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# BLASTHOLE STOPING FOR NARROW VEIN MINING

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## Résumé

Ce document décrit l'exploitation minière par chantiers longs trous pratiquée dans les gisements filoniens canadiens de puissance inférieure à deux mètres. Bien que la technique soit en utilisation depuis plus de dix ans, elle demeure au stade expérimental et des tests sont requis avant d'appliquer la méthode dans de nouvelles exploitations minières ou dans des conditions géologiques différentes. Cette méthode d'exploitation est intrinsèquement plus sécuritaire que les autres méthodes appliquées aux gisements filoniens.

L'étude décrit les principaux paramètres d'opération pour des exploitations minières récemment passées en revue. Les critères d'applicabilité de la méthode sont définis et les problèmes rencontrés sont étudiés. Il est conclu que l'exploitation par chantiers longs trous appliquée à l'exploitation des gisements filoniens est une méthode viable et moderne, qui promet de devenir plus populaire. La nouvelle technologie reliée à la fragmentation ainsi que la recherche relative au sautage peuvent augmenter davantage l'applicabilité de cette méthode d'abattage.

## Abstract

This paper examines the application of blasthole stoping to narrow vein mining in Canada, for ore deposits where widths are less than two meters. While the technique has now been utilized for more than ten years, it remains in the experimental stage and on-site testing is necessary for its adoption at new mining operations or in different geological settings. It is intrinsically a safer method than other narrow vein methods in current use.

The investigation describes the main operating parameters for mines recently surveyed. The criteria for method applicability are defined and the problems encountered are studied. It is concluded that blasthole stoping for narrow vein mining is a viable modern method which promises to gain popularity and that emerging fragmentation technology and blasting research can be used to further increase applicability of the stoping method.

## INTRODUCTION

The objective of this paper is to review and analyze the techniques and practices involved in blasthole stoping applied to narrow vein mining in Canada, with particular emphasis on the drilling and blasting aspects. A portion of the review included herein is based on a previous study, Lizotte (1989), sponsored by Echo Bay Mining Ltd. and the Northwest Territories through a Mineral Development Agreement with the Government of Canada. New information obtained and further analysis since the time of the previous study constitute updates to this present paper.

The study of blasthole stoping for narrow vein mining is of particular relevance at this time since:

1. Narrow vein mining is now regarded as remaining a major source of precious metals supply through this century and technological improvements are essential in order to maintain and improve the competitiveness of narrow vein mining.
2. A number of sub-economic ore deposits have been found and delineated, which require new productive and low cost mining methods implementation before they can be economically exploited.
3. Powerful self-propelled drilling equipment is now available for working in narrow openings. The difficulty of hiring skilled miners for conventional narrow vein mining (e.g. shrinkage, cut-and-fill) together with the intrinsic greater safety of blasthole stoping make it an attractive alternate to be closely considered.

Blasthole stoping for narrow vein mining, or longhole narrow vein blasthole stoping is herein *arbitrarily* defined as blasthole stoping applied to ore widths of less than two meters, with drilling of parallel holes from sublevels, with no more than three holes per row (width) and drillhole diameters not exceeding 80 mm. The stoping method is first succinctly described and the range of design and operating parameters are stipulated. Method applicability is discussed and particular problems are examined. The mining operations which apply or have applied narrow vein blasthole stoping are reviewed briefly. Technical and scientific amelioration procedures are presented. The discussion focusses mainly on particular aspects of implementation of narrow vein blasthole stoping in specific mining operations and the need for further studies related to particular aspects of the stoping method.

## DESIGN PARAMETERS

Stope development for blasthole stoping in narrow veins is quite similar to sublevel stoping in wider deposits. As previously stated the maximum stope width is restricted to 2 m, implying only one drift per sublevel and approximately parallel drilling of blastholes. This stoping method applies to vertical or subvertical deposits with vein dips at least $50^o$ to enable gravity flow. It is essential that the ore vein be fairly continuous with a well defined *planar regularity* between sublevels. The minimum recommended stope width is 0.5 m, Larsen et al (1989). Excluding requirements for *planar regularity* of the ore vein, parameters which impact the greatest on required development, drilling and blasting procedures and, consequently, the applicability of the method, are the competencies of the ore, the footwall and hangingwall. Because, by definition, it is an open stoping method, any amount of wall sloughing strongly marks dilution considering the ore widths involved. Additional care in drilling and blasting practices and added expenses and development for natural or artificial ground support in some instances prohibit profitable application of the method.

### General Description of Stoping Method

Figures 1 and 2 schematically illustrate the development and main features of narrow vein blasthole stoping. Figure 1 is a typical cross section through a blasthole stope for narrow vein mining showing the lowest sublevel drilling drift, the undercut and drawpoint. Holes are drilled up and down in most instances to minimize development while controlling wander/deviation. Downhole drilling is more accurate and productive, so upholes are usually shorter. Several operators prefer to *dump* upholes forward, $65^o$ to $75^o$, such as shown in figure 2, towards the open stope, providing more comfortable and secure drilling, better breakage and a smoother free face for either the downhole blasts or a sill pillar bottom. Access to the sublevels is gained through raises or with a ramp in the case of trackless mining, the later enabling narrow drill carriers to be easily relocated. The current trend is to dimension sublevels to accommodate these narrow drill carriers which require a minimum width of 2 m; larger sublevels are often driven because of the available development mucking equipment (a 1.7 $m^3$ Load-Haul-Dump has a width of about 1.6 m while a 3.8 $m^3$ LHD has a width of about 2.5 m). Bar & Arm setups are still used, particularly in very narrow veins. The recommended minimum sublevel height is 3 m allowing for efficient 1.2 m rod changes.

The sublevels are spaced according to the regularity of the ore, ground stability and the range of accurate drilling possible. Sublevels are spaced between 6 m (Golden Patricia Mine) and 20 m (Lupin Mine). Stope block heights vary between 30 m and 80 m; block height is strongly linked to sublevel interval, whereby blocks are generally planned for a maximum of four drilling sublevels. Block lengths are limited by operational efficiency, ore grades and the need to leave natural pillars. Maximum stope length encountered is 105 m. Mining retreats from a slot raise, shown in figure 2, normally at one end of the stope. Starting from a central slot, such as frequently applied in *wide* blasthole stoping and retreating in both directions provides for greater operational flexibility, i.e. more work areas and better grade control, but with added development access costs. The bottom of the stope is undercut the full stope length and drawpoints are spaced between 9 m and 13 m.

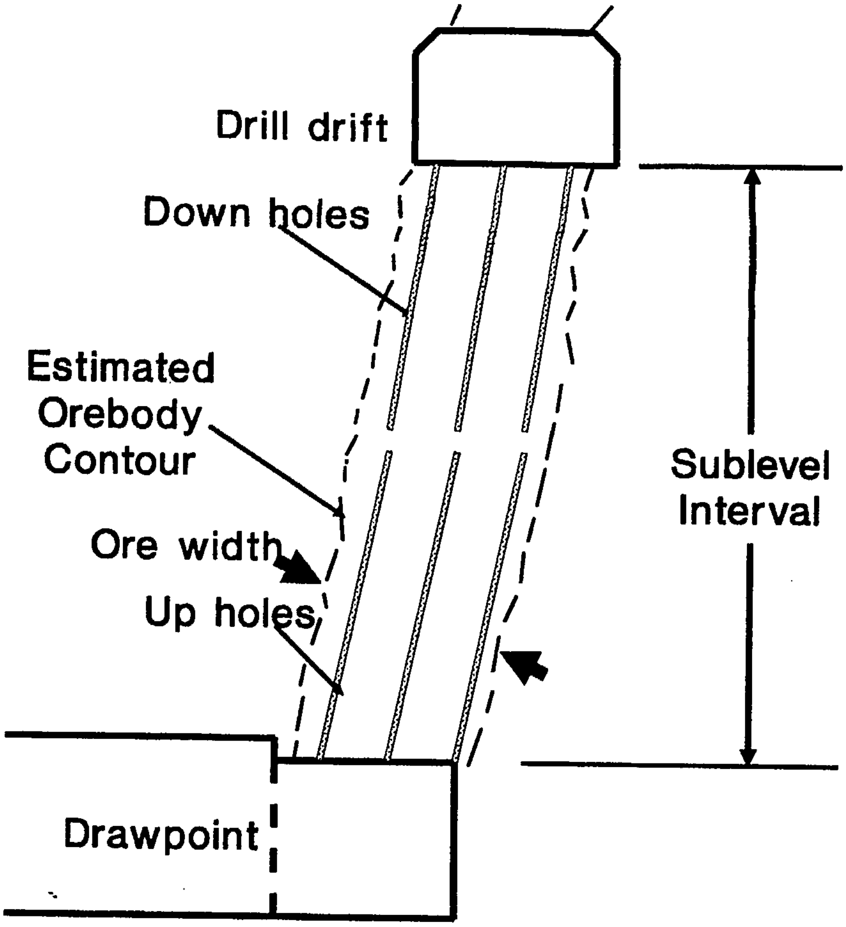


**Figure 1. Schematic Cross Section of Blasthole Stoping**

## Drilling and Blasting

Many of the mining operations surveyed developed blasthole stoping for narrow vein mining once blasthole stoping was established in wider veins or in thicker orebody widths. Consequently few attempts were made to optimize drillhole diameters. Drill-hole diameters characteristically range from 41 mm to 64 mm with most mines surveyed drilling 51 mm holes, with button bits. Round and hexagonal drill steels are used. A wide variety of drill types are used, such as Gardner Denver DH-99 and DH-123, Tamrock 322, BCI drill carrier with Secan S-36 drill and Atlas Copco BBC 120. The trend is towards drill carriers with electric-hydraulic drills. Up holes are drilled a maximum length of 12.2 m and down hole lengths vary between 7.3 and 16.8 m. When up and down holes are drilled the practice is to overlap lengths by slightly less than 1 m. Generally down holes are drilled parallel to the orebody cross-section and up holes are dumped towards the open stope.

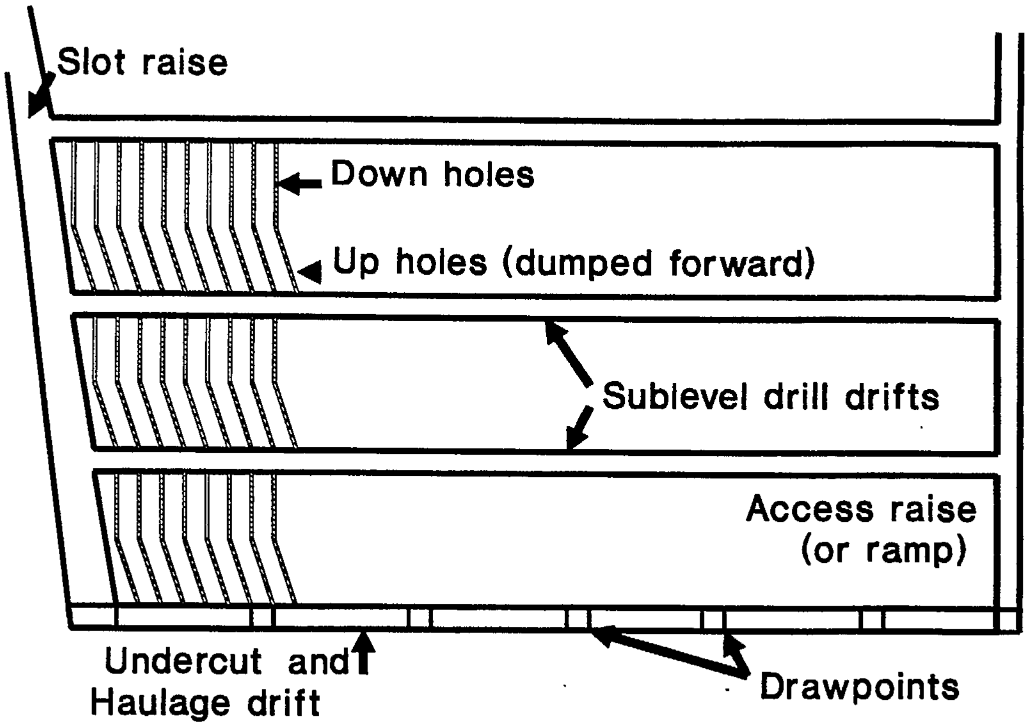


**Figure 2. Schematic Longitudinal Section of Blasthole Stoping**

The survey of mining operations revealed a surprising variety of different drilling geometries and blasting procedures for, what appears as, a gamut of similar orebody dimensions. The drilling patterns encountered in the survey are 3:3, 3:2, 3:1 and 2:1 layouts. Figure 3 illustrates typical 3:3 and 2:1 patterns which could be used in ore widths of 2.0 m and 1.5 m respectively. Variations are also noted regarding the firing sequences and the symmetries of the drilling sequences.

Explosives are detonated with short delay period (25 ms) electric detonators; the responsibility for the sequencing is given either to the engineering staff or to the miner, as a function of the mining operation. Figure 3 shows possible firing sequences. In the 3:3 pattern the middle hole is detonated first, the footwall hole second and the hangingwall hole is detonated last in the row; this ensures that the hangingwall hole will have a free face when it is fired, the objective being to protect the hangingwall as much as possible and reduce spalling induced by blast vibrations. In the 2:1 pattern shown in figure 3 all holes in a row are fired simultaneously; this tactic is also encountered in 3:3 and 3:2 patterns. The patterns are not always symmetrical, such as shown in figure 3 with the 2:1 pattern; since the holes in each row are fired simultaneously the pattern is designed so the burden carried is proportional to the explosive charge per delay number. When patterns have three holes per row it might also be beneficial to locate the central hole slightly off-centre, further from the hangingwall. Figure 4 illustrates this tactic initially used at the Dome Mine, Robertson (1986), to further protect the hangingwall. Certain mining operations place the holes directly on the ore/waste contacts while others tighten the pattern with the side holes 0.1 m to 0.2 m inside the desired breakage surface.

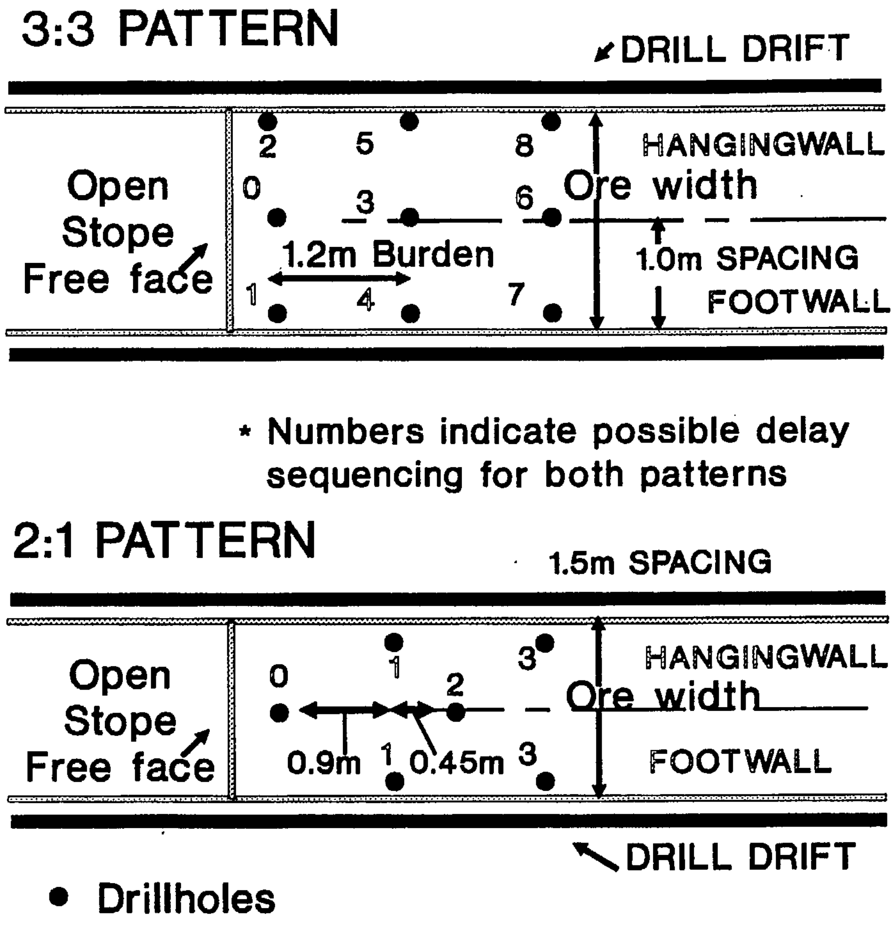


**Figure 3. Typical Drilling Layouts and Delay Sequencing**

The brief description of drilling and blasting practices clearly provides evidence that great care must be taken in the design of the drilling and blasting patterns and the accompanying operating procedures. Most problems described below are somewhat related to these practices. While certain mines appear to have few problems with the straightforward tactics they have adopted, the ability of a mining operation to rapidly determine adequate operating parameters can *make or break* narrow vein blasthole stoping for the mining operation.

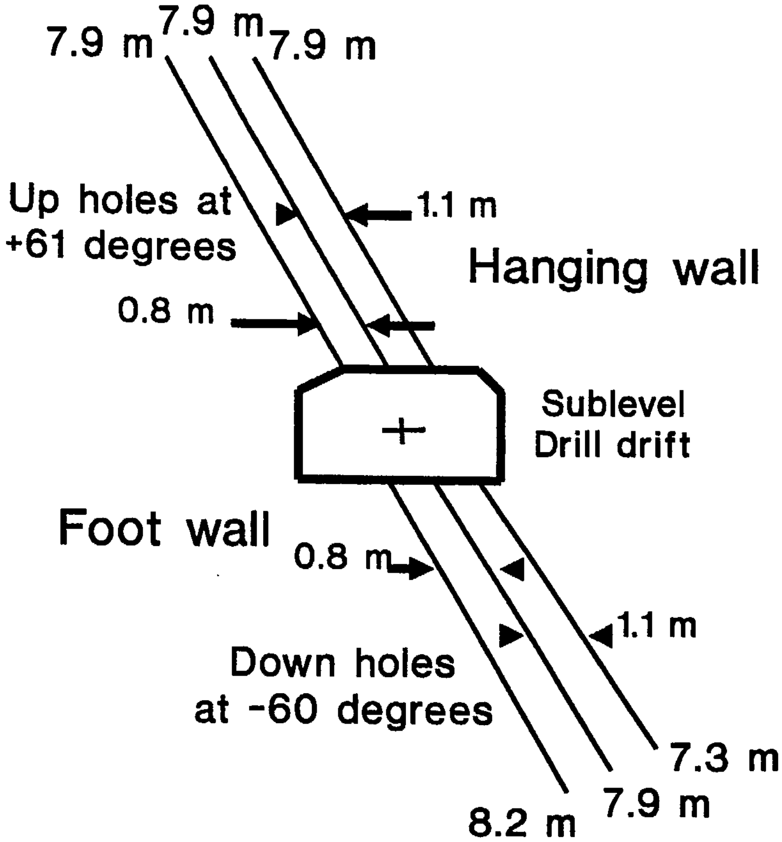


**Figure 4. Off-Centre Drilling Layout (Robertson (1986))**

## APPLICABILITY AND PROBLEMS

The essential features of a narrow vein deposit to apply blasthole stoping are: planar regularity, a dip sufficient for gravity flow of the ore and stability of the hangingwall and the footwall. The stoping method has been applied to vein widths as little as 0.3m (Golden Patricia Mine, Bond Gold Canada Inc.). A dilution of 0.5 m from each wall is not uncommon and quite acceptable in wide stopes; this represents 40% dilution for a 1.5 m wide stope ! Dilutions of up to 50% have been reported (the procedure for computing the reported dilution was not requested). The amount of dilution resultant from blasthole stoping is not the sole criterion for method applicability; it must be examined with regards to the dilution material, and possibly in comparison to the dilution resulting from alternative stoping methods.

The alternatives to blasthole stoping for narrow veins are shrinkage, cut-and-fill and Raise Platform Mining (RPM), Svensson (1976). When the stope walls are strong enough, shrinkage is more economical than cut-and-fill, but more dangerous because of possible voids occurring in the blasted rock and dangers associated with an unstable working surface. The RPM method was replaced by blasthole stoping at the Lupin Mine; this was justified on the basis that it is less labor intensive and cyclic than RPM and a higher level of productivity is obtained, Vatter (1989). Conversely, the RPM method is being considered to replace blasthole stoping for narrow veins at the Dome Mine. The change is motivated by the belief that a less skilled work force is required to efficiently apply RPM. The RPM method is slightly more selective than blasthole stoping and horizontal drilling may prove to be more accurate in some instances. Shrinkage and cut-and-fill are the most selective vein mining methods at present for mining narrow widths and irregular veins. Blasthole stoping is intrinsically safer since miners are never exposed to the open stope and a safe exit is ensured.

*Internal* dilution is caused by the irregularities of the ore vein and *stoping* dilution is caused by the minimum width which can be mined and not following ore contours. Dilution is increased by wall sloughing; this is the only source of dilution the on-going mining operation can attempt to reduce, given that the mining width cannot be reduced. In summary, most problems and concerns can be related to dilution and wall sloughing. Fragmentation, in terms of the distribution of fragment size, is very good for all operations surveyed, most mines reporting 80% to 90% passing 15 cm. To achieve this relatively fine fragmentation and avoid *benching*, mines probably over-blast using tighter patterns than necessary; this will induce sloughing but may be a necessary precaution, considering drilling accuracy, misfires and out-of-sequence detonations.

Drilling accuracy is an implicit concern associated with the stoping method. Skilled drillers are needed to ensure proper hole positioning, drill set-up and initial hole orientation. Drillhole deviation, induced by the drill or caused by the geology, may be crucial; it may dictate the maximum sublevel spacing and prevent use of a wider pattern and/or subtantial reduction in drilling and blasting costs. Initial alignment to attain acceptable toe spacing may actually be a cause of out-of-sequence detonation and excessive blast vibrations. Electric-hydraulic drills transmit more energy to the drill bit and, conceivably, can induce drillhole deviation. Present research concerned with the quality of drilling and aimed at reducing deviation will greatly affect the economics and applicability of the stoping method.

## MINING OPERATIONS

Most mining operations mentioned herein which apply or have applied blasthole stoping to narrow vein mining are fully reviewed in the previous report by Lizotte (1989). Consequently only salient features of these operations will be reported here.

The **Lupin Mine** (Echo Bay Mining Ltd.) is particularly active in trying to improve blasthole stoping; a substantial portion of their production is planned from a narrow zone of their ore deposit. The ore width averages 1.5 m, is sub-vertical, planar and very regular. Shrinkage and RPM were found to be less productive. Electric-hydraulic drills are used. Drillhole deviation is of concern, particularly with 20 m sublevel intervals. Dilution is controlled by drilling within the ore boundaries and careful blasting practices. Improved productivity is anticipated with a new drilling pattern, pending achievement of increased drilling accuracy.

The **Dome Mine** (Placer Dome Inc.) was among the first mines to experiment with longhole stoping for vein mining; the method has been in use since 1977, Robertson (1981). Their blasting procedures are *fine tuned* to protect the hangingwall, as previously illustrated in figure 4. Dilution and drillhole deviation are minimal in most cases and the method has proved to be more productive than shrinkage. As previously mentioned, the Dome Mine is now considering the RPM method to replace blasthole stoping.

Both the **Bell Creek Mine** (Canamax Resources) and the **Golden Patricia Mine** (Bond Gold Canada Inc.) have mined with blasthole stoping, Larsen et al (1989) and Lizotte (1989), but decided to abandon the method. Both mining operations now use shrinkage methods. Their motives for abandoning the blasthole method illustrate the typical problems which can be encountered with it; high dilution, high drillhole deviation (up to 0.6 m on 13.7 m holes for Bell Creek Mine), excessive hangingwall damage, dilution due to irregular ore vein, "method determined unsuitable due to sloughing during final stages of blasting" (Golden Patricia). Its attempted use was particularly challenging at the Golden Patricia Mine where ore widths vary between 0.3 m and 0.45 m and the nominal stope width is 0.8 m.

**Division Opémiska** (Minnova Inc.) successfully apply blasthole stoping with Bar & Arm drilling rigs in ore widths of 1.5 m. Drillhole deviation is less than 5% for holes up to 16.8 m long. Two different patterns are used, a 3:1 and 2:1, as a function of rock hardness. Un-symmetrical patterns, such as illustrated at the bottom of figure 3, have been divised to optimize the use of the explosive's energy. Estimated dilution is 30% and apparently acceptable for this mining operation.

The **Doyon Mine** (Lac Minerals Ltd.) use blasthole stoping in ore widths as narrow as 2 m. The ore is quite continuous and sublevels are spaced by 15 m. The equipment used in wider stoping areas are 2.7 $m^3$ LHDs and pneumatic drills for 64 mm diameter holes. Dilution is difficult to estimate because high grade samples are eliminated in reserve computations and higher tonnage than predicted is generally drawn from the stopes. Since the initial report by Lizotte (1989) a few more mining operations have been found to be using, or considering to use, blasthole stoping for narrow vein mining: **Macassa Division** (Lac Minerals) for recovering crown pillars as narrow as 1.9 m; **Forest Hill Mine** (Seabright) was considering use of the method and **Namew Lake Mine** (Hudson Bay Mining & Smelting) is now applying the method (no information is yet available concerning the later).

## TECHNICAL IMPROVEMENTS

From the previous presentation of blasthole applications it is apparent that a variety of adjustments and improvements can be made to the stoping method to meet particular requirements. It is hoped that the review by Lizotte (1989) and the present paper will serve to illustrate to potential users the adjustments that can be tried to improve its performance. The improvements suggested may require trial-and-error but do not necessitate additional technical tools.

Drilling accuracy is a problem for several operations and warrants particular attention. Operating procedures for initial drill positioning and hole orientation should be closely examined. Measure and monitoring of drillhole deviation, as a function of direction, rockmass characteristics and drilling parameters is worthwhile; limiting downpressure on electric-hydraulic drills could be considered.

The stability of the stope walls is paramount to the success of the stoping method. Increased stability can be gained with cable bolting; only **Division Opémiska** have reported use of 4.9 m cable bolts. Mandolin cable could also be tried to add support to the footwall, and may be more appropriate where geological discontinuities are parallel to the stope walls. A number of methods can be applied to reduce blast vibrations such as off-centre drillholes and detonation sequencing which have been previously described. Lower strength explosives (all mines presently use ANFO) have been tried and the following section demonstrates the benefits of using diluted ANFO. A simple test blast with single charged drill holes can be used to determine the breakage angle achieved in a rock type, crater shape to be expected and optimum burden. In general, given the nature of the stoping method, technical improvements should focus on elements of *quality control* to increase profitability and applicability of the method.

## 'SCIENTIFIC' IMPROVEMENTS

Improvements can also be made to narrow vein blasthole stoping with 'scientific' tools now available and applicable to the analysis and design of underground stoping operations.

Blast vibration monitoring instrumentation is useful in the analysis of blast efficiency in terms of controllable drilling and blasting parameters. The **Lupin Mine** study involved vibration analysis for the purpose of recommending modifications, once it was determined that the blasting was performing as planned through the measurement of energy levels in the near-field. Roy (1989) demonstrated how blast vibration monitoring can be applied as a diagnostic tool in the process of blast optimization. New developments in blast monitoring instrumentation technology now enable measurement of peak particle velocities, detonation velocities, the performance of each charge detonated <u>and</u> the real time of charge detonation. Monitoring instrumentation are not as such solutions to blasting problems. By acquired experiences, analysis and knowledge of parameters which affect blast performance, blast monitoring supplies a rational basis for modification of blast design.

Other 'scientific' tools available to aid blast design and site-specific improvements are several empirical procedures which are available to predict rock fragmentation by blasting. Empirical fragmentation estimation procedures are based on field experiments in which scaled or full scale blasts are performed, fragmentation measured and sets of equations calibrated to best fit resultant fragmentation. The procedures consider to various extents rock properties, blast parameters and layout, and predict either the mean fragment size or the full size distribution. Published procedures are reviewed by Lizotte (1990). The case study which follows illustrates the Kuz-Ram procedure developed by Cunningham (1987).

## <u>Case Study: Narrow Vein Blasthole Stoping Design</u>

The Kuz-Ram procedure is summarized in Appendix A. It is fully described by Cunningham (1987) and reviewed by Lizotte (1990). It involves estimation of the mean fragment size with the Kuznetsov equation [1] and application of the Rosin-Rammler equation [4] with an estimate of the Rosin-Rammler exponent [6] to take into consideration specific blast parameters. The Kuz-Ram procedure provides a *relative* fragmentation prediction tool rather than a prediction of actual fragmentation.

The first example compares three different drillhole diameters (51 mm, 64 mm and 76 mm) with three drilling patterns (3:3, 3:2 and 2:1) for blasthole stoping in a vein 1.8 m wide and drillhole lengths of 10 m. Table 1 summarizes available information and the Kuz-Ram analysis results. Figure 5 illustrates the predicted fragmentation size distributions. Note that the predicted mean sizes, approximately 15 cm, for the three patterns are nearly identical.

Table 1. Comparison of drilling and blasting techniques

| PARAMETERS | DRILLHOLE DIAMETER | | |
|---|---|---|---|
| DRILLHOLE DIAMETER | **51 mm** | **64 mm** | **76 mm** |
| DRILLING PATTERN | 3:3 | 3:2 | 2:1 |
| BURDEN (m) | 1.2 Rectangular | 1.7 Staggered | 1.1 Staggered |
| SPACING (m) | 0.9 | 0.9 | 1.8 |
| PREDICTED MEAN SIZE (cm) | 15.6 | 15.5 | 15.3 |
| ROSIN-RAMMLER COEFFICIENT | 1.18 | 1.36 | 1.62 |
| VOLUME PER BLASTHOLE ($m^3$) | 7.2 | 12.2 | 13.2 |
| EXPLOSIVES PER HOLE (kg) | 17.4 | 27.3 | 38.6 |
| EXPLOSIVES PER TONNE (kg/t) | 0.86 | 0.80 | 1.04 |
| DRILLING PER TONNE (m/t) | 0.495 | 0.292 | 0.270 |
| COMMON PARAMETERS | | | |
| EXPLOSIVE: ANFO Relative Weight Strength: 100 Specific Gravity: 0.85 | | | |
| LENGTH OF HOLES: 10.0m Drilling Accuracy: 0.4m Charge Length: 10.0m | | | |

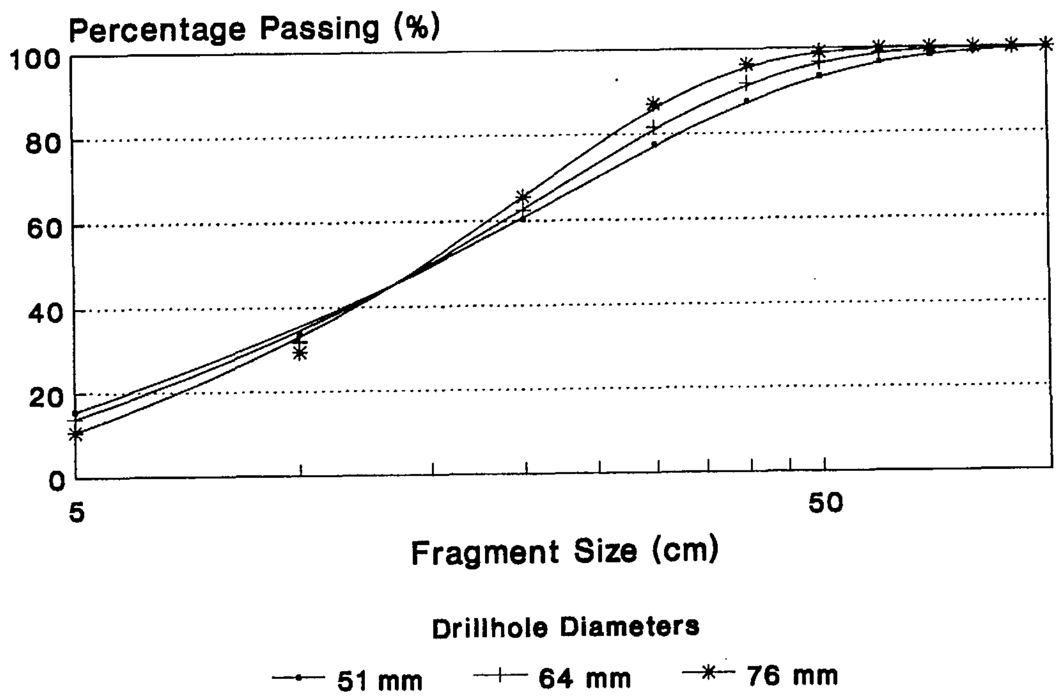


Figure 5. Fragmentation Prediction for Three Drilling Patterns

In this case the burdens for the 64 mm and 76 mm drillholes were adjusted to obtain the same mean fragment size as achieved with the 51 mm drillholes. The conventional practice in comparing different drillhole diameters might have involved setting the same *powder factor* for all three cases. This could result in excessive blasting in the case of 64 mm holes and coarse fragmentation for the 76 mm holes; comparison based on predicted mean fragmentation thus provides a rational basis for the analysis. Note that the *spread* of the distributions shown in figure 5 are different. This is due to the Rosin-Rammler coefficient estimate from equation [6] which takes into consideration the different blasting geometries. Further analysis could involve comparison of various patterns to obtain the same percentage of a given size. A complete comparison could also consider assumptions regarding different drillhole deviations as a function of drillhole diameter.

The next example illustrates pattern design with diluted ANFO. The initial pattern with 64 mm holes is simulated anew, but with polystyrene diluted ANFO. Input data and fragmentation predictions are summarized in table 2. Three different burdens are compared; 1.7 m as previously analyzed with full strength ANFO, 1.4 m and 1.1 m. The fragmentation size distributions are illustrated in figure 5. The three distributions have the same shape but the mean sizes differ. The 1.1 m burden produces approximately the same mean fragment size as the 1.7 m burden with full strength ANFO.

The objective of this analysis is to determine the added expense and predicted fragmentation achieved with diluted ANFO. In a full comparison the <u>benefits</u> obtained in terms of reduced spalling and, consequently, reduced dilution, would need to be measured against the added expense. Adding polystyrene to ANFO has the same effect as decoupling but less expensive and laborious (Day and Webster (1982)). The direct effect of decoupling is the reduction in velocity of detonation (VOD); a full strength ANFO will have a VOD of about 3300 m/s while a 50% ANFO-polystyrene mixture, such as used in the example, will have a VOD of about 2600 m/s (Nielson and Heltzen (1987)). A lower VOD will reduce detonation pressure and borehole pressure. Peak particle velocity (PPV) is proportional to detonation pressure and the PPV measured at a given point is indicative of the damage the rockmass may suffer at that point. Thus, diluted ANFO will reduce the damage envelope around the blast pattern. The example shows a different approach to blast design whereby fragmentation is predicted using an empirical procedure and blast parameters evaluated in terms of potential damage to stope walls. This integrated approach differs from conventional design which may only seek to minimize drilling and maximize explosive charge per meter drilled. A comprehensive analysis regarding diluted ANFO use would require <u>full</u> fragmentation analysis, as implied by Day and Webster (1982): "... Although the explosive energy was cut in half, fragmentation remained acceptable. This was attributed to prior overblasting and the fact that oversize frequently originates from overbreak".

Table 2. Predicted Fragmentation with diluted ANFO

| PARAMETERS | DRILL PATTERNS | | |
|---|---|---|---|
| DRILL PATTERN | SAME | INTERMEDIATE | TIGHTER |
| BURDEN (m) | 1.7 | 1.4 | 1.1 |
| PREDICTED MEAN SIZE (cm) | 24.8 | 21.2 | 17.5 |
| VOLUME PER BLASTHOLE ($m^3$) | 12.2 | 10.0 | 7.9 |
| EXPLOSIVES PER TONNE (kg/t) | 0.41 | 0.50 | 0.63 |
| DRILLING PER TONNE (m/t) | 0.292 | 0.357 | 0.452 |
| **COMMON PARAMETERS** | | | |
| EXPLOSIVE: Diluted ANFO Relative Weight Strength: 93 S.G.: 0.435 | | | |
| Charge Length: 10m Hole Length: 10m Rock Factor: 13 | | | |
| Drilling Accuracy: 0.4m S.G. Rock: 2.8 Rosin-Rammler: 1.36 | | | |
| Staggered Pattern Spacing: 0.9 m Weight Diluted ANFO: 14.0kg | | | |

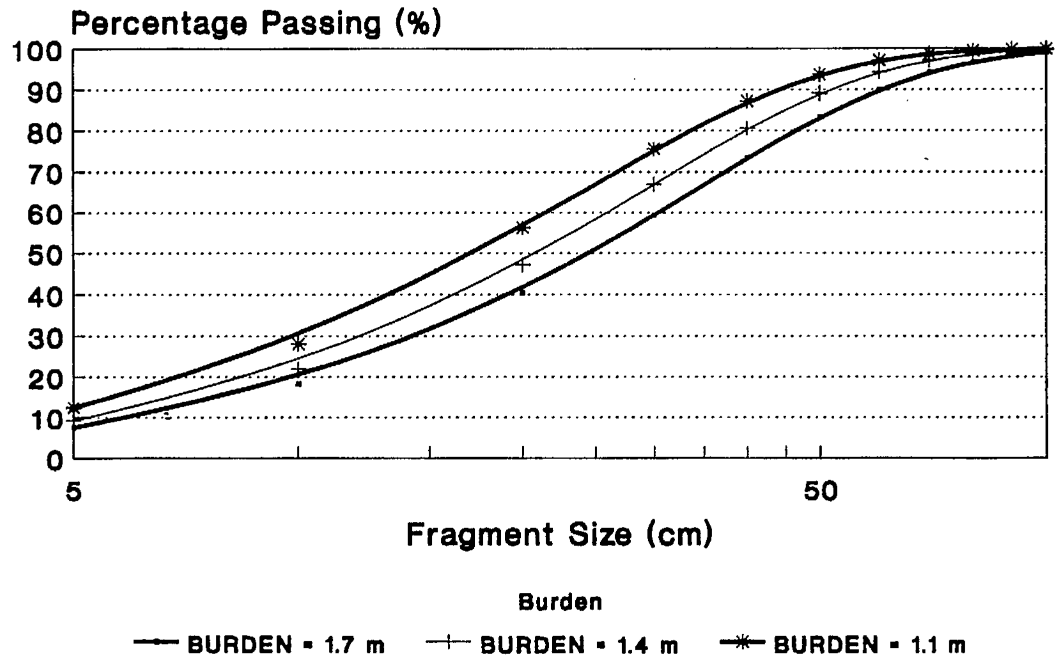


Figure 6. Fragmentation Prediction with Diluted ANFO

## CONCLUSIONS

Blasthole stoping is a safe and viable method of mining narrow vein-type ore deposits. The mining operations which have applied the stoping method have proved its economic effectiveness in comparison to alternative methods and demonstrated that adjustments through technological improvements increase its applicability and profitability. Increases in applicability and benefits can still be achieved, owing to new technological developments and analysis procedures, such as blast vibration monitoring and empirical fragmentation prediction. Future analytical design procedures, improved drilling accuracy and narrow drill carriers will undoubtably further increase profitability and applicability of blasthole stoping for narrow vein mining.

## ACKNOWLEDGEMENTS

The mining operations which have provided information to enable the survey of blasthole stoping applied to narrow vein mining in Canada are gratefully acknowledged.

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## APPENDIX A
## THE KUZ-RAM FRAGMENTATION PREDICTION MODEL

The Kuz-Ram model developed by Cunningham (1987) is based on the application of the Kuznetsov equation [1], the Rosin-Rammler equation [4] and a procedure derived by Cunningham to estimate the Rosin-Rammler exponent, equation [6].

The Kuznetsov equation is as follows:

$$\bar{x} = A\left(\frac{V_o}{Q}\right)^{4/5} Q^{1/6}\left(\frac{115}{E}\right)^{19/30} \qquad [1]$$

where,

$\bar{x}$ = mean fragment size in cm.

$A$ = Empirical rock factor

$V_o$ = Volume per blasthole = Burden x spacing x bench height, in $m^3$

$Q$ = Explosive mass per blasthole, kg above grade level, normally excluding explosive in subdrill section.

$E$ = Relative weight strength of explosive (ANFO=100)

Cunningham (1987) refined the estimation of the rock factor based on detailed experiments and precise actual assessments of fragmentation:

$$A = 0.06[RMD + JF + RDI + HF] \qquad [2]$$

where,

RMD = Rock Mass Description. Powdery, friable, RMD = 10. Vertically jointed, see JF. Massive, RMD = 50.

$$JF = JPS + JPA \qquad [3]$$

JPS = Vertical joint spacing. Less than 0.1, JPS = 10. From 0.1 to MS (defined oversize), JPS = 20. From MS to DP (drill pattern), JPS = 50.

JPA = Joint Plane Angle. Dip out of face, JPA = 20. Strike perpendicular to face, JPA = 30. Dip into face, JPA = 40.

RDI = Rock Density Influence RDI = 25RD - 50, RD = Density, tonnes per $m^3$, and

HF = Hardness Factor (Young's Modulus less than 50GPa, HF = Y/3; Young's Modulus greater than 50GPa, HF = UCS/5 where UCS is the uniaxial compressive strength in MPa).

The Rosin-Rammler equation is a well known equation used to characterize particle size distributions in a variety of applications. It is algebraically expressed as follows:

$$R = e^{-\left(\frac{x}{x_c}\right)^n} \qquad [4]$$

where,

$R$ = mass fraction larger than size $x$.

$x$ = diameter of fragment, cm.

$x_c$ = characteristic size, cm.

$n$ = Rosin-Rammler exponent

$e$ = base of natural logarithms, 2.7183..

The characteristic size, $x_c$, is approximately the 36.8% size retainment point on the size distribution function. The Rosin-Rammler exponent, $n$, is known as the uniformity coefficient; a wide variety of size distributions can be modelled with the Rosin-Rammler equation by changing the value of $n$ to fit the curve. Cunningham (1987) notes that for blasted rock, $n$ is generally in the range 0.8 to 1.5 and is normally around 1.0. The uniformity coefficient controls the spread of the distribution; in most cases a higher value of $n$ is preferred, signifying a more uniform distribution with less fines and oversize materials.

Since the mean size is obtained from the Kuznetsov equation, the characteristic size can be estimated once the value of 'n' is estimated with the following equations:

$$x_c = \frac{\bar{x}}{\sqrt[n]{-ln(0.5)}} \approx \frac{\bar{x}}{\sqrt[n]{0.6931472}} \qquad [5]$$

The most significant contribution of Cunningham was to relate the Rosin-Rammler exponent to blasting parameters. Given that the Kuznetsov equation accounts for explosive strength and rock mass characteristics, and that the mean size is related to the characteristic size of the Rosin-Rammler distribution, the only unkown left is the uniformity coefficient. Cunningham established the applicable uniformity coefficient through several investigations taking into consideration the impact of such factors as: blast **geometries**, hole diameter, burden, spacing, bottom and column charge lengths, hole length and drilling accuracy.

The exponent for the Rosin-Rammler equation is estimated as follows:

$$n = \left(2.2 - 14\frac{B}{d}\right)\left(\left[\frac{(1+S/B)}{2}\right]^{0.5}\right)\left(1-\frac{W}{B}\right)\left[\frac{|BCL-CCL|}{L+0.1}\right]^{0.1}\left(\frac{L}{H}\right) \qquad [6]$$

where,

$d =$ Hole diameter, mm.

$B =$ Burden, m.

$S =$ Spacing, m.

$BCL =$ Bottom charge length, m.

$CCL =$ Column charge length, m.

$L =$ Total charge length, m.

$W =$ Standard deviation of drilling accuracy, m.

$W$: Drilling accuracy is normally taken to have a standard deviation between 0.3m and 1.0m, depending on blasthole length, angle of drilling and local conditions.

$H =$ Bench height, m.

Cunningham (1987) suggests $n$ be increased by 10% for staggered patterns (compared to square patterns).

A complete description and analysis of the Kuz-Ram procedure and other empirical fragmentation prediction procedures are reviewed by Lizotte (1990).