, by the prevailing stress conditions. Indicating that in certain situations an A value less than one may be appropriate. However, as stated above, the existing CMS database contains no cases in which high stress conditions are suspected as being a contributing factor to the measured overbreak/slough.

Determine maximum induced tangential stress (compressive) acting at the centre of the stope face being considered. Obtain uniaxial compressive strength for the rock. Evaluate Stress Factor, *A*, using the graph below:

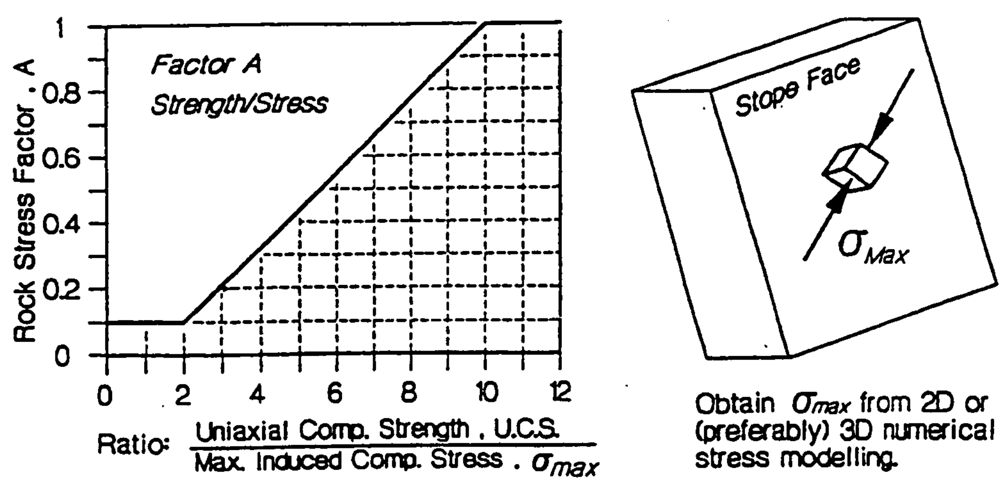


Figure 5.2 Chart for determining A factor (after Potvin, 1988)

It is this authors opinion that more research is needed to fully understand the effects of stress on hangingwall and footwall stability. For example, with regard to Figure 5.2, it does not seem intuitively obvious that a value of 1 (the best stability rating) should be assigned to stope surfaces in a relaxed stress state. From a dilution perspective a relaxed stress state would seem unattractive since there is no confining stress to aid with stability. Furthermore, since confining stress aids stability, within certain limits stress in stope walls should be beneficial to stability. Whether this is adequately accounted for in Figure 5.2 is uncertain. Other factors that should be considered include: the potential for high stresses to pre-condition the hangingwall and footwall rocks prior to mining; and the relationship between stress levels in a mining block and the rapid offloading off the stress immediately following a longhole blast. Villaescusa (1995) states that high pre-mining and/or mining induced stresses acting normal to the orebody cause greater offloading which increases the fissility of the hangingwall rocks. This has the effect of reducing the overall stability of the surface.

Joint Orientation Factor - B

This factor accounts for the orientation of structure relative to the stope walls. The factor is determined using the chart presented in Figure 5.3. The structure most critical to the stability of the wall is used to determine the appropriate B factor.

For all the stopes in the database, the critical structure with regard to stability was parallel to the stope walls. The parallel structures were either: joint sets; bedding planes; or foliation planes.

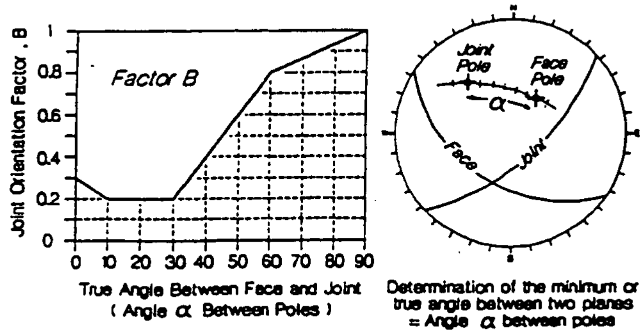


Figure 5.3 Chart for determining B factor (after Potvin, 1988)

Gravity Factor - C

The C factor accounts for the mode of failure (buckling/slabbing vs. sliding) and the effect of gravity. The buckling/slabbing mode of failure applies to hangingwalls whereas the sliding mode of failure applies to footwalls. The charts shown in Figure 5.4 are used to determine the appropriate C values.

Upon inspection the general shape of the two graphs make intuitive sense (i.e. hangingwalls become less stable as the dip decreases and footwalls become less stable as the dip increases). However, what does not make intuitive sense is the actual weighting factors. For example, a footwall with parallel structure dipping at 90° has a weighting of C=2, whereas a hangingwall with parallel structure dipping at 90° has a weighting of C=8. In reality, there is no difference between a 90° dipping footwall and a 90° dipping hangingwall therefore the C factors should be equal. This suggests that the minimum C value for a footwall should be 8 and as the dip decreases the C value should increase. This concept is shown schematically in Figure 5.5. This intuitively makes sense since it is well accepted that in a given rock type the hangingwall is less stable than the footwall except at 90° where there is no difference.

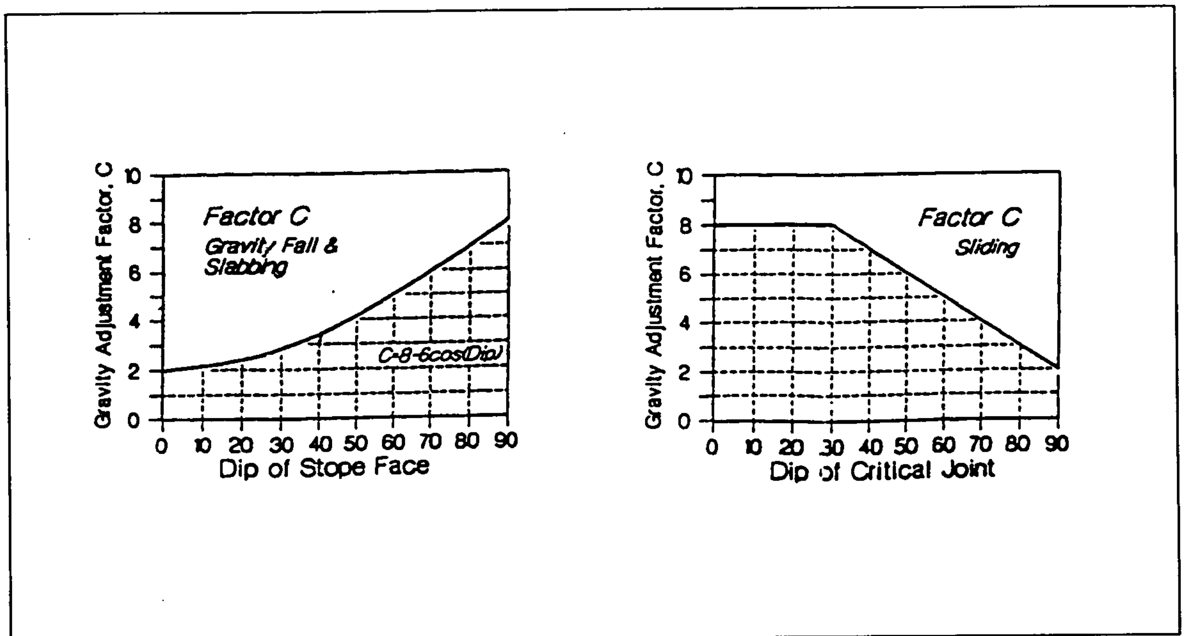


Figure 5.4 Chart for determining C factor (after Potvin, 1988)

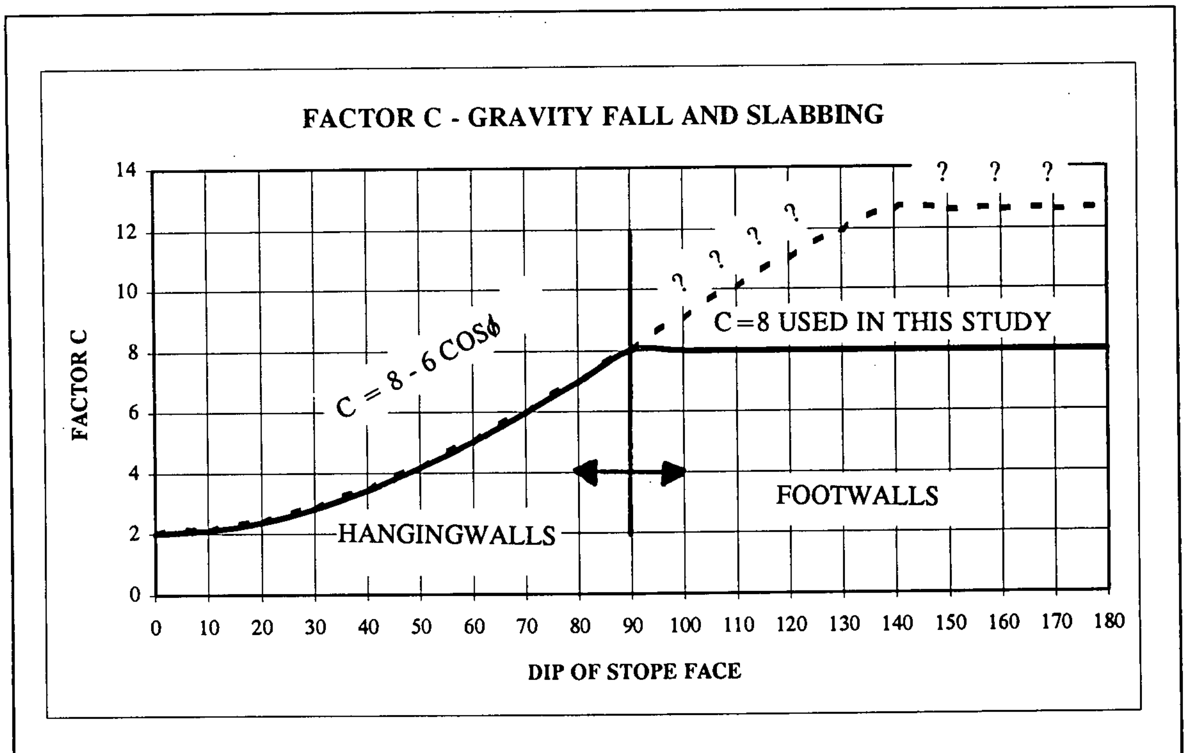


Figure 5.5 Schematic illustrating alternate interpretation of C factor for hangingwalls and footwalls

Field measurements support the above comments. Histograms of hangingwall and footwall ELOS are presented in Figure 5.6. The plot indicates that ELOS values greater than 1m are uncommon for footwalls, whereas hangingwalls have a much wider range of ELOS values. Figure 5.7 shows hangingwall and footwall ELOS plotted against stope dip. This plot clearly shows the adverse influence of dip on hangingwall stability. With regard to footwalls however, the graph shows that higher ELOS values were confined to stopes that dipped greater than 80° (i.e. approaching a vertical wall). Footwall stopes that dipped less than 80° all had ELOS less than 1.0m with the majority being less than 0.5m. Note that in all cases parallel structure was the critical structure with regard to stability.

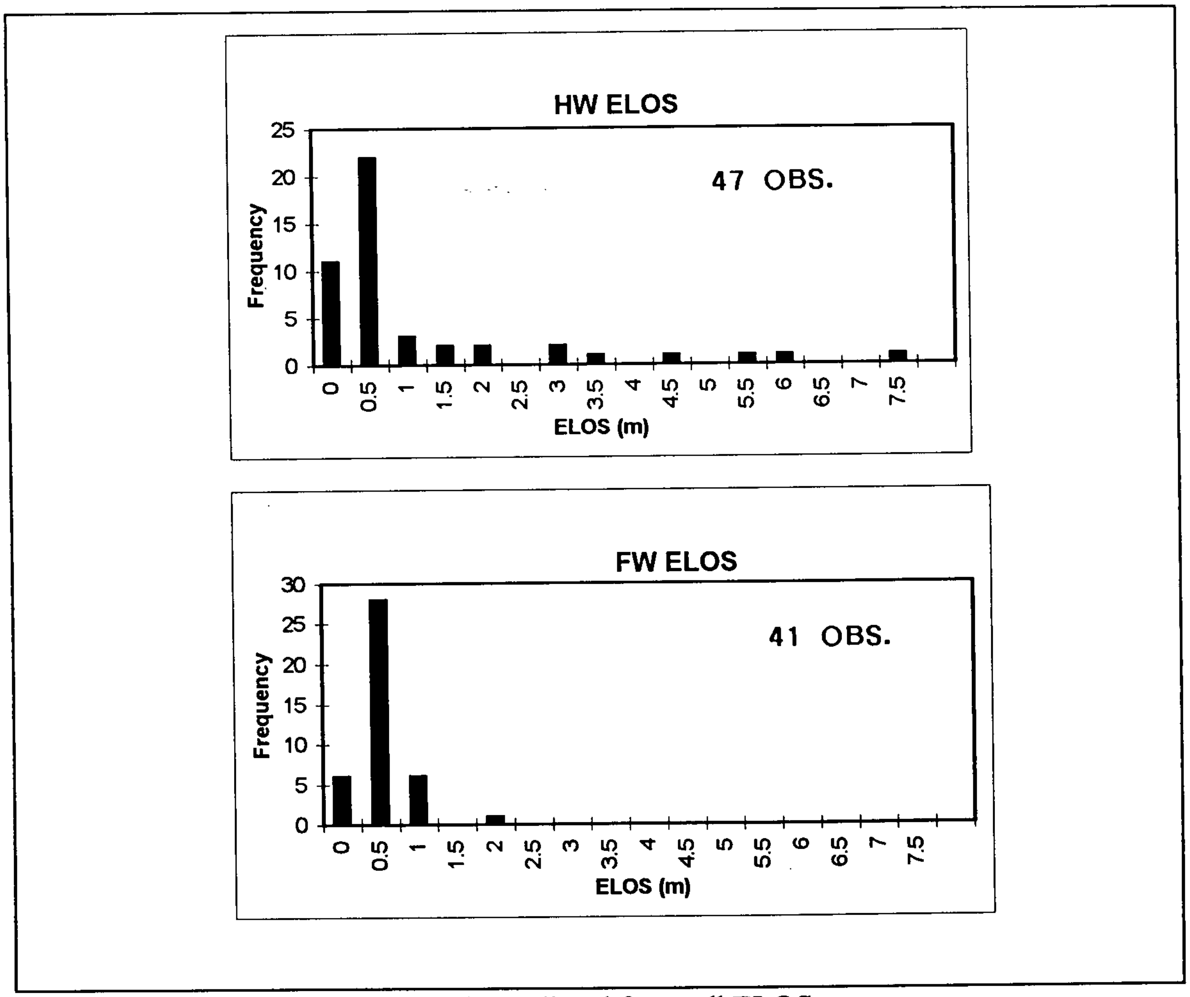


Figure 5.6 Histograms of hangingwall and footwall ELOS

47 OBS.

41 OBS.

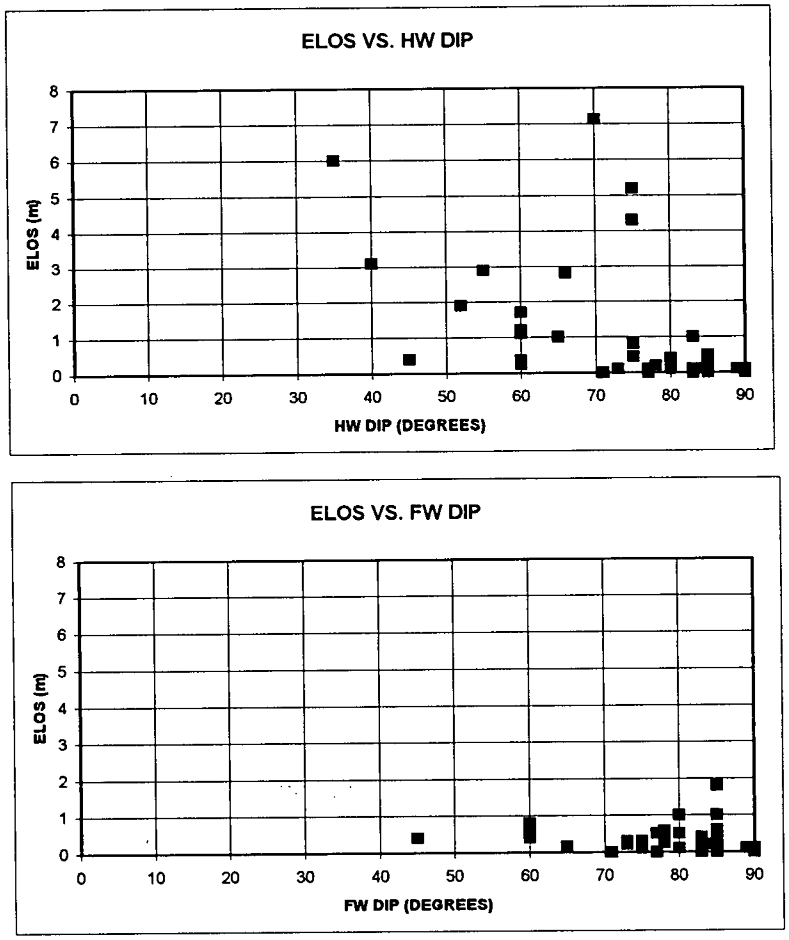


Figure 5.7 Hangingwall and footwall ELOS plotted against stope wall dip (Parallel structure critical with regard to stability)

The above supports the idea that footwalls are inherently more stable than hangingwalls. It also supports the idea that hangingwall stability decreases with decreasing stope dip while footwall stability decreases with increasing stope dip.

For this study the buckling/slabbing graph was used to determine the C values for the hangingwalls (C=8-6cos∅) and a weighting of C=8 was assigned to all the footwalls (which is the approach used with the Mathews Method). More work is required to better define the C factor for footwalls.

5.1.3 Presentation of Results

Figure 5.8 is a plot of the Modified Stability Graph (after Nickson, 1992) with ELOS values from the CMS database overlain.

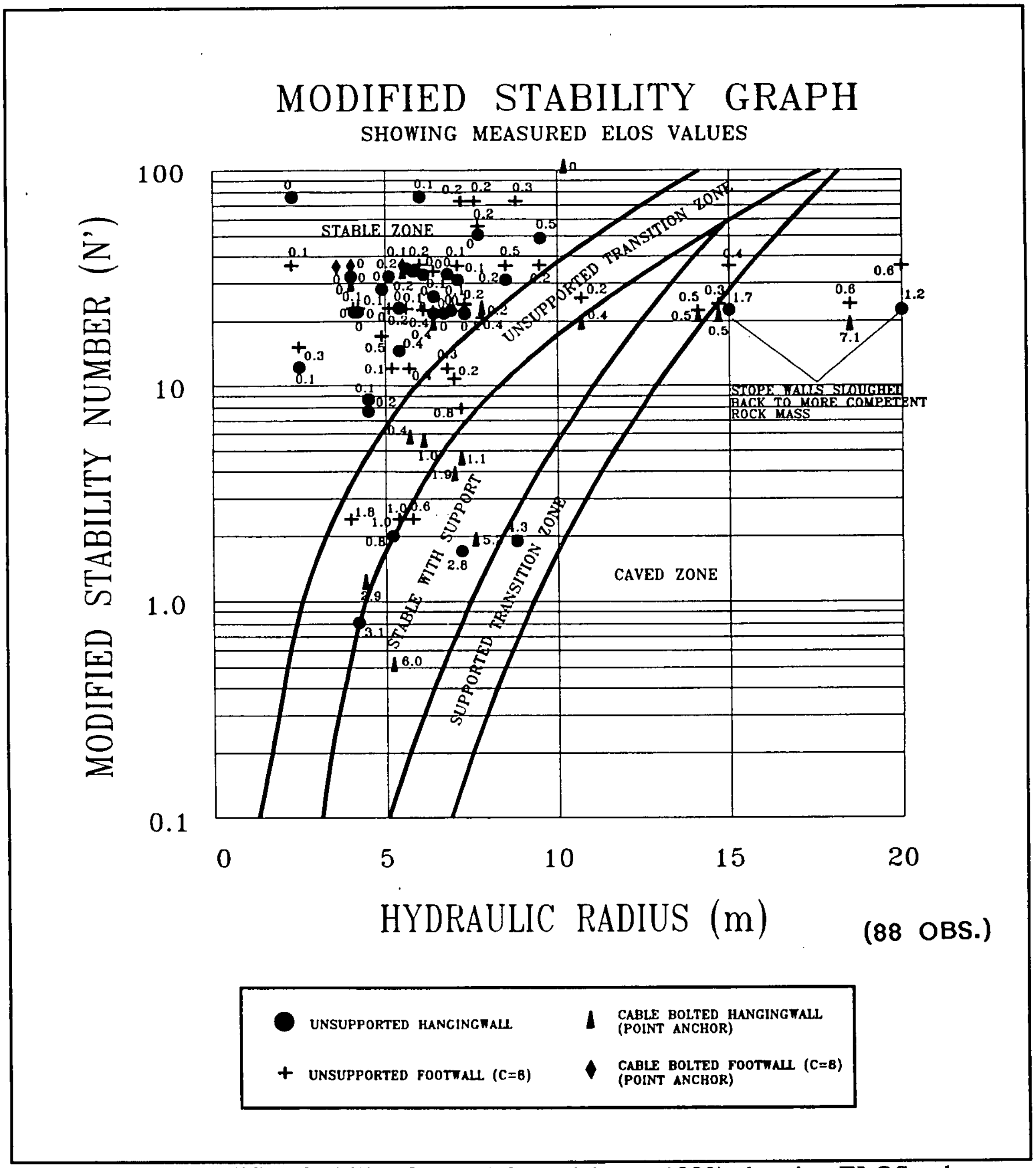


Figure 5.8 Modified Stability Graph (after Nickson, 1992) showing ELOS values from the CMS database

Figures 5.9 and 5.10 are similar plots showing the database points overlain on the Mathews Method Graph (Stewart and Forsyth, 1994) and the graph presented by Scoble and Moss (1994), respectively.

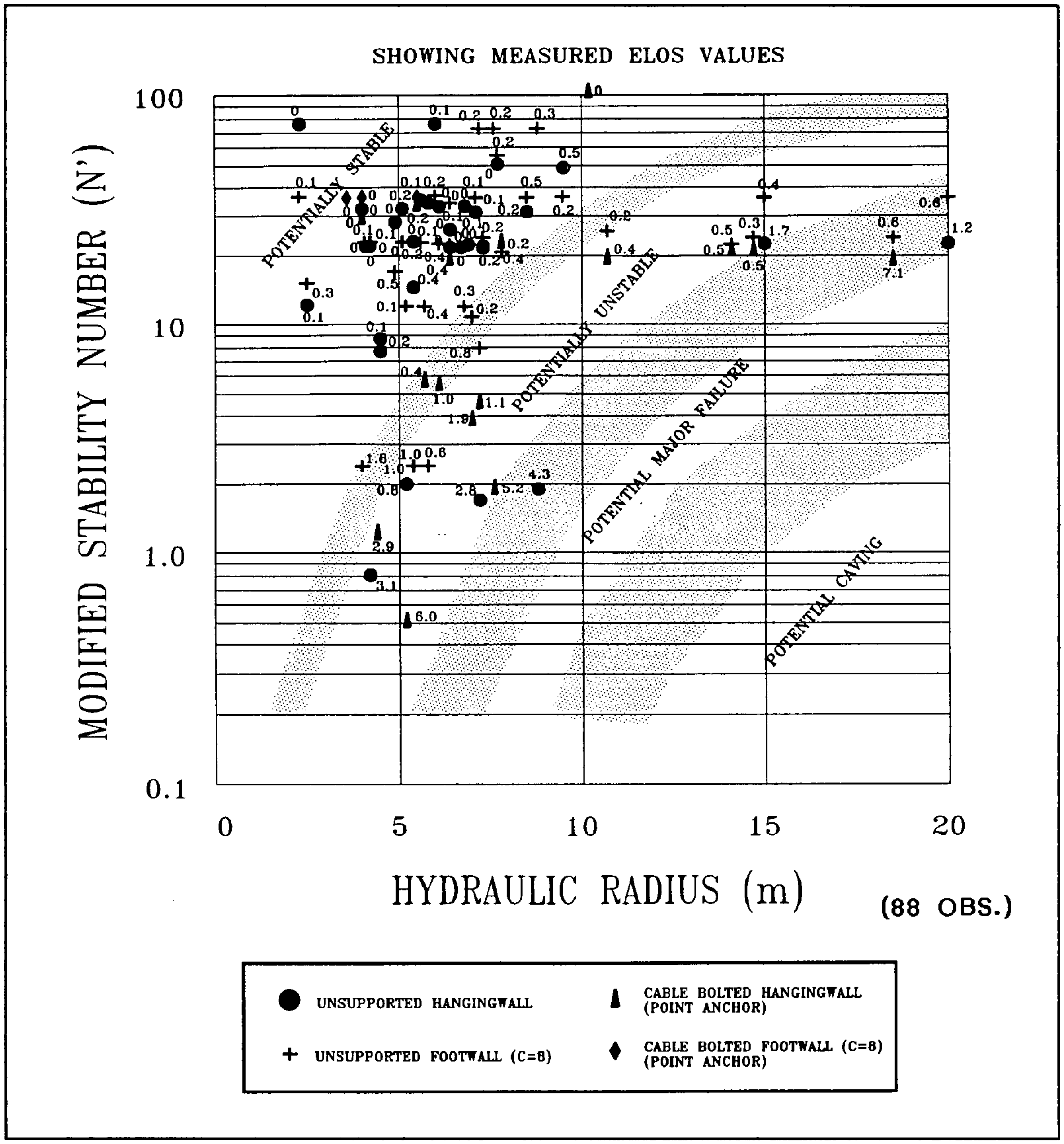


Figure 5.9 Mathews Stability Graph (after Stewart and Forsyth, 1994) showing ELOS values from the CMS database

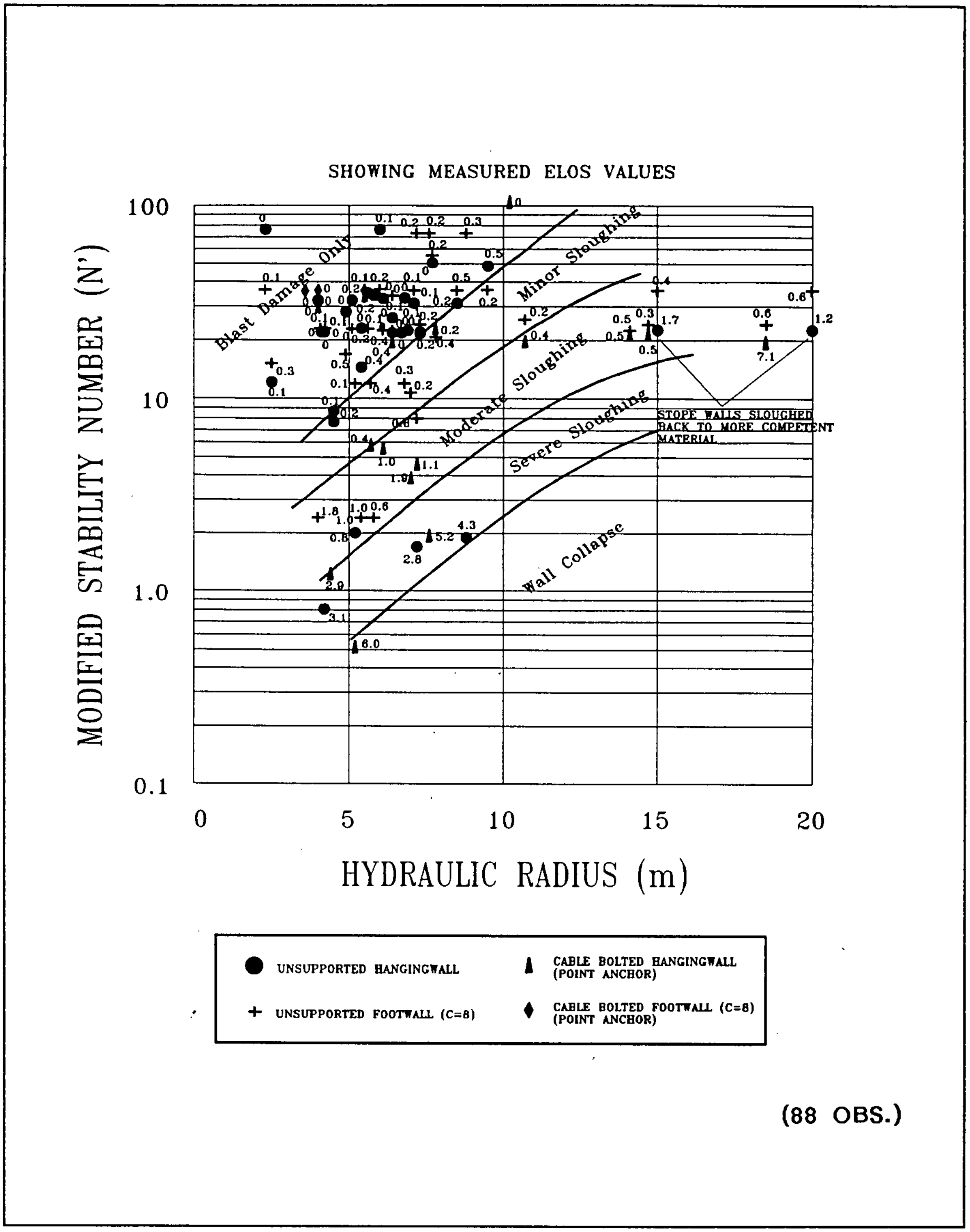


Figure 5.10 Scoble and Moss Stability Graph (1994) showing ELOS values from the CMS database

Technically it is not correct to plot the points on the Mathews Method Graph since the design zones are based on the Stability Number (N) not the Modified Stability Number (N'). However, it was done out of interest and realizing that the differences in the two N values are not large. In essence if the true Mathews Method Stability Number (N) had been used the data points would be shifted up slightly on the graph.

Referring to the figures, the data plots surprisingly well on all the graphs providing some quantifiable meaning to the design zones shown.

Design zones based on the measured ELOS values are proposed in Figure 5.11. The design zones were developed based on engineering judgement and using the existing stability lines as a guide. Note that the design zones apply to unsupported surfaces. There is not enough data to quantify the effects of the sub-level cablebolt support, however, the graph does provide some evidence that cable support may be ineffective for controlling unplanned dilution in rockmasses with N' < 6.

5.2 DEVELOPMENT OF A STABILITY GRAPH BASED ON RMR

5.2.1 General

It was decided to develop a stability graph based on RMR for the following reasons:

- some practitioners prefer using RMR over the Q system;
- converting RMR to Q using formulas (i.e. RMR = 9 ln Q + 44) can give erroneous results;
- RMR is easier to grasp for people not familiar with rockmass classification. This allows easier implementation of rock mechanics concepts into the production environment.

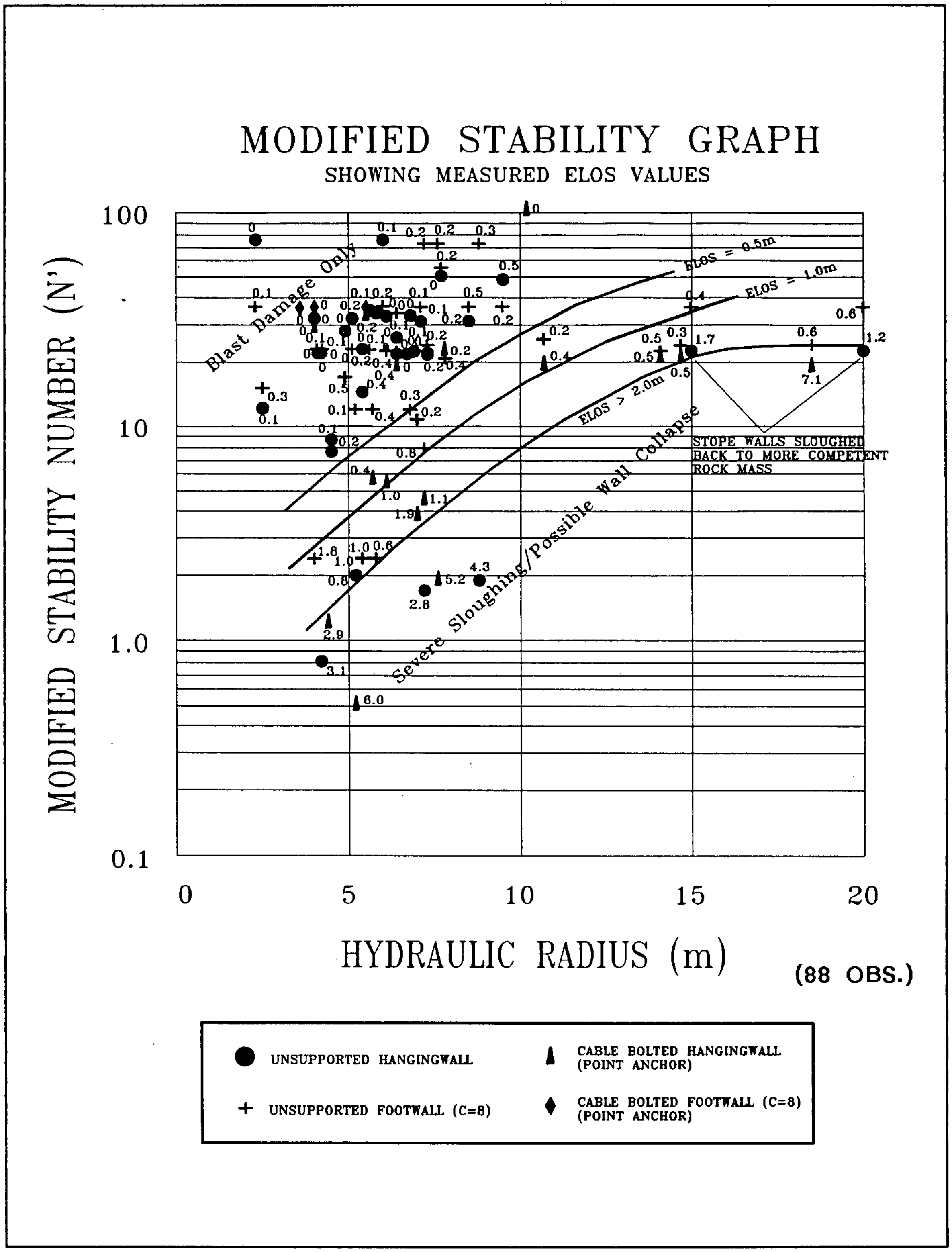


Figure 5.11 Proposed ELOS design zones - N' vs. HR

5.2.2 Adjustments for Stress, Structure, and Gravity

In order to develop an RMR based stability number that is roughly equivalent to the Modified Stability Number (N') adjustments for stress, structure, and gravity are required.

It can be seen from the discussion in Section 5.1.2 that the stress factor (A) and the joint orientation factor (B) are constants, with regards to this study. The gravity factor (C) is the only variable. Based on this, it can be appreciated that only a gravity factor is required to develop an RMR based stability number that is approximately equivalent to N'.

Initially, development of the RMR gravity factor involved using trial and error to get the data points to plot in a spatially similar manner to the data points on the Modified Stability Graph. Once a rough correction factor had been developed, corrections published in literature were examined to see if there were any similarities. Surprisingly, a correction published by Carter et.al. (1990) as part of an empirical method to predict crown pillar stability, was very close. The correction was to account for the intrinsically less stable nature of inclined stope surfaces which have paralleling structure (i.e. foliation planes). The correction is presented below in equation form and shown graphically in Figure 5.12.

$$\text{Corr.} = 1 - 0.4 \cos \varnothing \qquad \text{(Eq. 5.1)}$$

where :

Corr. = gravity adjustment

$\varnothing$ = hangingwall dip

With regard to this study, the adjusted RMR (RMR') was calculated as follows:

$$\text{Hangingwalls: } RMR' = RMR * \text{Corr.} \qquad \text{(Eq. 5.2)}$$

$$\text{Footwalls: } RMR' = RMR \text{ (no adjustment)} \qquad \text{(Eq. 5.3)}$$

Note that no adjustment is made to the footwall RMR. This is in line with making C = 8 when calculating the Modified Stability Number (N') for footwalls, since at a dip of 90°, Corr. = 1 and RMR' = RMR for both the hangingwall and footwall.

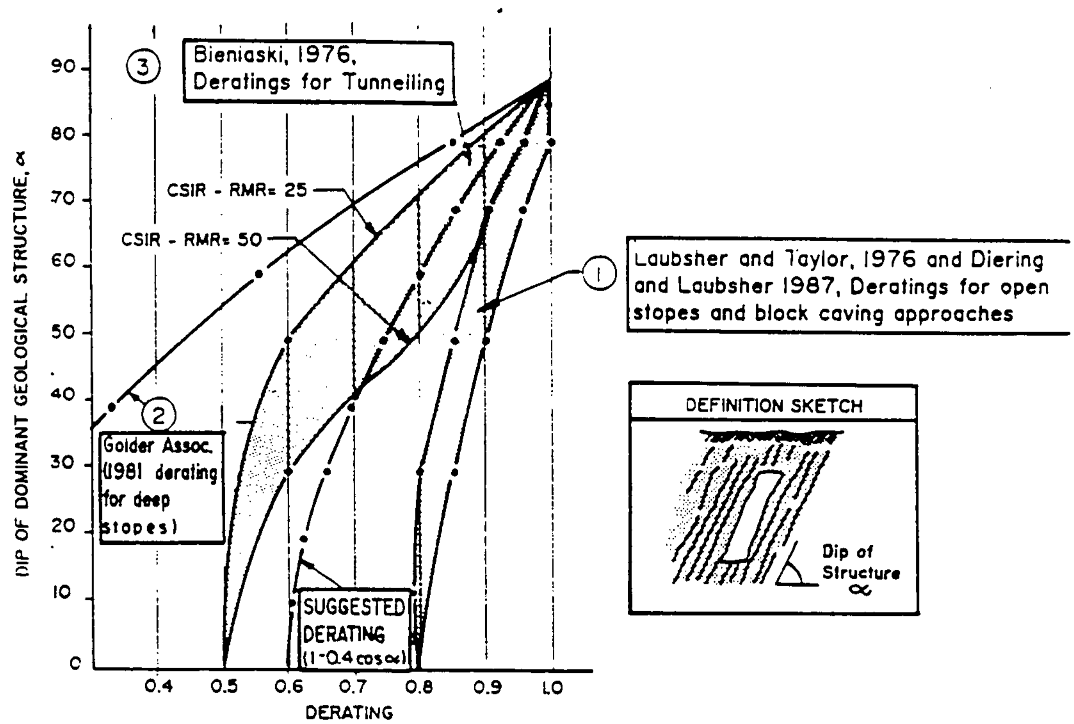


Figure 5.12 RMR gravity adjustment factor (after Carter et.al., 1990)

The relationship between RMR' and N' is shown in Figure 5.13. A strong correlation exists between the two parameters.

Given the assumptions made to arrive at RMR' it follows that the resulting stability graph will only be applicable for hangingwalls and footwalls that are either in relaxation or in a low stress environment, and, the critical joint set with regard to stability is parallel to the wall surface.

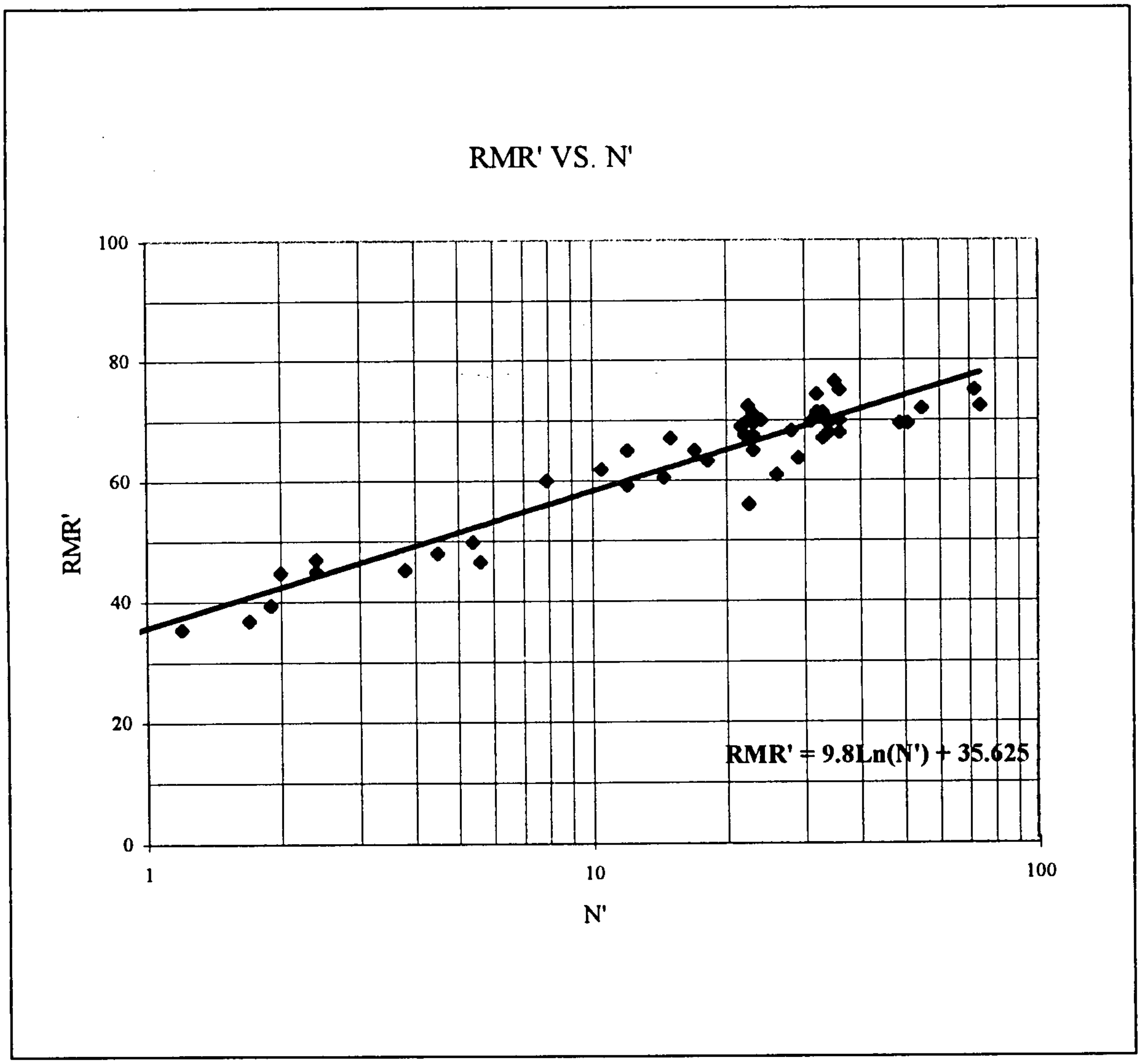


Figure 5.13 Relationship between RMR' and N'

5.2.3 Presentation of Results

Figure 5.14 shows how the ELOS data plots when the RMR values are not adjusted for hangingwall dip. The data points are quite tightly spaced and trends in the data are not apparent. Figure 5.15 shows the data after applying the adjustment to the RMR values. The data is more spread out and trends in the data are much more apparent.

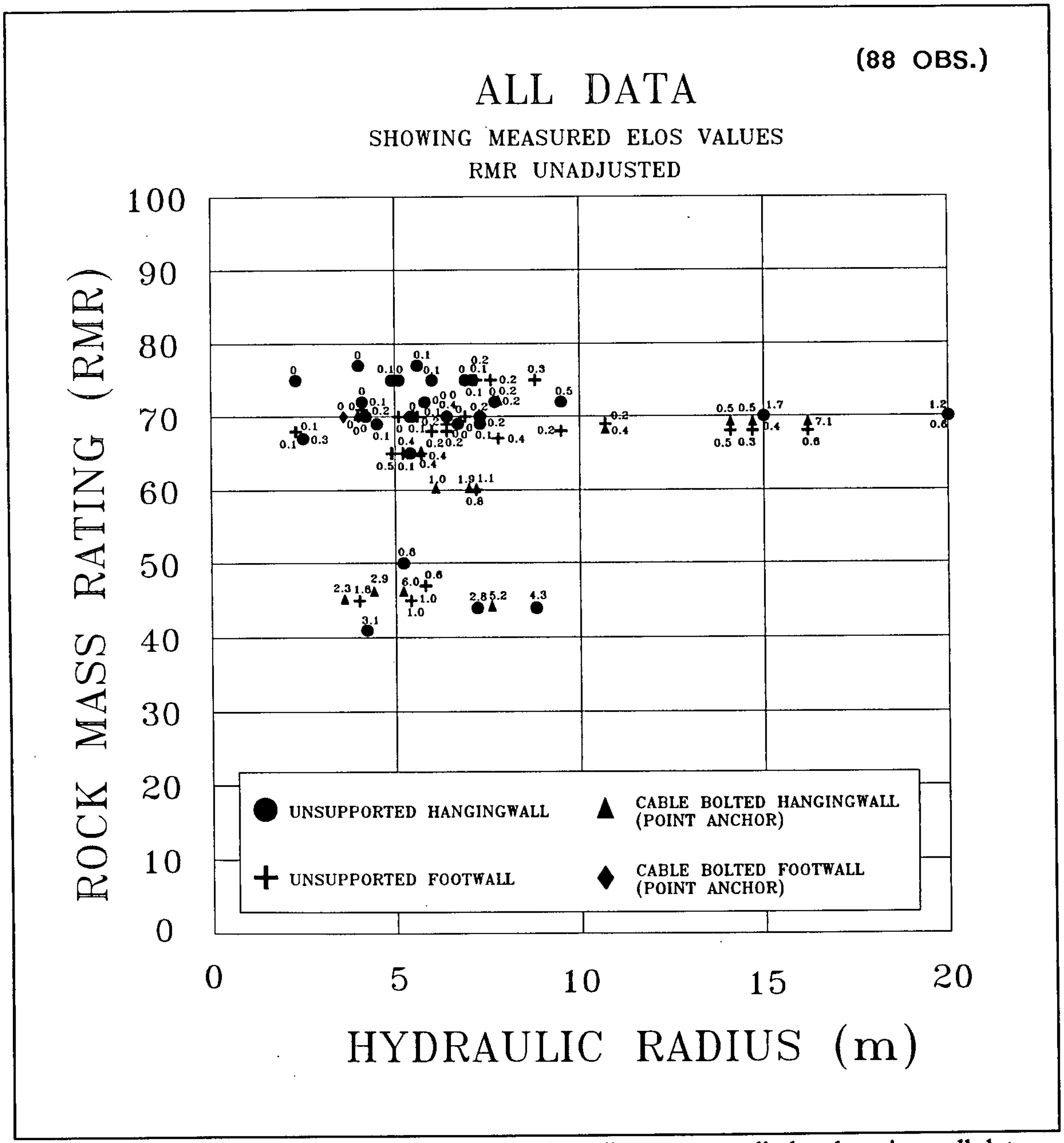


Figure 5.14 RMR Stability Graph - no gravity adjustment applied to hangingwall data

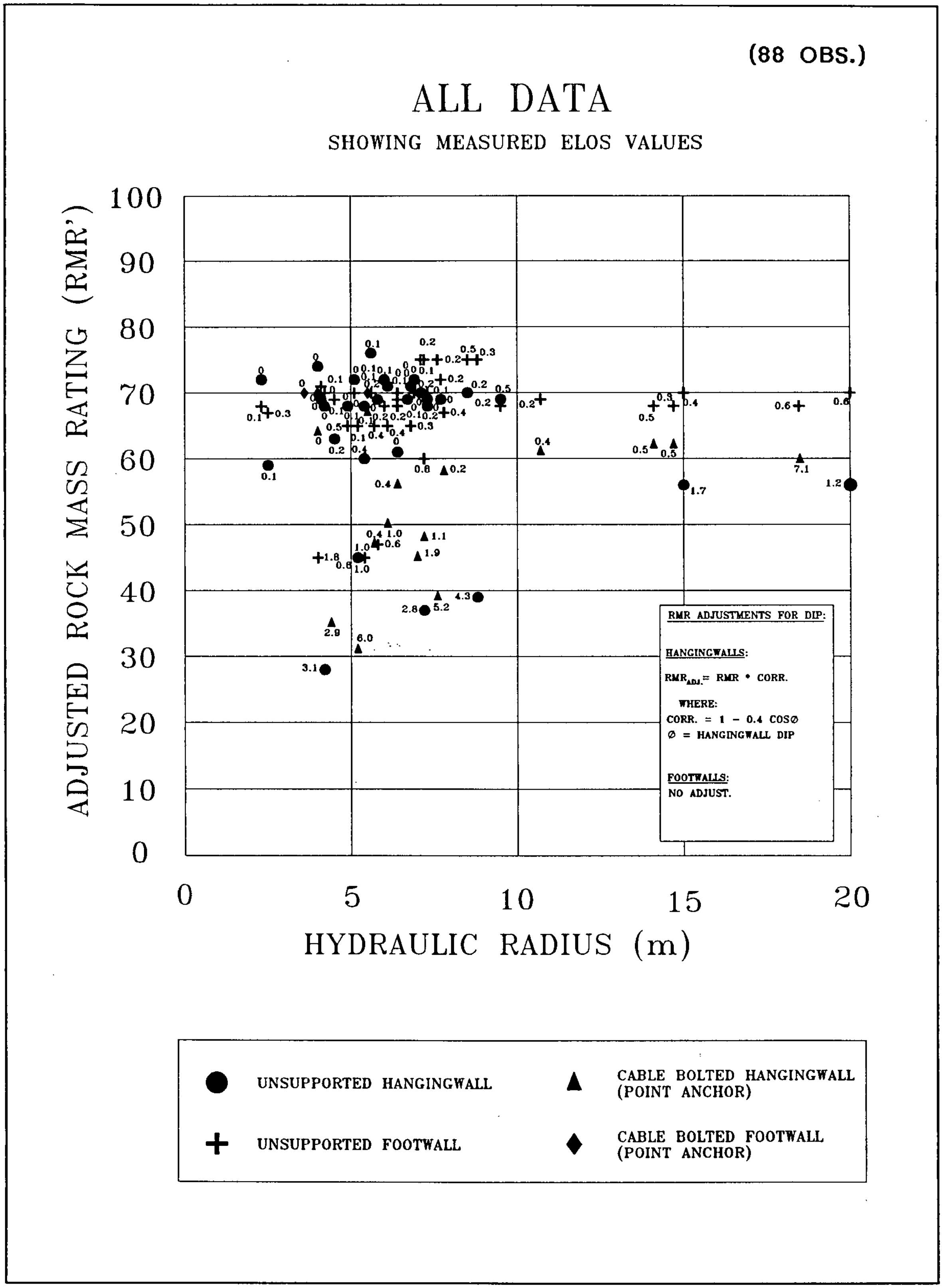


Figure 5.15 RMR Stability Graph - gravity adjustment applied to hangingwall data

Out of interest, a comparison was made with the Dilution Approach (Pakalnis, 1993) which uses RMR to characterize rock conditions. The graphs associated with the Dilution Approach are presented in Figure 2.2, Chapter 2. Referring to Figure 2.2, the dilution lines were converted to ELOS lines by assuming the dilution lines apply to 8m to 15m wide stopes (Pakalnis, 1993). For example, a 10% dilution line was assumed to approximate an ELOS line of 0.8m to 1.5m (avg. 1.2m). Figures 5.16 and 5.17 show the ELOS lines for "isolated stopes" and "rib stopes" respectively. Neither of the two sets of ELOS lines show strong correlation with the database points. The "rib stope" ELOS lines come closest to defining reasonable design zones. A strong correlation was not expected given that the dilution charts presented in Figure 2.2 use an unadjusted RMR and were developed based on data from only one mine.

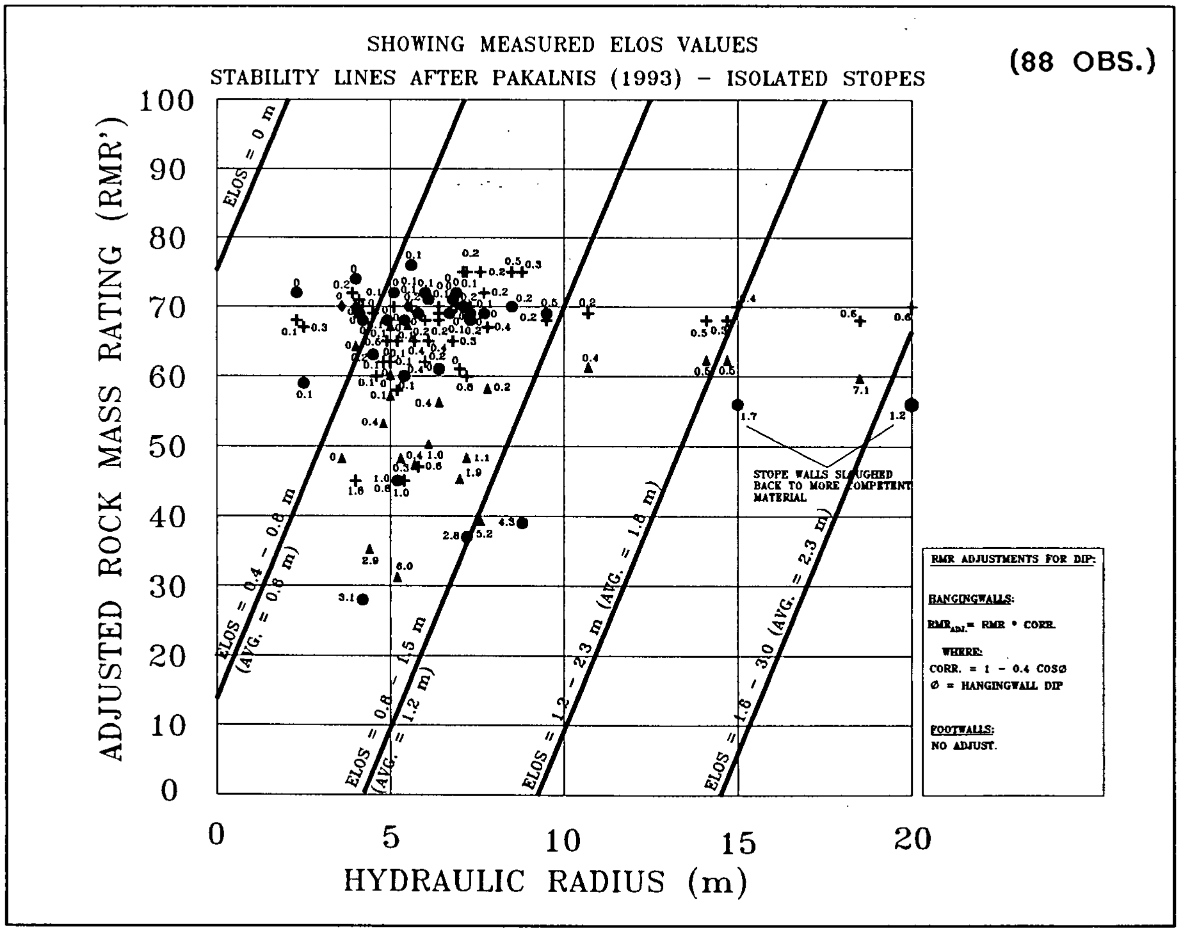


Figure 5.16 ELOS design zones determined from "Isolated Stopes" dilution design method (after Pakalnis, 1993)

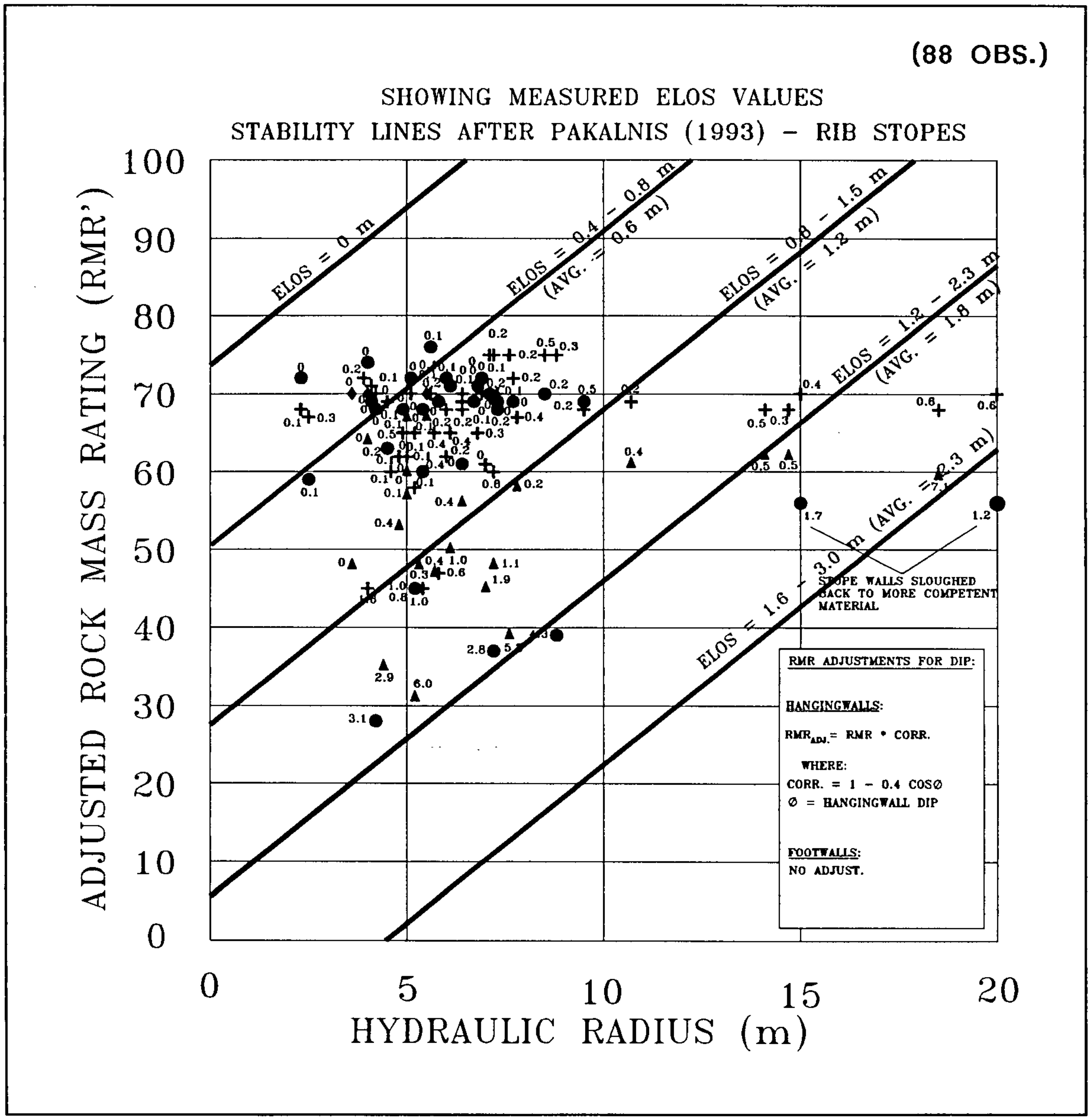


Figure 5.17 ELOS design zones determined from "Rib Stopes" dilution design method (after Pakalnis,1993)

Design zones based on engineering judgment are proposed in Figure 5.18. The design zones apply to unsupported surfaces. As mentioned in Section 5.2, there is not enough data to quantify the effects of sub-level cablebolt support, however, Figure 5.18 does provide some evidence that cable support may be ineffective for controlling unplanned dilution in rockmasses with RMR' < 55.

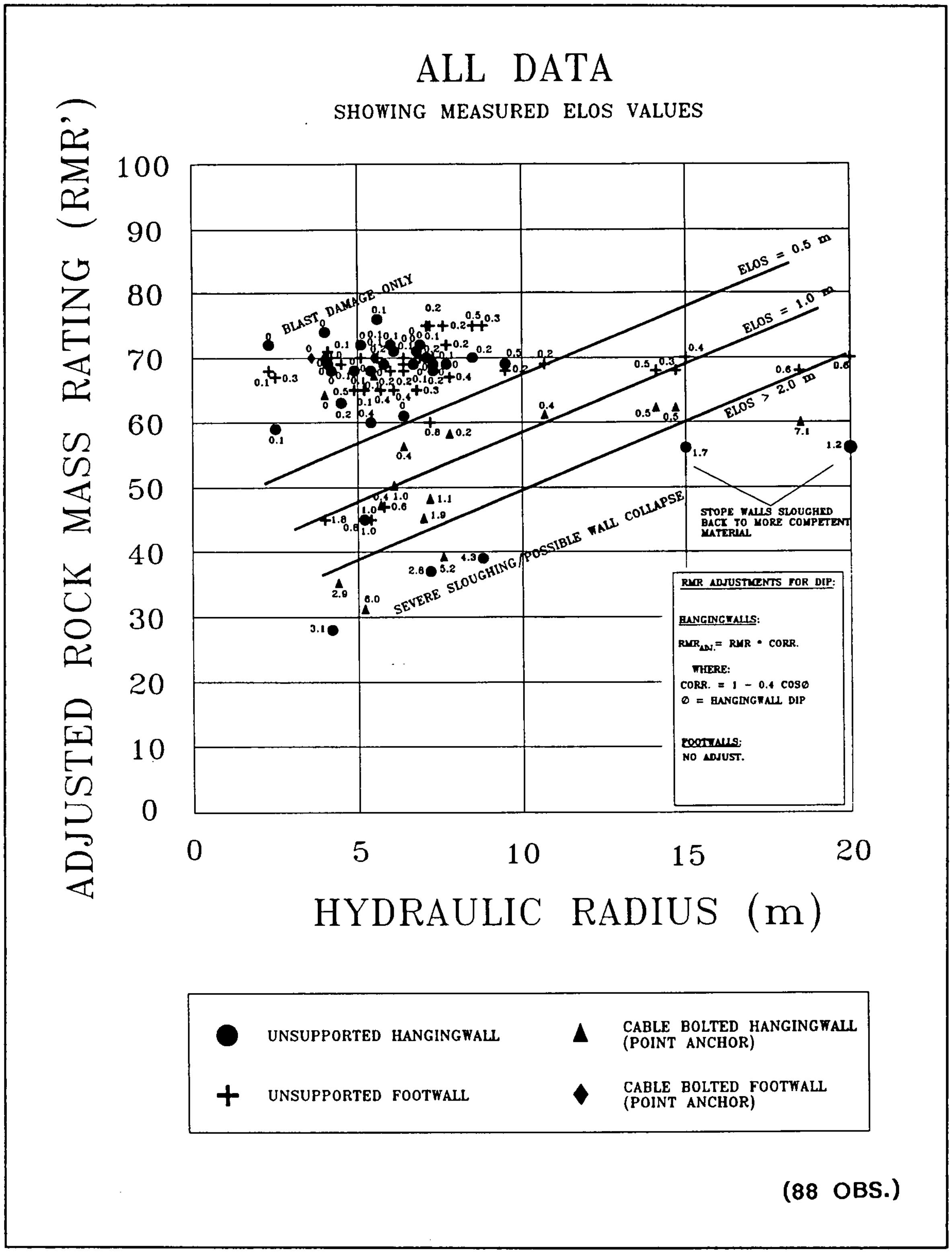


Figure 5.18 Proposed ELOS design zones - RMR' vs. HR

5.3 STATISTICAL DETERMINATION OF ELOS DESIGN ZONES

5.3.1 General

This section describes an attempt to statistically confirm the proposed ELOS design zones shown in Figures 5.11 and 5.18. The statistical analysis was performed by the Statisical Consulting and Research Lab (SCARL) at the University of British Columbia.

Due to the greater inherent stability of footwalls compared to hangingwalls and because there is some uncertainty in the appropriate gravity factor for footwalls, it was decided to use only hangingwalls and steeply dipping footwalls (>85°) in the statistical analysis. To eliminate any influence of stope support (i.e. cablebolts) only unsupported walls were used. This resulted in 46 points for the statistical analysis.

Initially it was hoped that four design zones could be determined: 1) ELOS $\leq$ 0.5m - Blast damage; 0.5m < ELOS $\leq$1.0m - Minor Sloughing; 1.0m < ELOS $\leq$ 2.0m - Moderate Sloughing; and ELOS > 2.0m - Severe Sloughing. However, given the limited data set it was only feasible to try and delineate the ELOS $\leq$ 0.5m zone (only 9 points had ELOS > 0.5m).

The statistical method used was logistic regression. Initially discriminant analysis was tried but the method requires that the predictor variables be normally distributed. This was not true for the data set and attempts at applying transformations were not successful. Logistic regression does not make the normality assumption. Logistic regression is typically used to predict the probability of an occurrence in situations with a dichotomous response. For example, in the case of a vote for some candidate, the probability of a vote or no vote is of interest. With regard to this study, logistic regression models were developed to discriminate between ELOS >0.5m and ELOS $\leq$ 0.5m. For details on logistic regression models refer to Hamilton (1992).

5.3.2 Development of Logistic Regression Models

Logistic regression models were fit to the data using both: N' and HR; and RMR' and HR, as predictor variables. A simple decision rule was used to derive a function of: N' and HR; and RMR' and HR, that discriminates between ELOS >0.5m and ELOS $\leq$ 0.5m. Two models were developed for each stability graph. The development of the models is described below.

If Y represents the response variable, Y=0 when ELOS is $\geq$ 0.5m, and Y = 1 when ELOS < 0.5m. The response is dependent on some subset of the predictor variables: N'(x_1) and HR(x_3); and RMR'(x_2) and HR(x_3). Four logisitic regression models were fit to the data:

Model 1	$\log (P / (1 - P)) = b_1x_1 + b_3x_3$	(Eq. 5.4)
Model 2	$\log (P / (1 - P)) = b_0 + b_1x_1 + b_3x_3$ (constant term)	(Eq. 5.5)
Model 3	$\log (P / (1 - P)) = b_2x_2 + b_3x_3$	(Eq. 5.6)
Model 4	$\log (P / (1 - P)) = b_0 + b_2x_2 + b_3x_3$ (constant term)	(Eq. 5.7)

Where: P = the predicted probability that Y=1

By using a simple decision rule: Y = 1 (ELOS < 0.5m) when P $\geq$ 0.5; and Y = 0 (ELOS $\geq$ 0.5m) when P < 0.5, a linear function of the predictor variables can be determined that discriminates between ELOS greater or less than 0.5m. Furthermore when P=0.5, log (P / (1 - P)) is equal to zero, therefore, the models can be solved for N' (Models 1 and 2) or RMR'(Models 3 and 4) and written in terms of HR.

The solved logistic regression models are presented in equation form below:

Model 1	N' = 1.86 (HR)	(Eq. 5.8)
Model 2	N' = 4.25 (HR) - 15.89	(Eq. 5.9)
Model 3	RMR' = 6.57 (HR)	(Eq. 5.10)
Model 4	RMR' = 1.59 (HR) + 39.01	(Eq. 5.11)

5.3.3 Discussion of Results

The logistic regression models are plotted on Figures 5.19 (models 1 and 2) and 5.20 (models 3 and 4). Note that all the database points are shown on the stability graphs, not just the points used in the statistical analysis. The proposed ELOS design zones from Figures 5.11 and 5.18 are also shown on the respective stability graphs. Referring to Figure 5.19, over certain ranges of hydraulic radius, models 1 and 2 both show a good correlation with the proposed ELOS = 0.5m line taken from Figure 5.11. Combining the lower portion of model 1 and the upper portion of model 2 would produce a line that correlates almost exactly. Referring to Figure 5.20, model 4 shows a reasonable correlation with the proposed ELOS = 0.5m line taken from Figure 5.18.

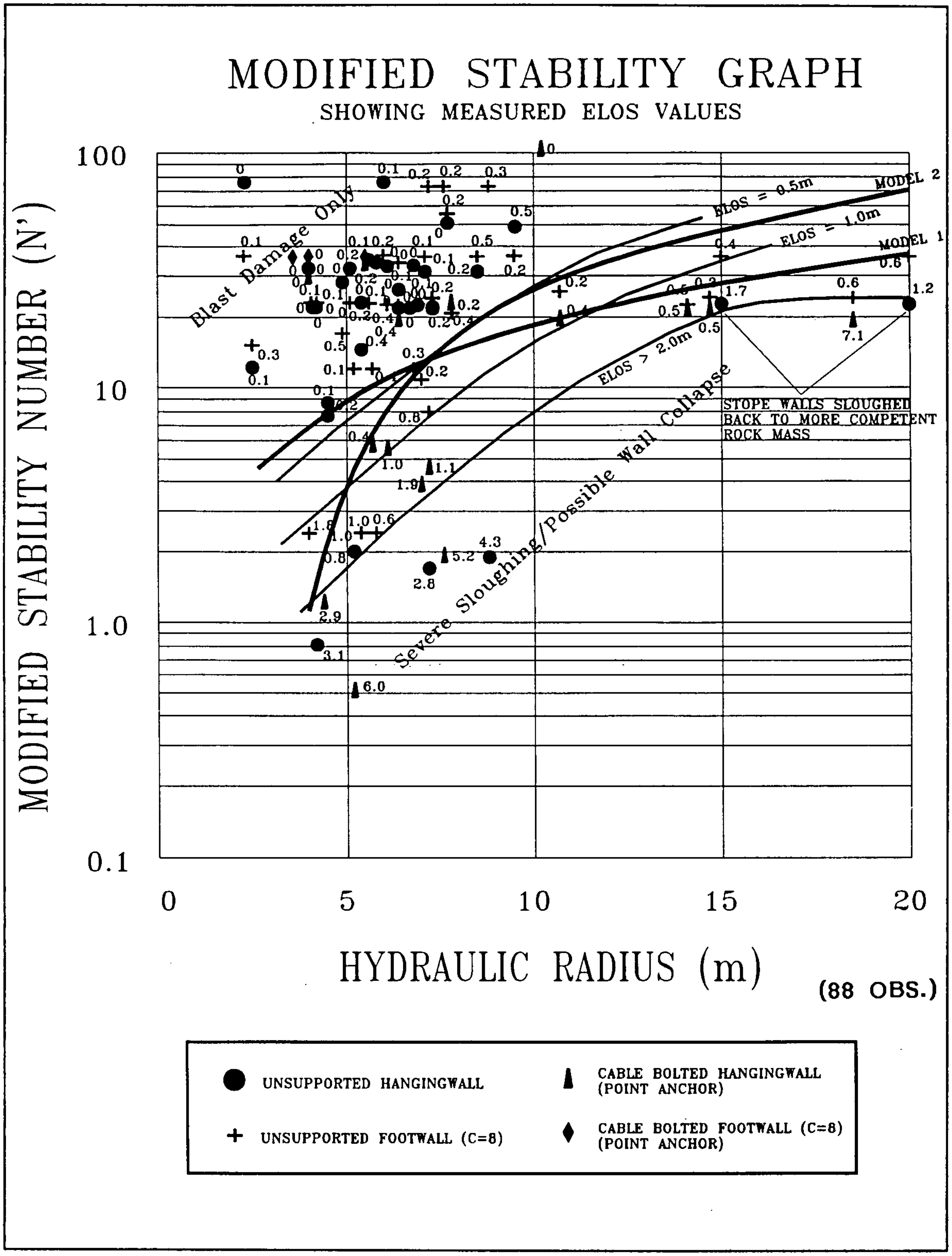


Figure 5.19 Logistic regression models for ELOS ≤ 0.5m - N' vs. HR

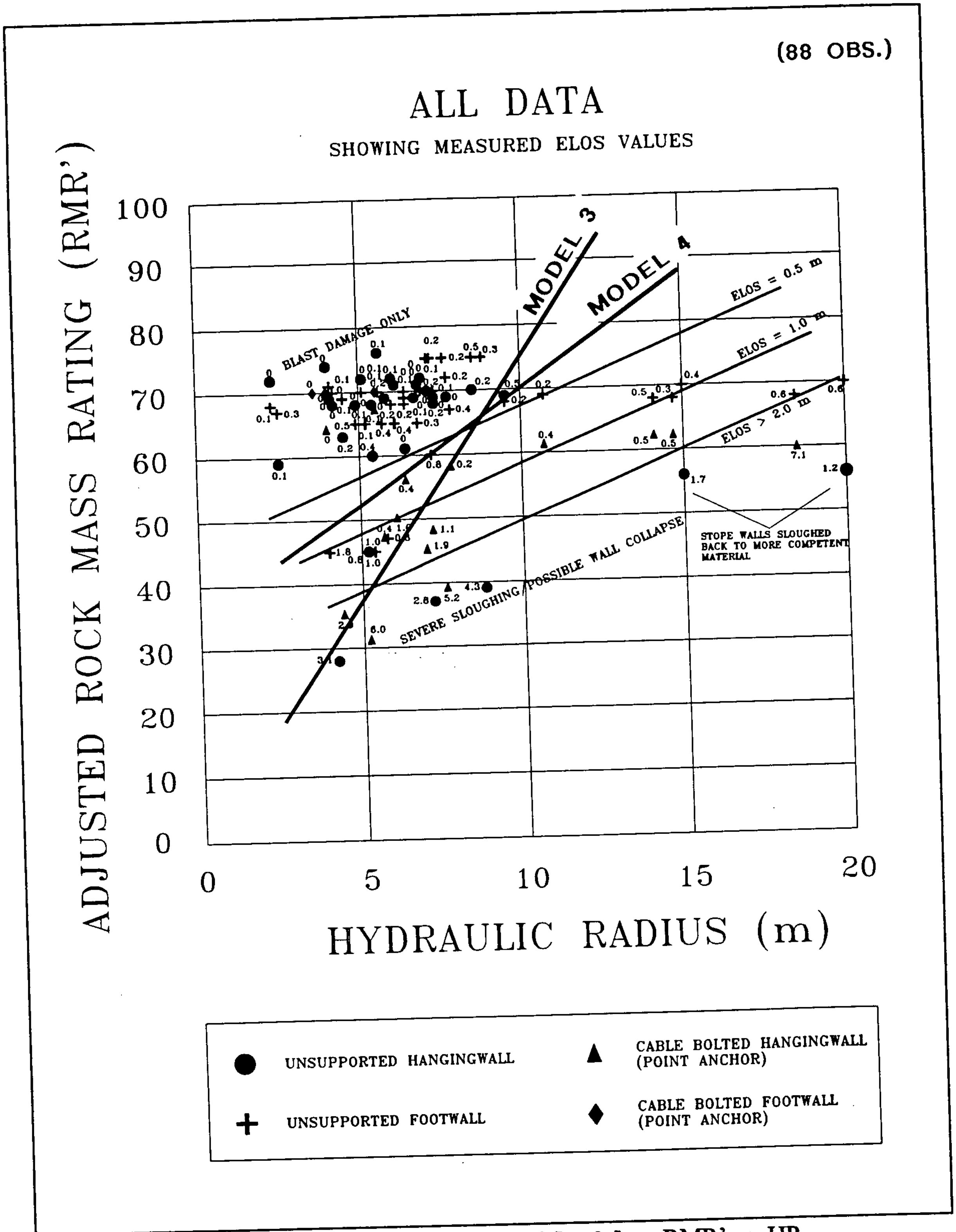


Figure 5.20 Logistic regression models for ELOS ≤ 0.5m - RMR' vs. HR

5.3.4 Conclusions From Statistical Analysis

The following conclusions can be drawn from this analysis:

- given enough data, logistic regression models can be used to successfully determine boundaries to ELOS design zones;
- models were developed which successfully discriminate between ELOS values greater and less than 0.5m;
- over certain ranges of hydraulic radius, models 1, 2 and 4 agreed reasonably well with the respective ELOS = 0.5m design boundary shown in Figures 5.11 and 5.18;
- there is not enough data in the database to statistically determine design zones for: ELOS = 1.0m; and ELOS > 2.0m;
- given the limited data set from which the logistic regression models were derived, the results from this statistical analysis do not warrant changing the proposed design zones shown in Figures 5.11 and 5.18.

5.4 DETERMINATION OF ELOS DESIGN ZONES USING NEURAL NETWORKS

5.4.1 General

Since the statistical analysis met with only limited success, neural networks were investigated as an alternate means of determining ELOS design zones. The following analysis forms a portion of a M.Sc. thesis being written by Logan Miller Tait (University of British Columbia, 1997).

Neural networks are used in situations where many data inputs make it difficult to accurately analyze the data. Neural networks can be "trained" to recognize patterns in data in much the same way as people are trained. Several factors can be analyzed at once to come up with an expert approximation. For a detailed description of neural networks refer to Beale and Jackson (1990).

The Braincel neural network program (V.2 - 1993) was used for this study. Braincel uses trial and error to find relationships between inputs and outputs and evaluates their relative importance on a scale from zero to one (or negative one) to predict an answer. For Braincel to work effectively there must be enough data available to analyze patterns and relations between numbers. More data would increase the prediction accuracy and diminish the effect of outlier data points (exceptions to the norm).

5.4.2 Description of Neural Network Analysis

For this analysis, a smaller database was created from the main CMS database. The neural network database only included the ELOS measurements which had data recorded for all the other database categories. This resulted in a database of 75 points (as opposed to 88 data points in the main CMS database). The neural network database was used for this particular analysis and a subsequent analysis in which relationships between ELOS and other database parameters were examined. The latter analysis is discussed in Chapter 7. Note that this data set differs from the one used in the statistical analysis in that it contains footwall data points which dip less than 85° as well as stope surfaces supported with cablebolts. This was required to ensure there was enough data to adequately train the neural network.

The way in which Braincel was used to determine ELOS design zones is described below:

1. Four neural networks were developed to predict ELOS: N' and HR - linear scale processing; N' and HR - log scale processing; RMR' and HR.- linear scale processing; RMR' and HR - log scale processing.
2. Initially, the database was divided into 60 data points for training data and 15 data points for test data.
3. The neural networks were trained to a seven percent error using the 60 data points.

4. The accuracy of the neural networks were then evaluated based on the ability to predict the measured ELOS values for the test data.

5. The more accurate of the N' and RMR' neural networks were then re-trained to a seven percent error using all 75 data points.

6. The neural networks were then used to predict ELOS values for a grid of 112 points on each of the respective stability graphs.

7. ELOS design zones were determined by contouring the grid of predicted ELOS values.

5.4.3 Discussion of Results

Accuracy of Neural Networks

Accuracy was evaluated based on how well the neural networks predicted the measured ELOS values from the test data. This was determined by plotting the results graphically and through calculation of the coefficient of determination (R^2). The R^2 factor is a measure of the correlation between the neural network ELOS predictions and the actual measured ELOS values. R^2 values can range between zero and one, the higher the value the greater the correlation (one is a perfect correlation).

The Braincel software allows for the option of linear or log scale processing of input factors. Both options were explored, resulting in four neural networks. The neural network using N' and linear scale processing could not reach the seven percent training error. This was not surprising given that N' is generally plotted on a log scale. Table 5.2 shown below presents the R^2 factors and the relative weightings given to the input factors, for the three successfully trained neural networks.

Table 5.2 - Accuracy of Neural Networks

Neural Network	N' or RMR' Weighting (%)	Hydraulic Radius Weighting (%)	Coefficient of Determination (R^2)
N', HR - Log	80	20	0.52
RMR', HR - Linear	57	43	0.13
RMR', HR - Log	59	41	0.76

Surprisingly, the neural network based on RMR' and computed using log scale processing was the most accurate. A possible explanation for why the log scale processing showed such a dramatic improvement over the linear scale processing is that a large portion of the RMR' data occurs between the narrow range of 60 and 75. Furthermore, most of the hydraulic radius values are between the narrow range of 4 and 7. Another interesting result is the low weighting given to hydraulic radius in the N' neural network. The reason for this is uncertain but perhaps it is related to the fact that most of the stope surfaces in the database are of small to moderate size, thus, having less of an influence on stability. Although this is also somewhat reflected in the RMR' neural network, hydraulic radius was given twice the weight that it was given in the N' neural net.

Determination of ELOS Design Zones

Based on the above analysis, the N'and RMR' neural nets were re-trained to seven percent error, using log scale processing, and all 75 data points. A grid of 112 points was then overlain on the N'and RMR' stability graphs and the respective neural networks used to predict ELOS values at the grid nodes. The ELOS values were then contoured to define ELOS design zones. Results for the N'and RMR' stability graphs are shown in Figures 5.21 and 5.22 respectively. Note that it is not recommended that these figures be used for design purposes. There is not enough data in the database to justify the number of ELOS design zones shown. Figures 5.23 and 5.24 compare the neural network results to the respective ELOS design zones proposed in Figures 5.11 and 5.18.

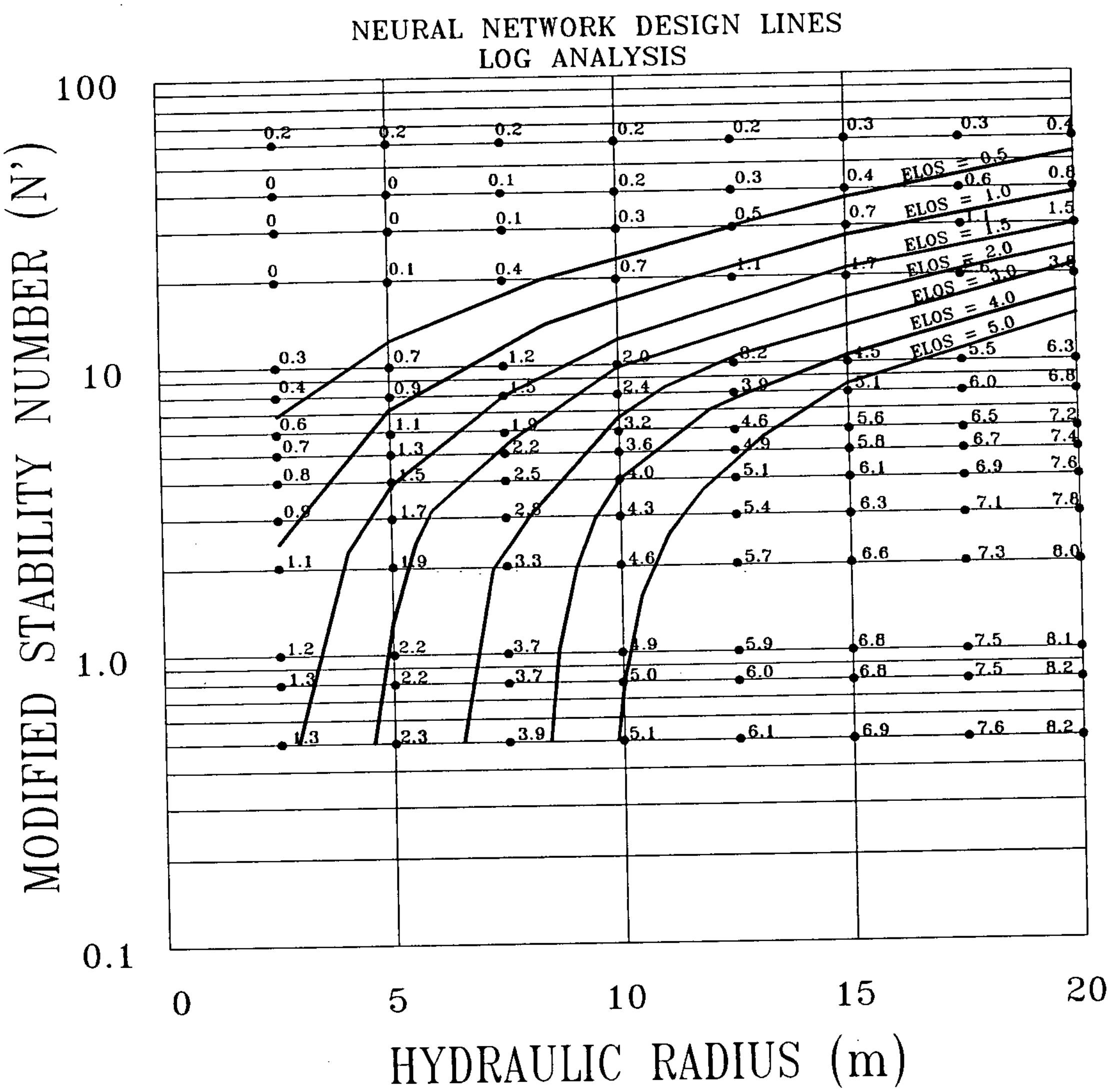
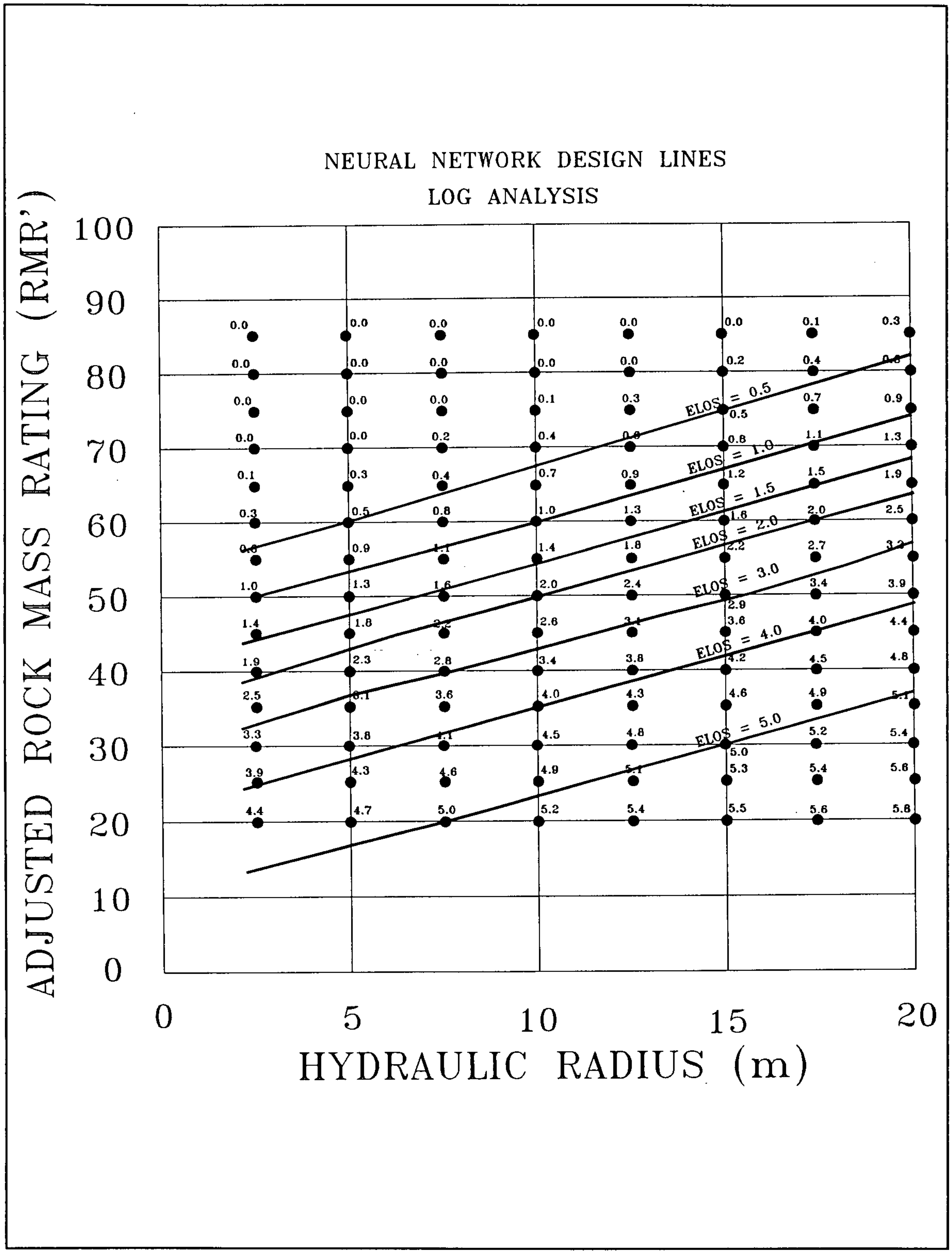
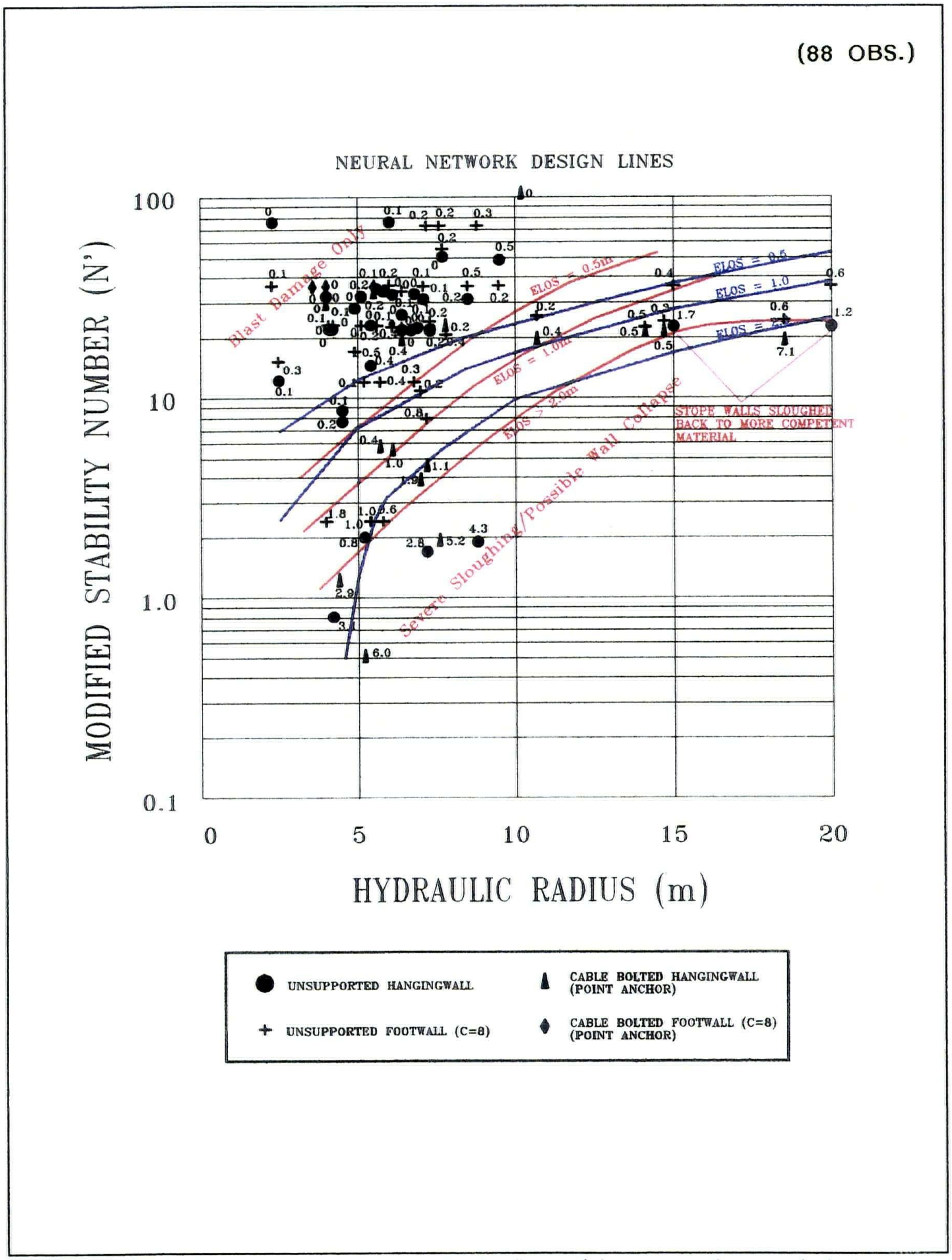


Figure 5.21 ELOS design zones determined from neural network analysis - N' vs. HR



5.22 ELOS design zones determined from neural network analysis - RMR' vs. HR



5.23 ELOS design zones from Figure 5.11 (red) overlain on neural network design zones (blue) - N' vs. HR

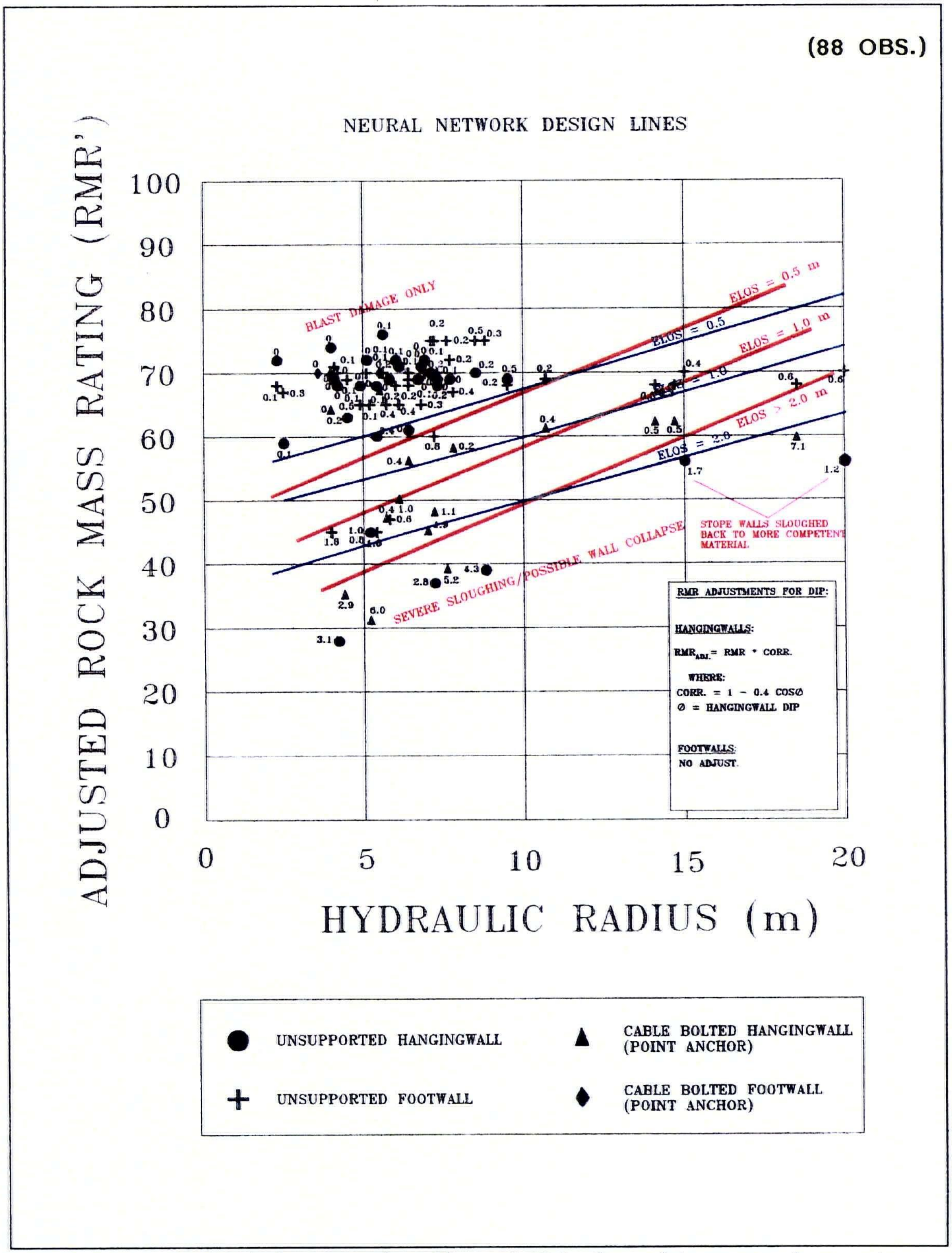


Figure 5.24 ELOS design zones from Figure 5.18 (red) overlain on neural network design zones (blue) - RMR' vs. HR

Referring to both Figures 5.23 and 5.24 the following observations can be made:

- the RMR' neural network results correlate well with the measured data and also show good correlation with the proposed design zones from Figure 5.18;

- the N' neural network results show a reasonable correlation with the measured data, although it appears that the both the ELOS = 0.5m and ELOS = 1.0m boundaries are plotting too high on the stability graph. The ELOS = 2.0m boundary correlates quite well with the proposed design zone from Figure 5.11;

- in general the ELOS boundaries determined using the neural networks have a flatter slope than the proposed design boundaries based on engineering judgement. This may be a function of having included the footwall data points in the neural network analysis. Note that the database contains some very large footwalls (HR = 14, 15, 18.5 and 20), dipping at 60-75°, that produced relatively low ELOS values. These points may be skewing the neural network results slightly. Note that based on the discussion in Section 5.12 and Figure 5.5, it can be argued that these points should possibly plot higher on the stability graphs.

5.4.4 Conclusions From Neural Network Analysis

The following conclusions can be drawn from the neural network analysis:

- this analysis showed that neural networks are a viable alternative to classical statistical methods for analysing trends in large data sets;

- neural networks were successfully developed that were able to delineate reasonable ELOS design zones;

- log processing of both the N' and RMR' data sets produced the most accurate neural networks;

- the neural network based on the RMR' data set was considerably more accurate than the neural network based on the N' data set ($R^2 = 0.76$ vs. $R^2 = 0.52$)

- the results of this analysis provide some theoretical justification for the proposed ELOS design zones shown in Figures 5.11 and 5.18. The results of this analysis do not warrant changing the design zones from those shown in the Figures.

5.5 ADDITIONAL CASE HISTORIES

As a further check on the proposed design zones, an addtional 10 CMS surveys were obtained from a mine that is not currently in the CMS database. From the 10 surveys, 14 ELOS measurements were obtained (5 supported hangingwalls and 9 unsupported footwalls). The following are brief notes concerning the mine:

- mining method is transverse open stoping with paste fill - primary and secondary stopes;
- the case histories obtained are primary stopes;
- HW cable bolt support consists of cables installed at sub-levels (fan of 3 cables every 2m);
- critical joint set with regard to stability parallels the HW and FW;
- blasthole diameter 4.5 in., blasthole offset approx. 0.6m..

The additional data points are presented in Table 5.3 below:

Table 5.3
Additional Case Histories (Not in Database)

Stope	Surface	Cables	Dip (Deg.)	HR	N'	RMR'	ELOS (m)
65301	HW	Yes	63	4.8	16	53	0.4
65281	FW	No	90	5.0	18	62	0
65261	HW	Yes	80	5.0	47	67	0
65261	FW	No	90-45	5.0	18	62	0
65241	FW	No	77	3.9	54	72	0.2
65221	FW	No	80	5.2	12	58	0.1
65222	FW	No	73	4.6	14	60	0.1
65201	HW	Yes	90	5.0	14	60	0
65201	FW	No	58	6.0	18	62	0.2
65181	FW	No	79	4.8	18	62	0.1
65161	HW	Yes	81	5.0	14	57	0.1
65161	FW	No	79	4.8	18	62	0.1
65141	HW	Yes	56	3.6	10	48	0
65141	FW	No	90	7.0	16	61	0

The additional case history data is shown plotted on Figures 5.25 and 5.26. The data shows no major discrepancies with the proposed design zones.

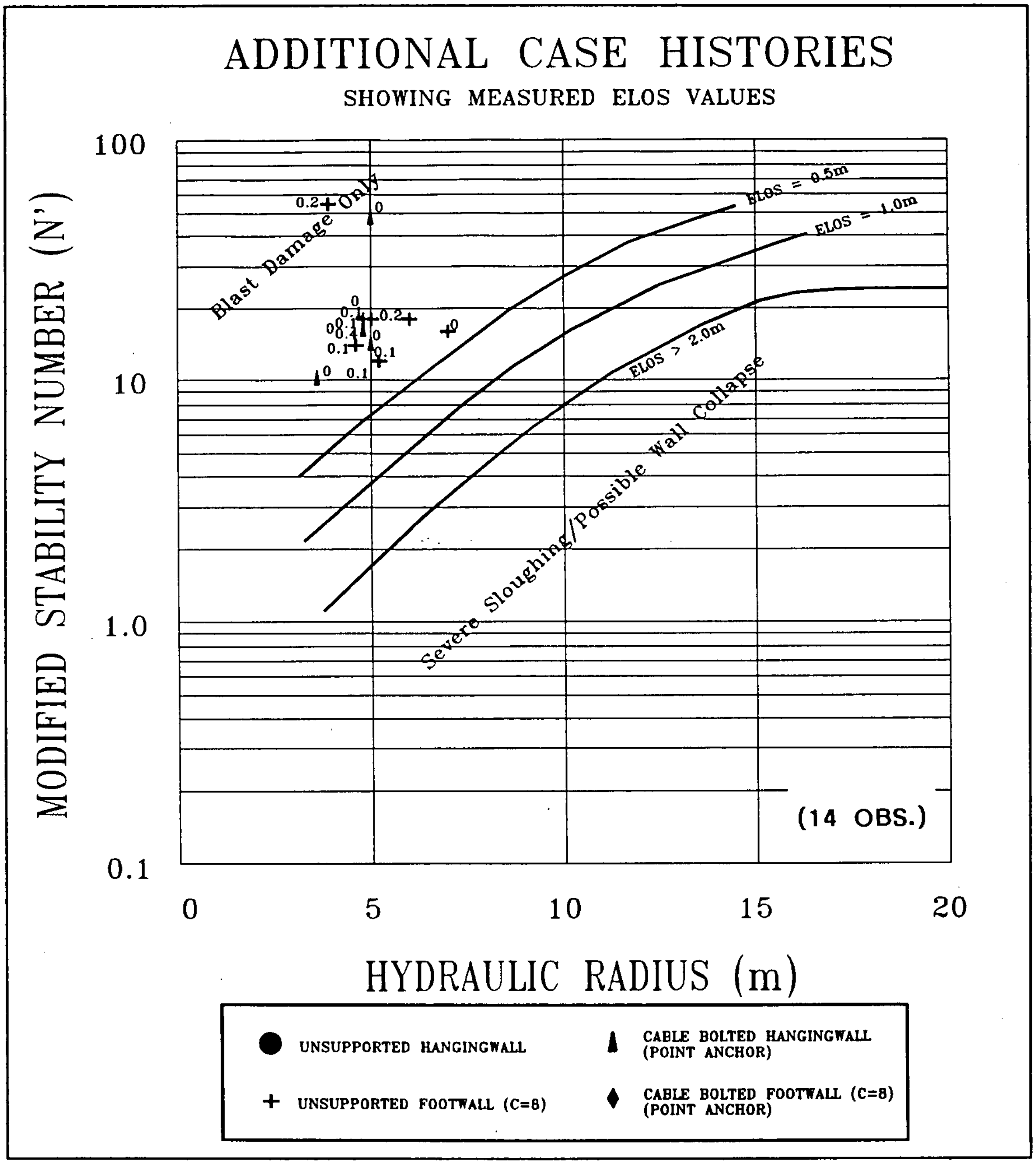


Figure 5.25 Additional case histories - N' vs. HR

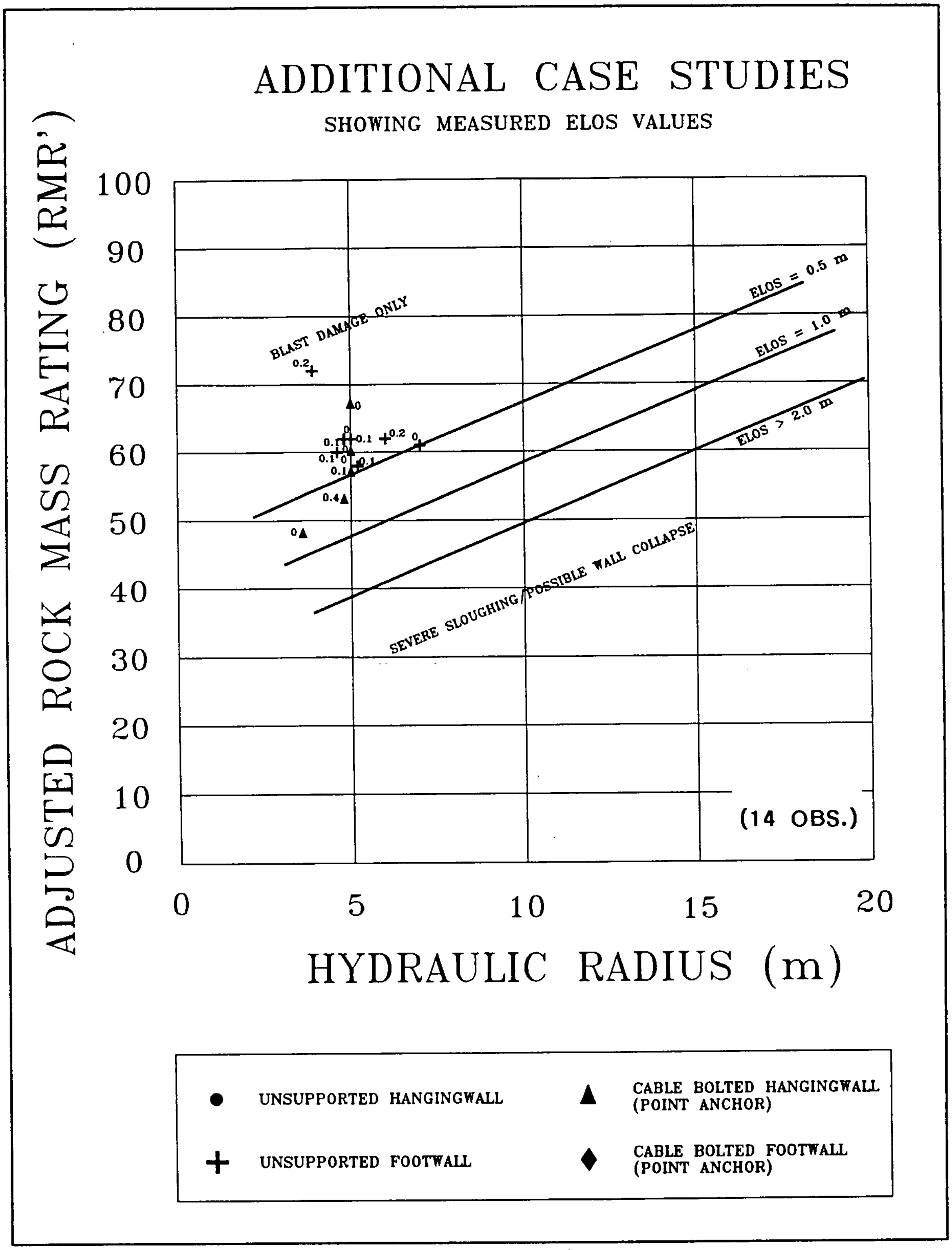


Figure 5.26 Additional case histories - RMR' vs. HR

5.6 SUMMARY

ELOS values have been incorporated onto the Modified Stability Graph (Potvin, 1988) and a new stability graph based on RMR and HR has been developed.

ELOS design zones have been proposed that are based on engineering judgement. Statistical methods, neural networks, and additional case histories were used to provide justification to the proposed design zones.

The ELOS design zones provide an empirical method of predicting the volume of overbreak/slough associated with a particular stope design. Unplanned dilution can be calculated based on the predicted volumes.

This empirical approach is only valid for unsupported hangingwalls and footwalls, in a relaxed or low stress state, and where the critical structure with regard to stability parallels the stope walls. More data is required to quantify the influence of sub-level cablebolt support (point anchor approach).

CHAPTER 6

A THEORETICAL EVALUATION OF THE ELOS PARAMETER

6.1 GENERAL

This chapter attempts to provide some theoretical justification for incorporating the ELOS parameter into stope design methodology. Numerical modelling was used to investigate the effect of: stress; stope size; and stope shape, on the formation of a destressed zone (zone of relaxation) in the hangingwall and footwall of open stopes, and the potential impacts with regard to stope wall stability.

Modelling results are compared to both the theoretical behaviour of stope walls and quantifiable measurements of overbreak/slough from the CMS database, and related to factors such as: geologic structure; rockmass quality; design of ground support; and prediction of unplanned dilution. This study does not address the impact of: drilling and blasting; stope wall undercutting; ground support; high stress; and time, on the stability of open stope hangingwalls and footwalls.

6.2 HANGINGWALL BEHAVIOR IN BEDDED AND FOLIATED ROCKS

In a large number of mines where open stoping is being used, the orebody is paralleled by a very persistent set of discontinuities. The discontinuities may represent bedding planes, foliation planes, or jointing. These parallel discontinuities are the critical set with regards to the stability of hangingwalls and footwalls and are the reason why hangingwall and footwall stability is so sensitive to undercutting.

The CMS database provides evidence that both hangingwalls and steeply dipping footwalls (>85°) are principal sources of unplanned dilution. The following discussion on hangingwall behavior is largely after Beer et al. (1983). It is assummed, that for the most part, the processes discussed below are also applicable to steeply dipping footwalls (>85°).

Consider an excavation being made in a pre-stressed rock. Assume $\sigma 1$ is the major principal virgin stress and is perpendicular to the hangingwall and footwall (based on stress measurements compiled by Arjang (1991) this is a reasonable assumption). Following creation of the excavation, the hangingwall and footwall will deform due to the release of stress at the free surface. The rock material will be completely destressed (relaxed) in the direction perpendicular to the hangingwall and footwall except at the abutments where a pressure arch forms. The destressed (relaxed) zone represents a zone of tensile stress *parallel* to the stope surface. Potvin (1988) states that since intact rock has a very low tensile strength and discontinuities have no strength in tension, tensile stress is not likely to build up in a rock mass medium. Instead, tensile stresses will open existing joints or induce new cracks through intact rock, creating a zone of relaxation. Inside this zone of relaxation individual rock blocks have more freedom to move because they are unconfined and thus become more sensitive to the action of gravity. Villaescusa (1995) states that high pre-mining stresses will result in greater offloading and increased relaxation. Within the zone of relaxation, gravitational forces and dynamic effects associated with blasting will induce tensile stresses in the hangingwall rock *normal* to the bedding or foliation. These tensions will cause the formation of micro-cracks along bedding or foliation planes. Also, because of the high shear stress near the abutments, slip on bedding or foliation planes may occur. This slip will cause the micro-cracks to open and initiate the formation of individual rock beams or plates. Depending on the thickness of the plates, frequency of cross joints, presence and location of ground support, and the stope dimensions, these plates are either stable or will collapse. Collapse will progress back into the hangingwall until a stable equilibrium is reached due to either an increase in rock beam thickness or a decrease in span.

A schematic illustrating the above concepts is presented in Figure 6.1.

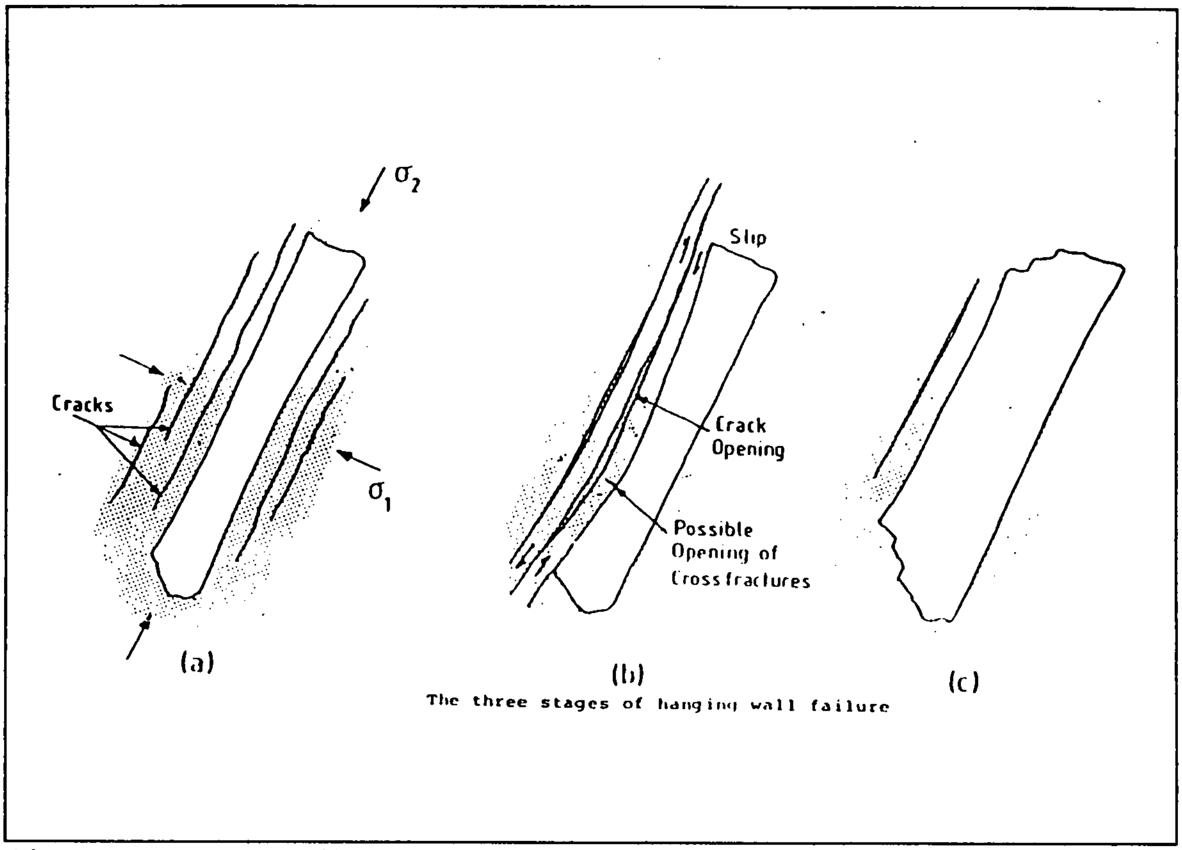


Figure 6.1 Stages of hangingwall failure (after Beer et.al, 1983)

From the above discussion it is logical to assume that the size of the zone of relaxation will have an effect on stope wall stability since it defines the zone where gravity driven block movements can occur. It is also logical to hypothesize that the larger the zone of relaxation the greater the chance of incurring significant amounts of wall slough.

Potvin (1988) states that the relative shape of the stope surface is the single most important parameter influencing the "theoretical" tensile zone (zone of relaxation). The remainder of this study will examine the relationship between stope size and shape and the size of the resulting zone of relaxation.

6.3 OBJECTIVES OF STUDY

The following lists the objectives of this numerical modeling study:

- determine the maximum depth of relaxation for a range of stope geometries and *in situ* stress regimes;

- determine the shape and size (volume) of the zone of relaxation for a range of stope geometries and *in situ* stress regimes;

- based on the modeling results develop relationships to predict the maximum depth of relaxation and the volume of the zone of relaxation;

- compare the volumes of the modeled zones of relaxation to measured values of overbreak/slough using the Equivalent Linear Overbreak Slough (ELOS) procedure;

- in reference to the modelling results, provide comments relating to unplanned dilution and design of ground support for stope walls.

6.4 DESCRIPTION OF NUMERICAL MODELS

6.4.1 Type of Numerical Analysis

Due to the large number of stope geometries to be examined, three dimensional stress analysis was considered necessary. This avoids errors associated with assuming plane strain conditions (which is necessary when using two dimensional stress analysis).

An elastic continuum approach to modeling was considered valid for this study since only the size and shape of the zone of relaxation was being examined, thus, highlighting a *potential* volume of rock that could become unstable. To analyse the actual stability of the volume of rock in the relaxed zone a discontinuum modeling method would have to be used. A benefit of using an elastic continuum is that run times are reduced thus allowing more models to be run in an allotted time period.

The Map3D modeling software was used for this study. Map3D is a comprehensive three-dimensional rock stability analysis package. Model formulation is based on the Indirect Boundary Element Method, and incorporates use of both fictitious force and displacement discontinuity elements. Map3D is user friendly and computationally very efficient allowing very detailed accuracy without excessive run times. A more detailed description of this modeling package can be found in Wiles (1995).

6.4.2 Model Geometries

To examine the effects of stope size and shape on the size of the zone of relaxation eleven (11) different stope geometries were modeled. Stope heights were varied between 20m and 100m and strike lengths were varied between 10m and 100m. For all of the stopes, the dip of the hangingwall and footwall was held constant at 90°. Potvin (1988) examined the effect of varying stope width on hangingwall and footwall stresses. The influence was found to be negligible. Based on his findings, stope widths were held constant at 10m for all the stopes modeled. Figure 6.2 is a schematic illustrating the stope geometries modelled.

Modeling was performed on isolated stopes only.

6.4.3 *In situ* Stresses

Based on several pre-mining stress measurements carried out at depths between 60m and 1890m, Arjang (1991) stated that a common feature at mines with near vertical orebodies is that the maximum horizontal stress acts perpendicular to strike while the minimum horizontal stress is aligned along strike. The vertical stress component approaches the gravitational overburden load. He presented the following average stress gradients:

Maximum Horizontal Stress, $\sigma_{Hmax} = 8.18 + 0.0422$ MPa/m depth

Minimum Horizontal Stress, $\sigma_{Hmin} = 3.64 + 0.0276$ MPa/m depth

Vertical Stress, $\sigma_V = 0.0266 \pm 0.008$ MPa/m depth

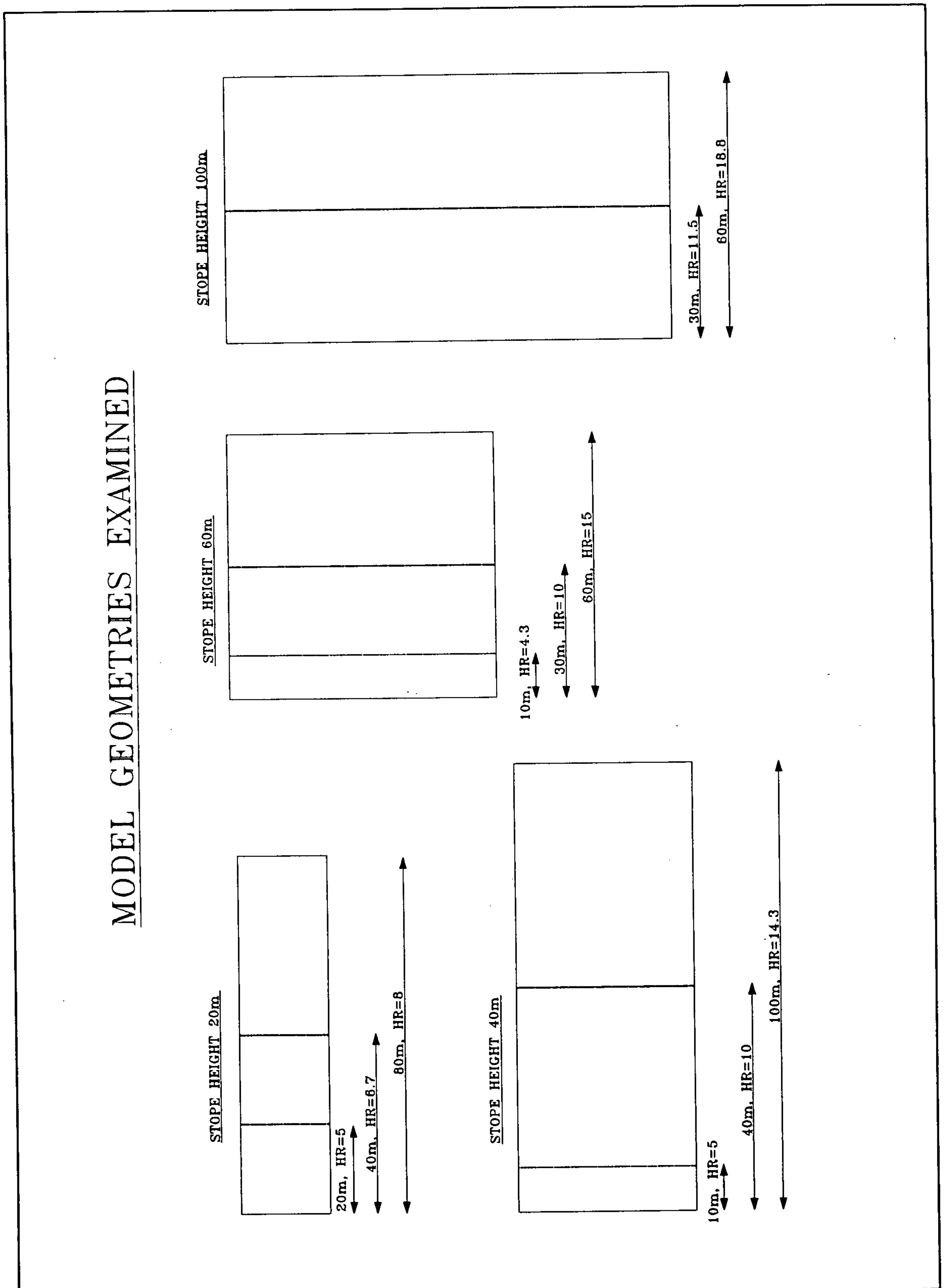


Figure 6.2 Schematic illustrating the different stope geometries modelled as part of this study

Three dimensional numerical modeling performed by Potvin (1988) showed the following:

- if the maximum horizontal stress is aligned perpendicular to strike, the hangingwall and footwall will be in a relaxed state and the back and endwalls will be in compression;
- if the maximum horizontal stress is aligned along strike, the hangingwall and footwall will not be in a relaxed state, the endwall may be in relaxation (depending on the ore width), and the back will be in compression;
- in a hydrostatic stress field ($\sigma_1 = \sigma_2 = \sigma_3$) none of the stope surfaces will be in relaxation.

Based on the above findings the following assumptions were made:

- scenarios will be investigated where the maximum horizontal compressive stress is oriented perpendicular to the strike of the hangingwall and footwall, this will give the maximum zone of relaxation;
- an hydrostatic stress regime will not be investigated;
- *in situ* stress regimes with the maximum horizontal stress aligned along strike will not be investigated.

Using the conventions: $Kh = \sigma_1/\sigma_2$ (where σ_1 is perpendicular to strike and σ_2 is aligned along strike); and $Kv = \sigma_1/\sigma_3$ (where σ_3 the vertical stress), the following *in situ* stress regimes were examined:

1. Kh = Kv = 1.5
2. Kh = Kv = 2.0
3. Kh = Kv = 2.5

These *in situ* stress regimes are in line with those presented by Arjang (1991).

A further assumption made for this study is that depth has a negligible effect on the size of the zone of relaxation. Test models were run which verified this assumption. To fulfill model input requirements all models were run assuming a depth of 500m.

6.4.4 <u>Rock Mass Properties</u>

Since only stresses (as opposed to displacements or strains) are being examined in this study, and an elastic continuum has been assumed, assumptions regarding rock mass properties will have negligible effect on the model results.

To fulfill the model input requirements a rock modulus of elasticity (E) of 50 GPa, and a Poisson's ratio (ν) of 0.3 were assumed.

6.4.5 <u>Model Accuracy</u>

Within Map3D, model results are calculated on user-defined grids that slice through the excavation geometry at any desired location and orientation. Model accuracy is largely controlled by how fine the surface of the excavation is discretized and by the detail of the grid (spacing of grid nodes) upon which the results will be contoured. Guidelines are given by Wiles (1995) which provide parameter settings for achieving: coarse; detailed; or high accuracy results. High accuracy results generally require significant run times.

For this study, high accuracy results were required. To cut down on run times, model size was reduced by using 2 axes of symmetry. This enabled a grid node spacing of 0.3m to be achieved and kept run times to below 8 hours per model.

Two result grids were used in each model. A vertical grid located midway down the stope strike length and a horizontal grid located at the mid-height of the stope.

6.5 MODEL RESULTS

6.5.1 Stresses Around Open Stopes

For all the stopes modeled the general shape of the stress contours was similar. Figures 6.3, 6.4, and 6.5 are plots of σ_1, σ_2, and σ_3 respectively for a 40m high by 40m long stope with K=2.5 (K = Kh = Kv). Referring to the figures, note that the vertical stress grid is actually located at mid-strike length, the other half of the stope cannot be viewed.

It can be seen from the figures that a zone of low σ_1 and tensile σ_2 and σ_3 occurs over most of the stope wall surface, this area represents the relaxed zone. Near the stope abutments σ_1 and σ_2 increase dramatically in magnitude, whereas σ_3 remains tensile, this area represents a zone of high shear stress. These results support the model for hangingwall behavior presented in Section 6.2.

6.5.2 Maximum Depth of Relaxation

For the purpose of this study, the maximum depth of relaxation will be defined as the largest distance from the excavation boundary such that: σ_3 remains tensile; σ_2 is tensile or low compressive; and σ_1 is low compressive. Referring to Figures 6.3,6.4, and 6.5 it can be seen that the maximum depth of relaxation occurs in the center of the stope.

Tables 6.1 through 6.4 (presented at the end of the chapter) document the maximum depth of relaxation for the various stope geometries modeled. Figure 6.6 shows a plot of the maximum depth of relaxation versus stope wall hydraulic radius (area/perimeter).

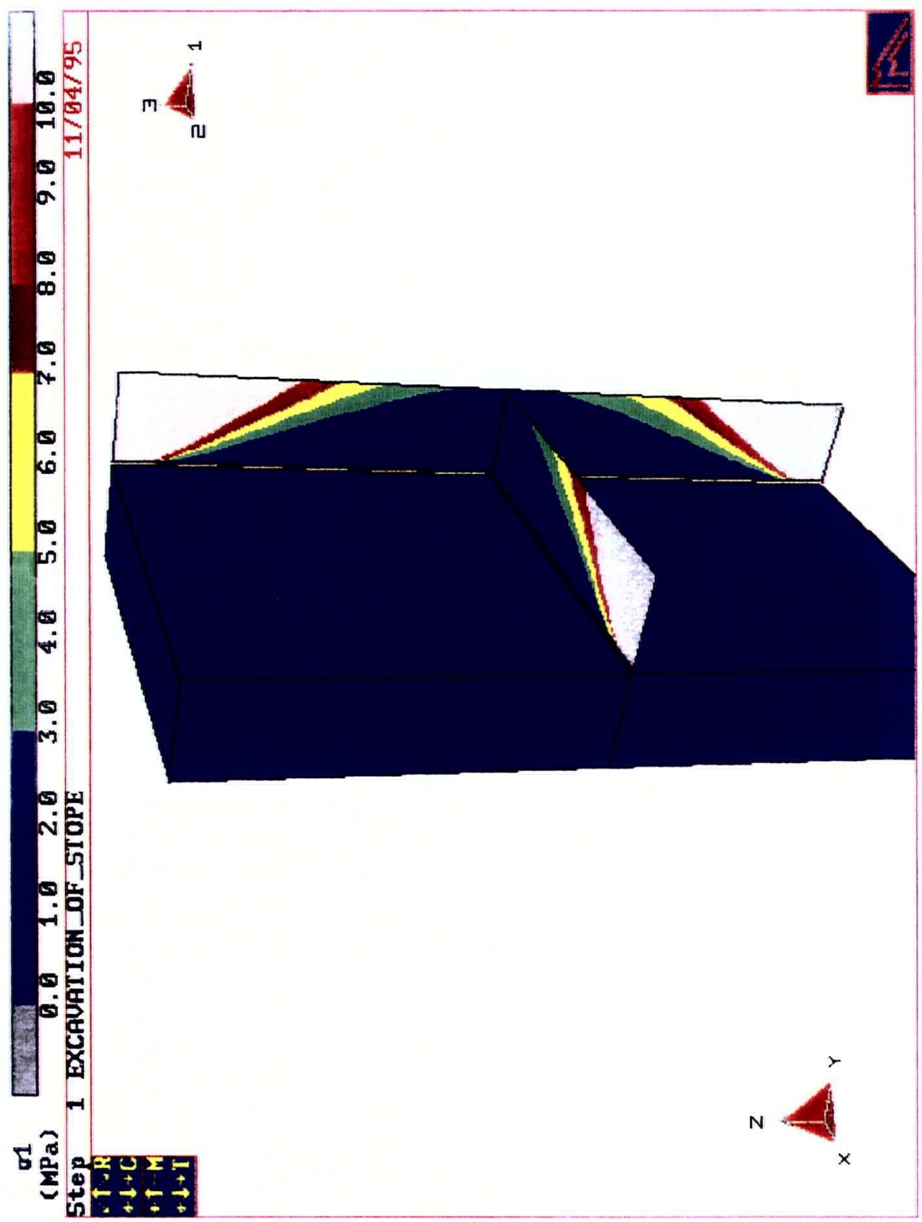


Figure 6.3 σ1 - Typical model results - stope 40m x 40m; K=2.5

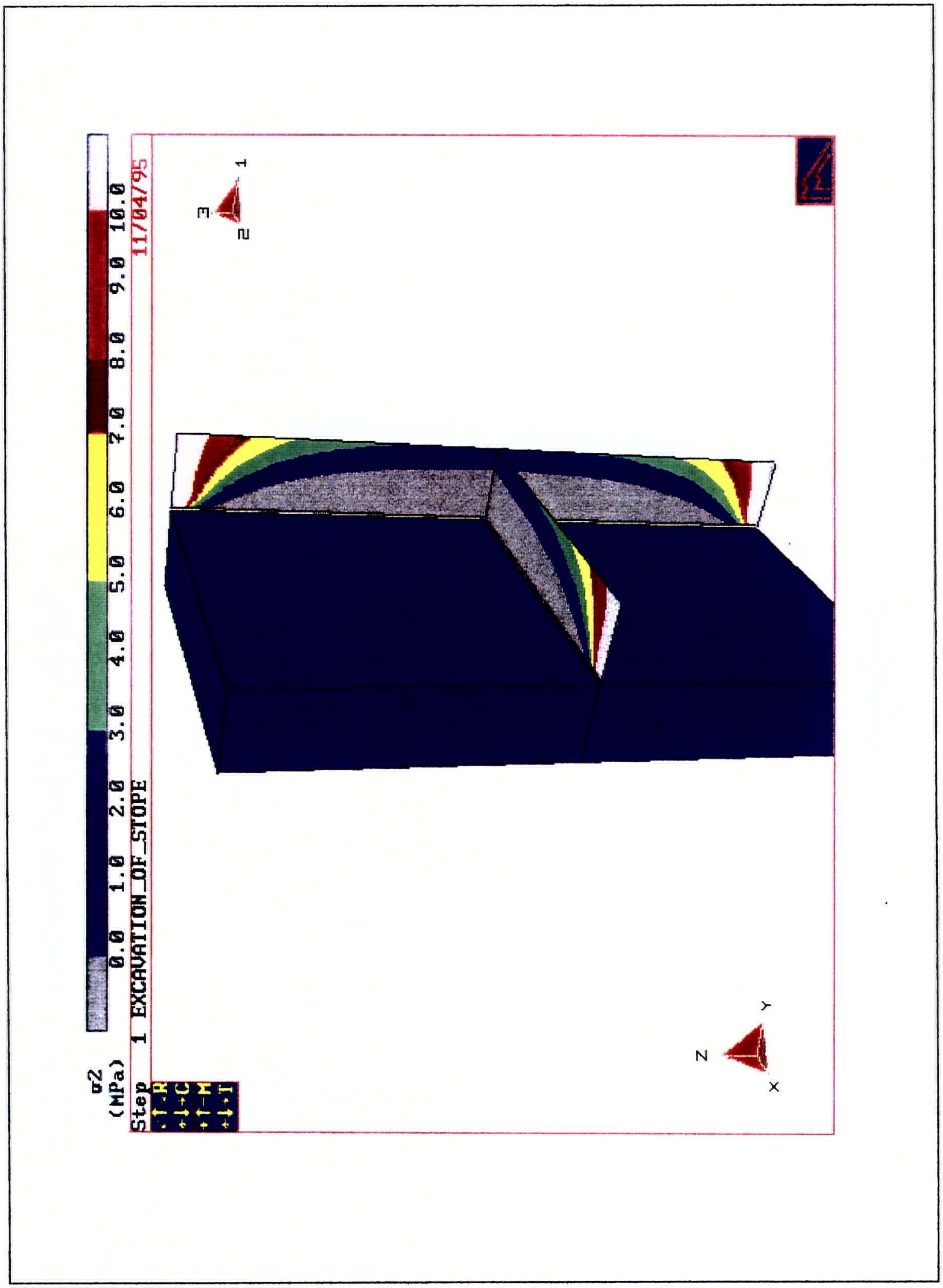


Figure 6.4 σ2 - Typical model results - stope 40m x 40m; K=2.5

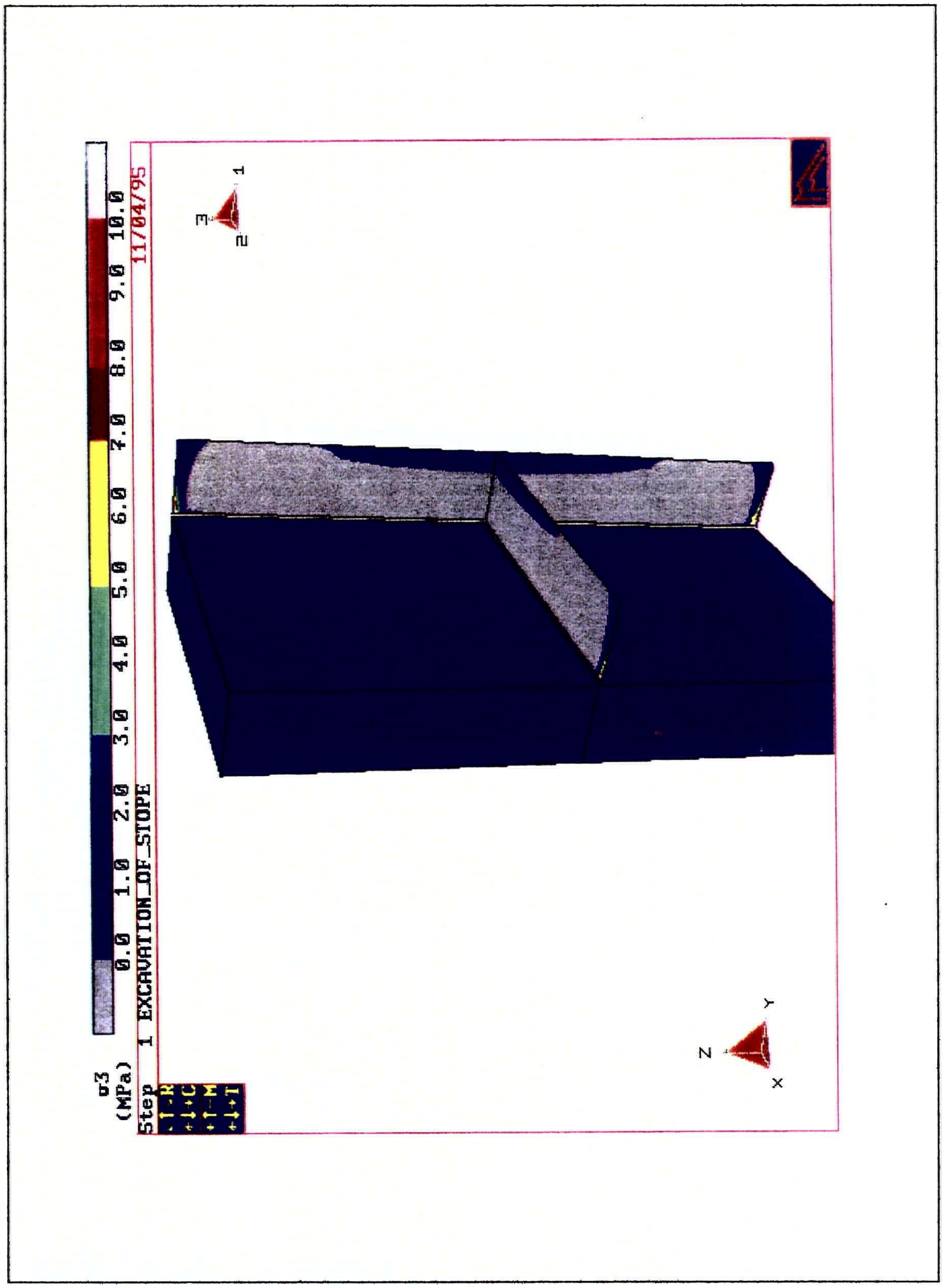


Figure 6.5 σ3 - Typical model results - stope 40m x 40m; K=2.5

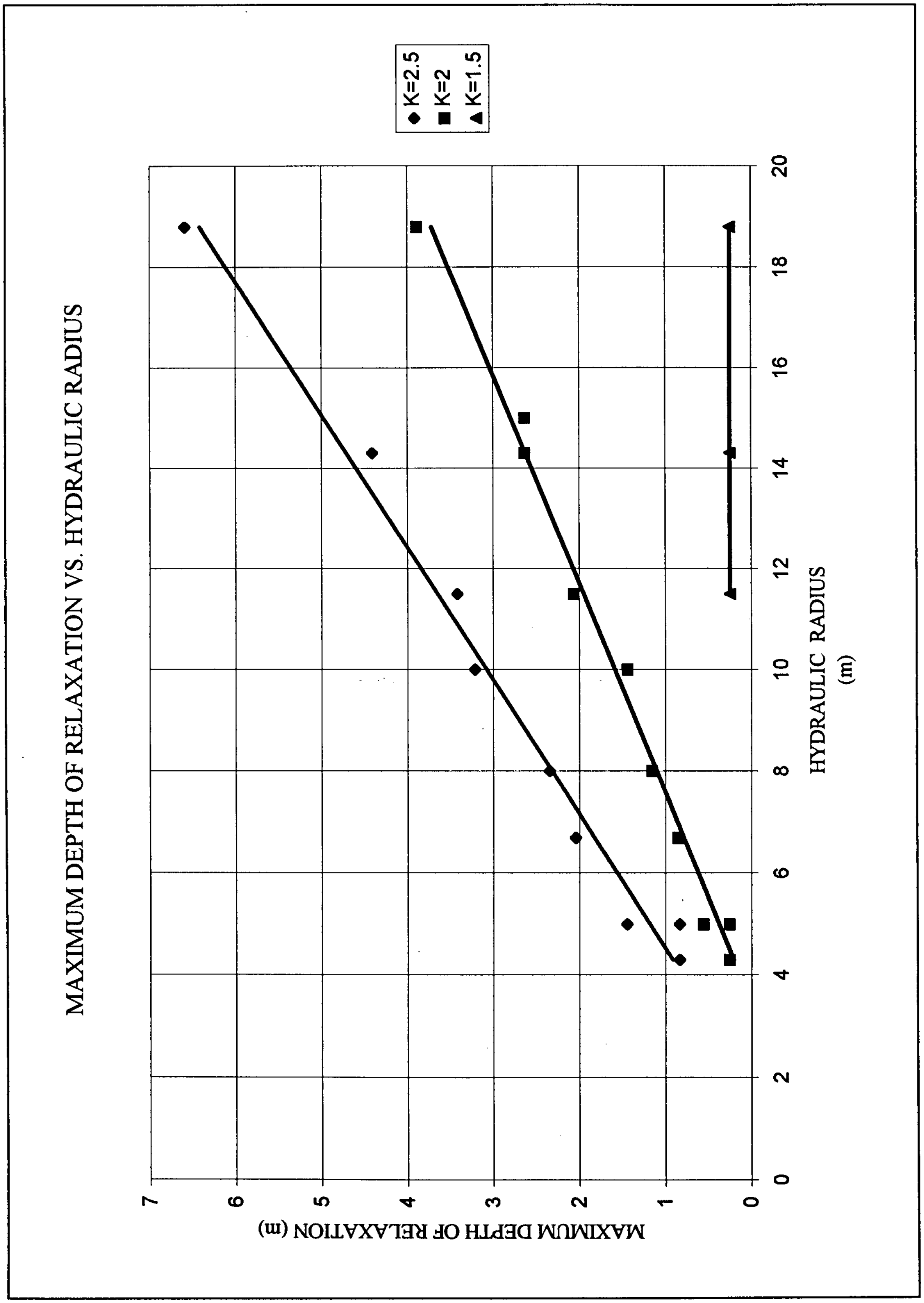


Figure 6.6 Maximum depth of relaxation versus hydraulic radius

Referring to the Tables and Figure 6.6 the following observations can be made:

- for K=1.5 the maximum depth of relaxation is approximately 0.25m. For all intensive purposes it can be assumed that no relaxed zone is formed under this stress regime;
- for K=2 and K=2.5 there exists a strong correlation between increasing depth of relaxation and increasing hydraulic radius;
- for K's >1.5 the depth of relaxation increases significantly as K increases;
- for K=2 and K=2.5 the maximum principal stress (σ_1) at the boundary of the relaxed zone is $\leq$ 10% of σ_1 *in situ*;
- a wide range of stope height to length ratios were modelled and the resulting relationship between the maximum depth of relaxation and hydraulic radius is very linear, suggesting that height to length ratio has little impact on the resulting maximum depth of relaxation for a given hydraulic radius. Refer to Figure 5.1, Chapter 5.

6.5.3 Volume of Zone of Relaxation

The method used to determine the volume of the zone of relaxation is as follows:

1. The maximum depth of relaxation is determined as outlined in Section 6.5.2.
2. The boundary of the zone of relaxation is defined by the σ_1 contour corresponding to the maximum depth of relaxation. This boundary is defined on both the vertical and horizontal stress grids. This concept is depicted graphically in Figure 6.7. The intersection of the σ_1 contours with the excavation boundary defines 4 points (V1,V3, H1, H3).
3. Referring to Figure 6.7 it can be seen that the shape of the zone of relaxation can be approximated using one half of an ellipsoid. Thus, the volume of the zone of relaxation can be determined using the equation given below:

$$\text{Volume of Relaxed Zone} = 1/2\ ((4/3\pi)(MDR)(V1)(H1)) \qquad \text{(Eq. 6.1)}$$

where:

MDR = maximum depth of relaxation
V1, H1 = as defined on Figure 6.7

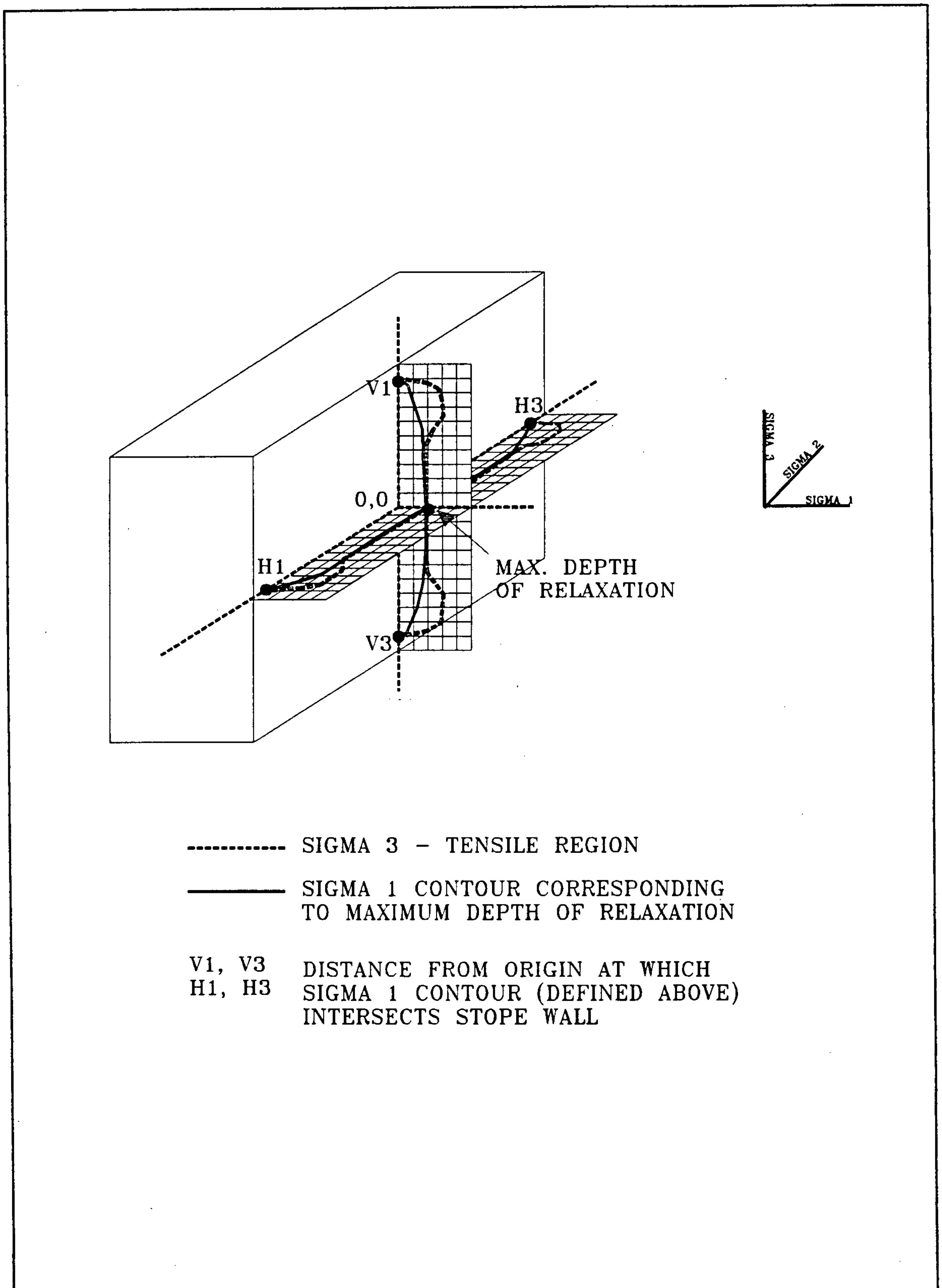


Figure 6.7 Schematic illustrating the parameters required to calculate the volume of the zone of relaxation

Relaxed zone volumes are summarized in Tables 6.1 through 6.4 (presented at the end of the chapter).

Since the zone of relaxation defines a region where gravity driven block failures may occur, it is logical to assume that the volume of the relaxed zone represents a potential volume of material that may be realized as unplanned dilution (overbreak/slough). To facilitate comparison with actual measurements of overbreak/slough from open stope hangingwalls and footwalls (to be discussed in Section 6.5.4) the modelled volumes for the zone of relaxation were converted to ELOS values. Model ELOS values are summarized in Tables 6.1 through 6.4. Figure 6.8 is a plot of the modeling results showing the model ELOS values plotted against hydraulic radius. Referring to the figure the following observations can be made:

- for K=1.5, model ELOS values are essentially zero;
- for K=2 and K=2.5, model ELOS values increase with increasing hydraulic radius, indicating the potential for increased unplanned dilution (overbreak/slough) with larger hangingwall hydraulic radii;
- for K values > 1.5, model ELOS increases with increasing K, suggesting a greater potential to incur unplanned dilution (overbreak/slough) under higher K stress regimes.
- a wide range of stope height to length ratios were modelled and the resulting relationship between the modelled ELOS and hydraulic radius is very linear, suggesting that height to length ratio has little impact on the resulting model ELOS for a given hydraulic radius. Refer to Figure 5.1, Chapter 5.

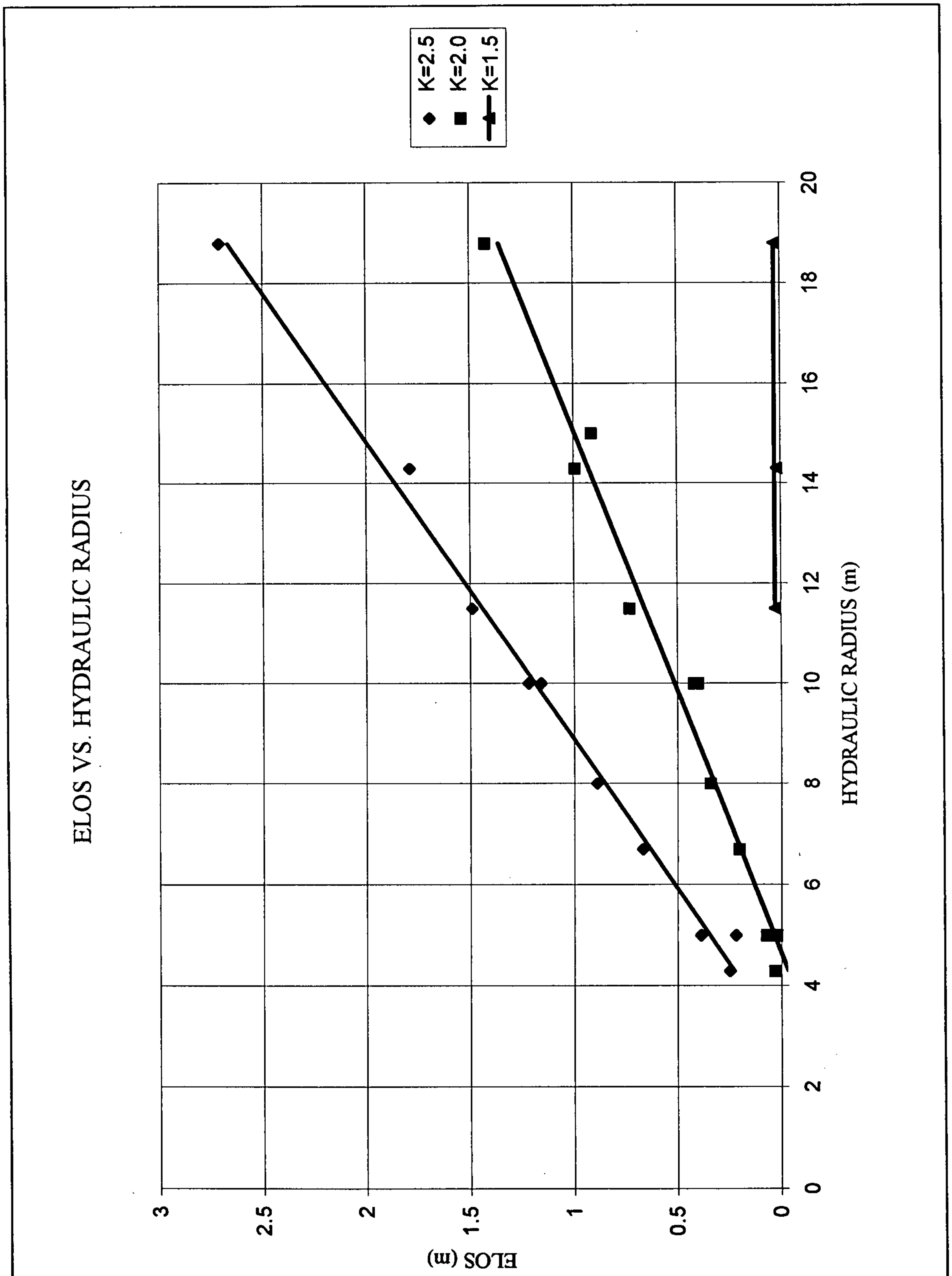


Figure 6.8 Model ELOS values (corresponding to the zone of relaxation) versus hydraulic radius

6.5.4 Comparison of Model ELOS with Measured ELOS

Figures 6.9 and 6.10 are stability graphs (N' vs. HR; and RMR' vs. HR, respectively) showing measured ELOS values from the CMS database. Only data points corresponding to hangingwalls and steeply dipping footwalls (>85°) are shown on the graphs.

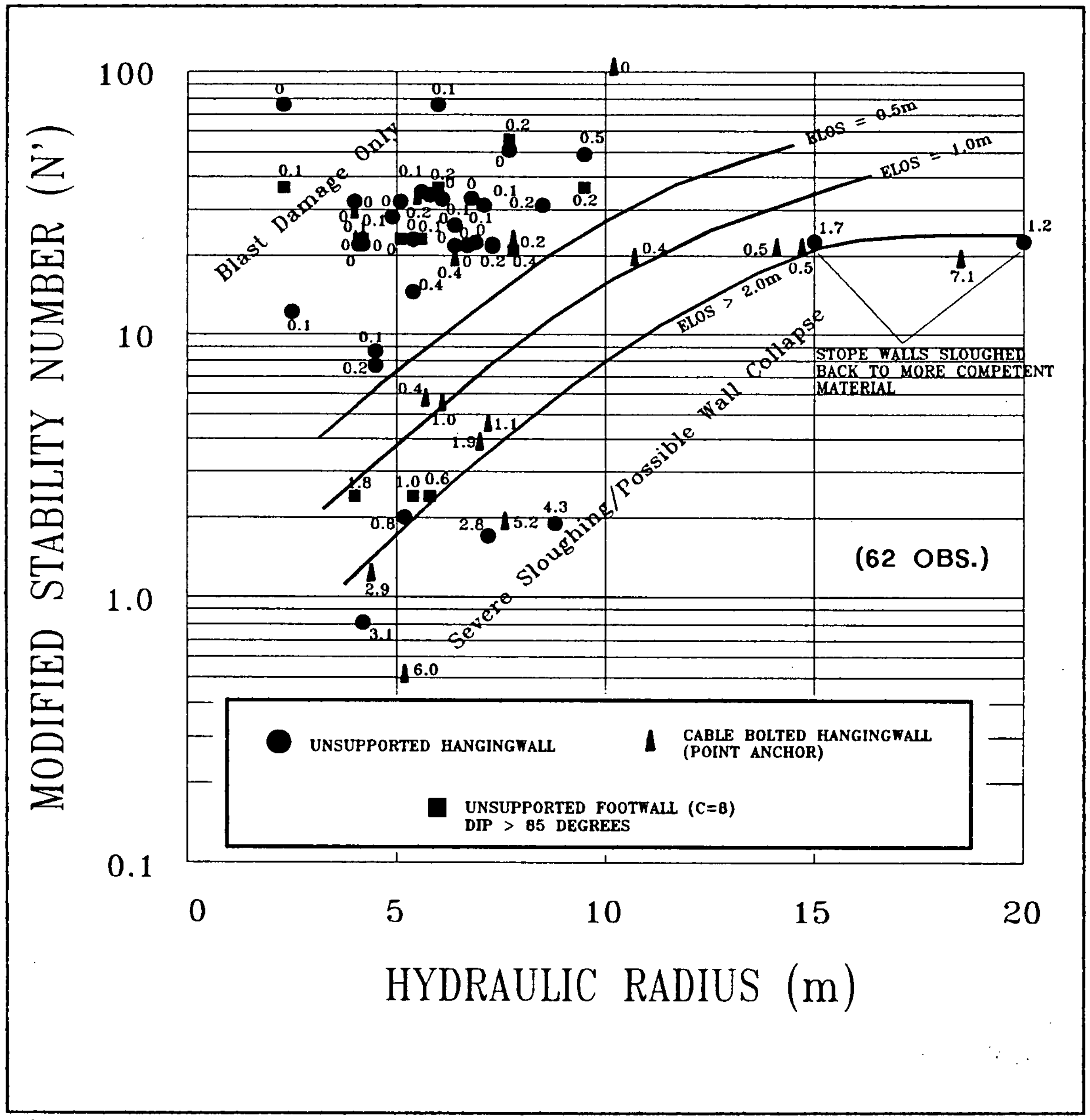


Figure 6.9 N' vs. HR: showing measured ELOS values corresponding to hangingwalls and steeply dipping footwalls (> 85°)

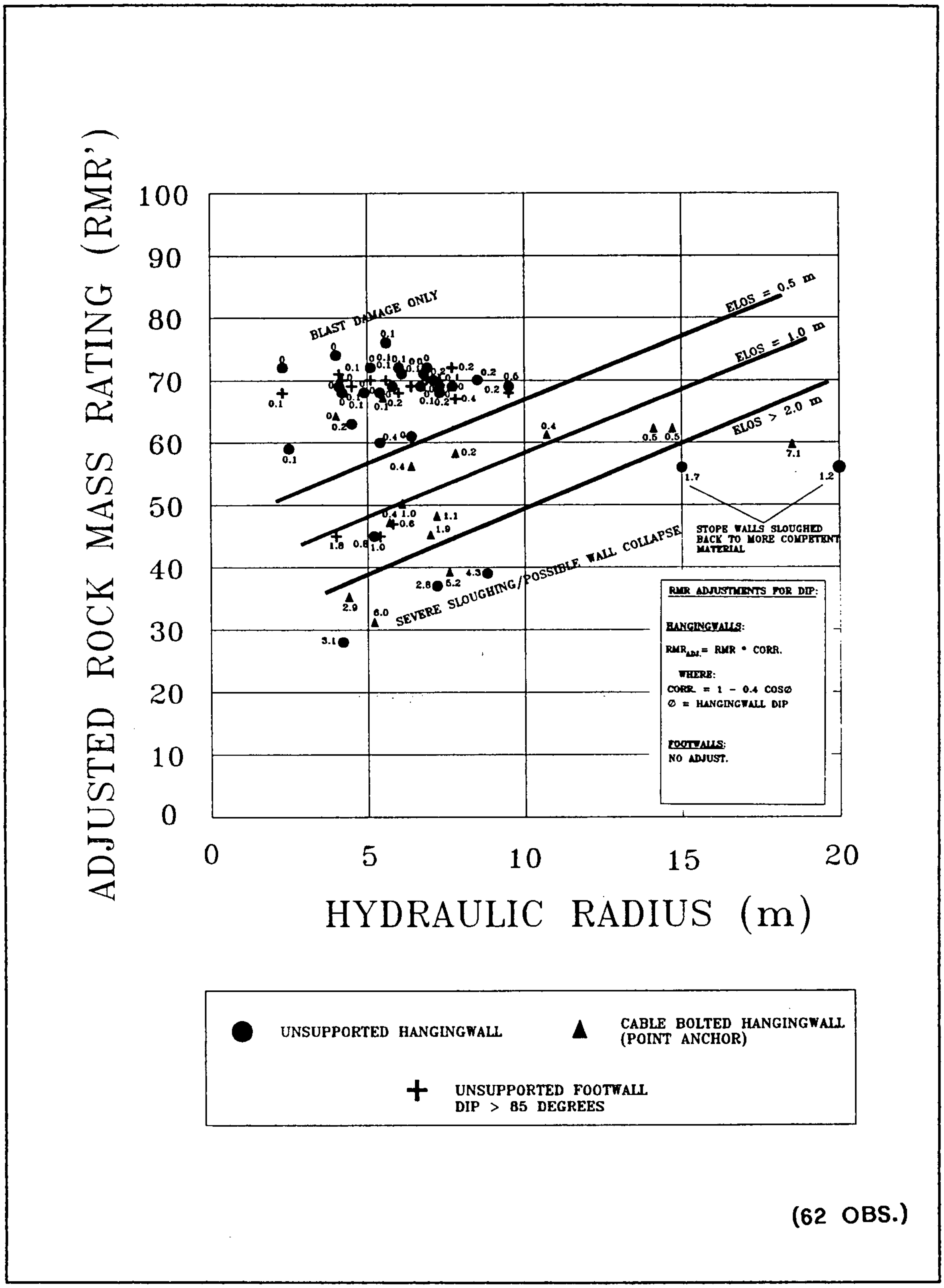


Figure 6.10 RMR'vs. HR: showing measured ELOS values corresponding to hangingwalls and steeply dipping footwalls (> 85°)

The difficulty in comparing the model ELOS values to the measured ELOS values is that a stability number (N' and/or RMR') must be assigned to the model results. Due to this uncertainty it was decided to use the measured values to investigate what stability number best fit the model data. To accomplish this task it was assumed that the modelled values represented a rock mass of constant stability number (which is logical since the models treat the rock mass as an homogeneous isotropic material). The modelled values were then moved up and down the graphs shown in Figures 6.9 and 6.10 until a "best fit" position was found. This procedure was done for both the K=2 and K=2.5 model ELOS values. Figures 6.11 and 6.12 shows the results of this exercise. Referring to Figures 6.11 and 6.12 the following observations can be made:

- the model ELOS values correlate reasonably well with the measured ELOS values between: N' values of 10 and 30; and RMR' values of 58 and 68;

- the N' and RMR' values noted above correspond to rock mass qualities of: Q'= 4.2 - 12.5 (Mathew's et al., 1981, Barton et al. 1974); and RMR = 58 - 68 (Bieniawski, 1976), this assumes vertical stope walls and the presence of discontinuities parallel to the hangingwall and footwall, which is common in most open stoping mines. Intuitively this range of rock mass qualities makes sense. Based on field experience, rock masses of this quality are generally very structured and would be susceptible to gravity driven block failures which could occur in a relaxed rock mass. Rock masses which exhibit high rock qualities likely do not contain enough structure to be susceptible to gravity driven failures. Rock masses which exhibit very low rock qualities would likely unravel beyond the limits defined by the zone of relaxation;

- the reasonable correlation between the model ELOS values and measured ELOS values (within a realistic range of N' and RMR') provides some theoretical justification for the ELOS design zones shown on Figures 6.11 and 6.12.

Having said the above, it is realized that an enormous amount of credence cannot be put on the correlation between the model and measured results. The model results represent isolated stopes with vertical hangingwalls, whereas, not all of the measured values represent isolated stopes and the dip of the hangingwalls range from 45° to 90°. Furthermore, the comparison assumes that the measured ELOS values were obtained from mines with *in situ* stresses in the range of K=2 to K=2.5. Although this a reasonable assumption, it cannot be verified with a great deal of accuracy.

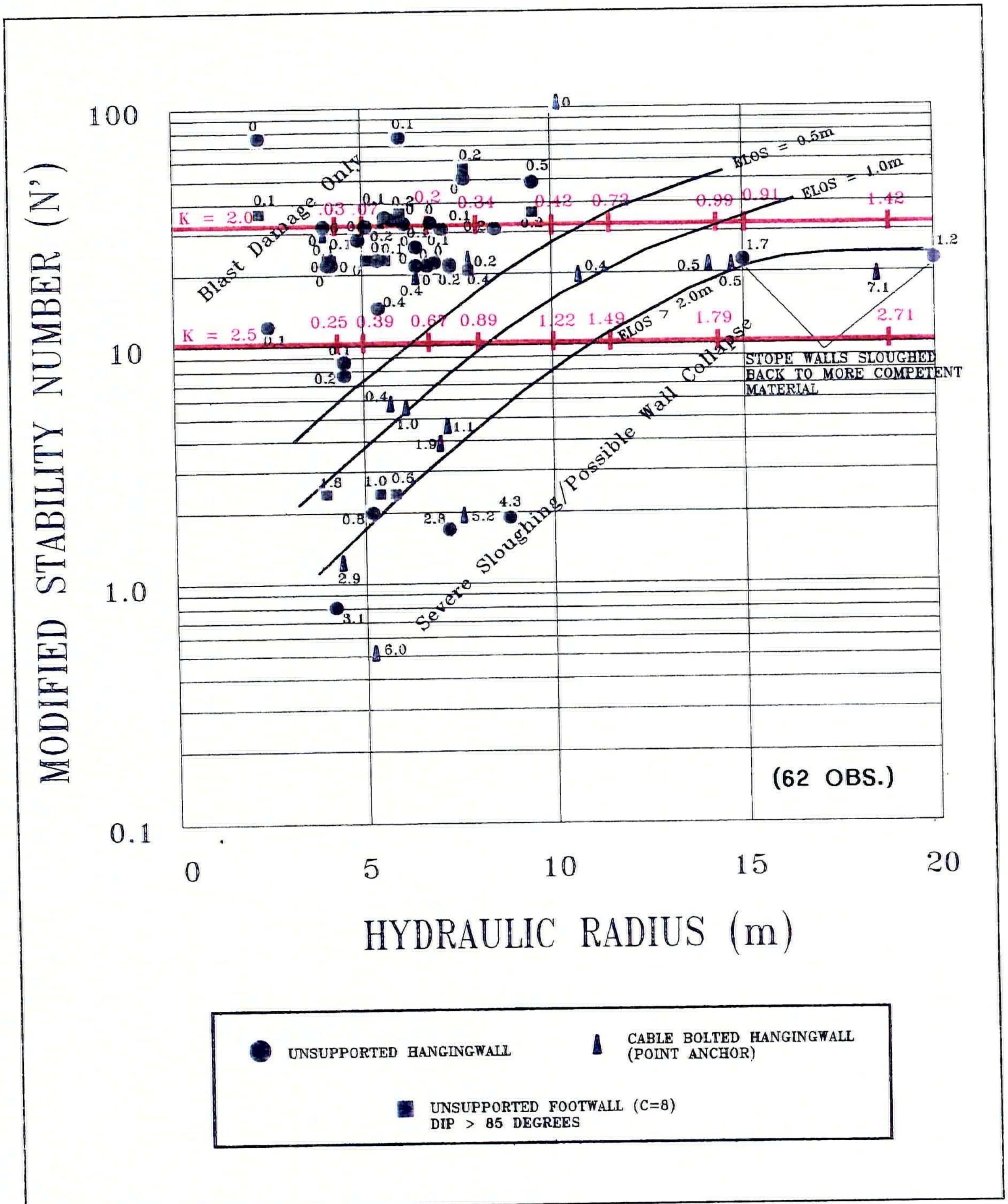


Figure 6.11 N' vs. HR: "Best Fit" for modelled ELOS values

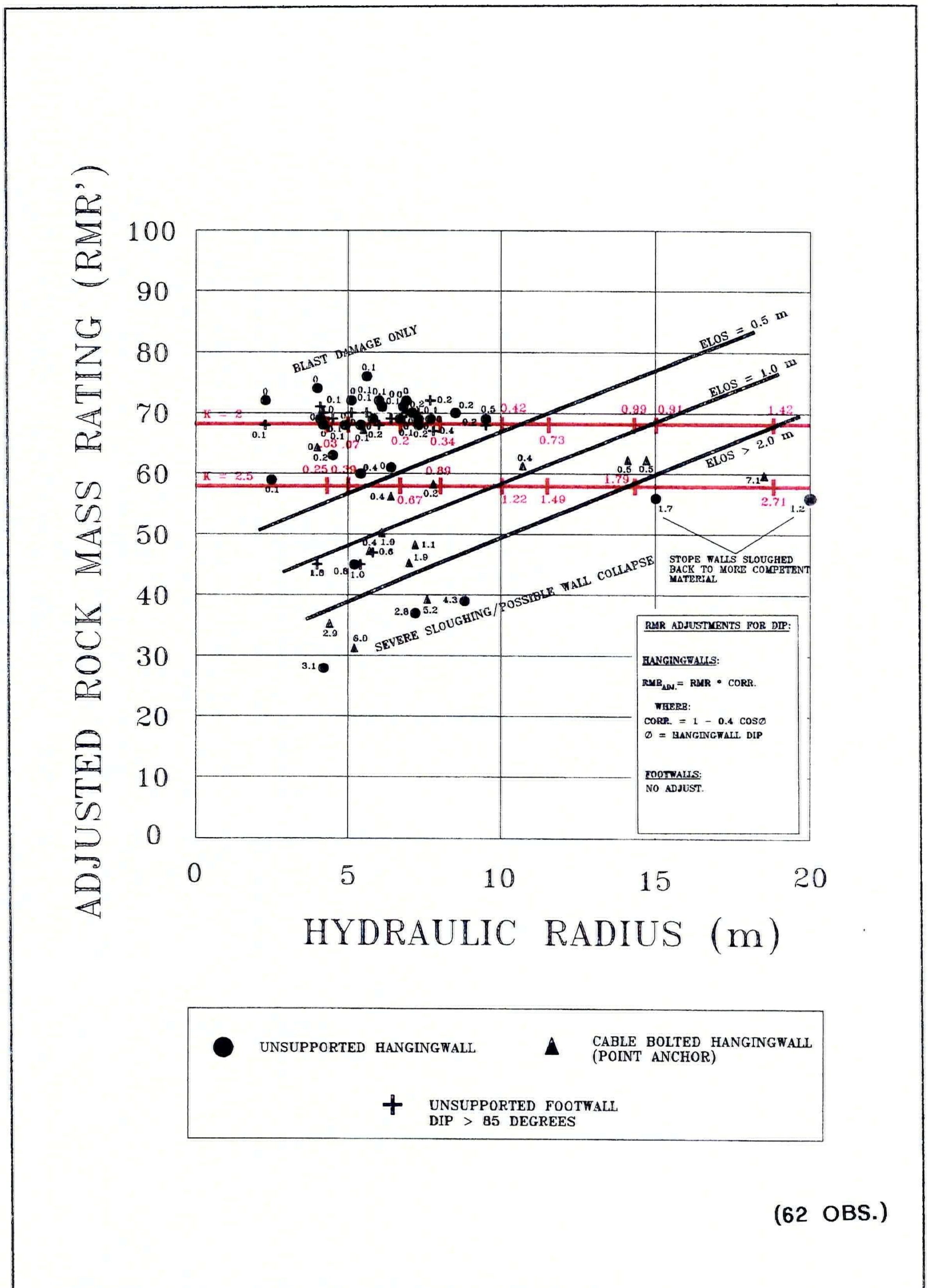


Figure 6.12 RMR' vs. HR: "Best Fit" for modelled ELOS values

6.6 POSSIBLE APPLICATIONS FOR MODEL RESULTS

6.6.1 Estimating Unplanned Dilution

Based on the data presented in Section 6.5.4, the model results shown on Figure 6.8 can be used to give a "ballpark" estimate of overbreak/slough (unplanned dilution) for stopes with vertical or near vertical walls in rock masses with a Q' of approximately 4.2 to 12.5 or RMR of approximately 58 to 68. In better quality rocks the curves will likely overestimate overbreak/slough and in poorer quality rocks the curves will likely underestimate overbreak/slough.

6.6.2 Cable Bolt Design

One possible use for the modelling results is in the design of cable spacing for hangingwall and footwall support. For example, given a particular stope geometry Figure 6.8 can be used to determine the volume of the zone of relaxation. This volume can then be converted into a deadload by applying the appropriate density of the hangingwall or footwall rocks. Cable spacing (bolt density) can then be designed to support the deadload of the zone of relaxation.

Another possible application of the modelling results is in the determination of cable lengths for hangingwall or footwall support. Using Figure 6.6 the maximum depth of the zone of relaxation can be determined. Cable lengths can then designed such that there is at least 2m of anchorage beyond the maximum depth of relaxation (Hutchinson and Diederichs, 1995). This approach is less conservative than current methods of determining cable length. Hutchinson and Diederichs (1995) present a semi-empirical method of determining cable length which amounts to cable length being equal to approximately 1.0-1.5 times the hydraulic radius of the surface being supported. For a hangingwall surface with a hydraulic radius of 10 this would suggest cable lengths of 10-15m. Using Figure 6.6 a conservative estimate of cable length would be 5m. A possible drawback of using Figure 6.6 is that the zone of high stress around the stope abutments is not taken into account, refer to Figures 6.3, 6.4, and 6.5. As can be seen from the figures this zone generally extends farther out from the stope surface than the maximum depth of relaxation. Thus, possibly the apparent conservatism built into the current methods of predicting cable bolt length is warranted. However, it could also be argued that the size of the zone of high stress is partially influenced by the sharp corners in the model geometry. In reality, the stresses would likely cause the sharp corners to spall to a smoother shape resulting in a smaller zone of high stress at the abutments of the stope. This should be studied in greater detail.

6.6.3 Limitations to Design Applications

The numerical modeling results presented in this study are only applicable to stopes with vertical walls which are located in stress environments similar to those assumed for the study. Further research is required to determine the effect of hangingwall and footwall dip on the size and shape of the zone of relaxation. It is postulated that as the dip decreases both the maximum depth of relaxation and the volume of the relaxed zone will decrease. Furthermore, it is expected that the location of the zone of relaxation will re-position further down on the hangingwall and further up on the footwall. This is supported by 2D stress analysis, refer to Figures 6.13, 6.14, and 6.15.

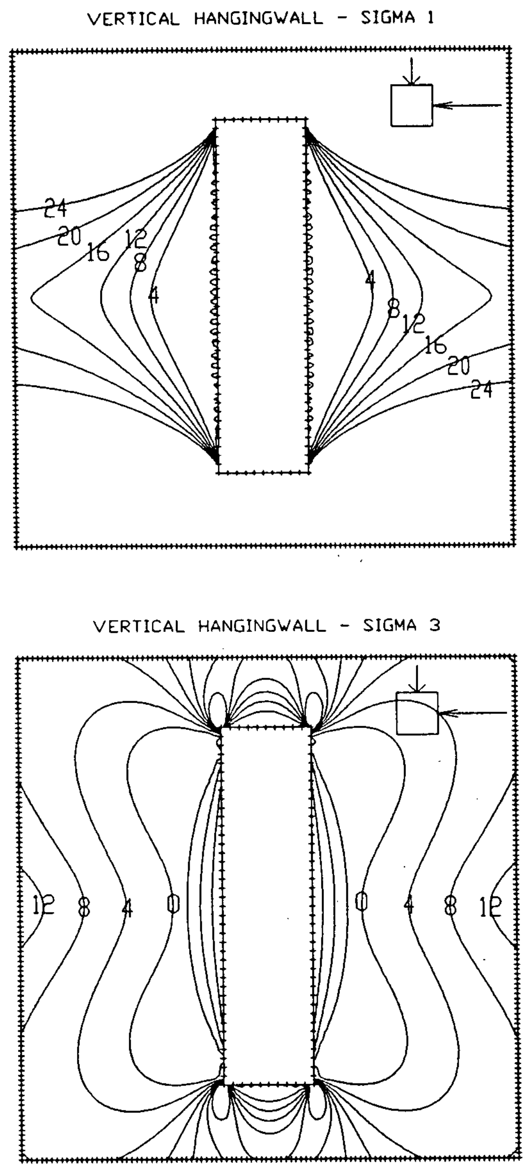


Figure 6.13 2D stress analysis - vertical stope walls

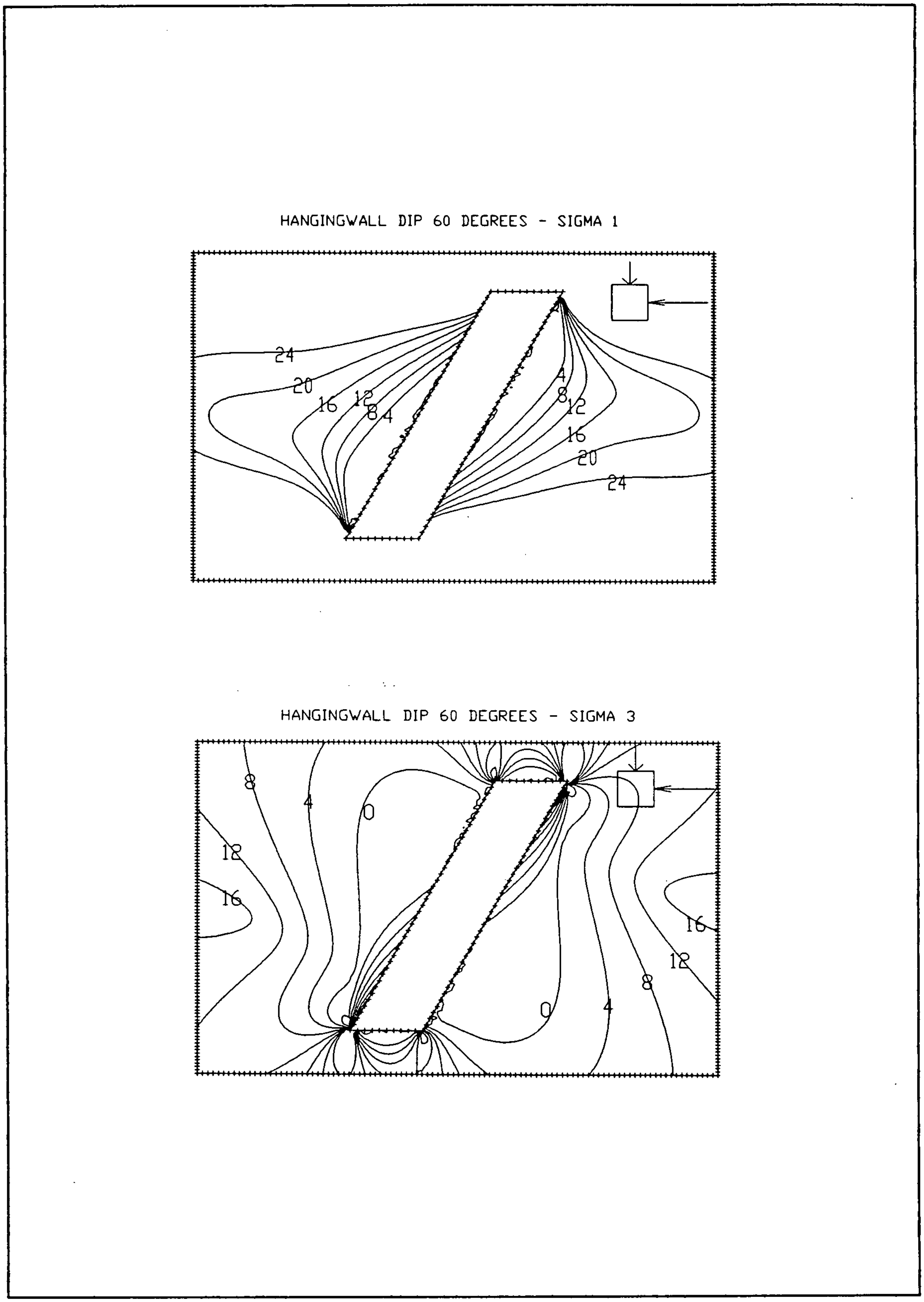


Figure 6.14 2D stress analysis - stope wall dip = 60°

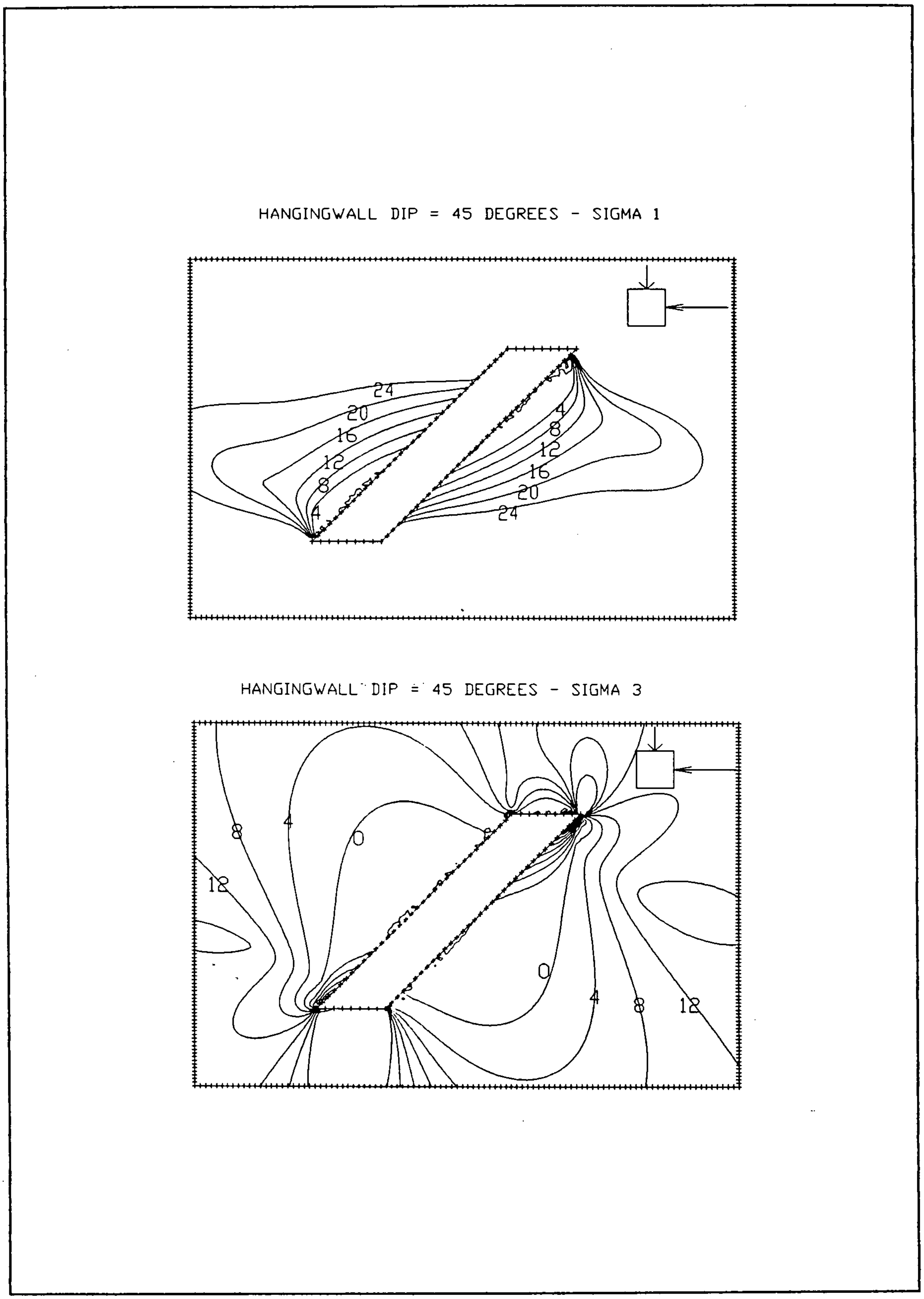


Figure 6.15 2D stress analysis - Stope wall dip = 45°

6.7 SUMMARY

A study of the zone of relaxation around open stope hangingwalls and footwalls has been carried out. The effect of various stope geometries in stress environments common to Canadian mines has been investigated. The following is a summary of the results:

- a model for hangingwall behavior was presented, results from the numerical stress analysis support the model;
- design curves were developed to predict the maximum depth of relaxation in vertical stope walls;
- design curves were developed to predict the volume (ELOS) of the zone of relaxation in vertical stope walls;
- model ELOS values were compared to measured ELOS values and a reasonable correlation was found for a specific range of rock mass qualities;
- application of the model results to estimating unplanned dilution and cablebolt design were discussed;
- areas requiring further research were discussed

TABLE 6.1

SUMMARY OF MAP3D RESULTS
20 m STOPE HEIGHTS

						VERTICAL GRID						HORIZONTAL GRID							
K	HR	L:H	MAX. DEPTH OF RELAXATION (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V1 (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H1 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	VOLUME OF RELAXED ZONE (m3)	ELOS (m)
1.5	5	1	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	5	1	0.55	0.84	3	5	0.6	2	5	0.6	2	5	0.83	3	5	0.83	3	28.8	0.07
2.5	5	1	1.45	1.29	4	7.2	1.25	4	7.2	1.25	4	7.2	1.17	3	7.2	1.17	3	157.4	0.39
1.5	6.7	2	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	6.7	2	0.85	1.68	6	5.6	1.7	6	5.6	1.7	6	15.63	1.52	6	15.63	1.52	6	156.7	0.2
2.5	6.7	2	2.05	1.06	3	7.5	0.95	3	7.5	0.95	3	16.56	0.93	3	16.56	0.93	3	533.3	0.67
1.5	8	4	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	8	4	1.15	2.8	10	6.3	2.76	10	6.3	2.76	10	36.25	2.52	9	36.25	2.52	9	545.7	0.34
2.5	8	4	2.35	1.6	5	7.8	1.49	4	7.8	1.49	4	37.19	1.5	4	37.19	1.5	4	1429.6	0.89

TABLE 6.2

SUMMARY OF MAP3D RESULTS
40 m STOPE HEIGHTS

						VERTICAL GRID						HORIZONTAL GRID							
K	HR	L:H	MAX. DEPTH OF RELAXATION (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V1 (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H1 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	VOLUME OF RELAXED ZONE (m3)	ELOS (m)
1.5	5	0.25	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	5	0.25	0.25	2.65	10	15	1.95	7	15	1.95	7	1.25	2.8	10	1.25	2.8	10	9.8	0.02
2.5	5	0.25	0.84	1.15	3	15.94	1.23	4	15.94	1.23	4	3.13	1.24	4	3.13	1.24	4	87.8	0.22
1.5	10	1	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	10	1	1.44	0.27	1	15	0.89	3	15	0.89	3	15	0.85	3	15	0.85	3	678.6	0.42
2.5	10	1	3.22	0.7	2	16.56	0.82	2	16.56	0.82	2	16.56	0.81	2	16.56	0.81	2	1849.4	1.16
1.5	14.3	2.5	0.25	3.8	19	5.25	4.2	21	5.25	4.2	21	28.13	3	15	28.13	3	15	77.3	0.02
2	14.3	2.5	2.63	2.1	8	15.63	2.17	8	15.63	2.17	8	46.09	1.12	4	46.09	1.12	4	3968.1	0.99
2.5	14.3	2.5	4.41	0.78	2	16.25	0	0	16.25	0	0	47.66	0.73	2	47.66	0.73	2	7153.3	1.79

NOTES:

K - for the above models, Kv = Kh = K, where Kv = sigma H1/ sigma vertical, and, Kh = sigma H1/sigma H2
L:H - stope strike length : stope height
Max Depth of Relaxation, V1, V3, H1, H3
Sigma 1 Induced - refers to the sigma 1 stress value at the point referenced in the preceding column
% Sigma 1 Insitu - ((Sigma 1 Induced / Sigma 1 Insitu) x 100)
ELOS - Equivalent Linear Overbreak / Slough
HR - Hydraulic Radius

TABLE 6.3

SUMMARY OF MAP3D RESULTS
60 m STOPE HEIGHTS

						VERTICAL GRID						HORIZONTAL GRID							
K	HR	L:H	MAX. DEPTH OF RELAXATION (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V1 (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H1 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	VOLUME OF RELAXED ZONE (m3)	ELOS (m)
1.5	4.3	0.17	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	4.3	0.17	0.25	3	11	25.31	2.16	8	25.31	2.16	8	1.25	3.11	12	1.25	3.11	12	16.6	0.03
2.5	4.3	0.17	0.84	1.38	4	27.66	1.82	5	27.66	1.82	5	3.13	1.35	4	3.13	1.35	4	152.3	0.25
1.5	10	0.5	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
2	10	0.5	1.44	1.52	6	25.31	1.78	7	25.31	1.78	7	9.38	1.54	6	9.38	1.54	6	716	0.4
2.5	10	0.5	3.22	0.95	3	26.72	1.16	3	26.72	1.16	3	12.19	0.6	2	12.19	0.6	2	2196.6	1.22
1.5	15	1																	
2	15	1	2.63	0.09	0.3	24.38	1.08	4	24.38	1.08	4	24.38	1.02	4	24.38	1.02	4	3274	0.91
2.5	15	1																	

TABLE 6.4

SUMMARY OF MAP3D RESULTS
100 m STOPE HEIGHTS

						VERTICAL GRID						HORIZONTAL GRID							
K	HR	L:H	MAX. DEPTH OF RELAXATION (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V1 (m)	SIGMA 1 INDUCED (MPa)	% SIGMA 1 INSITU (MPa)	V3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H1 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	H3 (m)	SIGMA 1 INDUCED (MPa)	%SIGMA 1 INSITU (MPa)	VOLUME OF RELAXED ZONE (m3)	ELOS (m)
1.5	11.5	0.33	0.25	4.26	21	28.91	3.63	18	28.91	3.63	18	6.56	4.46	22	6.56	4.46	22	99.3	0.03
2	11.5	0.33	2.06	2.57	10	46.88	2.11	8	46.88	2.11	8	10.78	2.11	8	10.78	2.11	8	2180.4	0.73
2.5	11.5	0.33	3.42	1.06	3	47.66	0.15	0.5	47.66	0.15	0.5	13.13	0.49	1	13.13	0.49	1	4482.3	1.49
1.5	18.8	0.6	0.25	2.5	12	21.48	2.21	11	21.48	2.21	11	16.88	3.15	15	16.88	3.15	15	189.8	0.03
2	18.8	0.6	3.88	1.12	4	42.19	1.23	5	42.19	1.23	5	24.84	1.11	4	24.84	1.11	4	8516.3	1.42
2.5	18.8	0.6	6.59	0.66	2	43.36	0.77	2	43.36	0.77	2	27.19	-0.02	0	27.19	-0.02	0	16272.1	2.71
1.5	25																		
2	25																		
2.5	25																		

NOTES:

K - for the above models, Kv = Kh = K, where Kv = sigma H1/ sigma vertical, and, Kh = sigma H1/sigma H2
L:H - stope strike length : stope height
Max Depth of Relaxation, V1, V3, H1, H3
Sigma 1 Induced - refers to the sigma 1 stress value at the point referenced in the preceding column
% Sigma 1 Insitu - ((Sigma 1 Induced / Sigma 1 Insitu) x 100)
ELOS - Equivalent Linear Overbreak / Slough
HR - Hydraulic Radius

CHAPTER 7

RELATIONSHIPS BETWEEN ELOS AND OTHER DATABASE PARAMETERS

7.1 GENERAL

The chapter attempts to quantitatively evaluate what factors, other than those accounted for in the stability graph method, influence hangingwall and footwall stability (i.e. undercutting; drilling; blasting; time; etc..). At present, the CMS database is not large enough to reliably determine additional factors which could be incorporated into the design approach presented in Chapter 5. However, it is hoped the following sections will provide insight into factors which may increase the probability of hangingwall and footwall instability, potentially resulting in higher ELOS values than would be estimated using the stability graphs.

Two approaches were used to explore for relationships between ELOS and the various CMS database parameters. The first approach used scatter plots and the second approach used neural networks. The following sections describe the two approaches taken and present the results from the analyses.

7.2 SCATTER PLOT ANALYSIS

7.2.1 Description of Analysis

Scatter plots were used to examine for correlations between ELOS and other database parameters. This is a simplistic approach since generally only one parameter is analyzed per plot (i.e. ELOS vs. some other database parameter). Using this method, only very obvious correlations with ELOS become apparent. Many of the factors controlling stope stability are inter-dependent which complicates trying to identify relationships. More sophisticated methods exist for performing multi-variable analysis. One such method is to use *neural networks*. This will be examined further in Section 7.3.

Due to the low occurrence of stability problems associated with footwalls, only hangingwall data and steeply dipping footwall data (≥ 85°) was used to explore for possible relationships between ELOS and other database parameters. Furthermore, to remove any effects of stope support, only ELOS values from unsupported stope surfaces were used. This resulted in 46 data points for the scatter plot analysis: 31 unsupported hangingwalls; and 15 footwalls with a dip ≥ 85°.

7.2.2 Presentation and Discussion of Scatter Plot Results

Depth

Figure 7.1 shows ELOS plotted versus stope depth.

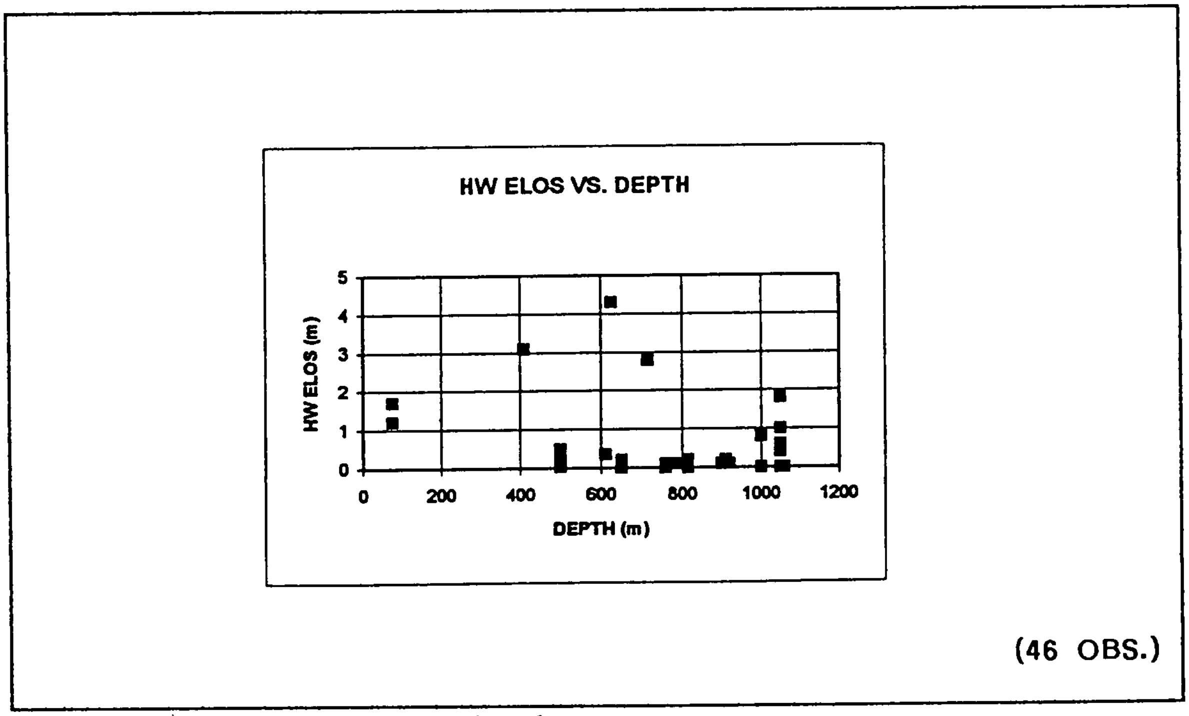


Figure 7.1 ELOS versus Stope Depth

No correlation can be observed. This provides some evidence that the hangingwalls and footwalls contained in the CMS database were generally in a low stress or relaxed state.

Stope Geometry

Figure 7.2 presents a series of scatter plots showing ELOS plotted versus: stope height to length ratio; stope height; stope strike length; and stope surface character.

There is some debate whether stopes with large height/length ratios are more stable than stopes with small height/length ratios. No obvious trend is observable. However, there is a weak correlation of increasing ELOS with increasing stope height (note: tall stopes do not necessarily have large height/length ratios). The relationship may be indicative of factors common to excavating tall stopes such as: greater chance of undercutting due to more sub-levels; greater likelihood of having an irregular stope surface (i.e. dog-leg); if upholes and downholes are being used the number of blasts required to excavate the stope may be large; and if only downholes are used the length of blastholes may be excessive resulting in significant hole deviation. No correlation is observable between ELOS and stope strike length.

A reasonable correlation exists between ELOS and stope surface character. This supports the idea that irregular stope walls (i.e. dog-legs along dip or strike; irregular ore contacts) are less stable than planar stope surfaces.

Figure 7.3 shows scatter plots of ELOS vs. hydraulic radius. Only a weak correlation is observable. A stronger correlation was expected since hydraulic radius is a main parameter in stope design. The weak correlation can be partially explained by the limited range of data in the database. The majority of stopes in the database range between hydraulic radius values of 4 to 8. Data from large stopes (HR>10) is limited. Furthermore, the concept of hydraulic radius which assumes that the distance to supporting abutments has an influence on the stability of the walls, is probably most significant for big stopes and stopes in poor quality rock, and likely less significant in small stopes in good rock (which form a significant portion of the database). In the latter case, factors such as blast damage and undercutting of stope walls are probably more significant.

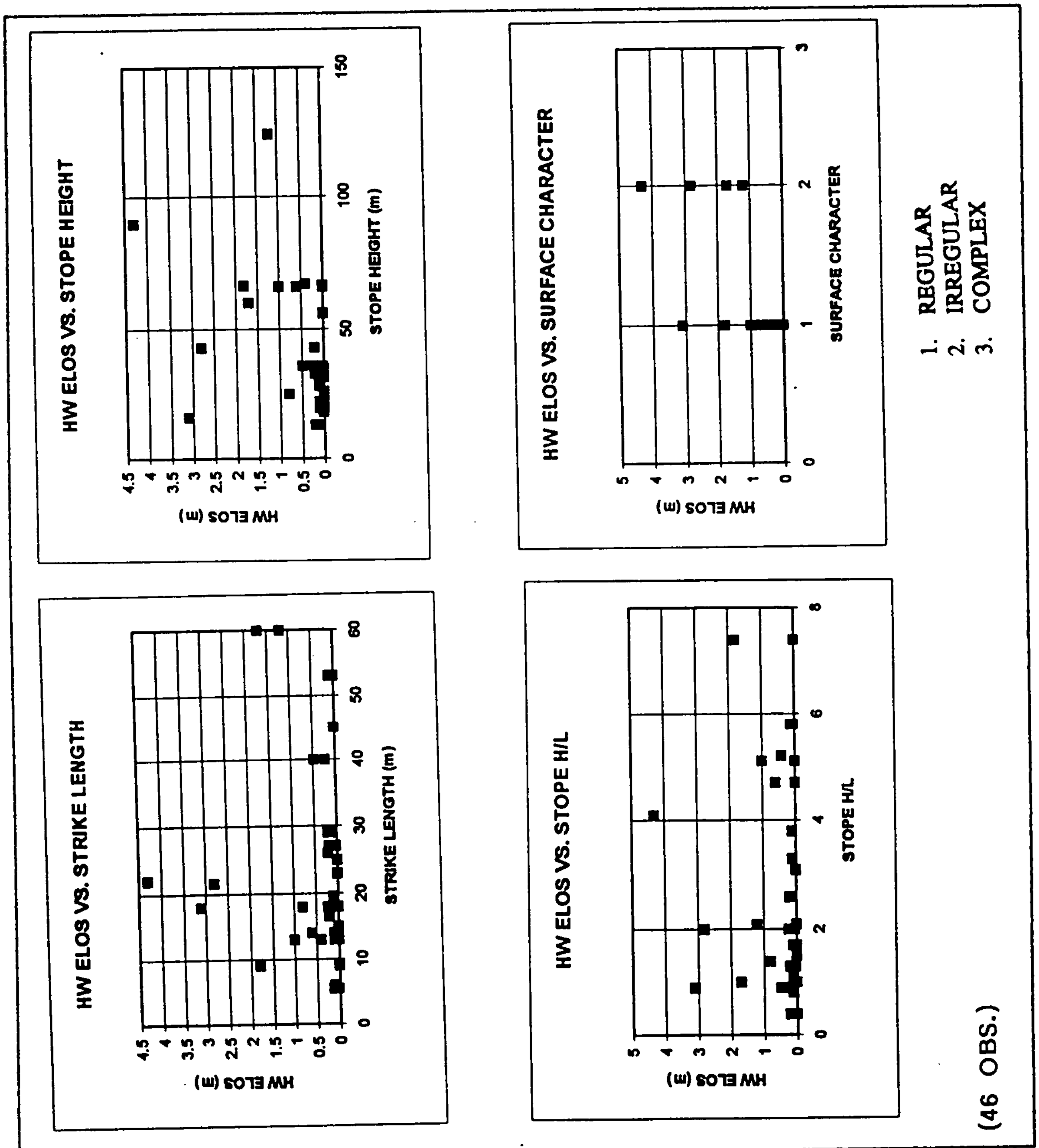
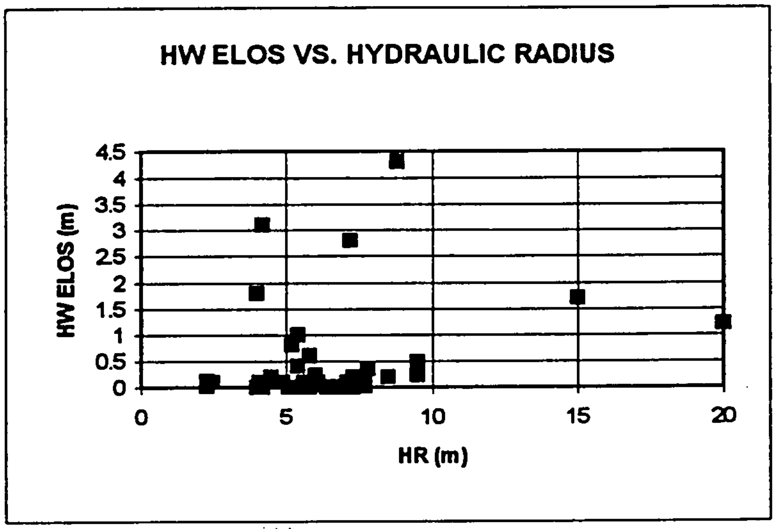
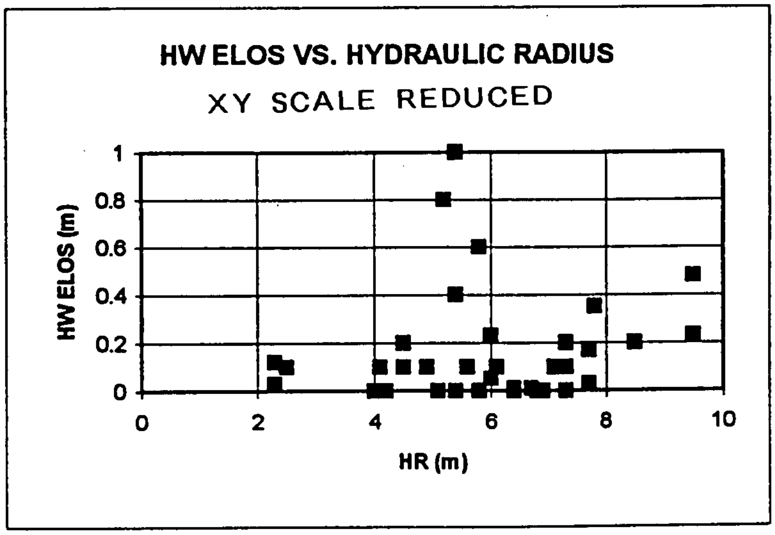


Figure 7.2 Scatter plots examining relationships between ELOS and various stope geometry parameters





(46 OBS.)

Figure 7.3 Scatter plots showing ELOS vs. Hydraulic Radius

Figure 7.4 presents scatter plots of: ELOS vs. N'; and ELOS vs. RMR'. Both N' and RMR' show good correlations with ELOS.

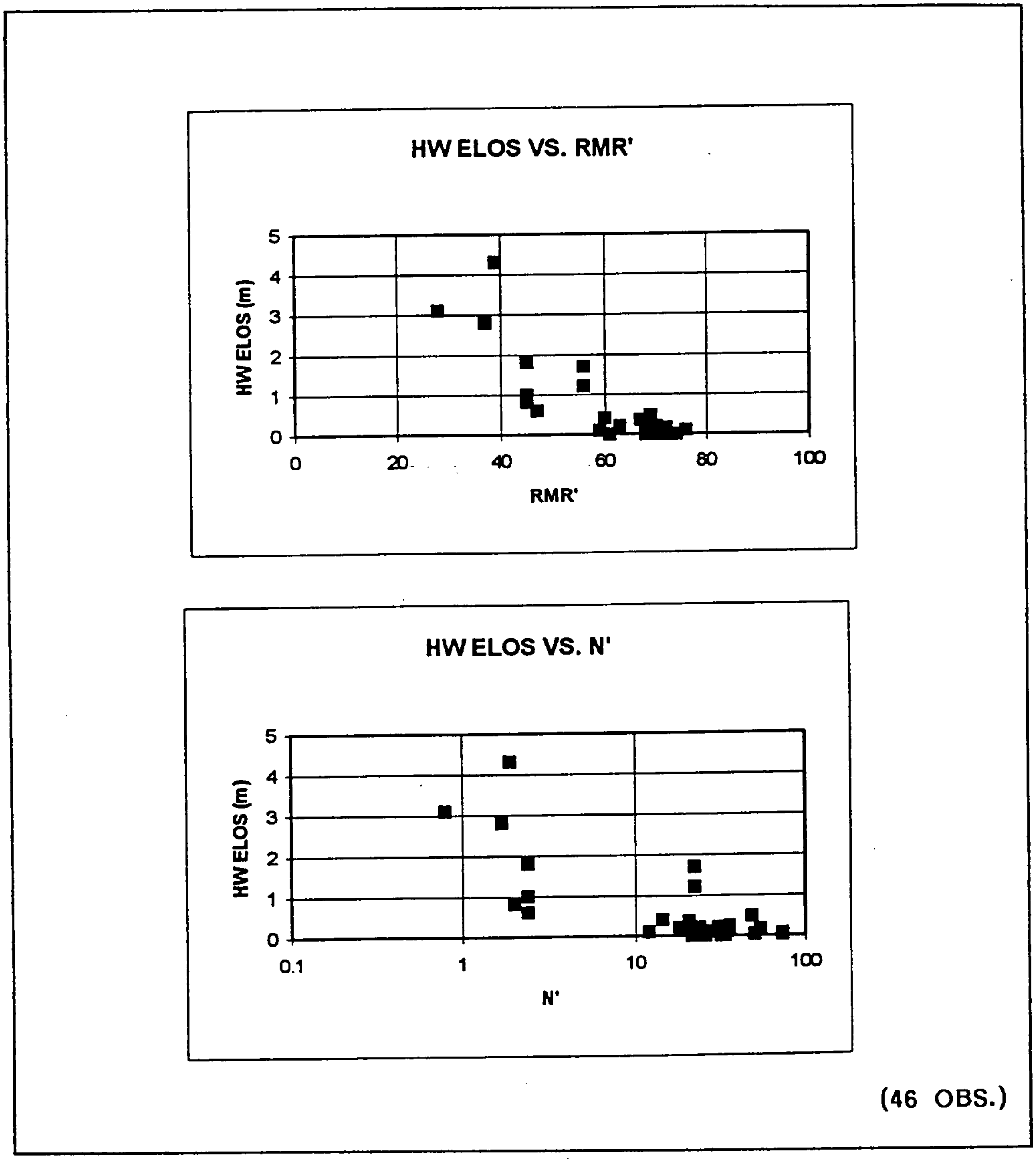


Figure 7.4 ELOS vs. N' and ELOS vs. RMR'

A good correlation with the Modified Stability Number (N') was expected since it has proven to be a reliable parameter for stope design.

The good correlation with RMR' provides additional confidence in this new parameter.

The good correlations with both rock mass classification systems highlights the major influence that rock mass quality has on the stability of open stopes.

Undercutting of Stope Walls

Undercutting of stope walls is well recognized as being a potential cause for wall instability. Observations underground indicate that the destabilizing effect of undercutting is largely dependent on the spacing, continuity, and surface characteristics of discontinuities which parallel the stope walls.

Figure 7.5 shows plots of: undercut depth vs. ELOS; and ELOS versus the undercut classification system presented in Section 4.2.4, Chapter 4. Note that the data shown in Figure 7.5 has been divided into ranges of RMR' and that the "depth of undercut" refers to the average depth of undercut along the entire strike length of the stope. The linear lines shown on two of the graphs are lines of linear regression corresponding to a specific range of RMR'.

A good correlation is observable between ELOS, the depth of undercut, and RMR'. The graph indicates that stope walls with a RMR' greater than approximately 65 appear to be less susceptible to undercutting than stope walls comprised of a lower quality rock mass. This graph is not meant for design, it is only meant to illustrate that in general lower quality rocks are much more susceptible to stability problems associated with undercutting than good quality rocks.

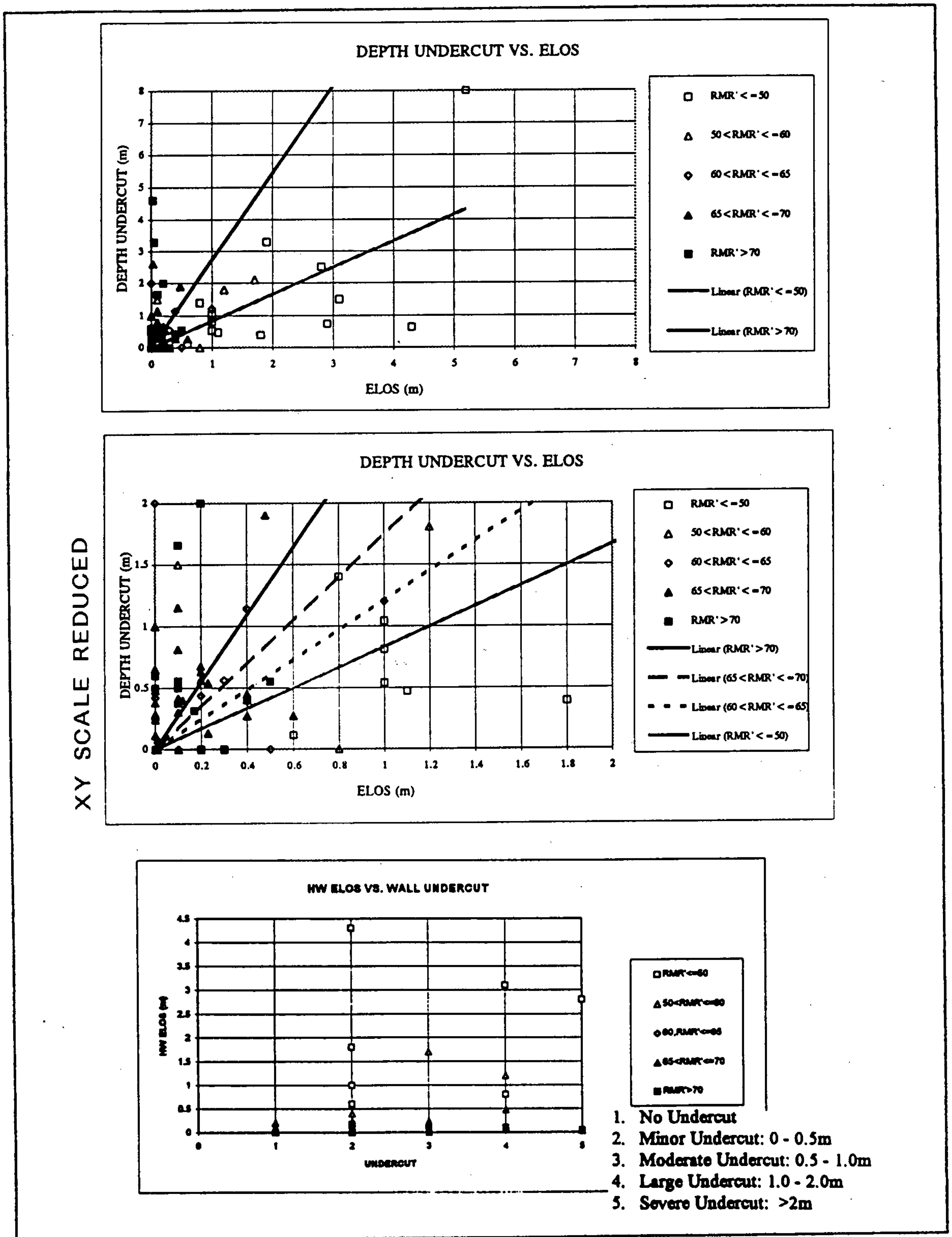


Figure 7.5 Plots examining the relationship between stope wall undercutting and ELOS

Drilling and Blasting

Figure 7.6 shows plots of: ELOS vs. blasthole diameter; and ELOS vs. average blasthole length.

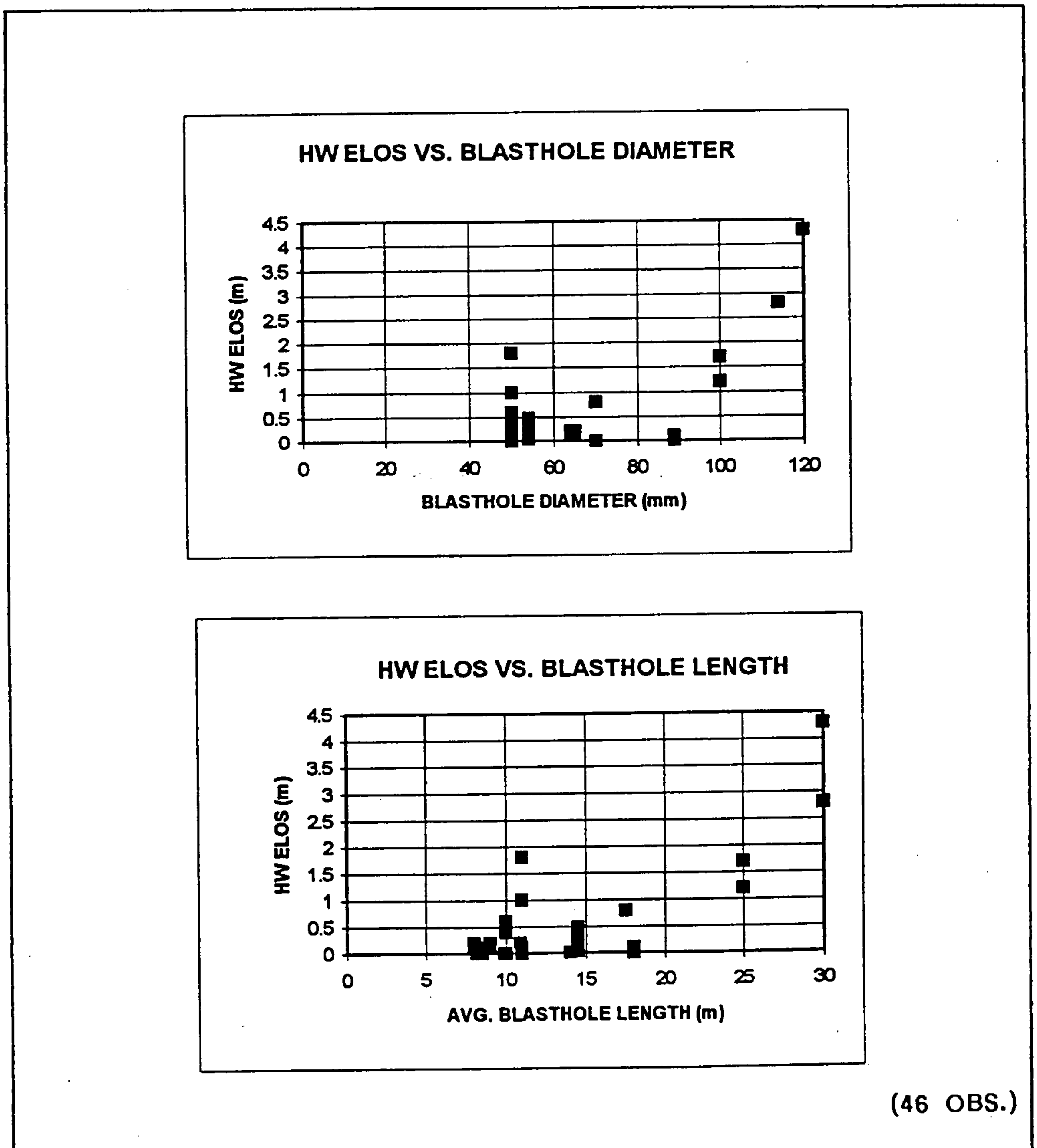


Figure 7.6 ELOS vs. Blasthole Diameter and ELOS vs. Blasthole Length

Good correlations are observed between ELOS and both blasthole diameter and blasthole length, which are themselves inter-dependent, as was demonstrated earlier (Figure 4.8, Chapter 4). This supports the fact that there is a greater potential for significant blast damage with larger diameter blastholes and that drillhole deviation associated with long blastholes can have an adverse effect on wall stability.

Figure 7.7 shows plots of ELOS versus: powder factor; blasthole offset; blasthole layout adjacent to the final wall (i.e. parallel blastholes; fanned; fanned/parallel); spacing to burden ratio; and number of blasts to excavate a stope.

No obvious correlation is observable between ELOS and powder factor, which suggests that it is not a reliable indicator of blast damage. Charge weight per delay and/or linear charge density would have likely been more appropriate parameters to record in the CMS database.

It is generally accepted that blastholes drilled parallel to the designed stope limits offer greater potential for less overbreak and a smoother final wall profile than blastholes which are fanned. This is not obvious from the scatter plot shown in Figure 7.7, however, the data points used for the scatter plot analysis are strongly biased towards parallel blastholes. There are an additional 8 stopes in the CMS database which were excavated with fanned/parallel blastholes but due to being supported with cable bolts are not shown in the scatter plot. Regardless, the database requires more stopes excavated using fans of blastholes before any quantitative conclusions can be drawn regarding overbreak potential of fanned versus parallel blastholes..

No correlation is observable between ELOS and spacing to burden ratio. This is not surprising since the parameter relates more to blast performance (i.e. number of misfires; and fragmentation).

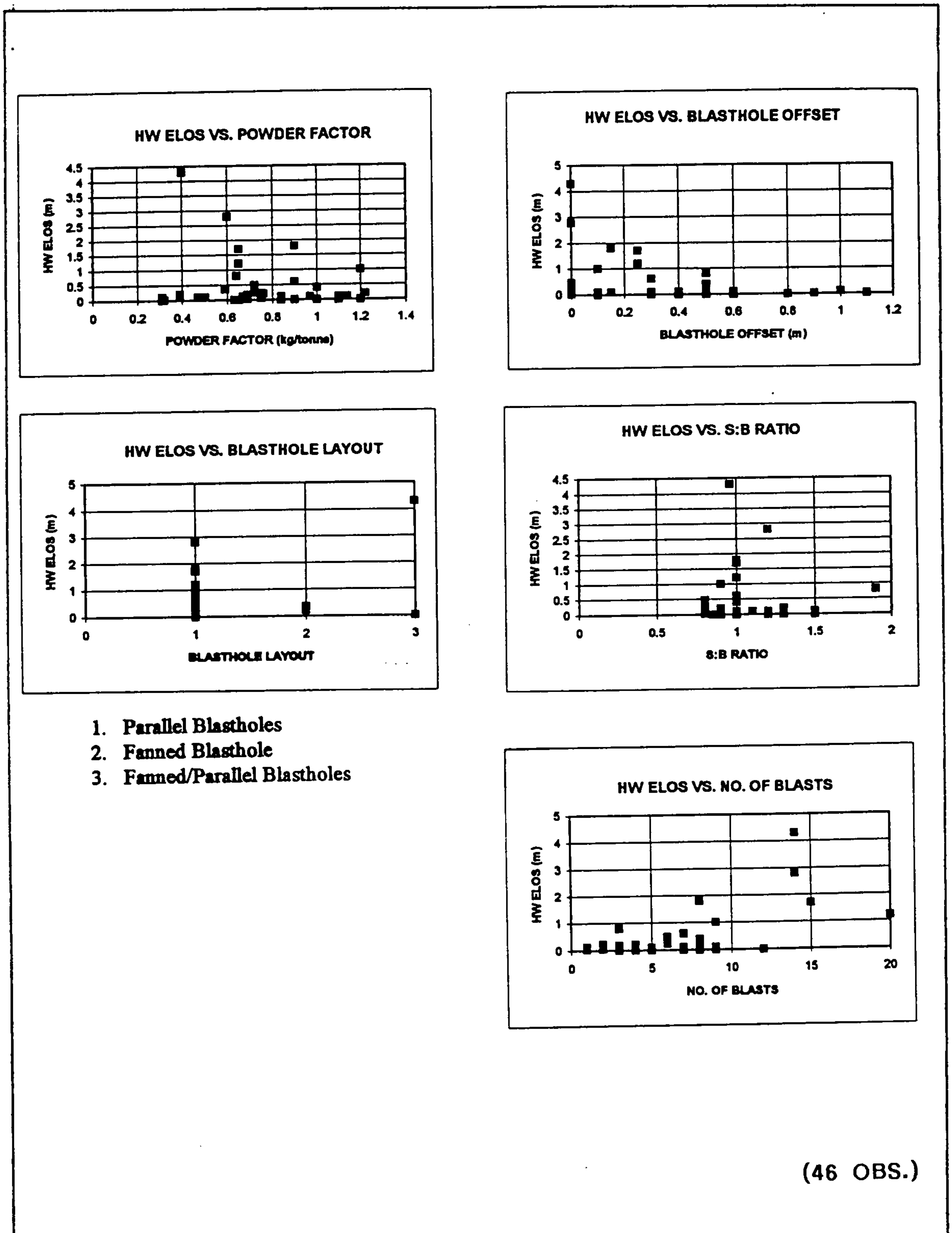


Figure 7.7 Scatter plots examining relationships between ELOS and various blasting related factors

To try and compensate for blast damage it is not uncommon to offset blastholes some distance from the designed stope limit, however, if the offset is too large ore may be left behind on the stope walls. An interesting correlation can be observed between ELOS and blasthole offset. The scatter plot shows that for offsets greater than 0.5m ELOS was almost always zero (0m), which usually indicates the stope has not completely broken to its planned limits. This is investigated further in Section 7.2.3.

A reasonable correlation can be observed between ELOS and the number of longhole blasts. This intuitively makes sense. Every blast subjects previously excavated portions of the stope wall to blast vibrations. This generally shakes down any pieces of rock that have loosened up since the last blast was taken and may initiate loosening of previously stable areas of the wall. It follows that stopes designed with a marginal level of stability will be more sensitive to the number of longhole blasts than a stope that is designed with a high initial level of stability. Furthermore, stopes that require a large number of blasts to excavate are generally open for a significant amount of time, which may in itself adversely impact stope wall stability.

Stope Support

To examine the effect of stope support on ELOS, the supported hangingwall data was plotted alongside the unsupported data used in the other scatter plots. Note that in all cases stope support consisted of cables installed from sub-levels (point anchor approach). RMR' was incorporated into the plots in an attempt to distinguish at what point the rock mass becomes so poor that cable bolts become ineffective (i.e. rock unravels around cables) and at what point the rock mass is so good that cables provide no benefit, refer to Figure 7.8.

Very little difference is observable between the supported and unsupported data. In other words, given the limited database of supported stope walls, and the simplistic analysis (i.e. no account is made for hydraulic radius of the stope walls), it is inconclusive whether the cable bolts had any beneficial effect with regard to stability. However, the analysis does agree with observations made underground. In most cases where sub-level cable support was used it appeared to have little impact with regard to controlling unplanned dilution between the sub-levels. The main benefit that the cables provided was that when a stope wall failed the cables stopped the failure from progressing up dip, preventing undercutting of the walls of the future overlying stope.

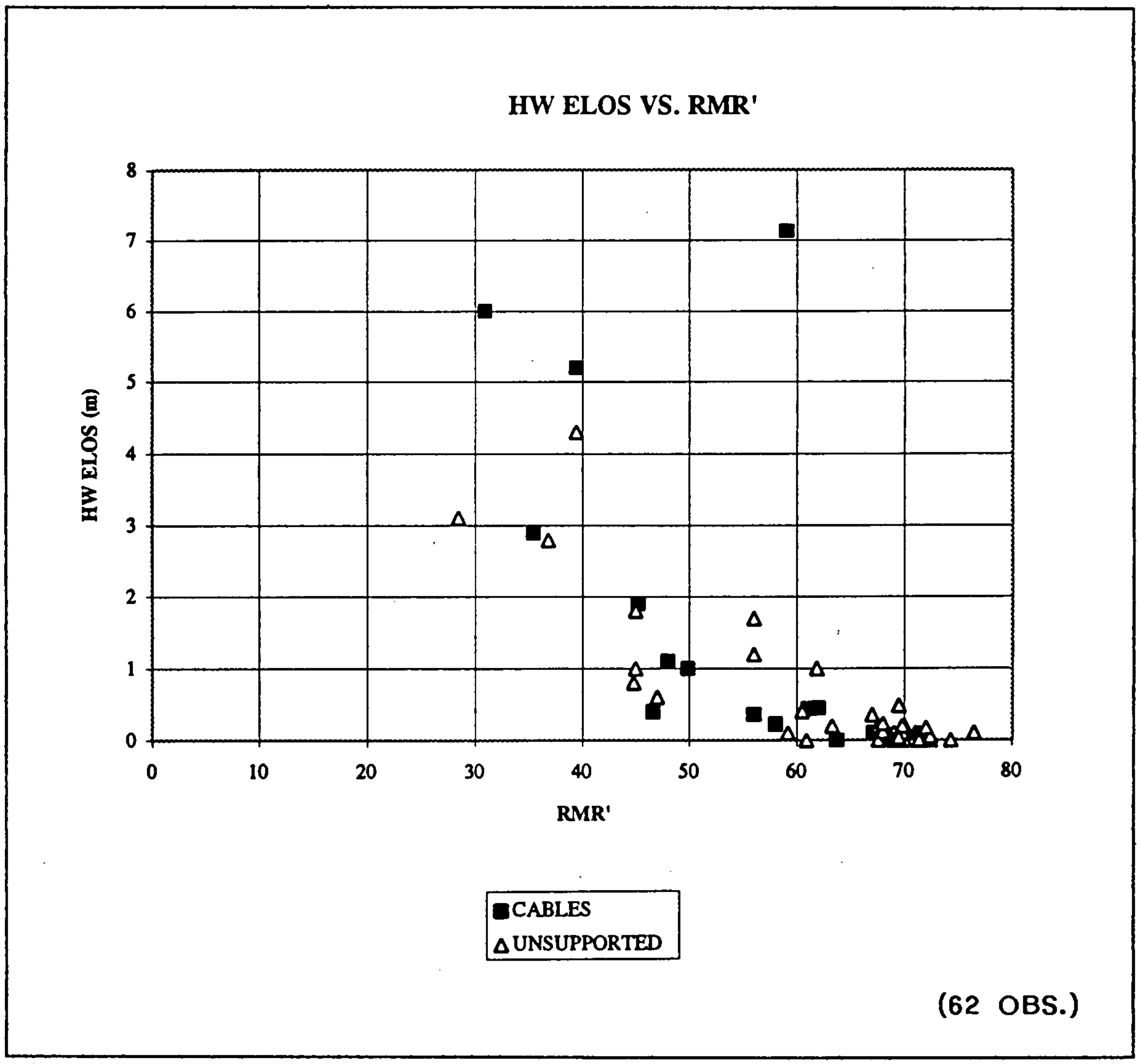


Figure 7.8 ELOS vs. RMR' showing data points from both supported and unsupported stope surfaces

Time

Figure 7.9 shows a plot of ELOS versus stope life (the time in days between the first stope blast and the stope survey).

Only a weak correlation can be observed between ELOS and stope life. A very tentative relationship is that stopes with a life greater than approximately 150 days all have ELOS values greater than 1.0m, however, the data set is very limited.

The time effect is very hard to quantify. In general, to minimize unplanned dilution, stopes should be mined as quick as practically possible. Stopes designed with a high initial level of stability will be less sensitive to the effects of time.

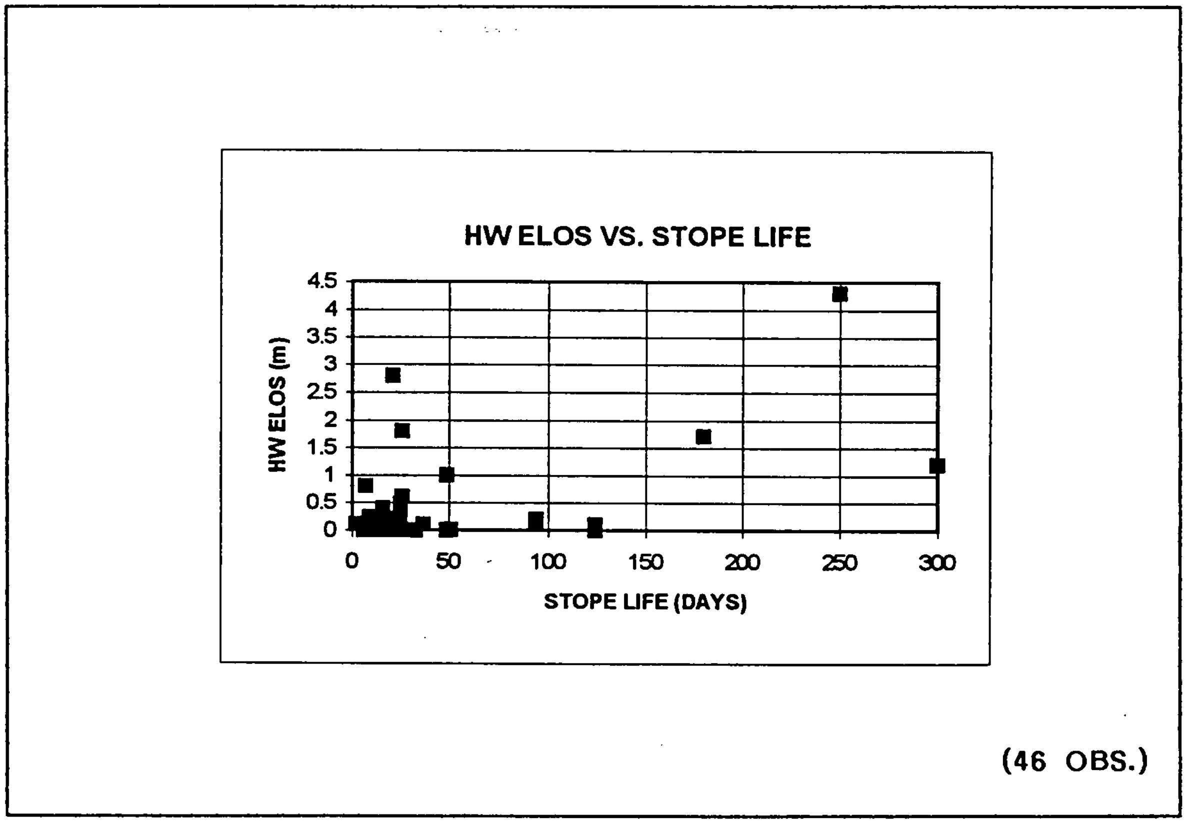


Figure 7.9 ELOS vs. Stope Life

7.2.3 ELLO vs. Blasthole Offset

An interesting relationship was observed between ELOS and blasthole offset (Section 7.2.2). To examine the effects of blasthole offsets in greater detail the relationship with ELLO (Equivalent Linear Lost Ore - Section 3.3, Chapter 3) was investigated. Figure 7.10 shows a plot of ELLO versus blasthole offset. Note that the data has been classified into ranges of RMR'. The linear lines are lines of linear regression pertaining to a range of RMR'.

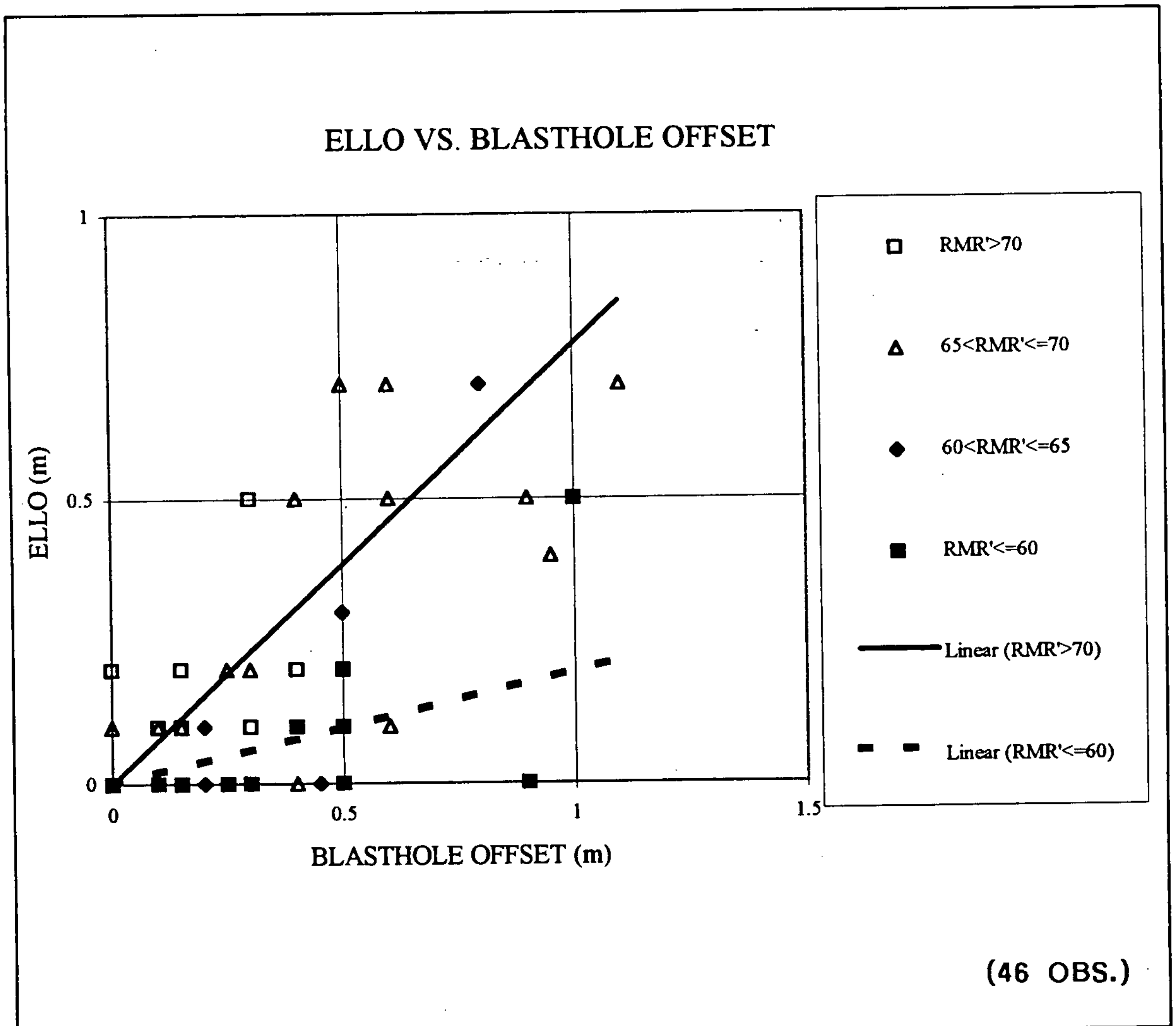


Figure 7.10 ELLO (Equivalent Linear Lost Ore) vs. Blasthole Offset

Comments regarding the plot are given below:

- the plot indicates a definite trend of increasing ELLO with increasing blasthole offset;
- the trend is much steeper for good quality rocks;
- out of all the cases where blasthole offset is greater than 0.5m only one has ELLO equal to zero (i.e. no ore left behind).

Figure 7.10 is not meant to be used for design since at this stage the database is limited. It is meant to demonstrate that blasthole offsets can result in considerable lost ore and that at the design stage careful consideration must be given to rock quality (i.e. small to no offsets in good quality rock, larger offsets in poor quality rock). CMS surveys should be used to verify that the designed blasthole offsets are appropriate.

7.2.4 Summary of Scatter Plot Analysis

The following summarizes the main findings from the scatter plot analysis:

- ELOS showed no obvious correlations with: depth; stope strike length; height/length ratio; powder factor; blasthole layout; spacing/burden ratio; and stope support.
- ELOS showed weak correlations with: stope height; hydraulic radius; and stope life.
- ELOS showed moderate to good correlations with: surface character; the Modified Stability Number (N'); RMR'; stope wall undercutting; blasthole diameter; blasthole length; blasthole offset; and number of stope blasts.

At this stage, given the simplistic analysis and the limited size of the database, the observed relationships between ELOS and the other database parameters should not be considered definite. In general, however, the findings make intuitive sense.

7.3 NEURAL NETWORK ANALYSIS

7.3.1 General

The neural network analysis was carried out using the same database used in Section 5.4, Chapter 5. The neural network database only includes ELOS measurements which have data recorded for all the CMS database categories. The neural network database contains 75 data sets (as opposed to 88 in the CMS database) and includes footwall surfaces which dip less than 85° and stope surfaces supported with cable bolts. The following analysis forms a portion of a M.Sc. thesis being written by Logan Miller Tait (University of British Columbia). A description of the analysis and a summary of the main findings are discussed in the subsequent sections.

7.3.2 General Method

Due to the limited size of the database, it was only considered appropriate to develop neural networks consisting of three input parameters (i.e. N' or RMR'; HR; and one other database parameter). To analyse more input parameters per neural network, a bigger database is needed if representative results are to be obtained.

The method used to evaluate the influence of other database parameters (i.e. parameters other than N', RMR', or HR) on predicting ELOS was as follows:

- the neural network database was divided into 60 data sets for training data and 15 data sets for testing data;
- neural networks consisting of three inputs (N' or RMR'; HR; and one other database parameter) were trained to a seven percent error using the training data;
- the trained neural networks were then used to try and predict the ELOS values contained in the test data set;
- the influence of the input parameters with regard to predicting ELOS was evaluated based on: the individual weightings given to the three input parameters during the

training stage; the R^2 value (coefficient of determination) pertaining to the prediction of the test data; and plots of the ELOS neural network predictions versus actual ELOS measurements. Typical results are presented in Figures 7.11, 7.12, and 7.13. Note that in the proceeding discussions the input weightings are normalized to a percentage to facilitate easier comparison between the different neural network runs.

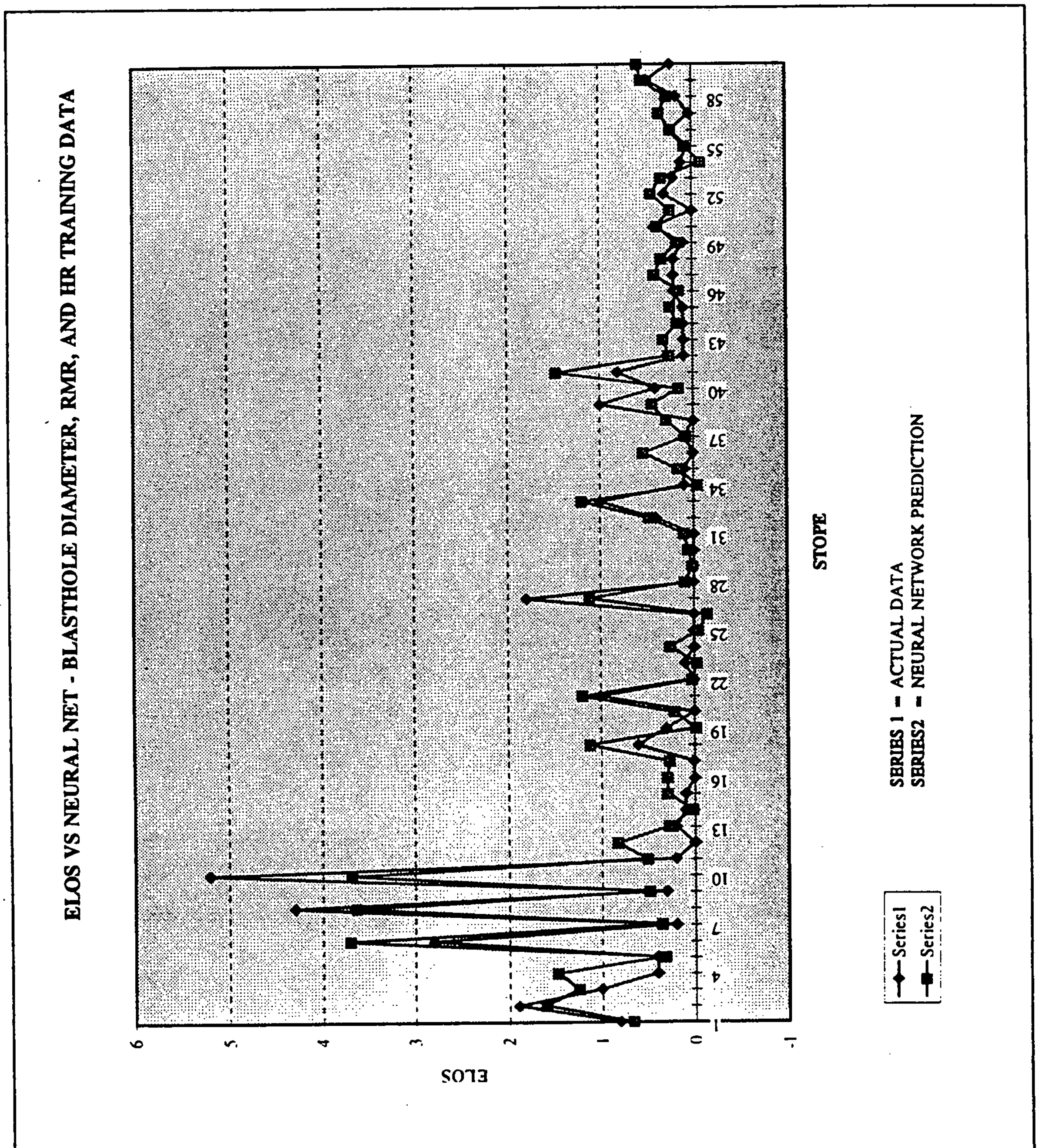


Figure 7.11 Typical plot showing training data - RMR'; HR; and Blasthole Diameter (linear scale processing)

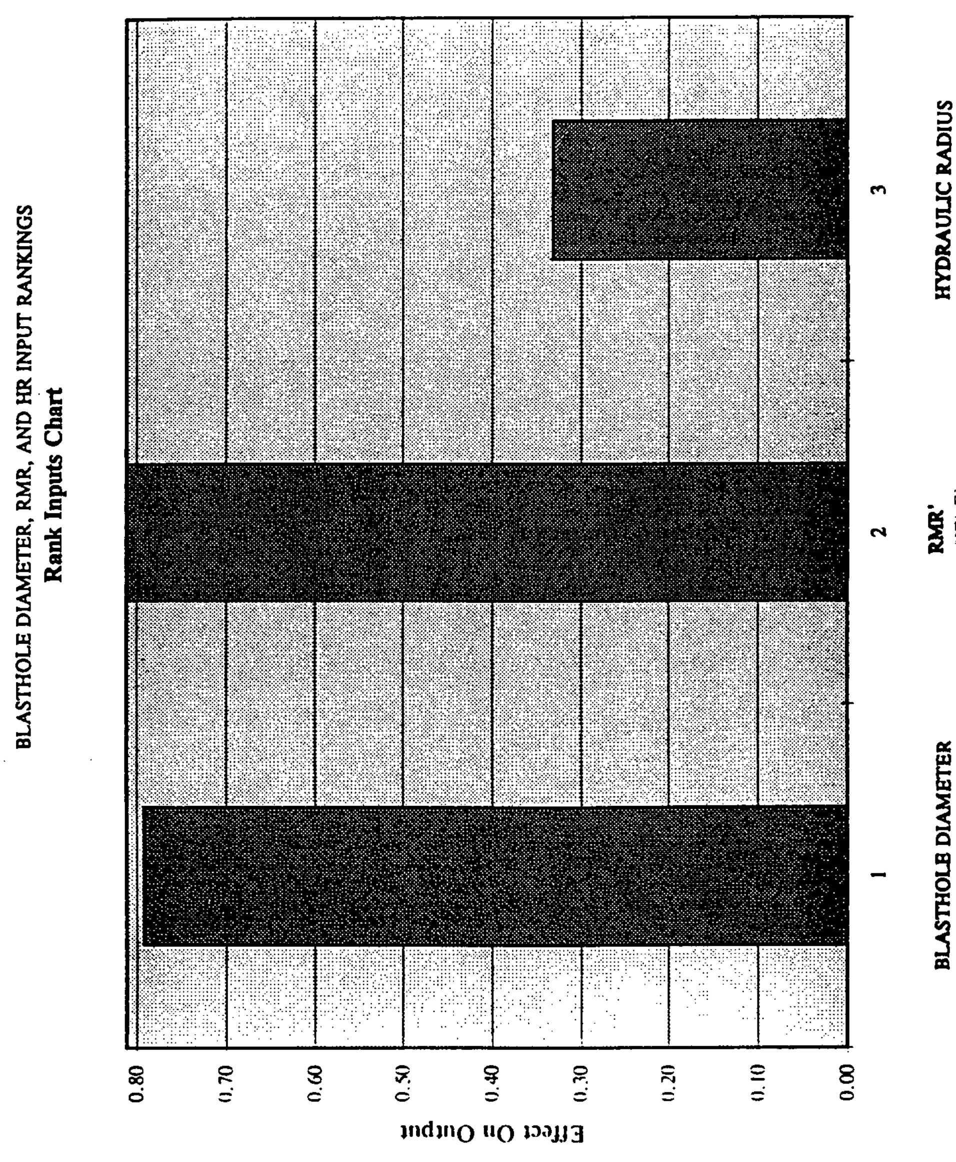


Figure 7.12 Typical plot showing input weightings developed during the training stage - RMR'; HR; and Blasthole Diameter (linear scale processing)

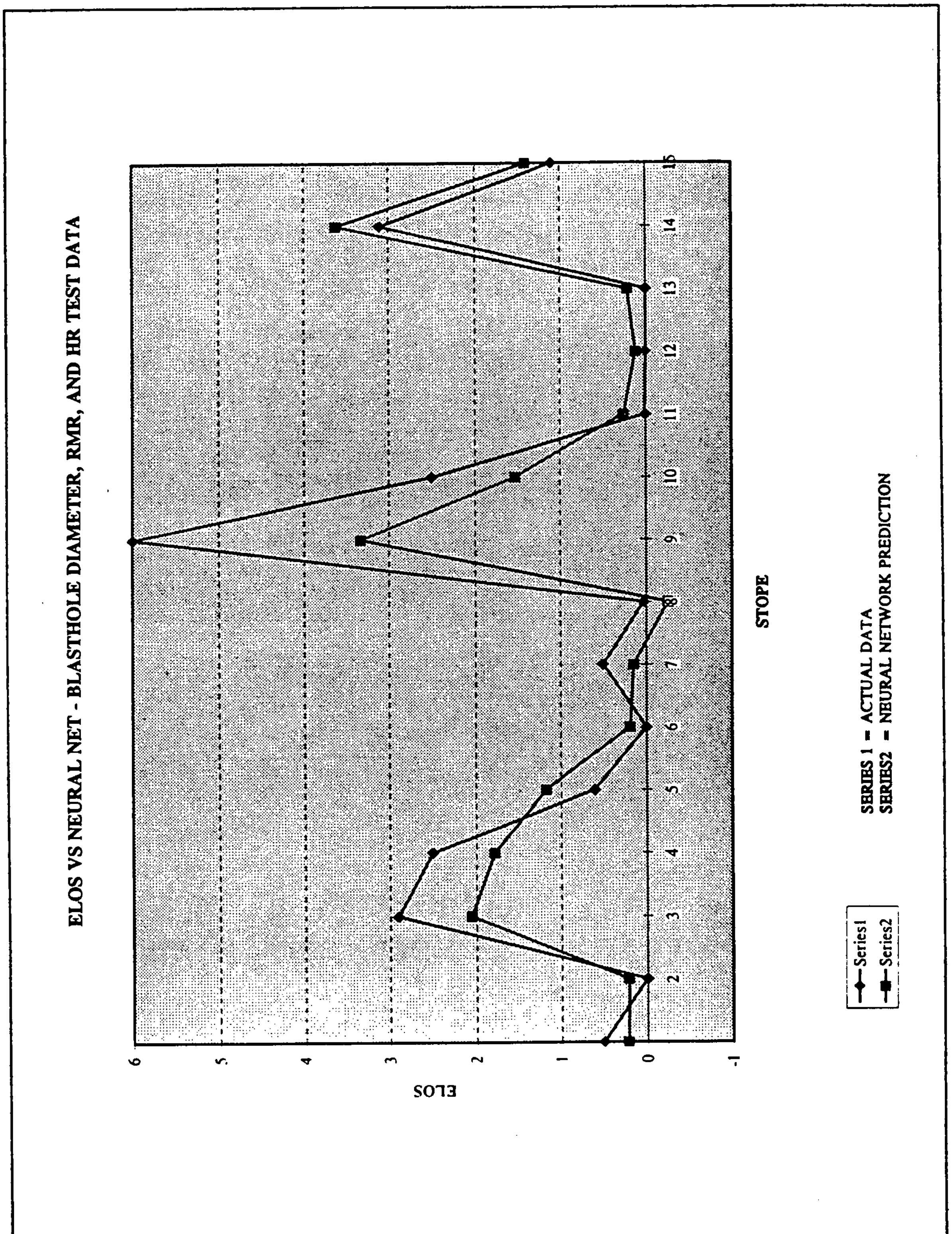


Figure 7.13 Typical plot of test data showing neural network predictions vs. actual ELOS measurements - RMR'; HR; and Blasthole Diameter (linear scale processing)

7.3.3 Description of Neural Networks

Review of Previous Neural Network Results

The neural network analysis presented in Section 5.4 (Chapter 5) showed that when log scale processing was used with two input parameters (RMR' or N' and HR) that RMR' gave a higher R^2 correlation than N' with regard to predicting ELOS (i.e. 0.76 vs. 0.52). Out of curiosity, a three input neural network containing RMR', N', and HR, was run using log scale processing. RMR' was given a much higher weighting than N' (i.e. N'- 19%; RMR'- 49%; HR - 32%). The R^2 value was slightly lower than the two input neural network containing just RMR' and HR (0.75 vs. 0.76).

The neural network analysis presented in Section 5.4 (Chapter 5) also showed that when linear scale processing was used the N' neural network could not be trained to a minimum seven percent error, whereas the RMR' neural network could. Although, the R^2 correlation was much lower than that achieved when using log scale processing (i.e. 0.13 vs. 0.76). It should be noted however, that the R^2 value can sometimes be misleading. Since its value is based on squared differences, a small number of outlier points (poorly predicted points) can have a large influence on its value. For example, often the neural networks were very accurate on predicting the test data except for 1 or 2 points. If the difference between the neural network prediction and the actual measured ELOS was quite large (i.e. $ELOS_{prediction} = 10$ vs. $ELOS_{actual} = 3$) a low R^2 value would result. It was for this reason that plots of ELOS predictions versus actual ELOS measurements were made for each neural network as opposed to relying solely on the R^2 value.

Neural Networks Developed for this Analysis

Based on the above, three sets of 18 neural networks were developed:

1. log scale processing: N'; HR; and one other database parameter;

2. log scale processing: RMR'; HR; and one other database parameter;

3. linear scale processing: RMR'; HR; and one other database parameter.

The additional database parameters investigated with each of the three sets of neural networks include: depth; orebody width; stope strike length; stope height; stope height to length ratio; stope surface character; undercutting; blasthole diameter; blasthole length; blasthole diameter to drill string diameter ratio; spacing to burden ratio; blasthole offset distance; powder factor; blasthole layout adjacent to the final wall; cable support support; stope life; number of longhole blasts to excavate stope; and whether the stope was in the vicinity of other mining.

7.3.4 Presentation and Discussion of Neural Network Results

Both the N' and RMR' log scale processed neural networks did not give promising results. In both cases the other database parameters were given relatively low input weightings in comparison to N' or RMR' and HR, suggesting that they did not greatly influence the neural network results. Based on these analysis it was concluded that the data ranges recorded for the additional database parameters were not suitable for log scale processing.

The linear scale processing gave much more promising results. While the overall R^2 correlations with the test data were on average lower than with the log processed neural networks, the input factors were given a greater weighting and, in general, an increase in weighting correlated with an increase in the R^2 value. Table 7.1 shown below summarizes the main results from the neural network analysis. Note that the highlighted sections correspond to runs where the additional database parameter was given a relatively high weighting and a significant improvement in the R^2 value resulted. A graphical presentation of the input weightings is presented in Figure 7.14.

Table 7.1
Summary of Neural Network Analysis

Two Input Parameter Results (N' or RMR' and HR)

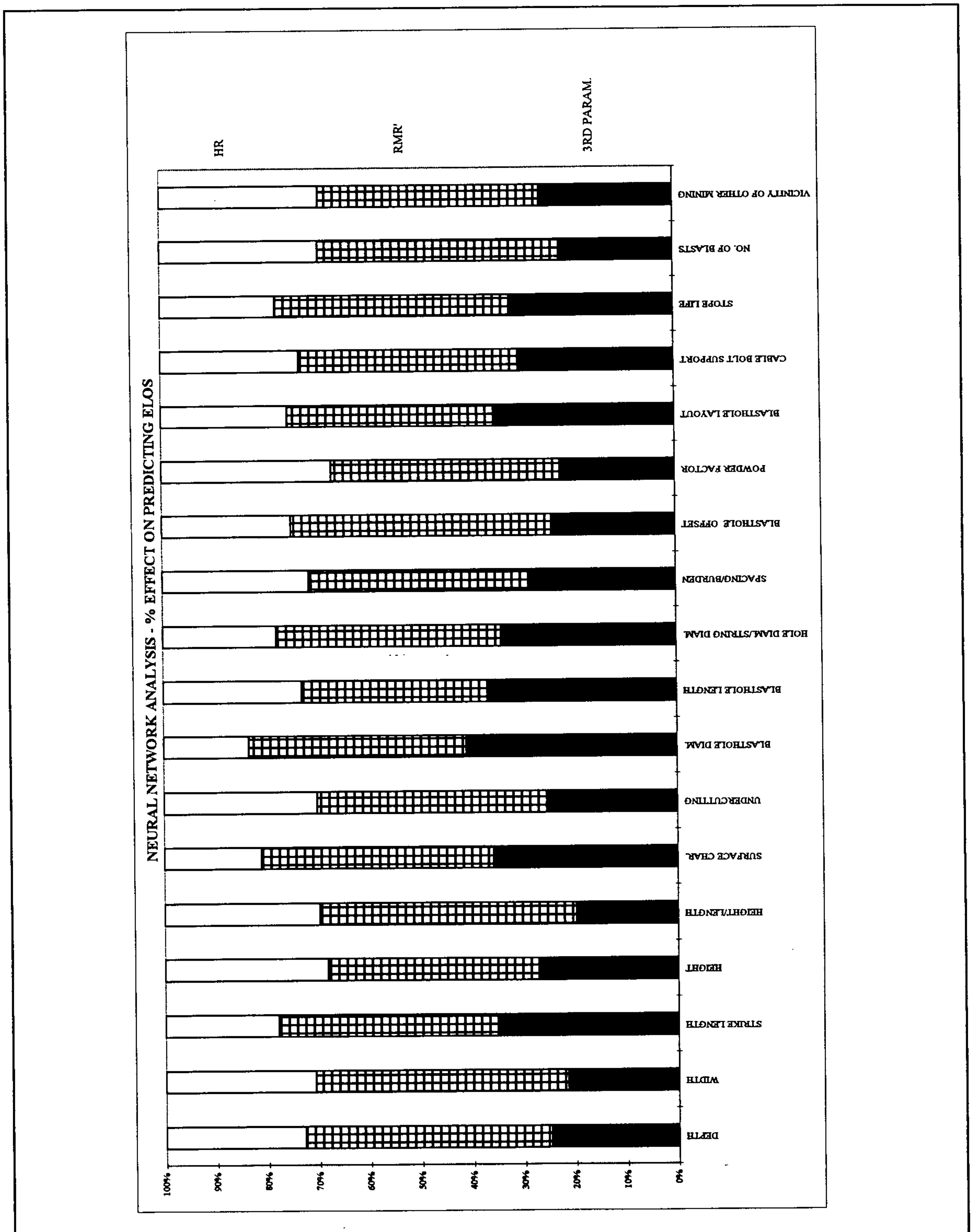
Log N' - Wt.	Log HR - Wt.	R^2 Coefficient
80%	20%	0.52
Log RMR' - Wt.	**Log HR - Wt.**	**R^2 Coefficient**
59%	41%	0.76
Linear RMR' - Wt.	**Linear HR - Wt.**	**R^2 Coefficient**
57%	43%	0.13

Three Input Parameter Results (N', RMR', & HR)

Log N' - Wt.	Log RMR' - Wt.	Log HR - Wt.	R^2 Coefficient
19%	49%	32%	0.75

Investigation into Database Parameters Which May Influence Estimated ELOS
Three Input Parameters (RMR'; HR; Database Parameter)

LINEAR SCALE PROCESSING	PARAM. Wt.	RMR' Wt.	HR Wt.	R^2 Coefficient
DEPTH	25%	48%	27%	0.31
WIDTH	22%	49%	29%	0.41
STRIKE LENGTH	35%	43%	22%	0.22
HEIGHT	27%	41%	32%	0.26
HEIGHT/LENGTH	20%	50%	30%	0.23
SURFACE CHAR.	36%	45%	19%	0.42
UNDERCUTTING	25%	45%	30%	0.14
BLASTHOLE DIAM.	41%	42%	17%	0.83
BLASTHOLE LENGTH	37%	36%	27%	0.61
BLASTHOLE DIAM. / DRILL STRING DIAM.	34%	44%	22%	0.28
SPACING/BURDEN	29%	43%	29%	0.21
BLASTHOLE OFFSET	24%	51%	25%	0.22
POWDER FACTOR	22%	45%	33%	0.46
BLASTHOLE LAYOUT	35%	41%	25%	0.73
CABLE BOLT SUPPORT	30%	43%	27%	0.22
STOPE LIFE	32%	46%	22%	0.56
NO. OF BLASTS	22%	47%	31%	0.23
VICINITY OF OTHER MINING	26%	43%	31%	0.34



7.14 Graphical representation of input weightings from 18 neural network analysis - Three inputs per neural network: RMR'; HR; and one other database parameter (linear scale processing)

7.3.5 Summary of Neural Network Analysis

The neural network analysis results are summarized below:

- RMR' was determined to have a stronger correlation with ELOS than N';
- with regard to the database parameter analysis, in all cases but one, RMR' had the strongest influence with regard to predicting ELOS. The exception was when blasthole length was included as an input, in this case RMR' and blasthole length had roughly equivalent influence with regard to predicting ELOS;
- of the database parameters other than RMR' and HR, the following were found to have a notably strong influence with regard to predicting ELOS and resulted in a significantly improved R^2 value: stope surface character (i.e. doglegs along dip or strike); blasthole diameter; blasthole length; blasthole layout (i.e. parallel, fanned; parallel/fanned); and stope life. In addition, all of the above parameters were weighted more heavily than hydraulic radius with regard to predicting ELOS;
- the neural network which consisted of: RMR'; HR; and blasthole diameter (linear scale processing), had the highest R^2 value of any of the neural networks run as part of this analysis (includes both log scale processed and linear scale processed neural networks).

The neural network analysis agrees reasonably well with the results obtained from the scatter plot analysis. In both the scatter plot analysis and the neural network analysis, strong correlations with blasthole diameter and blasthole length have been identified. It is envisioned that these factors will eventually be incorporated into the stability graph method of stope design. In addition, both the scatter plot analysis and the neural network analysis indicated that stope surface character and stope life have an influence on ELOS.

It was surprising that the neural network incorporating undercutting did not show a stronger influence with regard to predicting ELOS, since it is well accepted that this factor has a significant impact on wall stability, especially in poorer quality rocks. Perhaps the fact that the majority of the stope walls in the database were undercut to some degree reduced the effectiveness of this parameter with regard to helping predict ELOS. This may be why RMR' and HR were given the majority of the weighting and the reason for only a very minor improvement in the R^2 value over the two input parameter neural network (RMR' and HR).

It is expected that as more data is collected and the CMS database is expanded, neural networks will be a very valuable tool for refining the design approach presented in Chapter 5.

7.4 CHAPTER SUMMARY - FACTORS WHICH MAY INCREASE THE ESTIMATED ELOS

Based on the analysis presented in Sections 7.2 and 7.3, a number of factors, other than hydraulic radius and the factors accounted for in N' and RMR', have been identified which potentially influence the stability of open stope hangingwalls and footwalls. The following is a summary of these factors and rough guidelines that can be used to help determine if the ELOS estimated using the design charts in Chapter 5 will be optimistic (i.e. low).

- *Irregular Wall Geometry* - most of the stopes in the CMS database have regular wall profiles (i.e. relatively planar surfaces). Based on the irregular and complex wall profiles contained in the CMS database, the scatter plot analysis and the neural network analysis provide support to the idea that ELOS will tend to increase as the regularity of the wall geometry decreases.

- *Undercutting of Stope Walls* - the scatter plot analysis showed that the destabilizing effect of undercutting is somewhat dependent on the stability number (i.e. RMR' or N'). Stope walls with an RMR' < 50 (N' < 4) appear to be very sensitive to undercutting. An ELOS value equivalent to the undercut depth should be anticipated for rock masses with stability numbers lower than these values.

- *Blasthole Diameter and Blasthole Length* - Both the scatter plot and neural network analysis suggest a relationship of increasing ELOS with increasing blasthole length and blasthole diameter. The database is largely comprised of stopes that were excavated using blastholes less than 65mm diameter and lengths less than 20m. The design charts presented in Chapter 5.0 may underestimate ELOS where large blastholes of length greater than 20m are used.

- *Blasthole Layout* - the neural network analysis indicated a relationship between ELOS and blasthole layout. This indicates that there may be a tendency for ELOS to increase when using fanned blastholes as opposed to parallel blastholes. The majority of blastholes in the database were drilled parallel to the planned stope surface.

- *Blasthole Offset* - The majority of perimeter blastholes were offset 0 - 0.5m from the excavation perimeter, with the average being approximately 0.3m. The design charts presented in Chapter 5.0 may underestimate ELOS if small or no offsets are used in rock masses with lower stability numbers (i.e. RMR' < 60 or N' < 12). This effect will likely increase with increasing blasthole diameter.

- *Stope Life and Number of Stope Blasts* - The scatter plot analysis showed a correlation between ELOS and the number of stope blasts and both the neural network scatter plot analysis suggested a correlation between ELOS and stope life. It is expected that as stopes plot progressively below the *Blast Damage Only Zone* shown in Figures 5.11 and 5.18 (Chapter 5), wall stability will become increasingly sensitive to these parameters. The majority of the stopes in the database were open for less than 50 days and were excavated with less than 9 longhole blasts.

CHAPTER 8

STOPE DESIGN SUMMARY

AN EMPIRICAL DESIGN APPROACH FOR ESTIMATING UNPLANNED DILUTION FROM OPEN STOPE HANGINGWALLS AND FOOTWALLS

8.1 FINALIZING THE EMPIRICAL DESIGN APPROACH

Finalized versions of both the N' and RMR' stability graphs incorporating the ELOS design zones are presented in Figures 8.1 and 8.2, respectively. Areas of low design confidence are indicated by dashed lines and represent zones of limited data in the CMS database. A chart which relates: unplanned dilution; ELOS; and stope width, is included with each design chart to aid in designing stopes based on achieving a certain level of unplanned dilution. Formal definitions for each of the design zones are presented below:

Blast Damage Only (ELOS < 0.5m)

- Potential for minimal overbreak/slough. The quantity of overbreak/slough will be highly dependent on the quality of drilling and blasting.
- Surface is self supporting, no support is required to maintain a stable excavation.
- Time is expected to have a minimal effect with regard to stability.

Minor Sloughing (ELOS = 0.5m - 1.0m)

- If the surface is unsupported some wall failure should be anticipated before a stable configuration is reached.
- Stope support should be considered. The CMS database provides some evidence that sub-level cable support may be adequate in this design zone. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Wall stability may be sensitive to blasting vibrations and the effects of time. Stopes should be mined quickly and filled.

- Minor operational problems should be anticipated (i.e. some secondary blasting).

Moderate Sloughing (ELOS = 1.0m - 2.0m)

- If no stope support is installed, significant wall failure should be anticipated before a stable configuration is reached.

- Stope support should be considered. The CMS database provides some evidence that in this design zone sub-level cable support may provide little benefit with regard to controlling unplanned dilution in rock masses with RMR'< 55 (N'< 6). If feasible, pattern support should be installed. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Stope stability will be sensitive to blast vibrations and the effects of time. Stopes should be mined quickly and filled.

- If adequate support is not installed, significant operational problems should be anticipated (i.e. secondary blasting, plugged drawpoints, possible ore loss under sloughed material, erratic production)

Severe Sloughing / Possible Wall Collapse (ELOS > 2m)

- Potential for large and possibly unacceptable wall failures.

- Pattern support should be considered but may provide limited benefit. Sub-level cable support will likely provide little benefit. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Stope stability will be very sensitive to blast vibrations and the effects of time. Stopes should be mined quickly and filled.

- If stope surfaces cannot be adequately supported, significant operational problems should be anticipated (i.e. secondary blasting, plugged drawpoints, ore loss, erratic production, possible loss of stope)

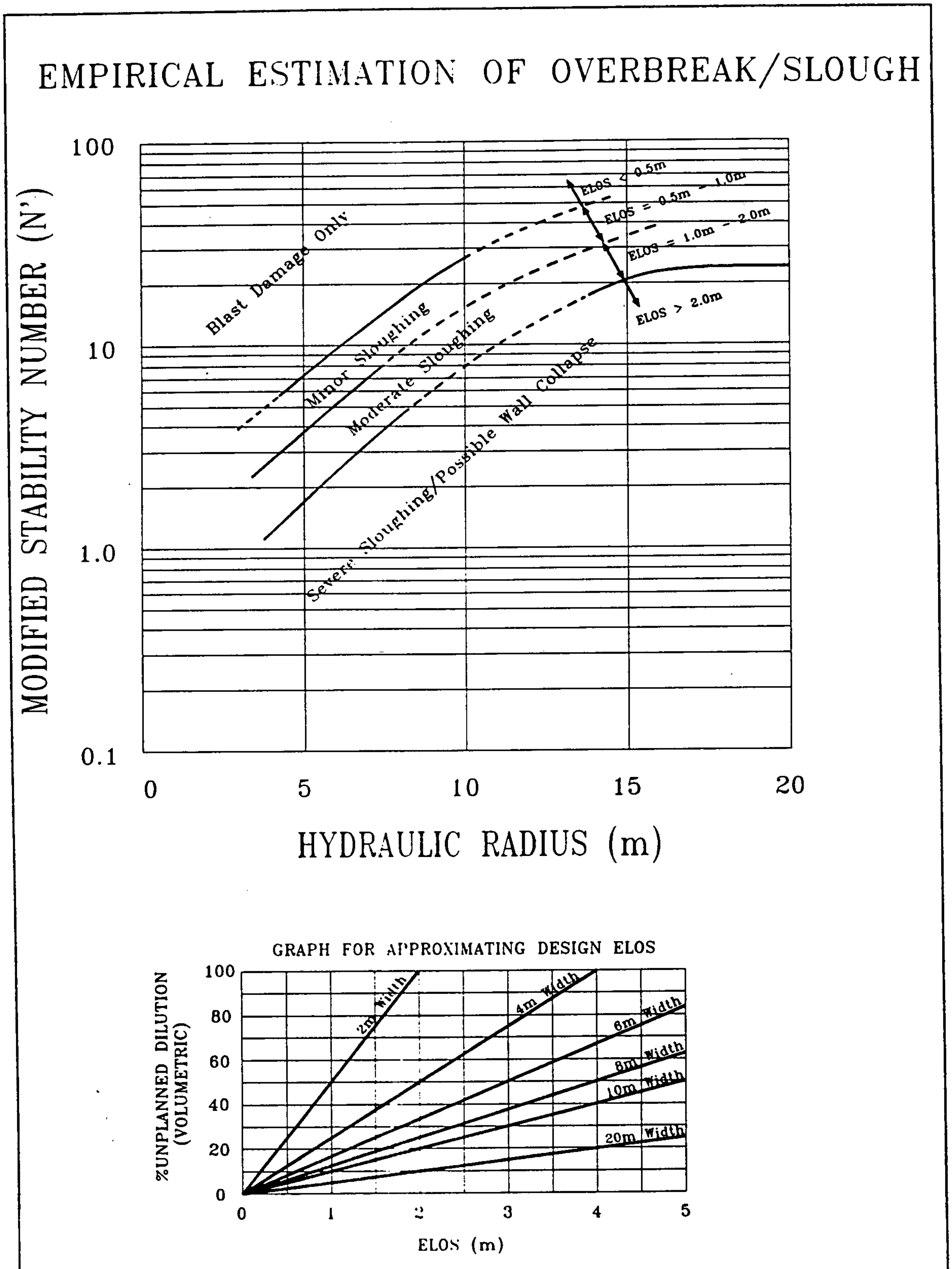


Figure 8.1 Finalized N' vs. HR stability graph incorporating ELOS design zones

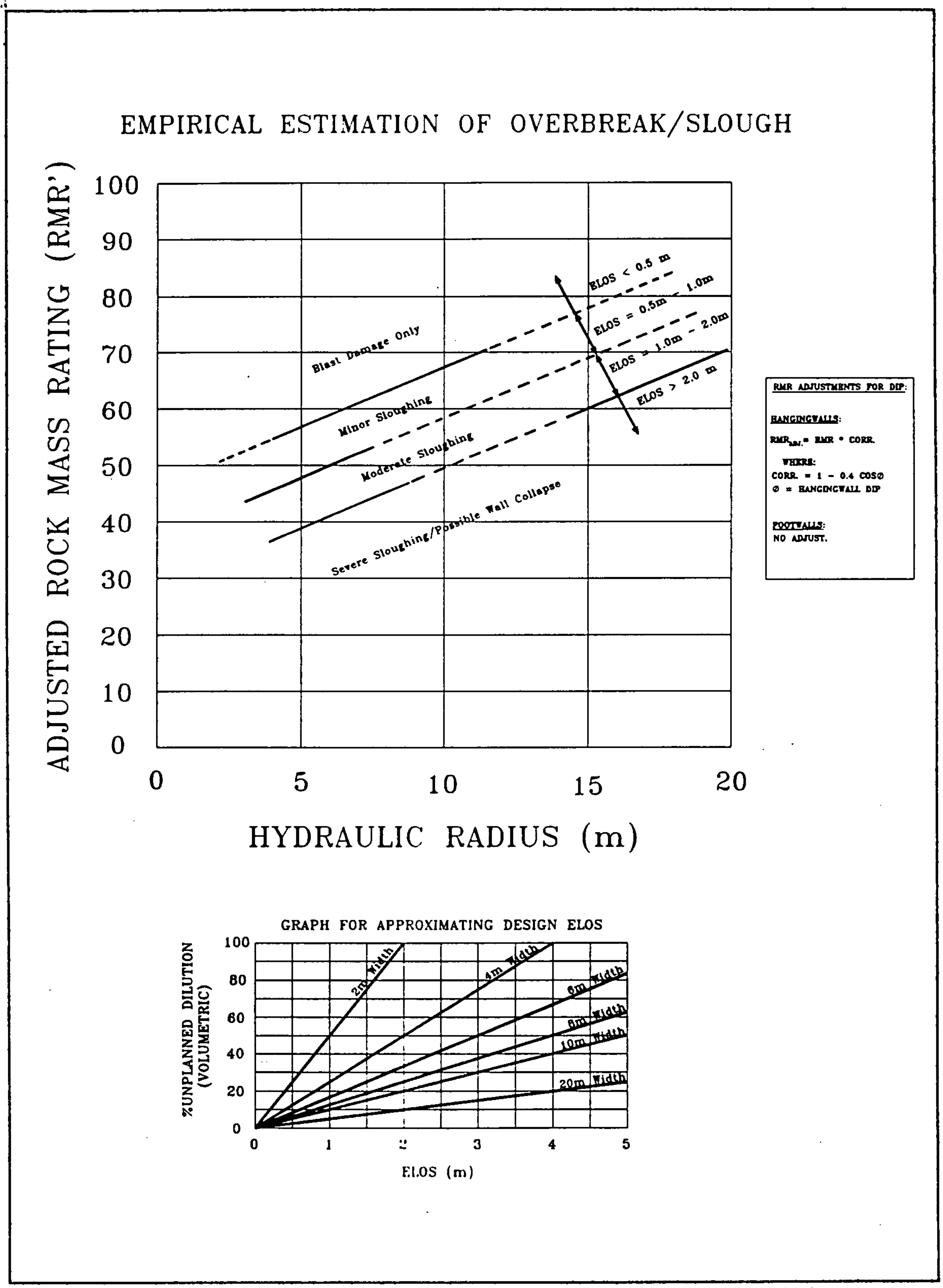


Figure 8.2 Finalized RMR' vs. HR stability graph incorporating ELOS design zones

A point to consider when using this design approach is that not all of the estimated overbreak/slough will necessarily be mucked out of the stope. Often the sloughed material sits on top of the blasted rock and is slowly drawn down to the mucking level and left behind in the stope, usually resulting in some ore loss. Other times, however, the slough preferentially finds its way to the drawpoint or slough material gets mixed in with the ore and everything gets mucked to the ore pass. This is influenced by factors such as drawpoint location and production requirements. This is a difficult factor to account for and is probably best handled on a mine by mine basis. At the design stage it is probably best to assume that all the estimated overbreak/slough will find its way to the ore pass.

Example stope design applications are presented in Appendix II.

8.2 LIMITATIONS OF THIS DESIGN APPROACH

Limitations of this design approach are as follows:

- the main limitation is the size of the database. More data is required to give confidence to the design zones and to refine the method;
- the method is limited to hangingwalls and footwalls in a low or relaxed stress state with parallel structure being critical with regard to stability;
- the ELOS design zones apply to unsupported hangingwalls and footwalls, more data is required to quantitatively address the effect of stope support;
- the database is biased towards mines that use relatively small diameter blastholes (i.e. <65mm). More data is needed from mines that utilize larger diameter blastholes;
- the database needs more large stopes and more stopes in poor quality rock;
- additional factors which influence stope stability have been identified but only broad guidelines regarding their influence on estimating ELOS have been given (Chapter 7).

8.3 SUMMARY

An empirical design approach has been presented that can be used to estimate the amount of overbreak/slough from open stope hangingwalls and footwalls. This in turn can be used to estimate the unplanned dilution associated with a particular design. The method is based on quantifiable measurements of overbreak/slough made with the Cavity Monitoring System (CMS).

Additional factors which influence hangingwall and footwall stability, but not presently incorporated in the design approach, have been identified through analysis of the CMS database using scatter plots and neural networks. Broad guidelines regarding their influence on the empirical design approach have been given.

The main limitation of the design approach is the size of the database. Additional data is required to verify the design zones and refine the approach. It must be recognized as with all empirical design approaches that the limitation of the predictive solution is largely governed by the existing database and how past observations relate to present input parameters in terms of predicting future behavior.

CHAPTER 9

INTRODUCTION

ASSESSMENT OF NARROW VEIN LONGHOLE BLAST PATTERNS AT THE LUPIN MINE

9.1 INTRODUCTION

From the preceding chapters concerning stope design it can be shown that even if stopes are sized such that they plot in the "Blast Damage Only" zone, the resulting ELOS may still be as high as 0.5m. For narrow vein deposits this relatively low ELOS value can represent significant unplanned dilution. For example, consider a 2m wide orebody, if 0.5m of blast induced overbreak is realized from both the hangingwall and footwall, the volumetric unplanned dilution amounts to 50%. It can be appreciated from this example that minimizing blast induced overbreak is extremely important in narrow vein mining.

This study examines the pros and cons of various narrow vein longhole blast patterns with regards to factors such as: charge interaction; overbreak potential; fragmentation; drilling and blasting costs; and the importance of quality control during the drilling and blasting process. The study was conducted at the Lupin Mine (Echo Bay Mines Ltd.) located in the Northwest Territories. For the purposes of this study, the term "narrow vein" applies to orebodies with widths less than or equal to 2.5m.

The following sections provide background information concerning the Lupin Mine and describe the historical development of the narrow vein blast patterns used at the mine. The objectives of the study, specific to the Lupin Mine, are presented at the end of the chapter.

9.2 LUPIN MINE- GENERAL DESCRIPTION

The Lupin Mine owned and operated by Echo Bay Mines Ltd., lies about 80km south of the Arctic Circle near the northern end of Contwoyto Lake (NWT), refer to Figure 9.1.

In plan, the Lupin orebody is "Z" shaped consisting of three main ore zones: the Center Zone (avg. 7m width); West Zone (avg. 1.5-2.0m width); and the East Zone (avg. 7m width), refer to Figure 9.2. The Center and West Zones dip steeply to the east at 75-90°, while the East Zone dips steeply to the west at 75-90°. Gold mineralization occurs in a sulphide bearing banded iron formation.

Mining is carried out using primarily longhole open stoping methods. The current rate of production is approximately 2000 stons/day. The Center Zone produces 50% of the daily tonnage, the East Zone produces 10%, and the West Zone provides the remaining 40%.

The focus of this study is on the longhole blast patterns used in the narrow West Zone.

9.3 WEST ZONE MINING

9.3.1 General Mining Method

The mining method used in the West Zone is sub-level retreat with paste backfill. A schematic depicting the main aspects of the mining method is shown in Figure 9.3. Note that the schematic shows the blastholes to be all breakthrough downholes, however, at present the majority of stoping blocks are drilled off using a combination of shorter length up and downholes. Trials using all breakthrough downholes are currently in progress.

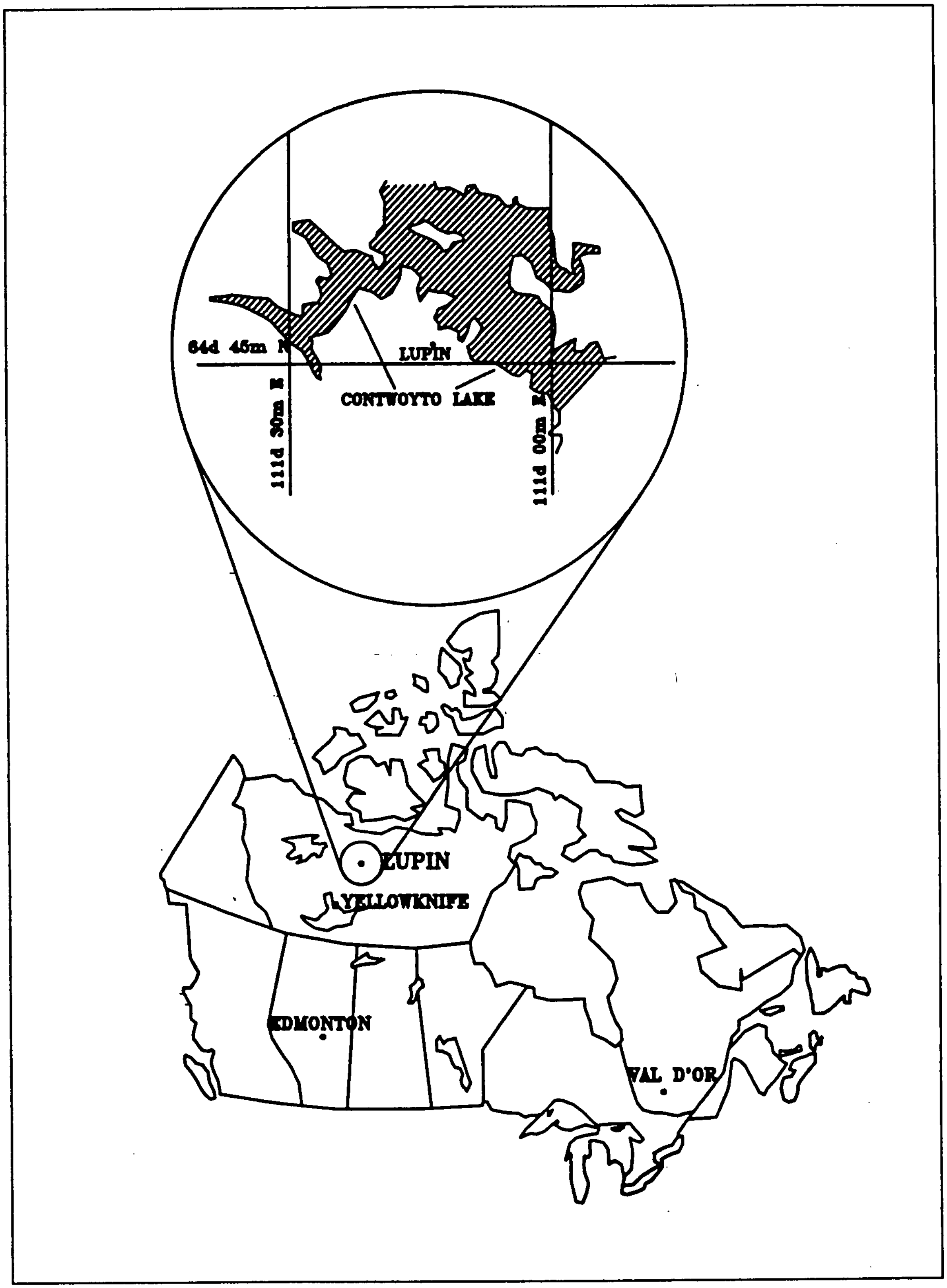


Figure 9.1 Location of Lupin Mine

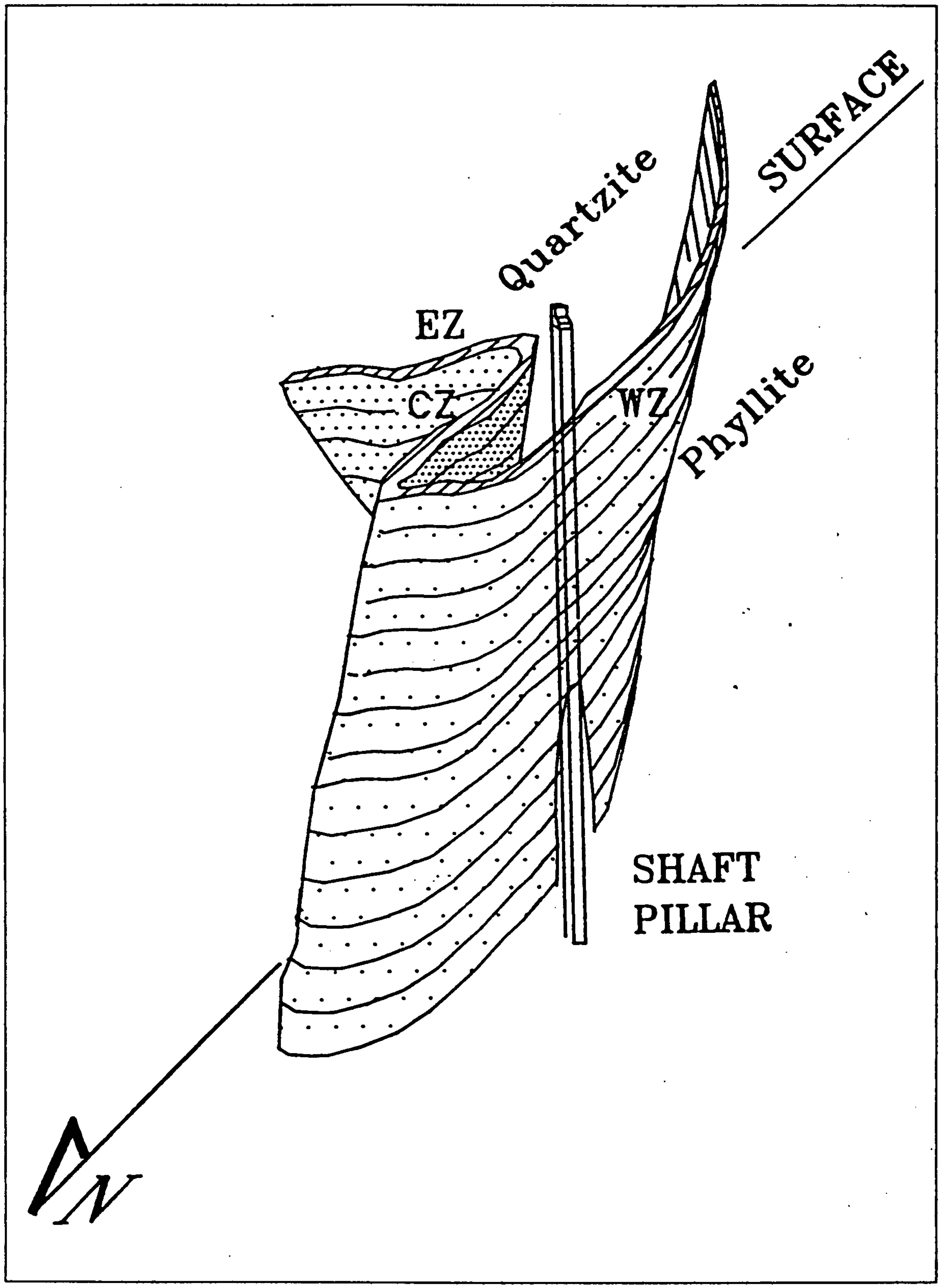


Figure 9.2 Schematic of Lupin Orebody

9.3.2 Mining Method Details

The sub-level interval in the West Zone is approximately 20m. Sub-level access is gained by driving a drift to the midpoint along strike of the ore zone. Ore drifts are driven north and south from the access. Ore drifts are driven a minimum of 2.1m wide by 3.4m high using Tamrock Micromatic single boom jumbos and Wagner 2cu.yd. scoops. Because of the narrow width of the ore drifts, safety bays are mandatory every 30m. The ore drifts are driven under geological control and an effort is made to have the ore centered in the face. West Zone ore width is on average 1.8m, however, it may vary from less than 1m to 4m in thickness. Ore drifts are typically 500-600m in length (north and south combined) with a mineable strike length of 350-400m. This yields approximately 2000 to 2400 stons/vertical meter.

Due to the narrow width of the ore, undercutting of one or both of the stope walls is unavoidable during ore drift development. This can adversely effect stope stability depending on the amount and type of rock undercut.

After drifting is complete, the geology department channel samples the drift on 2.5m centers. The channel samples along with drift mapping are used to determine the ore contacts on each level.

Longhole rings (parallel holes) are laid out using the ore contacts as a guide. The minimum stoping width is 1.5m. Longhole layouts utilize both vertical upholes (i.e. no dump angle) and downholes of approximately 10m length (upholes and downholes are overlapped 1.5m). A 3:2 blasthole pattern is used. Ring spacing (burden) is 0.75m. Typical blasthole spacing to burden ratios vary between 0.8 and 1.3. Tonnage per meter drilled is approximately 2 stons/m.

Longhole drilling is done using Tamrock MicroSolo drills. Blastholes are 50mm (2 inches) in diameter. Drill steel is 1.2m long (R32) with male/female connections. A single guide tube is used directly behind the bit for all holes. Since blasting is done utilizing up and downholes, no breakthrough holes are drilled. Thus drilling accuracy is difficult to check.

Blasting starts from the extreme north and south ends of a stoping panel. Stopes are typically mined 20m high with a strike length of 15-20m, after which they are filled with paste backfill. Once the paste has set, a slot is created through the paste for blasting of the next stope. Both the north and south stopes are retreated back to the center access.

Anfo is the primary explosive. The average powder factor is 0.9-1.0 kg/ston. Typical blast size is 500-1000 stons.

Broken ore is mucked from the ore drift using remote 2 cu.yd. scooptrams.

9.4 GEOTECHNICAL CHARACTERISTICS OF THE ORE AND WALL ROCKS

9.4.1 General

There are three basic rock types at the Lupin Mine: amphibolite (orebody); quartzite; and phyllite. In the West Zone, the hangingwall rock type is quartzite and the footwall rock type is phyllite.

The main geologic contributor to stope wall instability in the West Zone is a fault structure called the West Zone Fault. This structure is a graphitic slickensided shear zone which parallels the West Zone and is located in the immediate footwall.

A description of these rock units is given in the following sections.

9.4.2 Description of Main Rock Types

Amphibolite (Orebody)

The amphibolite is very strong (R5: > 200 MPa) and can be very massive in appearance.

In any particular area, the number of observed joint sets in the amphibolite varies from 1 to 3. The dominant joint set parallels the orebody. It is very persistent and forms smooth planar surfaces. Little to no joint alteration is present. The spacing of this joint set is typically around 0.5m, however, it can vary between approximately 0.3 - 1.0m. The remaining two joint sets are widely spaced (1 to >3m). One set is flat lying and the other strikes perpendicular to the orebody and generally dips steeply to the north. These joint sets have variable persistence and generally have smooth planar surfaces with little to no alteration.

Of the three basic rock types encountered at the Lupin Mine the amphibolite is the most competent from a geotechnical perspective.

Quartzites - West Zone Hangingwall

The quartzites are generally quite strong (R4: 100-200 MPa) but can be blocky.

Three well developed joint sets are generally observed in the quartzites. The dominant joint set parallels the orebody and has a spacing ranging from approximately 0.3 - 1.0m (avg. 0.5m). Joint surfaces are smooth and planar with little to no alteration. With regards to the remaining two joint sets, one is flat lying while the other strikes roughly perpendicular to the orebody and dips steeply to the north. The spacing of both sets varies between approximately 0.5 - 3m. Joint surfaces are typically smooth and planar with little to no alteration.

Occasionally a narrow zone of shearing (slickensided surface) is encountered at the contact between the amphibolite and quartzite.

Phyllites - West Zone Footwall

The phyllites are quite strong (R4: 100-200 MPa) but are typically quite slabby in appearance.

Three joint sets can be observed in the phyllite, however, generally only two are prominent. The main joint set which parallels the orebody is very persistent and quite tightly spaced (0.1 - 0.5m). Joint surfaces are smooth and planar and have little to no alteration. The second most prominent set is flat lying with a spacing of 0.5 - 2.0m. Joint surfaces are smooth and planar with little to no alteration. A joint set which strikes perpendicular to the orebody and dips steeply to the north is sporadically encountered.

As was the case with the quartzite, a narrow zone of shearing (slickensided surface) is occasionally observed at the contact with the amphibolite.

From a geotechnical perspective, the phyllites are the least competent of the three rock types discussed.

9.4.3 Rock Properties

A laboratory testing program to determine rock strengths was carried out by CANMET in 1990 (Gorski, 1990). Results from the study are summarized in Table 9.1 below:

Table 9.1
Rock Properties of Main Rock Types at the Lupin Mine

Material Properties	Units	Phyllite	Amphibolite	Quartzite
Modulus of Elasticity	GPa	52	101	49
Poisson's Ratio		0.21	0.24	0.42
Density	kg/m^3	2870	3210	2700
UCS	MPa	121	448	168
Brazilian Tensile	MPa	21	28	14
Friction Angle	deg.	27	51	35

9.4.4 Joint Sets

As can be inferred from Section 9.4.2, there are three main joint sets (roughly orthogonal) which are common to the three rock units described above. Table 9.2 presented below shows average orientations determined from underground drift mapping. The quoted orientations are with respect to mine north.

Table 9.2
Orientation of Major Joint Sets at the Lupin Mine

Joint Set	Dip	Dip Direction
1 (parallel)	80	110
2 (perpendicular)	75	005
3 (sub-horiz./flat)	13	195

Note that the orientations presented above apply to the north/south striking limbs of the orebody. In the nose of the fold the relative orientations remain similar (parallel, perpendicular, flat), however, the actual orientations rotate to maintain their relative orientation.

Random joints occur locally in areas intruded by dykes.

9.4.5 <u>West Zone - Major Structure</u>

<u>West Zone Fault</u>

The West Zone fault is a graphitic slickensided shear zone which parallels the West Zone and is located in the immediate footwall. The shear zone is on average 0.5 - 1.0m thick and is typified by weak rock (R2-R3: 25-100 MPa), tightly spaced strong parallel jointing, and graphitic slickensided joint surfaces. The fault meanders slightly and is generally located within 0-3m of the orebody contact.

The West Zone fault is very continuous between levels and is one of the main contributors to dilution in the West Zone. Underground observation indicates that significant footwall dilution is experienced when:

- the fault daylights in the stope wall;
- the fault is undercut by in-ore development;
- too thin a skin of competent rock (i.e. amphibolite) is left in front of the fault;
- large stope spans are opened.

Minor Cross-Cutting Faults

A number of minor faults cross-cut the orebody in the West Zone. These faults strike northeasterly and dip steeply. The faults are typically quite narrow (0.15m) and are occasionally water bearing. When exposed during stoping, these faults commonly initiate small local failures in both the hangingwall and footwall.

9.4.6 Rock Mass Classification

Based on underground drift mapping, the different rock types encountered at the Lupin Mine have been classified using the Q system (Barton et al., 1974) and the RMR system (Bieniawski, 1974). Tables 9.3 and 9.4 presented below, show typical values for the various rock types.

Table 9.3
Lupin Mine Rock Mass Classification - Q system

	Amphib.	Phyllite	Quartzite	WZ Fault
RQD	90 - 100	75 - 90	85 - 100	25 -50
Jn	2 - 6	6 - 9	6 - 9	6 - 9
Jr	1	1 - 0.5	1 - 0.5	0.5
Ja	1	1 - 2	1 - 2	4
Jw	1	1	1	1
SRF	1	1	1	1
Q (Q')	50 - 15 (typical: 23)	15 - 2.1 (typical: 6.3)	16.6 - 3.5 (typical: 10)	1 - 0.3 (typical: 1)

Note that Jw and SRF have arbitrarily been set to one (dry conditions and medium stress) which gives the rating termed Q' and is in accordance with existing empirical stope design methods developed by Mathews et al. (1981) and Potvin (1988).

Table 9.4
Lupin Mine Rock Mass Classification - RMR System

	Amphib.	Phyllite	Quartzite	WZ Fault
Strength	15 (447 MPa)	12 (121 MPa)	13 (168 MPa)	7-4 (R2-R4)
RQD	20 (90 - 100)	17 (75 - 90)	18 (85 - 100)	8 25 - 50
Spacing	20 (0.3 - 1m)	10 (50-300mm)	20 (0.3 - 1m)	10 (50-300mm)
Condition	20 - 12 (tight-open)	20 - 6 (tight-slick)	20 - 6 (tight-slick)	6 (slick-open)
Ground Water	10 (dry)	10 (dry)	10 (dry)	10 (dry)
RMR	85 - 77 (typical: 80)	69 - 55 (typical: 65)	81 - 67 (typical: 70)	41 - 38 (typical: 40)

9.5 GEOTECHNICAL CONSIDERATIONS REGARDING MINIMIZING OVERBREAK/SLOUGH

9.5.1 Stope Sizing

Given that only minor amounts of overbreak/slough are required to significantly dilute narrow orebodies, it follows that narrow vein stopes should be sized with a high inherent level of stability. In otherwords, stopes should be sized such that dilution due to blast induced overbreak is the main concern. Stopes should not be sized such that some wall failure is anticipated before a stable configuration is reached. Figure 9.4 presents plots of the Modified Stability Graph (Potvin, 1988, Nickson, 1992) and the corresponding ELOS design graph (presented in previous chapters), showing typical West Zone stope sizes and the predicted stability for stope surfaces comprised of the different rock types at the Lupin Mine. Referring to the figure, it can be seen that with the exception of stope surfaces exposing the West Zone Fault, stopes are sized with a high level of inherent stability. When exposure of the West Zone fault is unavoidable, the only means of minimizing unplanned dilution is to mine the stopes as small as is practically possible (i.e. strike lengths of approximately 10m).

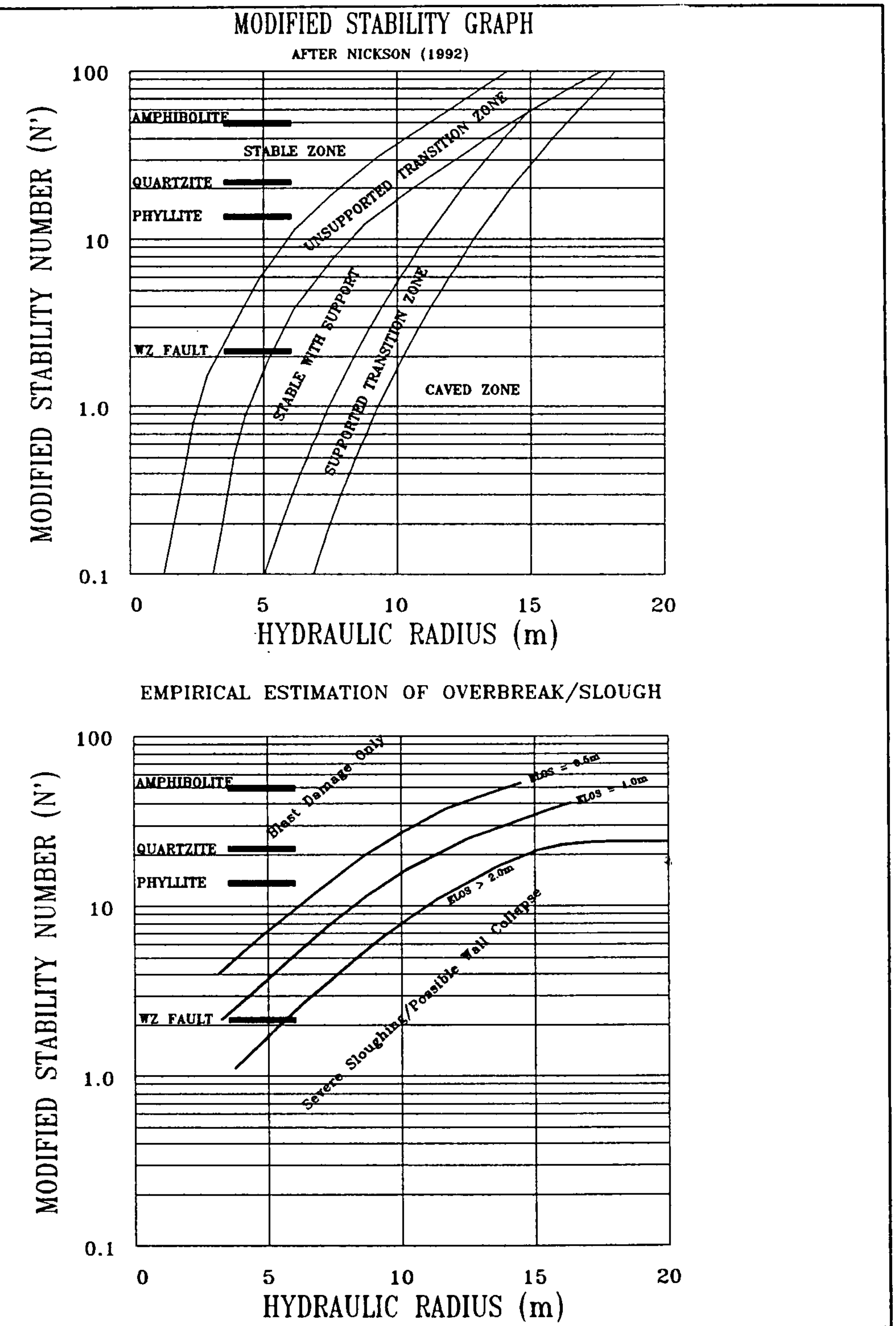


Figure 9.4 Stability graphs showing typical West Zone stope sizes

9.5.2 Influence of Rock Type on Overbreak Potential

In more cases than not, the main source of dilution in the West Zone is the footwall. This is largely due to: the weaker phyllites being more susceptible to blast damage; and stope wall instability associated with the West Zone Fault. The hangingwall quartzites are quite competent and generally very little blast induced overbreak occurs.

Generally, the ore bearing zone is narrower than the actual amphibolite unit, therefore, the main goal with regard to longhole blasting in the West Zone is to try and preserve a thick "skin" of competent amphibolite in front of the poorer quality footwall rocks. This is shown schematically in Figure 9.5. This is generally achievable when the West Zone Fault is located more than 1m away from the orebody contact and significant undercutting of the footwall has not occurred during ore drift development. When the West Zone Fault is located closer than 1m from the orebody contact, there has been some success with achieving minimal dilution, but, it is very dependent on the quality of the drilling and blasting, and, whether or not the fault has been undercut by the development.

9.6 HISTORICAL DEVELOPMENT OF THE CURRENT WEST ZONE BLAST PATTERN

Longhole mining was initially trialed in the West Zone in late 1985 and early 1986. At this time the large equipment used in the Centre Zone was utilized, resulting in ore drifts 3.5m wide. Subsequently, undercutting of the stope walls was a major problem. Blast patterns utilized 64mm blastholes (drilled to breakthrough $\approx$ 17m) on a 2:1 pattern in ore narrower than 2.5m and a 3:2 pattern in wider areas. The designed burdens were 1m and 1.2m respectively. To allow for some overbreak, perimeter blastholes were collared approximately 0.2m inside the ore contours.

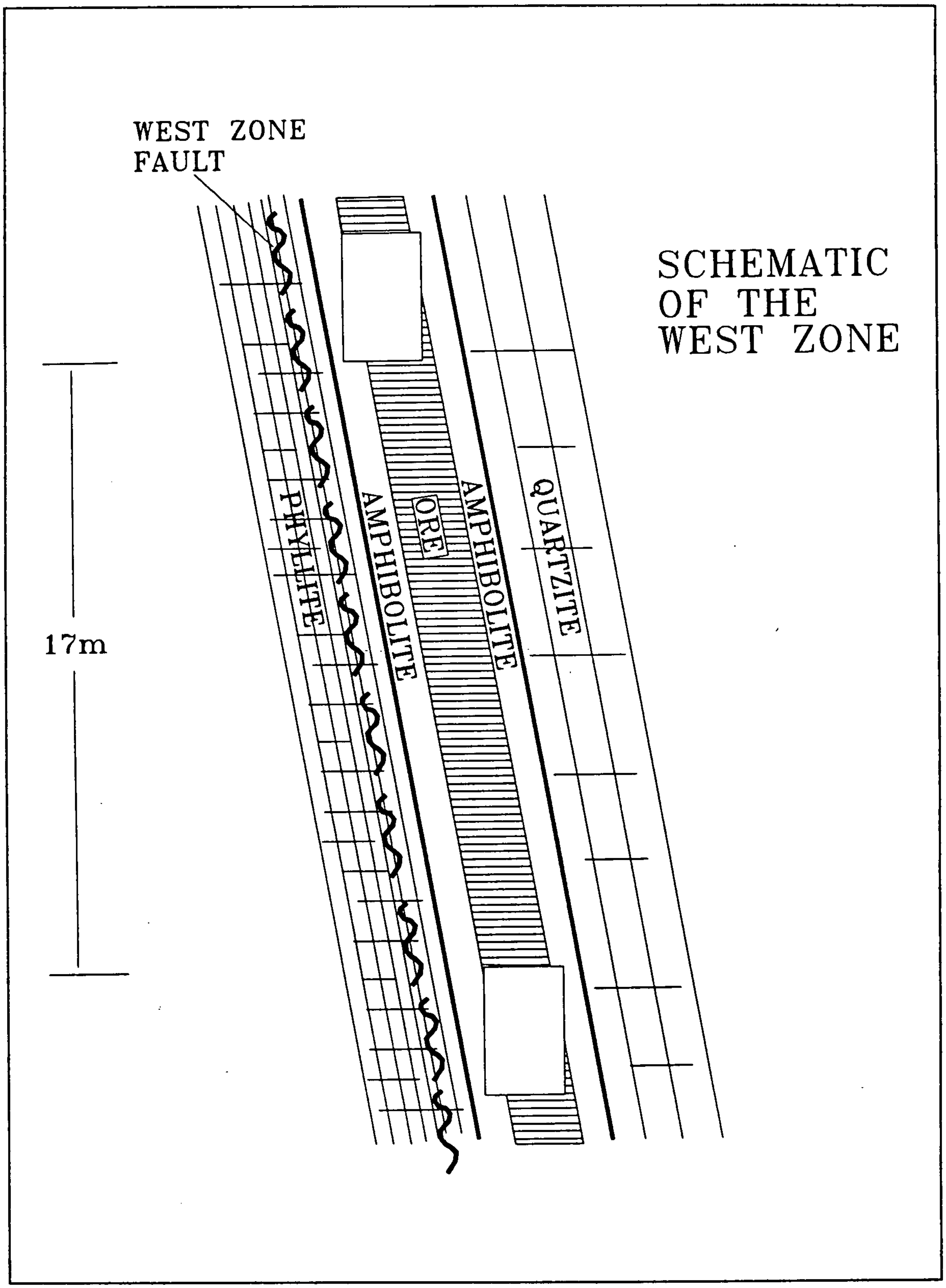


Figure 9.5 Schematic showing rock types in the West Zone

Dilution was estimated at 45% but can be partially attributed to the severe undercutting of the stope walls. Only one incident relating to a benched blast (blast did not break the ground) was documented. This longhole method was used along with raise platform mining (RPM) until 1988.

In 1988, the switch to smaller equipment was made. It was at this time that the blasthole diameter was reduced to 50mm. To reduce potential hole deviation problems, the blasthole length was shortened by converting to non-breakthrough upholes and downholes (blastholes were overlapped 1m). The upholes were dumped at 70°.

Initially a 2:1 pattern with a 0.75m burden was selected. Downhole blasts of 2-6 rings per blast benched at the toes on several occasions. Consequently, additional holes were drilled in the one-hole row, converting it to a 3:2 pattern. This broke more successfully but there was concern about stope wall damage due to the increased powder factor.

In 1989, Golder Associates Ltd. was contracted by Lupin/CANMET to do a blast monitoring program aimed at addressing the problem of blast induced wall damage. The program involved measuring near field blast vibrations as well as assessing wall conditions and fragmentation. With regard to the 3:2 pattern (0.75m burden) the monitoring program identified the following: significant charge interaction resulting in a substantial amount of misfires (i.e. charges not contributing to breakage process) and inefficient detonations; high energy levels at distances beyond the designed zone of influence of the blast; obvious wall sloughing; and bi-modal muck size which is characteristic of overcharging.

Based on the monitoring program, a 2:1 pattern with a 0.55m burden and 1.2m spacing was trialed and proved successful. No misfires were recorded although energy levels were similar to those measured for the 3:2 pattern. Following the trials, the pattern was expanded slightly such that the burden between the two hole rings was 1.3m with the Centre hole positioned 0.5m ahead of the two hole ring. The hole spacing was kept the same at 1.2m. The pattern proved effective in test blasts, but when implemented on a full scale, insufficient breakage was again experienced. It is expected that poor quality control with regard to drilling accuracy and loading practices was the main contributor to the failed implementation of this pattern. Figure 9.6 shows a survey pick-up of collar locations in an area drilled with the 2:1 pattern. It is obvious from the figure that the collaring errors alone are sufficient to cause benching problems.

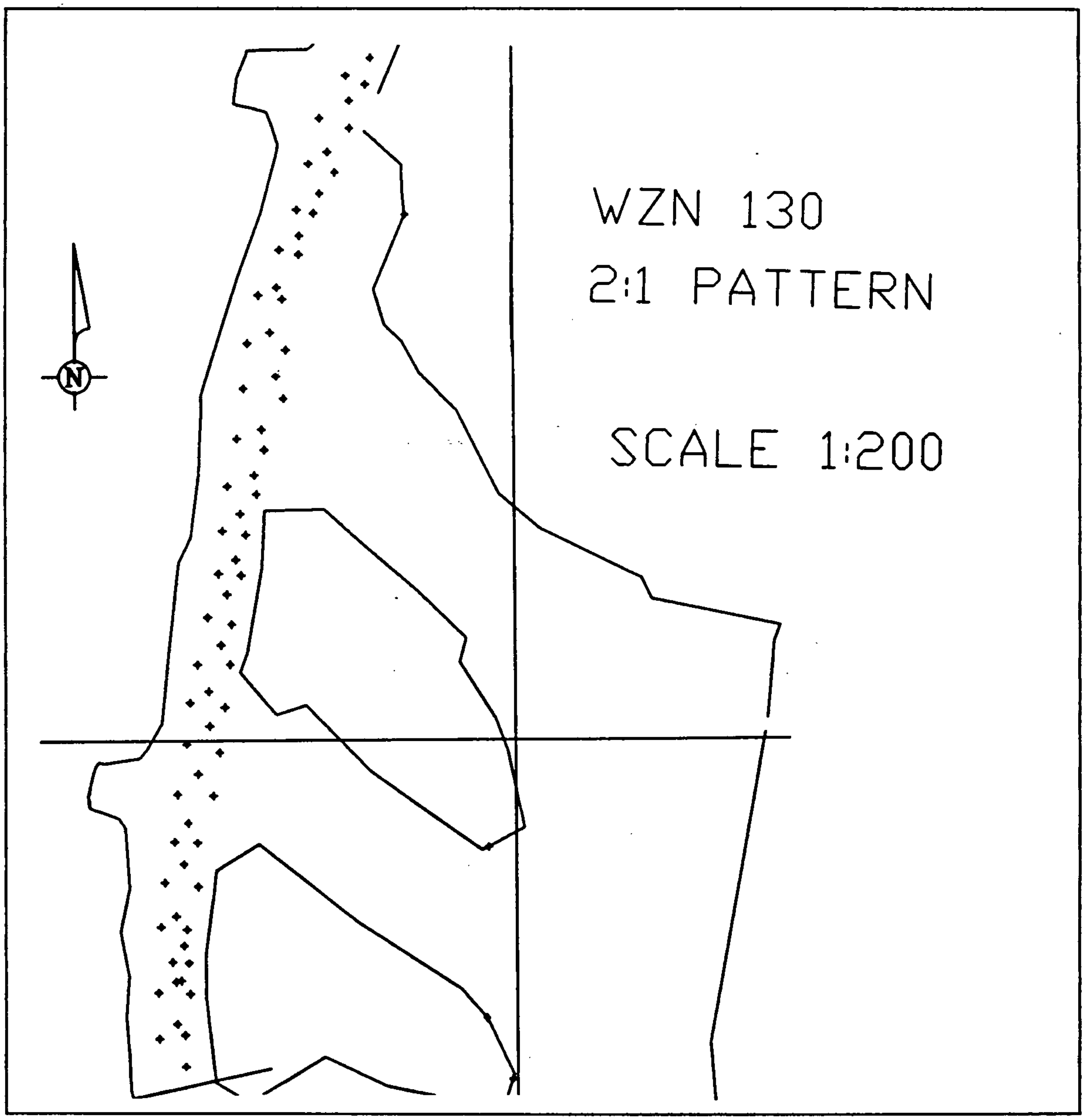


Figure 9.6 Survey of blasthole collars showing collaring errors - 2:1 pattern (1989/90)

Following the failed attempt at implementing the 2:1 pattern, a return was made to the 3:2 blast pattern with the 0.75m burden. This pattern is still being used today. Based on a number of laser stope surveys conducted in 1996, dilution in the West Zone currently ranges between approximately 2% to greater than 100%, with an average of 42%.

Figures 9.7 and 9.8 summarize the longhole blast patterns and timing used from 1985 to date.

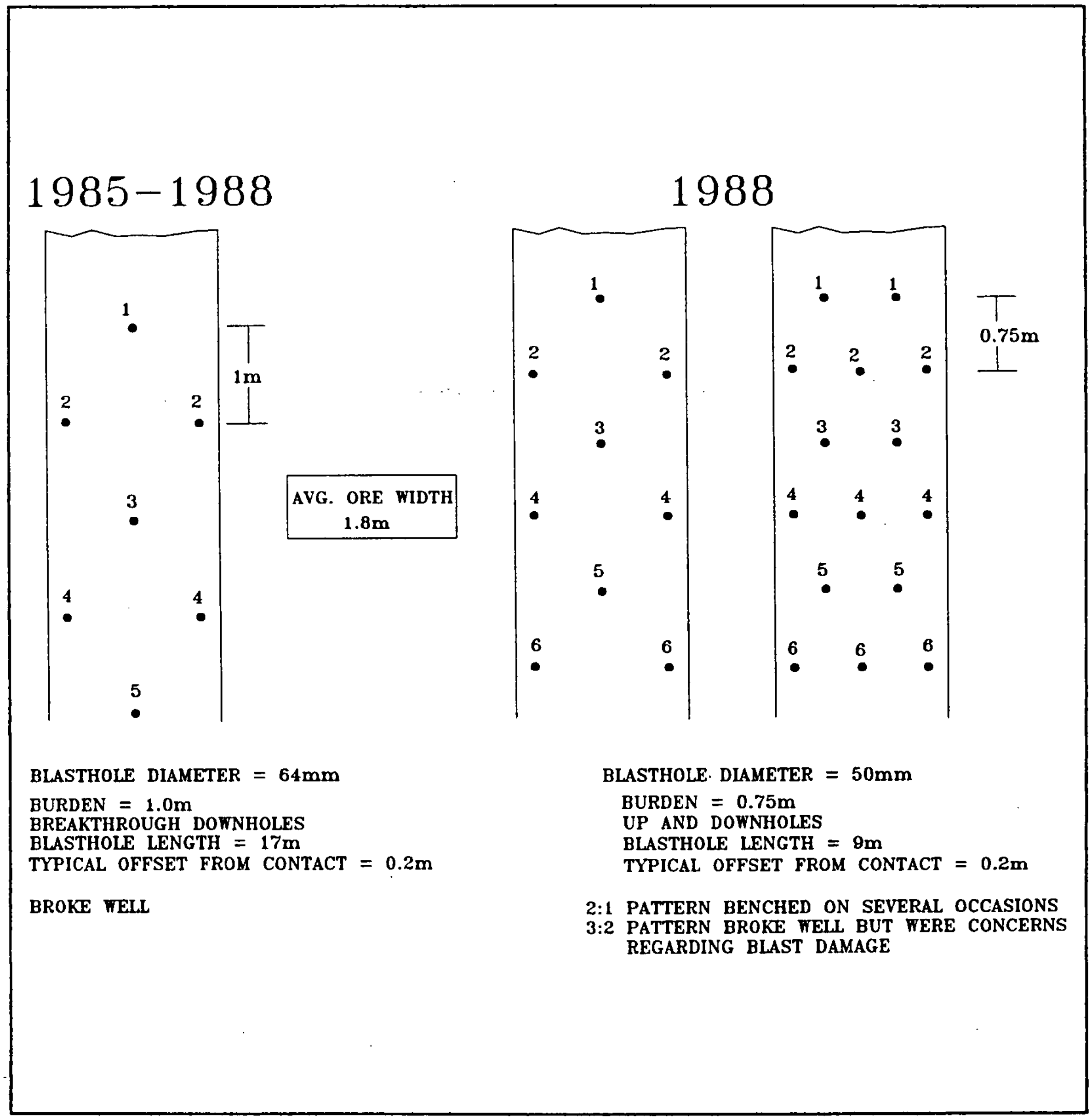


Figure 9.7 Historical development of West Zone blast pattern - 1985 to 1988

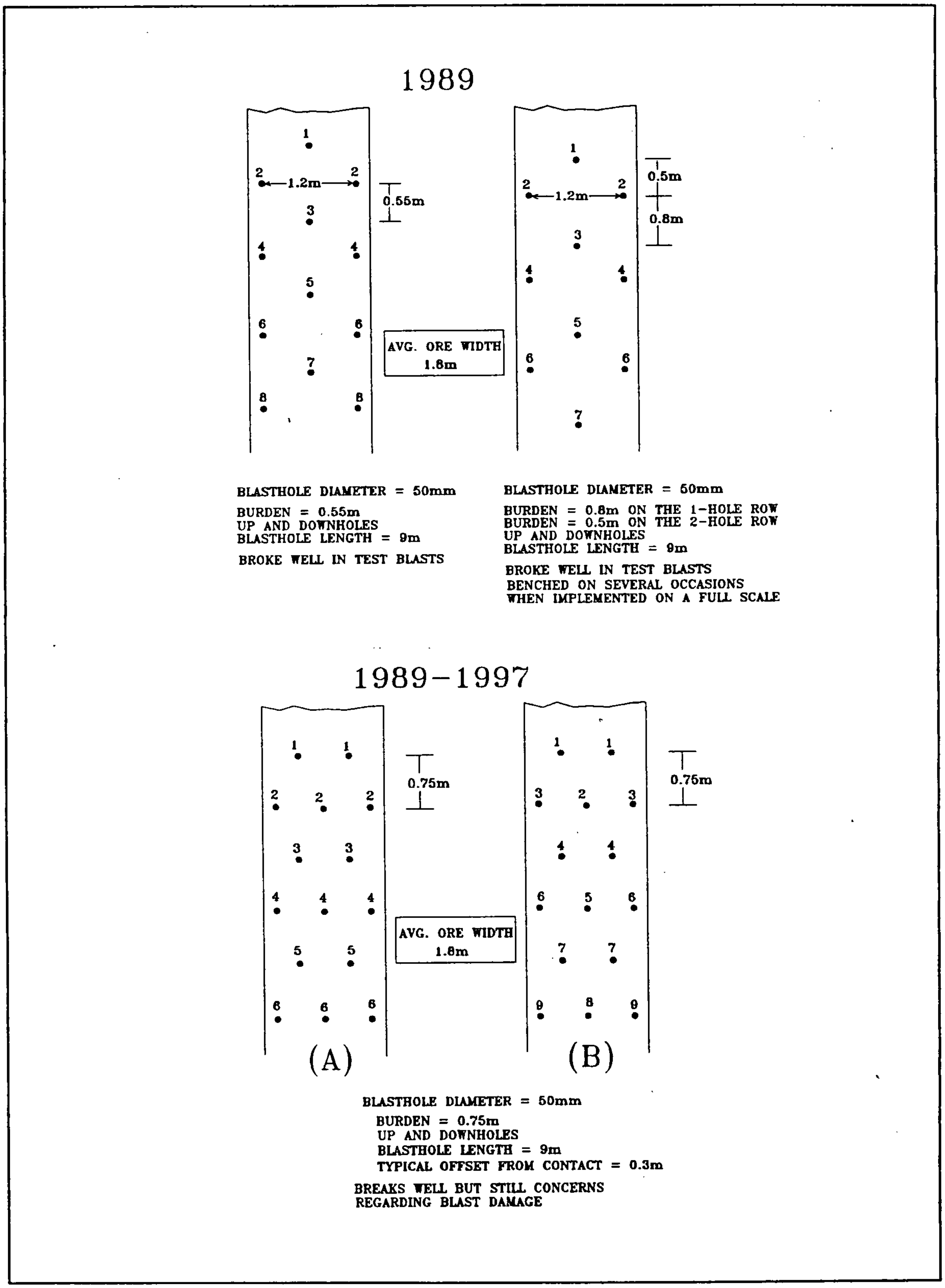


Figure 9.8 Historical development of West Zone blast pattern - 1989 to 1997

9.7 OBJECTIVES OF THIS STUDY

As mining progresses deeper at the Lupin Mine (currently the lowest mining horizon is at 1335m depth) there is considerable incentive to try and reduce certain mining costs to try and offset increased costs associated with activities such as ground support. Three areas where there is potential for cost reductions are: drilling; blasting; and reducing unplanned dilution. The objectives of this study are focused toward addressing these issues.

The objectives of this study are as follows:

- develop a baseline of current blast performance for the 3:2 pattern by monitoring a number of blasts;
- trial methods of reducing blast vibrations associated with the 3:2 pattern by examining different timing sequences. Trials will involve using both MS delays and 100/3800 dual delay detonators;
- summarize the pros and cons of the 3:2 pattern and the associated costs;
- trial 2:1 and 1:1 patterns using both 50mm and 64mm diameter blastholes;
- summarize pros and cons of the 2:1 and 1:1 patterns and the associated costs;
- make recommendations regarding potential methods of reducing drilling and blasting costs and reducing unplanned dilution.

CHAPTER 10

ASSESSING BLAST PERFORMANCE

10.1 WHAT IS GOOD BLAST PERFORMANCE?

Good blast performance should be measured relative to its required objectives with regard to mining costs and production. The following criteria are after Forsyth (1990):

- *Adequate Breakage* - muck should be broken just well enough to be readily loaded at the drawpoints with full bucket loads and without the necessity of secondary blasting in the drawpoints or plugging up the orepass grizzly.

- *Minimal Dilution* - overbreak beyond the ore contacts and/or wall sloughing dilute the ore thereby reducing the grade and increasing the cost per unit weight of metal mined.

- *Minimal Cost* - only the necessary amount of drilling and blasting should be done to achieve the required results. Over-drilling and over-blasting directly increase mining costs.

- *Minimal Benching* - if a blast fails for some reason and a remnant has to be re-accessed and re-blasted or has to be abandoned and a new slot raise established, costly delays to production as well as the additional costs to remedy the situation are incurred.

10.2 BLAST MONITORING

10.2.1 Blast Vibration Monitoring

A large portion of this study involved the measurement of blast vibrations (particle velocities - PV's) at relatively close distances to the blast ($\approx$30m). This information was used to assess the energy levels associated with each blasthole which correspondingly relates to the amount of work the blasthole is performing.

To measure blast vibrations an Omniprobe 1200 blast monitor developed by Instantel was used along with recoverable surface mounted geophones. The Omniprobe 1200 is a digital recorder that has the capacity to sample at a rate of 16000 Hz. This sampling rate is sufficient to record individual holes detonating thus allowing explosive energy to be assigned to specific blastholes.

The geophones used for this study were: OYO/Geospace McSEIS-128 High Frequency (1000 Hz). Two triaxial arrays of these geophones were used. Each triaxial array was mounted on an aluminum base (approx. 4"x 4" x 1") which was in turn bolted onto the ore drift wall utilizing short 1/2" diameter ready-rods and expansion shell anchors. It was realized from the onset that by using surface mounted geophones (as opposed to grouting them in a drill hole) there was a possibility of some inconsistency in the recorded energy levels. This can result due to: poor condition of drift walls; poor coupling of the aluminum base to the wall surface; and possible resonance of the mount and/or drift (Rowell et. al., 1984). However, it was felt that the benefits of using surface mounted geophones (i.e. reduced cost and increased flexibility) outweighed the disadvantages. To reduce the possibility of collecting inconsistent data, great care was taken to sound the walls to find "solid" locations to mount the geophones. Furthermore, the geophones were always mounted on planar surfaces within amphibolite (i.e. good coupling of the geophone to the rock surface). When monitoring, if there were doubts regarding the quality of the geophone mount the blast vibration data was not used.

To ensure all the monitoring data could be compared on a relative basis a very consistent approach to monitoring was taken. A typical monitoring session was as follows:

- the internal resistance of the geophones were checked before each blast to ensure they were functioning properly;

- mount the geophones on the ore drift wall, in amphibolite, approximately 30m plan distance back from the blast;

- use a wire-break trigger connected to the start of the blast to initiate the monitor;
- sample at 16000 Hz.

10.2.2 Analysis of Blast Vibration Data

Following a blast, vibration monitoring data was downloaded from the Omniprobe 1200 blast monitor to a PC, and analysed using software provided by Instantel. The following was documented for each blast:

- the detonation time and associated peak particle velocity (PPV) for each blasthole;
- the average peak particle velocity (PPV) for the blast;
- the maximum peak particle velocity (PPV) for the blast;
- the number of misfires;
- identification of the holes that misfired.

With regard to the recorded PPV's, for a particular triaxial array of geophones, the vector sum of the three orthogonal vibration levels was always used. This ensured that all the recorded vibration information was used, and that vibration levels were determined in a consistent manner.

The identification of misfires (i.e. charge malfunctions) is one of the main benefits of using near field blast monitoring techniques. Traditionally, a misfire was defined as an explosive which did not detonate due to an initiation system fault. Misfires of this type are relatively uncommon. More common, are misfires caused by charge interaction, which is related to the distance between adjacent charges. Misfires caused by charge interaction are generally due to one of three mechanisms:

- *Sympathetic Detonation*: where physical separation between charges is small, high temperature gas products can stream from one charge to the next, causing sympathetic detonation of the second charge. This interaction can occur between charges located in the same hole (decked charges) due to ineffective stemming, or, between charges in adjacent holes due to inadequate hole separation.

- *Explosive Desensitization*: most explosives become less sensitive at higher densities. This relationship is most pronounced for explosives which rely upon unreinforced air bubbles for sensitivity (i.e. air-sensitized watergels and emulsions). Charges can be densified and, hence, desensitized in three ways: hydrostatic pressure; dynamic (i.e. blast induced) pressures; or by a combination of hydrostatic and dynamic pressures. Of most concern in narrow vein longhole blasting is the dynamic effects imposed by earlier firing, closely spaced, adjacent charges. In this case, desensitization can occur due to:

  - the compressive strain wave passing through a later firing charge;

  - lateral deformation of adjacent blastholes and subsequent squeezing of a later firing charge;

  - explosive gasses or groundwater streaming through pre-existing or blast generated cracks into a later firing blasthole.

- *Overbreak*: this results when the action of an earlier firing charge breaks or removes the ground around a later firing charge. This can result in: dislodging of explosive product; and/or detonation of poorly confined or unconfined explosive product. Overbreak is a significant problem because it often results in oversize material.

10.2.3 Additional Field Information

In addition to the vibration monitoring data the following additional field information was collected for the blasts monitored:

- notes on loading practice and explosive products used;

- notes on the actual spacings and burdens;

- the timing sequence used;

- wall conditions before and after the blast;

- average size of blasted muck (refer to Figure 10.1) and distribution (i.e. uniform; bi-modal);

- overall success of blast (i.e. broke well; benched)

Post blast wall conditions were evaluated using primarily observational techniques. When time permitted, stope surveys were carried out using the Cavity Monitoring System (CMS). For the most part, stope surveying was only used to evaluate the overbreak potential for the test patterns (i.e. 1:1 pattern). A detailed database of West Zone stope surveys, excavated using the 3:2 pattern, had previously been compiled for the CMS database (Chapters 4 through 8). Thus there was already a relatively good understanding of overbreak potential with regards to the 3:2 pattern.

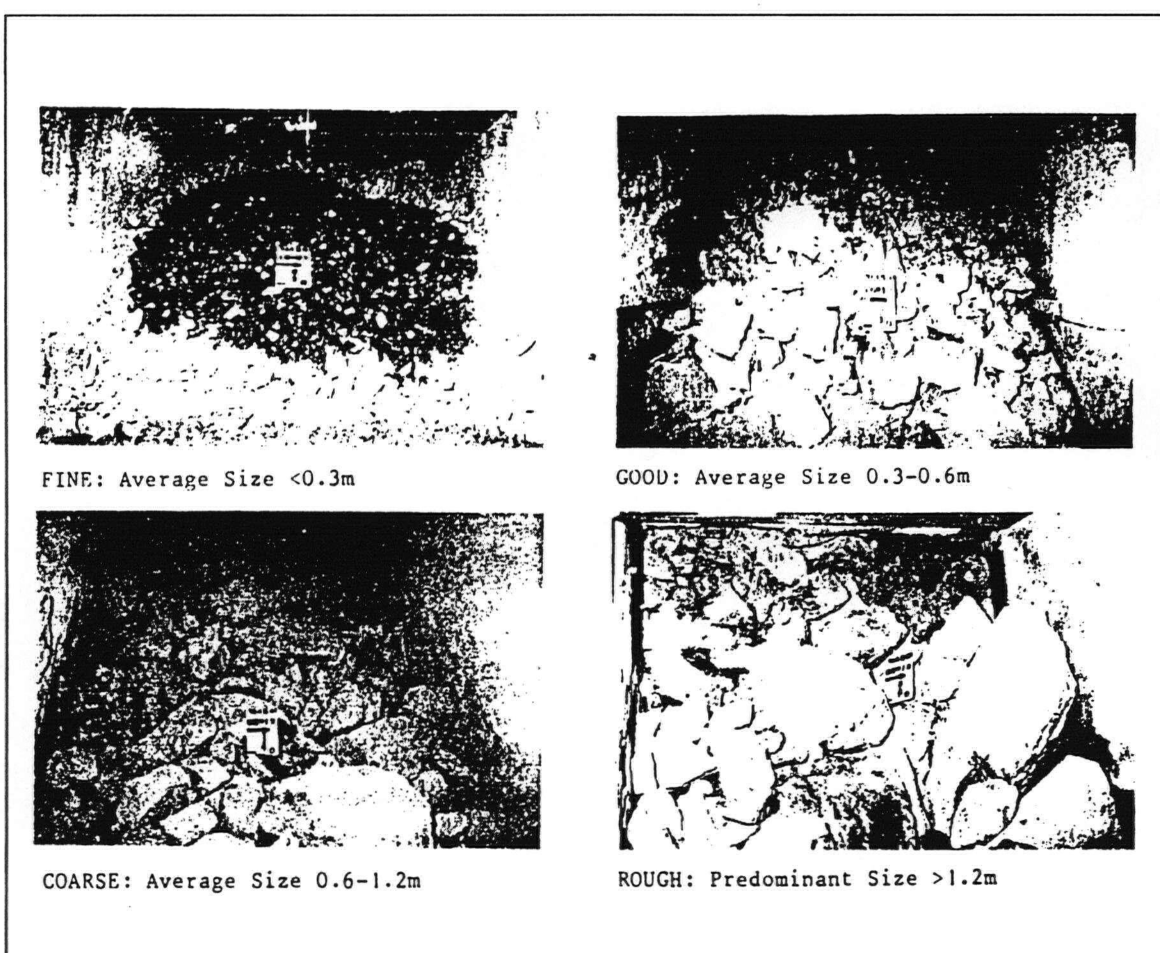


Figure 10.1 Fragmentation assessment guide (from Bohannan et.al., 1985)

10.3 INFLUENCE OF: DRILLING ACCURACY; LOADING PRACTICES; AND EXPLOSIVE PERFORMANCE ON MONITORING RESULTS

A study such as this generally has to assume that charge malfunctions (i.e. misfires) are largely a result of the blast design rather than poor quality workmanship and improper explosive performance. Fundamental blasting factors which can effect the success of the blast and how they relate to this study are discussed below:

- *Drilling Accuracy* - although it is expected that hole deviation was not a major problem due to the short length of the blastholes, significant errors in collar locations, which result in improper burdens and spacings, were occasionally observed. Inaccurate drilling can influence blast performance in a number of ways, for example: holes collared too close to one another may result in charge interaction resulting in misfires; increased burdens will result in higher vibration levels and hence increased potential for blast damage or benching; and if blastholes wander outside the ore contacts increased levels of dilution will result.

- *Explosive Product Performance* - a potential problem with working in cold climates is the effect of cold temperatures on the explosive products. Temperature cycling (above and below -18°C) can adversely effect Anfo by causing the prills to break down. This results in a product with a lot of fines which can cause problems when blow loading the product (i.e. significant blow back and poor packing in the blasthole). This was observed occasionally. Generally however, the temperature cycling does not effect the fuel content or explosive energy performance (verified by testing conducted by ICI). Some problems with low fuel content were identified, but were more related to poor storage practices and long storage times for some of the product (every winter a years supply of Anfo is brought up to the mine site via winter road). Cold temperatures can also effect other products such as emulsions. Emulsions if not completely thawed are insensitive.

- *Stope Preparation and Loading Techniques* - this aspect of blasting involves: ensuring that all holes are cleaned properly and that holes are redrilled if necessary; the right explosive products are used in the right application (i.e. do not use Anfo in wet holes); the blastholes are primed properly; and that the blastholes are sequenced properly. Early in the study, there were some concerns regarding loading practices due to a wide variation in techniques and views held by the blasters. This was addressed promptly by implementing detailed longhole blasting procedures, and by giving short courses to the drillers and blasters on blast design and the explosive products used at the mine. A longhole foreman was recently hired to help ensure quality control.

CHAPTER 11

3:2 PATTERN BLAST MONITORING RESULTS

11.1 BASELINE BLAST PERFORMANCE STUDY

To develop a baseline of current blast performance 6 blasts were monitored.

At the start of the study, two methods of sequencing were being used by the blasters, these are shown in Figure 11.1. Short period delays (MS-Delays - 25ms) were used to achieve the desired timing. Pneumatically loaded Amex II (Anfo) was the main explosive used. Priming methods were variable but generally involved either single priming or double priming with 40mm diameter Geldyne (dynamite).

The monitoring results can be summarized as follows:

- high degree of charge interaction as evidenced by a significant number of misfires and in-efficient detonations;
- the average number of misfires was 12%;
- with regard to the 12% misfires, 71% occurred on 3 hole rings, and 29% occurred on 2 hole (easer) rings;
- muck size was typically bi-modal (fine (blasted) and coarse (poorly fragmented ore and/or wall slough));
- the average PPV at 30m plan distance was approximately: 160mm/s for sequencing method (A); and 80mm/s for sequencing method (B), refer to Figure 11.1;
- wall damage was most prevalent with sequencing method (A);
- out of the 6 blasts monitored 1 bench was experienced .

Vibration monitoring results from two of the blasts are presented in Figures 11.2 and 11.3.

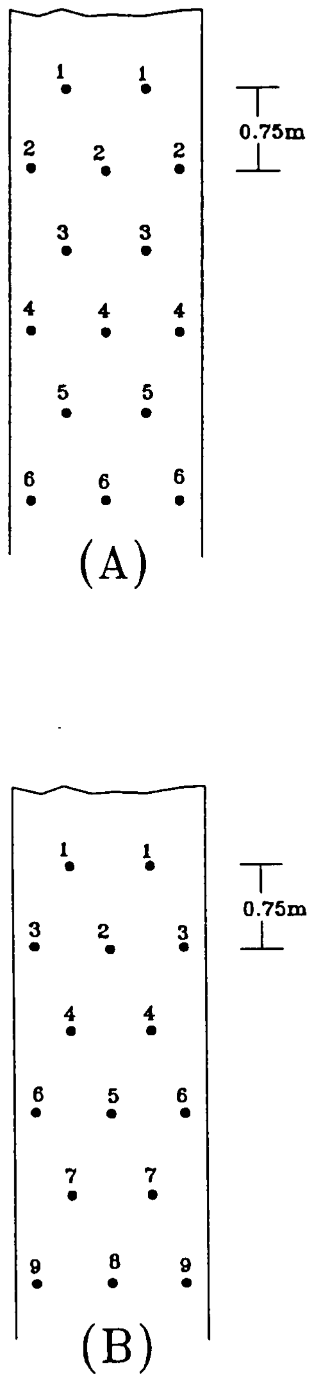


Figure 11.1 3:2 Pattern - sequencing methods (A) and (B) - baseline study

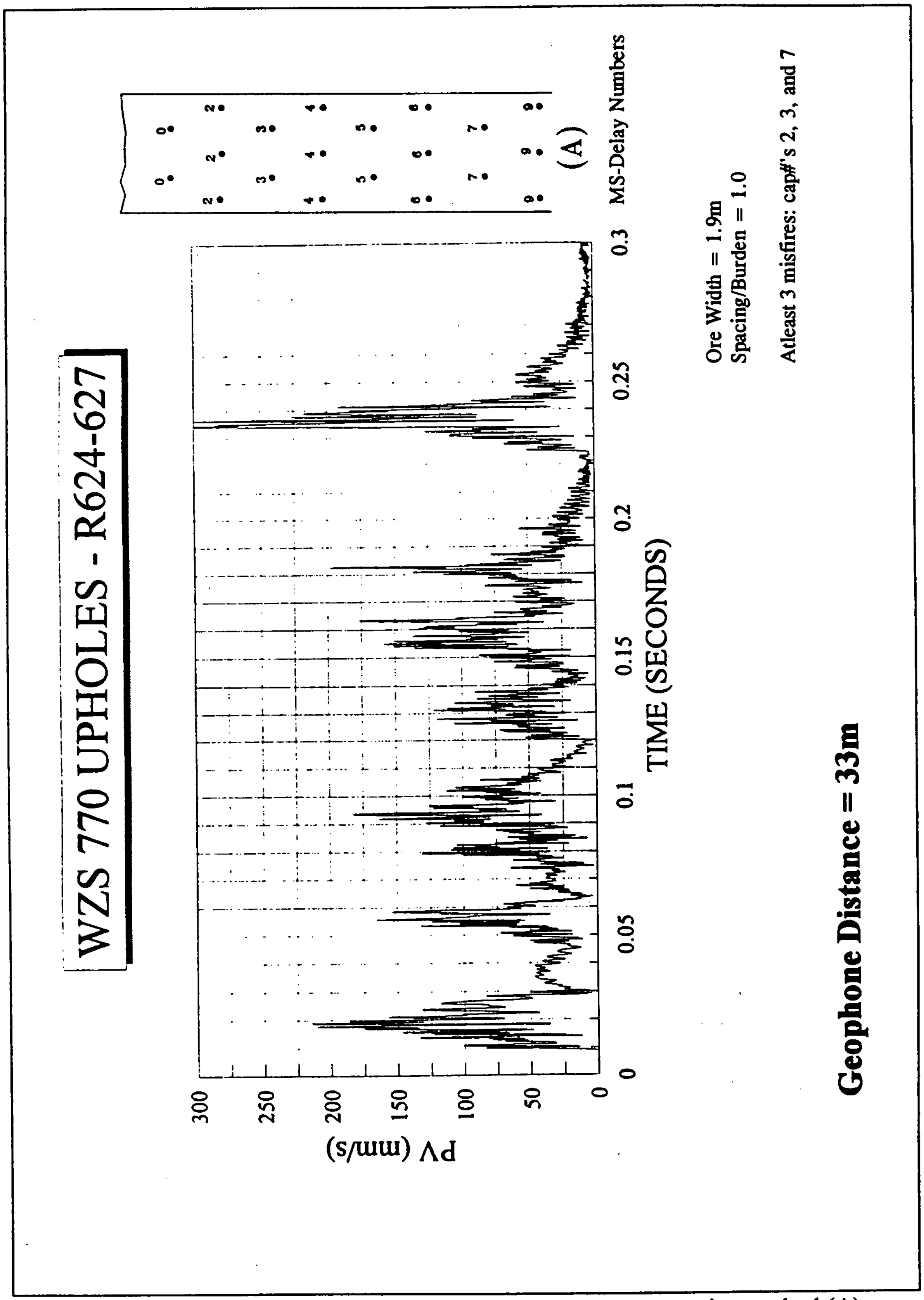


Figure 11.2 Typical blast monitoring results - 3:2 pattern - sequencing method (A)

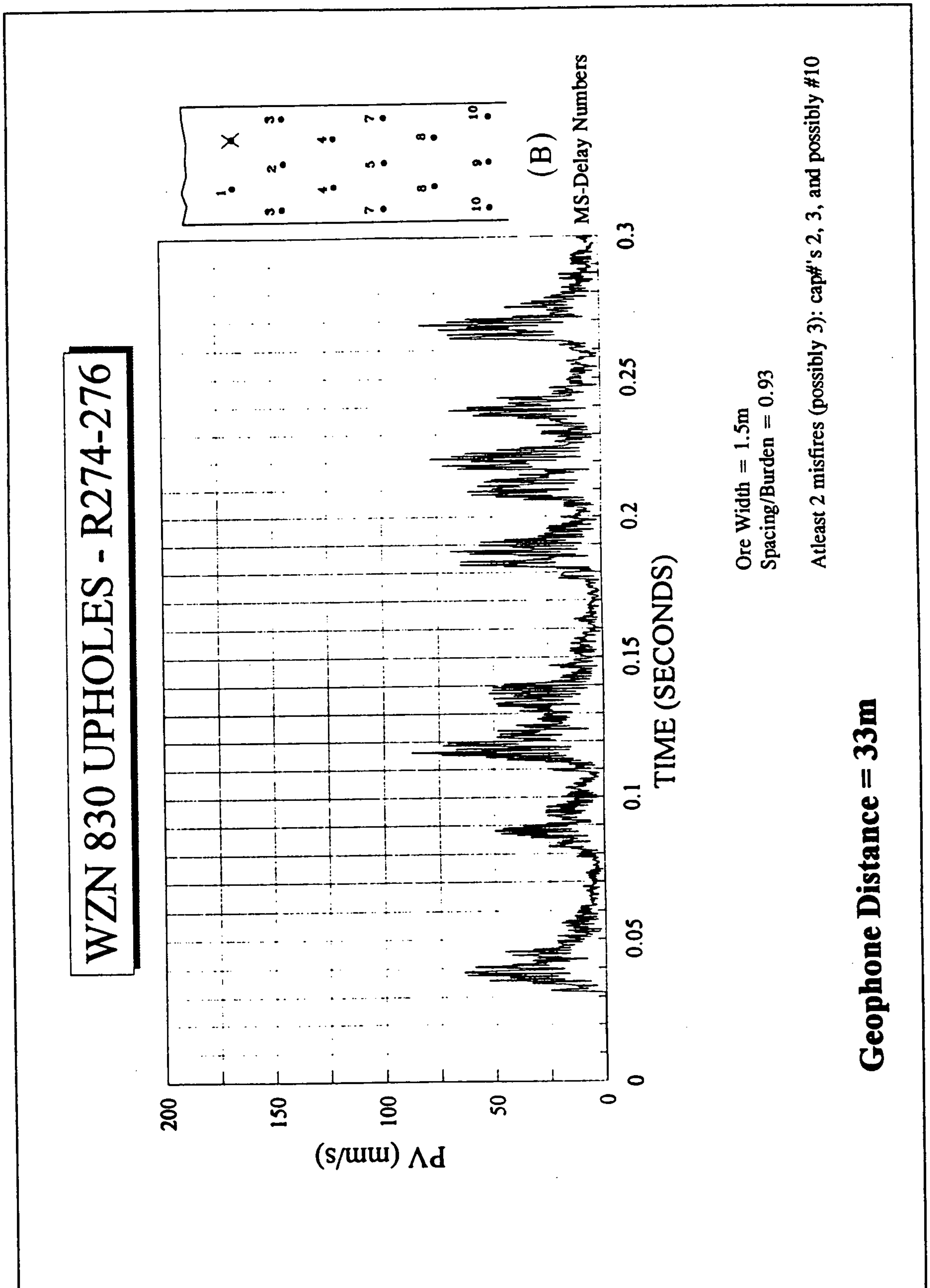


Figure 11.3 Typical blast monitoring results - 3:2 pattern - sequencing method (B)

Of particular concern was the high potential for stope wall damage when using sequencing method (A). When all holes on a 3 hole ring are on the same delay, and taking into account cap scatter, the probability of a wall hole detonating first is 67% (2 out of 3). The wall holes are the most confined, therefore, when a wall hole is first to detonate there is increased shearing along the stope walls and higher energy levels due to the poorer free face geometry (tighter breakout angle). Additionally, the greater the number of holes per delay (i.e. the higher the charge weight per delay) the greater the potential for energy enhancement and hence a higher potential for wall damage.

The monitoring results basically re-affirmed what had been determined from the Golder Associates Ltd./CANMET study done in 1989.

11.2 TRIALS WITH THE 100/3800 DUAL DELAY DETONATOR SYSTEM

11.2.1 General

The dual delay detonator system is a one product initiation system consisting of a common in-hole detonator and a common surface (relay) detonator that is contained in a plastic connector block. The surface detonator provides the timing for sequencing and initiates the shock tube. An example of a dual delay detonator is shown in Figure 11.4. Note that many different combinations of surface and in-hole detonators are available from suppliers (i.e. 100/3800; 25/350; 25/500; 25/700; 200/5000). With regard to the 100/3800 system, the 100 refers to a 100ms surface delay and the 3800 refers to a 3800ms in-hole delay. The in-hole delay dictates the "cooking time", in otherwords it determines how many surface caps will detonate, initiating the in-hole delays, before the first blasthole detonates. Generally, it is best to have all the surface caps detonated before the first blasthole detonates, this lessens the chance of getting a cut-off. With the 100/3800 system, this would permit 38 blastholes to be hooked in series. Figure 11.5 shows methods of hooking up the 3:2 pattern with this system.

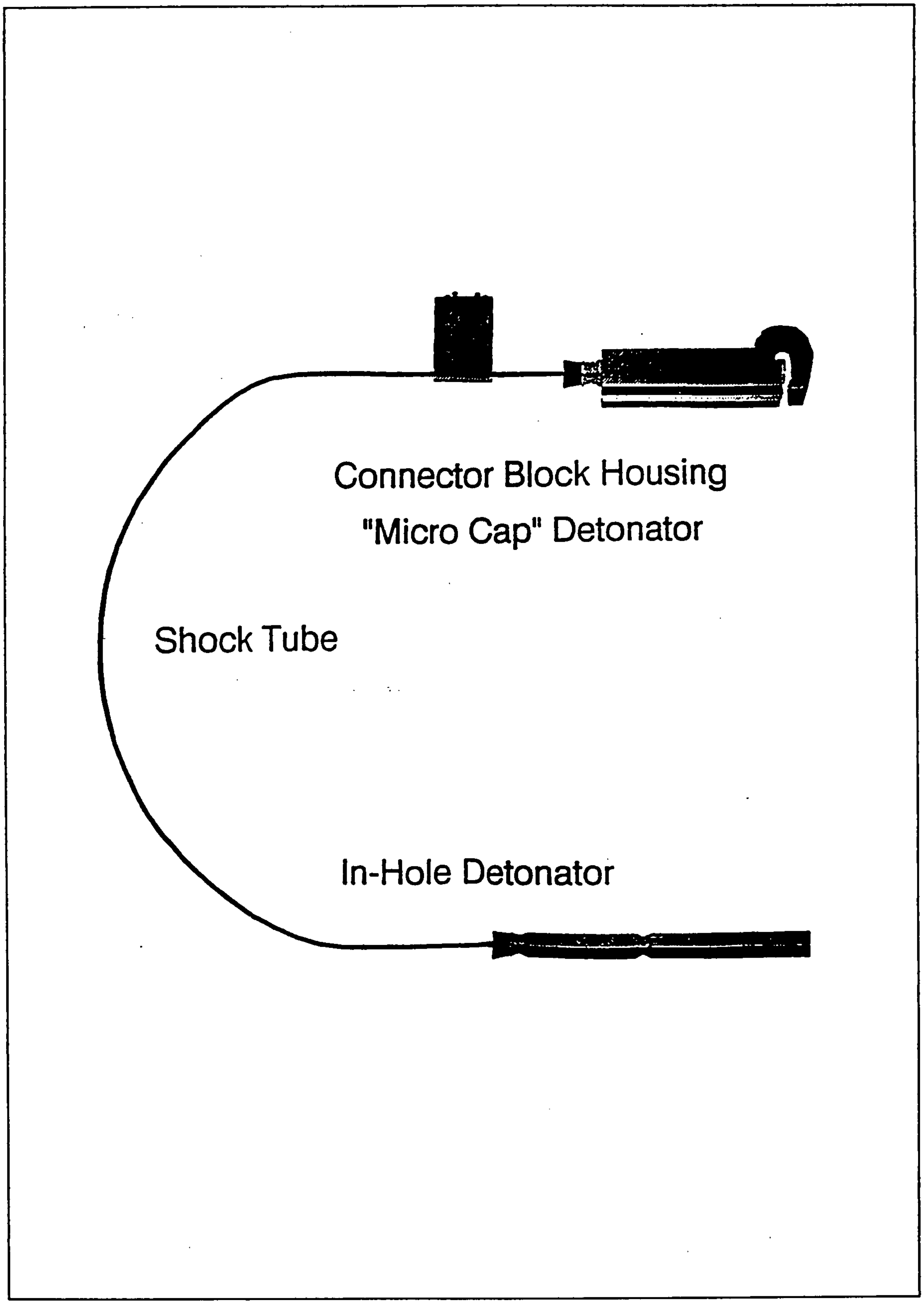


Figure 11.4 Example of a typical dual delay detonator

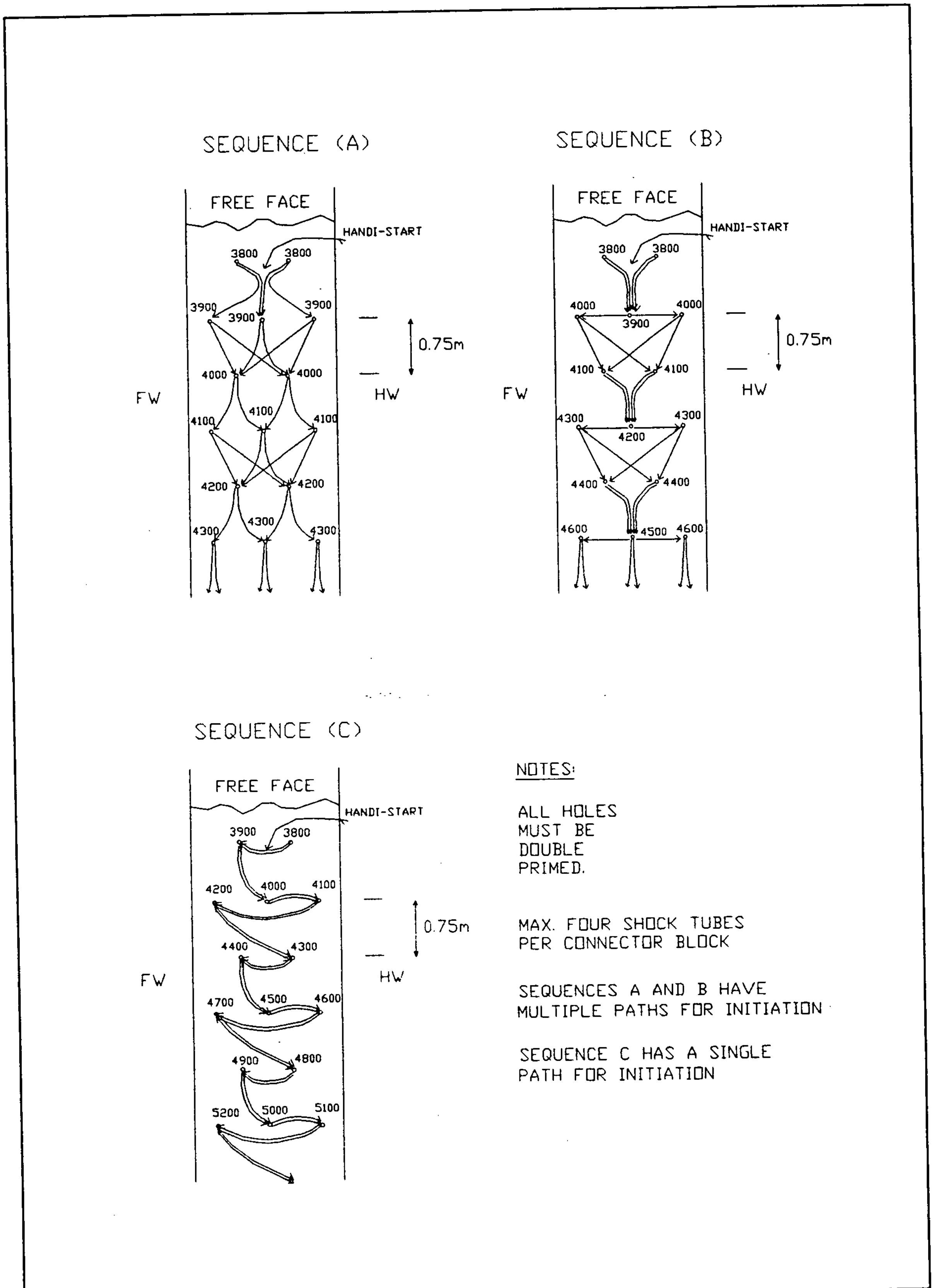


Figure 11.5 Methods of sequencing 3:2 pattern with dual delay detonators

The main benefit from the dual delay detonator system is the simplified inventory (i.e. only have to stock one type of delay in the cap magazine).

The 100/3800 combination is currently being used at a few mines in Ontario for narrow vein applications. From a blasting perspective, it was thought that the 100/3800 combination may offer some advantages over regular short period delays (25ms) for the following reasons:

- more time for burden relief (100ms as opposed to 25ms);
- the 3800ms in-hole delay allows 38 blastholes to be sequenced individually, as opposed to 18 with the typical range of short period delay numbers;
- reduced muck throw, making remote mucking more efficient;

Disadvantages of the system include:

- potentially too much delay scatter with the 100/3800 combination;
- the potential delay scatter problem could be worsened if different batches of 100/3800 detonators were mixed together in the cap magazine;
- when blasting one hole per delay, there is only one path for initiation, therefore, greater potential for a cut-off;
- the surface caps produce some shrapnel which can cause a cut-off;
- it is difficult to make a tidy hook-up, which makes it difficult to re-check the hook-up for errors;
- potentially coarser fragmentation due to the increased burden relief time.

The disadvantage of most concern was the potential for excessive delay scatter and subsequently the potential for blastholes shooting out of sequence. In narrow vein blasting, out of sequence firing can easily cause a blast to bench. This is largely because there are relatively few holes per row, therefore, an out of sequence hole may occur one or two rows behind the free face. If the out of sequence hole does not break its assigned burden it is likely the holes behind it will also bench.

Delay scatter is not a major issue with all dual delay detonator systems, just certain combinations. With the 100/3800 combination, the concern was that the scatter in the 3800ms in-hole delay may be too high for a 100ms separation between detonating blastholes. For example, with this combination there is a potential for holes shooting out of sequence if there is more than ± 50ms scatter between blasthole firing times. This amounts to 1.3% acceptable scatter in the 3800ms in-hole delay. In general, for pyrotechnic delay elements, delay scatter is typically in the range of ± 2.5% (Golder Associates Ltd., 1996).

11.2.2 Investigation Into Delay Scatter of the 100/3800 Combination

Tests for determining delay scatter were carried out in an old underground workshop with a concrete floor (170 lvl shop). The test procedure was as follows:

- a circle of approximately 4m diameter was painted on the floor;
- a triaxial array of geophones was bolted to the floor in the center of the circle;
- 3/8 inch diameter holes, 2.5 inches long, were drilled on an approximate 1m spacing around the perimeter of the circle (10 holes drilled per test);
- two lines of sash cord were then strung from the roof along the outside of the circle;
- twenty dual delay detonators were then hooked up in series with every second in-hole detonator placed in one of the holes drilled in the concrete floor. The remaining in-hole detonators were strung over the sash cord such that they detonated hanging in the air. All the surface connectors laid on the floor surface. This hook-up creates a theoretical 200ms separation between the in-hole detonators placed in the concrete floor;
- the dual delay detonators were initiated using an electric detonator (#0);

- the vibrations from the in-hole detonators located in the floor slab were measured using the blast monitor, the sampling rate used was 16000 Hz. The monitor was initiated using a wirebreak trigger attached to the electric starter detonator.

Twenty detonators from two different manufacturers were tested. An example of the test results are shown in Figure 11.6. Note that the air blast from some of the detonators on the sash cord was recorded by the monitor. An out of sequence firing can be seen near the end of the trace (i.e. strong signal - two air blasts - strong signal).

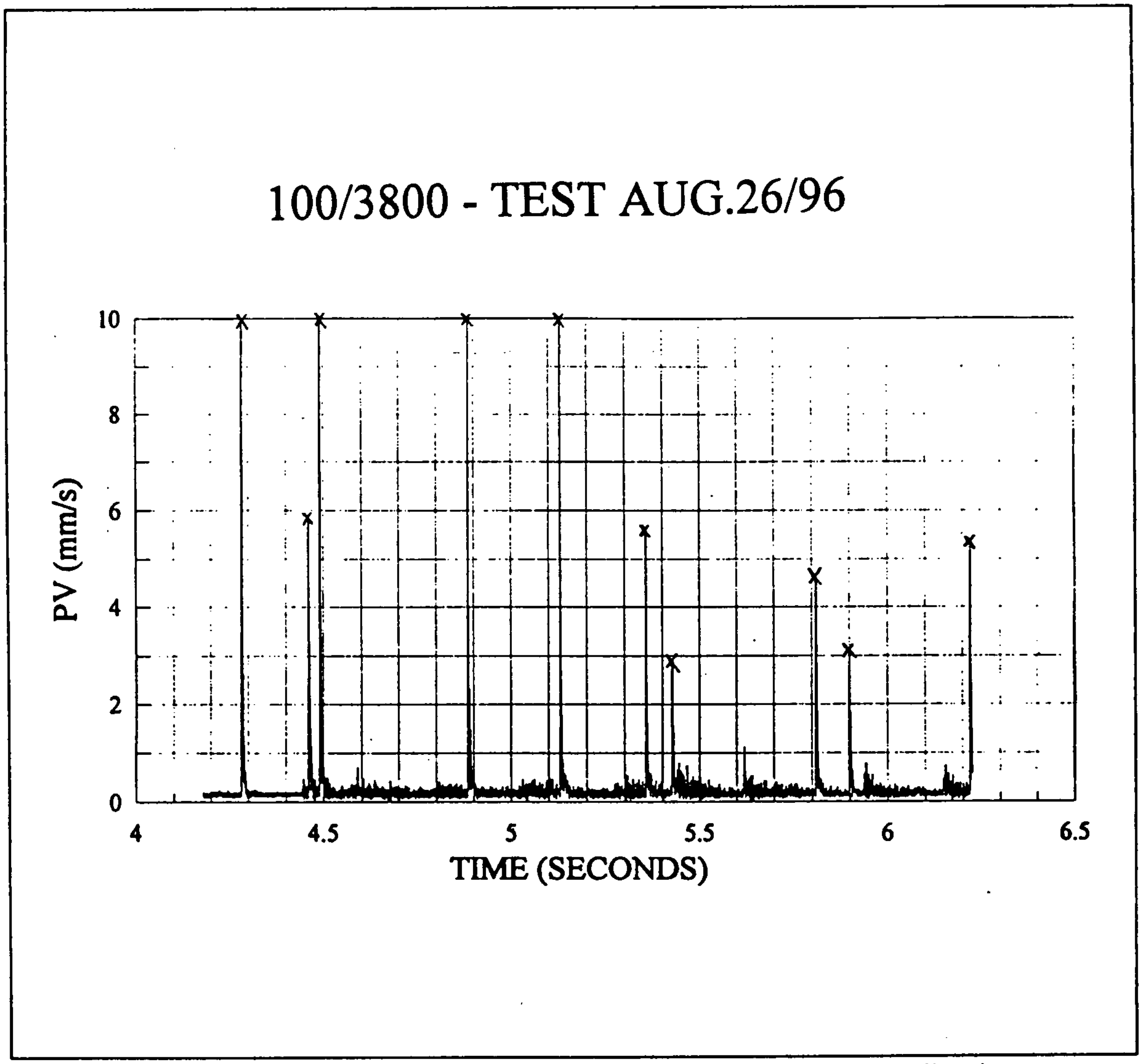


Figure 11.6 Investigation into delay scatter - 100/3800 combination - vibration monitoring results

The main findings from the tests are as follows:

- both tests confirmed a potential for out of sequence firings;
- the mean firing times varied significantly for each batch of detonators tested, indicating that mixing different batches could potentially cause serious sequencing problems;

Figure 11.7 shows histograms of delay scatter (%) for the two tests carried out. It is interesting to note that both batches exhibited mean firing times that were significantly different than the quoted 3800ms. For example Batch A was approximately 3950ms and Batch B was approximately 4200ms. As mentioned previously, acceptable delay scatter for the 100/3800 combination would be approximately $\pm$ 1.3% about the mean of a batch. Referring to Figure 11.7, it can be seen that both batches exceeded this value.

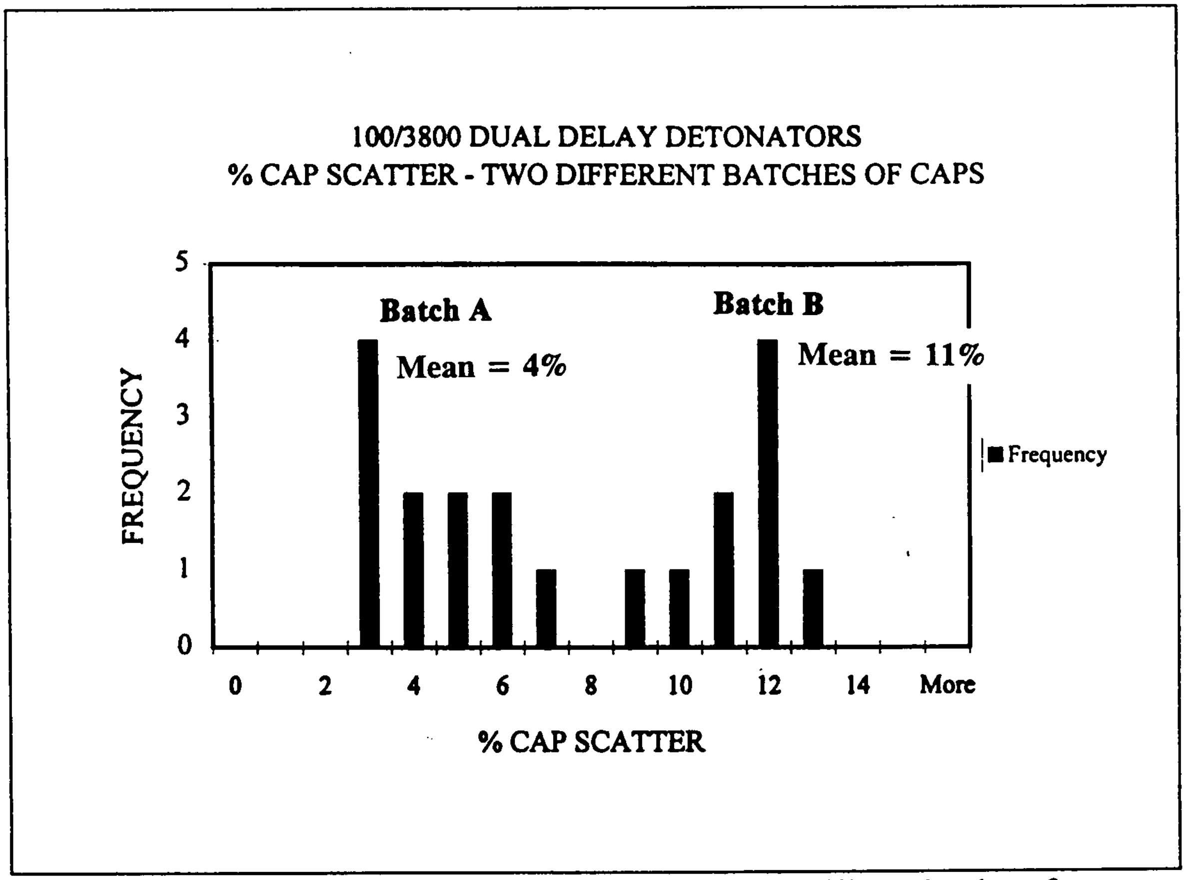


Figure 11.7 Histograms showing delay scatter (%) for two different batches of 100/3800 dual delay detonators

Although the amount of testing was not sufficient to say anything definitive, the results did prompt a discussion with one of the manufacturers who verified that their own plant statistics indicated a chance of out of sequence firing with the 100/3800 combination.

In spite of the testing, a decision was made to field trial the 100/3800 combination based on their apparent success at other operations.

11.2.3 100/3800 Field Trials

Field trials were conducted over a three month period (Oct. to Dec., 1996). During this time approximately 5 stopes were blasted using the 100/3800 dual delay detonators. Each stope on average required approximately 8 blasts to fully excavate. Typical stope size was 20m high with a strike length of 15m.

Initially the blasts were sequenced as before, using method (A) or method (B), refer to Figure 11.5. However, it was soon determined that due to delay scatter the blastholes were detonating as if they had all been sequenced individually, refer to Figure 11.8. Correspondingly the blast vibrations were considerably lower than those measured at the start of the study. These results made it relatively easy to convince the blasters that we should be designing to sequence the holes individually, since it was happening anyway. From the latter part of November until the end of December stope blasts were sequenced using method (C), refer to Figure 11.5.

During this period there was an observable improvement in the condition of the stope walls. Unfortunately during this period there was also a significant increase in the number of benched blasts. This increase in benching ultimately led to the discontinued use of the 100/3800 dual delay detonators. Although the increase in benching cannot be solely attributed to the detonators, it was considered to be a prime factor on a few occasions. The frequency of benches did decrease after the use of the 100/3800 dual delay detonators was discontinued.

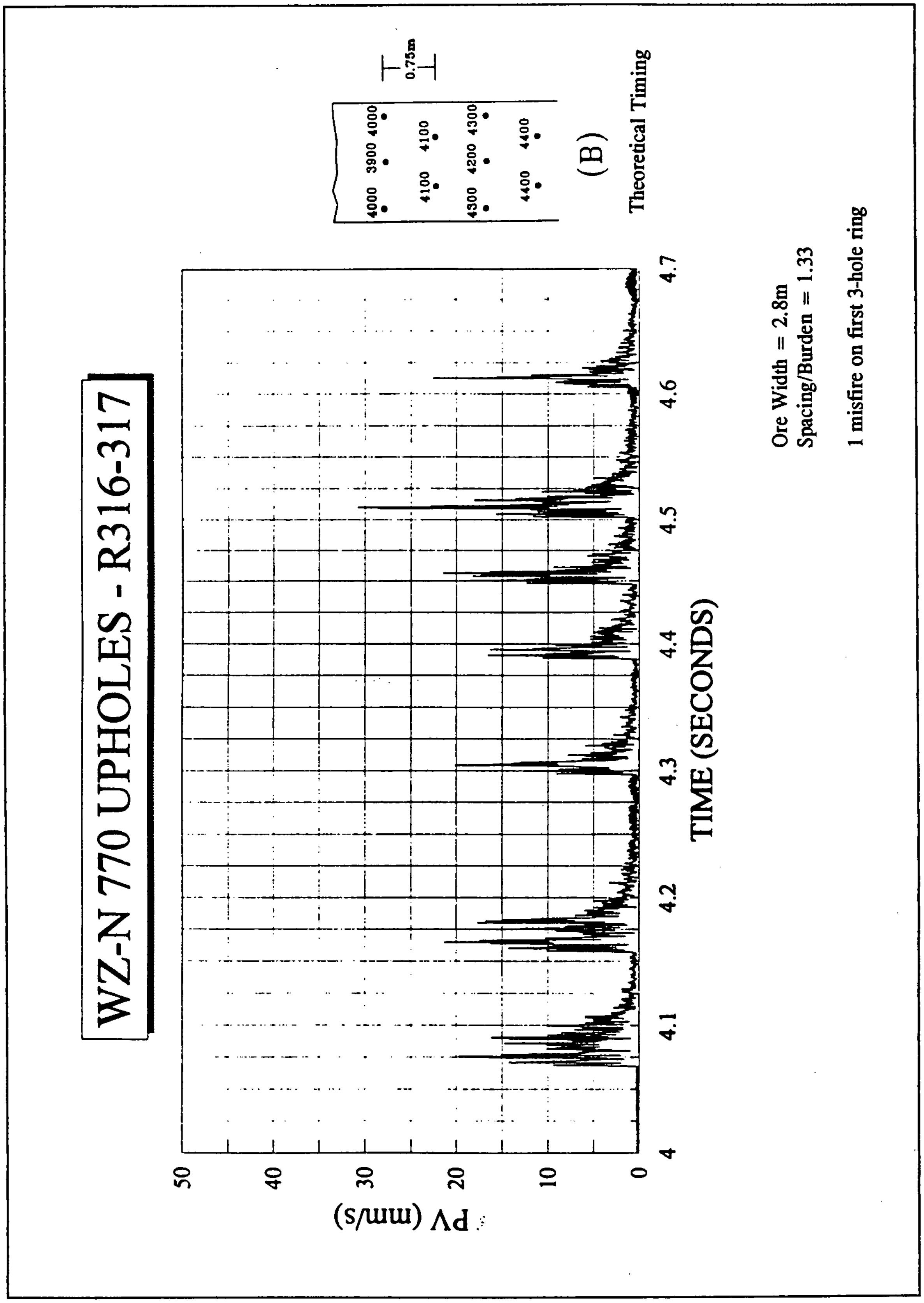


Figure 11.8 Example of 100/3800 field trial using sequencing method (B)

Results from the vibration monitoring are presented below:

- high degree of charge interaction as evidenced by a significant number of misfires and in-efficient detonations;
- the average number of misfires was 21%;
- with regard to the 21% misfires, 38% occurred on 3 hole rings, and 62% occurred on 2 hole (easer) rings;
- muck size was coarser than observed when using short period delays, in general muck size was good to coarse;
- the average PPV at approximately 35m plan distance was 31mm/s;
- wall damage was minimal;
- out of the 5 blasts monitored 3 benches were experienced .

11.3 TRIALS USING AN ALTERNATE TIMING SEQUENCE WITH B-LINE AND SHORT PERIOD DELAYS (MS-DELAYS)

Due to the benching problems experienced during the trials with the 100/3800 dual delay detonators, a switch back to MS-Delays was made. Initially the blasters reverted back to firing all the blastholes on a particular ring on the same delay number (sequencing method (A)). The argument used by the blasters was that they did not have enough confidence in the drilling to fire one hole at a time. In essence they were worried about holes crossing, largely because the spacing between the holes was so close. As a compromise, the sequencing shown in Figure 11.9 was proposed (sequencing method (D)). This timing sequence involves firing the blastholes on the 3-hole ring individually and firing the blastholes on the 2-hole ring on the same delay number. The logic is that if one of the holes on the 3-hole ring does not fire (due to: having the powder or primer dislodged, or, desensitization of the explosive column due to precompression) the 2-hole ring will have enough energy to break any remaining unbroken material.. The blastholes on the 2-hole ring are also offset from the orebody contacts on average 0.6m, therefore, there is more tolerance for blast damage resulting from having the two blastholes on the same delay number.

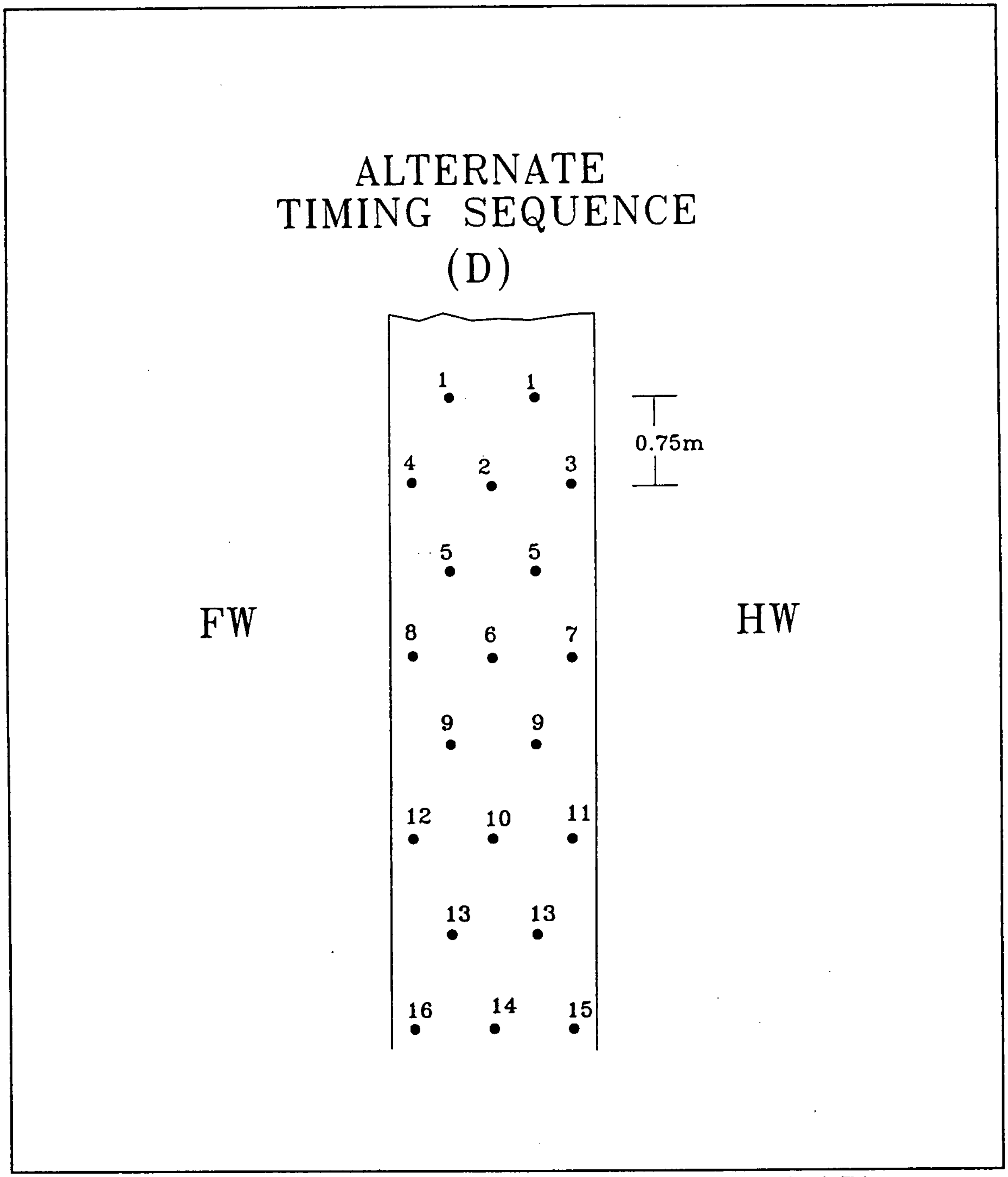


Figure 11.9 3:2 pattern - alternate timing sequence - sequencing method (D)

This timing sequence was trialed in Feb./97 and has since become the standard method of sequencing the 3:2 pattern in the WZ. Coincident with trialing the new timing sequence, the dynamite product which was being used for priming was replaced with 90g cast pentolite boosters. It was felt that the detonation pressure of the dynamite product was too low (VOD $\approx$ 3280 m/s), resulting in a long run-up distance for the Anfo to reach its steady state velocity of detonation (VOD $\approx$ 3300 m/s). This may be one reason poor breakage was being observed at the toes of upholes. Dynamites are also more susceptible to sympathetic detonation, thus, there is some risk of out of sequence firing with tight blasthole spacing, particularly if any hole deviation occurs. The 90g cast pentolite boosters have a high detonation pressure (VOD $\approx$ 8000 m/s) which results in a very short run-up distance for the Anfo and helps impart maximum energy to the toe area of the blastholes. The boosters also shield the detonators from dynamic shock and have the strength to overcome some precompression of the explosive column.

The new timing sequence and primers have been in use for approximately 9 months and the frequency of benches has been noticeably reduced. Occasionally the upholes will bench at the toes if too many rings are blasted (generally uphole blasts should not exceed 3 to 5 rings). This occurs largely because the holes are not dumped and thus have a poorer break-out geometry. Some blast damage to the walls still occurs but it is not as prevalent as it was when blasting each ring on the same delay number.

To date only two blasts have been monitored which were sequenced using the alternate timing described above. Preliminary results suggest vibration levels are similar to sequencing method (B), refer to Figure 11.1. The problem with misfires and in-efficient detonations remain.

11.4 SUMMARY OF 3:2 PATTERN BLAST MONITORING

Tables summarizing the blast monitoring results are presented in Appendix III. Additional monitoring information collected by Golder/CANMET (1989) and ICI/Lupin (1997) is also included.

11.4.1 Vibration Levels

The blast monitoring results show that the blast vibrations, and hence the potential for blast damage, are significantly impacted by the method used to sequence the blast. Figure 11.10 is a plot showing the average PPV (mm/s) versus plan distance to the center of the blast. A clear distinction in vibration levels can be seen between sequencing methods: (A) - MS Delays; (B) - MS Delays; and (C) - 100/3800 dual delay detonators. Sequencing method (D) appears to produce vibration levels similar to sequencing method (B).

Sequencing method (A) has high potential for blast damage, whereas sequencing method (C) has the lowest potential for blast damage. The difference in damage to the stope walls between methods (A) and (C) was easily observable in the field. Sequencing methods (B) and (D) fall somewhere between (A) and (C) and represent moderate blast damage potential. Rough design lines correlating charge weight per delay with observed blast damage are shown on the Figure.

A brief note of caution with regard to Figure 11.10 is warranted since it relates far field blast monitoring results to near field blast damage (i.e. the region close to the blasthole). Firstly, vibrations were measured at distances of approximately 20m to 40m from the blast. At these distances the blastholes can be considered "point sources" of blast energy (i.e. all the explosive in the blasthole has detonated prior to the blast energy reaching the geophone). Therefore, the vibrations measured are a function of both the charge weight per delay and the confinement felt by the charge (which is a function of the pattern design and timing). Secondly, it is important to realize that care must be taken when using charge weight per delay to compare blast damage potential in the near field. The reason being, is that in the near field blastholes do not act as "point sources" of energy due to the finite velocity of detonation of a given explosive. At any given point in time during the initiation of a blasthole, only a portion of the blasthole is contributing to strain energy (vibration) in the near field. Therefore, a better measure of blast damage potential is linear charge density (kg/m), refer to Holmberg and Perrson (1978). This is why from a near field blast damage perspective, there is little difference between a 9m long blasthole and an 18m long blasthole (given the same hole diameter, explosive type and pattern design) even though one has twice the charge weight of the other. However, away from the near field, where the blastholes can be considered "point sources", higher vibrations will be measured for the 18m long blasthole than the 9m long blasthole, because of the higher charge weight. The higher vibration level is not, however, indicative of greater blast damage potential in the near field. This is an important factor to keep in mind when comparing other far field blast monitoring results to Figure 11.10, otherwise erroneous conclusions might be drawn regarding blast damage potential.

A plot of PPV (mm/s) versus Scaled Distance ((distance to center of blast (m)) ÷ (charge weight per delay (kg))$^{1/2}$) is shown in Figure 11.11. The plot is shown more for interests sake than for design purposes. Plots such as this are generally used to try and predict far field blast damage (i.e. damage to surrounding drifts; and damage to equipment such as hoists, crushers, and conveyors). Since the plot is based on far field vibration measurements and incorporates charge weight per delay, it should be obvious based on the preceding discussion, that the results should not be extrapolated into the near field to try and predict damage in the region close to the blasthole.

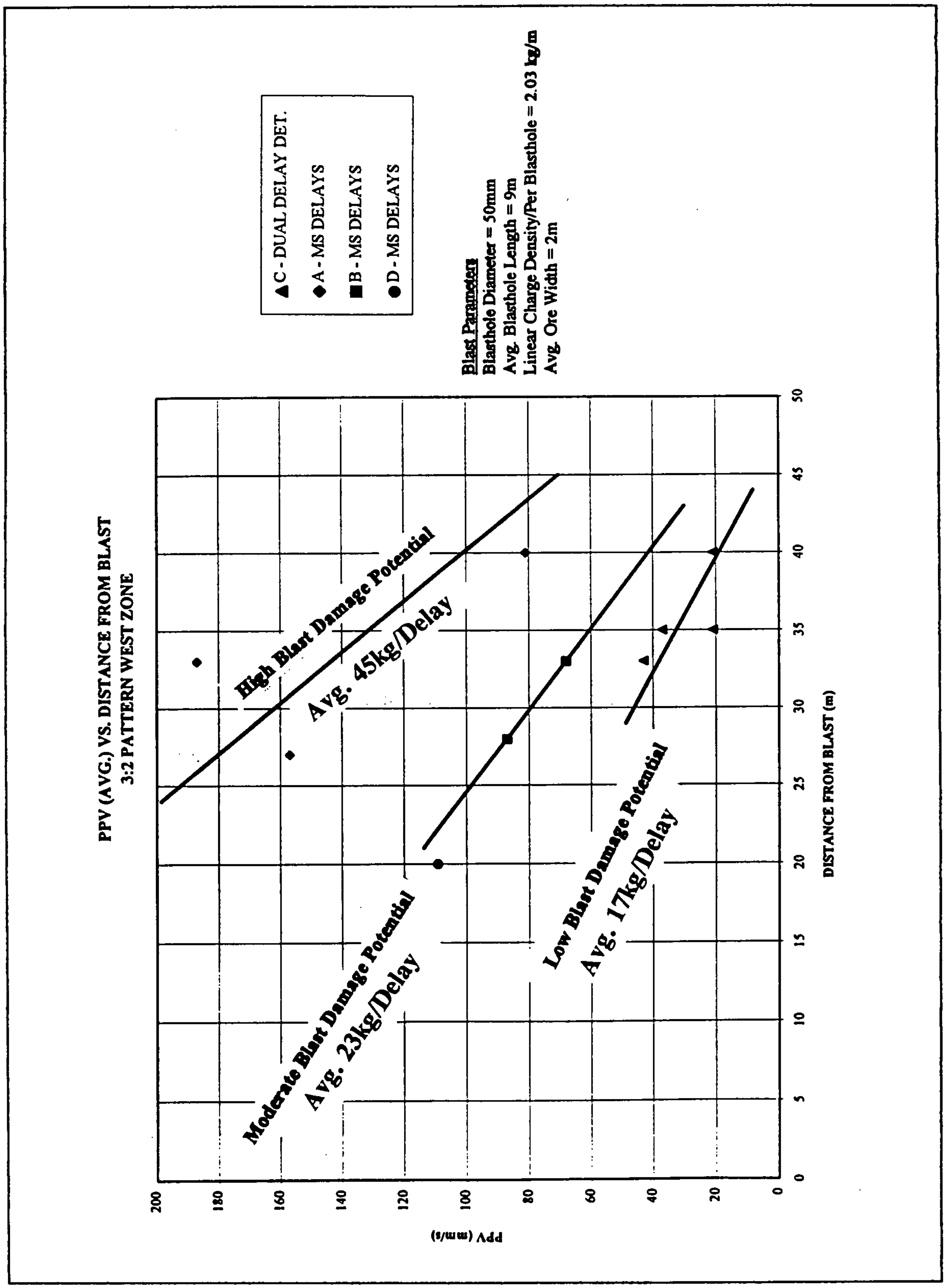


Figure 11.10 PPV (avg.) vs. Distance from Blast - 3:2 pattern blast monitoring results - showing effect of sequencing on blast vibrations and blast damage potential

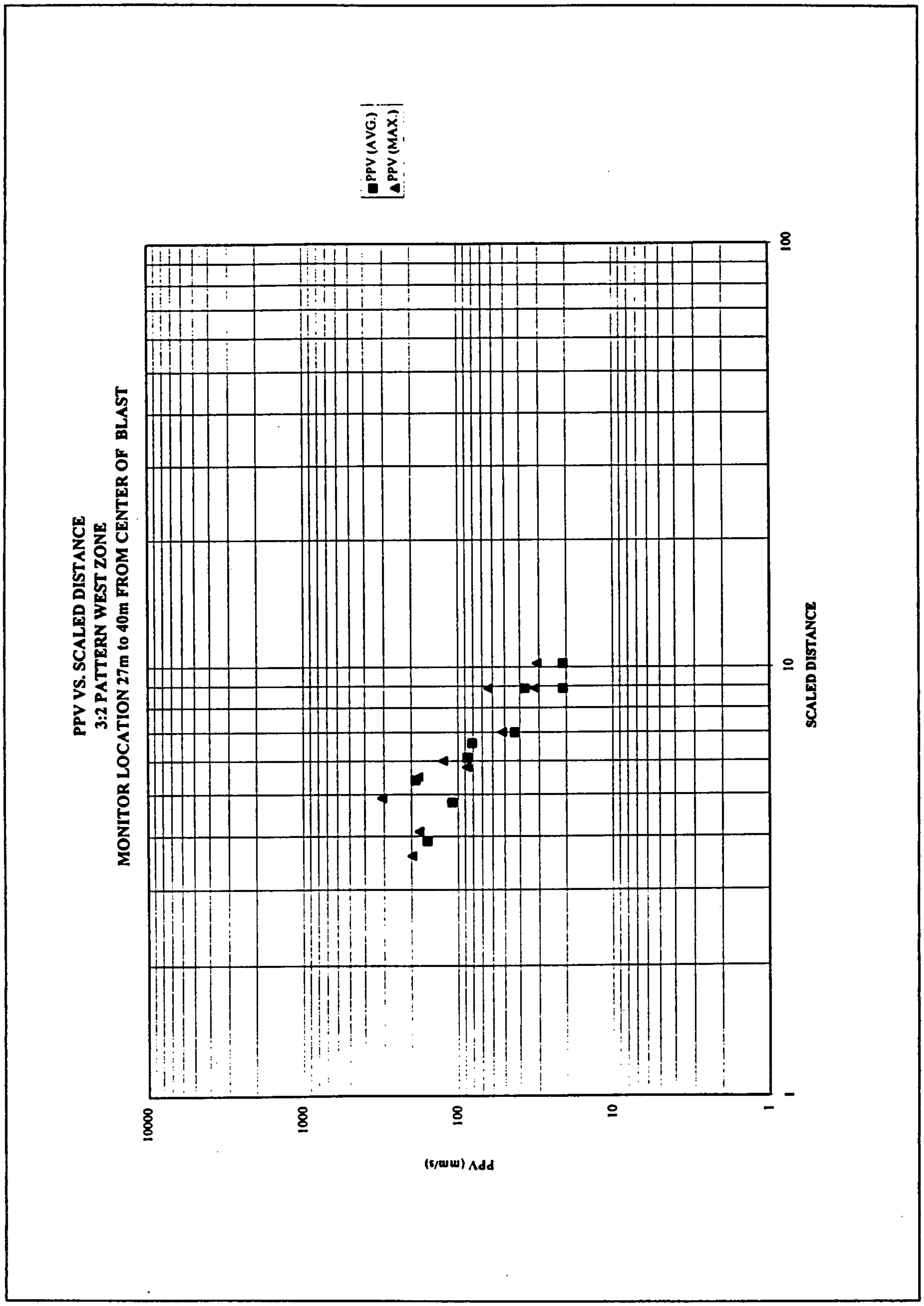


Figure 11.11 PPV vs. Scaled Distance - 3:2 pattern blast monitoring results

11.4.2 Charge Interaction (i.e. Misfires and Inefficient Detonations)

Although steps have been taken to reduce the vibration levels associated with the 3:2 pattern, the study has highlighted that the pattern is over-drilled (i.e. too many holes). This is evidenced by the high rate of misfires and inefficient detonations. Figure 11.12 shows a histogram of the "% Misfires" determined from blast monitoring. On average 14% misfires occurred, with a standard deviation of 9%. The misfires are largely a result of the tight blasthole spacing and are likely due to: dislodgement of adjacent charges; and/or desensitization of the explosive column due to precompression (dead-pressing).

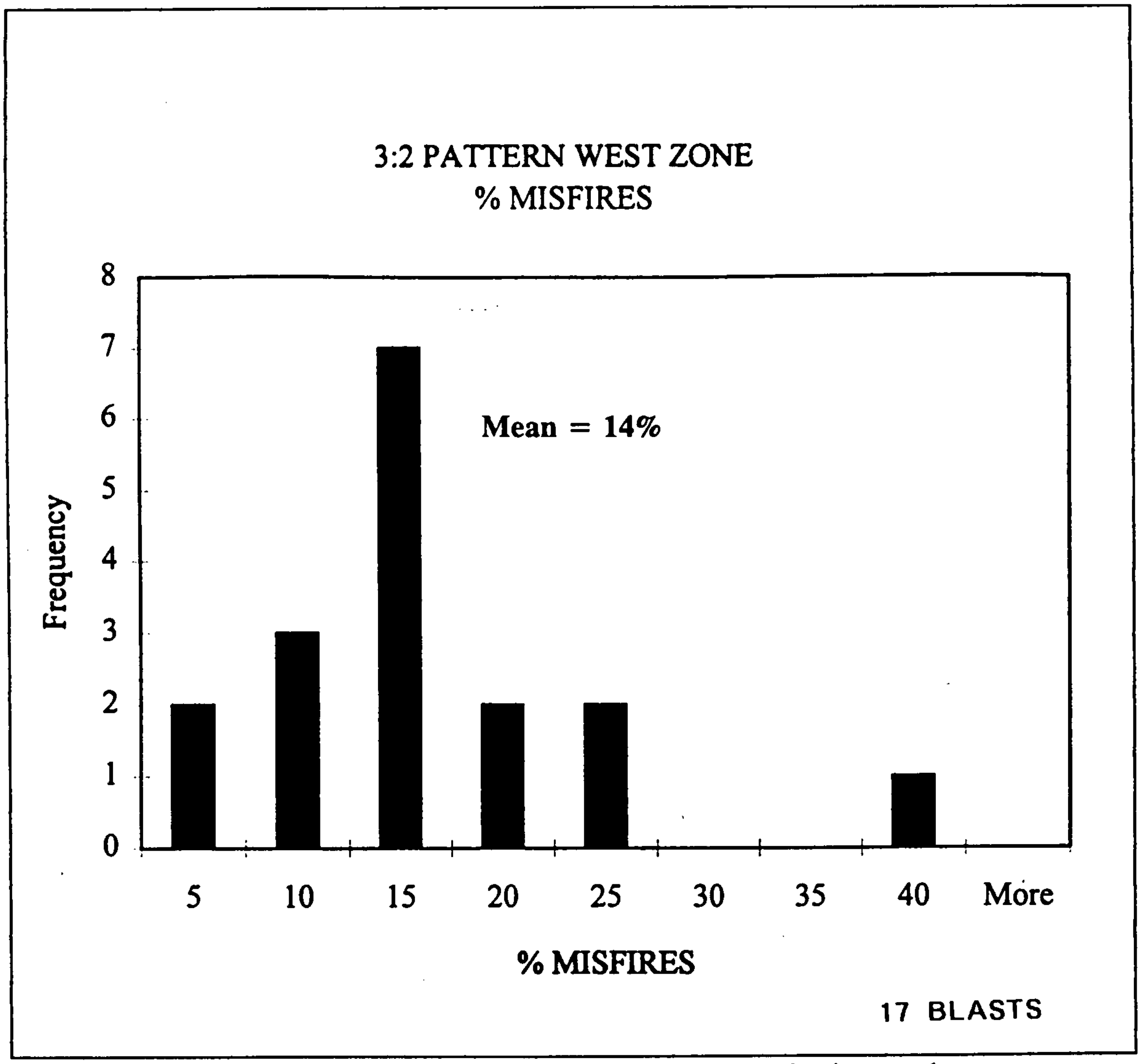


Figure 11.12 Histogram of % misfires - 3:2 pattern blast monitoring results

The high rate of misfires represents a significant cost. In essence, money is being spent on both drilling and explosives which are not being used to break the rock. In addition, undetonated Anfo finds its way into the mine water contributing to ammonia gas levels underground. Undetonated Anfo also adds additional contaminants such as nitrates to the mine water.

Given an average misfire rate of 14%, the direct cost of the misfires to the Lupin Mine can be calculated as follows:

Assume: WZ production = 204,000 stons/year (undiluted) ;
WZ longhole drilling = $15.98/m; stons/m drilled = 1.88 avg.(undiluted);
Explosive cost for a 9m blasthole = $22.39.

Note: Costs explained in greater detail in Chapter 12.

((204,000 stons/year ÷ 1.88 stons/m) x 14% misfires) x $15.98/m drilling = $242,760.00

((204,000 stons/year ÷ 1.88 stons/m) x 14% misfires) ÷ 9m hole length x $22.39/hole
= $37,793.00

Total Cost = $280,553.00

The above calculation demonstrates that there is a significant cost incentive to optimizing the blast pattern used in the WZ.

A prime factor to consider when optimizing, however, is that some of the flexibility in the pattern is eliminated. In otherwords, the quality of drilling and loading practices becomes critical, every blasthole is needed. One of the benefits of the current 3:2 pattern is that it does have some flexibility built into the pattern. For example, if a hole is plugged it is likely the blast will break adequately without bringing in a drill to redrill the hole. In addition, if loading practices are substandard or the drilling is particularly bad there is a good chance the blast will break adequately, mainly because there are more holes than required. If optimizing the blast pattern is to be successful a concerted effort has to be made to ensure blastholes are collared and drilled as designed and that proper cleaning and loading practices are enforced.

CHAPTER 12

POTENTIAL COST SAVINGS ASSOCIATED WITH IMPLEMENTING 1:1 (STAGGER) AND 2:1 (DICE-FIVE) BLAST PATTERNS

12.1 GENERAL - 1:1 AND 2:1 BLAST PATTERNS

As an alternative to the 3:2 pattern, both a 1:1 (stagger) pattern and a 2:1 (dice-five) pattern have been proposed. The 2:1 pattern has been used previously at the Lupin Mine, refer to Chapter 9, whereas the 1:1 pattern is new to the Lupin Mine. Figure 12.1 is a schematic showing the general layout of the 1:1 pattern.

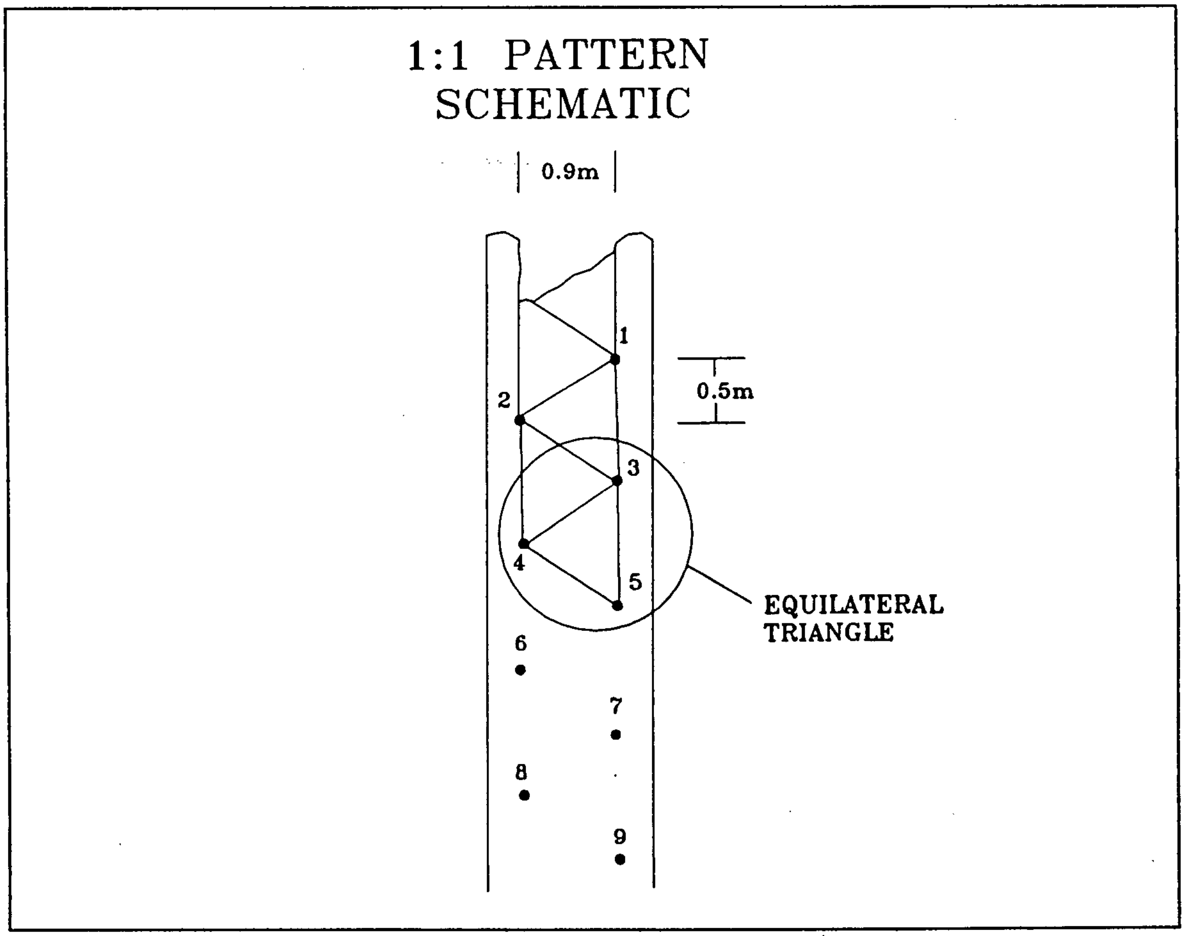


Figure 12.1 Schematic showing layout of 1:1 (stagger) pattern

The 1:1 pattern is being proposed for ore widths less than 1.8m and the 2:1 pattern is being proposed for ore widths between 1.8m and 2.5m. For ore widths greater than 2.5m the 3:2 pattern will be used.

It is also being proposed that a switch be made to 64mm diameter blastholes. The main reasons for the proposed change are as follows:

- the larger blasthole and stiffer drill string will reduce hole deviation;
- straighter blastholes will promote a move away from using a combination of up and downholes to all downholes which will break through into the level below;
- the drilling of break-through holes will allow easy checks of drilling accuracy;
- increased safety (i.e. blasters will not have to load upholes at open brows);
- the increased hole diameter will provide more tolerance towards squeezing of blastholes in stressed ground, which is becoming an issue at depth, and will reduce the amount of re-drilling required.

A potential drawback is that the larger diameter blasthole requires more explosive which could potentially result in increased levels of blast damage. However, through good pattern design (i.e. minimizing the confinement felt by the charge) and choice of appropriate offset distances, it is likely reductions in blast damage can be achieved. The following quote from Oriard (1970) in reference to reducing overbreak is appropriate:

"There does not have to be a decrease in the total quantity of explosives used, merely a change in the spatial distribution....The relationship would even apply if the powder factor were to be increased. Although this relationship seems obvious, it is unfortunate how often it is ignored in the field."

The actual pattern designs will be discussed in Chapter 13, the remainder of this section will examine potential cost savings associated with using the 1:1 and 2:1 patterns.

12.2 COST SAVINGS ASSOCIATED WITH REDUCED DRILLING AND EXPLOSIVE CONSUMPTION

12.2.1 Drilling (stons/m drilled)

The number of tons (short tons (stons) in the case of Lupin Mine) per meter drilled is a standard measure used for costing and planning purposes. Figure 12.2 is a plot showing short tons per meter drilled versus ore width. Referring to the plot, it can be seen that for the 2:1 pattern, even at very light burdens (i.e. 0.6m) significant reductions in the amount of drilling can be realized. With regard to the 1:1 pattern, it can be seen that even at an ore width of 1m the short tons per meter drilled is higher than the 3:2 pattern in ore 1.5m wide! The other point to bear in mind is that with the 3:2 pattern any ore narrower than 1.5m is not mined. Over the years a considerable amount of narrow ore has been left behind as pillars.

With regard to drilling 64mm diameter blastholes, the cost savings associated with the reduced drilling for the 1:1 and 2:1 patterns will be offset somewhat due to the increased cost of drill steel and bits (i.e. FI 38 drill rods vs. R32 drill rods).

12.2.2 Explosives Consumption

Figures 12.3 and 12.4 are plots of powder factor (kg's Anfo / short ton) versus ore width for 50mm and 64mm diameter blastholes respectively. Referring to the figures, it can be seen that with the 2:1 pattern and 50mm diameter blastholes, powder factors and hence explosive consumption are considerably lower than the 3:2 pattern, even at small burdens (i.e. 0.6m). With the 64mm diameter blastholes, the powder factor is approximately the same as the 3:2 pattern at a burden of 0.6m. For burdens greater than 0.6m the powder factor is lower than the 3:2 pattern. With regard to the 1:1 pattern and 50mm diameter blastholes, the powder factor is lower than the 3:2 pattern (1.5m width) up until ore widths of less than a 1m. With 64mm blastholes, the powder factor is lower up until ore widths of less than approximately 1.1m.

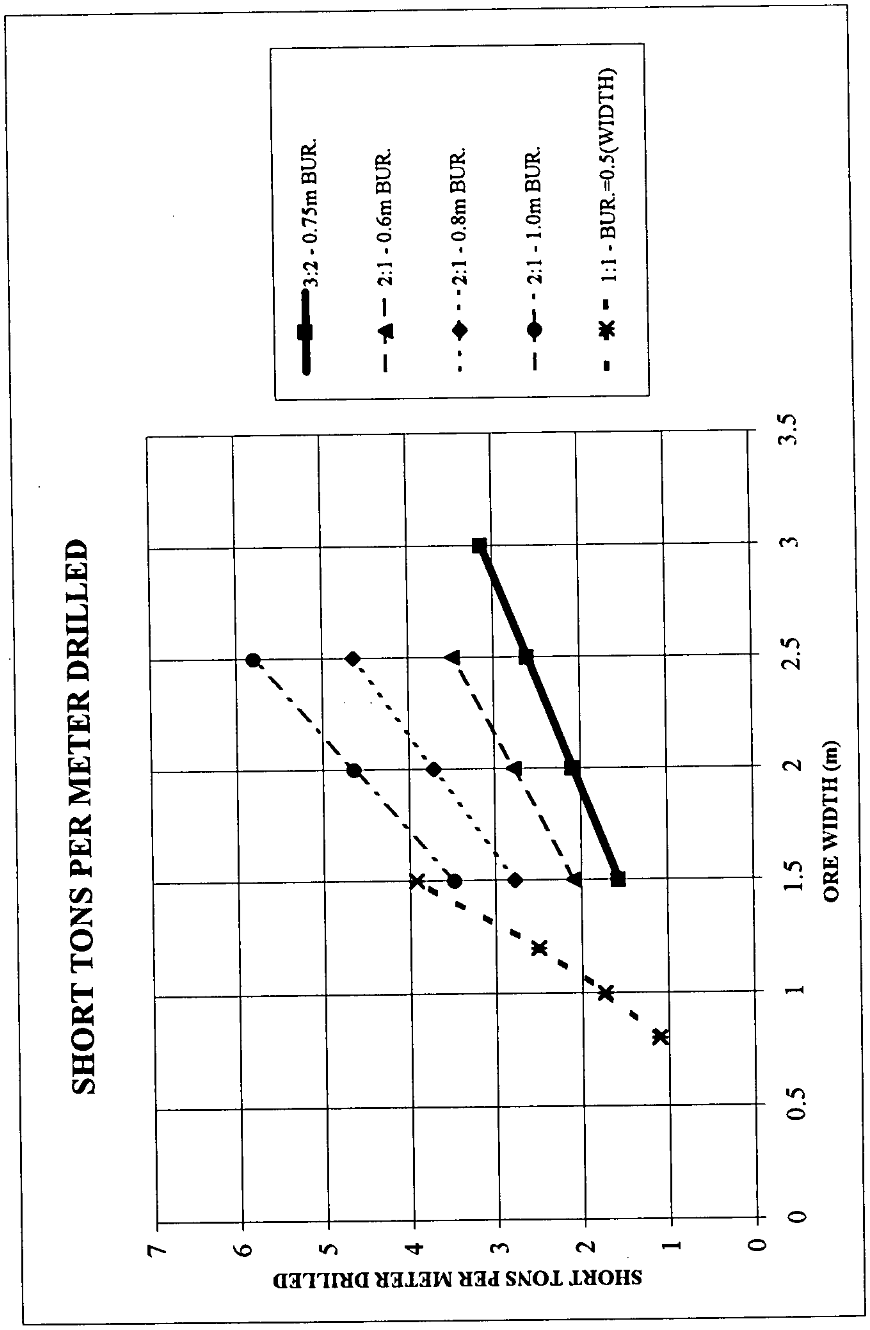


Figure 12.2 Short tons per meter drilled vs. Ore width - for various blast pattern geometries

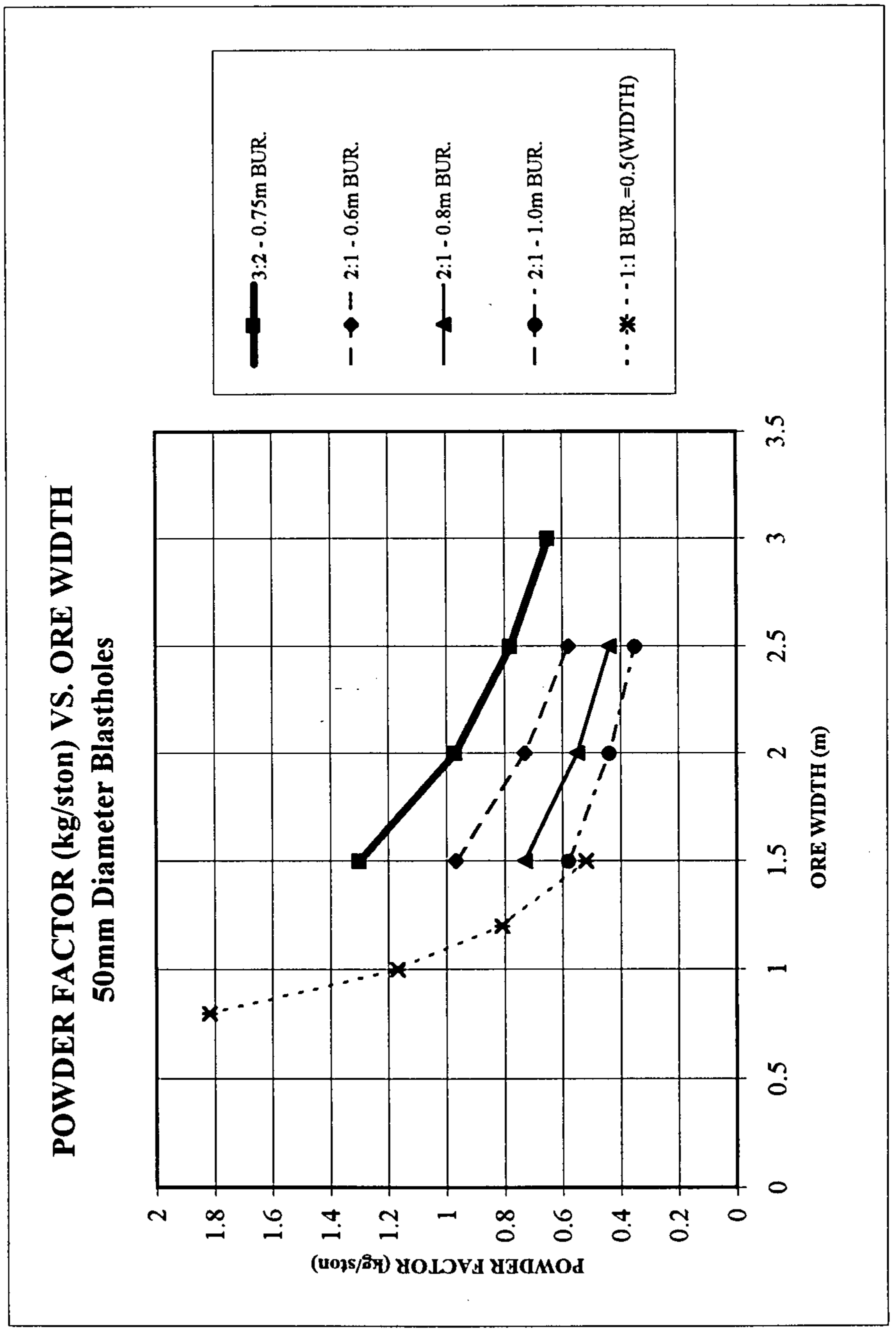


Figure 12.3 Powder factor vs. Ore width - 50mm diameter blastholes

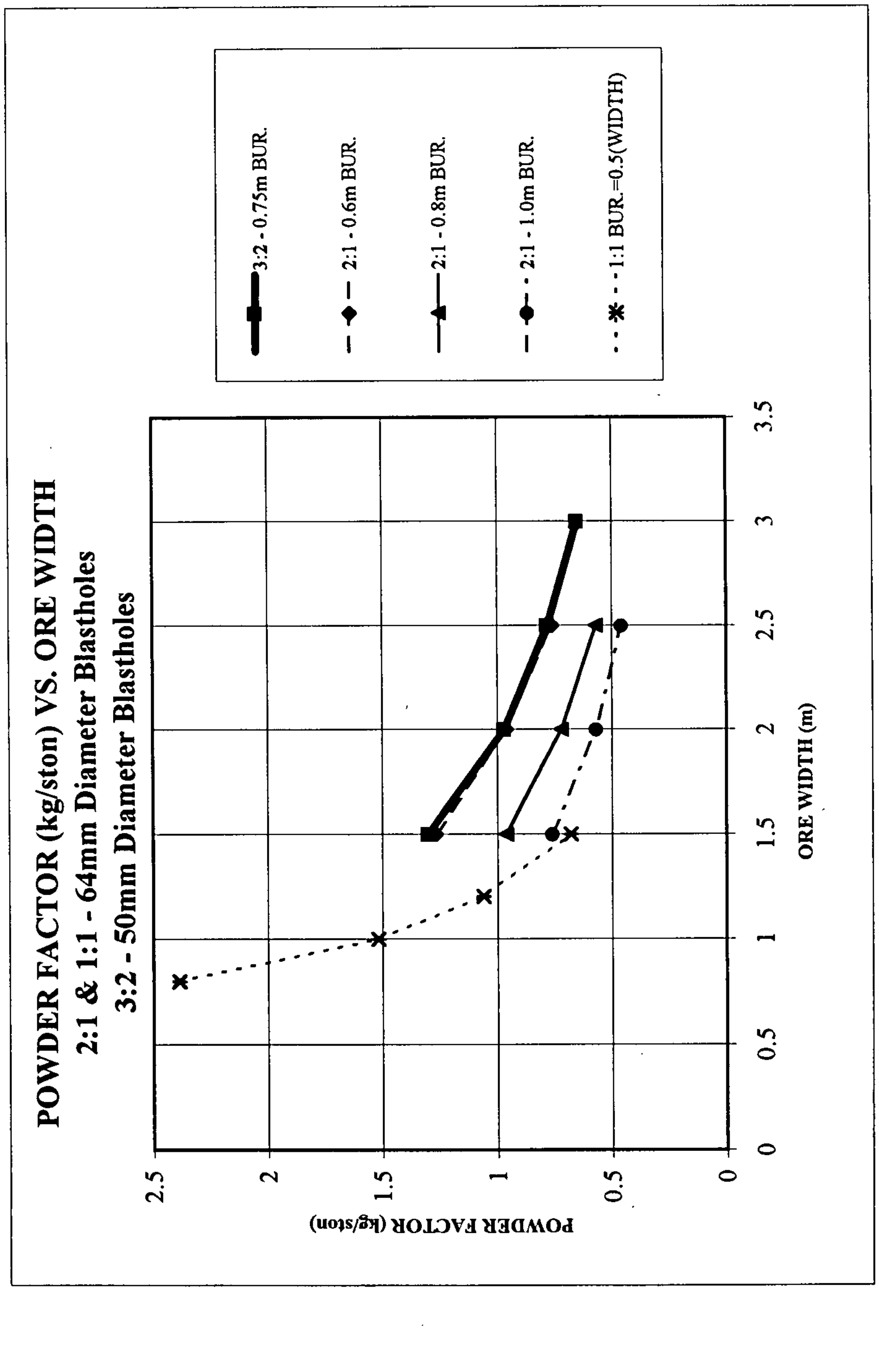


Figure 12.4 Powder factor vs. Ore width - 64mm diameter blastholes

Figure 12.5 is a bar chart showing the number of blastholes per meter of strike length for the various patterns and burdens. The chart is useful for determining what kind of savings can be made on products such as detonators and boosters. As can be seen, even at small burdens, there is a significant reduction in the numbers of detonators and boosters required with the 1:1 and 2:1 patterns, as compared to the 3:2 pattern.

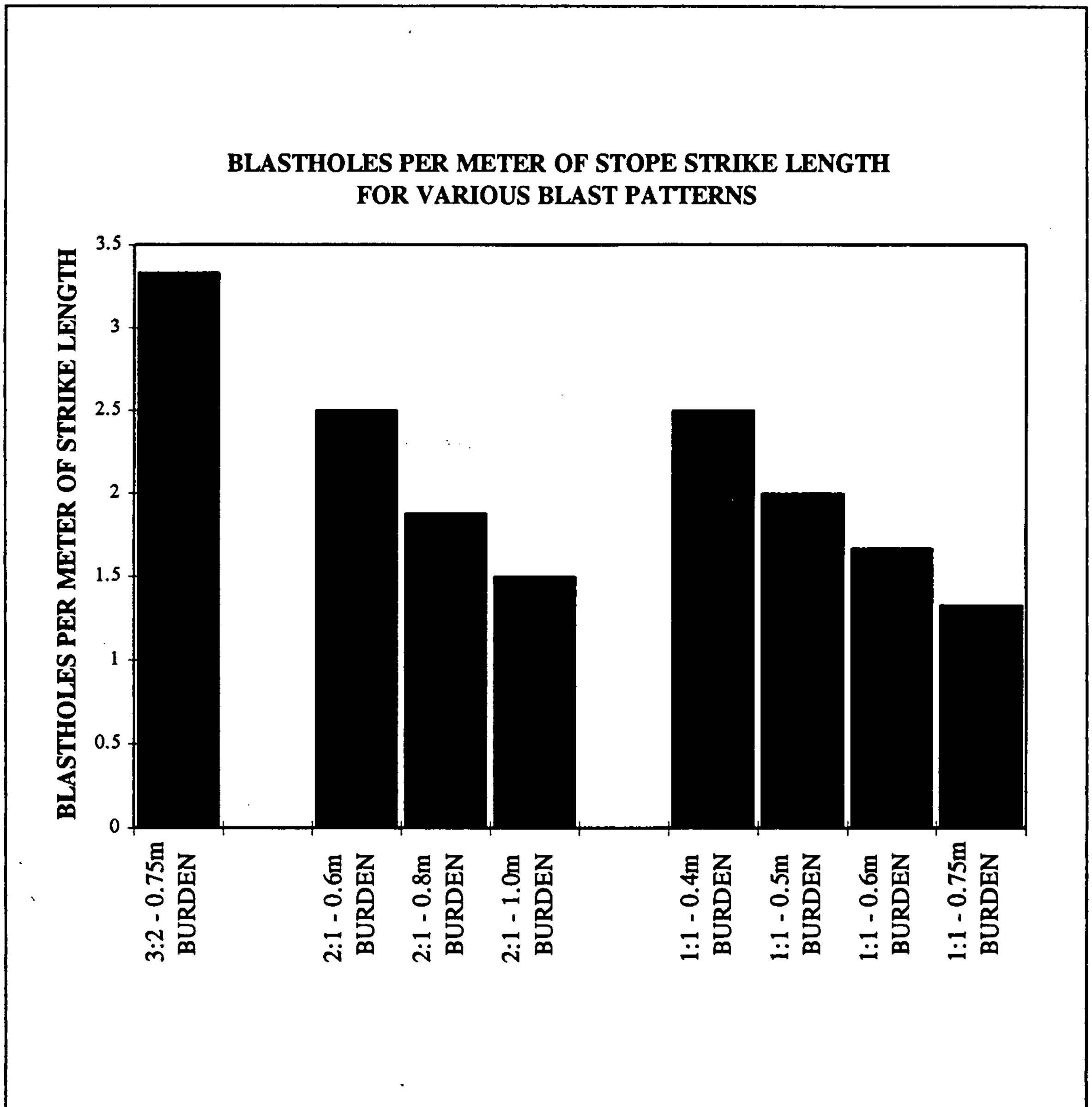


Figure 12.5 Bar chart showing number of blastholes per meter of stope strike length for various blast pattern geometries

12.2.3 Cost Comparison Between the 3:2, 2:1, and 1:1 Blast Patterns

The analysis presented in this section attempts to assign some real numbers to the cost savings that will be realized through the reduced drilling and explosive consumption associated with the 1:1 and 2:1 patterns. Figure 12.6 shows the breakdown of drilling and blasting costs used for the analysis. It can be seen from the costs that the majority of savings will be realized through reduced drilling. Note that the labour for cleaning and loading blastholes has not been included, however, it is expected that the reduced labour associated with the 1:1 and 2:1 patterns (fewer holes to load and clean) will result in additional cost savings that have not been accounted for in this analysis.

An example of how blast pattern costs were calculated is given below:

Example: <u>3:2 Pattern Cost: Ore Width = 1.8m; stons/m drilled = 1.88 (from Figure 12.2)</u>

1 ston ÷1.88 stons/m = 0.53m of required drilling.
Drilling Cost = 0.53m x $15.98/m (50mm blastholes) = $8.50
<u>Explosive Consumption = 0.53m x $2.49/m (50mm - up and down holes) = $1.32</u>

Total Cost = $9.82/ston

A summary of the analysis is presented in Tables 12.1 and 12.2. Note that an ore width of 1.8m (avg. WZ ore width) was assumed for the 2:1 pattern analysis.

With regard to the 2:1 pattern, all the scenarios investigated show a cost savings over the current 3:2 pattern. The lowest calculated savings was 10% which was associated with drilling 64mm diameter blastholes (up and downholes) with a 0.6m burden.

DRILLING AND BLASTING COSTS

BLASTING MATERIAL COSTS

	UNIT COST	50mm DIAM. BLASTHOLE 9m LENGTH	50mm DIAM. BLASTHOLE 18m LENGTH	65mm DIAM. BLASTHOLE 9m LENGTH	65mm DIAM. BLASTHOLE 18m LENGTH
*ANFO (50mm DIAM. BLASTHOLE)	$1.30/m	$10.40	$22.10	NA	NA
*ANFO (64mm DIAM. BLASTHOLE)	$2.03/m to $1.70/m	NA	NA	** $14.92	$28.90
BOOSTERS	$2.52 ea.	$5.04	$5.04	$5.04	$5.04
4m DETONATOR	$2.15 ea.	$2.15	NA	$2.15	NA
12m DETONATOR	$4.30 ea.	$4.30	$4.30	$4.30	$4.30
18m DETONATOR	$6.00 ea.	NA	$6.00	NA	$6.00
STARTER DETONATOR	$1.60 ea.	$0.10	$0.10	$0.10	$0.10
B-LINE	$0.40/m	$0.40	$0.40	$0.40	$0.40
COST/BLASTHOLE		$22.39/hole	$37.94/hole	$26.91/hole	$44.74/hole
COST/m OF BLASTHOLE		$2.49/m	$2.11/m	$2.99/m	$2.49/m

NOTES:

* ANFO - $16 / 25kg; Pneumatic loading density = 2.03 kg/m for 50mm diam. hole; 3.17 kg/m for 64mm diam. hole.
- poured density for 64mm diam. hole = 2.66 kg/m;
- a 1m collar (not loaded) was assumed;

** - represents the average cost between pour loaded and pneumatically loaded anfo. See notes below.

9m blasthole lengths correspond to mining with up and downholes. With the 50mm blastholes both up and downholes are pneumatically loaded. With the 64mm blastholes it is assumed the upholes are pneumatically loaded and the downholes pour loaded.

18m blasthole lengths correspond to mining entirely with downholes. It is assumed 50mm blastholes will be pneumatically loaded whereas 64mm blastholes will be pour loaded.

WZ LONGHOLE DRILLING COSTS

	UNIT COST	REMARKS
TAMROCK MICRO-SOLO 50mm DIAM. BLASTHOLES	$15.98/m	Cost Includes: labour; materials; maintenance; hourly rate for drill
TAMROCK MICRO-SOLO 64mm DIAM. BLASTHOLES	*$19.00/m	Cost Includes: labour; materials; maintenance; hourly rate for drill

NOTES:
- 50mm diam. blastholes drilled using R32 M/F drill steel.
- 64mm diam. blastholes drilled using FI38 M/F drill steel.

The unit cost for 50mm diam. blastholes is based on actual tracked costs for longhole drilling in the WZ.

*The unit cost for 64mm diam. blastholes is based on actual tracked costs for longhole drilling in the CZ using a regular SOLO drill. Costs are expected to be similar in the WZ. The WZ longhole drills are equipped with Montabert HC-80 drifters which are well suited for FI 38 components. As a note, FI 38 drill steel, guide rods, striker bars, and bits, cost approximately 45% more than R32 drill components.

Figure 12.6 Breakdown of drilling and blasting costs for cost analysis

TABLE 12.1
COST COMPARISON - 3:2 PATTERN VS. 2:1 PATTERN
(ASSUMES ORE WIDTH OF 1.8m)

	3:2 PATTERN	2:1 PATTERN		2:1 PATTERN		2:1 PATTERN	
	CURRENT	50mm Diam. Blastholes - Up and Downholes		64mm Diam. Blastholes - Up and Downholes		64mm Blastholes - Downholes (Breakthrough)	
		0.6m BURDEN	0.75m BURDEN	0.6m BURDEN	0.75m BURDEN	0.6m BURDEN	0.75m BURDEN
COST/STON	$9.82	$7.39	$5.90	$8.80	$7.03	$8.59	$6.87
SAVINGS (%)	NA	25%	40%	10%	28%	12.50%	30%

NOTES:
- Pattern costs determined using costs presented in preceeding table ("Drilling and Blasting Costs").
- Pattern costs do not include labour associated with cleaning and loading the blastholes.

TABLE 12.2
1:1 PATTERN COSTS

	50mm Diameter Blastholes Up and Downholes				64mm Diameter Blastholes Up and Downholes				64mm Diameter Blastholes Breakthrough Downholes			
ORE WIDTH:	0.8m	1.0m	1.2m	1.5m	0.8m	1.0m	1.2m	1.5m	0.8m	1.0m	1.2m	1.5m
COSTS/STON:	$16.64	$10.61	$7.36	$4.72	$19.81	$12.64	$8.76	$5.61	$19.36	$12.35	$8.56	$5.48

NOTES:
- Pattern costs determined assuming the burden for the 1:1 pattern is approximately equal to 1/2 the ore width.
- Pattern costs determined using costs presented in preceeding table ("Drilling and Blasting Costs").
- Pattern costs do not include labour associated with cleaning and loading the blastholes.

The cost of the current 3:2 pattern at an ore width of 1.5m = $11.76/ston

With regard to the 1:1 pattern, the cost per ston is lower than the 3:2 pattern (1.5m width) down to ore widths of approximately 0.9m when using 50mm diameter blastholes, and down to ore widths of approximately 1.1m when using 64mm diameter blastholes. At an ore width of 1.5m the 1:1 pattern shows a cost savings of approximately 60% when using 50mm diameter blastholes, and approximately 50% when using 64mm diameter blastholes.

Currently in the WZ, yearly drilling and explosive consumption costs (calculated using the method shown above) amount to just over $2,000,000.00. Even small cost savings (i.e. 10-20%) amount to $200,000.00 to $400,000.00 in savings. In addition, by using the 1:1 pattern, mining ore widths down to approximately 1m does not represent any cost increase over current costs. Thus it may be possible to mine ore that would otherwise be left behind, thus increasing mineable reserves.

12.3 COST SAVINGS ASSOCIATED WITH REDUCING UNPLANNED DILUTION

A very significant cost savings can be realized if blast damage to the stope walls, and hence the amount of unplanned dilution, can be minimized. Even though efforts have been made to reduce the blast damage potential associated with the 3:2 pattern, blast damage is still a concern, refer to Chapter 11.

The approximate cost of unplanned dilution in the WZ can be summarized as follows:

Activity	Cost/ston
Muck/Haul	$5.00
Crush/Conv./Hoist	$4.34
Milling	$12.10
Paste Backfill	$4.20
U/G Services	$7.54
Mine Power	$3.43
Mine Supervision	$2.28
Infrastructure	$32.00
Total:	**$70.89**

Total Minus:
Infrastructure; Services; Power; and Supervision: $25.64

The question that arises is, what cost should be assigned to unplanned dilution? If unplanned dilution is not prohibiting the mine from making its daily quota of ounces (which carries the cost of infrastructure, services, etc..), the cost should be that of mucking, crushing, hoisting, milling, and backfill ($25.64/ston). However, there is a lost opportunity cost since the unplanned dilution could have been ore. If the unplanned dilution is prohibiting the mine from making its daily quota of ounces, then some of the dilution must also bear the cost of infrastructure, services, power, and supervision ($70.89/ston). Again, there is also a lost opportunity cost due to mining waste instead of ore. There may also be additional costs if excess dilution effects mill recoveries.

With regard to this study, $25.64/ston will be used for the cost of unplanned dilution, recognizing that this is the most conservative approach and that greater cost savings may actually be realized if a significant reduction in unplanned dilution can be accomplished.

Unplanned dilution is estimated at approximately 40% (waste/ore) for the WZ. This amounts to approximately 136,000 stons/year of below cutoff grade material. At $25.64/ston, the annual cost of unplanned dilution is approximately $3,487,040.00.

From the above, it can be appreciated that if through optimizing the blast pattern a reduction in dilution can be realized, very significant cost savings will result.

12.4 OTHER POTENTIAL COST SAVINGS

If a change to drilling all downholes is implemented further cost savings will likely be realized. The cost savings will result from the fact that drilling accuracy can easily be determined by surveying in the toes and collars of breakthrough holes. If as a standard procedure, holes are checked and holes re-drilled when necessary, the number of benches should decrease. Fixing benches costs a significant amount of money in terms of time, possibly having to re-establish a slot, and lost reserves. In addition, damage to the stope walls invariably occurs during the process of trying to fix the bench. Furthermore, through elimination of the upholes and increased confidence in the downhole drilling accuracy, bigger blasts should be possible. This will permit more efficient use of time for the blasters and scoop operators, and should reduce the amount of rehab work required at the mucking brows (i.e. bigger blasts = less brows). Possibly the biggest benefit of switching to all downholes is the increased safety for workers, since it eliminates having to re-drill or load upholes near the stope brow.

CHAPTER 13

1:1 AND 2:1 PATTERN DESIGN
AND
FIELD TRIALS

13.1 1:1 PATTERN DESIGN

The 1:1 pattern (staggered) is commonly used in civil work for applications such as trenching. The only reference found in which the pattern was being used for longhole blasting was from Coolgardie Gold NL, Australia, Mills (1991). No information could be found in reference to the 1:1 pattern being used in Canadian mines. Lizotte (1989) states that the pattern has found application in narrow vein reef-type mining in South Africa.

At Coolgardie Gold NL, the following ore width / burden relationship is used:

Ore Width	Burden
0.8m	0.5m
1.0m	0.6m
1.2m	0.7m

The staggered blastholes are laid out such that they form equilateral triangles, resulting in design breakout angles of 60°. The blasthole diameter used is 57mm. Upholes are used exclusively (breakthrough) and have a length of approximately 11m. Anfo is the primary explosive. Overbreak was stated to be minimal. The 1:1 pattern is used only up to ore widths of 1.5m. For ore widths greater than 1.5m, either a 2:2 or 2:1 pattern is used.

For field trials at the Lupin Mine, test stopes were designed using both 50mm and 64mm diameter blastholes. In both cases, the blastholes were staggered such that they formed equilateral triangles. The trial using 50mm diameter blastholes used a burden (ring spacing) of 0.4m, which results in a design spacing of approximately 0.7m between the staggered blastholes. The trial using 64mm diameter blastholes used a burden (ring spacing) of 0.5m, which results in a design spacing of approximately 0.9m between the staggered blastholes.

One drawback of the 1:1 pattern is the tight breakout angles (i.e. 60°). This potentially results in higher vibrations and excessive shearing and tearing along the stope walls. To help compensate for the tight breakout angles, the blastholes were purposely given small burdens to help reduce the confinement felt by the charge. With regard to the 64mm diameter blasthole, the designed burden is very small, it was hoped that the low confinement would help compensate for the greater blast damage potential due to the higher charge weight per blasthole.

13.2 1:1 PATTERN FIELD TRIALS

13.2.1 Trial with 50mm Diameter Blastholes

The field trial using the 50mm diameter blastholes was conducted in an area that was initially planned to be left as a waste pillar (WZS 890). The trial involved drilling eighteen rings of upholes (1 hole per ring). The upholes were vertical (i.e. no dip or dump) with a designed length of 7m. The explosive used for the trial was pneumatically loaded Anfo. Each hole was double primed (at the toe and 3m from the collar) with 90 gram cast pentolite boosters. Uncharged collar lengths varied between 0m - 0.4m. The blastholes were sequenced using MS-Delays (25ms between each hole). Unfortunately, the stope blast previous to the trial (3:2 pattern) had benched approximately 1.8m at the toes. This effectively reduced the hole length for the test blast to 5.2m. Figure 13.1 is a long-section showing the blast geometry.

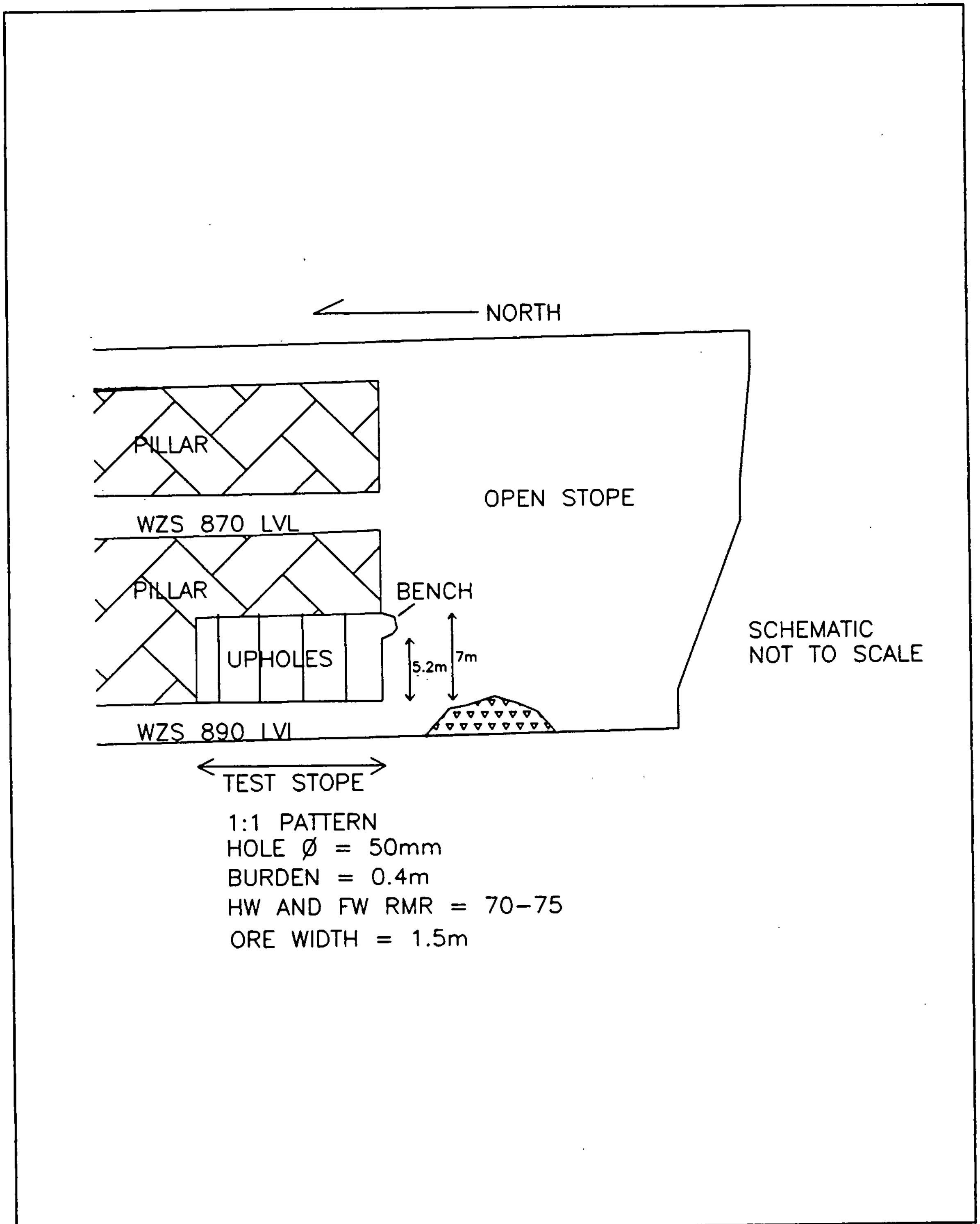


Figure 13.1 Long section schematically depicting WZS 890 trial area - 1:1 pattern; 50mm diameter blastholes; 0.4m burden

All eighteen holes were shot in one blast. A plot of the vibration monitoring results (measured 30m from blast) is shown in Figure 13.2. Results from a CMS survey of the stope are presented in Figure 13.3. Photographs taken after the blast are shown in Figure 13.4.

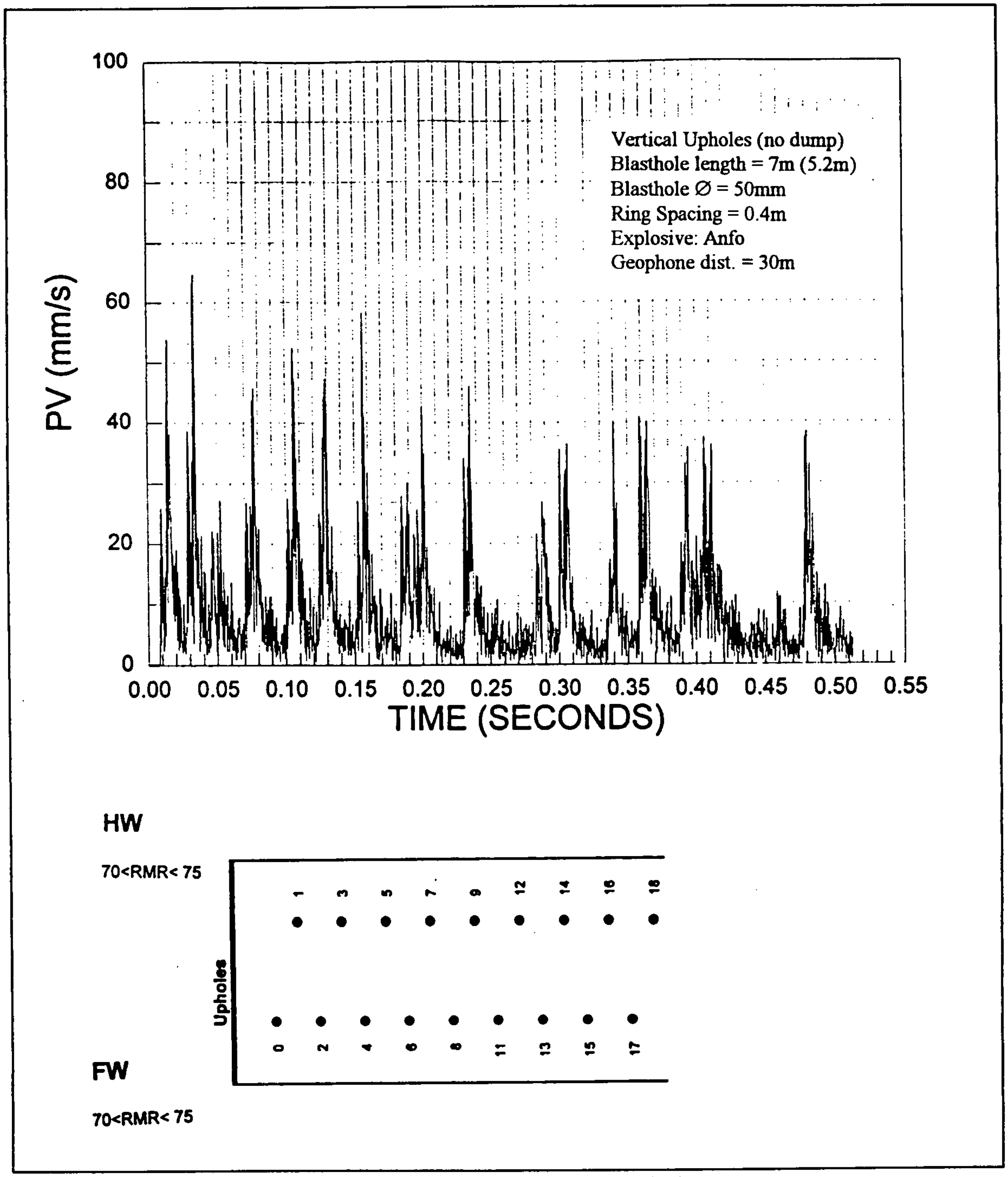


Figure 13.2 Vibration monitoring results - WZS 890 upholes; 1:1 pattern

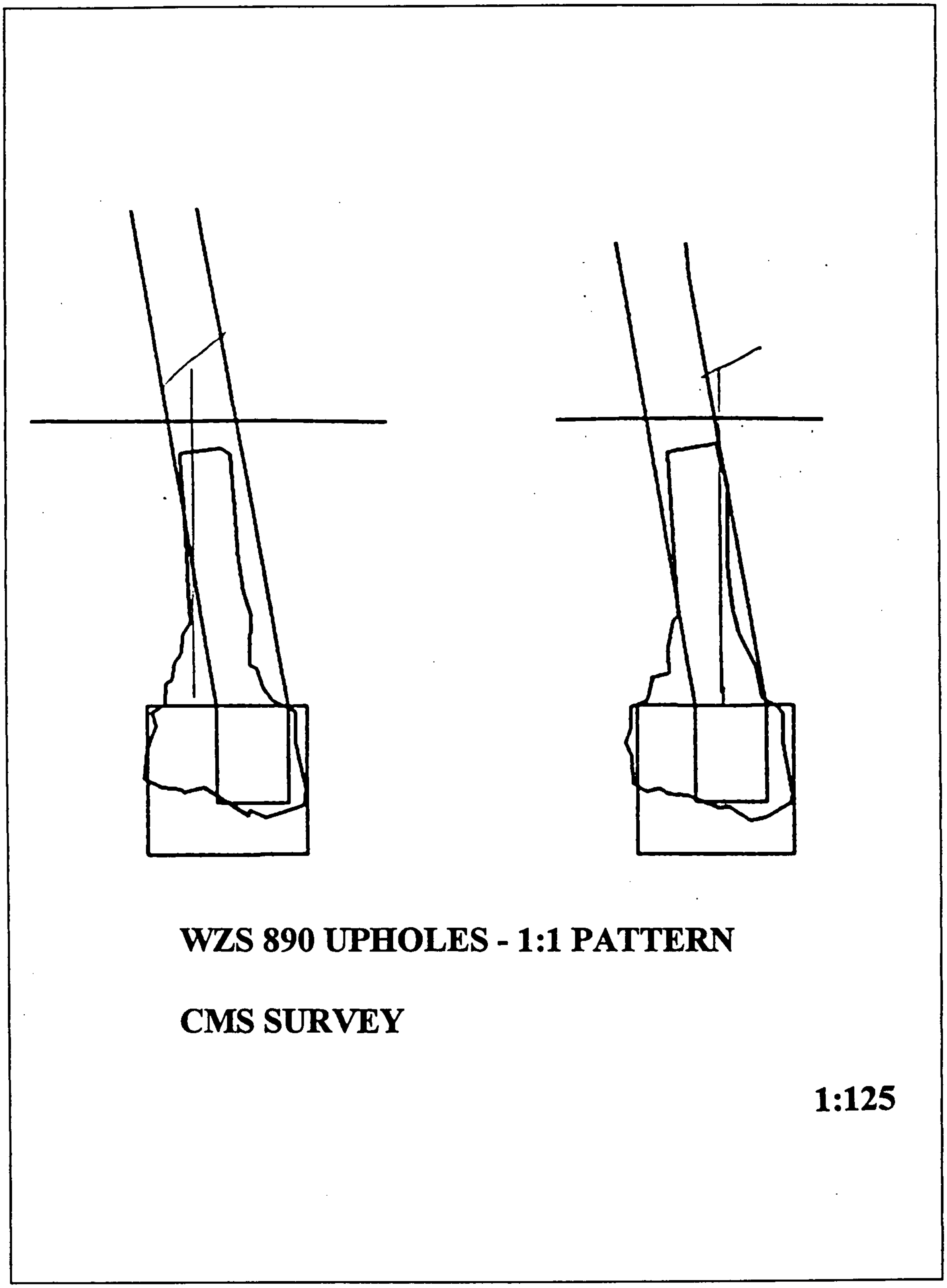


Figure 13.3 CMS stope survey - WZS 890 upholes; 1:1 pattern

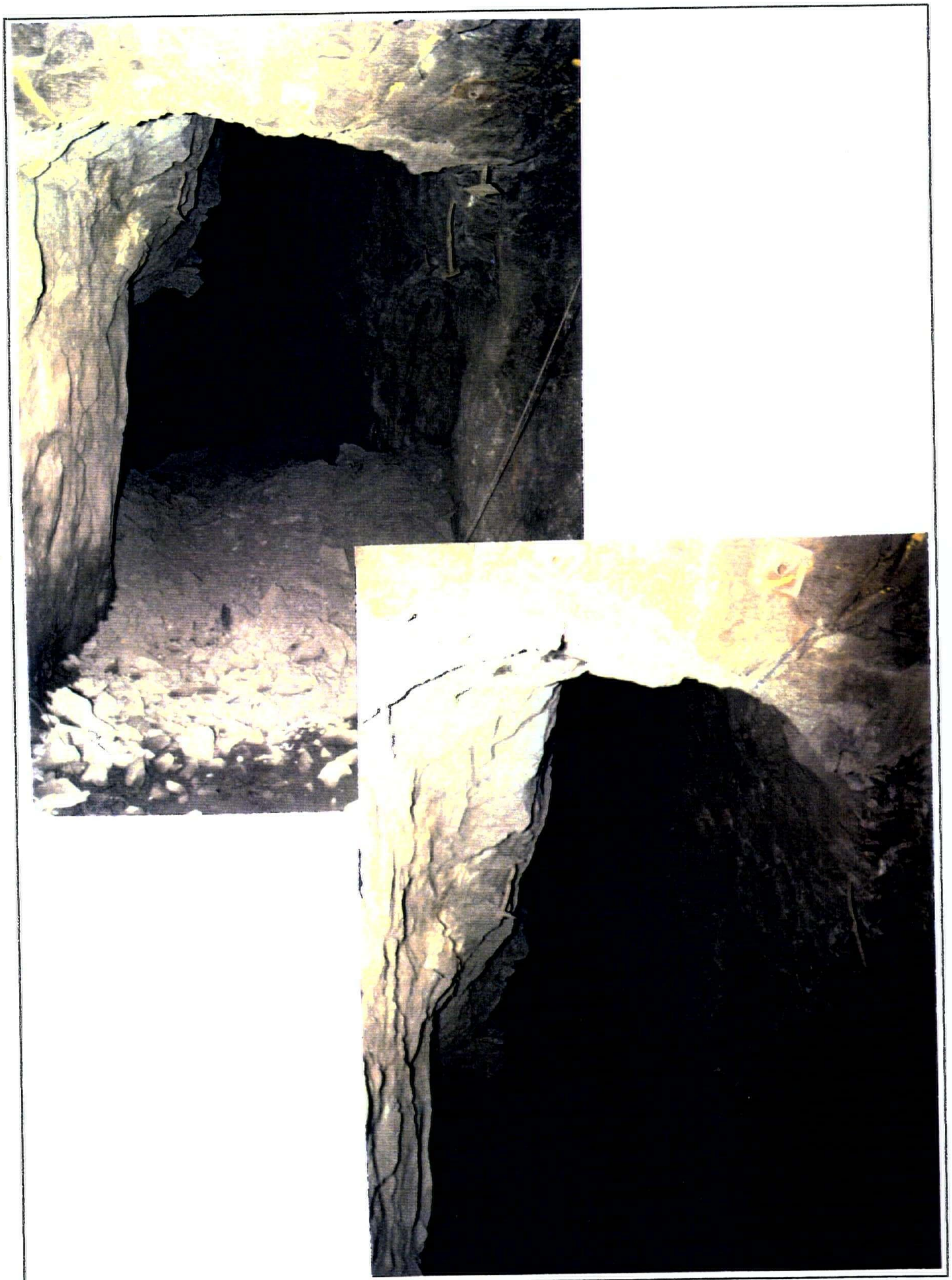


Figure 13.4 Photographs taken after blast - WZS 890 upholes; 1:1 pattern

The results are discussed below:

- no misfires, and an efficient distribution of explosives, as evidenced by consistent vibration levels for each hole (i.e. each blasthole performing similar amount of work);
- significant delay crowding at delay numbers: 7, 8, 16, and 17;
- blast broke well to 5.2m height with fine fragmentation of uniform size;
- walls looked very good;
- near the toes the avg. blasted width was approximately 1.1m, however, near the collars of the blastholes the avg. width was approximately 2m;
- appeared to be an equal amount of overbreak on the footwall as compared to the hangingwall;
- excluding the region around the collars of the blastholes, overbreak was on average 0.2m for both the hangingwall and footwall blastholes;
- the avg. PPV (mm/s) at a plan distance of approximately 30m was 39mm/s.

In general, the results from the test blast were considered very encouraging, however, further tests using the 50mm diameter blastholes were given low priority due to the proposed change to 64mm diameter blastholes.

13.2.2 <u>Trials with 64mm Diameter Blastholes</u>

As part of this study, WZS 840/860/870 Stope #5, was set aside specifically for testing the 1:1 and 2:1 blast patterns drilled with 64mm diameter blastholes.

The 1:1 pattern trial area consisted of 35 rings of both up and downholes (1 uphole and 1 downhole per ring) drilled from 860 lvl. The downholes were 8.8m length and drilled to breakthrough on 870 lvl. The upholes were non-breakthrough and were drilled to a depth of 9m (midway between 860 lvl and 840 lvl plus 1m overlap). The downholes dipped at approximately 80° and the upholes at approximately 85° (no dump angle). The ore width was on average 2m. The test pattern (0.9m wide) was centered between the ore contours (i.e. the blastholes were offset approximately 0.55m from the hangingwall and footwall ore contacts). Figure 13.5 is a long-section schematically depicting the trial stoping area.

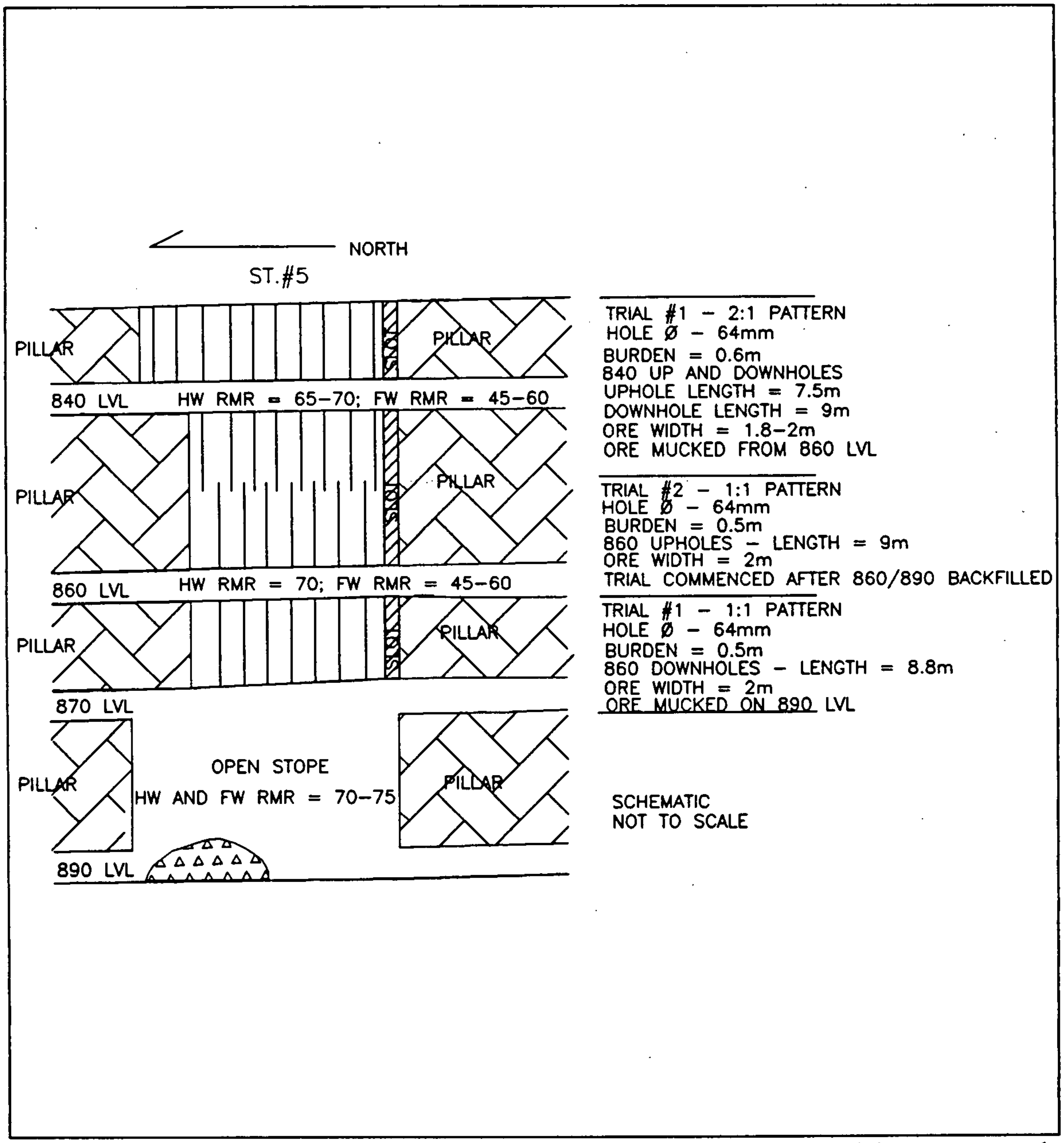


Figure 13.5 Long section schematically depicting WZS 840/860/870 trial area - 1:1 and 2:1 blast patterns; 64mm diameter blastholes

Trial #1 - WZS 860/870 Downholes

The first 1:1 pattern trial involved mining the downholes between 860 lvl and 870 lvl. Ultimately this resulted in a stope with a strike length of approximately 25m and a height of approximately 35m (hydraulic radius = 6.3), since at the time of the trial, the stoping block between 870 lvl and 890 lvl was mined but not yet backfilled. All mucking was done from 890 lvl. Following completion of the trial, the entire stope was backfilled up to the 860 lvl sill elevation to permit mining of the 840/860 stoping block.

Prior to any mining in this area, all of the breakthrough downholes between 860 lvl and 870 lvl were surveyed (collars and breakthrough locations), two redrills were required.

Once the 860/870 slot was open, the remaining downholes were shot in 3 blasts. Blasts #1 and #2 were 10 holes each, blast #3 was 16 holes. Anfo was the primary explosive used in each blast. Each hole was double primed (3m from the toe and 3m from the collar) with 90 gram cast pentolite boosters. Uncharged collar lengths varied between 0m - 0.5m. Blastholes were sequenced individually using MS-Delays (25ms between each hole).

A plot of the vibration monitoring results from the second blast (measured at a distance of 24m) is shown in Figure 13.6. Results from a CMS survey of the stope are presented in Figure 13.7, note the survey results only show the stope outline between 860 lvl and 870 lvl. Photographs taken before the second blast and after the third (last) blast are presented in Figure 13.8.

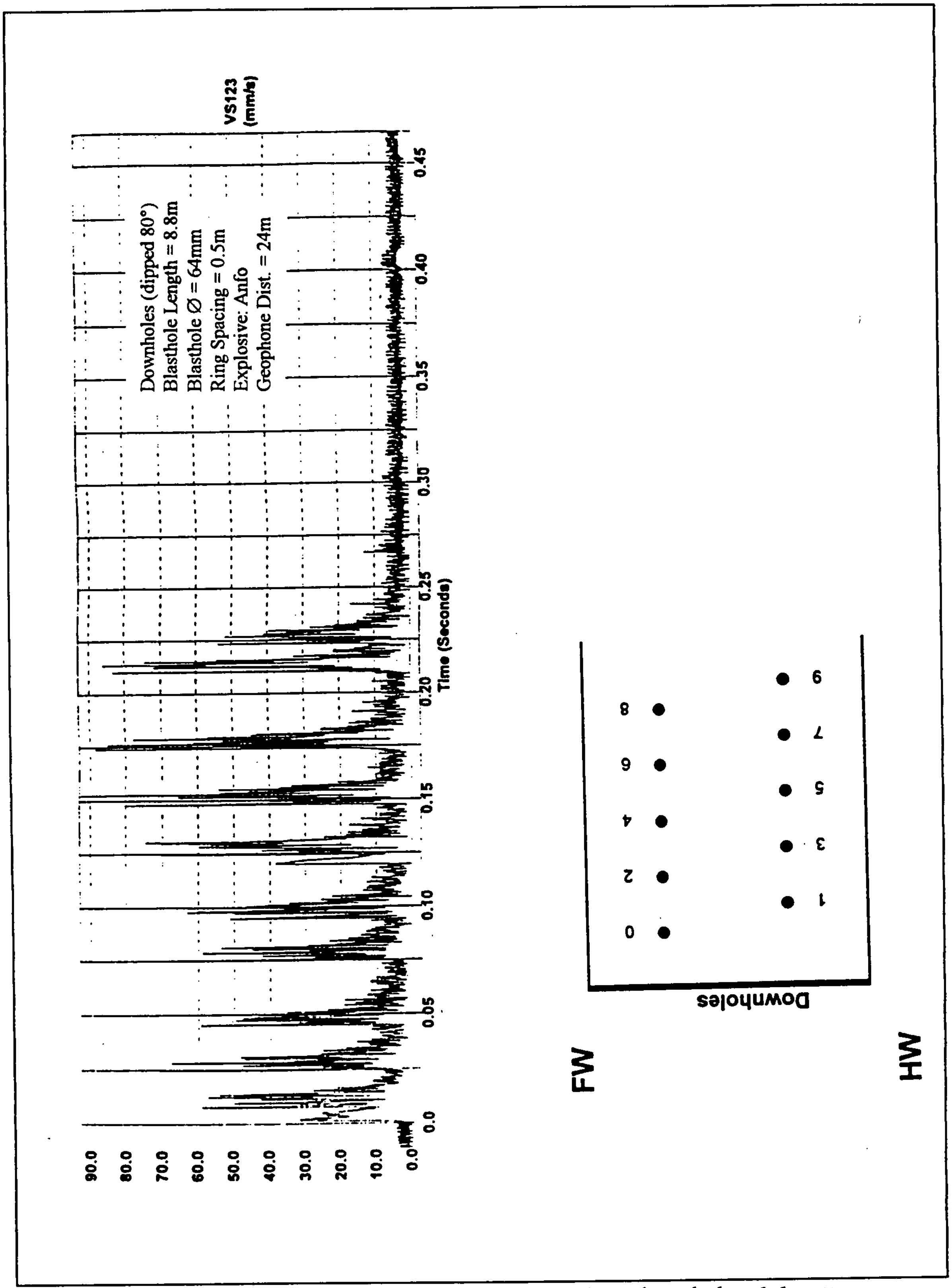


Figure 13.6 Vibration monitoring results - WZS 860/870 downholes; 1:1 pattern

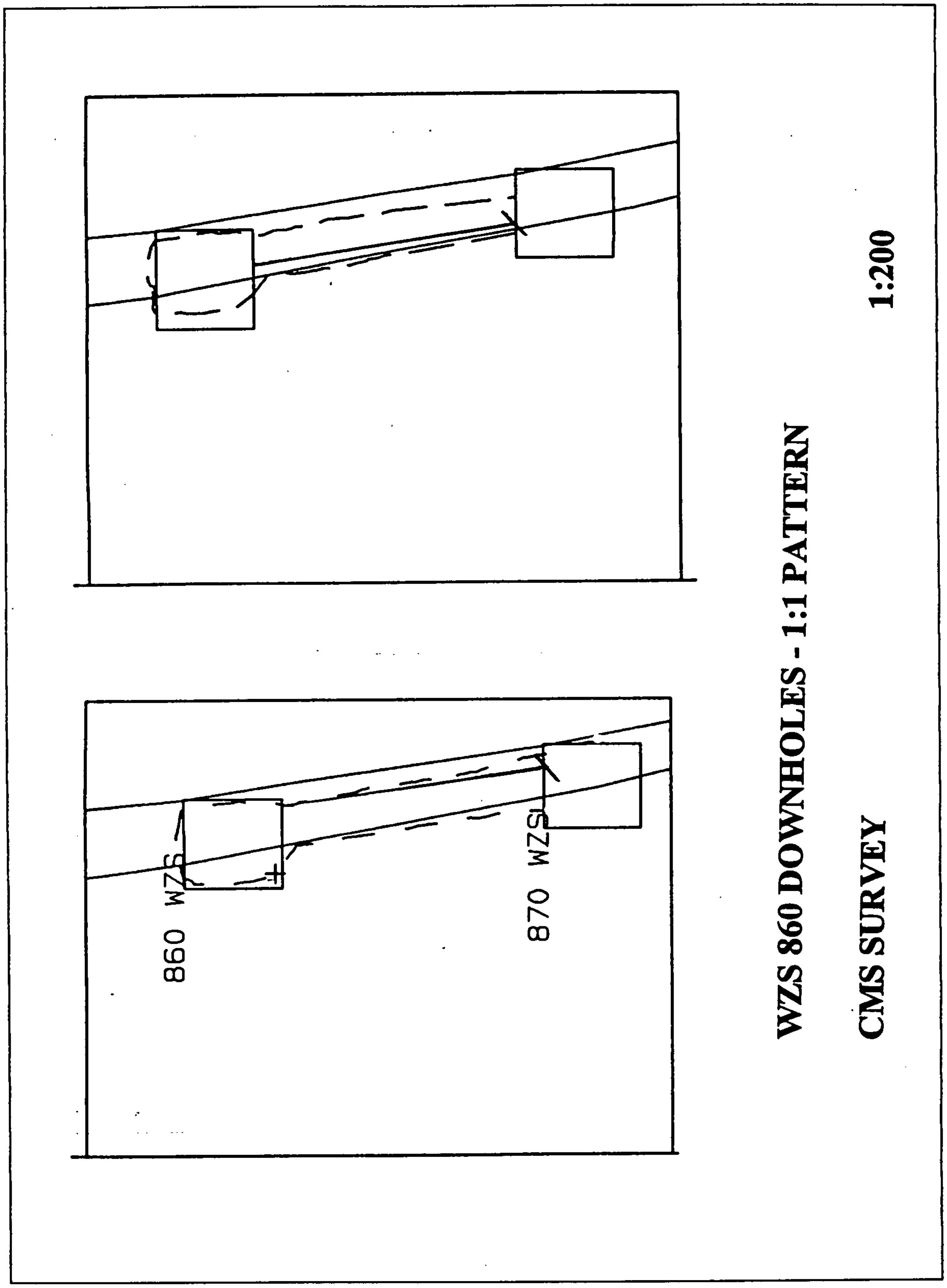


Figure 13.7 CMS stope survey - WZS 860/870 downholes; 1:1 pattern

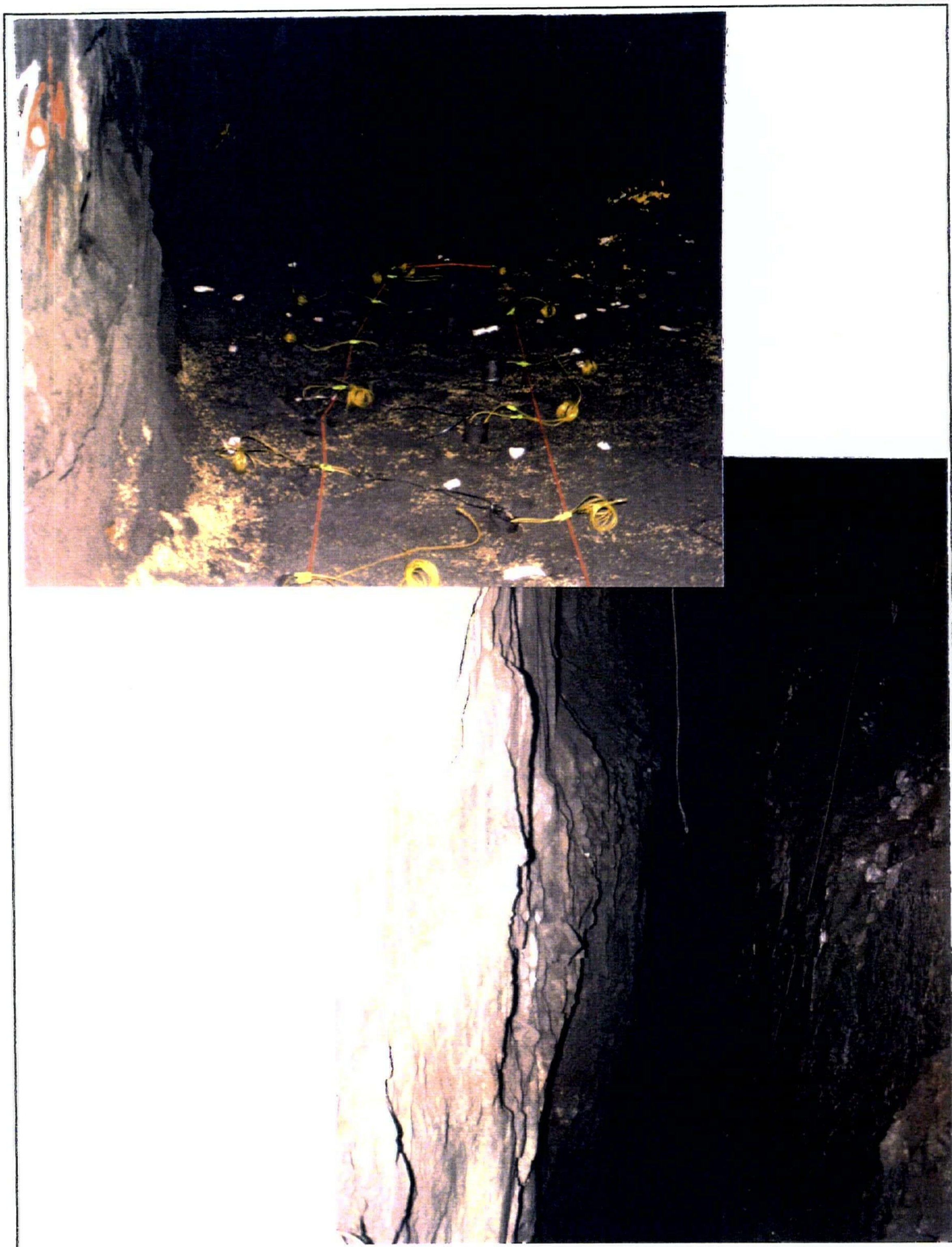


Figure 13.8 Photograph 1 - taken before second blast; Photograph 2 - taken after third blast - WZS 860/870 downholes; 1:1 pattern

The results are discussed below:

- blast monitoring results were only obtained for blasts #2 and #3;
- no misfires occurred in blast #2, 1 misfire occurred during blast #3 (a poor B-line connection is suspected as the reason rather than charge interaction);
- all three blasts broke well, even with a misfire in blast #3;
- the blasts indicated an efficient distribution of explosives, as evidenced by consistent vibration levels for each hole (i.e. each blasthole performing similar amount of work);
- blast #3 showed delay crowding at delay numbers 13 and 14;
- fragmentation was fine (<0.3m) and of uniform size;
- walls looked good but were jagged in appearance (likely due to the tight breakout angles);
- avg. blasted width was approximately 1.5m;
- the majority of the overbreak occurred on the footwall (approximately 0.5m);
- very little overbreak occurred on the hangingwall resulting in a skin of approximately 0.5m of unrecoverable ore;
- the avg. PPV (mm/s) at a plan distance of approximately 24m was 71mm/s.

The results were considered very encouraging for the following reasons:

- the vibration levels were comparable to those achieved using the dual delay detonators;
- the West Zone Fault was exposed along the drift wall (footwall) on 860 lvl, however, due to the combination of: low vibrations; good drilling; and not overbreaking beyond the amphibolite, wall stability was excellent;
- even with a misfire the third blast did not bench, indicating that there is some forgiveness in the pattern design, largely due to the very light burdens assigned to the 64mm blastholes.

On the negative side, approximately 0.5m of overbreak occurred on the footwall, indicating that blasting 1m wide may be difficult to achieve using 64mm diameter blastholes. Nonetheless, this pattern may be viable for ore that is 1.5-1.6m wide. A considerable amount of ore in the WZ has widths in this range. This would represent a significant cost savings over the current 3:2 pattern from both a drilling and explosives consumption perspective as well as a dilution perspective. The CMS survey highlighted that more testing is required to determine appropriate offset distances from the hangingwall and footwall contacts. Results from this trial suggest that very little offset is required from the hangingwall contact.

A potential method of reducing blast damage may be to pour load the Anfo in the downholes as opposed to pneumatically loading it. Pour loading is not recommended with 50mm diameter blastholes because it is difficult to achieve a sufficient packing density for the Anfo, which is required to ensure reliable initiation. However, with the 64mm diameter blastholes sufficient packing density can be achieved by pour loading. The linear charge density of pour loaded Anfo compared to pneumatically loaded Anfo in a 64mm diameter blasthole is 2.7 kg/m and 3.2 kg/m respectively. Pour loading achieves a 16% reduction in linear charge density which offers significant potential with regard to reducing blast induced overbreak.

A potential concern identified by the vibration monitoring is the delay crowding that begins to occur at the higher delay numbers. To minimize the chance of out of sequence firing and/or energy enhancement, it may be necessary to start skipping delay numbers (i.e. #14, #16, #18) when timing blasts of approximately 15 holes or more.

<u>Trial #2 - WZS 860 Upholes</u>

The second 1:1 pattern trial was carried out on 860 lvl as part of the extraction of the WZS 840/860 stoping block. Trials commenced after backfilling of the 860/890 stope

(Trial #1) was complete, refer to Figure 13.5. A typical mining sequence for a block such as this involves mining the upholes over a short distance and then squaring up the brow by blasting the up and downholes from the level above. The process then repeats itself. For this particular block the upholes from 860 lvl were drilled on a 1:1 pattern and the up and downholes from 840 lvl were drilled on a 2:1 pattern. All mucking was done from 860 lvl. Only the 1:1 pattern results will be discussed in this section. The 2:1 pattern results will be discussed in Section 13.4.

Once the slot was opened up the first blast consisted of shooting 11 rings (11 upholes). Drillhole accuracy was not checked prior to the longhole blasters being sent in to load the blastholes. A check on collaring accuracy was made after the holes were loaded and revealed that the actual ring spacings varied between 0.4m - 0.9m, as opposed to the designed 0.5m spacing. In addition to the poor drilling, loading practices were questionable. The blastholes were pneumatically loaded with Anfo, however, the Anfo was not packing well as evidenced by the significant blowback during loading. The poor packing was the result of poor quality Anfo (prills broken down) and poor condition of the loading hose. Furthermore, the uncharged collar lengths were typically 1m as opposed to the designed 0.5m. All the holes were double primed (3m from collar and 1m from the toe) with 90 gram cast pentolite boosters. The blastholes were sequenced individually using MS-Delays (25ms between each blasthole).

Despite the questionable drilling and loading practices the blast was fired. The results were as follows:

- the collars of the blastholes benched over the first 4 rings leaving a web of rock approximately 1-2m thick;
- the toes of the blastholes started benching on the 3rd ring (which had a burden of 0.9m);
- the last 7 rings had little to no free-face, thus only approximately 1m around the collars of the blastholes broke.

Figure 13.9 is a schematic showing the geometry of the bench.

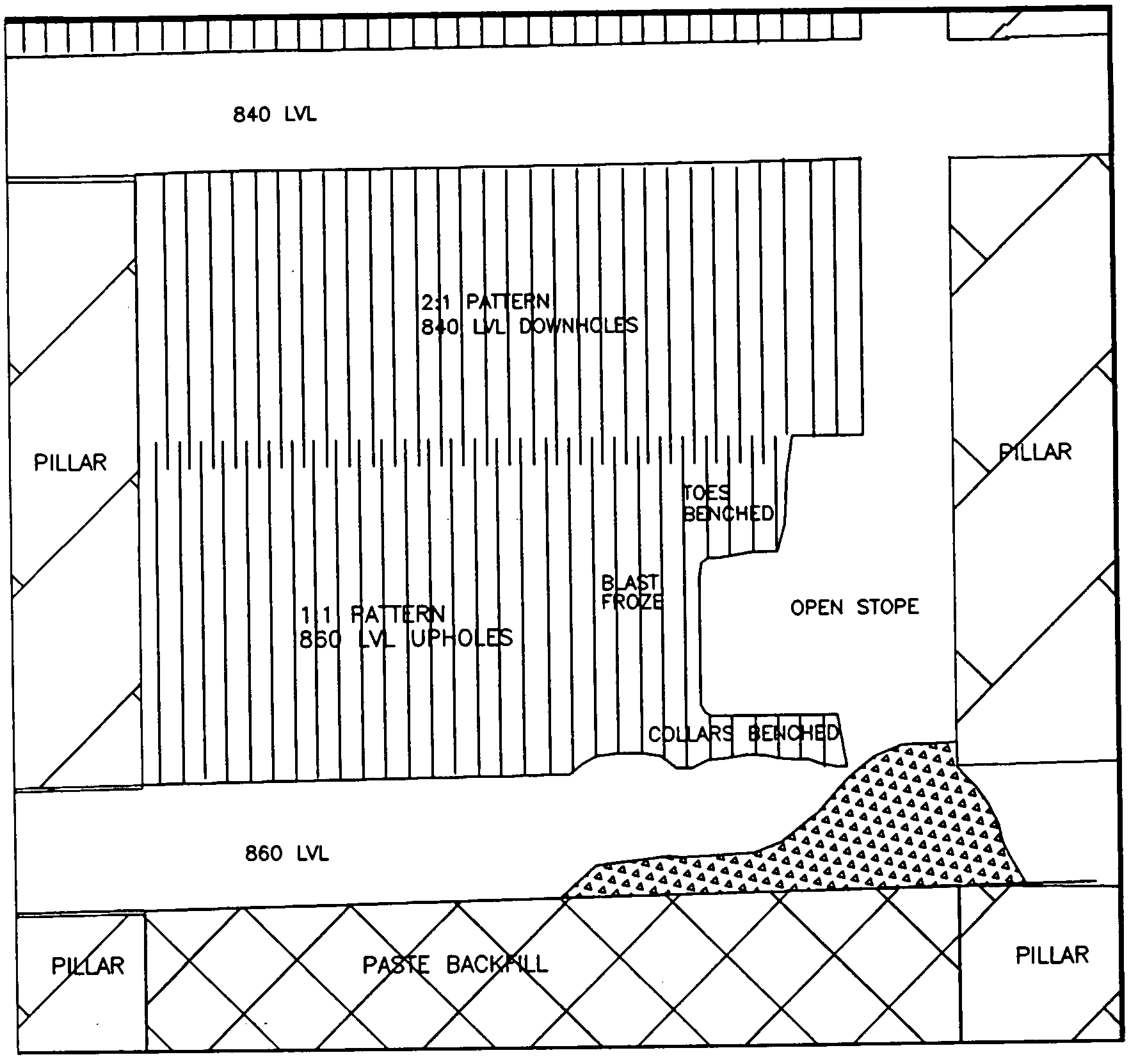


Figure 13.9 Geometry of benched blast - WZS 860 upholes; 1:1 pattern

The web of ground left at the collars of the blastholes and the remaining 7 rings were eventually re-blasted, however, all the downholes from 840 lvl had to be deepened (using remote drilling - since the deepening required drilling into blasted ground) to compensate for the benched toes.

At this point, the stope walls were still in amphibolite and stable, despite the re-blasting and the footwall being in close proximity to the West Zone Fault (i.e. fault exposed in drift wall on 860 lvl and 840 lvl).

Following the blasting of the up and downholes from 840 lvl to square up the brow, stope stability problems were experienced (to be discussed further in Section 13.4). The severity of the stability problems were such that a rib pillar had to be left and a new slot excavated to start the stope over. Once the new slot was developed, blasting of the upholes on 860 lvl resumed. By this time the collars of the remaining blastholes had been surveyed for collaring accuracy, although deviation at the toes was still uncertain. Collaring accuracy was reasonable with burdens (ring spacings) varying between 0.4m - 0.6m. The slot was opened with two small blasts consisting of 4 rings and 8 rings respectively. Both blasts broke narrow and to full height. On the last blast which consisted of 8 rings, the longhole blaster loading the holes missed the first hole (i.e. did not load) which resulted in a burden of 1.2m on the first hole. The blast was fired and not surprisingly approximately 3m of the toes benched starting at the first blasthole. The bench was eventually fixed by deepening the downholes from 840 lvl, using remote drilling.

Prior to blasting the up and downholes from 840 lvl, the stope walls looked excellent, despite the footwall being in close proximity to the West Zone Fault.

Although this trial experienced significant problems, some valuable lessons were learned. These are summarized below:

- the 1:1 pattern has limited tolerance for lapses in drilling and loading quality;
- the inability to survey non-breakthrough blastholes is a definite drawback if a high standard of drilling is not practiced by all longhole drillers;
- leaving the appropriate uncharged collar length is extremely important. As a rule of thumb, uncharged collar lengths should not be greater than the burden;
- burdens (ring spacings) as high as 0.7m broke successfully;
- the toes of blastholes benched where burdens (ring spacings) exceeded 0.9m;
- in all cases wall stability was excellent even when the footwall was in close proximity to the West Zone Fault.

13.3 2:1 PATTERN DESIGN

As discussed earlier (Chapter 9), the 2:1 pattern (dice-five) has been used at the Lupin Mine in the past, refer to Figures 9.7 and 9.8 (Chapter 9). At Lupin, the 64mm diameter blastholes met with more success than did the 50mm diameter blastholes. Although it is likely the 2:1 pattern with 50mm diameter blastholes could have been successful if a more concerted effort on quality control of drilling and loading practices had been implemented. For example, the Campbell Mine (Placer Dome Canada) has been successfully using the 2:1 pattern for a number of years. The blasthole diameter used is 54mm with a 1.2m burden between the 2-hole rings, the 1-hole ring is centered between the 2-hole rings (i.e. 0.6m burden). This pattern is used up to ore widths of 3m, after which a switch is made to a 3:2 pattern, O'Flaherty et.al. (1993). It is worthy to note that the 2:1 pattern used at Campbell Mine does not have as heavy a burden as what was trialed in the past at the Lupin Mine. Although the Lupin pattern worked when quality control was being monitored closely, it likely did not have enough forgiveness built into the pattern design. In addition, the Lupin Mine ore is considerably stronger than the Campbell Mine ore (i.e. Lupin Mine ore - UCS = 448 MPa; Campbell Mine ore - UCS = 175-200 MPa).

Current trials at the Lupin Mine are concentrating on the use of 64mm diameter blastholes. The initial trials have been designed using a 1.2m burden between the 2-hole rings. The 1-hole ring is centered between the 2-hole rings (0.6m burden). This is the same pattern being used at the Campbell Mine except with a larger diameter blasthole. The logic behind the design is the same as that used for the 1:1 pattern trials, low confinement and design some forgiveness into the pattern. Burdens as large as 1m have worked in the past at the Lupin Mine, so there is considerable opportunity to expand the pattern if warranted. Blastholes are to be sequenced individually to minimize the amount of explosive detonating per delay number. When sequencing the 2-hole rings, the blasthole adjacent to the more competent wall should be fired first. The reason being is that the first hole to fire on the 2-hole ring has a tight break-out angle and thus has higher potential for blast induced overbreak. In the West Zone, the hangingwall is generally more competent than the footwall, thus, the hangingwall blasthole should be sequenced to fire before the footwall blasthole. This concept is shown in Figure 13.10. This is different than past 2:1 pattern trials at Lupin where the blastholes on the 2-hole ring were given the same delay number.

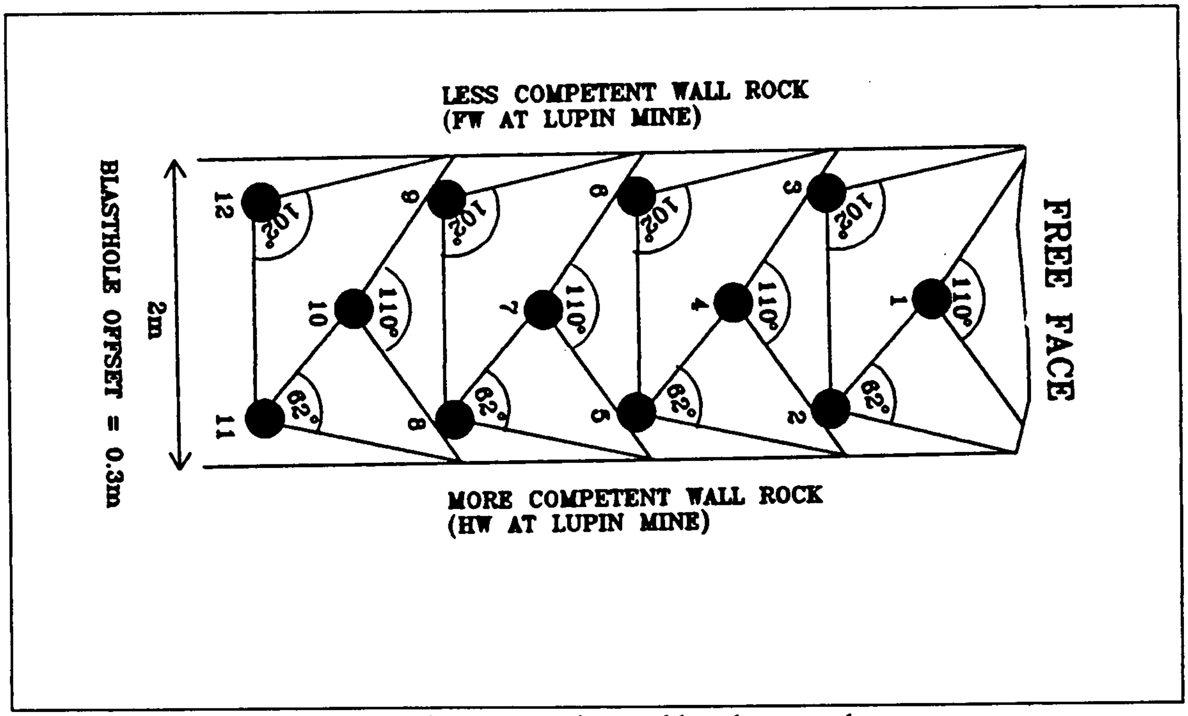


Figure 13.10 2:1 pattern showing sequencing and breakout angles

13.4 2:1 PATTERN FIELD TRIALS

To date, field trials of the 2:1 pattern have been conducted in three mining blocks: WZS 840/860 Stope #5; WZN 950/980 Stope #1; and WZN 840/860 Stope #1. Descriptions of the trials are presented in the following paragraphs. Unfortunately due to time constraints, no CMS surveys of stopes excavated with the 2:1 pattern have yet been performed.

Trial #1 - WZS 840 Stope #5 - Up and Downholes

The WZS 840/860 stoping block has already been discussed in some detail in reference to the 1:1 pattern field trials (Trial #2 - WZS 860 Upholes), refer to Figure 13.5. The 2:1 pattern trial area consisted of the up and downholes (non-breakthrough) drilled from 840 lvl.

Blasthole lengths were on average 7.5m for the upholes and varied between approximately 9m - 15m for the downholes. Note that the downholes in the south and north ends of the stope required deepening to compensate for the 860 upholes benching at the toes (refer to Section 13.2). The original design length for the downholes was 9m. Both the up and downholes dipped at approximately 85° with no dump angle on the upholes. Ore width varied between 1.8m - 2.0m. The blastholes on the 2-hole rings were designed with little to no offset from the hangingwall and footwall contact.

The collar locations of all the blastholes were surveyed in, and proved to be very accurate. Deviation at the toes of the holes was uncertain.

For the blasts monitored, both the up and downholes were pneumatically loaded with Anfo. All holes were double primed with 90 gram cast pentolite boosters. Uncharged collar lengths typically varied between 0.6m - 1.0m. Holes were sequenced using MS-Delays.

Once the slot was opened up, two small blasts were taken. Only the second blast was monitored which consisted of 3 rings of up and downholes. Although the design called for each blasthole to be sequenced individually, the blasters hooked it up such that the blastholes on the 2-hole rings were given the same delay number. Vibration monitoring results from the blast are presented in Figure 13.11.

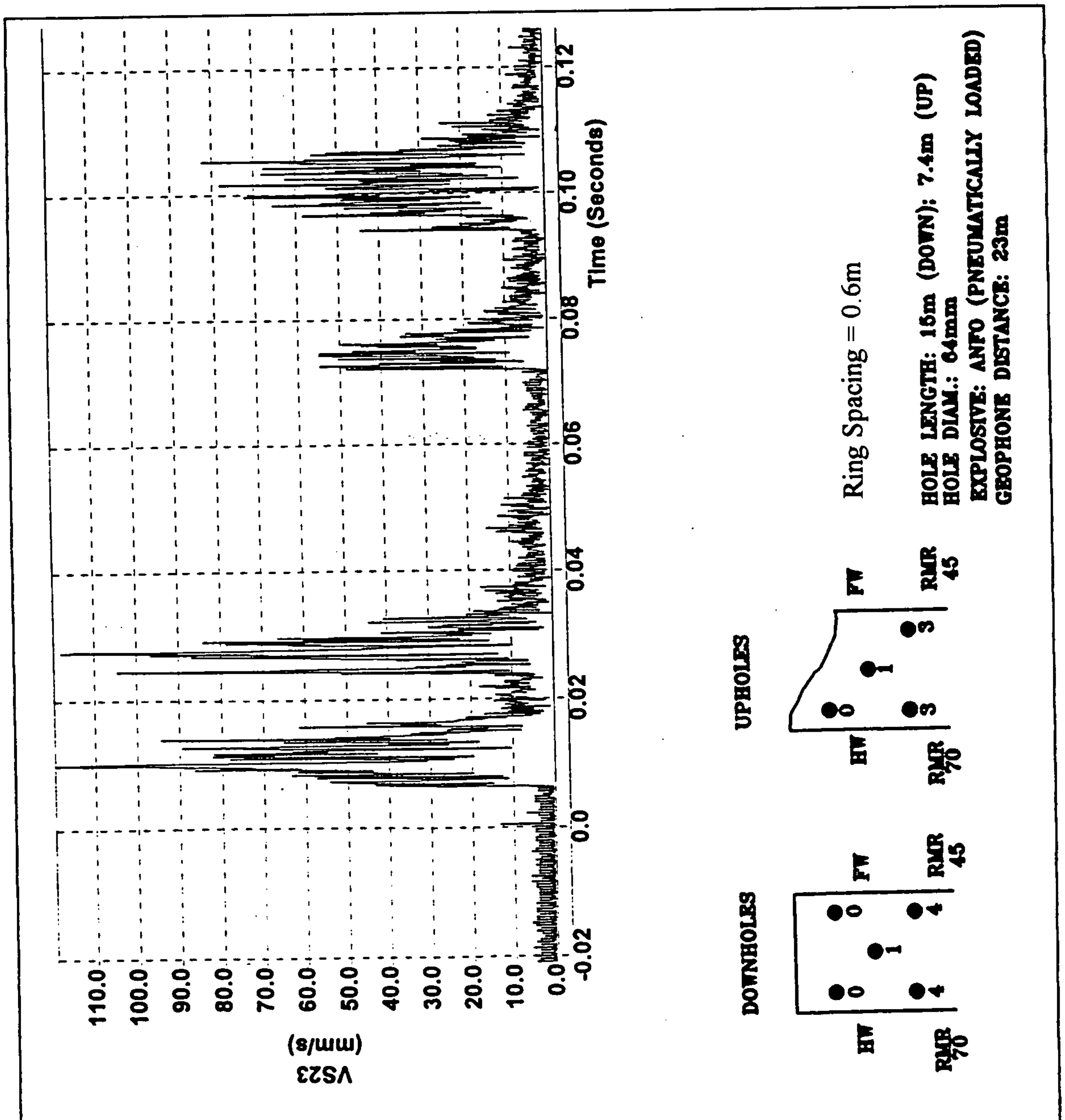


Figure 13.11 Vibration monitoring results - WZS 840 up and downholes; 2:1 pattern

Results from the blast are presented below:

- no misfires were recorded;

- the blast broke well, however, approximately 1.5m of overbreak occurred on the footwall, exposing the West Zone Fault. Note that the fault was exposed in the drift wall on 840 lvl prior to the blast;

- the average PPV (mm/s) at a plan distance of 23m was 93 mm/s, the maximum PPV (mm/s) was 120 mm/s.

After the second blast the exposed stope strike length was only 7m, however, excessive sloughing from the footwall necessitated that the stope be shut down. As was discussed earlier in Section 13.2, a rib pillar was left to regain control of the footwall. Leaving the rib pillar required that new slot raises be developed.

Due to scheduling constraints, the remaining 2:1 pattern blasts were not monitored in this stoping block. Operational personal stated that the remaining blasts had been sequenced as designed (i.e. each blasthole assigned its own delay #) and that no benches were experienced. An inspection of the stope showed that the footwall had again sloughed back into West Zone Fault. The depth of sloughing was approximately 2m. It is not known whether the blasting immediately overbroke into the West Zone Fault as with the previous blast, or, if the sloughing slowly developed with time.

The results from this trial were encouraging from the point of view that no benching problems were experienced. However, from a blast damage perspective the results achieved with the 2:1 pattern were no better than those typically achieved with the 3:2 pattern in similar circumstances (i.e. footwall in close proximity to the West Zone Fault).

This trial did show that blast induced overbreak can be very severe if the 2-hole rings are sequenced using the same delay number. If blast induced overbreak is to be minimized 2:1 pattern blasts should not be sequenced in this fashion.

As mentioned previously, the blastholes on the 2-hole rings were designed with little to no offset from the hangingwall and footwall contacts, thus, there is some potential to minimize the blast damage associated with the 2:1 pattern by determining appropriate offset distances. Likely a relatively large offset from the footwall contact (i.e. 0.5m) will be required when in close proximity to the West Zone Fault. Hangingwall overbreak was not observed to be severe, indicating that perhaps only a small offset from the hangingwall contact is required. CMS surveys will be a valuable tool for determining appropriate offset distances. It should be mentioned that the 2:1 pattern offers considerably more flexibility with regard to choosing offset distances than the 3:2 pattern due to the fewer number of blastholes.

As mentioned earlier in Section 13.2.2, a potential method of reducing blast damage may be to pour load the Anfo in the downholes as opposed to pneumatically loading it.

<u>Trial #2 - WZN 950/980 Stope #1 - Downholes</u>

The second trial for the 2:1 pattern was carried out in WZN 950/980 Stope #1. The 950/980 stope consisted of upholes (non-breakthrough) drilled on a 3:2 pattern from 980 lvl and downholes (non-breakthrough) drilled on a 2:1 pattern from 950 lvl. The stope had a planned strike length of 12m and a stope height of 30m (hydraulic radius = 4.3). No evidence of being in close proximity to the West Zone fault was apparent on either 950 lvl or 980 lvl. Figure 13.12 is a long-section schematically depicting the trial area.

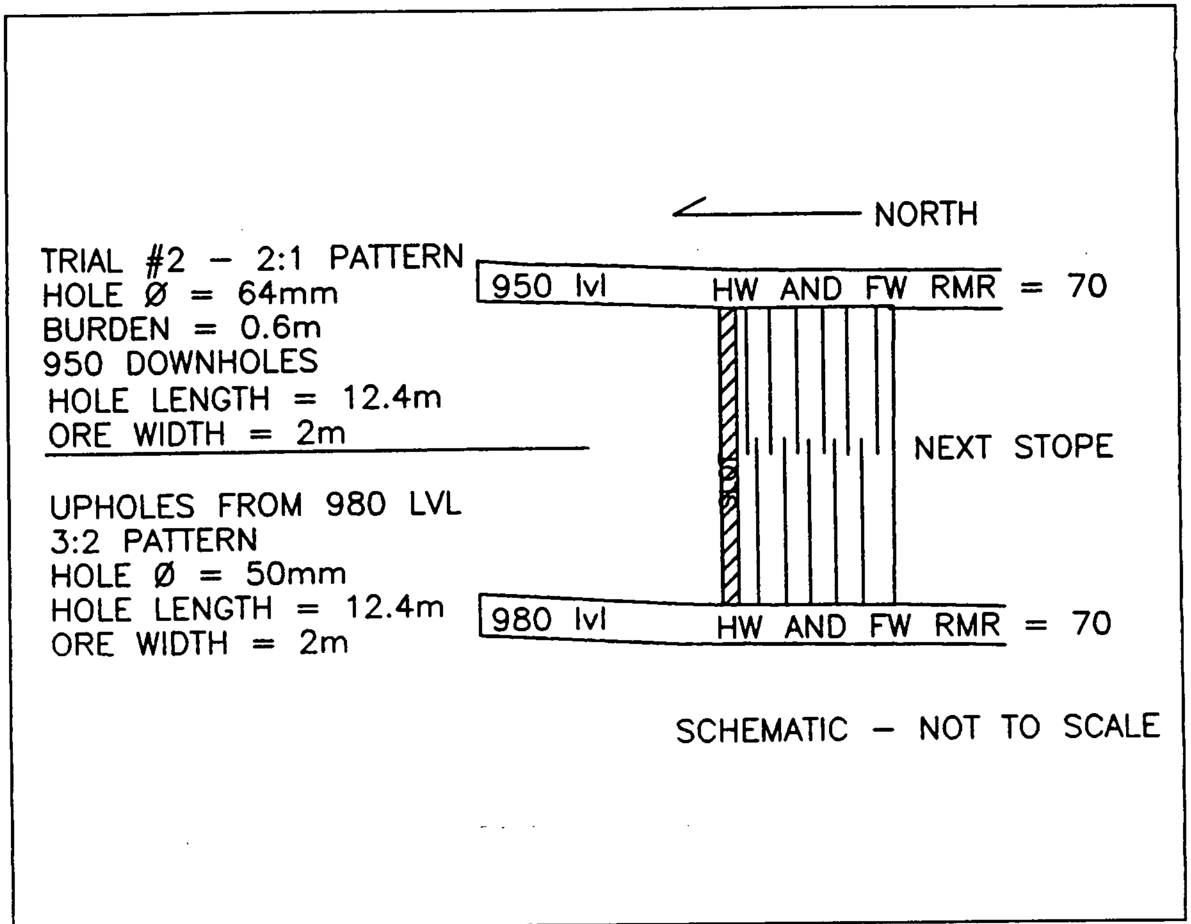


Figure 13.12 Long section schematically depicting WZN 950/980 trial area - 2:1 pattern; 64mm diameter blastholes; 0.6m burden

The stope was mined by first blasting the upholes from 980 lvl followed by blasting of the downholes from 950 lvl. Mucking was done from 980 lvl.

Blasthole lengths for the 2:1 pattern were on average 12.4m and were dipped at 79°. The average ore width in this area was approximately 2m. Blastholes on the 2-hole rings were offset from the hangingwall and footwall contacts 0.3m.

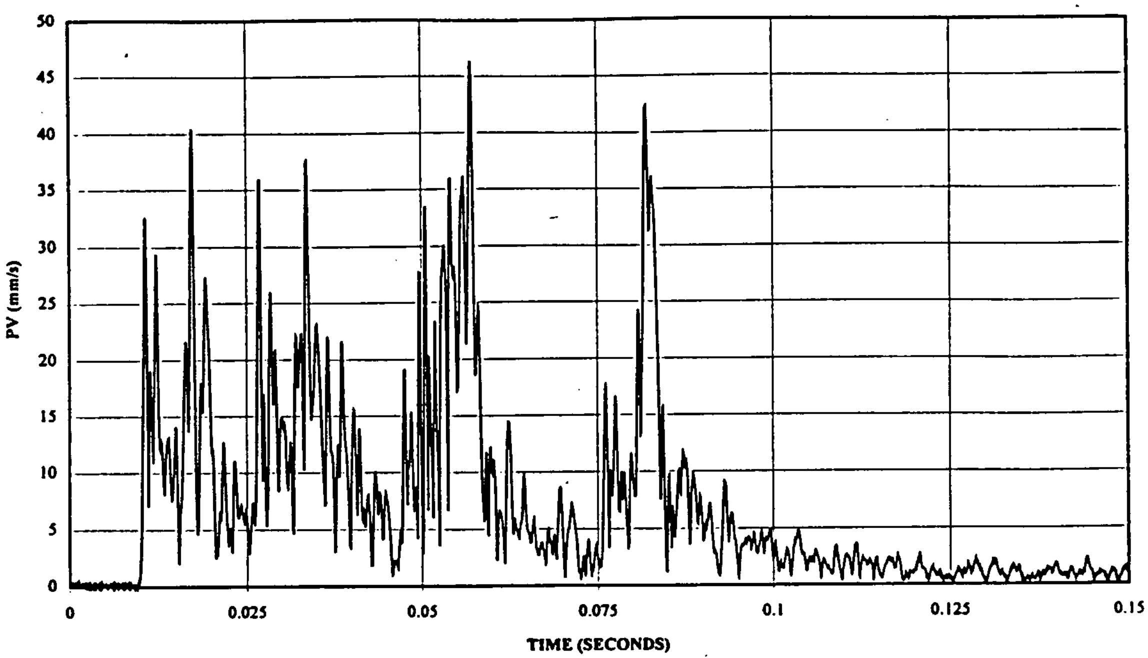
Drillhole collaring accuracy was not checked prior to mining in this area.

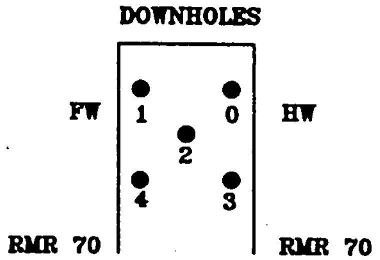
Once the upholes from the 980 lvl has been blasted and the slot opened up on 950 lvl the remaining downholes on 950 lvl were taken in 3 blasts. Due to scheduling constraints only the first two blasts were monitored.

The first blast consisted of 4 rings and the second blast consisted of 3 rings. A check on collaring accuracy was made after the holes had been loaded and showed that burdens (ring spacings) varied between 0.6m - 1.0m. For both blasts the blastholes were observed to be wet. With regard to the first blast, water resistant stick powder (Geldyne - 50mm x 400mm) was used in only 2 of the 6 blastholes. Pneumatically loaded Anfo was used in the remaining holes. For the second blast, stick powder was used in all the blastholes. Uncharged collar lengths for both blasts varied between 0m - 0.5m. All holes were double primed.

An example of the vibration monitoring results from the second blast (measured at a distance of 37m) are presented in Figure 13.13. Results from the two blasts are discussed below:

- both blasts broke well;
- 1 misfire occurred in each of the blasts. With regard to the first blast, the misfire occurred on a 1-hole ring loaded with Anfo. It is uncertain whether the misfire occurred due to charge interaction or as a result of having loaded Anfo in a wet blasthole. With regard to the second blast, the misfire occurred on a 2-hole ring and was likely due to charge interaction;
- only minor overbreak was observable on the hangingwall and footwall, neither blast overbroke beyond the amphibolite;
- fragmentation was very fine (<0.2m) and of uniform size;
- the avg. PPV (mm/s) for blast #1 was 36 mm/s, the max. PPV (mm/s) was 42 mm/s;
- the avg. PPV (mm/s) for blast #2 was 41 mm/s, the max. PPV (mm/s) was 46 mm/s;
- the stope walls looked very good.





Ring Spacing = 0.6m
Hole #4 misfired

HOLE LENGTH: 12.4m
HOLE DIAM.: 64mm
EXPLOSIVE: GELDYNE

GEOPHONE DIST: 37m
1.0m BURDEN ON CAP#1
DRILLING QUALITY: MODERATE TO POOR

Figure 13.13 Vibration monitoring results - WZN 950/980 downholes; 2:1 pattern

Despite the fact that the quality of both the drilling and loading practices were marginal both blasts broke well. This indicates that there is considerable forgiveness built into this pattern. The misfires, however, suggest potential charge interaction which may indicate that the burden (ring spacing) should possibly be increased to achieve a better distribution of explosive energy. Furthermore, the very fine fragmentation indicates overblasting. Continued blast monitoring will verify whether charge interaction is an issue with the pattern as it is currently designed.

The fact that the blasts did not overbreak beyond the amphibolite, and that the stability of the stope walls was very good, suggests that the blasthole offset distance of 0.3m from the hangingwall and footwall contact may be appropriate when in good quality rock and not in the vicinity of the West Zone Fault. However, CMS surveys need to be conducted to accurately verify that the offset distances are appropriate.

Trial #3 - WZN 840/860 Stope #1 - Downholes

This trial area represented the first attempt at drilling the 2:1 pattern using long breakthrough downholes (i.e. 840 lvl to 860 lvl). The average blasthole length was approximately 16.5m. The holes were dipped at 80°.

Figure 13.14 is a plot showing the results of a survey pick-up of breakthrough locations on 860 lvl. Referring to the figure, it can be seen that the accuracy of the drilling was poor. In fact, it is almost impossible to distinguish the 2:1 pattern.

This stoping block required a large number of re-drills subsequently resulting in a blast pattern which was significantly different than the initially designed 2:1 pattern. As such the longhole blasts were not monitored.

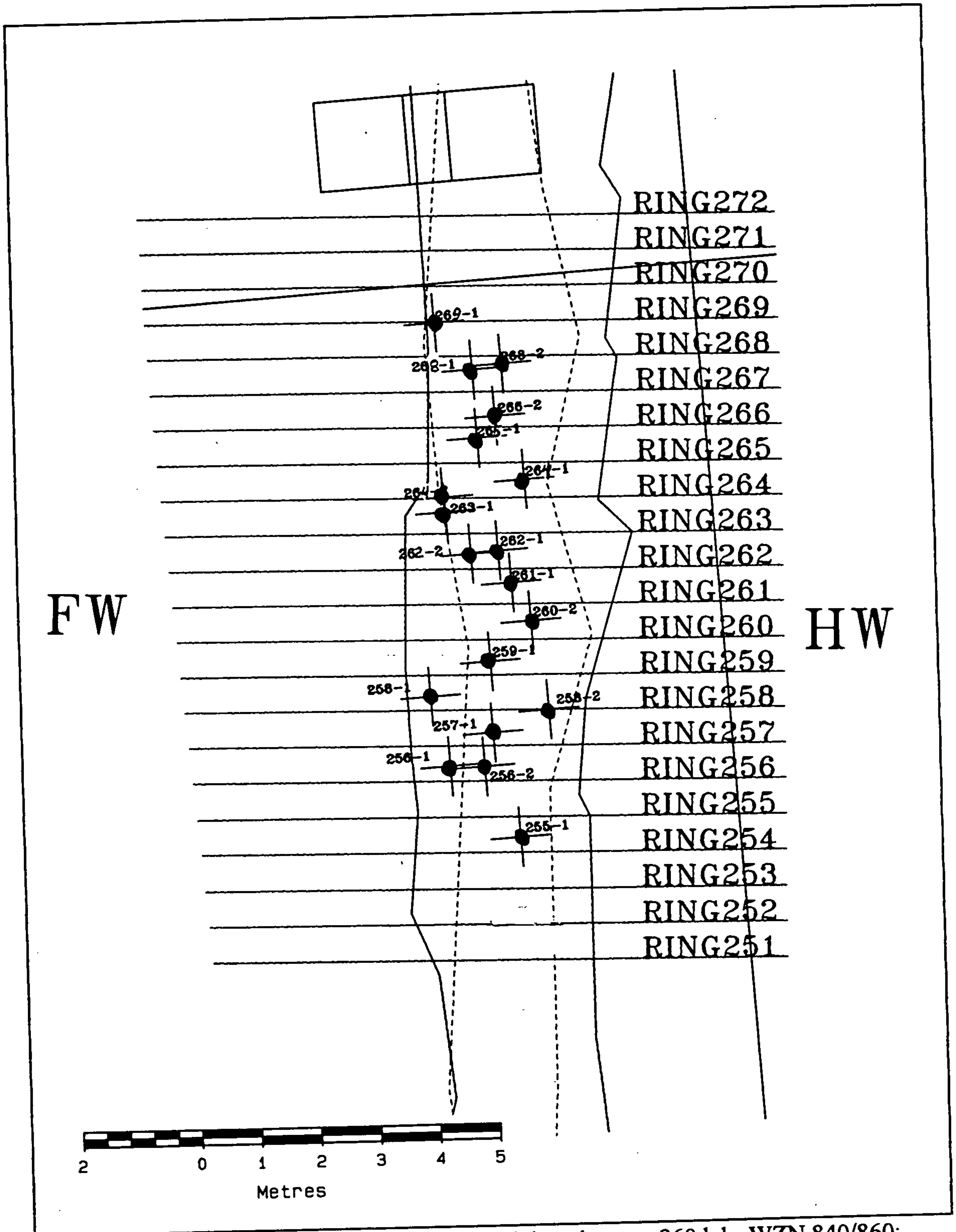


Figure 13.14 Survey of blasthole breakthrough locations on 860 lvl - WZN 840/860; 2:1 pattern; 64mm diameter blastholes

The results of this trial prompted a complete review of the longhole drilling practices at Lupin. A longhole foreman was recently appointed to help improve the quality of longhole drilling and blasting. Improving drillhole accuracy is currently the top priority.

13.5 CHAPTER SUMMARY

This chapter has presented the results of the initial field trials of both the 1:1 (stagger) pattern and the 2:1 (dice-five) pattern at the Lupin Mine. Although some problems were experienced, the field results and potential cost savings are considered promising enough to continue on with further trials. Upcoming trials include expanding the 1:1 pattern to a 0.6m burden (ring spacing) and expanding the 2:1 pattern, possibly to a 0.75m burden (ring spacing). It is envisioned that when fully adopted, the 1:1 pattern will be used for ore widths of between 1.4m - 1.8m, the 2:1 pattern for ore widths between 1.8m - 2.5m; and the 3:2 pattern will be used for ore widths greater than 2.5m. Appropriate blasthole offset distances still need to be determined. More CMS surveys are required to accurately determine appropriate offset distances.

An important development that arose from the trials was the appointment of a longhole foreman. Blast monitoring during the trials showed that there are definite concerns regarding the quality of drilling and loading practices. In order for these patterns to be successfully implemented, a high quality of workmanship is required. The longhole foreman's role is to help improve drilling and blasting practices and to help ensure quality work is being performed.

CHAPTER 14

SUMMARY
ASSESSMENT OF NARROW VEIN LONGHOLE BLAST PATTERNS

14.1 COMPARISON OF BLAST MONITORING RESULTS: 1:1 AND 2:1 PATTERNS VS. THE 3:2 PATTERN

Tables summarizing the blast monitoring results are presented in Appendix III. Additional monitoring information collected by Golder/CANMET (1989) is also included.

14.1.1 Blast Vibrations - Blast Damage Potential

Figure 14.1 is a reproduction of the 3:2 pattern vibration monitoring results shown in Figure 11.10 (Chapter 11) with the 1:1 and 2:1 vibration monitoring results overlain. Note that descriptions of the A, B, C (dual delay detonator), and D sequencing methods for the 3:2 pattern are given in Chapter 11. A discussion of the monitoring results is given in the following paragraphs.

1:1 (Stagger) Pattern

The vibration levels measured for both the 50mm and 64mm diameter blasthole patterns were comparable to those measured for the 3:2 pattern (50mm diameter blastholes) when blasting each hole on its own delay (sequencing method C). This is very interesting, especially with regard to the 64mm diameter blastholes, since the linear charge density and charge weight per delay are approximately 35% higher (assuming pneumatically loaded Anfo). Furthermore, the breakout angles are very tight with the 1:1 pattern. The fact that light burdens were assigned to the blastholes must have contributed significantly to the low vibration levels. This is direct evidence that confinement plays an important role with regard to the vibration levels produced by a particular blast pattern, and that charge weight per delay and linear charge density are not the only factors to consider.

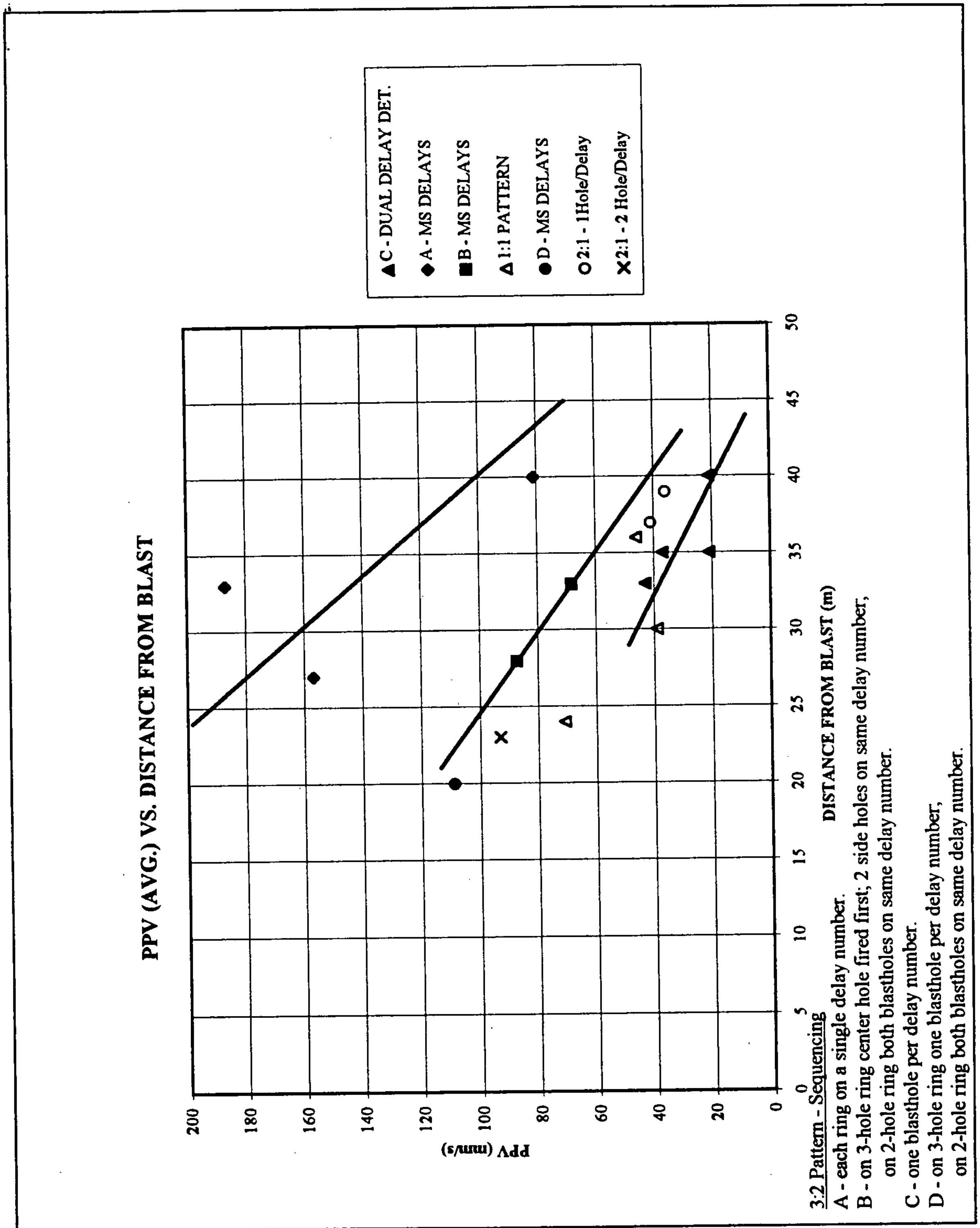


Figure 14.1 PPV (Avg.) vs. Distance From Blast - showing all the data from this study

Referring to Figure 14.1, the vibration levels and hence blast damage potential of the 1:1 pattern (64mm diameter blastholes) is lower than that of the 3:2 pattern (50mm diameter blastholes) as it is currently being sequenced (method D). This is supported by field measurements and observations. With regard to achieving stable stope walls and minimizing dilution, the 1:1 pattern provided the best results of all the patterns monitored. The stopes consistently broke narrow and did not overbreak beyond the competent amphibolite, even when in close proximity to the West Zone Fault. However, there was some immediate overbreak beyond the blastholes with the 1:1 pattern. The 50mm diameter blastholes (vertical upholes with no dump angle - 0.4m burden) overbroke on average 0.2m beyond the blastholes along both the hangingwall and footwall. The average blasted stope width was approximately 1.1m. With the 64mm diameter blastholes (downholes holes dipped at 80° - 0.5m burden) immediate overbreak was approximately 0.5m beyond the blastholes along the footwall and virtually no overbreak beyond the blastholes along the hangingwall. The average blasted width was approximately 1.5m. Continued trials and CMS surveying are required to determine appropriate offset distances from the hangingwall and footwall contacts which will in turn minimize dilution and ore loss.

In addition to optimizing the blasthole offset distances, there is also opportunity to reduce blast damage through reductions in linear charge density (i.e. kg/m of explosive). Examples of this include pour loading Anfo in the downholes as opposed to pneumatically loading it, or, using smaller diameter decoupled explosives.

<u>2:1 (Dice-Five) Pattern</u>

Referring to Figure 14.1, when the 2:1 blasts were sequenced such that there was only one blasthole per delay number, the blast vibrations were similar to those measured for the 1:1 pattern. When the blastholes on the 2-hole rings were sequenced on the same delay number, vibration levels were comparable to the 3:2 pattern when sequencing methods B or D were used. It is interesting to note that the blast vibrations associated with the current 2:1 pattern (64mm diameter blastholes) are considerably lower than those measured by Golder/CANMET (1989) during past trials of the 2:1 pattern utilizing 50mm diameter blastholes. Even when the 2-hole rings are sequenced on the same delay number. The past trials with 50mm diameter blastholes showed vibration levels comparable to the 3:2 pattern when using sequencing method A.

Figure 14.2 is a plot of PPV (mm/s) versus scaled distance ((distance from blast / (charge weight per delay)$^{1/2}$) showing measured blast vibrations associated with the:

- 3:2 pattern (past and current monitoring - 50mm diameter blastholes);
- 2:1 pattern (past trials - 50mm diameter blastholes)
- 1:1 and 2:1 patterns (current trials - 64mm diameter blastholes).

Referring to the figure, the results from the current trials with 64mm diameter blastholes plot distinctively lower and with a flatter slope than the other results, indicating lower vibration levels and hence less blast damage potential. This further shows that that the low confinement with the current pattern has a profound impact on the resulting vibration levels, since both the linear charge density and charge weight per delay are significantly higher than with the 50mm diameter blastholes.

With regard to maintaining stope stability and minimizing dilution, one 2:1 pattern trial was unsuccessful, resulting in excessive dilution (estimated at >100%), and one trial proved successful, with the blasts breaking narrow and not overbreaking beyond the amphibolite. Unfortunately no CMS surveys were conducted so no quantifiable measurements of overbreak/slough or underbreak were made.

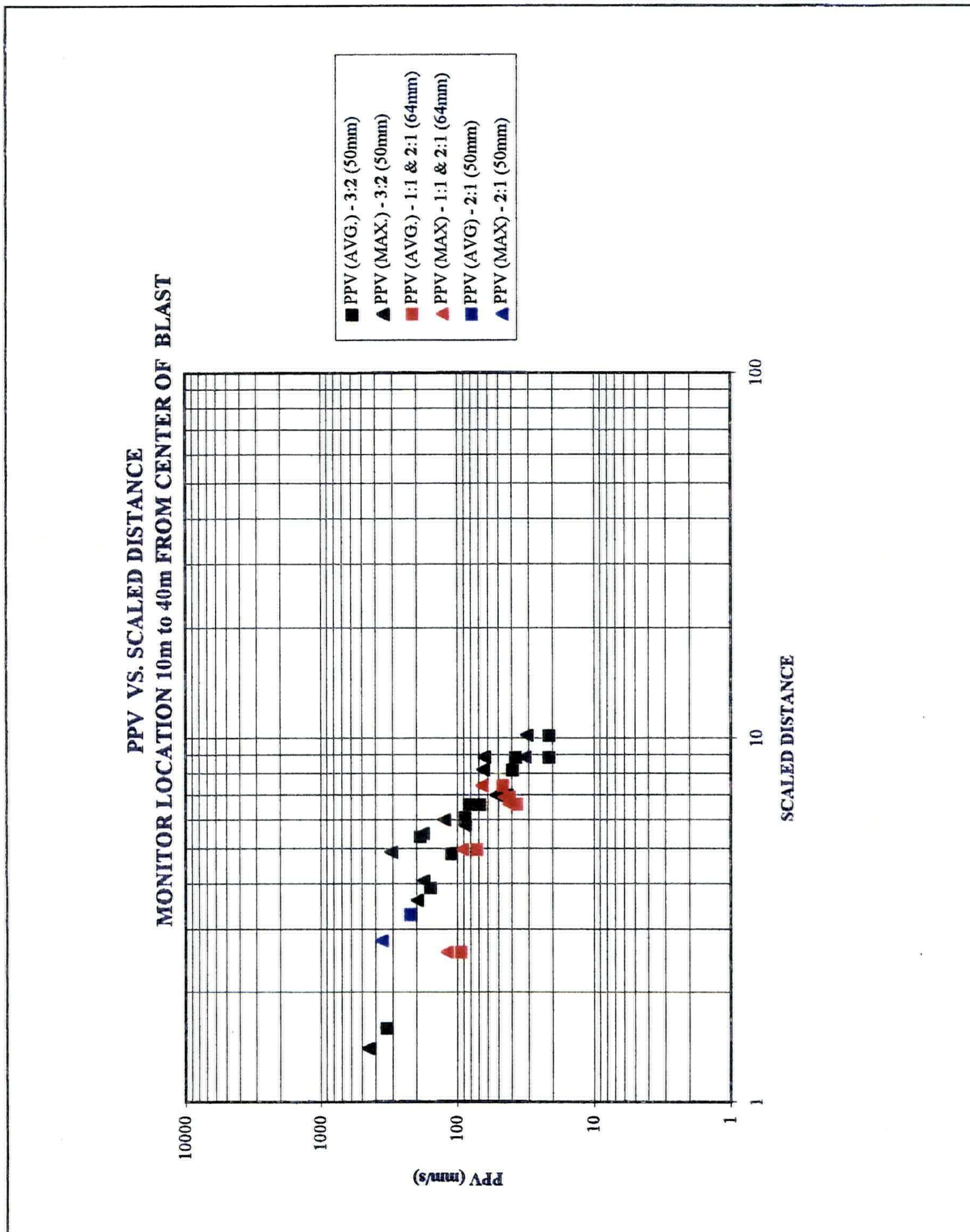


Figure 14.2 PPV vs. Scaled Distance - comparing 50mm diameter and 64mm diameter blastholes, 3:2; 2:1; and 1:1 blast monitoring results are shown

With regard to the unsuccessful trial, there were a number of factors which may of had an adverse impact on the results of the trial. These are listed below:

- the blastholes on the 2-hole rings were designed with little to no offset from the hangingwall and footwall contact;
- prior to blasting, the West Zone Fault was observable in the footwall of the ore drift on both the overcut and mucking level;
- some of the blasts were sequenced improperly (i.e. blastholes on the 2-hole rings fired on the same delay #);
- some of the downholes required deepening to compensate for the upholes (from the level below) benching at the toes. The deepening resulted in blasthole lengths of up to 15m, therefore, hole deviation may have been an issue.

In addition to the above, the majority of the blasts in this stoping block were not monitored, therefore, it is difficult to say with certainty what contributed the most to the stope instability.

In general, stope performance results from the 2:1 pattern, thus far, seem similar to those being achieved with the 3:2 pattern. There is not as obvious a difference in stope stability as there was with the 1:1 pattern. Note however that the 1:1 pattern blasts are designed narrower than the 2:1 pattern blasts. More trials are needed to better determine if the 2:1 pattern will result in significant improvements to stope stability. Based purely on blast vibration levels, the 2:1 pattern should show an improvement over the 3:2 pattern. Note that even if significant improvements in stope stability are not achieved, the cost savings due to reduced drilling make implementation of this pattern worthwhile.

Having said the above, there are definite opportunities to reduce the blast damage potential of the 2:1 pattern. CMS surveys of stopes blasted with the 2:1 pattern are needed to determine quantifiable measurements of overbreak and underbreak so that appropriate blasthole offset distances can be determined. At present blastholes on the 2-hole rings are being offset 0.3m from both the hangingwall and footwall contacts. As was mentioned in Chapter 13, there is considerably more opportunity to experiment with offset distances with the 2:1 pattern as opposed to the 3:2 pattern, since the blasthole are spaced much wider with the 2:1 pattern. As was mentioned earlier in reference to the 1:1 pattern, there is also potential to reduce blast damage by experimenting with reducing linear charge density (kg/m of explosive).

14.1.2 Charge Interaction

The blast monitoring results presented in Chapter 11 show that the 3:2 pattern suffers from a high degree of charge interaction resulting in misfires and inefficient detonations, largely due to the tight spacing of the blastholes. The average number of misfires per blast was determined to be 14% (based on 17 blasts).

With regard to the 1:1 pattern, all three blasts which were monitored showed very consistent energy levels for each of the blastholes (i.e. each blasthole is performing a similar amount of work). This indicates a good distribution of explosive energy. This was rarely observed with the 3:2 pattern. For comparison, Figure 14.3 shows vibration traces from one of the 1:1 pattern trials and a typical 3:2 pattern.

One misfire was recorded during the 1:1 pattern trials, although it is a concern, a poor B-line connection is suspected rather than charge interaction.

With regard to the 2:1 pattern, misfires were recorded in two of the three blasts monitored (1 misfire in each blast). In one case, the cause is suspected to be poor loading practices (i.e. Anfo loaded in a wet hole). In the other case, charge interaction is suspected. More blasts need to be monitored to determine if charge interaction is an issue. If it is, expansion of the pattern should be considered, to improve the distribution of explosive energy. Note there is considerable flexibility with regard to the burden of the 2:1 pattern. Burdens as large as 1m have been used in the past. Trials with a burden of 0.75m are currently planned. Note that if charge interaction is not an issue with the 0.6m burden, expanding the pattern will likely result in higher blast vibrations since the charges will be more confined. In this case, the cost savings associated with further reducing the amount of drilling and explosive consumption will have to be weighed against the cost of possibly increasing the level of blast damage.

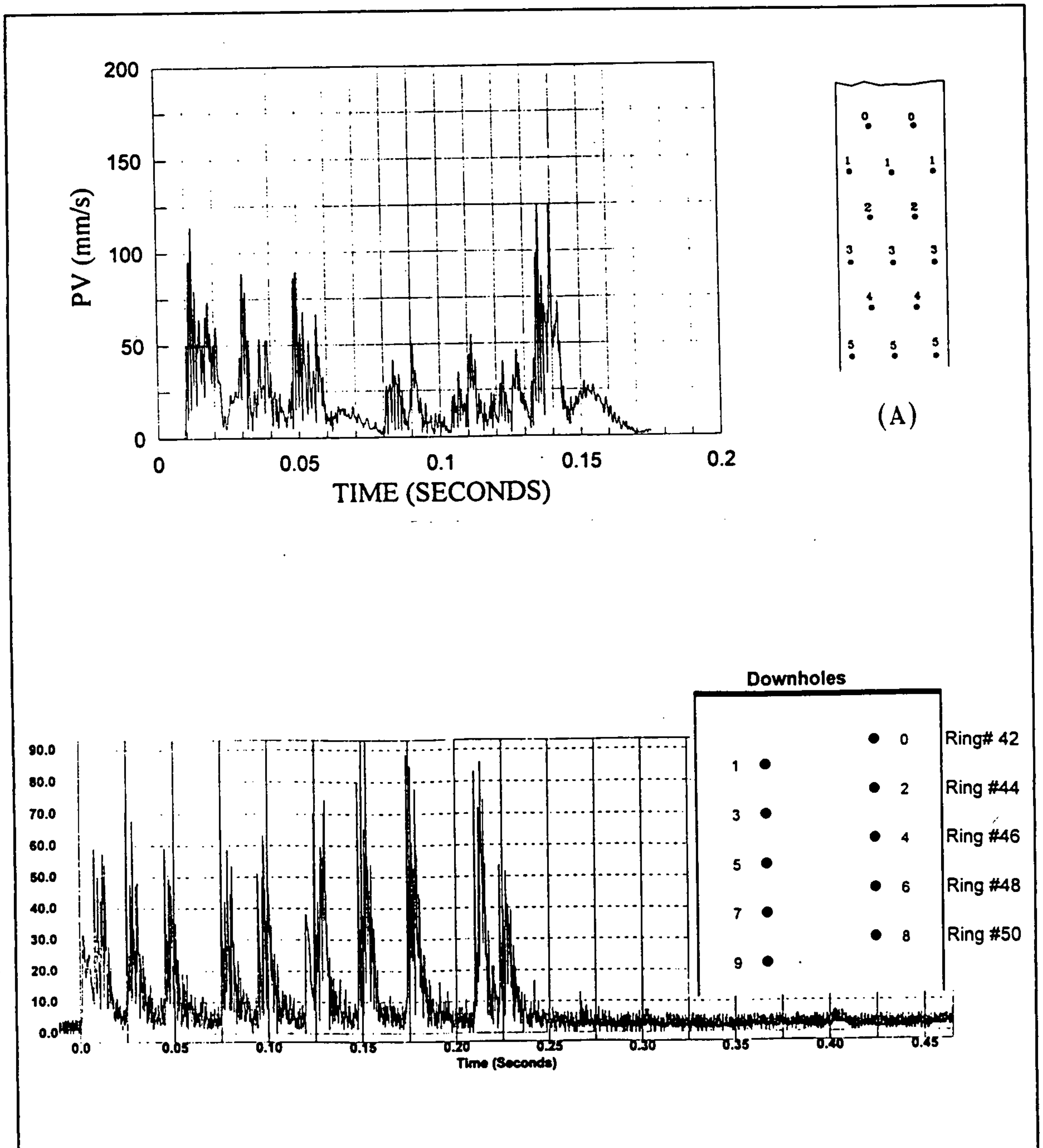


Figure 14.3 Comparison of vibration monitoring results (1:1 pattern vs. 3:2 pattern) - demonstrating a good distribution of explosive energy (1:1 pattern)

14.1.3 Fragmentation

In general, fragmentation can be classified as "fine" (<0.3m) for all 3 patterns, which is smaller than what is considered "good" (0.3m - 0.6m). The 3:2 pattern typically exhibited a bi-modal muck size (i.e. predominantly fine with some coarse fragments). The coarse fragments were likely a result of charge interaction and/or blast related wall instability. With the 1:1 and 2:1 patterns the muck size distribution was typically uniform, which is likely a reflection of the reduced charge interaction. The muck size associated with the 2:1 pattern was noticeably smaller than either the 1:1 or 3:2 patterns indicating significant over-blasting. This suggests that expansion of the pattern (i.e. increasing the burden) should be considered.

With regard to the 3:2 pattern, an exception to the "fine" fragmentation was observed when the 100/3800 dual delay detonators were trialed. During these trials fragmentation was considered "good" (0.3m - 0.6m). Likely because the blast energy was spread over a longer time period resulting in less opportunity for the blasted muck to aid in the fragmentation process (i.e. in-air collisions).

14.1.4 Impact of Drilling and Loading Practices

The main benefit of the 3:2 pattern is the built in forgiveness to occasional lapses in the quality of drilling, and loading practice. The forgiveness is a result of the pattern being over drilled (i.e. too many holes) which ironically is also its biggest drawback.

The 1:1 pattern is the most optimized of the 3 patterns and the least tolerant to lapses in quality of drilling and loading practice. Although, a misfire did occur during one of the trials and the blast still broke, demonstrating that the pattern does have some forgiveness. On the otherhand, 2 benches were experienced during the trials. One bench was caused solely by poor loading practice, and the other by a combination of poor drilling and poor loading practice. The following is knowledge gained from the benched blasts:

- ensuring the proper uncharged collar length is very important, otherwise a web of unbroken rock at the collar elevation may result. The uncharged length should be no greater than the burden distance;

- a burden (ring spacing) of 0.9m (caused by collaring error) is too large (i.e. toes of blastholes may bench).

Note that burdens (ring spacings) as large as 0.7m broke successfully.

The 2:1 pattern as it is currently designed (0.6m burden) appears to be relatively forgiving. Misfires occurred in 2 of the 3 blasts monitored yet both blasts broke well. No benches have been experienced with this pattern. Note however, that if the pattern is expanded (i.e. burden increased) the forgiveness of the pattern will decrease.

14.2 SUMMARY

The following points are a summary of the main findings from this study:

3:2 Pattern

- monitoring of the 3:2 pattern showed a high degree of charge interaction resulting in significant numbers of misfires (avg. 14%) and inefficient detonations. The charge interaction is due to the tight spacing of the blastholes. Bi-modal fragmentation was commonly observed which is another indication of over charging. It is estimated that approximately $280,500.00 annually is being spent on drilling and explosives which is not contributing to fragmentation of the ore;

- the 100/3800 dual delay detonator system was trialed. The system is not recommended due to excessive delay scatter resulting in a high potential for out of sequence firing;

- blast damage potential to the stope walls is highly dependent on the way the 3:2 pattern is sequenced. Blasting each ring on a single delay number has the highest potential for blast damage. Sequencing each hole separately has the lowest potential for blast damage. The difference was obvious in the field. However, due to concerns over drilling accuracy a compromise between the two methods has been developed. Preliminary blast monitoring results indicate the new timing sequence results in energy levels which fall between those generated blasting a ring per delay, and sequencing each hole individually. Blast damage potential is moderate;

- the main benefit of the 3:2 pattern is the built in forgiveness to occasional lapses in quality of drilling, loading practices, and explosive product;

<u>1:1 (Stagger) and 2:1 (Dice-Five) Patterns</u>

- significant cost savings can be realized through implementation of a more optimal blast pattern such as the 2:1 (dice-five) or 1:1 (staggered) pattern. Savings will be realized through decreased drilling and reduced explosive consumption. Additional savings may also result through decreases in unplanned dilution. For example, a 15% reduction in drilling and blasting costs and a 10% reduction in unplanned dilution could result in savings of approximately $650,000.00 annually in the WZ;

- the 1:1 pattern trials have shown very encouraging results. The patterns tested have demonstrated energy levels lower than the 3:2 pattern (as it is currently being sequenced) and the resulting wall stability has been excellent. The pattern has demonstrated only limited forgiveness to lapses in drilling and loading practices. Vibration monitoring indicates a good distribution of explosive energy and minimal charge interaction. The 64mm diameter blasthole pattern (0.5m burden) may be viable for mining ore widths of 1.5m -1.6m. Trials are currently being conducted using a burden of 0.6m. This may prove viable for mining ore widths up to 1.8m wide. Implementation of the 1:1 pattern for ore widths of between 1.5m - 1.8m represents a cost savings (reduced drilling and explosive consumption) of approximately 40% compared to the 3:2 pattern. This does not include cost savings associated with reducing unplanned dilution;

- the 2:1 pattern trials have also shown encouraging results. Energy levels associated with the pattern are lower than the 3:2 pattern (as it is currently being sequenced) although wall conditions thus far seem similar to those being achieved with the 3:2 pattern. However, significant opportunity exists to reduce blast damage associated with the 2:1 pattern through choosing appropriate blasthole offset distances and by reducing linear charge density. No benched blasts were experienced during the 2:1 pattern trials and the pattern as it is currently designed (0.6m burden) appears to be relatively forgiving to lapses in quality of drilling and loading practice. There are indications that charge interaction may be a concern suggesting that the pattern may need to be expanded. It is envisioned that the 2:1 pattern will be used for mining ore widths of between 1.8m - 2.5m. With the 2:1 pattern as it is currently designed, this represents a cost savings (reduced drilling and explosive consumption) of approximately 10% over the 3:2 pattern. This does not include cost savings associated with reducing unplanned dilution. If the pattern was expanded to a 0.75m burden the cost savings would increase to approximately 30% over the 3:2 pattern;

- no trials with long breakthrough downholes have successfully been carried out due to problems with drilling accuracy. The problems are currently being addressed and trials should be carried out in the near future;

- the field trials highlighted some concerns regarding the quality of drilling and blasting practices. The successful implementation of the more optimized 1:1 and 2:1 blast patterns requires a high quality of workmanship. Given the potential cost savings associated with: reduced drilling; reduced explosive consumption; reduced dilution; and fewer benches, there is considerable incentive to successfully implement these patterns. To help ensure quality workmanship a longhole foreman was recently appointed;

- the fact that the energy levels (PPV) associated with both the 1:1 and 2:1 patterns (64mm diameter blastholes) are lower than the 3:2 pattern (50mm diameter blastholes) is direct evidence that confinement plays a large role with regard to the vibration levels produced by a particular blast pattern and that linear charge density and charge weight per delay are not the only factors to consider.

CHAPTER 15

CONCLUSIONS

15.1 INTRODUCTION

The focus of this research was minimizing dilution in open stope mining. The research encompassed both stope design and narrow vein longhole blasting.

A new empirical design approach has been introduced for estimating unplanned dilution from open stope hangingwalls and footwalls. The resulting design charts are based on quantifiable measurements of overbreak/slough made with the *Cavity Monitoring System (CMS),* and were developed from a comprehensive database of stoping histories (CMS database) compiled from six (6) Canadian underground open stoping operations. A new parameter termed ELOS (equivalent linear overbreak/slough) has been introduced and incorporated into the design charts as a measure of unplanned dilution. Theoretical justification for the design methodology has been demonstrated through a numerical modelling study examining the zone of relaxation around open stopes. Statistical methods, neural networks, and additional case histories have been used to validate the proposed design zones. This new approach to stope design is an improvement over existing methods in that it allows stopes to be sized based on an "acceptable" level of dilution, as opposed to, sizing stopes based on qualitative descriptions of stability such as: "stable"; "transition zone"; or "potentially unstable".

The CMS database shows that even if stopes are sized to be inherently stable, blast induced overbreak of up to 0.5m from both the hangingwall and footwall is not uncommon. In narrow vein mining, this represents significant unplanned dilution. Consider a 2m wide orebody, if 0.5m of overbreak occurred on both the hangingwall and footwall, the unplanned dilution would be 50%. To examine blast damage potential in narrow vein longhole mining, an assessment of narrow vein blast patterns was carried out at the Lupin Mine (NWT). The study assessed the performance of three narrow vein blast patterns: the

3:2 pattern; the 2:1 (dice-five) pattern; and the 1:1 (stagger) pattern. The patterns were evaluated on the basis of: cost; blast damage potential; charge interaction; fragmentation; and tolerance to lapses in quality of drilling and loading practice. Guidelines are presented regarding narrow vein pattern selection, design, and implementation.

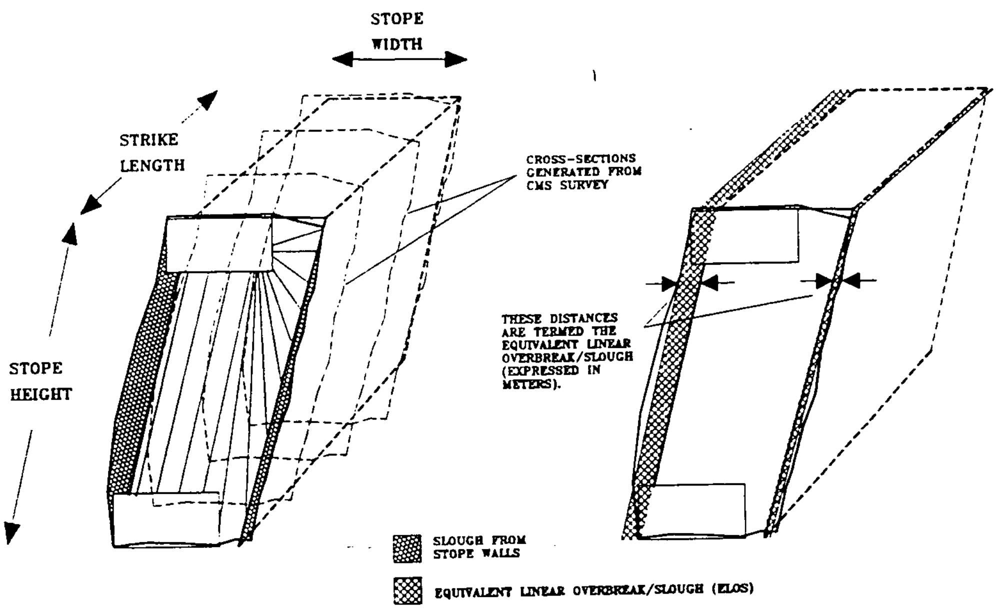
15.2 CONCLUSIONS

15.2.1 Estimating Dilution with Existing Methods of Stope Design

Based on the empirical design techniques reviewed, there is currently no general design method which can be used to size stopes based on some acceptable level of dilution. The existing methods either, provide only qualitative measures of stability, or, are too site specific. The stability graph methods (Mathews et.al, 1981, Potvin, 1988) have however, proven to be useful tools for dimensioning open stopes and have been well accepted by industry. For this reason, quantification of the design zones associated with either of the stability graph methods in terms of unplanned dilution, was considered a good starting point for the development of an empirical method for estimating unplanned dilution. For this study, the Modified Stability Graph Method (Potvin, 1988) was used as a starting point, mainly because of the larger database of published information associated with it.

15.2.2 CMS Database

Development of a quantitative design approach for estimating unplanned dilution required that a database of actual measurements of overbreak/slough be compiled. This was accomplished by conducting stope surveys with the Cavity Monitoring System (CMS) and by compiling detailed stoping histories for the stopes surveyed. Stope performance was measured in terms of two new parameters termed ELOS (equivalent linear overbreak/slough) and ELLO (equivalent linear lost ore) which are alternate ways of expressing the volumetric measurements (m^3) of overbreak/slough and underbreak. They represent conversions of the true volumetric measurements into an average depth (ELOS) or thickness (ELLO) over the entire stope surface. A schematic describing ELOS and the method of calculation is shown in Figure 15.1. ELLO is calculated the same way except "volume of overbreak/slough" is replaced with "volume of underbreak".



PROCEDURE FOR CALCULATING EQUIVALENT LINEAR OVERBREAK/SLOUGH (ELOS)

1) CUT SECTIONS THROUGH THE CMS SURVEY ALONG THE LONGHOLE SECTION LINES.

2) ON EACH SECTION CALCULATE THE AREA (m^2) OF OVERBREAK/SLOUGH FROM EACH OF THE STOPE SURFACES. THE OVERBREAK/SLOUGH SHOULD BE MEASURED RELATIVE TO THE ORE CONTACTS OR MINING LINES. UNPLANNED DILUTION TAKEN DURING DEVELOPMENT OF THE ORE DRIFTS SHOULD NOT BE INCLUDED.

3) BASED ON THE AREAS CALCULATED FROM THE SECTIONS, CALCULATE A VOLUME OF SLOUGH (m^3) FOR EACH STOPE SURFACE. THIS WILL REQUIRE THAT A THICKNESS BE ASSIGNED TO EACH SECTION.

4) THE EQUIVALENT LINEAR OVERBREAK/SLOUGH (m) FOR A GIVEN SURFACE CAN BE CALCULATED USING THE EQUATION GIVEN BELOW:

$$\text{EQUIVALENT LINEAR OVERBREAK/SLOUGH} = \frac{\text{VOLUME OF SLOUGH FROM STOPE SURFACE}}{\text{STOPE HEIGHT X WALL STRIKE LENGTH}}$$

Figure 15.1 Schematic describing ELOS (Equivalent Linear Overbreak/Slough)

The primary database consists of 47 stope surveys from six (6) Canadian open stoping operations, yielding 88 measurements of hangingwall and footwall ELOS. Average values for the main parameters in the CMS database are listed in Table 15.1 below.

Table 15.1 - Average Values for Main Parameters in CMS Database

CMS Database Parameter	Average Value	Remarks
Depth	740 m	
Stope Dip	76 degrees	
Stope Width	7 m	53% of stopes have widths ≤ 4 m.
Stope Strike Length	27 m	
Stope Height	39 m	
Stope Height/Length	2.1	
Hydraulic Radius	7.3	
Wall Surface Character	Regular	
Stope Wall Undercutting	Minor/Moderate	Typical undercut 0.5 - 1 m.
RMR	67	
Q'	12	
Blasthole Diameter	65 mm	63% of stopes mined with blastholes ≤ 65 mm diam.
Blasthole Length	15 m	
Blasthole Offset Distance	0.3 m	
Explosive Type	Anfo	
Powder Factor	0.7 kg/tonne	
Perimeter Blasthole Orientation	Parallel	
Cablebolt Support	34% of HW's supported; 5% of FW's supported	Cables installed on sub-levels; no pattern support
Time Between First Stope Blast and CMS Survey	51 days	53% of stopes mined in ≤ 30 days.
Number of Longhole Blasts to Excavate Stope	7	

15.2.3 An Empirical Design Approach for Estimating Unplanned Dilution from Open Stope Hangingwalls and Footwalls

ELOS design zones have been incorporated onto the Modified Stability Graph (Potvin, 1988) and a new design graph based on RMR has been developed. Plots of the respective design charts are presented in Figures 15.2, 15.3, 15.4, and 15.5. The ELOS design zones are based on engineering judgment. Statistical methods, neural networks, and additional case histories (14 data points) were used to provide justification to the proposed design zones.

The ELOS design zones provide an empirical method of predicting the volume of overbreak/slough associated with a particular design. Unplanned dilution can be calculated based on the predicted volumes. Note that when calculating unplanned dilution, estimates of ELOS from both the hangingwall and footwall need to be considered.

Formal definitions for each of the design zones are presented below:

Blast Damage Only (ELOS < 0.5m)

- Potential for minimal overbreak/slough. The quantity of overbreak/slough will be highly dependent on the quality of drilling and blasting.
- Surface is self supporting, no support is required to maintain a stable excavation.
- Time is expected to have a minimal effect with regard to stability.

Minor Sloughing (ELOS = 0.5m - 1.0m)

- If the surface is unsupported some wall failure should be anticipated before a stable configuration is reached.
- Stope support should be considered. The CMS database provides some evidence that sub-level cable support may be adequate in this design zone. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Wall stability may be sensitive to blasting vibrations and the effects of time. Stopes should be mined quickly and filled.

- Minor operational problems should be anticipated (i.e. some secondary blasting).

Moderate Sloughing (ELOS = 1.0m - 2.0m)

- If no stope support is installed, significant wall failure should be anticipated before a stable configuration is reached.

- Stope support should be considered. The CMS database provides some evidence that in this design zone sub-level cable support may provide little benefit with regard to controlling unplanned dilution in rock masses with RMR'< 55 (N'< 6). If feasible, pattern support should be installed. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Stope stability will be sensitive to blast vibrations and the effects of time. Stopes should be mined quickly and filled.

- If adequate support is not installed, significant operational problems should be anticipated (i.e. secondary blasting, plugged drawpoints, possible ore loss under sloughed material, erratic production)

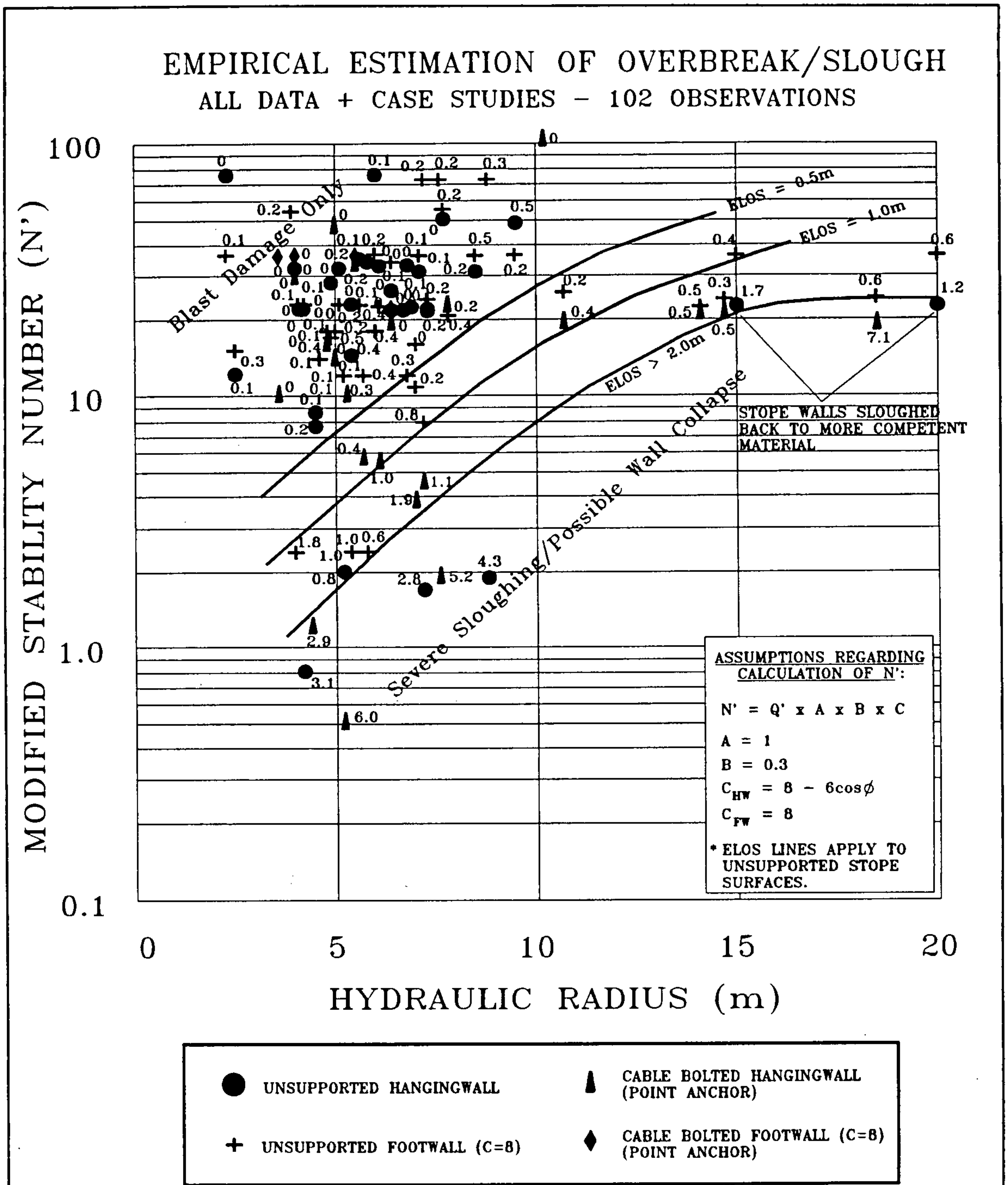
Severe Sloughing / Possible Wall Collapse (ELOS > 2m)

- Potential for large and possibly unacceptable wall failures.

- Sub-level cable support will likely provide little benefit. Pattern support should be considered but may provide limited benefit. For design of cable bolt support refer to Hutchinson and Diederichs (1995).

- Stope stability will be very sensitive to blast vibrations and the effects of time. Stopes should be mined quickly and filled.

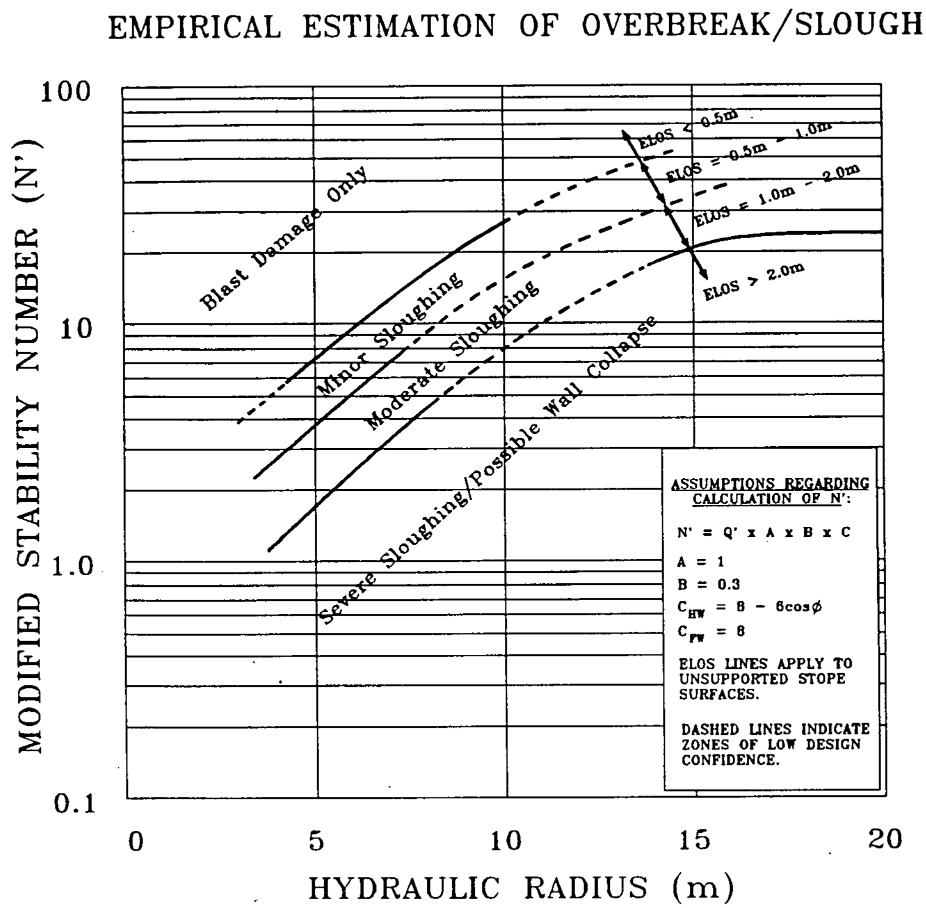
- If stope surfaces cannot be adequately supported, significant operational problems should be anticipated (i.e. secondary blasting, plugged drawpoints, ore loss, erratic production, possible loss of stope)

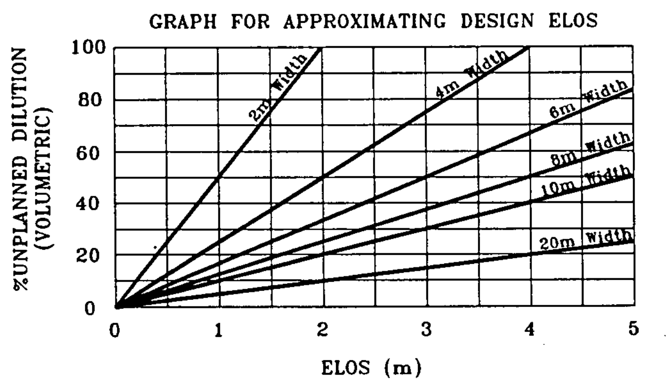


Avg. Values for Select Database Paramaters:

Stope Depth:	740m	Days Stope Open:	51
Stope Dip:	76°	Stope Width:	*7m
Blasthole Ø:	50-65mm	*(53% of stopes ≤4m width)	
Blasthole Length:	15m	Blasthole Layout:	Parallel holes

Figure 15.2 Finalized design chart showing all data - N' vs. HR

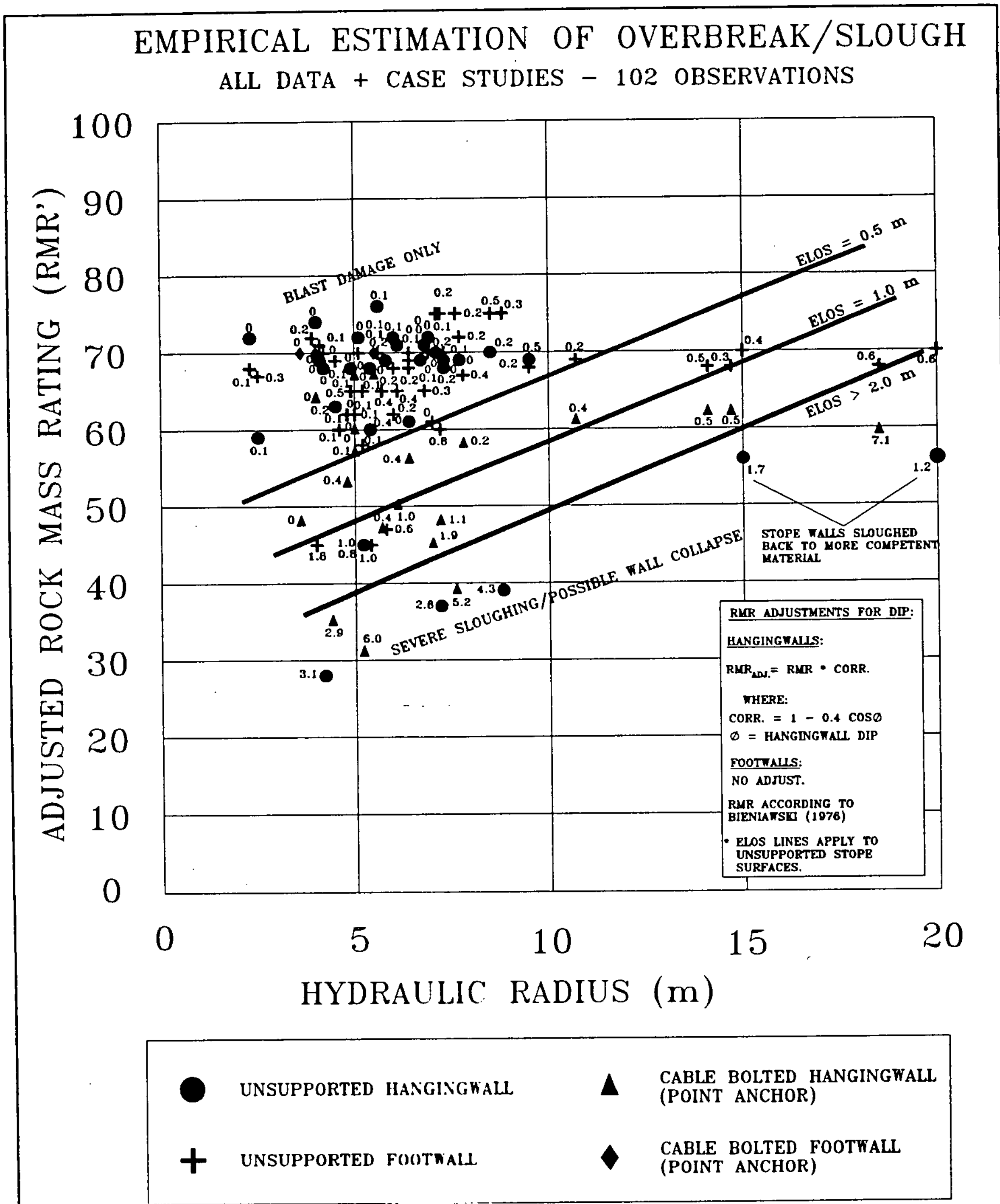




Avg. Values for Select Database Paramaters:

Stope Depth:	740m	Days Stope Open:	51
Stope Dip:	76°	Stope Width:	*7m
Blasthole Ø:	50-65mm	*(53% of stopes ≤4m width)	
Blasthole Length:	15m	Blasthole Layout:	Parallel holes

Figure 15.3 Finalized design chart just showing ELOS design Zones - N' vs. HR



Avg. Values for Select Database Paramaters:

Stope Depth:	740m	Days Stope Open:	51
Stope Dip:	76°	Stope Width:	*7m
Blasthole Ø:	50-65mm	*(53% of stopes ≤4m width)	
Blasthole Length:	15m	Blasthole Layout:	Parallel holes

Figure 15.4 Finalized design chart showing all data - RMR' vs. HR

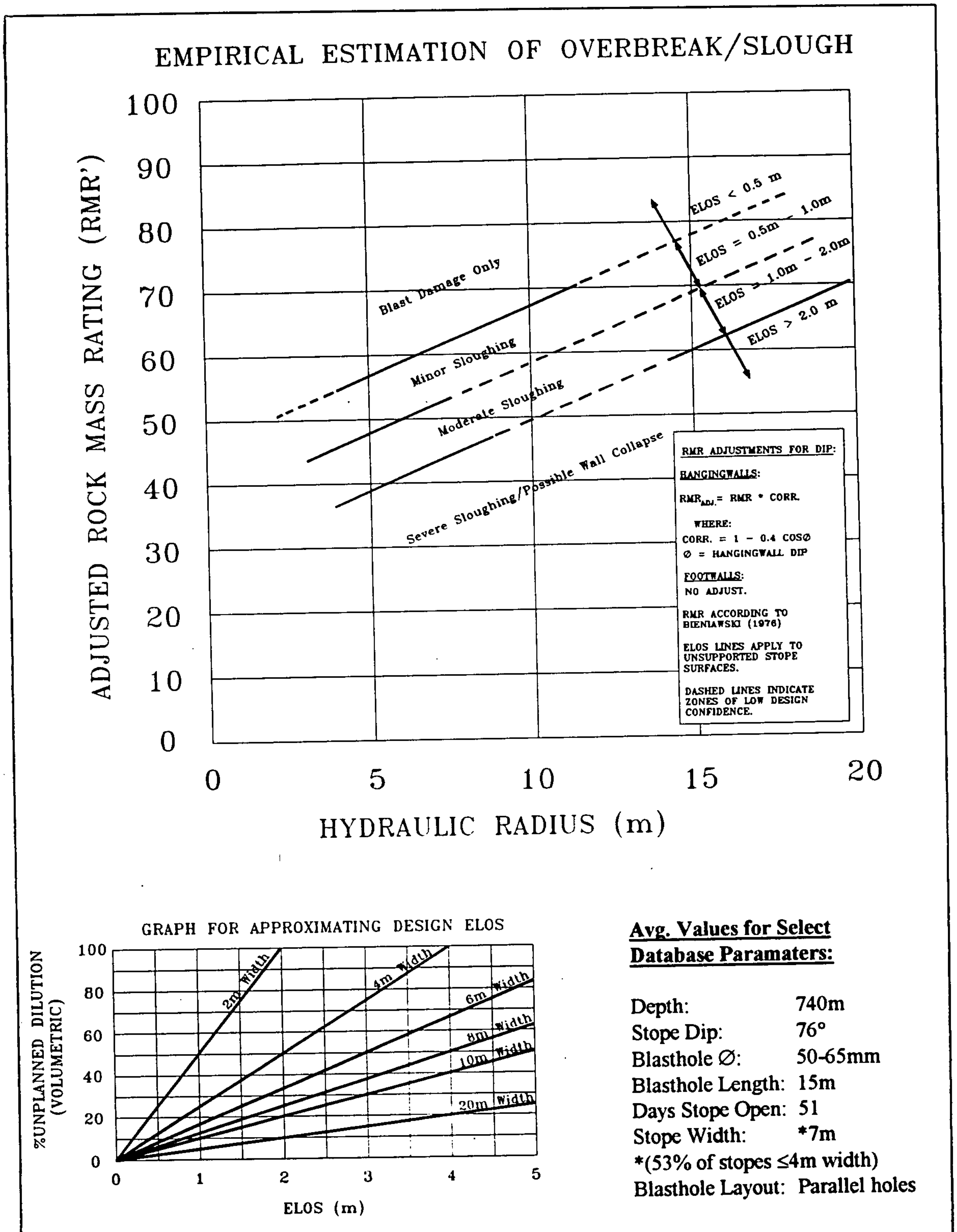


Figure 15.5 Finalized design chart just showing ELOS design Zones - RMR' vs. HR

Limitations of this design approach are as follows:

- the main limitation is the size of the database. More data is required to give confidence to the design zones and to refine the method;
- the method is limited to hangingwalls and footwalls in a low or relaxed stress state with parallel structure being critical with regard to stability;
- the ELOS design zones apply to unsupported hangingwalls and footwalls, more data is required to quantitatively address the effect of cable bolt support;
- the database is biased towards mines that use relatively small diameter blastholes (i.e. <65mm). More data is needed from mines that utilize larger diameter blastholes;
- the database needs more large stopes (i.e. HR>10) and more stopes in poor quality rock;
- there is some uncertainty with regard to the method of calculating N' and RMR' for footwalls. Although a reasonable approach has been proposed, the topic is deserved of more research.

15.2.4 Theoretical Justification for ELOS Design Methodology

Theoretical justification for the ELOS design methodology was investigated through a numerical modelling study (Map3D) examining the zone of relaxation around open stopes. Modelling results were compared to both the theoretical behavior of stope walls and ELOS measurements from the CMS database.

Two design charts were developed: 1) maximum depth of relaxation vs. hydraulic radius; and 2) model ELOS vs. hydraulic radius, refer to Figures 15.6 and 15.7 respectively. Model ELOS values correspond to the volume of the zone of relaxation. The design charts are applicable for vertical or near vertical stope walls where the maximum principal stress is horizontal and oriented perpendicular to the strike of the stope.

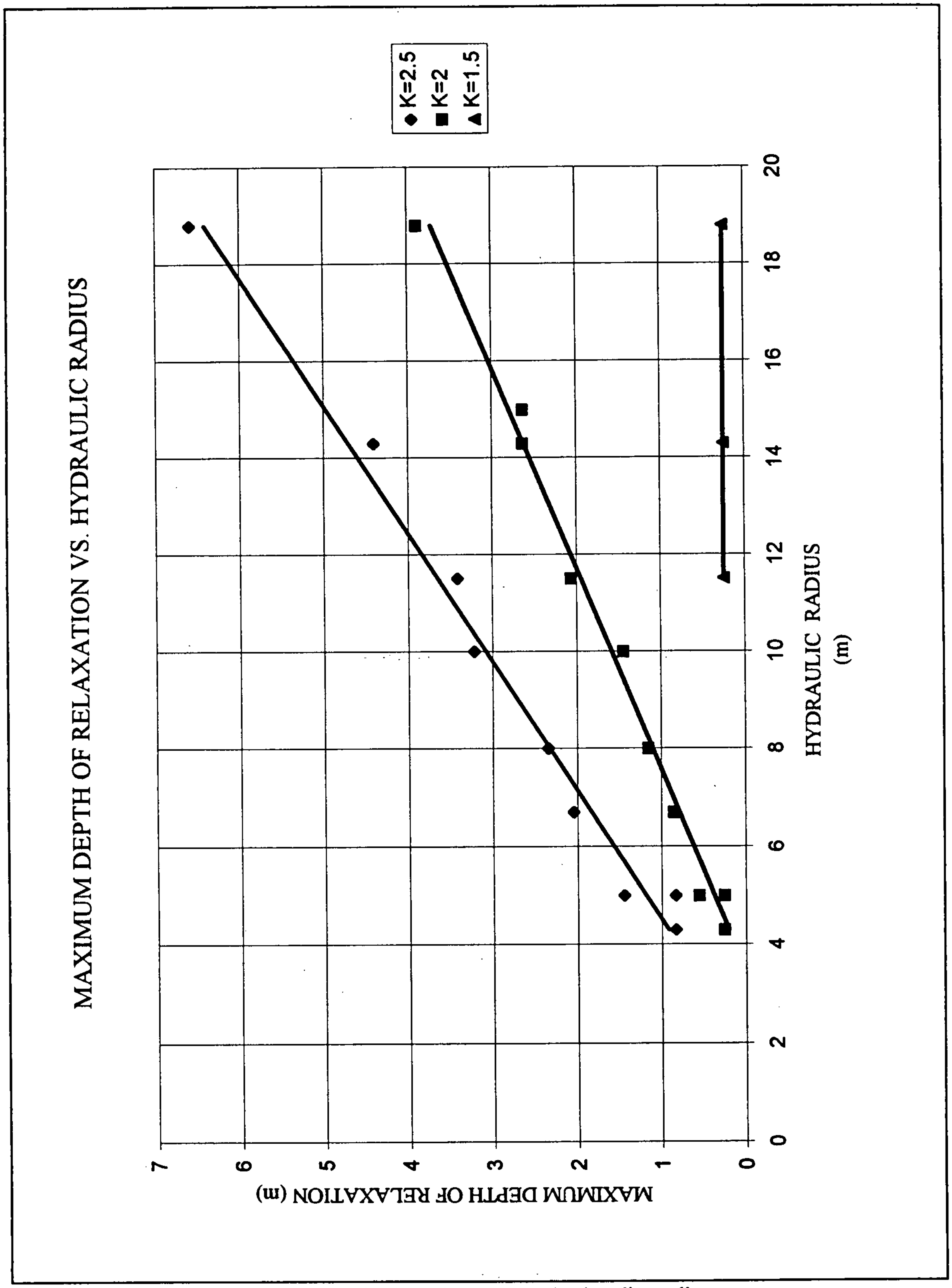


Figure 15.6 Maximum depth of relaxation versus hydraulic radius

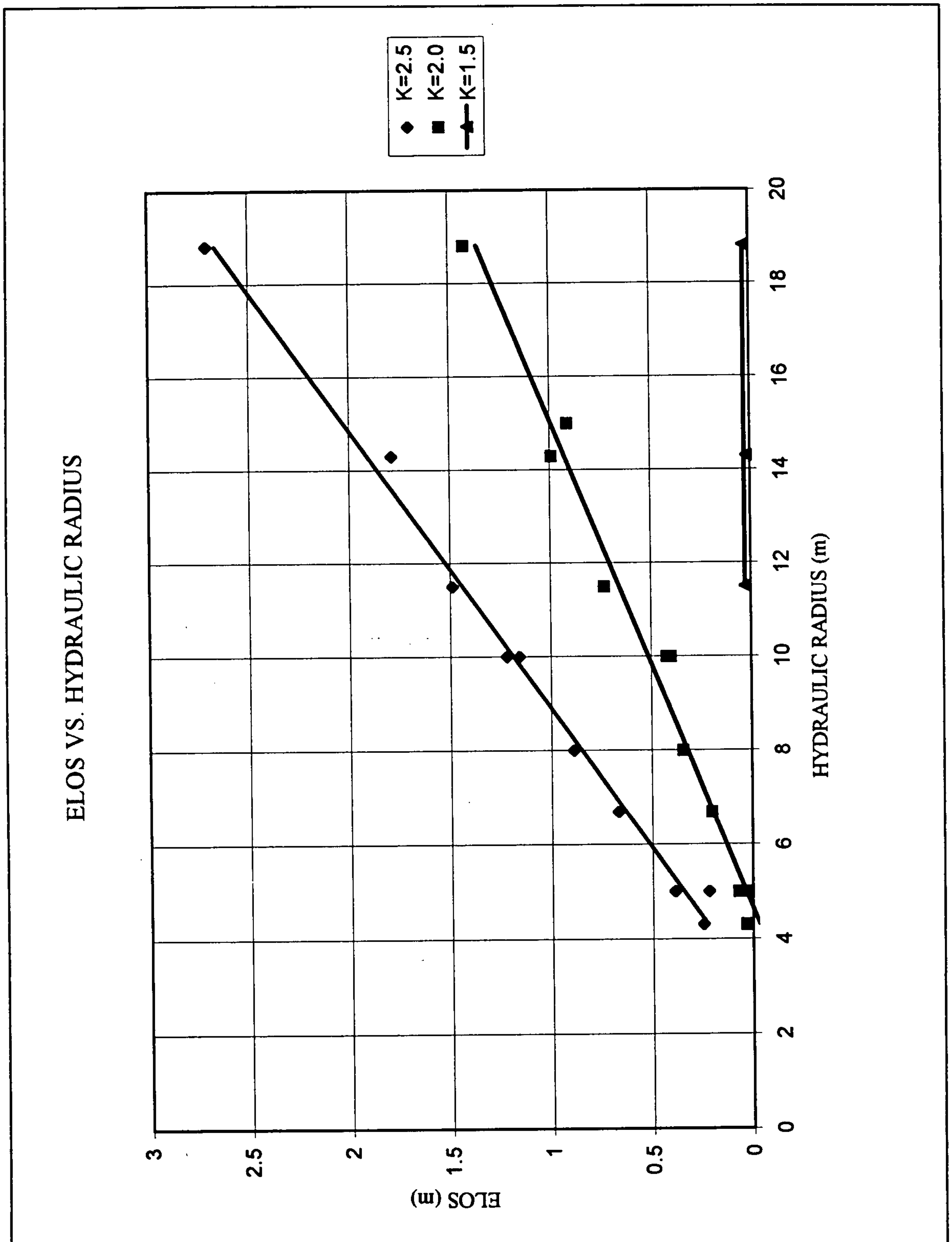


Figure 15.7 Model ELOS values (corresponding to the zone of relaxation) versus hydraulic radius

Both figures have potential application for design of cablebolt support (i.e. design of cable density; and cable length). Figure 15.7 may also have application for estimating ELOS from vertical or near vertical stope walls within a certain range of rock quality. This is discussed in the following paragraph.

A reasonable correlation was found between model ELOS values and measured ELOS values for rock masses with: N' values between 10 and 30; RMR' values between 58 and 68; and assuming K values ($\sigma h/\sigma v$) of between 2 to 2.5. The N' and RMR' values noted above correspond to rock mass qualities of: Q'= 4.2 - 12.5; and RMR = 58 - 68, this assumes vertical stope walls and the presence of discontinuities parallel to the hangingwall and footwall. Intuitively this range of rock mass qualities makes sense. Based on field experience, rock masses of this quality are generally very structured and would be susceptible to gravity driven block failures which could occur in a relaxed rock mass. Rock masses which exhibit high rock qualities likely do not posses enough structure to be susceptible to gravity driven failures. Rock masses which exhibit very low rock qualities would likely unravel beyond the limits defined by the zone of relaxation. The reasonable correlation between the model ELOS values and measured ELOS values (within a realistic range of N' and RMR') provides some theoretical justification for the ELOS design zones.

15.2.5 Relationships Between ELOS and other Database Parameters

Using both scatter plots and neural networks an analysis of the CMS database was undertaken to try and determine which factors, other than those accounted for in N' and RMR', influence hangingwall and footwall stability.

The following is a summary of the findings and rough guidelines that can be used to help determine if the ELOS estimated using the design charts presented in Figures 15.2 through 15.5 will be optimistic (i.e. low).

- *Irregular Wall Geometry* - most of the stopes in the CMS database have regular wall profiles (i.e. relatively planar surfaces). Based on the irregular and complex wall profiles contained in the CMS database, the scatter plot analysis and the neural network analysis provide support to the idea that ELOS will tend to increase as the regularity of the wall geometry decreases.

- *Undercutting of Stope Walls* - the scatter plot analysis showed that the destabilizing effect of undercutting is somewhat dependent on the stability number (i.e. RMR' or N'). Stope walls with an RMR' < 50 (N' < 4) appear to be very sensitive to undercutting. An ELOS value equivalent to the undercut depth should be anticipated for rock masses with stability numbers lower than these values.

- *Blasthole Diameter and Blasthole Length* - Both the scatter plot and neural network analysis suggest a relationship of increasing ELOS with increasing blasthole length and blasthole diameter. The database is largely comprised of stopes that were excavated using blastholes less than 65mm diameter and lengths less than 20m. The design charts may underestimate ELOS where large blastholes of length greater than 20m are used.

- *Blasthole Layout* - the neural network analysis indicated a relationship between ELOS and blasthole layout. This indicates that there may be a tendency for ELOS to increase when using fanned blastholes as opposed to parallel blastholes. The majority of blastholes in the database were drilled parallel to the planned stope surface.

- *Blasthole Offset* - The majority of perimeter blastholes were offset 0 - 0.5m from the excavation perimeter, with the average being approximately 0.3m. The design charts may underestimate ELOS if small or no offsets are used in rock masses with lower stability numbers (i.e. RMR' < 60 or N' < 12). This effect will likely increase with increasing blasthole diameter.
- *Stope Life and Number of Stope Blasts* - The scatter plot analysis showed a correlation between ELOS and the number of stope blasts and both the neural network scatter plot analysis suggested a correlation between ELOS and stope life. It is expected that as stopes plot progressively below the *Blast Damage Only Zone* shown on the design charts, wall stability will become increasingly sensitive to these parameters. The majority of the stopes in the database were open for less than 50 days and were excavated with less than 9 longhole blasts.

15.2.6 Design Guidelines - Empirical Estimation of Unplanned Dilution from Open Stope Hangingwalls and Footwalls

Guidelines on how to use this design approach are presented through worked examples shown in Appendix II.

15.2.7 Assessment of Narrow Vein Longhole Blast Patterns at the Lupin Mine (NWT)

An assessment of three (3) narrow vein longhole blast patterns was carried out at the Lupin Mine (NWT). The patterns assessed were the: 3:2 pattern; 2:1 (dice-five) pattern; and 1:1 (stagger) pattern. The patterns were evaluated based on: costs; blast damage potential; charge interaction; fragmentation; and tolerances to lapses in quality of drilling and loading practice. Costs were determined based on drilling and explosive costs at the Lupin Mine, the remaining criteria were evaluated based on field trials. Field performance was monitored using: underground observation; near field blast vibration monitoring; and when time permitted CMS stope surveys.

The following are brief notes concerning the field trials:

- 3:2 pattern: blasthole diameter was 50mm; burden (ring spacing) was 0.75m; average blasthole length was 9m; pneumatically loaded Anfo was the primary explosive used; either short period delays (25ms non-electric) or dual delay detonators (100/3800 non-electric) were used for sequencing; blastholes were single or double primed with either Geldyne (NG-based explosive) or 90gram cast pentolite boosters; blast vibrations were recorded for 14 blasts;

- 2:1 (dice-five) pattern: blasthole diameter was 64mm; burden (ring spacing) was 0.6m; average blasthole length was 12m; pneumatically loaded Anfo was the primary explosive used; short period delays (25ms non-electric) were used for sequencing; blastholes were double primed with 90gram cast pentolite boosters; blast vibrations were recorded for 3 blasts;

- 1:1 (stagger pattern): the majority of trials used 64mm diameter blastholes; burden (ring spacing) was 0.5m; note that one trial using a 0.6m burden (ring spacing) has been successfully completed since the writing of Chapters 13 and 14; one trial with 50mm diameter blastholes was conducted; burden (ring spacing) was 0.4m; pneumatically loaded Anfo was the primary explosive used in all the trials; average blasthole length was 8.4m; short period delays (25ms non-electric) were used for sequencing; blastholes were double primed with 90gram cast pentolite boosters; blast vibrations were recorded for 4 blasts;

No trials with full length breakthrough downholes (i.e. 17m length) were conducted due to problems with drillhole accuracy. Hole deviation problems at the Lupin mine are currently being addressed, and future trials are planned.

The average ore width for the field trials was 1.9m.

The following points are a summary of the main findings from this study:

3:2 Pattern

- monitoring of the 3:2 pattern showed a high degree of charge interaction resulting in significant numbers of misfires (avg. 14%) and inefficient detonations. The charge interaction is due to the tight spacing of the blastholes. Bi-modal fragmentation was commonly observed which is another indication of over charging. It is estimated that at the Lupin Mine, approximately $280,500.00 annually is being spent on drilling and explosives which are not contributing to fragmentation of the ore;

- the 100/3800 dual delay detonator system was trialed. The system is not recommended due to excessive delay scatter resulting in a high potential for out of sequence firing;

- blast damage potential to the stope walls is highly dependent on the way the 3:2 pattern is sequenced. Blasting each ring on a single delay number has the highest potential for blast damage. Sequencing each hole separately has the lowest potential for blast damage. The difference in overbreak was obvious in the field. However, due to concerns over drilling accuracy and potential problems with misfires and sympathetic detonations, a compromise between the two methods was developed, refer to Figure 15.8. Preliminary blast monitoring results indicate the new timing sequence results in energy levels which fall between those generated blasting a ring per delay, and sequencing each hole individually.

- the main benefit of the 3:2 pattern is the built in forgiveness to occasional lapses in quality of drilling, loading practices, and explosive product, which is essentially a result of there being more blastholes than required;

- with regard to drilling and explosive consumption costs, the 3:2 pattern was the most expensive of the three patterns evaluated. Table 15.2 below, compares costs for the three patterns assuming an ore width of 1.8m.

Table 15.2
Narrow Vein Longhole Pattern Cost Comparison
(Assumes Ore Width of 1.8m)

3:2 Pattern **Blasthole Ø = 50mm** **Burden = 0.75m**	**2:1 Pattern** **Blasthole Ø = 64mm** **Burden = 0.6m**	**2:1 Pattern** **Blasthole Ø = 64mm** **Burden = 0.75m**	***1:1 Pattern** **Blasthole Ø = 64mm** **Burden = 0.6m**
\$9.82/ston	\$8.70/ston (11% savings)	\$6.95/ston (29% savings)	\$5.83/ston (40% savings)

Costs do not include labour associated with cleaning and loading blastholes
Costs savings are relative to the 3:2 pattern
*1:1 pattern assumes 0.3m overbreak on HW and FW, which is reasonable given results form field trials

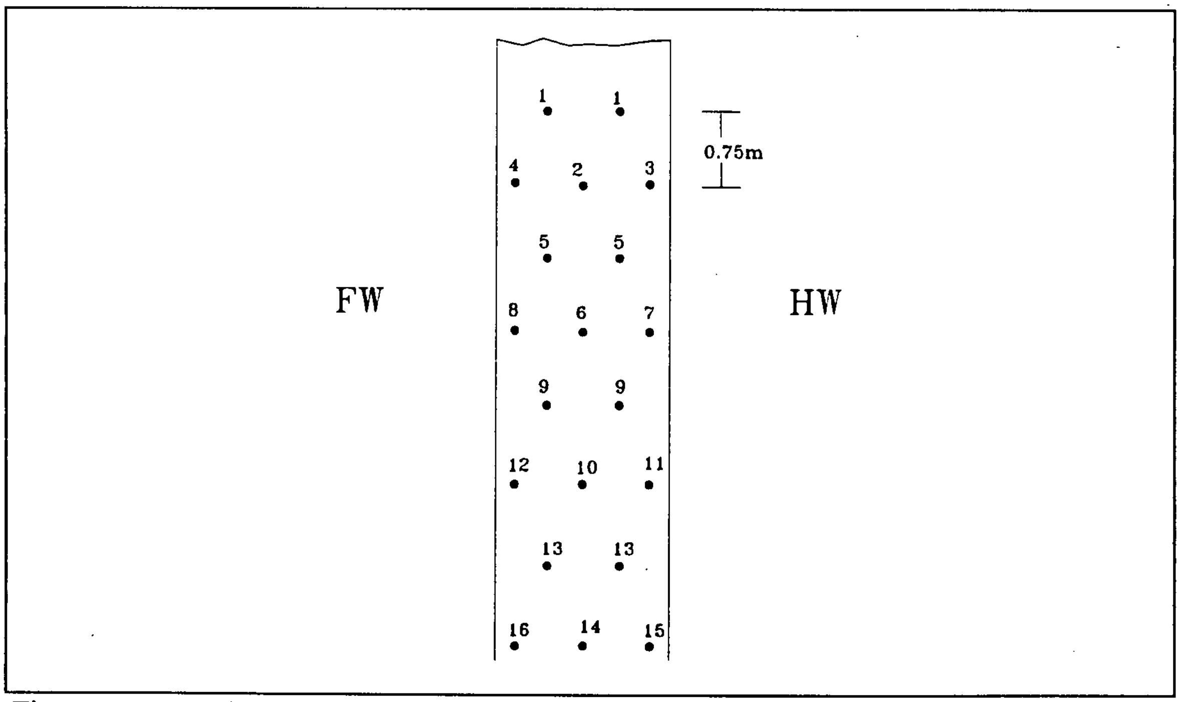


Figure 15.8 Alternate timing sequence for 3:2 pattern

1:1 (Stagger) and 2:1 (Dice-Five) Patterns

- significant cost savings can be realized through implementation of a more optimal blast pattern such as the 2:1 (dice-five) or 1:1 (staggered) pattern. Savings will be realized through decreased drilling and reduced explosive consumption. Additional savings may also result through decreases in unplanned dilution. For example, a 15% reduction in drilling and blasting costs and a 10% reduction in unplanned dilution could result in savings of approximately $650,000.00 annually in the West Zone;

- the 1:1 pattern trials have shown very encouraging results. The patterns tested have demonstrated energy levels lower than the 3:2 pattern (as it is currently being sequenced - Figure 15.8) and the resulting wall stability has been excellent. Two benches were experienced during the field trials which demonstrates the pattern has only limited forgiveness to lapses in drilling and loading practices. Vibration monitoring indicates a good distribution of explosive energy and minimal charge interaction. Fragmentation was fine and of uniform size. The 64mm diameter blasthole pattern (0.5m burden) may be viable for mining ore widths of 1.5m -1.6m. Since the preliminary field trials (Chapters 13 and 14) a trial has been conducted using a burden of 0.6m. Preliminary results suggest it may be viable for mining ore widths of up to 1.8m. Implementation of the 1:1 pattern for ore widths up to 1.8m represents a cost savings (reduced drilling and explosive consumption) of approximately 40% compared to the 3:2 pattern. This does not include cost savings associated with reducing unplanned dilution;

- the 2:1 pattern trials have also shown encouraging results. When the pattern is sequenced such that there is only one blasthole firing per delay, energy levels associated with the pattern are lower than the 3:2 pattern (as it is currently being sequenced - Figure 15.8), although, wall conditions thus far seem similar to those being achieved with the 3:2 pattern. However, significant opportunity exists to reduce blast damage associated with the 2:1 pattern through choosing appropriate blasthole offset distances and by reducing linear charge density. No benched blasts were experienced during the 2:1 pattern trials and the pattern as it is currently designed (0.6m burden) appears to be relatively forgiving to lapses in quality of drilling and loading practice. There are indications that charge interaction may be a concern suggesting that the pattern may need to be expanded. Further blast monitoring is required to confirm whether or not charge interaction is an issue. Fragmentation is typically of uniform size but noticeably finer than that produced by the 3:2 or 1:1 patterns. This also suggests that expansion of the pattern may be warranted. It is being proposed that the 2:1 pattern be used for mining ore widths of between 1.8m - 2.5m. With the 2:1 pattern as it is currently designed, this represents a minimum cost savings (reduced drilling and explosive consumption) of approximately 10% over the 3:2 pattern. This does not include cost savings associated with reducing unplanned dilution. If the pattern was expanded to a 0.75m burden the cost savings would increase to approximately 30% over the 3:2 pattern;

- no trials with long breakthrough downholes have successfully been carried out due to problems with drilling accuracy. The problems are currently being addressed and trials should be carried out in the near future;

- the field trials highlighted some concerns regarding the quality of drilling and blasting practices. The successful implementation of the more optimized 1:1 and 2:1 blast patterns requires a high quality of workmanship. Given the potential cost savings associated with: reduced drilling; reduced explosive consumption; reduced dilution; and fewer benches, there is considerable incentive to successfully implement these patterns. To help ensure quality workmanship a longhole foreman was recently appointed;

- both the 1:1 and 2:1 patterns were designed with relatively light burdens. The fact that the blast vibrations associated with both the 1:1 and 2:1 patterns (64mm diameter blastholes) are lower than the 3:2 pattern (50mm diameter blastholes) is direct evidence that confinement plays a large role with regard to blast vibration levels, and hence the blast damage potential associated with a particular blast design, and that linear charge density and charge weight per delay are not the only factors to consider, refer to Figure 15.9. Note that expanding a pattern (i.e. increasing the burden) increases the confinement felt by a charge, which subsequently results in increased blast vibrations. Cost savings associated with reducing the amount of drilling and explosive consumption, which result from expanding a pattern, need to be weighed against the cost of potential increases in blast damage.

15.2.8 <u>Design Guidelines - Narrow Vein Pattern Selection and Implementation</u>

Based on the field trials, Table 15.3 presents the guidelines being implemented at the Lupin Mine, for narrow vein pattern selection and design. The guidelines apply to 64mm diameter blastholes.

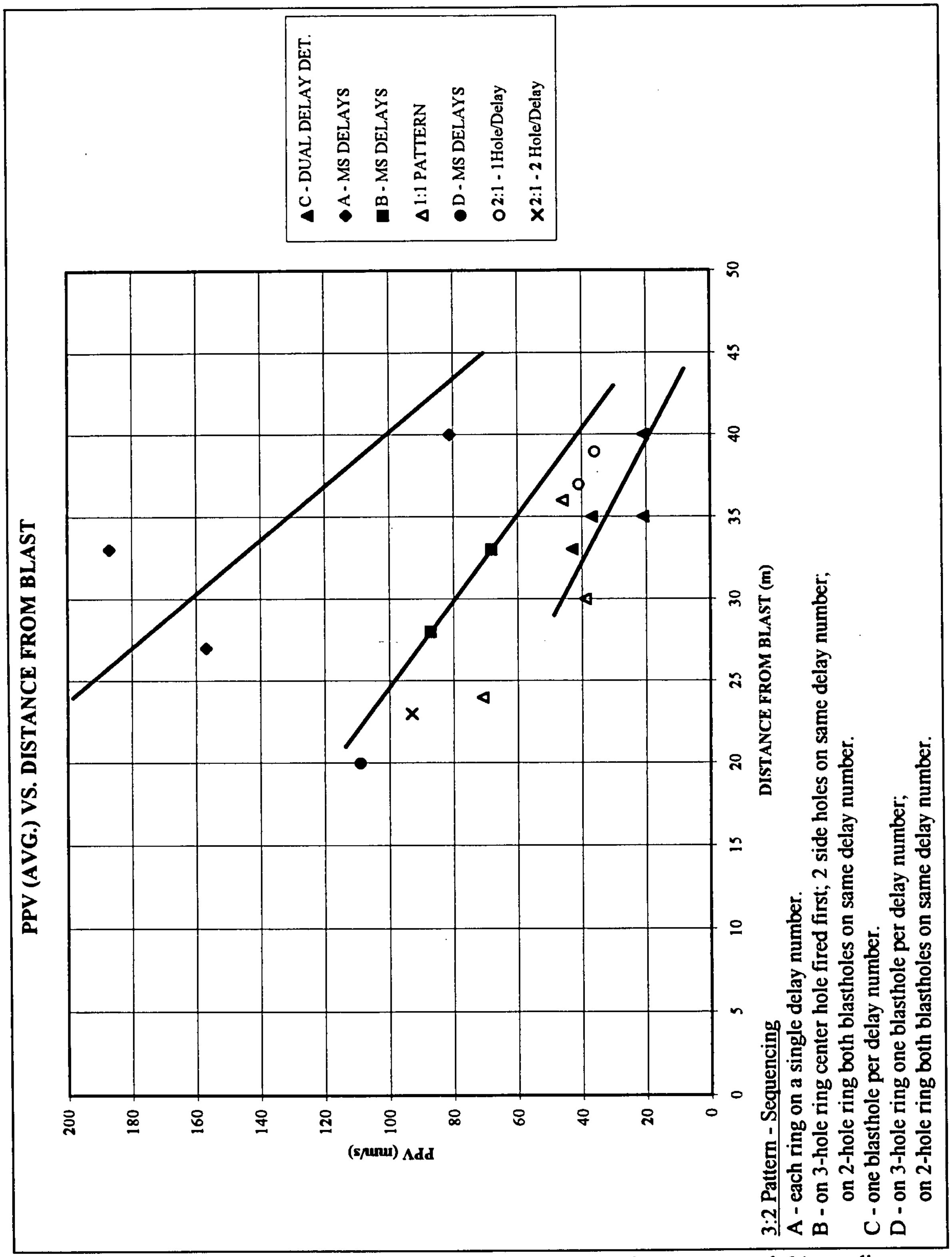


Figure 15.9 PPV (Avg.) vs. Distance from blast - comparing 50mm and 64mm diameter blastholes, 3:2; 2:1; and 1:1 blast monitoring results are shown

Table 15.3
Pattern Selection and Design for 64mm Diameter Blastholes

Pattern	Ore Width (m)	Burden (Ring Spacing) (m)	Offset Distance from HW & FW Contact (m)	Remarks
1:1 (stagger)	1.4m≤width<1.8m	0.6m	Pattern centered between ore contacts	Only one trial with this burden has been completed
2:1 (dice-five)	1.8m≤width<2.5m	0.6m	0.3m	Future trials are being planned with a burden of 0.75m
3:2	2.5m≤width<3.5m	0.75m	0.3m	

All blasts are to be sequenced such that there is only one blasthole firing per delay number.

When sequencing 2:1 patterns, on the 2-hole rings, the blasthole adjacent to the more competent wall should be fired first. The reason being is that the first hole to fire on the 2-hole ring has a tight break-out angle and thus higher potential for blast damage.

As of now the guidelines are still tentative, more blast monitoring and CMS surveying are required to finalize burdens and offset distances.

With regard to pattern selection for 50mm diameter blastholes only very tentative guidelines can be given, since only the 3:2 pattern was evaluated in detail. The following points can be used as a guide:

- the 3:2 pattern was used for approximately 8 years at the Lupin Mine for mining ore with widths of between 1.5m - 3.0m. On average, unplanned dilution was approximately 40-45%. For ore widths less than 2m, significant charge interaction should be expected due to the close spacing of the blastholes. Blast damage potential is dependent on how it is sequenced. Blast damage can be minimized by limiting the number of blastholes per delay. In narrow ore, in order to maintain a reasonable burden/spacing relationship, it may not be possible to offset the blastholes from the ore contacts, which may limit efforts to reduce blast induced overbreak. Drilling and explosive costs are high relative to the 2:1 or 1:1 patterns. The pattern has considerable tolerance to lapses in quality of drilling and loading practice because essentially there are more blastholes than necessary.

- 2:1 patterns with burdens ranging between 0.75m - 0.8m have not been successful in the past at the Lupin mine. It is likely that the patterns were too optimized and did not have enough forgiveness built into the patterns for the level of quality control that was being practiced at that time. At the Campbell Mine (Placer Dome Canada Ltd.), a 2:1 pattern utilizing 54mm diameter blastholes and a burden of 0.6m has been used successfully for many years for ore widths up to 3m. With regard to minimizing overbreak, the 2:1 pattern offers considerably more opportunity to experiment with offset distances compared to the 3:2 pattern.

- during this study only a single 1:1 pattern trial was conducted using 50mm diameter blastholes. A burden (ring spacing) of 0.4m was used. The pattern broke to an average width of 1.1m. It is suspected that the burden (ring spacing) could be increased to 0.5m - 0.6m and that ore of up to 1.5m could be mined successfully.

A potential drawback of the 50mm diameter blastholes, is that it may prove difficult to maintain drillhole accuracy for blastholes longer than 10-15m. In addition, the smaller diameter makes it less tolerant to problems associated with squeezing and shifting of blastholes in stressed ground.

The following are points to consider regarding narrow vein pattern design and implementation:

- when selecting and designing narrow vein patterns the level of optimization should reflect the level of quality control over drilling and loading practices. For example, implementing a 1:1 (stagger) pattern is likely not a good idea if drilling accuracy is not going to be monitored closely;

- at the Lupin Mine, it was found beneficial to design some forgiveness into the patterns (i.e. design the burdens to be light rather than heavy) to help compensate for occasional lapses in drilling or loading quality. Using lighter burdens also lessens the confinement felt by the blasthole which in turn lessens blast induced vibrations and hence lowers the potential for blast damage. However, pattern costs and potential for charge interaction need to be carefully evaluated;

- successful implementation of optimized patterns such as the 2:1 (dice-five) and 1:1 (stagger) patterns requires a high quality of drilling and loading practice. Hole accuracy needs to be checked and holes re-drilled if necessary. Loading practice should be monitored to ensure: all the holes have been cleaned; the proper explosives are being used, appropriate priming practices are being followed; unloaded collar lengths are as designed; the sequencing is as designed; and proper hook-up procedures are being followed. To help ensure quality workmanship at the Lupin Mine, a longhole foreman was recently appointed;

- implementation of any blast pattern will likely require some degree of "field fitting". Near field blast monitoring and CMS surveys are invaluable tools for determining if blasts are performing as planned and for helping to develop methods of improving the design.

15.3 FUTURE WORK

15.3.1 Empirical Estimation of Unplanned Dilution From Open Stope Hangingwalls and Footwalls

Future work should focus on expanding the CMS database. More data is needed to refine and calibrate the ELOS design zones presented in this study. At present the database needs more measurements associated with large stopes (i.e. HR > 10) and more measurements from stopes in poor quality rock masses. The database is also biased towards mines that use small diameter blastholes. More data is needed from mines that utilize larger diameter blastholes (i.e. >64mm).

Once a larger database has been established the following topics are deserved of further research:

- footwalls: the calculation of N' and RMR' for footwalls needs to be looked at in greater detail;

- cable bolt support: the effect of cablebolt support needs to be quantitatively addressed;

- incorporation of additional factors: this study has identified factors, other than those accounted for in the calculation of N' and RMR', which influence stope stability. Only broad guidelines regarding there influence have been presented. More work is required to incorporate these factors into stope design;

- stress - the effect of stress on hangingwall and footwall stability needs to studied in greater detail.

15.3.2 Narrow Vein Longhole Blasting

With regard to the study conducted at the Lupin Mine, more blast monitoring and CMS surveying is required to finalize burdens and appropriate offset distances for the 1:1 (stagger) and 2:1 (dice-five) patterns. Improvements in drilling accuracy are needed before these pattern can be implemented with full length breakthough downholes (i.e. 17m length). Future work should also examine methods of reducing blast damage through reducing linear charge density (i.e. pour loading Anfo; using decoupled explosives).

In a more general sense, guidelines for selection of narrow vein longhole patterns need to be developed for blasthole diameters other than 50mm and 64mm. Furthermore, more work needs to be done on determining limitations of drilling accuracy with different *blasthole diameter / drill string* combinations. For example, is it impractical to design 64mm diameter blastholes, of 20m length, drilled using FI 38 rods, on a 1:1 (stagger) pattern with a burden of 0.6m, or, can you only achieve the desired accuracy with a larger diameter blasthole and tube steel.

More research is needed linking blast induced overbreak to factors such as rockmass quality, rockmass strength, explosive properties, confinement; charge weight per delay, and linear charge density.

Future work should also examine the performance of other narrow vein longhole patterns such as the 2:2 pattern.

15.4 FINAL REMARKS

As discussed in Chapter 1, there are many factors which can influence the dilution and recovery realized for a particular open stope. This study has tried to address many of the factors pertaining to unplanned dilution. Planned dilution was only discussed briefly, but is equally as important, and deserved of more research.

The CMS instrument has proven to be a valuable engineering tool enabling quantifiable measurements of stope performance and providing insight into many of the factors which influence stope stability. As demonstrated in this study, it has permitted a move away from qualitative methods of stope design to design approaches based on actual measurements of overbreak/slough. As more CMS information is gathered, and the effects of many of the factors influencing unplanned dilution are quantified, stope design will rely less on intuition and experience and more on engineering principles. The key will be to ensure that the CMS database is continually expanded upon and re-analyzed. It would be beneficial to have a research institution such as a University dedicated to carrying out this task, thus becoming a sort of repository for CMS data.

The CMS instrument also proved to be a valuable tool for longhole blast design. When coupled with near field blast monitoring it provides valuable insight into factors such as blast induced overbreak and underbreak. This allows blast design parameters such as linear charge density, charge weight per delay, confinement, and blasthole offset distances to be quantitatively evaluated. This greatly improves the design process, resulting in better blast designs, which subsequently result in reduced dilution and increased recovery.

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APPENDIX I

CMS DATABASE

GENERAL					GEOMETRY						
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	AVG. DIP	AVG. WIDTH (m)	STRIKE LENGTH (m)	EXPOSED HEIGHT (m)	H/L RATIO	HYDRAULIC RADIUS (m)	SURFACE CHARACTER
Trout Lake	7251	HW	700	10% MUCK	40 - 80	15	20	51	2.6	7.2	IRREGULAR
Trout Lake	7251	FW	700	10% MUCK	60	15	20	51	2.6	7.2	REGULAR
Trout Lake	7010T	HW	700	10% MUCK	52	27	30	26	0.9	7	COMPLEX
Trout Lake	6810T	HW	670	35% FILL	65	37	22	15-27 (27)	1.2	6.1	REGULAR
Trout Lake	7451	HW	730	5% MUCK	45	2.5	17	35	2.1	5.7	REGULAR
Trout Lake	7451	FW	730	5% MUCK	45	2.5	17	35	2.1	5.7	REGULAR
Ruttan	740 - 5B	HW	715	35% MUCK	66	7	21.6	43	2	7.2	IRREGULAR
Ruttan	740 - 5B	FW	715	35% MUCK	73	7	21.6	43	2	7.2	IRREGULAR
Ruttan	630 - 13B	HW	625	25% FILL	75	22	22	69 (90)	4.1	8.8	IRREGULAR
Ruttan	630 - 13B	FW	625	25% FILL	75	22	22	69 (90)	4.1	8.8	COMPLEX
Ruttan	630 - 9B	HW	625	30% MUCK	75	14	22	50	2.3	7.6	IRREGULAR
Ruttan	630 - 9B	FW	625	30% MUCK	75	14	22	50	2.3	7.6	IRREGULAR
Ruttan	950 - 26J4	HW	925	45% FILL	90	26	28.2	40 (75)	2.7	10.2	REGULAR
Westmin	K381-B5	HW (N)	650	EMPTY	78	11	29	13	0.4	4.5	REGULAR
Westmin	K381-B5	FW (S)	650	EMPTY	90	11	29	13	0.4	4.5	REGULAR
Westmin	K381-B1	HW (W)	650	10% MUCK	90	21	53	20	0.4	7.3	REGULAR
Westmin	K381-B1	FW (E)	650	10% MUCK	90	21	53	20	0.4	7.3	REGULAR
Westmin	K381-B3	HW (W)	650	5% MUCK	90	18	45	19	0.4	6.7	REGULAR
Westmin	K381-B3	FW (E)	650	5% MUCK	90	18	45	18	0.4	6.4	REGULAR
Westmin	S410-D3	HW	410	EMPTY	40	9	18	16	0.9	4.2	REGULAR
Westmin	S351-B1	HW	470	EMPTY	55	3	20	16	0.8	4.4	REGULAR
Westmin	C390-2E	HW	650	EMPTY	35	19	14	45	3.2	5.2	REGULAR
Contact Lake	481-29 -July/95	HW	75	50% MUCK	60	7	60	60	1	15	IRREGULAR
Contact Lake	481-29 - July/95	FW	75	50% MUCK	60	7	60	60	1	15	IRREGULAR
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	60	7	60	124	2.1	20	IRREGULAR
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	60	7	60	124	2.1	20	IRREGULAR
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	80-85	2.6	14	66	4.7	5.8	REGULAR
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	80-85	2.6	14	66	4.7	5.8	REGULAR
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	73	5	6	20	3.3	2.5	REGULAR
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	73	5	6	20	3.3	2.5	REGULAR
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	85	1.8	13	66	5.1	5.4	REGULAR
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	85	1.8	13	66	5.1	5.4	REGULAR
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	85	1.7	13	22	1.7	4.1	REGULAR
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	85	1.7	13	22	1.7	4.1	REGULAR
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	77	4	11.25	28	2.5	4	REGULAR
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	77	4	11.25	28	2.5	4	REGULAR
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	85	2.2	9	66.5	7.4	4	REGULAR
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	85	2.2	9	66.5	7.4	4	REGULAR
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	85	1.8	13.5	22	1.6	4.2	REGULAR
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	85	1.8	13.5	22	1.6	4.2	REGULAR
Lupin	WZS 800-May2/96	HW	815	EMPTY	80-85	3	15	31.5	2.1	5.1	REGULAR
Lupin	WZS 800-May2/96	FW	815	EMPTY	80-85	3	15	31.5	2.1	5.1	REGULAR
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	80	2.3	13	44 (67)	5.2	5.4	REGULAR
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	80	2.3	13	44 (67)	5.2	5.4	REGULAR
Lupin	WZN 770-May10/96	HW	785	10% MUCK	89	1.6	9-19	22-34	3.8	5.6	REGULAR
Lupin	WZN 770-May10/96	FW	785	10% MUCK	89	1.6	9-19	22-34	3.8	5.6	REGULAR
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	71	3.5	25	21.25 (26)	1	6.4	REGULAR
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	71	3.5	25	21.25 (26)	1	6.4	REGULAR
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	77	4.75	19.5	12 (19.5)	1	4.9	REGULAR
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	77	4.75	19.5	12 (19.5)	1	4.9	REGULAR
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	85	2.5	23	35	1.5	6.9	REGULAR
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	85	2.5	23	35	1.5	6.9	REGULAR
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	83	1.7	15	45	3	5.6	IRREGULAR
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	83	1.7	15	45	3	5.6	IRREGULAR
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	75	4	18	25	1.4	5.2	REGULAR
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	75	4	18	25	1.4	5.2	REGULAR
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	80	2.2	27	23.5 (30)	1.1	7.1	REGULAR
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	80	2.2	27	23.5 (30)	1.1	7.1	REGULAR
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	84	1.8	20	18 (24)	1.2	5.5	REGULAR
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	84	1.8	20	18 (24)	1.2	5.5	REGULAR
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	85	1.4	26	28 (33)	1.3	7.3	REGULAR
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	85	1.4	26	28 (33)	1.3	7.3	REGULAR
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	83	1.8	27	22	0.81	6.1	REGULAR
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	83	1.8	27	22	0.81	6.1	REGULAR
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	83	1.9	18	56	3.1	6.8	REGULAR
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	83	1.9	18	56	3.1	6.8	REGULAR
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	80	3.5	16.5	43	2.6	6	REGULAR
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	80	3.5	16.5	43	2.6	6	REGULAR
Detour Lake	935 Q120 F6	HW	500	35% MUCK	85	3.5	5.5	32	5.8	2.3	REGULAR
Detour Lake	935 Q120 F6	FW	500	35% MUCK	85	3.5	5.5	32	5.8	2.3	REGULAR
Detour Lake	935 Q120 F7	HW	500	20% MUCK	85	3.5	18	36	2	6	REGULAR
Detour Lake	935 Q120 F7	FW	500	20% MUCK	85	3.5	18	36	2	6	REGULAR
Detour Lake	935 Q120 F8	HW	500	EMPTY	85	4	27	36	1.3	7.7	REGULAR
Detour Lake	935 Q120 F8	FW	500	EMPTY	85	4	27	36	1.3	7.7	REGULAR
Detour Lake	935 Q120 F11	HW	500	5% MUCK	85	4	40	36	0.9	9.5	REGULAR
Detour Lake	935 Q120 F11	FW	500	5% MUCK	85	4	40	36	0.9	9.5	REGULAR
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	60	7	45	18	0.4	6.4	REGULAR
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	65	7	45	18	0.4	6.4	REGULAR
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	60	17	45	24	0.5	7.8	REGULAR
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	85	17	45	24	0.5	7.8	REGULAR
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	75	7	45	41	0.9	10.7	IRREGULAR
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	75-80	7	45	41	0.9	10.7	IRREGULAR
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	75	7-17	45	85	1.9	14.7	IRREGULAR
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	75-80	7-17	45	85	1.9	14.7	IRREGULAR
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	75	7-17	45	75	1.7	14.1	IRREGULAR
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	75-80	7-17	45	75	1.7	14.1	IRREGULAR
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	70	7-17	98	58	0.6	18.5	IRREGULAR
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	75-80	7-17	98	58	0.6	18.5	IRREGULAR

GENERAL					ROCK MASS				UNDERCUTTING		
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	RMR	Q'	RMR'	STABILITY NUMBER (N')	AVERAGE UNDERCUT DEPTH (m)	% OF STRIKE LENGTH UNDERCUT	% OF WALL HEIGHT UNDERCUT
Trout Lake	7251	HW	700	10% MUCK	60	3.3	48	4.5	0.8	59%	50%
Trout Lake	7251	FW	700	10% MUCK	60	3.3	60	7.9	0	0	0
Trout Lake	7010T	HW	700	10% MUCK	60	3.3	45	3.8	10	33%	100%
Trout Lake	6810T	HW	670	35% FILL	60	3.3	50	5.4	2	52%	100%
Trout Lake	7451	HW	730	5% MUCK	65	5	47	5.6	1	41%	100%
Trout Lake	7451	FW	730	5% MUCK	65	5	65	12	1.5	76%	100%
Ruttan	740 - 5B	HW	715	35% MUCK	44	1	37	1.7	2.5	100%	100%
Ruttan	740 - 5B	FW	715	35% MUCK	75	30	75	72	2	100%	65%
Ruttan	630 - 13B	HW	625	25% FILL	44	1	39	1.9	2	32%	25%
Ruttan	630 - 13B	FW	625	25% FILL	75	30	75	72	0	0	0
Ruttan	630 - 9B	HW	625	30% MUCK	44	1	39	1.9	8	100%	40%
Ruttan	630 - 9B	FW	625	30% MUCK	75	30	75	72	0	0	0
Ruttan	950 - 26J4	HW	925	45% FILL	68	14.2	68	34	0.6	45%	33%
Westmin	K381-B5	HW (N)	650	EMPTY	69	9	63	18.2	1.5	29%	100%
Westmin	K381-B5	FW (S)	650	EMPTY	69	9	69	21.6	0	0	0
Westmin	K381-B1	HW (W)	650	10% MUCK	69	9	69	21.6	1.3	32%	100%
Westmin	K381-B1	FW (E)	650	10% MUCK	69	9	69	21.6	0.6	47%	100%
Westmin	K381-B3	HW (W)	650	5% MUCK	69	9	69	21.6	0.3	27%	100%
Westmin	K381-B3	FW (E)	650	5% MUCK	69	9	69	21.6	0	0	0
Westmin	S410-D3	HW	410	EMPTY	41	0.7	28	0.8	1.5	100%	100%
Westmin	S351-B1	HW	470	EMPTY	46	1.3	35	1.2	0.75	100%	100%
Westmin	C390-2E	HW	650	EMPTY	46	1.3	31	0.45	NOT AVAIL	NOT AVAIL	NOT AVAIL
Contact Lake	481-29 -July/95	HW	75	50% MUCK	70	15	56	22.5	2.1	100%	30%
Contact Lake	481-29 - July/95	FW	75	50% MUCK	70	15	70	36	0.9	30%	30%
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	70	15	56	22.5	1.8	100%	50%
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	70	15	70	36	0.9	30%	30%
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	72	15	69	34	1.3	50%	75%
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	47	1	47	2.4	0.8	14%	75%
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	67	6.3	59	12	1.5	100%	100%
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	67	6.3	67	15	0	0	0
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	70	10	68	23	1	38%	60%
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	45	1	45	2.4	0.7	77%	30%
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	72	10	69	22	0.8	30%	100%
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	71	9.4	71	23	1.8	92%	100%
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	70	15	64	29	4	50%	100%
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	70	15	70	36	0	0	0
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	77	14.2	74	32	1	60%	60%
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	45	1	45	2.4	1	39%	100%
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	70	9.4	68	22	0	0	0
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	70	9.4	70	23	1	100%	100%
Lupin	WZS 800-May2/96	HW	815	EMPTY	75	15	72	32	0.5	100%	100%
Lupin	WZS 800-May2/96	FW	815	EMPTY	70	9.4	70	23	0	0	0
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	65	7	60	14.5	0.4	100%	35%
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	45	1	45	2.4	1.5	54%	100%
Lupin	WZN 770-May10/96	HW	785	10% MUCK	77	15	76	35	0.5	100%	30%
Lupin	WZN 770-May10/96	FW	785	10% MUCK	70	9.4	70	23	1	53%	100%
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	70	14.2	61	26	0.7	60%	100%
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	70	14.2	70	34	1	48%	100%
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	75	14.2	68	28	2.5	46%	100%
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	65	7	65	17	0	0	0
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	75	10	72	22.4	0	0	0
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	70	9.4	70	23	0.75	15%	100%
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	65	4.7	62	10.5	1.5	80%	100%
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	70	9.4	70	23	0.75	60%	50%
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	50	1	45	2	2	70%	100%
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	65	5	65	12	0	0	0
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	75	15	70	31	1.25	65%	100%
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	75	15	75	36	0.5	75%	33%
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	70	15	67	33	0.4	75%	100%
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	70	15	70	36	0.9	70%	100%
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	70	10	68	22	0.75	75%	100%
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	70	10	70	24	0.75	90%	33%
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	75	15	71	33	1.5	37%	100%
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	65	9.4	65	23	0.75	52%	100%
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	75	15	71	33	0	0%	0%
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	65	5	65	12	1	56%	100%
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	75	15	70	31	0	0%	0%
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	75	15	75	36	1	55%	100%
Detour Lake	935 Q120 F6	HW	500	35% MUCK	75	33	72	74	4.6	100%	50%
Detour Lake	935 Q120 F6	FW	500	35% MUCK	68	15	68	36	0.4	100%	50%
Detour Lake	935 Q120 F7	HW	500	20% MUCK	75	33	72	74	3.3	100%	50%
Detour Lake	935 Q120 F7	FW	500	20% MUCK	68	15	68	36	0.4	32%	50%
Detour Lake	935 Q120 F8	HW	500	EMPTY	72	23	69	51	2.6	100%	50%
Detour Lake	935 Q120 F8	FW	500	EMPTY	72	23	72	55	0.8	39%	50%
Detour Lake	935 Q120 F11	HW	500	5% MUCK	72	23	69	49	1.9	100%	50%
Detour Lake	935 Q120 F11	FW	500	5% MUCK	68	15	68	36	1.2	45%	50%
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	70	19	56	19.2	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	68	14	68	22.4	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	72	23	58	22.5	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	67	13	67	20.8	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	68	15	61	19.3	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	69	16	69	25.6	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	69	16	62	21	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	68	15	68	24	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	69	16	62	21	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	68	14	68	22.4	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	69	16	59	19	NOT AVAIL	NOT AVAIL	NOT AVAIL
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	68	15	68	24	NOT AVAIL	NOT AVAIL	NOT AVAIL

GENERAL					DRILLING					
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	DRILL TYPE	BLASTHOLE DIAMETER (mm)	BLASTHOLE AVG. LENGTH (m)	DRILL STRING DIAM. (mm)	HOLE DIAM. STRING DIAM. RATIO	GUIDE ROD (S) (Y/N)
Trout Lake	7251	HW	700	10% MUCK	HTH-SIMBA 278	76	20	T45	1.7	Y
Trout Lake	7251	FW	700	10% MUCK	HTH-SIMBA 278	76	20	T45	1.7	Y
Trout Lake	7010T	HW	700	10% MUCK	HTH-SIMBA 278	76	17	T45	1.7	Y
Trout Lake	6810T	HW	670	35% FILL	HTH-SIMBA 278	76	18	T45	1.7	Y
Trout Lake	7451	HW	730	5% MUCK	HTH-SIMBA 278	76	23	T45	1.7	Y
Trout Lake	7451	FW	730	5% MUCK	HTH-SIMBA 278	76	23	T45	1.7	Y
Ruttan	740 - 5B	HW	715	35% MUCK	ITH-CUBEX	76-114	20-40	76 - 89	1.3	NA
Ruttan	740 - 5B	FW	715	35% MUCK	ITH-CUBEX	76-114	20-40	76 - 89	1.3	NA
Ruttan	630 - 13B	HW	625	25% FILL	ITH-CUBEX	76-120	15-45	76 - 89	1.3	NA
Ruttan	630 - 13B	FW	625	25% FILL	ITH-CUBEX	76-120	15-45	76 - 89	1.3	NA
Ruttan	630 - 9B	HW	625	30% MUCK	ITH-CUBEX	120	15-55	76 - 89	1.3	NA
Ruttan	630 - 9B	FW	625	30% MUCK	ITH-CUBEX	120	15-55	76 - 89	1.3	NA
Ruttan	950 - 26J4	HW	925	45% FILL	ITH-CUBEX	114	20-40	76 - 89	1.3	NA
Westmin	K381-B5	HW (N)	650	EMPTY	HTH-SOLO	64	9	R32	2	N
Westmin	K381-B5	FW (S)	650	EMPTY	HTH-SOLO	64	9	R32	2	N
Westmin	K381-B1	HW (W)	650	10% MUCK	HTH-SIMBA	89	18	T51	1.7	N
Westmin	K381-B1	FW (E)	650	10% MUCK	HTH-SIMBA	89	18	T51	1.7	N
Westmin	K381-B3	HW (W)	650	5% MUCK	HTH-SIMBA	89	14	T51	1.7	N
Westmin	K381-B3	FW (E)	650	5% MUCK	HTH-SIMBA	89	14	T51	1.7	N
Westmin	S410-D3	HW	410	EMPTY	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.
Westmin	S351-B1	HW	470	EMPTY	HTH-SOLO	64	12	R32	2	N
Westmin	C390-2E	HW	650	EMPTY	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL
Contact Lake	481-29 -July/95	HW	75	50% MUCK	ITH-CUBEX	100	15-35	76	1.3	NA
Contact Lake	481-29 - July/95	FW	75	50% MUCK	ITH-CUBEX	100	15-35	76	1.3	NA
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	ITH-CUBEX	100	15-35	76	1.3	NA
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	ITH-CUBEX	100	15-35	76	1.3	NA
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	HTH-SOLO	50	10	R32	1.6	Y
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	HTH-SOLO	50	10	R32	1.6	Y
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	HTH-SOLO	65	11	T38	1.7	Y
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	HTH-SOLO	65	11	T38	1.7	Y
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	HTH-SOLO	50	8.5	R32	1.6	Y
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	HTH-SOLO	50	8.5	R32	1.6	Y
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	HTH-SOLO	70	19	T38	1.8	Y
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	HTH-SOLO	70	19	T38	1.8	Y
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	HTH-SOLO	50	8.5	R32	1.6	Y
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	HTH-SOLO	50	8.5	R32	1.6	Y
Lupin	WZS 800-May2/96	HW	815	EMPTY	HTH-SOLO	50	8.2	R32	1.6	Y
Lupin	WZS 800-May2/96	FW	815	EMPTY	HTH-SOLO	50	8.2	R32	1.6	Y
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	HTH-SOLO	50	10	R32	1.6	Y
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	HTH-SOLO	50	10	R32	1.6	Y
Lupin	WZN 770-May10/96	HW	785	10% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	WZN 770-May10/96	FW	785	10% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	HTH-SOLO	70	18	T38	1.8	Y
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	HTH-SOLO	70	18	T38	1.8	Y
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	HTH-SOLO	65	11	T38	1.7	Y
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	HTH-SOLO	65	11	T38	1.7	Y
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	HTH-SOLO	50	11	R32	1.6	Y
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	HTH-SOLO	70	17.5	T38	1.8	Y
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	HTH-SOLO	70	17.5	T38	1.8	Y
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	HTH-SOLO	50	9.6	R32	1.6	Y
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	HTH-SOLO	50	9.6	R32	1.6	Y
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	HTH-SOLO	50	8	R32	1.6	Y
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	HTH-SOLO	50	8.6	R32	1.6	Y
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	HTH-SOLO	50	8.6	R32	1.6	Y
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	HTH-SOLO	50	9.9	R32	1.6	Y
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	HTH-SOLO	50	9.9	R32	1.6	Y
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	HTH-SOLO	65	10.9	T38	1.7	Y
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	HTH-SOLO	65	10.9	T38	1.7	Y
Detour Lake	935 Q120 F6	HW	500	35% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F6	FW	500	35% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F7	HW	500	20% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F7	FW	500	20% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F8	HW	500	EMPTY	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F8	FW	500	EMPTY	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F11	HW	500	5% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	935 Q120 F11	FW	500	5% MUCK	HTH-STOPE MASTER	54	14.5	R32	1.7	Y
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	HTH-STOPE MASTER	54	15-20	R32	1.7	Y
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	HTH-STOPE MASTER	54	15-20	R32	1.7	Y

GENERAL					BLASTING							
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	BURDEN (m)	SPACING (m)	SPACING BURDEN RATIO	OFFSET DISTANCE (m)	EXPLOSIVE	POWDER FACTOR (Kg/Tonne)	BLASTHOLE LAYOUT ADJACENT TO CONTACT	CONTROLLED BLASTING ADJACENT TO CONTACT (Y/N)
Trout Lake	7251	HW	700	10% MUCK	2	2.5	1.25	1	EMULSION	0.3	FAN/PARALLEL	Y - IRESPLIT
Trout Lake	7251	FW	700	10% MUCK	2	2.5	1.25	0	EMULSION	0.3	FAN/PARALLEL	N
Trout Lake	7010T	HW	700	10% MUCK	2.3	3	1.3	0.5	EMULSION	0.25	FAN/PARALLEL	N
Trout Lake	6810T	HW	670	35% FILL	2.4	3	1.3	0.9	EMULSION	0.25	PARALLEL	N
Trout Lake	7451	HW	730	5% MUCK	2	1.4 -2.8	0.7 - 1.4	0	ANFO	0.45	PARALLEL	N
Trout Lake	7451	FW	730	5% MUCK	2	1.4 -2.8	0.7 - 1.4	0	ANFO	0.45	PARALLEL	N
Ruttan	740 - 5B	HW	715	35% MUCK	2.4	2.8	1.2	0	ANFO	0.6	PARALLEL	N
Ruttan	740 - 5B	FW	715	35% MUCK	2.4	2.8	1.2	0	ANFO	0.6	PARALLEL	N
Ruttan	630 - 13B	HW	625	25% FILL	2.4	2.3	0.96	0	ANFO	0.4	FAN/PARALLEL	N
Ruttan	630 - 13B	FW	625	25% FILL	2.4	2.3	0.96	0	ANFO	0.4	FAN/PARALLEL	N
Ruttan	630 - 9B	HW	625	30% MUCK	2.4	2.5 - 3	1.04 - 1.25	0	ANFO	0.55	FAN/PARALLEL	N
Ruttan	630 - 9B	FW	625	30% MUCK	2.4	2.5 - 3	1.04 - 1.25	0	ANFO	0.55	PARALLEL	N
Ruttan	950 - 26J4	HW	925	45% FILL	2.4 - 2.7	2.5 - 3.4	0.9 - 1.4	0	ANFO	0.4	PARALLEL	Y - LOMEX
Westmin	K381-B5	HW (N)	650	EMPTY	1.8	2.3	1.3	0	ANFO	0.39	FAN	N
Westmin	K381-B5	FW (S)	650	EMPTY	1.8	2.3	1.3	0	ANFO	0.39	PARALLEL	N
Westmin	K381-B1	HW (W)	650	10% MUCK	2.5	3.2	1.3	0.6	ANFO	0.31	PARALLEL	N
Westmin	K381-B1	FW (E)	650	10% MUCK	2.5	3.2	1.3	0.6	ANFO	0.31	PARALLEL	N
Westmin	K381-B3	HW (W)	650	5% MUCK	2.4	3.5	1.5	1.1	ANFO	0.32	PARALLEL	N
Westmin	K381-B3	FW (E)	650	5% MUCK	2.4	3.5	1.5	0.9	ANFO	0.32	FAN/PARALLEL	N
Westmin	S410-D3	HW	410	EMPTY	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.	NOT AVAIL.
Westmin	S351-B1	HW	470	EMPTY	1.6	1.3	0.8	0	ANFO	0.7	FAN	N
Westmin	C390-2E	HW	650	EMPTY	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL
Contact Lake	481-29 -July/95	HW	75	50% MUCK	2	2	1	0.25	ANFO/EMUL	0.65	PARALLEL	N
Contact Lake	481-29 - July/95	FW	75	50% MUCK	2	2	1	0.25	ANFO/EMUL	0.65	PARALLEL	N
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	2	2	1	0.25	ANFO/EMUL	0.65	PARALLEL	N
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	2	2	1	0.25	ANFO/EMUL	0.65	PARALLEL	N
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	0.75	0.75	1	0.6	ANFO	0.9	PARALLEL	N
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	0.75	0.75	1	0.3	ANFO	0.9	PARALLEL	N
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	0.75	0.75 - 1.5	1-2	0.4	ANFO	0.5	PARALLEL	Y - LOMEX
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	0.75	0.75 - 1.5	1-2	0.4	ANFO	0.5	PARALLEL	Y - LOMEX
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	0.75	0.4 - 0.8	0.7 - 1.1	0.5	ANFO	1.2	PARALLEL	N
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	0.75	0.4 - 0.8	0.7 - 1.1	0.1	ANFO	1.2	PARALLEL	N
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	0.75	0.7	0.9	0.1	ANFO	1.1	PARALLEL	N
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	0.75	0.7	0.9	0.1	ANFO	1.1	PARALLEL	N
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	0.75	1.2 - 1.5	1.6 - 2	0.5	ANFO	0.58	PARALLEL	Y - LOMEX
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	0.75	1.2 - 1.5	1.6 - 2	0.4 - 1.5	ANFO	0.58	PARALLEL	Y - LOMEX
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	0.75	0.7 - 0.8	0.9 - 1.1	0.3	ANFO	0.9	PARALLEL	N
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	0.75	0.7 - 0.8	0.9 - 1.1	0 - 0.3	ANFO	0.9	PARALLEL	N
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	0.75	0.6 - 0.7	0.8 - 0.9	0.1	ANFO	1.2	PARALLEL	N
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	0.75	0.6 - 0.7	0.8 - 0.9	0.1	ANFO	1.2	PARALLEL	N
Lupin	WZS 800-May2/96	HW	815	EMPTY	0.75	1 - 1.3	1.3 - 1.7	0.3	ANFO	0.63	PARALLEL	N
Lupin	WZS 800-May2/96	FW	815	EMPTY	0.75	1 - 1.3	1.3 - 1.7	0.4	ANFO	0.63	PARALLEL	N
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	0.75	0.7 - 0.8	0.9 - 1.1	0.3 - 0.7	ANFO	1	PARALLEL	N
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	0.75	0.7 - 0.8	0.9 - 1.1	0 - 0.3	ANFO	1	PARALLEL	N
Lupin	WZN 770-May10/96	HW	785	10% MUCK	0.75	0.7 - 0.8	0.9 - 1.1	0	ANFO	1.14	PARALLEL	N
Lupin	WZN 770-May10/96	FW	785	10% MUCK	0.75	0.7 - 0.8	0.9 - 1.1	0	ANFO	1.14	PARALLEL	N
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	0.75	1.1	1.5	0.6 - 1	ANFO	0.65	PARALLEL	Y - LOMEX
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	0.75	1.1	1.5	0.5	ANFO	0.65	PARALLEL	Y - LOMEX
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	1.5	1.4	0.9	0.5 - 1.5	ANFO	0.47	PARALLEL	N
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	1.5	1.4	0.9	0.4 - 0.5	ANFO	0.47	PARALLEL	N
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	0.75	0.7 - 1.1	0.9-1.5	0.3-0.5	ANFO	0.84	PARALLEL	N
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	0.75	0.7 - 1.1	0.9-1.5	0.3	ANFO	0.84	PARALLEL	N
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	0.75	0.7	0.9	0.3	ANFO	1.19	PARALLEL	N
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	0.75	0.7	0.9	0	ANFO	1.19	PARALLEL	N
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	0.75	1.4	1.9	0.5	ANFO	0.64	PARALLEL	Y - LOMEX
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	0.75	1.4	1.9	0.5	ANFO	0.64	PARALLEL	Y - LOMEX
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	0.75	0.8	1.1	0 - 0.3	ANFO	0.84	PARALLEL	N
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	0.75	0.8	1.1	0 - 0.3	ANFO	0.84	PARALLEL	N
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	0.75	0.6 - 1.0	0.8 - 1.3	0 - 0.3	ANFO	1.05	PARALLEL	N
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	0.75	0.6 - 1.0	0.8 - 1.3	0 - 0.3	ANFO	1.05	PARALLEL	N
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	0.75	0.7	0.9	0	ANFO	1.22	PARALLEL	N
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	0.75	0.7	0.9	0	ANFO	1.22	PARALLEL	N
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	0.75	0.7 - 1.1	0.9 - 1.5	0.3	ANFO	0.97	PARALLEL	N
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	0.75	0.7 - 1.1	0.9 - 1.5	0.2	ANFO	0.97	PARALLEL	N
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	0.75	0.6 - 0.9	1	0.5	ANFO	1	PARALLEL	N
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	0.75	0.6 - 0.9	1	0.2	ANFO	1	PARALLEL	N
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	0.75	1	1.3	0 - 1.0	ANFO	0.76	PARALLEL	N
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	0.75	1	1.3	0 - 0.3	ANFO	0.76	PARALLEL	N
Detour Lake	935 Q120 F6	HW	500	35% MUCK	1.5	1.2	0.8	0	ANFO	0.67	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F6	FW	500	35% MUCK	1.5	1.2	0.8	0	ANFO	0.67	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F7	HW	500	20% MUCK	1.5	1.2	0.8	0	ANFO	0.75	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F7	FW	500	20% MUCK	1.5	1.2	0.8	0	ANFO	0.75	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F8	HW	500	EMPTY	1.5	1.2	0.8	0	ANFO	0.69	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F8	FW	500	EMPTY	1.5	1.2	0.8	0	ANFO	0.69	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F11	HW	500	5% MUCK	1.5	1.2	0.8	0	ANFO	0.72	PARALLEL	Y - TRACED
Detour Lake	935 Q120 F11	FW	500	5% MUCK	1.5	1.2	0.8	0	ANFO	0.72	PARALLEL	Y - TRACED
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.62	PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL.	ANFO	0.62	PARALLEL	NOT AVAIL.
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.59	PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.59	FAN	NOT AVAIL
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	ANFO	0.7	FAN/PARALLEL	NOT AVAIL

GENERAL					STOPE SUPPORT						
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	SUPPORT PATTERN	SPACING (m)	LENGTH (m)	BOLT TYPE (S)	# OF CABLES PER HOLE	FIXTURES	W:C RATIO
Trout Lake	7251	HW	700	10% MUCK	POINT ANCHOR	3 every 2m	7.5	PLAIN	2	PLATES	0.4 -0.35
Trout Lake	7251	FW	700	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Trout Lake	7010T	HW	700	10% MUCK	POINT ANCHOR	3 every 2m	4.5 - 6	PLAIN	2	PLATES	0.4 -0.35
Trout Lake	6810T	HW	670	35% FILL	POINT ANCHOR	3 every 2m	4.5 - 6	PLAIN	2	PLATES	0.4 -0.35
Trout Lake	7451	HW	730	5% MUCK	POINT ANCHOR	3 every 2m	4.5	PLAIN	2	PLATES	0.4 -0.35
Trout Lake	7451	FW	730	5% MUCK	NA	NA	NA	NA	NA	NA	NA
Ruttan	740 - 5B	HW	715	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Ruttan	740 - 5B	FW	715	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Ruttan	630 - 13B	HW	625	25% FILL	NA	NA	NA	NA	NA	NA	NA
Ruttan	630 - 13B	FW	625	25% FILL	NA	NA	NA	NA	NA	NA	NA
Ruttan	630 - 9B	HW	625	30% MUCK	POINT ANCHOR	7 every 2.4m	NOT AVAIL	PLAIN	2	PLATES	NOT AVAIL
Ruttan	630 - 9B	FW	625	30% MUCK	NA	NA	NA	NA	NA	NA	NA
Ruttan	950 - 26J4	HW	925	45% FILL	POINT ANCHOR	3 every 2.4m	NOT AVAIL	PLAIN	2	PLATES	NOT AVAIL
Westmin	K381-B5	HW (N)	650	EMPTY	NA	NA	NA	NA	NA	NA	NA
Westmin	K381-B5	FW (S)	650	EMPTY	NA	NA	NA	NA	NA	NA	NA
Westmin	K381-B1	HW (W)	650	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Westmin	K381-B1	FW (E)	650	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Westmin	K381-B3	HW (W)	650	5% MUCK	NA	NA	NA	NA	NA	NA	NA
Westmin	K381-B3	FW (E)	650	5% MUCK	NA	NA	NA	NA	NA	NA	NA
Westmin	S410-D3	HW	410	EMPTY	NA	NA	NA	NA	NA	NA	NA
Westmin	S351-B1	HW	470	EMPTY	POINT ANCHOR	3 every 1.6m	9.5 - 15.5	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL
Westmin	C390-2E	HW	650	EMPTY	POINT ANCHOR	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL	NOT AVAIL
Contact Lake	481-29 -July/95	HW	75	50% MUCK	NA	NA	NA	NA	NA	NA	NA
Contact Lake	481-29 - July/95	FW	75	50% MUCK	NA	NA	NA	NA	NA	NA	NA
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	NA	NA	NA	NA	NA	NA	NA
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	POINT ANCHOR	3 every 2m	8	PLAIN	1	PLATES	NOT AVAIL
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	POINT ANCHOR	3 every 2m	8	PLAIN	1	PLATES	NOT AVAIL
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 800-May2/96	HW	815	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 800-May2/96	FW	815	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 770-May10/96	HW	785	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 770-May10/96	FW	785	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	POINT ANCHOR	3 every 2m	4	PLAIN	1	PLATES	NOT AVAIL
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	POINT ANCHOR	3 every 2m	4	PLAIN	1	PLATES	NOT AVAIL
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	NA	NA	NA	NA	NA	NA	NA
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F6	HW	500	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F6	FW	500	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F7	HW	500	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F7	FW	500	20% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F8	HW	500	EMPTY	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F8	FW	500	EMPTY	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F11	HW	500	5% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	935 Q120 F11	FW	500	5% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	NA	NA	NA	NA	NA	NA	NA
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	POINT ANCHOR	3 every 3m	8	PLAIN/FLARED	2	PLATES	NOT AVAIL
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	NA	NA	NA	NA	NA	NA	NA

GENERAL					TIME			PERFORMANCE	
MINE	STOPE	SURFACE	DEPTH (m)	% FULL OF MUCK OR FILL	TIME BETWEEN INITIAL BLAST AND CMS SURVEY	# OF BLASTS TO CREATE SURVEYED STOPE	IN VICINITY OF OTHER MINING (Y/N)	ELOS	ELLO
Trout Lake	7251	HW	700	10% MUCK	60 DAYS	5	Y	1.1	0.5
Trout Lake	7251	FW	700	10% MUCK	60 DAYS	5	Y	0.8	0
Trout Lake	7010T	HW	700	10% MUCK	30 DAYS	3	Y	1.9	0.1
Trout Lake	6810T	HW	670	35% FILL	30 DAYS	2	Y	1	0
Trout Lake	7451	HW	730	5% MUCK	28 DAYS	2	Y	0.4	0
Trout Lake	7451	FW	730	5% MUCK	28 DAYS	2	Y	0.4	0
Ruttan	740 - 5B	HW	715	35% MUCK	21 DAYS	APPROX. 14	Y	2.8	0
Ruttan	740 - 5B	FW	715	35% MUCK	21 DAYS	APPROX. 14	Y	0.2	0
Ruttan	630 - 13B	HW	625	25% FILL	250 DAYS	APPROX. 14	Y	4.3	0
Ruttan	630 - 13B	FW	625	25% FILL	250 DAYS	APPROX. 14	Y	0.3	0.2
Ruttan	630 - 9B	HW	625	30% MUCK	120 DAYS	APPROX. 14	Y	5.2	0
Ruttan	630 - 9B	FW	625	30% MUCK	120 DAYS	APPROX. 14	Y	0.2	0
Ruttan	950 - 26J4	HW	925	45% FILL	210 DAYS	APPROX. 14	Y	0	0
Westmin	K381-B5	HW (N)	650	EMPTY	94 DAYS	2	Y	0.2	0
Westmin	K381-B5	FW (S)	650	EMPTY	94 DAYS	2	Y	0.1	0
Westmin	K381-B1	HW (W)	650	10% MUCK	124 DAYS	5	Y	0.1	0.1
Westmin	K381-B1	FW (E)	650	10% MUCK	124 DAYS	5	Y	0	0.5
Westmin	K381-B3	HW (W)	650	5% MUCK	51 DAYS	3	Y	0.01	0.7
Westmin	K381-B3	FW (E)	650	5% MUCK	51 DAYS	3	Y	0.01	0.5
Westmin	S410-D3	HW	410	EMPTY	NOT AVAIL	NOT AVAIL	NOT AVAIL	3.1	0
Westmin	S351-B1	HW	470	EMPTY	30 DAYS	2	NOT AVAIL	2.9	0
Westmin	C390-2E	HW	650	EMPTY	NOT AVAIL	NOT AVAIL	Y	6	0
Contact Lake	481-29 -July/95	HW	75	50% MUCK	180 DAYS	>20	N	1.7	0
Contact Lake	481-29 - July/95	FW	75	50% MUCK	180 DAYS	>20	N	0.4	0
Contact Lake	481-29 - Nov/95	HW	75	EMPTY	300 DAYS	>20	N	1.2	0
Contact Lake	481-29 - Nov/95	FW	75	EMPTY	300 DAYS	>20	N	0.6	0.2
Lupin	WZS 1010 S7-Jan15/96	HW	1050	EMPTY	26 DAYS	7	Y	0	0.7
Lupin	WZS 1010 S7-Jan15/96	FW	1050	EMPTY	26 DAYS	7	Y	0.6	0
Lupin	CZ 905-930-Jan.17/96	HW	920	15% MUCK	2 DAYS	2	Y	0.1	0.1
Lupin	CZ 905-930-Jan.17/96	FW	920	15% MUCK	2 DAYS	2	Y	0.3	0
Lupin	WZN 1010 S2- Feb15/96	HW	1050	EMPTY	49 DAYS	9	Y	0	0.2
Lupin	WZN 1010 S2- Feb15/96	FW	1050	EMPTY	49 DAYS	9	Y	1	0
Lupin	WZN 750-Feb24/96	HW	760	EMPTY	14 DAYS	4	N	0	0.1
Lupin	WZN 750-Feb24/96	FW	760	EMPTY	14 DAYS	4	N	0.1	0.1
Lupin	CZ 1010 S5-Feb24/96	HW	1020	EMPTY	35 DAYS	3	Y	0	0.3
Lupin	CZ 1010 S5-Feb24/96	FW	1020	EMPTY	35 DAYS	3	Y	0	0.4
Lupin	WZS 1010 S7-Mar22/96	HW	1050	EMPTY	26 DAYS	8	Y	0	0.5
Lupin	WZS 1010 S7-Mar22/96	FW	1050	EMPTY	26 DAYS	8	Y	1.8	0
Lupin	WZN 750-Mar29/96	HW	760	EMPTY	19 DAYS	4	Y	0	0.1
Lupin	WZN 750-Mar29/96	FW	760	EMPTY	19 DAYS	4	Y	0	0.1
Lupin	WZS 800-May2/96	HW	815	EMPTY	21 DAYS	3	N	0	0.1
Lupin	WZS 800-May2/96	FW	815	EMPTY	21 DAYS	3	N	0	0.5
Lupin	WZS 1010 S7-May9/96	HW	1050	35% MUCK	16 DAYS	8	Y	0.4	0
Lupin	WZS 1010 S7-May9/96	FW	1050	35% MUCK	16 DAYS	8	Y	1	0
Lupin	WZN 770-May10/96	HW	785	10% MUCK	37 DAYS	7	N	0.1	0
Lupin	WZN 770-May10/96	FW	785	10% MUCK	37 DAYS	7	N	0.1	0
Lupin	CZ 990-May19/96	HW	1000	20% MUCK	6 DAYS	4	Y	0	0.7
Lupin	CZ 990-May19/96	FW	1000	20% MUCK	6 DAYS	4	Y	0	0.1
Lupin	CZ 890-May 19/96	HW	900	40% MUCK	17 DAYS	5	Y	0.1	0.5
Lupin	CZ 890-May 19/96	FW	900	40% MUCK	17 DAYS	5	Y	0.5	0
Lupin	WZN 750-Jun15/96	HW	760	EMPTY	17 DAYS	7	N	0	0.2
Lupin	WZN 750-Jun15/96	FW	760	EMPTY	17 DAYS	7	N	0	0.2
Lupin	WZN 1050-Jun18/96	HW	1050	EMPTY	30 DAYS	7	Y	1	0
Lupin	WZN 1050-Jun18/96	FW	1050	EMPTY	30 DAYS	7	Y	0.4	0
Lupin	CZ 990-Jun22/96	HW	1000	10% MUCK	7 DAYS	3	Y	0.8	0
Lupin	CZ 990-Jun22/96	FW	1000	10% MUCK	7 DAYS	3	Y	0.1	0.2
Lupin	WZS 780-Jul30/96	HW	780	20% MUCK	21 DAYS	9	Y	0.1	0.1
Lupin	WZS 780-Jul30/96	FW	780	20% MUCK	21 DAYS	9	Y	0.1	0.1
Lupin	WZS 1070-Aug7/96	HW	1080	25% FILL	24 DAYS	8	Y	0.1	0.1
Lupin	WZS 1070-Aug7/96	FW	1080	25% FILL	24 DAYS	8	Y	0.2	0
Lupin	WZN 800-Aug14/96	HW	815	15% MUCK	20 DAYS	8	Y	0.2	0
Lupin	WZN 800-Aug14/96	FW	815	15% MUCK	20 DAYS	8	Y	0.2	0
Lupin	WZS 750-Sept2/96	HW	760	EMPTY	19 DAYS	8	Y	0.1	0.5
Lupin	WZS 750-Sept2/96	FW	760	EMPTY	19 DAYS	8	Y	0.4	0.1
Lupin	WZS 1050-Sept19/96	HW	1060	EMPTY	33 DAYS	12	Y	0	0.2
Lupin	WZS 1050-Sept19/96	FW	1060	EMPTY	33 DAYS	12	Y	0.3	0
Lupin	CZ 890-Oct8/96	HW	910	EMPTY	16 DAYS	4	Y	0.2	0.7
Lupin	CZ 890-Oct8/96	FW	910	EMPTY	16 DAYS	4	Y	0.5	0.2
Detour Lake	935 Q120 F6	HW	500	35% MUCK	8 DAYS	7	N	0.0	0
Detour Lake	935 Q120 F6	FW	500	35% MUCK	8 DAYS	7	N	0.1	0
Detour Lake	935 Q120 F7	HW	500	20% MUCK	9 DAYS	8	N	0.1	0
Detour Lake	935 Q120 F7	FW	500	20% MUCK	9 DAYS	8	N	0.2	0
Detour Lake	935 Q120 F8	HW	500	EMPTY	14 DAYS	9	N	0.0	0
Detour Lake	935 Q120 F8	FW	500	EMPTY	14 DAYS	9	N	0.2	0
Detour Lake	935 Q120 F11	HW	500	5% MUCK	25 DAYS	12	N	0.5	0
Detour Lake	935 Q120 F11	FW	500	5% MUCK	25 DAYS	12	N	0.2	0.1
Detour Lake	885-PANELB-CB	HW	610	50% MUCK	7 (40) DAYS	NOT AVAIL	Y	0.4	NOT AVAIL
Detour Lake	885-PANELB-CB	FW	610	50% MUCK	7 (40) DAYS	NOT AVAIL	Y	0.2	NOT AVAIL
Detour Lake	885-PANELB-CE	HW	610	70% MUCK	3 (140) DAYS	NOT AVAIL	Y	0.2	NOT AVAIL
Detour Lake	885-PANELB-CE	FW	610	70% MUCK	3 (140) DAYS	NOT AVAIL	Y	0.4	NOT AVAIL
Detour Lake	885-PANELB-CG	HW	610	60% MUCK	10 (187) DAYS	NOT AVAIL	Y	0.4	NOT AVAIL
Detour Lake	885-PANELB-CG	FW	610	60% MUCK	10 (187) DAYS	NOT AVAIL	Y	0.2	NOT AVAIL
Detour Lake	885-PANELB-CH	HW	610	45% MUCK	19 (222) DAYS	NOT AVAIL	Y	0.5	NOT AVAIL
Detour Lake	885-PANELB-CH	FW	610	45% MUCK	19 (222) DAYS	NOT AVAIL	Y	0.3	NOT AVAIL
Detour Lake	885-PANELB-CI	HW	610	35% MUCK	33 (236) DAYS	NOT AVAIL	Y	0.5	NOT AVAIL
Detour Lake	885-PANELB-CI	FW	610	35% MUCK	33 (236) DAYS	NOT AVAIL	Y	0.5	NOT AVAIL
Detour Lake	885-PANELB-CJ	HW	610	60% SL/MU	58 (261) DAYS	NOT AVAIL	Y	7.1	NOT AVAIL
Detour Lake	885-PANELB-CJ	FW	610	60% SL/MU	58 (261) DAYS	NOT AVAIL	Y	0.6	NOT AVAIL

NOTES:

- footwall N' values were calculated assuming C=8
- bracketed numbers in the "Exposed Height (m)" column refer to the true stope height
- "Surface Character" classification is after Germain et.al. (1996)
- "Average Undercut Depth (m)" is not averaged over the entire stope strike, it corresponds to the "Percentage of Strike Length Undercut"
- bracketed numbers in the "Time Between Initial Blast and CMS Survey" column refer to the total time the stope has been in production. The unbracketed numbers pertain to the wall exposure time for what could actually be measured with the CMS.

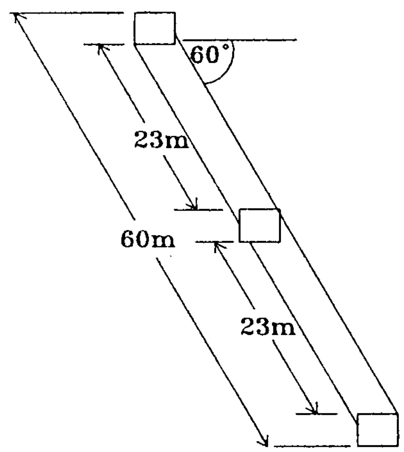
APPENDIX II

EXAMPLE PROBLEMS

EXAMPLE 1

A new mine is being developed which is to use open stoping as the primary mining method. To meet certain economic criteria, dilution during open stoping must be kept below 20%. Given the data shown in Figure 1, comment on whether this is achievable for stopes with heights of 60m and strike lengths of 40m.

TYPICAL SECTION



Strike Length = 40m
Ore width = 5m
RMR of HW and FW = 70-75
Ore = 3 tonnes/m3
Wall Rocks = 2.7 tonnes/m3 (no grade)
Ore development = 4m high x 5m wide
Blasthole diameter = 3.5 inches
Blasthole length = 23m (breakthrough holes)
Parallel blastholes

Parallel structure (foliation planes) in HW and FW
HW and FW in low stress state

Figure II-1 Example 1 - known data

Solution

1) Calculate: Hangingwall and Footwall RMR'

HW RMR' = (1- 0.4cos60°) (70 to 75) = 56 - 60

FW RMR' = no adjustment = 70 - 75

2) Calculate: Hydraulic Radius

HR = Surface Area / Perimeter = 60m x 40m / 60m + 60m + 40m + 40m = 12

3) Estimate Hangingwall and Footwall ELOS

Referring to Figure 2 shown below, the estimated ELOS is: HW ELOS = 1 - 2m; FW ELOS = 0 - 0.5m (Blast Damage Only). Note that this assumes unsupported stope surfaces.

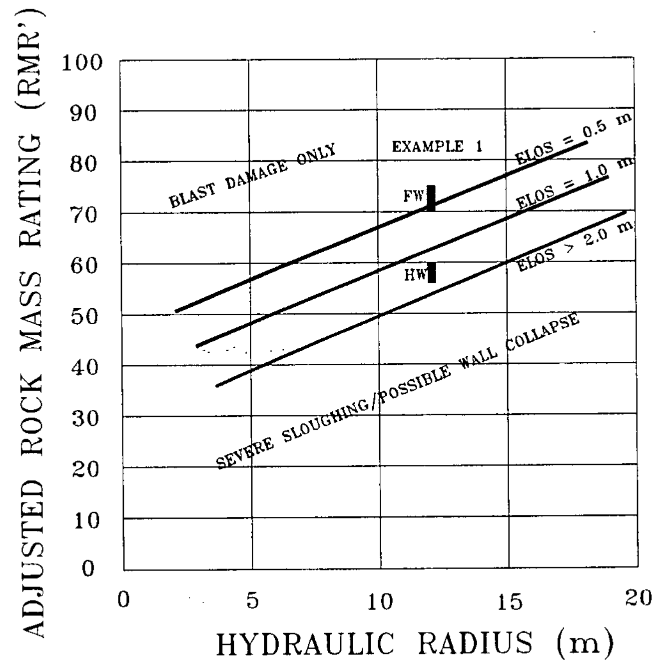


Figure II-2 Example 1 - estimation of HW and FW ELOS

4) Calculate Unplanned Stoping Dilution

Dilution = (Tonnes Waste / Tonnes Ore) x 100
= ((HW Waste + FW Waste) / (Total Ore - Develop. Ore)) x 100

Dilution = ((6480 to 12960 tonnes) + (0 to 3240 tonnes) / (36000 - 7200 tonnes)) x 100

Unplanned Stoping Dilution Estimate = 23 % to 56%

5) Comments

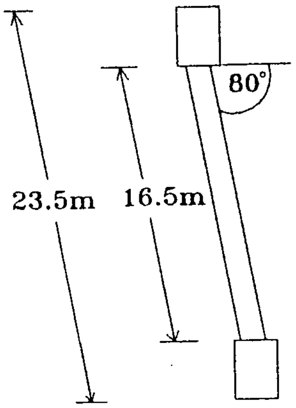
The above analysis suggests that if the stope surfaces are not supported, achieving less than 20% dilution is unlikely. Furthermore, given the blasthole diameter and length, ELOS values may be slightly underestimated, refer to Chapter 7. The relatively large sub-level interval also results in reduced information regarding the location of ore contacts and increases the probability of incurring some planned dilution. Given these considerations, dilution should be anticipated to be on the high side of the estimate shown.

There is potential to limit unplanned dilution by cable bolting the hangingwall. Sub-level cable support (point anchor approach) may be adequate, refer to Nickson (1992). It should be noted however, that even with cable bolts, achieving less than 20% dilution may be optimistic. If 0.5m ELOS from both the hangingwall and footwall is realized (i.e. blast damage), the resulting unplanned dilution is still 23%. In this situation, a reduction in stope size and/or a reduction in sub-level spacing (to achieve better control on drilling and blasting and more information regarding ore contacts) should be considered.

EXAMPLE 2

For a new narrow vein gold deposit the following typical stope size has been proposed: stope height 23.5m high; and strike length 20m. Achieving less than 25% dilution during open stoping is critical for economic success. Given the information presented in Figure 3 below, comment on the likelihood of meeting the dilution criteria.

TYPICAL SECTION



Strike Length = 20m
Ore Width = 1.5m
Q' of HW and FW = 15
Ore = 3.5 tonnes/m3
Wall rocks = 2.7 tonnes/m3 (no grade)
Ore development = 3.5m high x 2.5m wide
Blasthole diameter = 2 inches
Blasthole length = 16.5m (breakthrough holes)
Parallel blastholes

Parallel structure (bedding) in HW and FW
HW and FW in low stress state

Figure II-3 Example 2 - known data

Solution

1) Calculate N' for the Hangingwall and Footwall

HW - N' = Q' x A x B x C = 15 x 1 x 0.3 x (8 - 6cos80°) = 31

FW - N' = Q' x A x B x C = 15 x 1 x 0.3 x 8 = 36

2) Calculate: Hydraulic Radius

HR = Surface Area / Perimeter = 23.5m x 20m / 23.5m + 23.5m + 20m + 20m = 5.4

3) Estimate Hangingwall and Footwall ELOS

Referring to Figure 4 shown below, the estimated ELOS is: HW ELOS = 0 - 0.5m (Blast Damage Only); FW ELOS = 0 - 0.5m (Blast Damage Only).

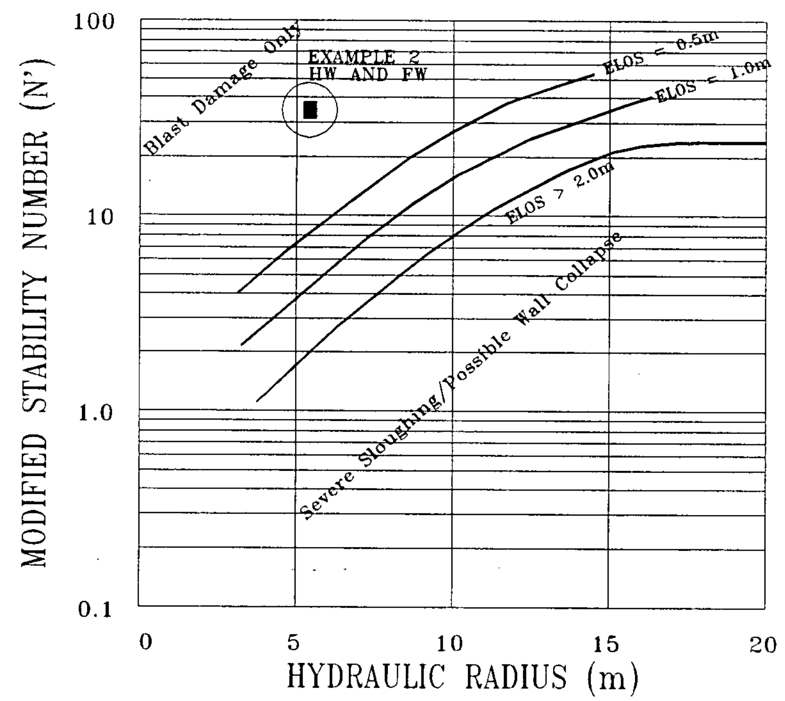


Figure II-4 Example 2 - estimation of HW and FW ELOS

4) Calculate Unplanned Stoping Dilution

Dilution = (Tonnes Waste / Tonnes Ore) x 100
= ((HW Waste + FW Waste) / (Total Ore - Develop. Ore)) x 100

Dilution = ((0 to 635 tonnes) + (0 to 635 tonnes) / (1733 tonnes)) x 100

Unplanned Stoping Dilution Estimate = 0% to 73%

5) <u>Comments</u>

The above analysis shows that the stopes are sized correctly (i.e. plot in "Blast Damage Only" zone) however due to the narrowness of the ore, unplanned dilution may still vary between 0% to 73% depending on the quality of the drilling and blasting. In this case, relatively long 2 inch diameter blastholes are being used. The breakthrough locations should be surveyed to verify drilling accuracy. When longhole blasting, the charge weight per delay should be minimized (i.e. one blasthole per delay).

A possible other factor which could effect wall stability is the undercutting of the stope walls during ore drift development (undercut depth approx. 0.5m). In this particular case, however, both the hangingwall and footwall rocks are good quality and thus there is a good chance the undercutting may not effect wall stability significantly. The CMS database provides some support for this statement, refer to Chapter 7.

APPENDIX III

BLAST MONITORING RESULTS:

3:2 PATTERN
1:1 (STAGGER) PATTERN
2:1 (DICE-FIVE) PATTERN

BLAST MONITORING RESULTS - 3:2 PATTERN - 50mm BLASTHOLES

UBC\CANMET 1996/97 - 3:2 PATTERN MONITORING RESULTS

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC. / BUR. RATIO	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	**SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
AUG.14/96	WZS 770 UPHOLES R624-627 (VERTICAL UPHOLES)	33	1.9	1	8.4	20	ANFO	GELDYNE SP-CEN	EXEL-MS	A	187	37.5	309	45	15	AT LEAST 3 MISFIRES: 2 EASERS: 1 ON 3 HOLE RING BIMODAL MUCK SIZE: FINE AND COARSE
AUG.18/96	WZS 750 DOWNHOLES R625-627	40	1.9	0.98	8.2	15	ANFO	GELDYNE SP-TOE	EXEL-MS	A	81	36.5	125	43.8	7	AT LEAST 1 MISFIRE: 1 ON 3 HOLE RING OBSERVED SOME HOLES MAKING WATER BEFORE BLAST OBSERVED 1 BOOTLEG IN FW AFTER BLAST
AUG.22/96	WZS 1050 UP&DWN R609-610 (UPHOLES DUMPED 70 DEG.)	27	2	0.93	9.2 - 11	19	ANFO	GELDYNE DP-COL&TOE	EXEL-MS	A	157	46 - 78	198	55 - 94	11	AT LEAST 2 MISFIRES: 1 EASER: 1 ON 3 HOLE RING OBSERVED SOME HOLES MAKING WATER BEFORE BLAST BIMODAL MUCK SIZE: FINE AND COARSE
SEPT.26/96	WZN 1050 DOWNHOLES R262-267	42	1.8	0.8	9.6	25	ANFO	GELDYNE DP-COL&TOE	EXEL-MS	A	NA	44	NA	52	12	AT LEAST 3 MISFIRES: ALL ON 3 HOLE RINGS VIBRATIONS ANOMALOUSLY LOW DUE TO POOR GROUND
AUG.12/96	WZN 830 UPHOLES R274-276 (VERTICAL UPHOLES)	33	1.5	0.93	9	14	ANFO	GELDYNE SP-TOE	EXEL-MS	B	68	25.3	88	32.5	14	AT LEAST 2 MISFIRES: BOTH ON 3 HOLE RING BIMODAL MUCK SIZE - FINE AND COARSE
AUG.13/96	WZS 1110 DOWNHOLES R232-228	28	2.2	1.15	9.5	23	GELDYNE (40mm)	GELDYNE SP-TOE	EXEL-MS	B	87	21.4	180	26	13	AT LEAST 3 MISFIRES: 1 EASER: 2 ON 3 HOLE RINGS SMALL BENCH LEFT FROM PREV.BLAST - ABLE TO LOAD BLAST BENCHED - COLLARS AND TOES BROKE 1-1.5m ALONG ENTIRE STRIKE LENGTH DID NOT BREAK
AUG.23/96	WZS 770 UPHOLES R636-640 (VERTICAL UPHOLES)	35	2.7	1.33	8.6	25	ANFO	GELDYNE DP-COL&TOE	HANDI-DETS 100/3800	A-HD	21	15.4	32	15.4	12	AT LEAST 3 MISFIRES: ALL ON 3 HOLE RINGS COLLARS BENCHED OVER MOST OF BLASTED STRIKE LENGTH OBSERVED SOME HOLES MAKING WATER BEFORE BLAST TWO DETONATORS PUT IN EACH HOLE (COLLAR AND TOE) BUT ONLY ONE PRIMER (COLLAR) ANFO HAD LOST COLOR AND WAS DUSTY
SEPT.30/96	WZN 770 UPHOLES RJ16-J17 (VERTICAL UPHOLES)	40	2.8	1.33	8.6	10	ANFO	GELDYNE SP-CEN	HANDI-DETS 100/3800	B-HD	21	15.4	31	15.4	10	ONE MISFIRE ON A 3 HOLE RING MUCK COARSER THAN OBSERVED WITH 25MS DELAYS
OCT.4/96	WZN 770 UPHOLES RJ19-J23 (VERTICAL UPHOLES)	35	2.6	1.33	8.6	23	ANFO	GELDYNE DP-COL&TOE	HANDI-DETS 100/3800	B-HD	37	15.4	64	15.4	22	5 MISFIRES: 4 EASERS: 1 ON A 3 HOLE RING COLLARS BENCHED OVER ENTIRE BLASTED STRIKE LENGTH HIGH COLLARS WERE NOTED BEFORE BLAST
NOV.27/96	WZN 1030 UPHOLES R266-270 (DUMPED UPHOLES 70 DEG.)	33	1.9	1	11.2	24	ANFO	GELDYNE DP-COL&TOE	HANDI-DETS 100/3800	C-HD	43	22.3	52	22.3	38	9 MISFIRES: 5 EASERS: 4 ON 3 HOLE RINGS BIMODAL MUCK SIZE - IN GENERAL GOOD TO COARSE
NOV.29/96	WZN 1110 DOWNHOLES R202-208	30	1.5	0.8	8.9	32	ANFO	GELDYNE SP-CEN	HANDI-DETS 100/3800	C-HD	NA	16	NA	16	25	8 MISFIRES: 7 EASERS : 1 ON 3HOLE RING BLAST BENCHED AT R207: COLLARS AND TOES BROKE POOR DRILLING: BURDENS 1m - 0.3m: SPACING 0.5-0.7m MONITORED FROM WZ ACCESS DRIFT
JUN.4/97	WZS 1050 UPHOLES R529-531 (VERTICAL UPHOLES)	28	1.7	0.8	9.7	15	ANFO	90g TROJAN DP-COL&TOE	EXCEL-MS	D	NA	22.1	NA	35.3	13	2 MISFIRES: BOTH ON 3 HOLE RINGS ON PREV. BLAST UPHOLES BENCHED 5m ANFO "DUSTY" - BLOWBACK AND PACKING PROBLEMS BLAST BROKE WELL BUT ONLY BROKE TO HEIGHT OF PREVIOUS BENCH.
JUN.9/97	WZN 890 UPHOLES R299-300 (VERTICAL UPHOLES)	20	1.6	1.1	7	10	ANFO	90g TROJAN DP-COL&TOE	EXCEL-MS	D	109	17.4	178	24	0	NO MISFIRES BLAST BROKE WELL

BLAST MONITORING RESULTS - 3:2 PATTERN - 50mm BLASTHOLES

ICI\LUPIN 1997 - 3:2 PATTERN MONITORING RESULTS

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC./BUR. RATIO	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	**SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
APR./97	CZ 1180 UPHOLES R96-99 (VERTICAL UPHOLES)	17m 43m	2	1.1	10.7	18	ANFO	90g TROJAN DP-COL&TOE	EXCEL-MS	D	NA	23.6	NA	39.4	0	NO MISFIRES BLAST BROKE WELL 9 VOD'S TESTED DURING BLAST - AVG.=3260m/s

GOLDER ASSOCIATES LTD./CANMET 1989/90 - 3:2 PATTERN MONITORING RESULTS

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC./BUR. RATIO	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	**SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
FEB.16/89	WZ 130 DOWNHOLES	NA	2	1.33	9	35	ANFO	CILGEL 70% SP-CEN	CXA MS DELAYS	A	NA	40	NA	48	9	3 MISFIRES: 2 EASERS: 1 ON 3 HOLE RING BACK BREAK OF APPROX. 1/3 BURDEN WALL DAMAGE
FEB.17/89	WZ 130 DOWNHOLES	10	2	1.33	9	27	ANFO	CILGEL 70% SP-CEN	CXA MS DELAYS	A	328	40	450	48	19	5 MISFIRES: 3 EASERS: 2 ON 3HOLE RINGS BACK BREAK OF APPROX. 1/3 BURDEN WALL DAMAGE ENERGY LEVELS > 900mm/s AT 3m DISTANCE
MAR.4/89	WZ 130 DOWNHOLES	NA	2	1.33	9	18	ANFO	CILGEL 70% SP-CEN	CXA MS DELAYS	A	NA	40	NA	48	17	3 MISFIRES: 2 EASERS: 1 ON 3HOLE RING BACK BREAK OF APPROX. 1/3 BURDEN WALL DAMAGE

NOTES:

PRIMING: SP-CEN (SINGLE PRIMED AT CENTER OF EXPLOSIVE COLUMN): SP-TOE (SINGLE PRIMED AT TOE OF EXPLOSIVE COLUMN): DP-COL&TOE (DOUBLE PRIMED AT COLLAR AND TOE OF EXPLOSIVE COLUMN)

SEQUENCING:

A = EACH RING OF BLASTHOLES ON SAME DELAY NUMBER (MS DELAYS)
B = CENTER HOLE ON 3 HOLE RING IS ON SEPARATE DELAY (MS DELAYS)
A-HD = EACH RING OF BLASTHOLES ON SAME DELAY NUMBER (100/3800 HANDI-DETS)
B-HD = CENTER HOLE ON 3 HOLE RING IS ON SEPARATE DELAY (100/3800 HANDI-DETS)
C-HD = EACH HOLE IS ON SEPARATE DELAY (100/3800 HANDI-DETS)
D = EACH HOLE ON 3 HOLE RING IS ON SEPERATE DELAY. BLASTHOLES ON 2 HOLE RING (EASER RING) ARE ON SAME DELAY (MS DELAYS)

BLAST MONITORING RESULTS - 2:1 PATTERN

UBC/CANMET 1997 - 2:1 PATTERN MONITORING RESULTS - 64mm DIAMETER BLASTHOLES

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC. & BUR.	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
SEPT.14/97	WZS 840 UP AND DOWNHOLES	23	2	0.6m BUR. 1.4m SPAC.	15m DWN 7.5m UP	5 DWN 4 UP	ANFO (PNEU. LOADED)	90g TROJAN DP-COL&TOE	EXCEL-MS	2-HOLE RINGS ON SAME DELAY #	93	77	120	111	0	FW RMR = 45; HW RMR = 70 COLLAR LOCATIONS ACCURATE SEQUENCED IMPROPERLY-SHOULD HAVE BEEN 1 HOLE/DELAY NO MISFIRES OVERBROKE 1.5m ON FOOTWALL STOPE SHUT DOWN DUE TO EXCESSIVE DILUTION
OCT.17/97	WZN 950 - R961-958 DOWNHOLES	39	1.6	0.6m BUR. 1.2m SPAC.	12.4	6	ANFO (PNEU. LOADED) & GELDYNE (WET HOLES)	90g TROJAN DP-COL&TOE	EXCEL-MS	1 HOLE/DELAY#	36	35	42	39	17	HW RMR = 70; FW RMR = 70 DRILLING MODERATE TO POOR BENCH WET - SOME WET HOLES LOADED WITH ANFO 1 MISFIRE - WET HOLE LOADED WITH ANFO - CENTER HOLE HW OVERBROKE APPROXIMATELY 0.5m
OCT.19/97	WZN 950 - R957-955 DOWNHOLES	37	1.6	0.6m BUR. 1.2m SPAC.	12.4	5	GELDYNE (WET HOLES)	DP-COL&TOE	EXCEL-MS	1 HOLE/DELAY#	41	29	46	29	20	HW RMR = 70; FW RMR = 70 DRILLING MODERATE TO POOR BENCH WET - HOLES LOADED WITH 50mmX400mm GELDYNE 1 MISFIRE - 2 HOLE RING - HOLE SPACING WAS 0.8m FRAGMENTATION - FINE/UNIFORM LITTLE TO NO OVERBREAK - WALLS LOOKED EXCELLENT

GOLDER ASSOCIATES LTD./CANMET 1989/90 - 2:1 PATTERN MONITORING RESULTS - 50mm DIAMETER BLASTHOLES

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC. & BUR.	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
MAR.5/89	WZ 130 DOWNHOLES	NA	2	BUR. = 0.75m SPAC. = 1.0 - 1.2m	9	10	ANFO	CILGEL 70% SP-CEN	CXA-MS DELAYS	2 HOLE RINGS ON SAME DELAY	NA	24	NA	32	0	NO MISFIRES OBSERVABLE BLAST DAMAGE
JULY 23/89	WZ 270 DOWNHOLES	16m	1.5-2.0	BUR. = 0.55m SPAC. = 1.1 - 1.4m	9	12	ANFO	CILGEL 70% SP-CEN	CXA-MS DELAYS	2 HOLE RINGS ON SAME DELAY	222	24	360	32	0	NO MISFIRES MIN. TO MOD. OBSERVABLE BLAST DAMAGE ENERGY LEVELS IN EXCESS OF 900mm/s 4m FROM BLAST
JULY 24/89	WZ 270 DOWNHOLES	NA	1.5-2.0	BUR. = 0.55m SPAC. = 1.1 - 1.4m	9	12	ANFO	CILGEL 70% SP-CEN	CXA-MS DELAYS	2 HOLE RINGS ON SAME DELAY	NA	24	NA	32	0	NO MISFIRES MIN. TO MOD. OBSERVABLE BLAST DAMAGE

BLAST MONITORING RESULTS - 1:1 PATTERN

UBC/CANMET 1997 - MONITORING RESULTS - 50 mm & 64mm DIAMETER BLASTHOLES

				BLASTING DETAILS							BLAST PERFORMANCE					
DATE	LOCATION	GEOPHONE DISTANCE (m)	ORE WIDTH (m)	DESIGN SPAC. & BUR.	BLASTHOLE LENGTH (m)	NUMBER OF BLASTHOLES	EXPLOSIVE	*PRIMING	DETONATORS	SEQUENCING	PPV (AVG.) (mm/s)	AVERAGE CHG. WT./DELAY (kg)	PPV (MAX.) (mm/s)	MAX. CHG. WT./DELAY (kg)	% MISFIRES	REMARKS
FEB.11/97	WZS 890 UPHOLES R28-45 (50mm BLASTHOLE DIAM.)	30	1.5	BUR. = 0.4m SPAC.= 0.8m	7	18	ANFO	90g TROJAN DP-CEN&TOE	EXEL-MS	EACH HOLE ON OWN DELAY	39	13.4	65	13.4	0	NO MISFIRES GOOD LOADING PRACTICES GOOD FRAGMENTATION OVERBREAK 0m TO 0.4m - MINOR BLAST DAMAGE
JULY 11/97	WZS 860 DOWNHOLES R32-41 (64mm BLASTHOLE DIAM.)	NA	2	BUR. = 0.5m SPAC. = 1.0m	8.8	10	ANFO	90g TROJAN DP-CEN&TOE	EXEL-MS	EACH HOLE ON OWN DELAY	NA	23.4	NA	23.4	NA	BLAST BROKE WELL; UNIFORM FRAGMENTATION WALLS JAGGED IN APPEARANCE BUT STABLE; OVERBROKE ON FW APPROX. 0.5m - BLAST DAMAGE
JULY 13/97	WZS 860 DOWNHOLES R42-51 (64mm BLASTHOLE DIAM.)	24	2	BUR. = 0.5m SPAC. = 1.0m	8.8	10	ANFO	90g TROJAN DP-CEN&TOE	EXEL-MS	EACH HOLE ON OWN DELAY	71	23.4	92	23.4	0	BLAST BROKE WELL; UNIFORM FRAGMENTATION WALLS JAGGED IN APPEARANCE BUT STABLE; OVERBROKE ON FW APPROX. 0.5m - BLAST DAMAGE
JULY 14/97	WZS 860 DOWNHOLES R52-67 (64mm BLASTHOLE DIAM.)	36	2	BUR. = 0.5m SPAC. = 1.0m	8.8	16	ANFO	90g TROJAN DP-CEN&TOE	EXEL-MS	EACH HOLE ON OWN DELAY	46	23.4	66	23.4	6	1 MISSFIRE BUT BLAST STILL BROKE WELL; UNIFORM FRAGMENTATION WALLS JAGGED IN APPEARANCE BUT STABLE; OVERBROKE ON FW APPROX. 0.5m - BLAST DAMAGE SOME DELAY CROWDING AFTER DELAY NO.12.