PIT SLOPE MANUAL

chapter 7

PERIMETER BLASTING

This chapter has been prepared as part of the

PIT SLOPE PROJECT

of the

Mining Research Laboratories
Canada Centre for Mineral and Energy Technology
Energy, Mines and Resources Canada

MINERALS RESEARCH PROGRAM
MINING RESEARCH LABORATORIES
CANMET REPORT 77-14

Available by mail from:

Printing and Publishing
Supply and Services Canada,
Ottawa, Canada K1A 0S9

CANMET
Energy, Mines and Resources Canada,
555 Booth St.
Ottawa, Canada K1A 0G1

or through your bookseller.

Catalogue No. M38-14/7-1977 Price: Canada: $2.25
ISBN 0-660-01011-9 Other countries: $2.70

Price subject to change without notice.

En vente par la poste:

Imprimerie et Édition
Approvisionnements et Services Canada,
Ottawa, Canada K1A 0S9

CANMET
Énergie, Mines et Ressources Canada,
555, rue Booth
Ottawa, Canada K1A 0G1

ou chez votre libraire.

N° de catalogue M38-14/7-1977 Prix: Canada: $2.25
ISBN 0-660-01011-9 Autres Pays: $2.70

Prix sujet à changement sans avis préalable.

THE PIT SLOPE MANUAL

The Pit Slope Manual consists of ten chapters, published separately. Most chapters have supplements, also published separately. The ten chapters are:

1. Summary
2. Structural Geology
3. Mechanical Properties
4. Groundwater
5. Design
6. Mechanical Support
7. Perimeter Blasting
8. Monitoring
9. Waste Embankments
10. Environmental Planning

The chapters and supplements can be obtained from the Publications Distribution Office, CANMET, Energy, Mines and Resources Canada, 555 Booth Street, Ottawa, Ontario, K1A 0G1, Canada.

Reference to this chapter should be quoted as follows:

Calder, P. Pit Slope Manual Chapter 7 - Perimeter Blasting; CANMET (Canada Centre for Mineral and Energy Technology, formerly Mines Branch, Energy, Mines and Resources Canada), CANMET REPORT 77-14; 82 p; May 1977.

FOREWORD

Open pit mining accounts for some 70% of Canada's ore production. With the expansion of coal and tar sands operations, open pit mining will continue to increase in importance to the mineral industry. Recognizing this, CANMET embarked on a major project to produce the Pit Slope Manual, which is expected to bring substantial benefits in mining efficiency through improved slope design.

Strong interest in the project has been shown throughout its progress both in Canada and in other countries. Indeed, many of the results of the project are already being used in mine design. However, it is recognized that publication of the manual alone is not enough. Help is needed to assist engineers and planners to adopt the procedures described in the manual. This need for technology transfer will be met by a series of workshops for mine staff. These workshops will be held in various mining centres during the period 1977-81 following publication of the manual.

A noteworthy feature of the project has been its cooperative nature. Most organizations and individuals concerned with open pit planning in the country have made a contribution to the manual. It has been financed jointly by industry and the federal government.

Credit must be given to the core of staff who pursued with considerable personal devotion throughout the five-year period the objectives of the work from beginning to end. Their reward lies in knowing that they have completed a difficult job and, perhaps, in being named here: M. Gyenge, G. Herget, G. Larocque, R. Sage and M. Service.

D.F. Coates
Director-General
Canada Centre for Mineral and
Energy Technology

SUMMARY

Perimeter blasting techniques limit the damage to final pit walls and benches. This is done by lowering the explosive energy concentration at the pit perimeter, and by controlling the energy concentration at the pit wall due to the main production blast. The more common forms of blast damage are backbreak, crest fracture or loose face rock on the immediately adjacent pit wall. However, blast vibration may damage pit walls and buildings some distance from the blast.

Explosives for open pit mining have a wide range of borehole pressures. For a given loading density, backbreak and vibration will increase with borehole pressure of the explosive used. Thus wall damage can be reduced by using a lower pressure explosive.

For a given explosive, borehole pressure increases with charge diameter. AN/FO and permissible explosives produce a fracture radius two to four times smaller than dynamite.

Borehole pressure can be reduced by decoupling and/or decking charges. In decoupling, a space is left between the explosive charge and the borehole. In decking, short charges are taped to a primacord line or spacers are alternated with explosives to produce a discontinuous explosive column. However, if the blastholes are water filled, the effectiveness of decoupling is greatly reduced.

To minimize backbreak and vibration, blasts should be sequenced so that each row or hole can break to a free face. The vibration level depends on the charge weight per delay. In general, all charges not separated by at least a 15 millisecond delay act together in causing vibration. Some additive effects do occur between delayed charges. However, the maximum increase is twice the single charge effect.

The depth of subgrade drilling and blasthole collar affect crest fracturing. In competent rock a collar of 12 charge diameters should limit wall damage. In medium strength rock the collar should be about 22 times the charge diameter. In soft or incompetent rock a collar of 30 charge diameters should be used.

Subgrade drilling should be seven to ten times the charge diameter. It is not required in horizontally bedded or jointed rock and should be avoided over future haul roads or final berms.

Rock properties must be taken into account if successful perimeter blasting procedures are to be developed. Most significant are the frequency and orientation of structural features, and in situ dynamic rock strength.

Backbreak occurs when the in situ dynamic compressive strength of the rock mass is exceeded. This strength can be determined by test blasts. The dynamic compressive strength is equal to the maximum explosive pressure which does not produce crushing in the borehole wall.

Dynamic tensile strength is determined from blasts in paired holes. Using an explosive loading density that will not crush the borehole

wall, the maximum spacing is established at which a good connecting crack is produced between the paired holes. The dynamic tensile strength of the rock mass is then determined from the explosive pressure on the hole and the area of rock fractured.

Presplit faces in heavily jointed and fractured rock show considerably more backbreak than faces in competent rock. Areas where joints are tight or in-filled have less backbreak than areas with open joint systems. Smooth faces are easily achieved when a weak structural feature parallels the desired face. Undercutting of joints or faults almost parallel to the final pit wall can produce loose face rock and overbreak. Good wall conditions should be attainable where steeply dipping joints are parallel to the final pit face. With steeply dipping joints normal to the pit face, some backbreak involving cross bedding fractures can be expected. Backbreak problems rarely occur in flat-lying sedimentary deposits if cross bedding fractures are not excessive.

The frequency of discontinuities has a major influence on backbreak and crest fracture. Discontinuities interfere with control blasting when their spacing is less than hole spacing.

Buffer blasting is the cheapest form of perimeter blasting. Drilling costs for buffer blasting are slightly higher than for production blasting because of reduced hole spacing.

The burden and spacing for the buffer row should be 0.5 to 0.8 times that for the adjacent production row. Hole spacing for the buffer row should be 1.25 times the burden. The charge per hole should result in an effective powder factor about 0.6 times that used for production blasts.

In cushion blasting, holes are detonated after the production blast to trim the slope to the planned excavation limit. The charge is designed to create a low borehole pressure and limit backbreak. For best results all cushion blastholes should be fired together.

The burden/spacing ratio in competent rock should be 0.8 to 1.25. In highly fractured or soft rock it should be 0.5 to 0.8. In the case of cushion rows adjacent to a final production row, the burden is measured from the backbreak line of the production row. Guide holes may be helpful if it is difficult to obtain a good wall with cushion blasting.

In pre-splitting, a line of lightly charged holes is fired prior to the production blast to produce a continuous fracture along the planned excavation line.

For best results, accurate drilling is required as the holes should be in the same plane. Bad toe from preceding blasts should be removed so that the pre-split holes will be easier to drill.

The explosive charge must not crush the surrounding rock. A buffer row of blastholes is used to shield the pre-split line from the effects of the production blast.

Pre-split holes should if possible be fired 50 milliseconds before the main blast. However, if hole caving is likely, loading and firing in groups of not more than six holes as they become available is recommended. Operational constraints may also require the firing of groups of pre-split holes as they become available.

In line drilling, a row of closely spaced non-blast holes is drilled at the planned boundary. This produces a plane of weakness to which the final production row breaks. A buffer row is required to give protection from the production blast. The most common hole sizes are 1.5 and 3 in. (4 and 8 cm), but large diameter holes can be used.

Line drilling is the most expensive method of control blasting, followed in decreasing order by pre-splitting, cushion blasting and buffer blasting. Line drilling costs are an order of magnitude greater than costs of the other perimeter blasting techniques.

Production blasts must be designed to minimize ground vibration to protect nearby structures such as buildings or underground openings or to prevent minor falls of loose rock. Control of vibration can be exercised through the delays, firing sequence, blast geometry, explosives and stemming used in the production blast.

Delays between rows of blastholes should be greater than 15 milliseconds, particularly for the final two production rows before the pit limit. V-cuts allow individual charges to break to a free face; this reduces vibration. Square patterns produce higher vibration than rectangular patterns. The explosive with the lowest pressure consistent with the necessary rock breaking power should be used in production blasting.

Vibration from blasting will be slightly reduced if there is no stemming. However, in most cases this decrease is not enough to warrant the increased airblast and flyrock.

Ground particle velocity gives the best indication of probable blast damage. Damage should not occur at a particle velocity below 2 in./sec (5 cm/sec). An empirical relationship between particle velocity, charge weight per delay and distance from blast can be used in blast design to ensure this velocity is not exceeded. Alternatively, ground vibrations can be measured using a seismograph to establish the time relationship between charge weight per delay, distance and particle velocity.

ACKNOWLEDGEMENTS

Mr. G. Larocque was responsible for production of this chapter. Address enquiries to him at: 555 Booth St, Ottawa, Ontario, K1A 0G1.

The chapter was written by staff members of the Mining Engineering Department of Queen's University under the direction of Dr. Peter Calder.

The sole contractor has been:

Queen's University, Kingston, Ontario

The Pit Slope Project is the result of five years' research and development cooperatively funded by the Canadian Mining Industry and the Government of Canada.

The Pit Slope Group has been successively led by D.F. Coates, M. Gyenge, and R. Sage; their colleagues have been G. Herget, B. Hoare, G. Larocque, D. Murray and M. Service.

CONTENTS

FIGURES

TABLES

PURPOSE AND SCOPE

1. The scale-up in size of open pit operations in recent years has resulted in major improvements in efficiency which have been very beneficial to the industry. Increased use of higher bench heights, larger diameter blastholes and more powerful explosives have all played an important part in reducing mining costs. These measures have also resulted, however, in an increased energy concentration in the blast area which can result in severe backbreak problems for final pit walls.

2. If backbreak is not controlled it ultimately necessitates a decrease in the overall pit slope angle, with such major adverse economic consequences as decreased recoverable ore reserves and increased waste-to-ore ratios. More face loose rock will be produced and planned safety berms will be narrower and less effective or non-existent. Hazardous working conditions result. Remedial measures such as scaling of large areas and the use of wire mesh or other artificial support are very expensive and difficult to implement.

3. It is apparent there must be a trade-off between money saved by using larger blasts and money spent to maintain pit wall quality. The best approach is to control the effects of blasting so that the inherent strength of the walls is not destroyed. Methods of accomplishing this are known as "control blasting" and are the central theme of this chapter.

4. Four basic control blasting techniques are used in open pit mines - pre-splitting, cushion blasting, buffer blasting and line drilling. All of these are designed to create a low explosive energy concentration per square foot of wall at the perimeter of the pit. The energy concentration of the main production blast must also be controlled so that it does not damage the final pit wall. This low energy concentration at the final wall can be obtained by using decoupling charges, by using decking charges, by using less powerful explosives, by decreasing blasthole diameters, and by changing the burden and spacing.

The effect of changing these controllable variables will be discussed later in this section.

5. The properties of the rock being blasted also influence the success of a blast fragmentation and final state of the pit wall. The most important properties to consider are in situ rock strength and the nature, frequency, and orientation of structural features. Since these variables cannot be controlled, they must be evaluated by suitable field tests, and a control blast designed using such controllable variables as spacing, burden, hole diameter, etc so that the blast will be successful for those particular rock conditions.

6. A control blasting program can easily be incorporated into a mining operation. In this chapter, the various control blast techniques are described in terms of how they work, the kind of results they produce, and their costs. All of the procedures, tests, and technical details necessary for a mine operator to develop optimal wall control procedure, based on site conditions, at each stage of mine development are given. The required steps for reducing ground shock to protect buildings or underground openings are outlined. Current Canadian control blasting practice is summarized.

7. Optimization of pit slopes should be an integral part of mine design. Thorough planning is essential. The information on control blasting in this chapter prescribes what to do under various conditions, when to do it, and how to do it.

THE CONTROLLABLE VARIABLES

8. Four basic types of damage can result from blasting:

a. damage to pit walls immediately adjacent to a blast (eg, backbreak, crest fracture, face loose rock)

b. damage to pit walls not adjacent but still close to a blast (eg, shaking down of loose or weathered rock)

c. damage to buildings or underground openings close to a blast (eg, cracking of foundations or walls, spalling)

d. damage to pit walls due to a blast outside but near the pit (eg, blasting rock to install a crusher)

9. The mine operator has a number of tools at his disposal with which he can minimize or eliminate these problems. He can control the type of explosives, loading density, blasthole diameter, burden and spacing, depth of subgrade drilling, height of collar, and height of stemming.

10. For example, consider the direct damage to final pit walls caused by blasting. Backbreak, crest fracture, and face loose rock are typical of this type of damage. Loading density could be varied to minimize backbreak and face loose rock by changing the type of explosives and blasthole diameter, by decoupling and decking, and by altering burden and spacing. Changing the depth of subgrade drilling or height of collar will reduce crest fracture and subsequent narrowing of the surface of safety berms.

11. The following is a more detailed discussion of the controllable variables and how they can reduce damage due to blasting.

Explosive Type

12. One way of rating explosives is by comparing borehole pressures produced on detonation. The peak pressure exerted by the expanding gases from the explosion depends primarily on the

explosive density and the detonation velocity. Figures 1(a) and 1(b) provide the borehole pressures produced by various C.I.L. and DuPont explosives. Pressures were calculated using the following equation:

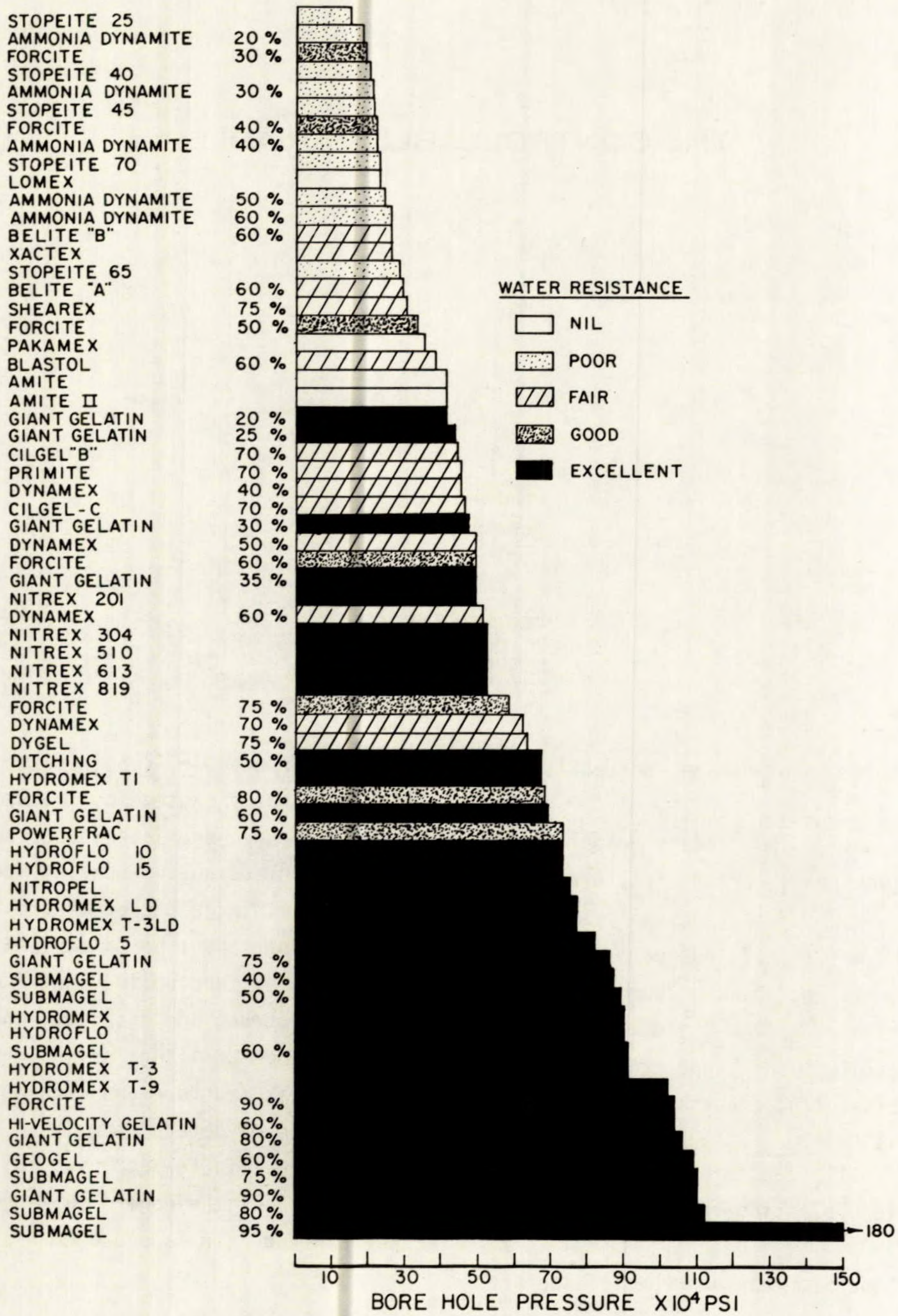

Fig 1(a) - Calculated borehole pressures for various C.I.L. explosives.

$$P_b = N\rho D^2 \qquad \text{eq 1}$$

where

P_b = borehole pressure of a charge completely filling the blasthole (psi)

N = constant, determined from Fig 2

ρ = specific gravity of the explosive

D = detonation velocity for a confined explosive having a specific gravity ρ (ft/sec)

13. The lower the borehole pressure, the less backbreak that will occur. This principle is illustrated in Fig 3. The rupture radius, as measured from the centre of gravity of the charge, produced by the AN/FO and permissible explosives is 2 to 4 times less than the radius of rupture from the same volume of dynamite in the same rock type. The borehole pressure produced by AN/FO and permissible explosives is several times less than that produced by dynamite. The curves for C-4 and

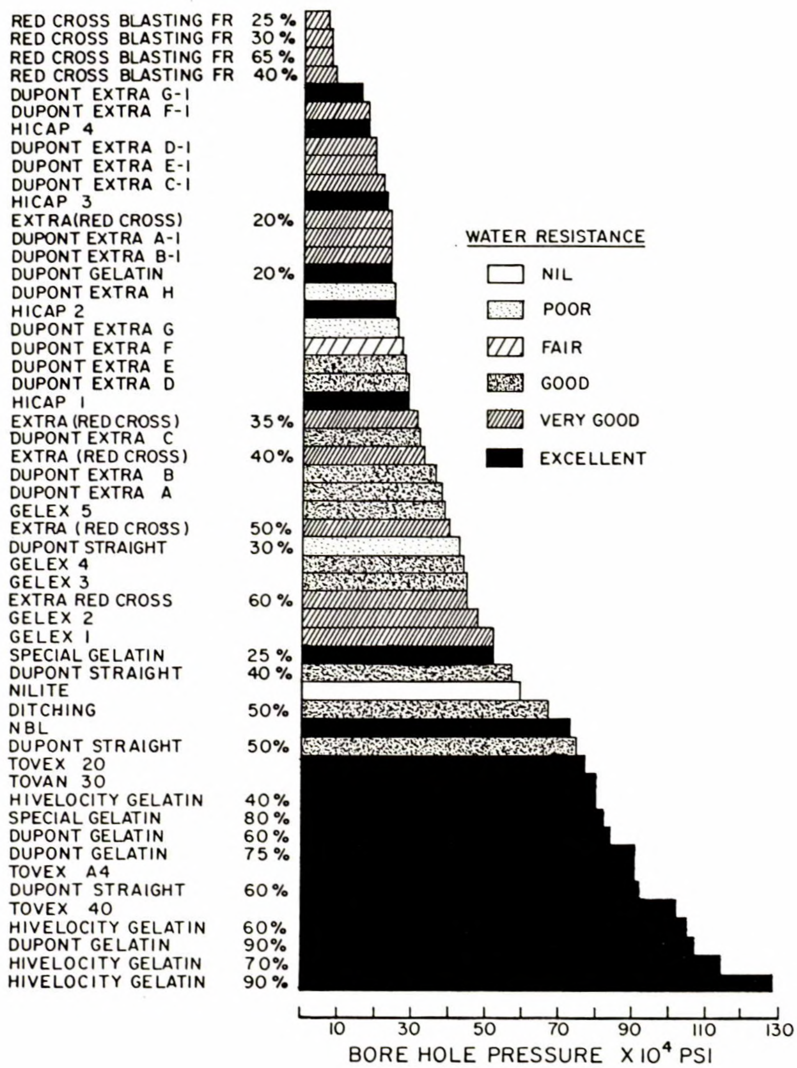

Fig 1(b) - Calculated borehole pressures for various Dupont explosives.

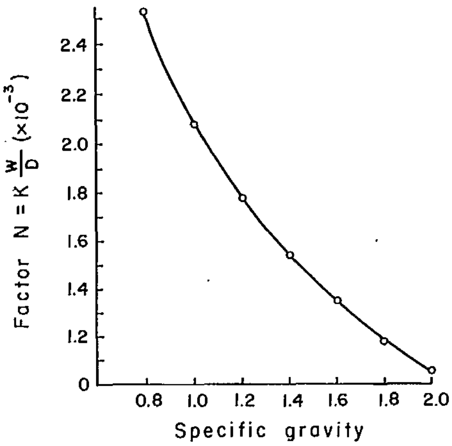

Fig 2 - Graph to determine constant, N, for use in equation 1.

AN/FO in granite cannot be compared as the limit of cracking from AN/FO charges was measured approximately 1/6 of the distance up the charge. It should also be noted that the rupture limits of Fig 3 are not necessarily applicable to other rock types as the rupture radius is largely dependent on rock strength and degree of jointing.

14. The lower the borehole pressure, the lower the level of ground vibration produced. For example, work by Larocque et al indicated that a confined charge of Geogel 60% would produce a level of ground vibration 2.5 times greater than a confined charge of Cigel B 70% of the same diameter (1). The borehole pressure produced by Geogel 60% is 2.5 times greater than that by Cilgel B 70%.

15. Low density explosives will produce low borehole pressures. The density of an explosive can be lowered by:

a. gassing (natural, mechanical, or chemical)

b. adding material containing entrapped air (eg,

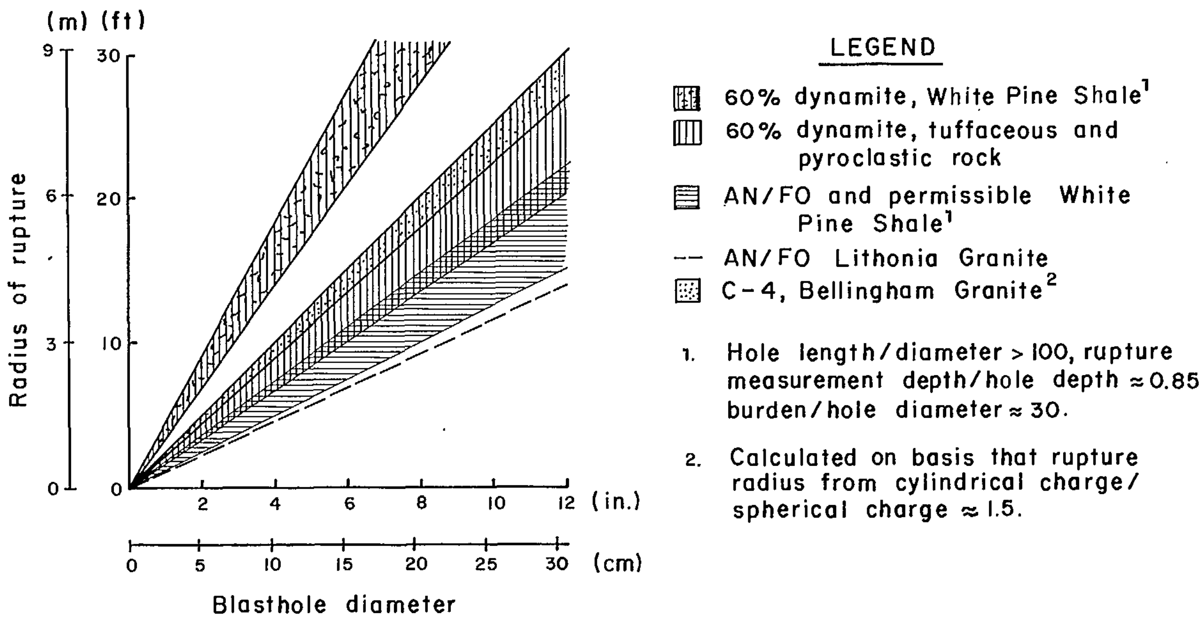

Fig 3 - Radius of rupture vs hole diameter for cylindrical charges of various explosives.

perlite, styrofoam, woodmeal, microballoons, hollow glass beads, etc)

Figure 4 illustrates the effect that lowering the density of AN/FO has on detonation velocity and hence on borehole pressure. These low density products can be economically bulk-loaded. They have great potential in control blasting because of the low borehole pressure produced.

16. Figure 4 illustrates the importance of charge diameter on borehole pressure. As the charge diameter is reduced below a critical size (approximately 4 in. (10 cm) for AN/FO), the detonation velocity and resulting borehole pressure are reduced. The detonation velocity of AN/FO is particularly sensitive to charge diameters where small diameter charges are involved but becomes stable at charge diameters larger than 4 in. This consideration is not important unless AN/FO products are being used in 2.5 in. (6.4 cm) to 4 in. (10 cm) diameter holes.

17. The borehole pressure created by an aluminized explosive cannot be calculated using equation 1. The velocity of detonation of the explosive is reduced due to the fact that the initial reactions with the aluminum are endothermic. However, outside of the detonation zone, the equilibria shifts with rapid formation of exothermic products. The borehole pressure is therefore higher than that predicted from calculations based solely on the velocity of detonation. While these explosives would not normally be recommended for wall control the borehole pressure of the basic slurry or dry mix systems can be estimated to increase by 2% for each percentage of aluminum added, up to an addition of 13% aluminum.

Decoupling and Decking

18. Borehole pressure, and hence backbreak, can be reduced by decoupling or decking of charges. Charges are decoupled when they do not touch the borehole wall. The ratio of the charge radius to the hole radius is a measure of the coupling of a charge. Charges are decked by separating portions of the explosive column by wooden or cardboard spacers or by taping charges onto primacord. Decking results in further decoupling of a charge. The net coupling ratio is:

$$C.R. = (\sqrt{C} \cdot \frac{r_c}{r_h}) \qquad \text{eq 2}$$

where

C = percentage of explosive column that is loaded

r_c = radius of charge

r_h = radius of borehole

Figure 3 illustrates the effect of decoupled and decked charges. The borehole pressure is drastically reduced by decoupling. The mathematical relationship is:

$$(P_b)_{dc} = (P_b)_c \cdot (C.R.)^{2.4} \qquad \text{eq 3}$$

where

$(P_b)_{dc}$ = borehole pressure for a decoupled and/or decked charge (psi)

$(P_b)_c$ = borehole pressure for the same type of explosive coupled with the borehole (psi), Fig 1

C.R. = coupling ratio, from equation 2

Figure 5 can be used to determine $(C.R.)^{2.4}$ if C.R. is known.

19. Reduction of borehole pressure by decoupling and decking of charges is important in control blasting. By ensuring that the borehole pressure is sufficiently low, crushing around boreholes on the final pit wall can be avoided and backbreak can be reduced. Vibrations can also be reduced.

Water in Blastholes

20. When a decoupled charge is surrounded by water, its effective strength on detonation increases. Field studies in granodiorite have indicated that water around decoupled charges can increase the level of ground vibration several times. If blastholes can not be kept dry, this becomes a consideration when designing a blast to protect buildings or tunnels from ground vibrations.

	Density (g/cc)	Explosive type
1	1.1	AN/FO, pneumatically loaded
2	1.0	AN/FO, pneumatically loaded
3	0.8	AN/FO, pneumatically loaded
4	0.4	AN/FO and Microballoons, pneumatically loaded
5	0.4	AN/FO and Microballoons
6	0.3	AN/FO and Microballoons
7	0.25	AN/FO and Microballoons
8	0.2	AN/FO and Microballoons
9	0.3	AN/FO and Perlite
10	0.2	AN/FO and Perlite

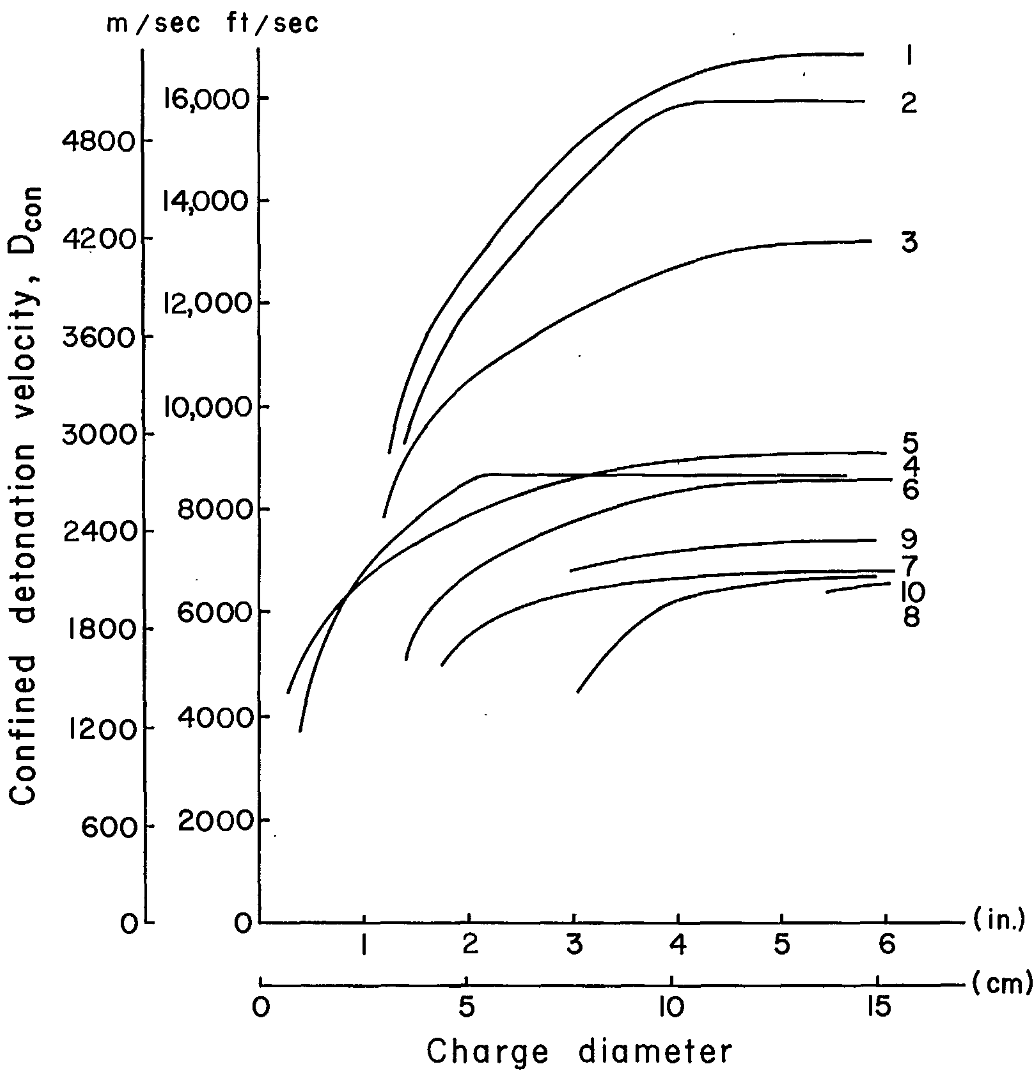

Fig 4 - Detonation velocity vs charge diameter for charges of AN/FO in aluminum pipe.

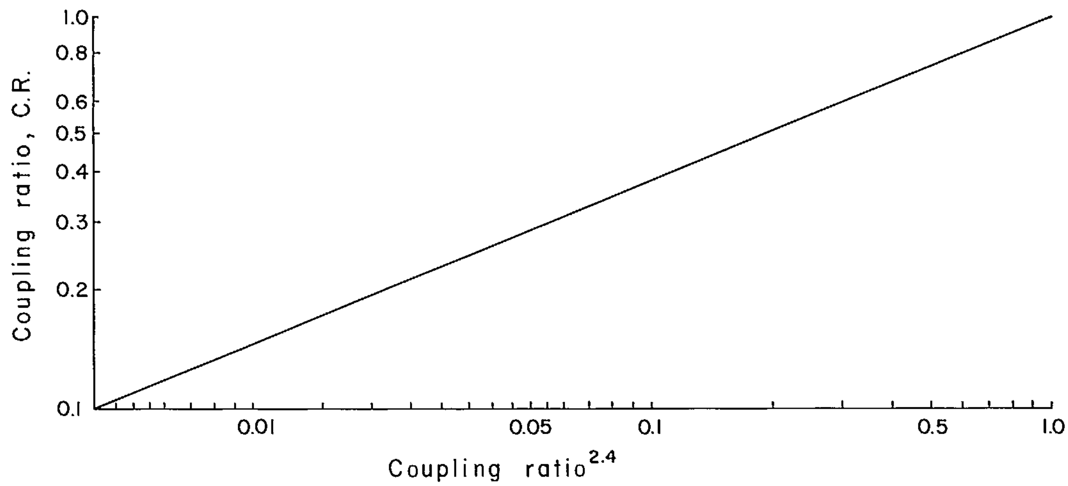

Fig 5 - Graph of coupling ratio vs (coupling ratio)$^{2.4}$.

Hole Diameter

21. Figure 3 illustrates the variation of rupture radius with hole diameter for a particular explosive and ratio of hole length to diameter. Doubling the hole diameter doubles the rupture radius, assuming that the coupling ratio is kept constant. This illustrates that small diameter drill holes will create less damage to final pit walls than larger holes. The radius of rupture for a given rock mass will be largely governed by local rock properties - particularly by rock strength and the nature and orientation of joints or faults. Open fractures oriented perpendicular to the outgoing shock wave front will cause a significant increase in the radius of rupture.

22. The use of small diameter blastholes also means using smaller hole spacings. A better wall surface results, but holes are more subject to wander or to caving in incompetent ground.

Burden and Spacing

23. Burden and spacing have no effect on borehole pressure, but they do have an influence on backbreak and face loose rock. To minimize these two possibilites, the spatial dispersion of charges at the pit perimeter should be as great as possible. This means using a lower hole spacing and charge per blasthole than for normal production blasting. The use of a larger number of smaller charges decreases the radius of fracture around blastholes and lessens the likelihood that large volumes of explosive gases from a single charge will be channelled into a joint or fracture, causing serious backbreak. Common sense dictates that when the borehole pressure is lowered to reduce backbreak in a blast, that the burden and spacing must also be reduced accordingly to maintain the same powder factor. Frequently, when blasting up to a row of pre-split or line-drilled holes it is necessary to also use a lower powder factor, in addition to reducing the burden and spacing. This will be discussed further in Appendix C.

24. A general rule of thumb to follow in buffer blasting is that the burden should not exceed the hole spacing. A burden/spacing ratio of 0.8 is commonly used. If the burden is too large compared with spacing, large muck and cratering around the blasthole may result. The charge is over-confined and cannot break to a free face. If the spacing is too great, a protrusion may be left on the wall between each pair of holes. In cushion blasting, a burden/spacing ratio of less than one is desirable to minimize backbreak. However, a large ratio can be used with simultaneous detonation of blastholes to provide a smoother face.

25. Reducing the burden and spacing of a blast

to reduce backbreak will not reduce the vibration level from the blast, providing the total charge weight per delay is not changed.

Delays and Sequencing

26. To minimize backbreak and vibration from blasting, the blast should be sequenced so that each row of holes can break to a free face.

27. The level of vibrations from an explosion depends largely on the charge weight per delay. For example, Fig 6 is a graph of peak particle velocity (a measure of ground vibration) versus scaled distance (shot point to monitoring point distance divided by the square root of the charge weight per delay). The graph is for cylindrical charges of Powerfrac 75% (length/diameter = 10) confined in granodiorite. At a shot-to-monitoring point distance of 100 ft (30 m), detonation of a 50 lb (23 kg) per delay will result in a peak particle velocity of 8 in./sec (20 cm/sec). At the same distance of 100 ft (30 m), detonation of a 10 lb (4.5 kg) per delay will result in a peak particle velocity of 2 in./sec (5 cm).

28. The number of charge delays also has an effect on the level of vibrations. Figure 7 shows that there are additive effects between delays. Due to the erratic firing times of smaller delays, 15 msec should be used as the minimum delay period. Otherwise, larger cumulative effects might result.

Drilling

29. Accurate drilling is important in control blasting - particularly in pre-splitting, line drilling and staggered hole depth techniques. Drill hole wander can result in scallop or backbreak break at the toe. Figure 8 illustrates minor backbreak caused by drill hole wander. Not drilling deep enough may result in backbreak at the toe. Inaccurate drilling is especially undesirable when large diameter blastholes are being used.

30. Some application of control blasting require that holes be drilled at an angle corresponding to that of the final pit wall. Some form of equipment which can drill back under itself such as a small diameter percussive drill must be used. Accurate drilling becomes important when blasting up to a joint or fault plane.

Collar and Subgrade Drilling

31. Depth of subgrade drilling and blasthole collar both affect crest fracturing.

32. Crest fracturing can be caused directly by the natural tendency of an explosive column to crater or break out towards the free surface. The depth of collar varies from 12 times the charge diameter for hard competent rock with static compressive strength > 30,000 psi (2.1×10^8 Pa) to 22 times the charge diameter for softer rock with static compressive strength approximately 15,000 psi (1.0×10^8 Pa to 30 times the charge diameter for soft or incompetent rock with static compressive strength approximately 5000 psi (3.4×10^7 Pa). Frequent open joints necessitate use of a larger collar since this type of rock is more apt to crater at the top (explosive has a large radius of rupture in jointed rock).

33. Subgrade drilling and blasting may fracture the rock composing the surface of an underlying bench, thereby weakening it and making it susceptible to crest fracture. It is recommended that no subgrade be used over the crest of a haul road or berm. Subgrade drilling in holes at the pit perimeter, adjacent to a haul road, will help promote drainage. Common practice is to use 3 ft (0.91 m) to 5 ft (1.5 m) of subgrade drilling or to set the depth of subgrade drilling equal to 7 times the charge diameter.

34. In horizontally bedded or jointed rocks it is often unnecessary to employ heavy toe loads or subgrade drilling. Under these conditions the rock has a larger radius of rupture and will break out more easily.

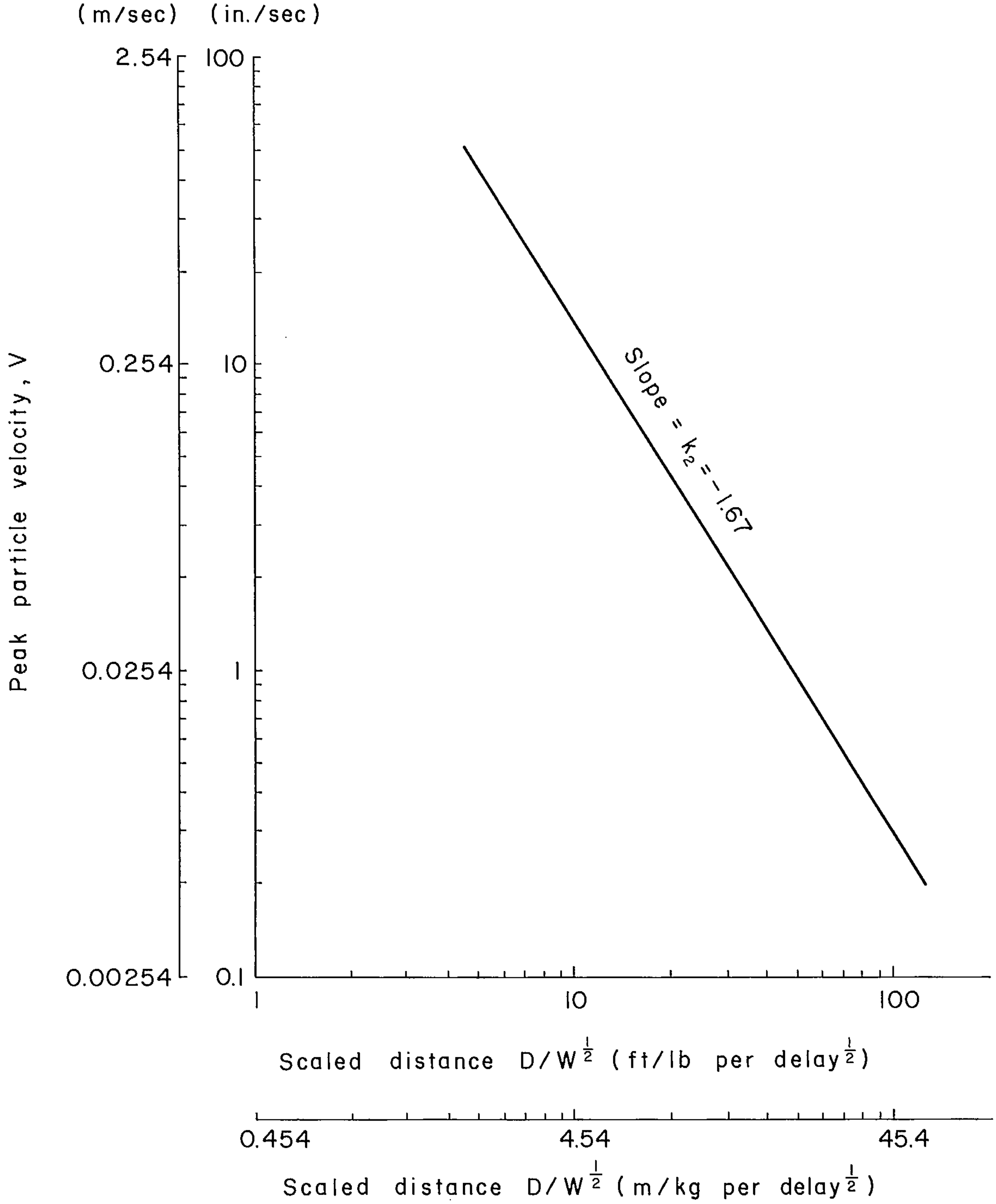

Fig 6 - Graph of peak particle velocity vs scaled distance for confined cylindrical charges of Powerfrac 75% in granodiorite.

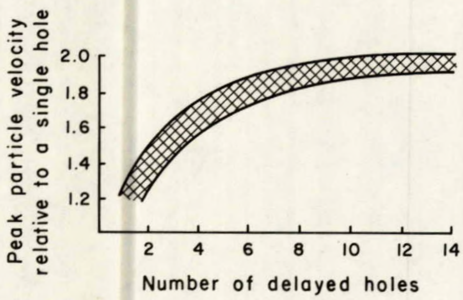

Fig 7 - Graph showing the effect of successively fired shot holes with 15 msec delays on peak particle velocity level.

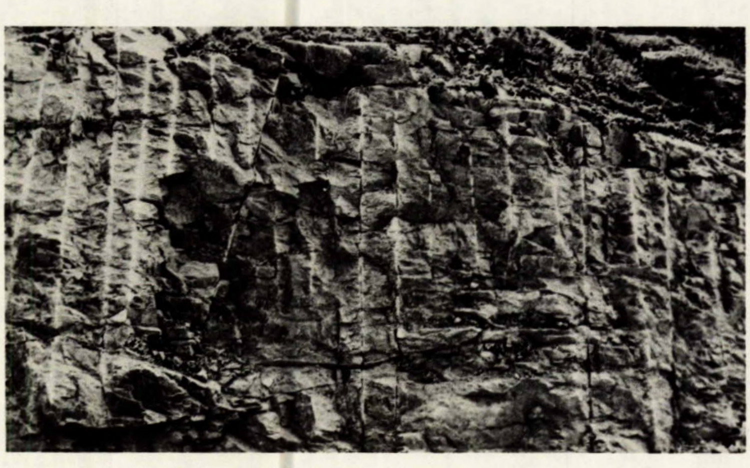

Fig 8 - Local backbreak due to drill hole wander.

INFLUENCE OF SITE CONDITIONS

35. Rock properties contribute significantly to the degree of success achieved by a control blast. Failure to consider rock properties when designing a blast may result in serious backbreak, crest fracture, face loose rock, or sliding of weak portions of the pit wall. Evaluation of rock properties is necessary when determining safe blasting limits to structures or underground opening.

36. The most important rock properties are: in situ dynamic rock strength; the nature, frequency and orientation of structural features; Young's Modulus of Elasticity; rock density; and longitudinal wave velocity.

In Situ Dynamic Rock Strength

37. Dynamic rock strength refers to strength of the rock when it is subjected to a changing load such as a ground shock wave.

38. In situ rock strength refers to the rock strength as measured in situ rather than in the laboratory. By making in situ measurements, the effects of weathering or of any observed structure on the rock strength can be assessed. If rock specimens were transported to the laboratory, the weakest portions would fall apart, so that only the strongest parts of the rock would get tested. The in situ dynamic strength of the rock depends on more than just the strength of the rock type. A strong rock type, (eg taconite), can be considerably weakened by weathering, groundwater, alteration, the presence of structures (eg frequent open joints, prominent bedding or foliation planes), or fractures due to previous blasting. Useful tests for determining compressive and tensile dynamic rock strengths for purposes of control blasting design are given in para 95.

39. Backbreak - crushing and radial cracking around the borehole - results when the ground stress from an explosive charge exceeds the in situ dynamic compressive strength of the rock mass. Extensive backbreak often results when heavily fractured or jointed rock is blasted using

the same powder factor, etc, that gave good results in competent rock. Figures 9 and 10 illustrate pre-split walls in the same rock type, both pre-split blasts being set up in the same way. However, much backbreak occurred on the wall shown in Fig 10 because of incompetent, highly

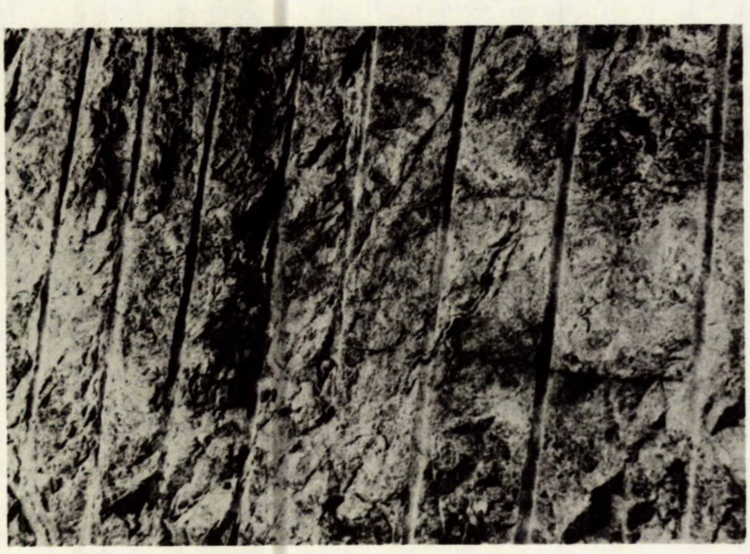

Fig 9 - A smooth clean pre-split surface in a competent rock.

Fig 10 - Backbreak and face loose rock on pre-split surface in intensely fractured rock.

fractured ground.

40. Backbreak or face loose rock due to structural features largely depends on the nature of the structure. Tight or infilled joints result in less backbreak than open joints. Figure 11 shows highly jointed rock which has been successfully pre-split aided by the natural strength of the joints. The chance of face rock being bumped out from the pit wall by nearby blasting operations is less if the joints are infilled and have some tensile strength.

41. The orientation of structures with respect to the final pit wall has a great influence on backbreak and face loose rock. When the structure is parallel to the final pit wall, a clean smooth face can be quite readily provided (Fig 12). The pre-split is designed to conform with the jointing. Problems can arise when structures are

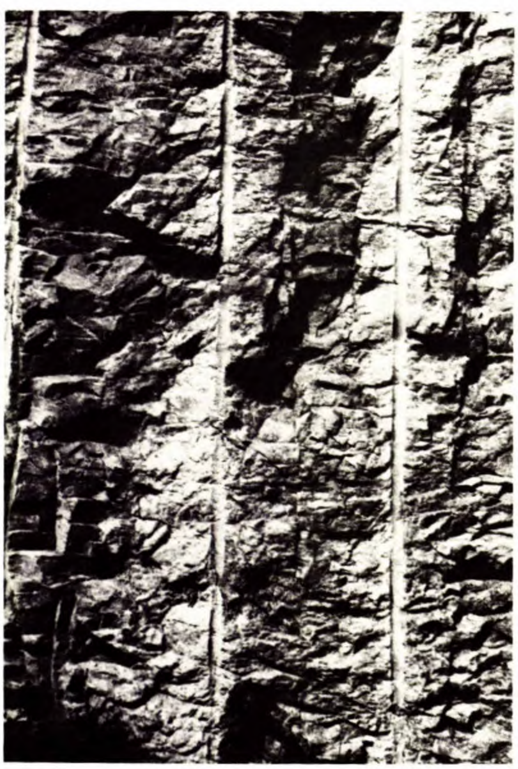

Fig 11 - A good clean pre-split surface in jointed rock.

undercut by the final pit wall. Figure 13 shows undercut joints which are almost parallel to the final pit wall; face loose rock and backbreak are evident. Figure 14 illustrates a shear zone which also fits into this category. Figure 15 shows steeply dipping joints which are at a 45 degree angle to the pit wall. The rock has been broken back about 6 ft (1.8 m) at the bottom of the photo. Figure 16 illustrates vertical joints which are almost perpendicular to the rock face. Some backbreak due to cross-jointing is evident, but not as much as in previous photos. In Fig 17 steeply dipping joints strike parallel to the rock face. There are no backbreak or sliding problems in this instance. Figure 18 illustrates tight flat-lying bedding which is perpendicular to the rock face. Structures with this orientation seldom affect the results of control blasting.

42. The frequency or density of structures has a major influence on backbreak, face loose rock, and crest fracture. Joints interfere with perimeter blasting when the joint spacing is less than the hole spacing. Comparing Fig 9 with Fig 19 illustrates how joints with a similar orientation but greater joint density, can increase the amount of backbreak. Crest fracture due to frequent jointing is a common problem (Fig 20). To counteract this, the collar height must be increased or the upper column load must be decreased.

43. To summarize, damage to pit walls at the site of a control blast (ie backbreak, crest fracture, face loose rock) is caused when the borehole pressure exceeds the in situ dynamic compressive strength of the rock. The nature, orientation, and frequency of structures can weaken a rock type, so that this damage occurs. Similarly, the rock may be rendered weak by weathering, groundwater or fracturing due to previous blasting.

44. Slabbing or spalling at a free face near a blast occurs when the stress due to the reflected ground shock wave exceeds the in situ dynamic tensile strength of the rock mass. The presence of joints, bedding or foliation planes parallel to a free face greatly increases the possibility of slabbing. Slabbing is a potential hazard where a

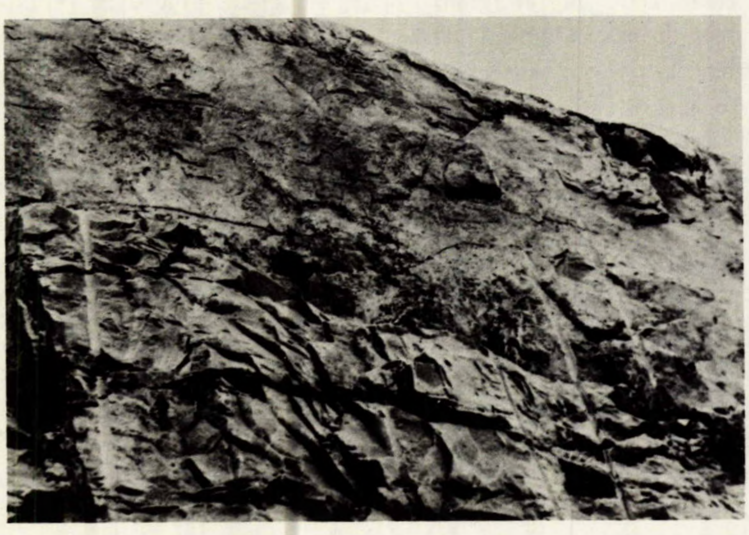

Fig 12 - Pre-split surface corresponds to major jointing oriented parellel to rock face. Note the clean smooth surface which results.

Fig 13 - Backbreak and face loose rock due to undercut jointing oriented almost parallel to pre-split line.

Fig 14 - Backbreak and face loose rock due to undercut fault oriented almost parallel to pre-split line. Blasting or weathering may cause face loose rock.

Fig 15 - Open joints oriented at 45° to rock face with some backbreak.

Fig 16 - Open joints oriented at 90° to rock face with some backbreak.

Fig 17 - Steeply dipping joints strike parallel to rock face but are not undercut, with no sliding or backbreak problem.

Fig 18 - Good pre-split surface in rock consisting of thin horizontal beds of quartzite.

Fig 19 - Frequent vertical jointing oriented at 90° to rock face with considerable backbreak.

tunnel is in close proximity to open pit blasts or where blasting operations in one pit are close to the walls of another pit.

45. Ground vibrations from blasting can have detrimental effects on nearby rock which has been weakened by jointing or faults, weathering, previous blasting, etc. In the case of joints, bedding, or intersecting joints ie, wedge-shaped structures, which have been undercut by pit slopes (Fig 21), vibrations from nearby blasting operations can supply the necessary force to cause the rock to slide along planes of weakness. Large slides can be triggered in this way. Figure 22 illustrates this type of structure. Where face rock has been weakened by groundwater, weathering, or previous blasting, vibrations from nearby blasting operations can cause face loose rock to fall.

Other Rock Properties

46. In regions where rock breakage is particularly undesirable, as in final walls the rock properties which relate to in situ strength are most important. The Modulus of Elasticity is a

Fig 20 - Example of crest fracture. Rock is broken back to major joints.

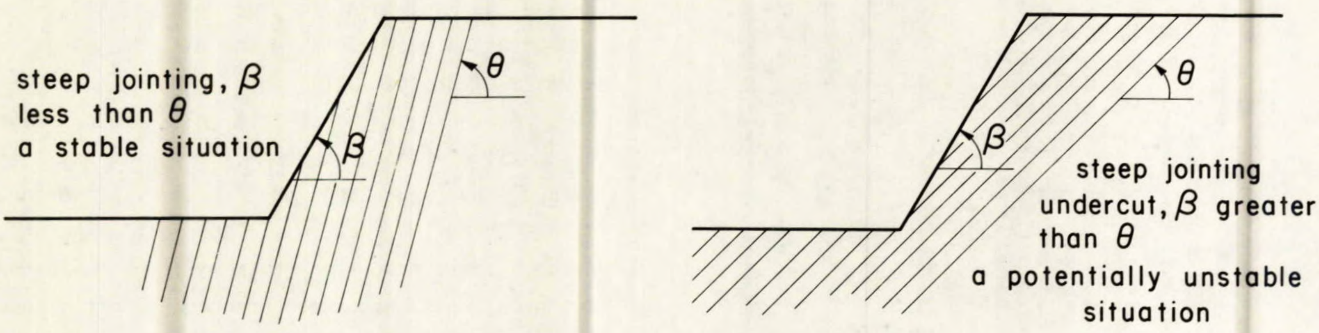

Fig 21 - Steep-jointing, stable and undercut.

Fig 22 - Steeply dipping major faults and joints which make possible a wedge type failure.

measure of the brittleness of a rock or its susceptibility to backbreak. Frequently, rocks which have a high Modulus of Elasticity also have a high compressive strength and so are harder to break. Similarly, rocks which have a higher longitudinal wave velocity are observed to be stronger. Rocks which are weakened by weathering, alteration, fracturing due to dense jointing or previous blasting have a lower longitudinal velocity. This observation provides the basis of seismic techniques for determining depth of overburden or broken rock, radius of rupture, jointing density, etc.

47. At greater distances from the blast the mine operator's concern shifts from backbreak control to preventing damage due to vibrations from blasting such as from spalling, sliding or falling of loose rock or damage to buildings or tunnels. Rock properties affecting attenuation of vibrations become of interest.

Table 1: Rock properties

Rock type location	Uniaxial compressive strength psi(Pa)	Tensile strength Brazilian psi(Pa)	Modulus of elasticity psi(Pa)	Poisson's ratio	Longitudinal wave velocity ft/sec(m/sec)	Density $lb/ft^3 (kg/m^3)$	Site factor K_2	M_2
Granodiorite Hinchinbrooke, Ont.	32,000 (2.2×10^8)	1,700 (1.2×10^7)	7.44×10^6 (5.1×10^{10})	0.33	20,000 (6,100)	167 (2.7×10^3)	160	-1.67
Qtz. carbonate gneiss Smallwood Mine	33,852 (2.3×10^8)	1,209 (8.3×10^6)	1.24×10^7 (8.5×10^9)	--	21,000 (6,400)	237 (3.8×10^3)	490 (av)	-2.93 (av)
Magnetite Carol Lake, Lab. (spherical charges)	--	1,800 (1.2×10^7) (Hopkinson bar test)	15.4×10^6 (1.1×10^{11})	0.30	21,000 (6,400)	--	--	-1.35
Granite gneiss Lithonia, Ga. (spherical charges)	30,000 (2.1×10^8)	450 (3.1×10^6)	9.2×10^6 (6.3×10^{10})	0.26	18,700 (5,700)	164 (2.6×10^3)	--	$\sim$-2.08
Magnetite pilot Knob, Mo.	42,000 (2.9×10^8) to 48,000 (4.0×10^8)	2,300 (1.6×10^7) to 2,800 (1.9×10^7)	12.0×10^6 (8.3×10^{10}) (dynamic)	--	15,000 (4,600) (bar velocity)	243 (3.9×10^3)	--	$\sim$-2.21
Shale White Pine, Mich.	29,900 (2.1×10^8)	820 (5.7×10^6)	5.4×10^6 (3.7×10^{10})	--	15,500 (4,700)	170 (2.7×10^3)	--	$\sim$-1.91

48. Figure 6 is a plot of peak particle velocity versus scaled distance for charges of Powerfrac 75% detonated in granodiorite. The equation of the straight line is

$$V = K_2 \cdot \left[\frac{D}{\sqrt{W}}\right]^{m_2}$$

where

V = peak particle velocity (in./sec)

$D/\sqrt{W}$ = scaled distance-(ft/lb$^{\frac{1}{2}}$)

K_2 = 750

M_2 = -1.67

K_2 and M_2 represent site factors. The value of K_2 depends upon the properties of the explosive charge and the impedance of the rock around the charge. The impedance is the product of the specific gravity and longitudinal wave velocity of the rock. The value of M_2 depends upon the properties of the rock between the charge and the monitoring point. The presence of fractures will result in lower values of longitudinal wave velocity and Modulus of Elasticity, and in greater attenuation of ground vibrations. Typical rock properties are shown in Table 1.

CONTROL BLASTING TECHNIQUES

Buffer Blasting

49. Buffer blasting is possibly the most simple method of control blasting and involves a modification to the last row of the main blast pattern. Modifications are limited to reduced burden, spacing, and explosive loads. The aim is to limit ground shock from the blast. The method is usually employed in conjunction with some other control blasting technique, such as pre-splitting, and its results are quite economical. Buffer blasting can only be used by itself when the ground is fairly competent. It may produce minor crest fracturing or backbreak but the amount of damage is less than would be produced by the main production blast if no control blasting was used at all.

Cushion Blasting

50. Cushion blasting involves splitting along the planned excavation limits, but the boreholes are detonated after the main production blast. The aim is to slash or trim excess material from the walls and to improve their stability.

51. Boreholes are drilled in a line along the planned excavation limits, loaded lightly, and exploded to remove the undesirable material. Holes are commonly 4 in. (10 cm) to 7 in. (18 cm) in diameter and spaced 5 ft (1.6 m) to 8 ft (2.4 m) apart. A reduced explosive load can be obtained by using low density bulk-loaded explosives thereby improving the economics of the method.

52. Cushion blasting gives similar results to pre-splitting. In competent rock, the exposed pit wall surface after blasting is clean and smooth and the backs of the boreholes are visible.

Pre-splitting

53. Pre-split blasting is by far the most successful and widely used control blasting technique. A row of closely spaced holes is drilled on the planned excavation limit and the holes are loaded lightly with suitable explosives and blasted before arrival of the main shock wave.

This is believed to create an open fracture necessary to dissipate the expanding gases from the production blast.

54. The aim of pre-splitting is to load the holes in such a way that for a particular rock type and spacing, the borehole pressure will split the rock yet not exceed its in situ dynamic compressive strength and cause crushing around the borehole. Because most explosives produce borehole pressures greater than 100,000 psi (6.9 x 10^8 Pa) although most rocks are not stronger than 60,000 psi (4.1 x 10^8 Pa), the borehole pressure must be lowered. As discussed in this section on control variables this can be done by using decoupled or decked charges or low density explosives.

55. Much pre-splitting is done using small diameter blastholes drilled with common percussion equipment. Hole diameters range from 2 in. (5.1 cm) to 4 in. (10 cm). Typical spacings would be 2 ft (0.61 m) to 4 ft (1.2 m), with a hole depth limit of 50 ft (15 m) to 60 ft (18 m). Explosive loads usually consist of gelatin or semi-gelatin dynamites in cartridge form, taped to primacord down-lines in a decked manner. No stemming is used.

56. More mines are now attempting to use large diameter holes and greater spacing for pre-splitting because of the economic advantages. Holes 9 in. (23 cm) to 10 in. (25 cm) are spaced 8 ft (2.4 m) to 12 ft (3.7 m) apart. Explosive loads consist of either a toe load or a combination of toe load and column load. AN/FO and/or slurries are commonly used.

57. Figure 9 illustrates a typical pre-split surface in competent rock. Note the clean smooth surface on which the backs of the boreholes are clearly visible. If the proper depth of collar is used, pre-splitting does not cause any significant crest fracture. Figure 23 shows the results of pre-splitting in blocky ground. To achieve best results, good drilling is essential.

Line Drilling

58. Line drilling is similar to pre-splitting because it involves the drilling of closely spaced, small diameter holes along the limit of excavation. The object is to create a plane of weakness to which the production blast will break. Unlike pre-splitting, the holes are not loaded or are loaded lightly with primacord. Holes of 2 in. (5.1 cm) to 3 in. (7.6 cm) diameter are spaced 2 to 4 hole diameters apart. This method produces the best surface - a smooth, clean face with no backbreak or crest fracture. Because of its high drilling cost, the method has not been commonly used in open pit work.

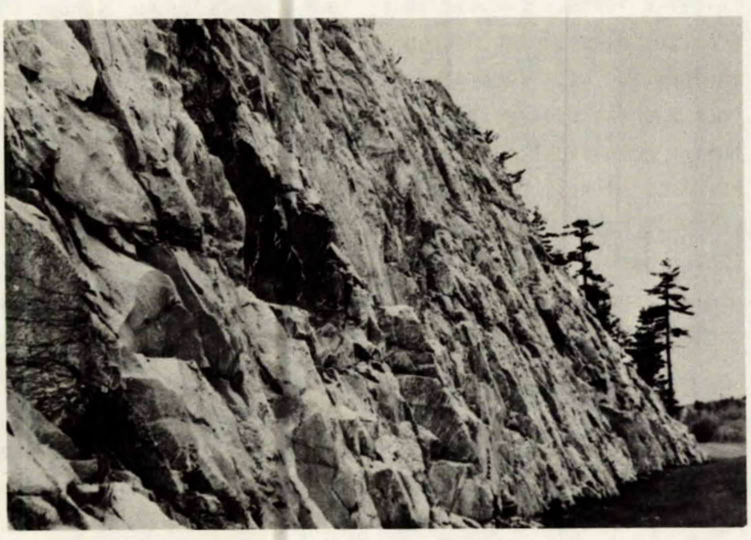

Fig 23 - A clean pre-split face in blocky ground.

COST AND BENEFITS

59. The need to optimize open pit operations today has resulted in a scale-up of operations. Larger blastholes and higher benches are used to increase the volume of ore broken and yet reduce time and cost. More powerful explosives are being used to achieve better fragmentation. Consequently, production blasts can have a devastating effect on pit walls, safety berms, haulage roads, or nearby structures. One cost of scaling-up operations, then, is that incurred in reducing this type of damage to an acceptable level.

60. The objective of control blasting is to reduce damage from blasting at minimum cost. Control blasting can result in a number of benefits as described below.

Benefits

61. The stripping ratio can be increased. Since backbreak is reduced or eliminated, the waste rock into which the backbreak may have extended will not have to be removed. For example, consider a conical-shaped pit, 500 ft (152 m) deep, with a desired bottom radius of 600 ft (183 m) and a slope angle of 45 degrees. The ore is chalcopyrite disseminated in granodiorite and has a density of 180 lb/ft^3(2.9×10^3 kg/m^3). Suppose that heavy powder loads resulted in 6 ft of overbreak at the pit limit. The total amount of extra rock to be removed would be:

$$\Delta V = \gamma(V_1 - V_2)$$

Where

V_1 = volume of pit with extra waste rock removed

V_2 = volume of desired pit

γ = rock density

Therefore, for this example,

$$\Delta V = \left\{\frac{3.14 \times 500}{3}\left[(1106)^2 - (606)^2\right] - \frac{3.14 \times 500}{3}\left[(1100)^2 - (600)^2\right]\right\} \times 180$$

$$= 5.6 \times 10^8 \text{ lb}$$

$$= 2.8 \times 10^5 \text{ tons } (2.5 \times 10^8 \text{ kg}) \text{ (or 0.7\% of the total pit volume)}$$

62. Mechanical support, scaling, and secondary blasting costs can be reduced by control blasting. Use of these methods is sometimes complicated by loss of access to pit walls due to further mining or to infilling of safety berms with material from minor slides. With control blasting, this is no problem.

63. The berm interval can be increased because the pit walls and berms are more sound. This in effect, increases the slope angle of the pit thereby increasing available ore reserves. In the conical-shaped pit in the previous example a 1° increase in the slope of the pit wall will result in a 9.9×10^5 ton rise in ore reserves. This represents a substantial amount of money.

64. Costly damage to buildings or tunnels can be prevented by controlling vibrations from blasting. Reducing ground vibrations from blasting in the final pit wall may eliminate the possibility of failure of a potentially unstable portion of the wall, thereby avoiding a costly clean-up.

65. Safety is improved when control blasting techniques are used. Pit walls are smoother and less fractured, so rock falls are reduced. Also, in the event of a rock fall, safety berms will be more effective in catching the rock because they have not been narrowed due to overbreak or crest fracture from production blasting.

Cost

66. The additional expense of control blasting is due to drilling smaller blast patterns. Methods employing small hole spacings and hole diameters are generally more expensive than for large rotary-drilled holes, but they produce a better rock face. The reason for this, as mentioned in the section on burden and spacing is that the explosive charge is distributed more evenly.

67. The actual cost of control blasting is the cost over and above that of breaking the same volume of ground by using straight production blasting. In the case of pre-splitting or line drilling, the true cost of the technique must therefore include the cost of the buffer row used in connection with that technique. Control blasting costs are expressed in terms of dollars per square foot of final pit wall surface.

68. The following is a discussion of the cost of each control blasting technique. Costs for three rock types and for both percussive - and rotary-drilled holes are included for purposes of comparison. Cost analyses are based on drilling costs shown in Fig 24 and on blast layouts determined by the methods outlined in Appendix C.

Buffer Blasting

69. Buffer blasting is the cheapest form of control blasting. The powder factor is essentially the same as for production blasting so explosives costs are the same. Drilling costs are slightly higher because of reduced burden and spacing. Coupled charges produce high borehole pressures, which are usually greater than 300,000 psi (2.1×10^9 Pa). This is not considered excessive as breaking of the rock is desirable for buffer blasting.

70. Typical costs are shown in Fig 25. For taconite and copper ore, costs are lowest for large diameter rotary-drilled holes followed by small diameter and large diameter percussive-drilled holes respectively. In asbestos ore, small diameter percussive-drilled holes cost approximately the same as large diameter percussive-drilled holes.

Cushion Blasting

71. Figure 26 illustrates typical cost curves for cushion blasting in three rock types. Costs are minimized in all three by using large diameter rotary-drilled holes. In taconite, costs are higher for small diameter percussive-drilled holes than for large diameter ones. The opposite is true for copper ore. It is interesting to note that cushion blasting in asbestos ore appears to be cheaper than production blasting. The reason is that the powder factor for cushion blasting is less than half of that for production blasting.

Pre-splitting

72. Cost curves for pre-splitting in three rock types are shown in Fig 27. Since it is necessary to design a pre-split blast with a buffer

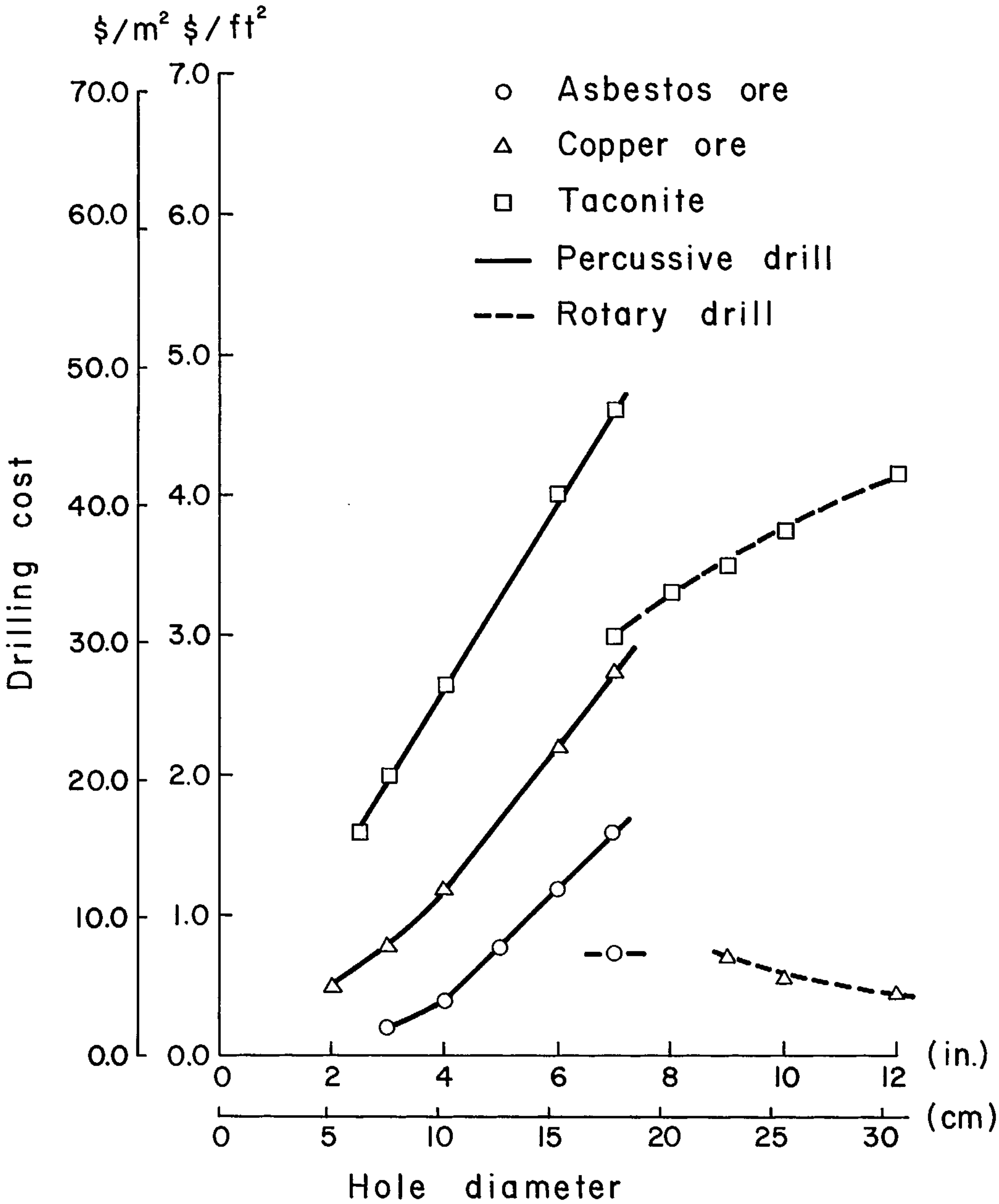

Fig 24 - Drilling costs used for cost estimates of control blasting techniques.

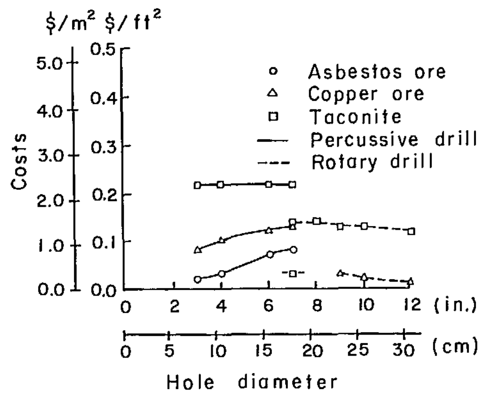

Fig 25 - Cost of buffer blasting (minus cost of production blasting using same hole size) vs hole diameter for various rock types.

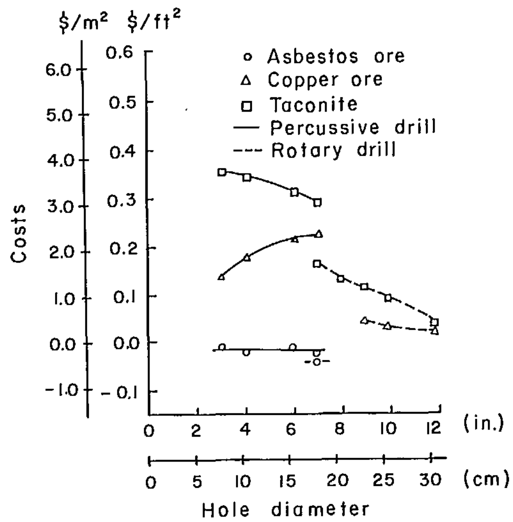

Fig 26 - Cost of cushion blasting (minus cost of production blasting necessary to break same volume of ground) vs hole diameter for various rock types assuming an explosives cost of $0.25/lb.

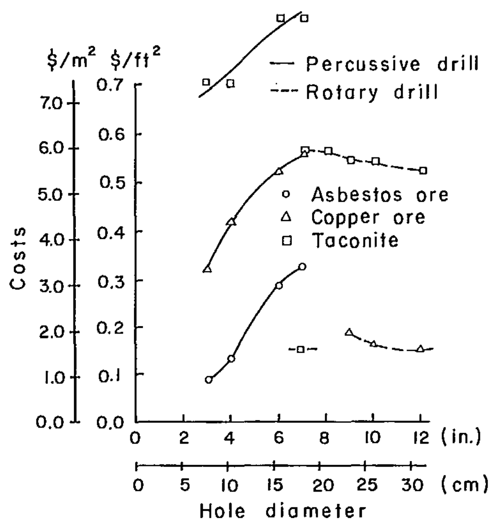

Fig 27 - Cost of pre-splitting and buffer blasting (minus cost of production blasting necessary to break same volume of ground) vs hole diameter for various rock types assuming an explosives cost of $0.25/lb.

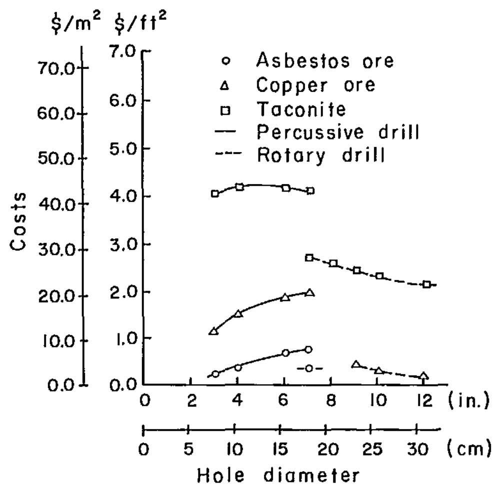

Fig 28 - Cost of line drilling and buffer blasting (minus cost of production blasting necessary to break same volume of ground) vs hole diameter for various rock types assuming an explosives cost of $0.25/lb.

row in front of the pre-split line, the cost of buffer blasting has been added to the basic pre-splitting cost. In doing this, it was assumed that hole diameters are the same in both.

73. Pre-splitting costs in taconite and copper ore are less for large diameter rotary-drilled holes than for percussive-drilled holes. Small diameter percussive-drilled holes cost less to use than those of larger diameter. In asbestos ore, costs are minimized by using small diameter percussive-drilled holes, with large diameter rotary-drilled holes and large diameter percussive-drilled holes costing more respectively.

74. As with cushion blasting, costs could be reduced by increasing hole spacings and using a higher loading density or greater coupling ratio. The reduction in drilling cost is greater than the increase in explosives cost. However, since the costs in Fig 27 were determined using the maximum borehole pressure which would not crush the rock, use of a greater coupling ratio, ie, higher loading density, would cause the rock around the borehole to be crushed.

Line Drilling

75. Cost curves for line drilling in three rock types are shown in Fig 28. Line drilling is the most expensive technique because it is largely a function of drilling costs. Cost figures were determined using constant hole diameter to hole spacing ratios for the various rock types. As with pre-splitting, the cost of buffer blasting has been added to the basic cost of line drilling.

76. Costs in taconite and copper ore are less for large diameter rotary-drilled holes than for percussive-drilled holes. In copper ore, costs decrease with increasing size of percussive-drilled holes, whereas in taconite the cost does not change appreciably. In asbestos ore, costs are minimized by using small diameter percussive-drilled holes. Large diameter rotary-drilled holes and large diameter percussive-drilled holes are more expensive respectively.

Cost Comparison of Control Blasting Methods

77. Line drilling costs are much higher than those of other methods. Ranked in order of decreasing cost, the other methods are: pre-splitting, cushion blasting and buffer blasting.

78. With the exception of asbestos ore, costs are lowest for large diameter rotary-drilled holes. Costs decrease slightly for large hole sizes, eg, 10 in. (25 cm) hole compared with 7 in. (18 cm) hole. In taconite and copper ore, small diameter percussive-drilled holes are second lowest in cost. However, in asbestos ore the opposite is true, and methods employing small diameter percussive-drilled holes are cheapest. In all three rock types, costs for methods using percussive-drilled holes generally increase with increasing hole size.

Additional Costs

79. There are several additional factors which can influence the cost of control blasting. Small diameter holes may cave in poor ground. This necessitates costly redrilling of holes. This can be a problem with long slash or cushion blast holes. In poor ground, it may prove as costly to use small diameter precussive-drilled holes which may cave as it is to use larger diameter percussive-drilled holes.

80. Equipment for control blasting is often used for some form of production drilling as well, so its cost has already been written off. Where special equipment must be purchased for control blasting as for example a small diameter percussive or small rotary drill, the additional cost must be included in any cost analyses.

DEVELOPING A CONTROL BLASTING PROGRAM

EXPLORATION-EVALUATION STAGE

81. Most work done at the exploration-evaluation stage consists of diamond drilling. Logging of core and drillholes can also give useful information for setting up a control blasting program.

82. When possible, the following details should be noted when logging core:

a. principal rock types, presence of weathering or alteration
b. nature, frequency, orientation of structures such as joints, faults, foliation, bedding
c. location or orientation of major discontinuities such as faults, shear zones, contacts between rock types (particularly between ore and country rock)
d. location of porous zones or aquifers.

Much of the above information depends on ability to get oriented core samples. If a borehole camera is available, orientation of structures, location of rock boundaries and aquifers, and observations of the nature of structures can be obtained directly.

83. The rate of inflow of water into each diamond drill hole should be estimated. In addition, the depth to the water table should be measured for each hole.

84. Core samples of the principal rock types and any weathered or altered sections of core should be tested for the following properties:

a. static compressive strength
b. static tensile strength
c. Young's Modulus of Elasticity
d. longitudinal wave velocity.

The results of this information should be compiled in a series of core logs, such as the one shown in Fig 29. This information will be used in slope design as well as to aid in the control blasting program.

MINE DESIGN STAGE

85. At this stage, a preliminary pit design is drawn up according to the location of known mineable ore reserves.

86. Using the information gathered from logging of core and drill holes, the pit is divided into areas according to:

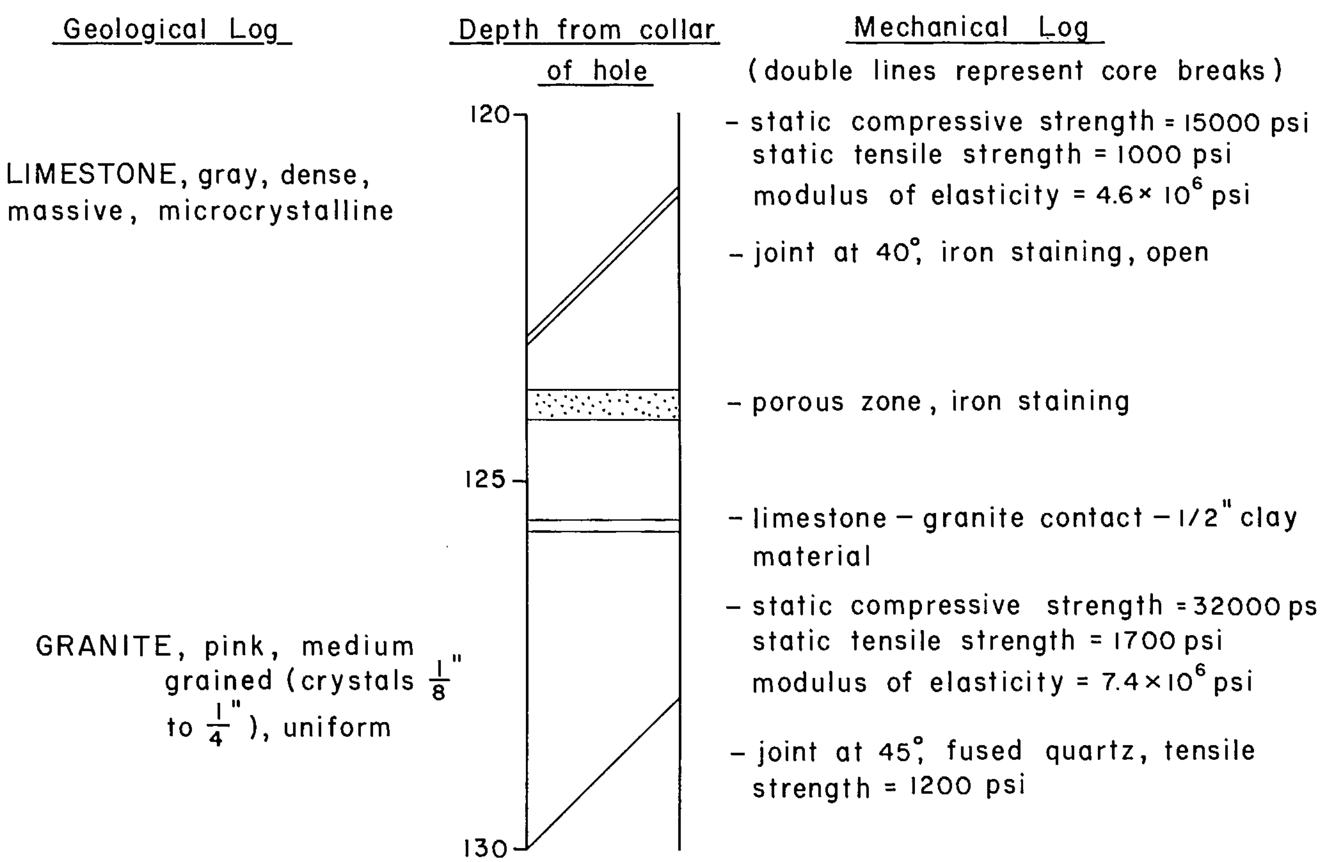

Fig 29 - Sample core log.

a. major structures undercut by the pit wall
b. rock type and rock condition
c. areas where buildings, crusher, tunnels, etc, will be located.

Major Structures

87. If major joints, bedding, faults, or wedge-shaped structures are undercut by the proposed pit wall, a calculation should be done to determine the factor of safety against sliding. Three methods can be used if the factor of safety is unacceptably low: (1) monitor slope movement once mining has begun, (2) use artificial support, or (3) redesign the pit slope so that it corresponds to the dip of the undercut structure.

88. The first two alternatives are treated in the manual chapters on design, artificial support and monitoring in this manual. If the third alternative is chosen, a control blasting method must be selected. Buffer or cushion blasting can be employed if the structure is open; if it dips at less than 60°, a staggered drill hole technique can be used. Pre-splitting can be used if the structure is tight.

89. For example, preliminary pit design may call for a slope of 70 degrees. Core logging has shown that in one area of the pit, a major fault runs parallel to the proposed pit wall, but has a gentler dip. Calculations show that the proposed pit wall will not be stable. By utilizing control blasting, the pit slope can be redesigned to correspond with the dip of the structure providing the location of the structure at depth is known.

Rock Type and Rock Condition

90. The pit must first be divided into areas

according to rock type and rock condition. The object is to design a blast to minimize backbreak. The material in Appendix C can be used to design the blast layout for each area. Once the mine is operating, more information on rock type, etc, will be available and better control blast layouts will be possible.

91. If the mine operator wishes to get a control blast layout that he knows will work, he can perform some preliminary blasting using small diameter holes. The blasts are designed according to Appendix C. Blast patterns are drilled off at various angles to the major jointing. The loading density for each pattern can be varied every 3 or 4 holes from slightly above to slightly below the suggested loading density. The patterns are fired off. The loading density which gives the best results is selected for each pattern. To scale up the blast patterns, hole spacing and burden are increased by the same percentage that the hole diameter is to be increased. The loading density is increased by the square of the percentage increase in hole size.

Protection of Buildings and Underground Openings

92. If crusher, buildings or tunnels are to be located near the pit, vibrations may have to be controlled to prevent damage. Similarly, if surface blasting operations are to be located near the pit, vibrations during excavations may have to be controlled to prevent damage to nearby pit walls. Any necessary changes in either production or control blasting should be determined. This will enable more accurate cost estimates to be made. The section of the chapter treating ground shock damage outlines what type of damage can occur and how to design a blast or locate a structure to prevent damage. Appendix B explains how to design a blast to minimize vibration from blasting.

93. Once the pit design has been arrived at, and control blasts have been designed for various areas of the pit, accurate cost estimates can be made.

MINE RE-DESIGN

94. Most operating mines are at this stage. Once production has started, the location of faults, weak ground and rock types is much more evident. The following investigations should be carried out at this stage:

a. in situ testing of rock types
b. location of water table
c. determination of nature, spacing, and orientation of structures
d. observation of backbreak from production blasting near pit limits.

95. In situ testing of rock types consists of determining the dynamic tensile strength and dynamic compressive strength of the rock mass. The dynamic compressive strength is determined by setting off charges having various borehole pressures ranging upwards from the static compressive strength of the rock. The burden/hole diameter ratio should be roughly the same as the burden/hole diameter ratio for the designed blast, since the apparent compressive strength of the rock varies with the burden/hole diameter ratio. The in situ dynamic compressive strength of the rock mass for a particular burden/hole diameter ratio will be equal to the maximum borehole pressure which does not cause crushing around the borehole. The dynamic tensile strength is used mainly in pre-splitting calculations. To determine its value, drill off several sets of holes at various spacings. The largest spacing which still allows a good crack to form between the pre-split holes is then substituted back into equation C-1 to determine an accurate figure for the dynamic tensile strength. A ballpark figure for spacing can be determined by substituting values for T from Table C-1 and borehole pressure or compressive strength of the rock for an infinite burden/hole diameter ratio in equation C-1. Values for T also can be estimated as 1/15 of the compressive rock strength as measured in the laboratory at high confining pressure, ie, triaxial testing.

96. The observation of backbreak from a production row near the pit limit is used to calculate the position of the buffer row. If stereophotos are taken of the bench face at this location they will approximate the type of structure that will occur at the pit limit.

97. For a mine which has not been using control blasting, these tests will provide the basic information necessary to design a control blast (Appendix C). For a mine which is already using control blasting, this information may help to modify control blasting methods and result in more accurate cost analyses.

98. Pit operators in the past have found that developing an effective control blasting program is mostly a trial-and-error process. A blast is set up one way and if this does not give the desired results, hole spacing or powder load, etc, is changed. Given a certain problem, such as backbreak from a pre-split line, which variables should be adjusted so that good results can be achieved as quickly as possible? - Spacing? Explosives load? Hole size? Table 2 lists a number of common problems and the variables which

Table 2: Variable to be used for solving problems at the mine re-design stage

Problem	Problem cause	Solutions
Backbreak throughout wall (no boreholes showing)	(a) buffer row overloaded or too close; (b) control blast may be overloaded	(a) move buffer row further from pit limit, reduce borehole pressure of buffer charge, use 15 msec delay between buffer charges (if not already being done) (b) increase hole spacing or decrease powder load by decoupling, decking or using cushion or pre-split holes
Backbreak around boreholes	borehole pressure greater than in situ dynamic compressive rock strength	decouple or use deck charges in cushion or pre-split holes, decrease burden (for cushion blasting)
Backbreak between boreholes	buffer hole too close	move buffer row back
Jointing interferes between blastholes	(a) spacing too great (b) burden insufficient (c) delays between perimeter holes too large	(a) reduce spacing and powder load (b) make burden larger than spacing (c) detonate holes on perimeter row simultaneously
Very poor fragmentation at pit limit or blast fails to break to pre-split or line	buffer row too far from pit limit	decrease distance from buffer row to pre-split or line drilled holes
Crest fracture	collar insufficient or rock exceptionally weak, fractured or weathered, at crest	increase the height of collar, eliminate subgrade in drill holes overlying the crest of a berm, use spacers in the upper portion of the explosive column, drill small diameter guide holes (10 (3 m) - 20 (6 m) ft deep)

can be changed to solve the problem.

99. Many perimeter blasts have failed to give good results because the holes adjacent to the perimeter were overloaded. Backbreak on the pit wall occurred not because the perimeter blast holes were overloaded, but because the adjacent row of holes was too close, destroying whatever wall surface the perimeter blast might have formed. How can you tell whether the perimeter blast or the production blast was overloaded? First, observe the rock face in question. Crushing or cratering around the borehole indicates an overloaded perimeter blast. Equal backbreak throughout the wall or immediately behind blastholes of the row immediately in front of the pit perimeter suggests that this buffer row or production row in the event that the buffer row is to provide the final pit wall surface, was overloaded. Check to see if the buffer to perimeter distance agrees with that calculated by the method in Appendix C. If the calculated distance exceeds the distance used, then the production row was overloaded. The cheapest way to decrease backbreak in this case is to move the buffer row back from the perimeter. Find this distance using equation C-3. Reducing the explosive load will decrease backbreak as well. Use equation C-3 for this, setting D_{BUF} equal to the buffer row to pit perimeter distance and solving for $P_{b_{BUF}}$. Since you already know the type of explosive charge in the buffer row, find the coupling ratio that you should use from equation 3. If, on the other hand, the perimeter blast was overloaded, reduce the explosive load. Increasing the hole spacing will not be effective unless the major backbreak occurs between the blastholes.

GROUND SHOCK DAMAGE

Buildings and Equipment

100. Buildings, crushers, or electrical instrumentation near pits are commonly subjected to ground shock from blasting. Vibrations from blasting can cause costly damage if proper care is not taken to locate buildings far enough away or to design blasts with minimum ground vibration.

101. A number of studies have been done to evaluate damage criteria for buildings subjected to ground shock from blasting. It has been shown that peak particle velocity is closely associated with building damage. For example, the onset of building damage corresponds to a peak particle velocity of 2 in./sec (5.1 cm/sec) in any direction. Table 3 shows vibration levels at which various types of damage occur to equipment and buildings.

102. Designers for mines in the design stage will want to know where buildings can be safely located; those for mines in the re-design or operating stage will wish to know how to design a blast to protect buildings already located near the pit.

103. The first step is to determine the safe scaled distance between the blasts and building, etc, ie, the scaled distance at which the peak particle velocity becomes less than 2 in./sec (5.1 cm/sec). The safe scaled distance will vary, depending upon charge characteristics and the rock type. Once the safe scaled distance is known, the maximum charge weight per delay can be calculated. Two methods can be used to find the safe scaled distance:

a. assume a value of 50 $ft/lb^{\frac{1}{2}}$ ($23\ m/(kg)^{\frac{1}{2}}$)

b. monitor blasts using particle velocity gauges (Appendix D)

104. In the first method, a safe scaled distance of 50 $ft/lb^{\frac{1}{2}}$ ($23\ m/kg^{\frac{1}{2}}$) is assumed for

Table 3: Type of damage related to the peak particle velocity in the ground waves from blasting

Type of structure	Type of damage	Peak particle velocity threshold at which damage starts in./sec	cm/sec
Rigidly mounted mercury switches	trip out	0.5	1.3
Houses	plaster cracking	2	5.1
Concrete block as in a new house	cracks in blocks	8	20.
Cased drill holes	horizontal offset	15	38
Mechanical Equipment Pumps, Compressors	shafts misaligned	40	100
Prefabricated metal building on concrete pads	cracked pads, building twisted and distorted	60	150

buildings. Work by the U.S.B.M. has shown that this will be a safe scaled distance for all rock types and unconfined explosive charges. If the charges are to be confined or if the building is located in overburden, a safe scaled distance of 100 $ft/lb^{\frac{1}{2}}$ (45 $m/(kg)^{\frac{1}{2}}$) should be assumed.

105. The maximum charge weight per delay is found by substituting values of safe scaled distance, S.S.D., and blast-to-building distance, S.B.D., into the equation

$$S.S.D. = \frac{S.B.D.}{\sqrt{W}} \qquad \text{eq 4}$$

where S.S.D. = safe scaled distance, ft/(lb per delay)$^{\frac{1}{2}}$

S.B.D. = safe blasting distance, or blast-to-building distance (ft)

W = maximum charge weight per delay (lb per delay); delay must be at least 15 msec.

The blast-to-building distance should be measured from the closest charge to the building.

106. In the second method, blast vibrations are monitored as blasting operations approach the structure which must be protected. Initially, the scaled distance from blast to building should not exceed 50 ft/(lb per delay)$^{\frac{1}{2}}$ (23 m/(kg per delay)$^{\frac{1}{2}}$) in rock or 100 ft/(lb per delay)$^{\frac{1}{2}}$ (45 m/(kg per delay)$^{\frac{1}{2}}$) in overburden. The monitoring gauges should be placed at approximately the same orientation with respect to the blast layout as the structure to be protected will be oriented with respect to blasts close to it. The distance from blast to monitoring gauges should be such that the peak particle velocities obtained are of the same

order of magnitude as those which would damage the structure. When 3 or 4 blasts have been monitored, the peak particle velocities are plotted against the corresponding scaled distances (Fig 30). A best fit line is drawn through the points. The scaled distance which corresponds to 2 in./sec (5.1 cm/sec) is read off the graph. The maximum charge weight per delay is found using equation 4. The blast-to-building distance is measured from the building to the blasthole (in the next blast pattern) which will be closest to it. The next blast is set up using this maximum charge weight per delay. The blast is fired and the peak particle velocity is recorded. The peak particle velocity is plotted on the graph at the appropriate scaled distance. A new best fit line is drawn through the points. The location of the next blast is decided upon. Substituting the new blast-to-building distance and the safe scaled distance found using the new best fit line into equation 4, the maximum charge weight per delay for the next blast is determined. The procedure is repeated as blasting approaches the building. Blasts can be brought closer to buildings using this method than if the conservative safe scaled distances of 50 or 100 ft/(lb per delay)$^{\frac{1}{2}}$ (25 or 45 m/(kg per delay)$^{\frac{1}{2}}$) were used.

107. Methods for minimizing vibrations are outlined in Appendix B.

108. Allowing a safety factor of 50% in the safe scaled distance will eliminate the risk of additive effects from delays.

Underground Openings

109. Underground openings are sometimes located near open pit blasting operations and must be protected from possible damage. In the case of an ore pass, for instance damage could result in a costly delay in production or stop it entirely.

110. The first step to ensure integrity of the underground opening is to establish the peak particle velocity at which tensile slabbing or spalling the most common form of damage occurs. To do this, the tensile strength of the rock must be known. The static tensile strength as determined in the laboratory will give a figure far too low. Procedures to determine the in situ dynamic tensile strength of a rock has been described. Or alternatively, the dynamic tensile strength can be estimated as 1/15 of the compressive strength as measured in the laboratory at high confining pressure by triaxial test. The dynamic tensile strength as found using a modified Hopkinson Bar apparatus as described by Larocque et al, (1) is more representative, but the testing procedure and equipment are more complex. The peak particle velocity which will cause spalling is found using the equation

$$V = \frac{1728\ S_T}{\rho_M\ C_\ell} \qquad \text{eq 5}$$

where

V = peak particle velocity (in./sec)

S_T = dynamic tensile strength of the rock mass (psi)

ρ_M = mass density of the rock ($lb.sec^2/ft^4$)

C_ℓ = longitudinal wave velocity in the rock (ft/sec)

111. The rest of the procedure is basically the same as that used for protecting buildings. Initial values of 25 and 50 ft/(lb per delay) (11 and 23 m/(kg per delay)) should be used as safe scaled distances for unconfined and confined charges in rock (or charges in overburden) respectively. A safety factor of 50% should be included in the safe scaled distance to eliminate any risk of cumulative effects of delays.

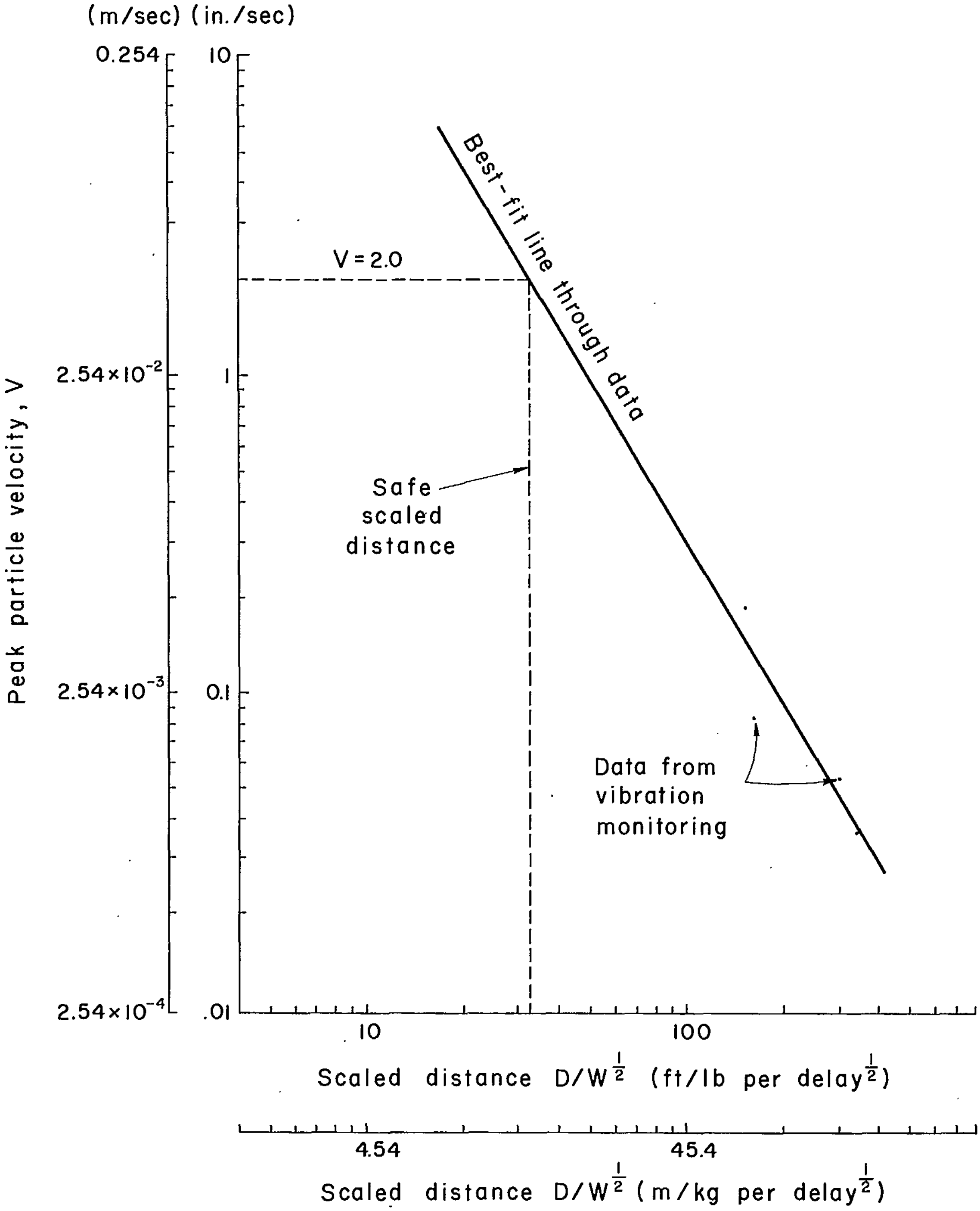

Fig 30 - Sample graph for determining the safe scaled distance for blasting near buildings, using confined charges of Powerfrac 75% in granodiorite.

REFERENCES

1. Larocque, G.E., Sassa K., Darling J.A., Coates, D.F. "Field blasting studies"; Proceedings of the Fourth Rock Mechanics Symposium, Ottawa, March 29-30, 1967; Mines Branch, Department of Energy, Mines and Resources; p. 169-203; 1967.

BIBLIOGRAPHY

Bauer, A. "Open pit explosives, drilling and blasting"; Mining Engineering Department, Queen's University, Kingston, Ont.; 168 pp; 1973.

Bauer, A. and Calder, P.N. "The influence and evaluation of blasting on stability"; Stability in Open Pit Mining; Ed. C.O. Brawner and V. Milligan, First International Conference on Stability in Open Pit Mining; the American Institute of Mining, Metallurgical and Petroleum Engineers; New York; p 83; 1971.

Calder, P.N. and Morash, B.J. "Pit wall control at Adams Mine"; Mining Congress Journal; pp 34-42; Aug. 1971.

Cook, M.A. "The science of industrial explosives"; IRECO Chemicals; pp 449; 1974.

Dupont of Canada Ltd. "Blaster's handbook"; E.I. duPont de Nemours and Company Inc., Delaware; fifteenth edition; pp 524; 1966.

Dupont of Canada Ltd. "Controlled blasting"; DuPont of Canada Ltd; pp 16; 1964.

Gladwin, D.B. "Characteristics of blast-generated ground motion"; M.Sc. Thesis; Queen's University, Kingston, Ont.; pp 168; Dec. 1973.

Oriard, L.L. "Blasting effects and their control in open pit mining"; Geotechnical Practice for Stability in Open Pit Mining; Ed. C.O. Brawner and V. Milligan; Second International Conference on Stability in Open Pit Mining; the American Institute of Mining, Metallurgical and Petroleum Engineers; New York; pp 197-222; 1972.

Richings, M.B. "Low density and pneumatically loaded ammonium nitrate/fuel oil; Detonation velocity diameter relationships"; M.Sc. Thesis; Queen's University, Kingston, Ont.; pp 104; Jan. 1971.

Sanden, B.H. "Pre-slit blasting"; M.Sc. Thesis; Queen's University, Kingston, Ont.; pp 125; 1974.

Siskind, D.E. and Fumanti, R.R. "Blast-produced fractures in lithonia granite"; U.S. Department of the Interior; Bureau of Mines; R.I. 7901; 1974.

Workman, J.L. "An explosive slurry development and a study of priming practices"; M.Sc. Thesis; Queen's University, Kingston, Ont.; pp 111; Sep. 1973.

APPENDIX A

TABULATION OF PERIMETER BLASTING PRACTICES

AT CANADIAN OPEN PIT MINES

Key to Evaluation

1.* Condition of the final pit walls after excavation

1. No noticeable backbreak or fracturing
2. Minor backbreak (a) mainly near the crest
   (b) mainly near the toe
   (c) throughout the wall
3. Some excess rock left at the toe
4. Extensive backbreak over some areas of the wall
5. Extensive breakage over the entire wall

2.* Condition of the perimeter control blastholes

1. Fully intact on the final wall with no cracking or crushing evident
2. Mainly intact on the final wall
3. Intact over some areas of the final wall
4. Not visible at the final wall

3.* Effect of weathering on the walls

1. No noticeable change in condition after _________ months.
2. Some minor surface deterioration after _________ months.
3. Surface deterioration requiring scaling after _____ months.
4. Extensive deterioration of the rock after _________ months.

Name of mine, Location	Material mined	Perimeter blast technique	Hole diameter (in.)	Bench height (ft)	Subgrade depth (ft)	Hole spacing (ft)	Explosives type		Explosives load (lb)		Charge dia/hole dia
							toe	column	toe	column	
Bethlehem Copper Corp. Ltd., Highland Valley, B.C.	Cu	pre-splitting buffer	9-7/8	33	0	12	AN/FO	AN/FO	120-160	-	1.0
Brenda Mines Ltd., Peachland, B.C.	Cu, Mo	trim-blast-ing	12-1/4	50	3	15	AN/FO, Nitrex 201	AN/FO Nitrex 201	150	300 or 380	0.37
Brunswick Mining and Smelting Corp. Ltd., Bathurst, N.B.	Pb, Zn	buffer	6-3/4 9	36	0 or 6	8, 10	Tovex A2E Tovex A2E	Trimtex C Trimtex C	150 200	60 - 70	0.52 0.39
Canada Tungsten Mining Corp. Ltd. Tungsten, N.W.T.	W	cushion	3-1/2	16-18	3	5	Cilgel 70% Cilgel 70%	Cilgel 70% Cilgel 70%	3	10 - 12	0.43
Carey Canadian Mine Ltd., East Broughton, P.Q.	asbestos	cushion	4	45	5	7	60% High Primaflex	60% High Primaflex	10	20	0.13
Cassiar Asbestos Corp. (Clinton Mine) Clinton Creek, Yukon	asbestos	pre-splitt-ing	9	30 or 45	0	10	AN/FO	none used	125	none	1.0
Pine Point Mines Ltd., Pine Point, N.W.T.	Pb, Zn	pre-splitt-ing	9-7/8, 9 6-3/4	25	0	6	none	75% Forcite	none	22	0.032 0.039 0.069
Canex Placer Ltd., Endako Mines Division Endako, B.C.	Mo	pre-splitt-ing	9	33	7 - 10	10	Hydro-mex	Nil	100	Nil	1.0

Charge length/ spacer length	Stemming material	Height of stemming (ft)	Condition of final pit wall 1.*	Condition of perimeter blastholes 2.*	Effect of weathering 3.*	Comments	Name of mine, Location
N.A.	drill cuttings	?	3	4	2, 3 months	- every 2nd hole loaded	Bethlehem Copper Corp. Ltd., Highland Valley, B.C.
N.A.	drill cuttings	2.3	2a, 3, 4	4	2, 18-24 months 4, 6 months	- 25 ft spacing, 5 ft subgrade on 2nd to last row - extensive backbreak, weathering occur on east and west walls	Brenda Mines Ltd., Peachland, B.C.
N.A.	nil	4-5 ft	2a	2	2, 8 months	- last two rows adjacent to limit have half the normal powder load; 6-3/4 in. holes form limit row, have 8 ft burden and 16 ft spacing; 9 in. holes from adjacent row, have 10 ft burden and 20 ft spacing; pattern kept approx 50 ft wide; maximum of three limit holes per blast; no subgrade over berms	Brunswick Mining and Smelting Corp. Ltd., Bathurst, N.B.
N.A.	drill cuttings	5	2c, 3	3	1, 2 months	- 5 ms delay between rows - little difference in perimeter results when 5 or 10 ms delay is used	Canada Tungsten Mining Corp. Ltd., Tungsten, N.W.T.
N.A.	crushed rock	20	2a	2	2, 10 months		Carey Canadian Mines Ltd., East Broughton, P.Q.
N.A.	nil	all but 5 ft of hole	2a	2	4, summer months	- extensive breakage over entire wall in faulted zones, extensive weathering in faulted zones. - experimenting with drill holes 9° off vertical	Cassiar Asbestos Corp. (Clinton Mine) Clinton Creek, Yukon
3.0	nil	4	2c, 3, 4	3	-	- some of backbreak may be due to proximity of production holes to preshear line - most of rock (except for 55 ft ring around the pit) is removed prior to pre-splitting - results of pre-splitting depend upon competency of the rock, which varies greatly over short distances	Pine Point Mines Ltd., Pine Point, N.W.T.
1.0	nil	N/A	2a	1	2 - spring	- extensive wall scaling is done shortly after excavation with dozer and ship anchor chain - very good results - pit walls deteriorate during springtime only	Canex Placer Ltd., Endako Mines Division Endako, B.C.

Name of mine, Location	Material mined	Perimeter blast technique	Hole diameter (in.)	Bench height (ft)	Subgrade depth (ft)	Hole spacing (ft)	Explosives type toe	Explosives type column	Explosives load (lb) toe	Explosives load (lb) column	Charge dia/hole dia
Steel Company of Canada Griffith Mine, Red Lake, Ontario	Fe	cushion	9	35	0	12	Al AN/FO	none	250	none	1.0
Steel Company of Canada Hilton Mines Ltd., Bristol, P.Q.	Fe	cushion buffer (termination)	7 7	33 33	0 2	12 15	slurry slurry	slurry AN/FO	125 50	50 25 - 30	0.36 1.0
Indusmin Ltd., Nephton, Ontario.	nepheline syenite	cushion	3, 4-1/2	30	3	3-1/2, 5	Powerfrac Watergel	Belite	4 - 8	30 - 65	0.83 0.78
Ecstall Mining Ltd. (Kidd Creek Mine), Timmins, Ontario.	Zn, Cu Pb, Ag, Sn, Cd	line drilling	3	40	0	6		Watergel	-	100	0.83
Sherritt Gordon Mines Ltd. Ruttan Mine North Central Manitoba	Cu, Zn	pre-split	9-7/8	40	3	8	Tovex A2 or A4	TL1 or AN/FO	50	360	0.41
Granby Mining Ltd., Phoenix Copper Division, Grand Forks, B.C.	Cu	pre-splitt-ing buffer	9	33 and 66	0	9 - 10	Hydromex T9	AN/FO	25	225	0.44 or 1.0

Charge length/ spacer length	Stemming material	Height of stemming (ft)	Condition of final pit wall 1.*	Condition of perimeter blastholes 2.*	Effect of weathering 3.*	Comments	Name of mine, Location
N.A.	drill cuttings	24	2c	3	1 month	- smooth wall (cushion) blasting begun in 1973 - limit holes have 15 ft burden; adjacent row of holes has 24 ft spacing, 24 ft burden, toe load of 390 lb of Al AN/FO; next row has 24 ft spacing 22 ft burden, toe load of 550 lb Al AN/FO	Steel Company of Canada Griffith Mine, Red Lake, Ontario
1.33	nil nil	12 - 13 12 - 13	2a	1, 3	2, 12 months	- bottom initiation of blastholes improve pit bottom, decreases fly rock, reduces backbreak - there are two buffer rows	Steel Company of Canada Hilton Mines Ltd., Bristol, P.Q.
1.0	drill cuttings	8	2a	2	12 months	- spacing for post-split holes is half of spacing for production holes	Indusmin Ltd., Nephton, Ontario
N.A.	drill cuttings	5	2a	2	2, 12 months	- final pit wall configuration is 80 ft high with 40 ft safety catchment berm - originally, upper 40 ft of wall was pre-split using 2-1/2 in. diameter holes spaced 2 ft - line drilling gave equivalent results with reduced drilling and blasting costs	Ecstall Mining Ltd. (Kidd Creek Mine), Timmins, Ontario.
1.0	drill cuttings	5	2a 2c 3	3	1, 29 months	- condition of final pit wall depends on local geology and on the weathered state of the rock - mostly the lower section of perimeter blast holes are visible (never in the top 20 ft); 20% of holes have a portion of hole that is still visible. Final pit walls are poor where blasted in weathered rock	Sherritt Gordon Mines Ltd., Ruttan Mine North Central Manitoba
N.A.	crushed rock	15	2a, 3	3	1, 6 months 2, 1 month	- two 33 ft benches are blasted sequentially; top bench first; limit holes are 66 ft deep so they act as pre-split holes for the lower bench - no noticeable backbreak or fracturing except in arkose	Granby Mining Ltd., Phoenix Copper Division, Grand Forks, B.C.

Name of mine, Location	Material mined	Perimeter blast technique	Hole diameter (in.)	Bench height (ft)	Subgrade depth (ft)	Hole spacing (ft)	Explosives type toe	Explosives type column	Explosives load (lb) toe	Explosives load (lb) column	Charge dia/hole dia
Granisle Copper Ltd., Granisle, B.C.	Cu	pre-splitting	9	35	0	10	Al AN/FO	none	125	none	1.0
Cliffs of Canada Ltd., Adams Mine Kirkland Lake, Ont.	FE	pre-splitting	2-1/2	40	5	4	Nil	Forcite	Nil	35	0.50
Asbestos Corp. Ltd., Normandie Mine Black Lake, Quebec	Asbestos	pre-splitting	4	40	Nil	5	PowerFrac 75%	Forcite 40%	10	18	0.50
Asbestos Corp. Ltd., British Canadian Mine Black Lake, Quebec	Asbestos	pre-splitting	4	50	Nil	4	Nil	Forcite 40%	Nil	30	0.28
Asbestos Corp. Ltd., King Beaver Mine Thetford Mines, Quebec	Asbestos	pre-splitting	4-1/2	50	Nil	4	Nil	Hi-Cap 75%	Nil	32	0.33
Lake Asbestos of Quebec, Black Lake, P.Q.	asbestos	cushion	6-1/2	40	3-5	18	N.L.B.3B9	70% Cilgel-C	100-200	85-120	0.46 (column)
Marmoraton Mining Co., Marmora, Ontario	Fe	buffer	7	55	3	5	15%Al NCN +NCN blasting agent	ammonia dynamite	150	160	0.5 (column)
Utah Mines Ltd.	Cu, Mo	cushion	9-7/8	40	8	10	AN/FO	AN/FO	100	157	0.41 (column)

Charge length/ spacer length	Stemming material	Height of stemming (ft)	Condition of final pit wall 1.*	Condition of perimeter blastholes 2.*	Effect of weathering 3.*	Comments	Name of mine, Location
N.A.	nil	nil	2c	4	1 month	- control blasting just begun - 25 ft collar on production row next to pre-split line - pre-splitting not completely successful because of too heavy toe load, fractured surface rock	Granisle Copper Ltd., Granisle, B.C.
N.A.	-	5 ft collar	2a 4	2	2	- extensive backbreak occurs when there are strong fracture planes behind the pre-shearing - about 60% of the perimeter control blast holes are visible; water has minor effect on condition of walls	Cliffs of Canada Ltd., Adams Mine Kirkland Lake, Ont.
N.A.	-1/2 in. material	full column	varies with rock type	3	1, 12 months 2, 24 months 3, 36 months	- condition of the final pit walls vary with rock type and geological conditions	Asbestos Corp. Ltd., Normandie Mine Black Lake, Quebec
N.A.	1/4 in. rock	3 feet above column charge	2a	3	2, 12 months	- final pit walls are relatively stable, some scaling is required; scaling is done in summertime only to remove overhanging or loose rock - pre-splitting gives excellent results for pit wall control	Asbestos Corp. Ltd., British Canadian Black Lake, Quebec
1.0	fine gravel	10	4	3, 4	1, 24 months 2, 60 months	- pre-splitting experiments have given disappointing results because of the relative softness of the rock	Asbestos Corp. Ltd., King Beaver Mine Thetford Mines, Quebec
N.A.	screen tailings	11 - 12	2a (10-20 ft) 2b(0-5 ft)	4	2, 6 months	- 90° pit faces can be obtained in granite but only 55° to 70° in serpentine or shear zones - backs of holes are sometimes visible in granite - surface deterioration appears to be due to weathering (eg freezing and thawing cycles)	Lake Asbestos of Quebec, Black Lake, P.Q.
1.0 (for 1/2 column)	1/2 in. crushed rock	10	2a	2	2, 12 months	- holes are drilled to 1/2 depth	Marmoraton Mining Co., Marmora, Ontario
N.A.	drill cuttings	10	see comments	-	-	- results of control blasting not available	Utah Mines Limited

Name of mine, Location	Material mined	Perimeter blast technique	Hole diameter (in.)	Bench height (ft)	Subgrade depth (ft)	Hole spacing (ft)	Explosives type: toe	Explosives type: column	Explosives load (lb): toe	Explosives load (lb): column	Charge dia/hole dia
Steep Rock Iron Mines Ltd., Atikokan, Ontario.	Fe	buffer?	7-7/8	37-1/2	3	12	nil	AN/FO		190	0.52
Wesfrob Mines Ltd., Tasu, B.C.	Fe	pre-split, line drill-ing	3 3	35 35	3 3	1.3 - 2 5	nil	Forcite 40% Forcite 40%			0.33-0.40
Unidentified	Fe	staggered hole depth	9-7/8, 12-1/4	45,65	5, 9	23 - 30	Hydromex M-210	Hydromex T3, AN/FO, Hydromex M-210	700	1300, 2000	1.0

Charge length/ spacer length	Stemming material	Height of stemming (ft)	Condition of final pit wall 1.*	Condition of perimeter blastholes 2.*	Effect of weathering 3.*	Comments	Name of mine, Location
N.A.	drill cuttings	5	2a	2	1	- no toe problems or weathering problems; only top 10 ft is scaled - 90% of holes visible on wall - stability has increased 100% since plastic pipes were used to decouple charges	Steep Rock Iron Mines Ltd. Atikokan, Ontario.
0.53-0.80	drill cuttings	10 10	2a,c,3,4 2a,c,3,4	2, 3, 4 2, 3, 4	1 1	- poorest results occur in No. 2 pit where rock is heavily fractured and faulted (wedge-shaped faults) - best results in pit No. 3 - bottom 75 ft of 105 ft wall in pit No. 2 and all of 70 ft wall in pit No. 1 are line drilled	Wesfrob Mines Ltd., Tasu, B.C.
3.0	drill cuttings	12, 16	1	see comment	1 12 months	- staggered hole depth is used to obtain dip control - control blast holes for final pit wall are drilled on same pattern as production blast, with depth depending upon the desired dip of the final pit wall. - last two rows are drilled with reduced dip and shot with the main blast - perimeter blastholes not visible, but final surface is clear	Unidentified

APPENDIX B

DESIGN OF PRODUCTION BLASTS

TO REDUCE WALL VIBRATION

INTRODUCTION

1. When trying to protect structures such as buildings or underground openings or to prevent minor falls of loose rock, it is often necessary to minimize wall vibration from blasting. To accomplish this, the mine operator can adjust a number of variables. In order of importance the variables are: delays and sequencing, blast geometry, explosive type, and stemming.

Delays and Sequencing

2. Delays between adjacent rows of blastholes should be greater than 15 msec - particularly for the two production rows nearest the pit limit as otherwise ground shock from adjacent rows may accentuate vibration level in the final pit wall.

3. Reducing the charge weight per delay is one of the most effective methods of reducing vibration. How to calculate the maximum charge weight per delay that can safely be used near buildings has been indicated in the chapter proper. One way of reducing the charge weight per delay with V cuts is to stagger the firing of each arm. To further reduce it, only part of each row of holes can be fired per delay. Eventually, as a building is approached, it may be necessary to fire as few as one hole per delay. If smaller hole diameters or decoupling or decking are used, the charge weight per delay, as well as the burden and spacing, will be reduced still further.

Blast Geometry

4. V-cuts should be used so that all charged break to a free face. This will reduce vibrations.

5. Square patterns produce more vibration than rectangular patterns. The reason for this is that a charge is less confined with a smaller burden. Reducing the burden or charge confinement will decrease vibration.

6. The rows of blastholes oriented parallel to tunnels, building faces, etc, contribute most to blasting vibration. The rows of blastholes oriented obliquely to tunnels, etc, contribute less to vibrations because of the destructive interference between shock waves and the longer average travel time for the shock wave from blasthole to tunnel.

7. Where a blast is set up so that one arm of a V-cut is situated in front of the other arm with respect to a building or tunnel, vibrations from the rear row can be reduced by firing the front row of holes first. The firing of the row of holes closer to the building fractures the ground and vibrations from the rear row are reduced when passing through the fractured ground.

Explosive Type

8. Vibration can be reduced by substituting AN/FO for slurries or dynamites which produce high borehole pressure. AN/FO has good breaking ability but does not produce as high a level of ground vibration.

Stemming

9. Vibration from blasting will be reduced slightly if no stemming is used. In most cases though, the difference is not enough to warrant the increased airblast and flyrock.

APPENDIX C

DESIGNING A CONTROL BLAST

PRE-SPLITTING

Drilling

1. Accurate drilling of pre-split holes is critical. Holes must lie in a single plane. Bad toes should be removed so that holes can follow a straight line. Several drill hole collars should be surveyed in before the blast. This will aid in determining the extent of any ensuing backbreak. Careful observation of the rock face should be made after the blast to determine acceptable tolerances of drilling accuracy.

2. The effects of drill hole wander are most prevalent at the toe, since this is where variations in alignment will be largest. If wander is producing unaccceptable backbreak or leaving excess rock at the toe, drilling accuracy must be improved. Wander can be reduced by drilling larger diameter holes or by reducing hole length. Holes should be less than 60 ft (18 m) to 80 ft (24 m).

3. Drill holes must be inclined when pre-splitting to a tight fault or joint surface. Small diameter percussive drilling equipment should be employed for this, and care must be taken to drill accurately.

Explosives

4. One of the primary considerations in selecting an explosive charge for a pre-split hole is its borehole pressure. As calculated by equation 3, para 18 this must not exceed the in situ dynamic compressive strength of the rock. In most cases, this will involve decoupling or decking the explosive charge.

Example

5. A rock having a dynamic compressive strength of 50,000 psi is to be pre-split in a 3 in. diameter drill hole. What size cartridge of Cilgel-B will do the job without crushing the back of the borehole?

First, set the borehole pressure equal to the dynamic compressive rock strength:

$$(P_b)_{dc} = 50{,}000 \text{ psi}$$

from Fig 1a,

$$(P_b)_c \text{ of Cilgel B} = 450{,}000 \text{ psi}$$

from Eq 3,

$$\text{C.R.}^{\,2.4} = (P_b)_{dc}/(P_b)_c = 0.11$$

from Fig 5,

$$\text{C.R.} = 0.40$$

from Eq 2,

$$r_c = \frac{\text{C.R.} \times r_h}{\sqrt{C}}$$

$$= \frac{0.40 \times 1.5}{1}$$

$$= 0.6 \text{ in. } (1.5 \text{ cm})$$

the charge diameter must be 1.2 in. (3.0 cm) Taking the closest cartridge size below 1.2 in. (3.0 cm), a value of 1-1/8 in. (2.9 cm) is obtained.

6. Another important consideration in designing the explosive charge is the depth of water table in the immediate blasting area. Wet portions of drill holes should be loaded with a water resistant explosive. The required water resistance will depend upon the delay between loading and blasting which ideally should be no more than several days. Where a decoupled charge is to be surrounded by water, the loading density should be reduced by 10 to 20%.

Burden and Spacing

7. The pre-split blast is fired before the production burden in front of it. In effect, the pre-split line has an infinite burden.

8. The spacing between pre-split holes can be calculated by the equation:

$$S = \frac{2r\,(P_b + T)}{T} \qquad \text{eq C-1}$$

where

- S = hole spacing (in.)
- r = hole radius (in.)
- P_b = borehole pressure of the explosive charge from eq 3 (psi)
- T = dynamic tensile strength of the rock (psi)

The borehole pressure should not exceed the dynamic compressive strength of the rock. A first approximation for the value of T may be obtained from Table C-1. An accurate determination could be made using the testing procedure outlined in the chapter.

9. For best results, the hole spacing should be less than twice the spacing of major open joints. This results in a more even powder distribution and less chance of backbreak along joint planes.

Table C-1: First approximation values for dynamic tensile strength of rock

Rock type	T (psi)	T (Pa)
Taconite	2500-6000	$1.7 \times 10^7 - 4.1 \times 10^7$
Copper ore	4000	2.7×10^7
Asbestos ore	700	4.8×10^6
Limestone	1000-2000	$6.8 \times 10^6 - 1.4 \times 10^7$

Example

10. A rock with a dynamic tensile strength of 2500 psi (1.7×10^7 Pa) and a dynamic compressive strength of 40,000 psi (2.8×10^8 Pa) is to be pre-split using a continuous column of 1-1/4 in. (3.2 cm) cartridges of Cilgel B in a 4 in. (10 cm) hole. What hole spacing should be used?

First calculate the coupling ratio from eq 2

$$C.R. = \left(\sqrt{C}\, \frac{r_c}{r_h}\right)$$

$$= 1.0 \times \frac{1.25/2}{4.0/2}$$

$$= 0.31$$

from Fig 5,

$$C.R.^{\,2.4} = 0.061$$

using Eq 3,

$$\begin{aligned}(P_b)_{dc} &= (P_b)_c \times 0.061\\ &= 450{,}000 \times 0.061\\ &= 27{,}450 \text{ psi } (1.9 \times 10^8 \text{ Pa})\end{aligned}$$

which is well below the dynamic compressive strength of the rock.

using Eq C-1,

$$\begin{aligned}S &= \frac{2r(P_b + T)}{T}\\ &= \frac{4(27{,}450 + 2500)}{2500}\\ &= 47.9 \text{ in.}\\ &= 4.0 \text{ ft } (1.2 \text{ m})\end{aligned}$$

If the rock were highly jointed, its dynamic compressive strength would be lower, perhaps in the 20,000 psi (1.4×10^5 Pa) range, calling for a lower borehole pressure to avoid backbreak, and thus a closer spacing.

Stemming

11. The collar should be determined using the criteria given in para 31 to 34. Holes need not be stemmed unless air blast must be controlled.

Subgrade Drilling

12. Depth of subgrade drilling may vary from 3 to 5 ft (0.9 - 1.5 m). This will help promote drainage beside haul roads.

Delays and Sequencing

13. For effective splitting action, pre-split holes should be fired simultaneously with a 50 msec delay before the main blast. A primacord trunkline should be used, or if noise and vibration control is necessary, MS delay caps can be used.

14. It is often not possible to shoot the entire pre-split line with a 50 msec delay before the main blast. Air-track holes may cave due to frequent blasting operations. To minimize hole caving and consequent re-drilling, holes must be loaded soon after drilling. If water is present, these holes must be blasted before the explosives deteriorate. Holes should be fired in groups of at least six. Also, labour and equipment problems may prevent a large number of pre-split holes from being fired simultaneously 50 msec before the main blast.

15. Drilling of drainage holes will help to reduce water problems in blastholes. This will also reduce the problem of water seepage into an open pre-split fracture. When groups of pre-split holes are fired days before the main blast, water can fill the open pre-split fracture and reduce its effectiveness.

Buffer Row

16. To prevent the production blast from breaking back past the pre-split fracture, a buffer row must be incorporated in the design.

CUSHION BLASTING

Drilling

17. Accurate drilling is important for slashing or cushion blasting with long blastholes. Holes may be up to 120 ft (37 m) deep; however, the usual depth is one bench.

18. For best results, holes should be drilled at the final pit slope angle which requires percussive drilling equipment. In fractured rock, this type of equipment has several disadvantages. Return of drill cuttings is difficult in very long holes, especially if the compressed air can escape into open fractures or joints. Small diameter holes are also subject to wander or caving. Thus, it is necessary to either use free-pouring explosives or to drill more costly large diameter percussive drill holes.

Explosives

19. Charges should have a coupling ratio of 0.5 or less to provide a "cushion", ie, to reduce the borehole pressure, so that backbreak does not occur. Wedges or stemming should be used to ensure that the cartridges are pushed against the excavation side of the blasthole thus providing a better "cushion" effect. Spacing of charges near the top of the column will help reduce crest fracture, particularly in unconsolidated rock.

20. Some average powder factors for various rock types are suggested in Table C-2 as first approximations. Add or subtract approximately 20 per cent to these figures for competent rock and highly fractured rock respectively.

Table C-2: Powder factors for use as first approximation in cushion blasting

Rock type	Powder factor (lb/ton)	Powder factor (kg/tonne)
Taconite	0.30	0.15
Copper ore	0.28 - 0.38	0.14 - 0.19
Asbestos ore	0.16	0.08

Burden and Spacing

21. Make the spacing in feet equal to 1.25 times the hole diameter in inches for taconite or copper ore. Use a value of 2.0 for asbestos ore. The burden may be made equal to the spacing in competent rock or 0.6 - 0.8 times the spacing in very fractured rock. If the blastholes are inclined, a heavy crest burden can be tolerated since the weight of the undercut rock will help bring down the top burden.

22. Spacing should be reduced but a constant powder factor should be maintained when performing cushion blasting around curved areas. Line drilling or pre-splitting should be used when blasting around 90° corners.

Use of Guide Holes

23. Guide holes can be used between cushion holes to provide better blasting results where the ground is unconsolidated, (ie, where it is weathered or highly fractured). However, this will greatly increase drilling costs. Large diameter rotary drill holes should be used to reduce costs as much as possible.

24. Try cushion blasting without guide holes first. If this is unsatisfactory, reduce the spacing by 25% and reduce the powder load. If this is still unsatisfactory, go back to the original spacing and powder load and use guide holes midway between the cushion holes.

Stemming

25. Use 20 ft (6.1 m) of stemming where the rock is soft or highly jointed. In competent rock, use 10 - 15 ft of stemming.

Delays and Sequencing

26. For best results, cushion blastholes should be fired simultaneously. Use a primacord trunkline, or if noise and vibration control are necessary, use MS delay caps.

LINE DRILLING

Drilling

27. The most commonly used hole size in line drilling is 2-1/2 in. (4 cm) or 3 in. (7.6 cm). Large diameter rotary drill hole can also be used.

28. If a constant web is maintained between adjacent holes as opposed to a spacing which is some multiple of the hole diameter, costs for large diameter rotary holes are still comparable to those of small diameter holes. If the larger hole spacings are used, then large diameter rotary drill holes are more economical.

29. For 2 in. (5.1 cm) to 3 in. (7.6 cm) holes, the depth should not exceed 30 ft (9.1 m) to 40 ft (12 m) as otherwise, hole wander is too great. No subgrade drilling is necessary.

30. Close control over drilling is essential, more than in any other control blasting method. Holes must be drilled so that they all lie in one plane corresponding with the dip of the final pit wall. Careful observation of the rock face should be made after the blast to determine acceptable tolerances of drilling accuracy.

Hole Spacing

31. To get hole spacing in feet, multiply the value in Table C-3 by the hole diameter in feet.

Table C-3: Values of hole spacing for use as first approximation in line drilling

Rock type	Hole spacing factor
Taconite	2.0
Copper ore	2.5
Asbestos ore	4.0

Buffer Row

32. For line drilling to be effective, it must be used in conjunction with a buffer row.

Production Blast

33. For best results, the main excavation charges should be 1 - 3 rows from the pit limit.

BUFFER BLASTING

Drilling

34. Hole size and depth should be the same as for holes in the production blast. Depth of subgrade should be 7 to 10 times the hole diameter to eliminate scallop at the toe and to promote drainage for haul roads. Holes positioned directly above or near the crest of an underlying berm (usually production holes) should not have any subgrade, to reduce crest fracturing.

Explosives

35. Common practice in designing buffer rows consists of reducing the powder factor by as much as 0.5. Examples of typical powder loads for different rock types are given in Table C-4. These factors are equal to 0.6 time the values of the powder factors for production blasting which is the most usual reduction.

Table C-4: Typical powder factors for buffer blasting

Rock type	Powder factor lb/ft	kg/tonne)
Taconite	0.39	0.19
Copper ore	0.28	0.14
Asbestos ore	0.20	0.10

36. The burden and spacing for the buffer row should be 0.5 to 0.8 times that of the adjacent production row. As a general rule, the burden on a buffer row should be less that the hole spacing, H_{spac}. If the burden is too large in relation to

hole spacing, over-confinement can occur causing the production of oversize muck. Too large a hole separation, however, can result in protrusions being left midway between the buffer holes in the back wall. It is recommended that hole separation H_{spac} be 1.25 the burden in a buffer row.

37. Once the burden and spacing of the buffer row has been established, it is necessary to find the charge per hole which will effectively reduce the powder factor to about 0.6 times that used for production blasts. First, the borehole pressure of the buffer charge is found as

$$P_{b_{buff}} = \frac{0.6 P_{b_{prod}} \, r_{prod} \, S_{buff} \, B_{buff}}{r_{buff} \, S_{prod} \, B_{prod}} \qquad \text{eq C-2}$$

where $P_{b_{buff,\ prod}}$ = borehole pressure of buffer or production charge

$r_{buff,\ prod}$ = radius of buffer or production blasthole (in.)

$S_{buff,\ prod}$ = spacing on buffer or production row (ft)

$B_{buff,\ prod}$ = burden on production row (ft)

38. Knowing the necessary borehole pressure for each buffer charge and the borehole pressure that would be produced if the same explosive filled the borehole completely, the coupling ratio can be calculated from equation 3. The necessary buffer charge diameter or charge length/spacer ratio can then be determined. The resulting charge per hole is larger than that found by a straight powder factor calculation, as decoupling reduces the effective breaking power of the explosive.

Example

39. The following information is available concerning the final row of a production blast in which a buffer row must be designed:

a. blast hole diameter, 10 in. (25 cm)
b. explosive velocity, 17,000 fps (5,200 m/s)
c. explosive specific gravity, 1.2
d. loading density, 0.60 lb/ton (0.30 kg/tonne)
e. pattern: burden = 18 ft (5.5 m) spacing = 20 ft (6.1 m)
f. sub-grade drilling, 5 ft (1.5 m)
g. average measured backbreak, 16 ft (4.9 m)
h. rock density 200 lb/ft^3, (3.2×10^3 kg/m^3)

40. The same amount of sub-grade drilling will be used in the buffer row as in the production rows, ie, 5 ft (1.5 m).

41. The burden used in the production blast should be reduced by a factor of 0.5 - 0.8; in this case assume a factor of 0.55.

$$B_b = 0.55 \times 18 = 10 \text{ ft } (3.0 \text{ m})$$

The spacing should be 1.25 times the burden:

$$S_b = 1.25 \times 10 = 12.5 \text{ ft } (3.8 \text{ m})$$

The borehole pressure of the production charges from equation 1 would be:

$$P_{b\ prod} = .00177\ (1.2)\ (17{,}000)^2 = 614{,}000 \text{ psi } (4.2 \times 10^9 \text{ Pa})$$

42. The required borehole pressure in the buffer holes, using equation C-2 is:

$$P_{b\ buff} = \frac{0.6\ (614{,}000) \times 10 \text{ in.} \times 10 \text{ ft} \times 12.5 \text{ ft}}{10 \text{ in.} \times 18 \text{ ft} \times 20 \text{ ft}} = 127{,}900 \text{ psi } (8.8 \times 10^8 \text{ Pa})$$

From equation 3:

$$C.R.^{2.4} = \frac{127{,}900}{614{,}000} = .21$$

$$C.R. = 0.52$$

43. From equation 2, a 5 in. diameter charge in a 10 in. (2.5 cm) diameter hole will provide the necessary charge size for a 10 ft (3.0 m) x 12.5 ft (3.8 m) pattern.

44. Assuming a collar of 24 hole diameters, ie, 10 ft (3.0 m), 5 ft (1.5 m) of sub-grade drilling, a 35 ft (10.7 m) bench and 10 lb/ft (15 kg/m) for an explosive with a specific gravity of 1.2, the charge weight per hole is

$$W_{buff} = (35 + 5 - 10) \times 10$$
$$= 300 \text{ lb } (136 \text{ kg})$$

The above is an approximation as a 5 in. (13 cm) charge in a 10 in. (15 cm) borehole only gives a coupling ratio of 0.5 rather than 0.52.

Positioning of the Buffer Row

45. When the buffer row is to be located in front of pre-split or line-drilled holes, the distance from the buffer row to these holes can be found using this equation:

$$D_{BUF} = \left(\frac{P_{b_{BUF}}}{P_{b_{PROD}}}\right)^{0.5} \frac{F.R. \times r_{BUF}}{r_{PROD}} \qquad \text{eq C-3}$$

where

D_{BUF} = distance from buffer row to pre-split or line drilled holes

$P_{b_{BUF}}$ = borehole pressure of buffer charge

$P_{b_{PROD}}$ = borehole pressure of production charge

F.R. = fracture radius or backbreak from production row

r_{BUF} = hole radius of buffer charge

r_{PROD} = hole radius of production charge

Example

46. Consider the location of a pre-split line with respect to the designed buffer row of the previous example. The production row back break has been established as 16 ft (4.9 m). Using eq C-3 we have:

$$D_{BUF} = \left(\frac{1.28 \times 10^5}{6.14 \times 10^5}\right)^{0.5} \times 16 \times \frac{10}{10}$$
$$= 7.3 \text{ ft } (2.2 \text{ m})$$

Delays and Sequencing

47. The buffer row is to be fired after the production holes with at least a 15 msec delay. In the case of pre-splitting, the buffer row, as well as the production rows, should be fired after the pre-split blast. It is preferable to have at least a 15 msec delay between adjacent holes on the buffer row.

Stemming

48. The depth of collar should be 12 charge diameters for hard competent rock to 30 charge diameters for soft incompetent rock. Holes may be stemmed using drill cuttings or crushed rock.

Staggered Hole Depth Technique

49. This technique is a form of buffer blasting. Rows of blastholes have a reduced powder-load, burden and spacing. These rows are drilled at various depths close to an underlying structural plane (bedding, joint, fault) without penetrating the plane. The object is to fragment the wedge of rock between the holes and the plane without undercutting the plane. Use the same explosives load that would be used in a buffer row.

50. The required vertical distance from the toe of the buffer row to the bedding plane should be found using eq C-3. The vertical distance from the hole bottom to the bedding plane can be determined using eq C-4.

$$D_{VERT} = \frac{D_{BUF}}{\cos\theta} \qquad \text{eq C-4}$$

where

D_{BUF} = distance from toe of buffer row to structural plane (from eq C-3)

θ = dip of structural plane

51. Precise drilling of blastholes is critical. The exact position of the structural plane must be known as nearly as possible, so that the holes can be drilled to within the correct distance of the plane. Holes should be drilled in a chevron configuration (Fig C-1).

52. The buffer rows should be laid out in this manner. Draw a cross section of the bench to be blasted, showing the strucural plane. Find the intersection between the plane and the level of the next lowest bench. The first buffer row should be positioned 5 ft (1.5 m) in front of this point. Lay out the positions of the other buffer rows relative to it and with one buffer row in front of it. It should be drilled with 6 ft (1.8 m) to 9 ft (2.7 m) of subgrade.

53. The depth of stemming is less than for a

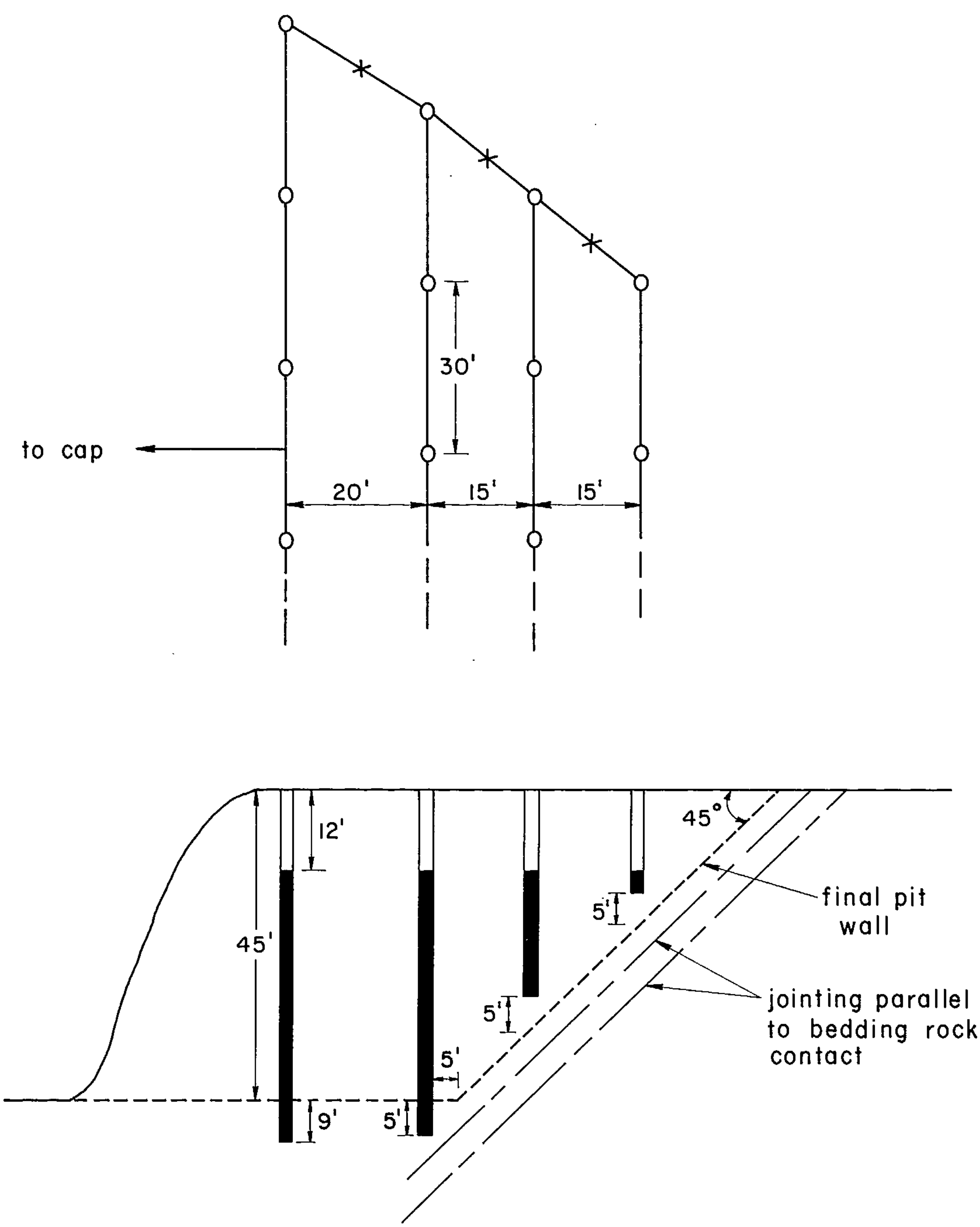

Fig C-1 - Typical blast layout for staggered hole depth technique.

Fig C-2 - Excavation of footwall at Carol Lake Mine. A 125 ft stretch of clean smooth wall was produced using the staggered hole depth technique.

regular buffer blast, since crest fracture or cratering is actually beneficial. Use 5 ft (1.6 m) to 10 ft (3.0 m) of stemming in competent rock and 10 ft (3.0 m) to 15 ft (4.6 m) in fractured rock.

54. There should be at least a 15 msec delay between adjacent rows.

55. Figure C-1 illustrates a blast laid out using these principles. The object was to blast to a bedding plane. The final pit wall had to be free of undercut rock and face loose rock, as there was to be no berm for 125 ft (38 m). Figure C-2 indicates the excellent condition of the wall as seen after the bulk of the blasted rock had been mucked out.

APPENDIX D

SHOCK AND VIBRATION MEASUREMENTS

INTRODUCTION

1. Vibration measurements are used to define the rate of attenuation of ground shock in a particular rock type. Knowledge of how the ground shock wave attenuates is necessary for protecting structures and for designing control blasts.

2. Three systems for monitoring ground motion due to blasting measure particle acceleration, velocity and displacement respectively. However, damage and ground motion are best correlated with peak particle velocity. Consequently, peak velocity is monitored to make shock or vibration measurements.

Instrumentation

3. Pre-packaged seismographs are recommended for use by mine operators. These instruments are compact, portable and relatively easy to set up.

4. A good example of the type of engineering instrument currently available in the mining and construction industry is the Sprengnether Engineering and Research Seismograph, Model VS-1200.

Velocity Sensing Unit

5. The sensing elements of the 3-component seismometer in the Sprengnether instrument are velocity transducers. The seismometer is contained in a small waterproof cube for remote placement. The cube must either be buried in overburden or firmly placed on rock by means of weight to be coupled with the medium undergoing vibration.

6. Specifications for this system are given in Table D-1. The frequency response for the unit covers a range of 1.8 to 250 cps. This precludes use of the instrument close to large blasts, because ground motion frequency in this region may exceed the 250 cps upper tolerance.

7. To operate the instrument, the seismometer is plugged into the main unit and the type of measurement to be taken, in this case, velocity, is switch-selected. Other quantities can be measured in addition to particle velocity. Particle acceleration or displacement is obtained by transmitting the electrical output from the velocity transducers through a hi-pass or low pass filter, respectively. The power for the system is supplied by an internal rechargeable battery providing three hours of continuous operation.

8. For some applications, such as velocity monitoring on the wall of a tunnel, the 3-component seismometer is not suitable as it is difficult to mount. An appropriate velocity

Table D-1: Specifications for Sprengnether seismograph

System

Frequency response	Flat (3db) 1.8 Hz to 250 Hz	
Sensitivities		
Displacement	10, 50, 200, 1000, 5000, 20,000 in./in.	
Velocity	1, 5, 20, 100, 500, 2000 in./in./sec	
Acceleration	0.1, 0.5, 2, 10, 50, 200 in./g	
Fourth channel	Voltage sensitivity of 30 mv/in.	
Calibration	Step in acceleration (33 ma in calibrate coils) automatically applied 1 second after start of recording can be applied manually at any time	

Physical

	Seismometer	Camera Case
Size	7x7x7 in.	9x11x13 in.
Weight	19.5 lbs	33 lbs
Density	1.6 gm/cc	

Seismometer

Three orthogonal components with identical characteristics

Natural frequency	2 Hz
Inertial Mass	0.5 kg
Range of motion	6 mm peak-to-peak
Signal coils	300 ohms - 45 volts/m/sec
Damping	0.6 critical
Calibration coil	0.44 newtons/ampere
Temp. range	
Between stops	-20°F to 140°F

Camera

Paper	70 mm (2.75 in.) standard photographic or direct write in up to 200 ft rolls
Paper speed	100 mm/sec (approx 10 min. recording), 400 mm/sec optional
Timing lines	Standard - each 0.02 sec, 0.005 at 400 mm/sec.
Galvanometer	200 Hz
Power	12V DC Internal Battery (approx 3 hours or 15 rolls (200 ft) continuous recording), built-in 110V, 60 Hz charging unit.

Sprengnether Seismometer

sensing unit can be put together using off-the-shelf components. Velocity transducers (eg Tektronix 331015 velocity transducers, see Table D-2) are mounted on short lengths of aluminum bar stock. The transducers are insulated from the aluminum gauge pins by lucite spacers to eliminate ground looping due to stray electrical currents. Velocity transducers are mounted on the gauge pins in three mutually perpendicular orientations. A potting compound (eg Devca Flexane^R 60) is used to waterproof the gauges. Gauge pins on which transducers have been mounted are clamped onto wall-anchored rock bolts on the tunnel wall.

Table D-2: Specifications for the Textronix 331015 horizontal velicity transducers

Weight	5.8 oz (0.16 kg)
Type:	Inductive, self-generating, mechanical
Displacement range:	0.050 in. (0.13 cm) (peak-to-peak)
Temperature range:	-40°C to 71°C
Voltage sensitivity:	550 mV/in./sec (nominal) 216 mV/cm/sec
Natural frequency:	8 cycles/sec
Flat frequency response range:	10 - 1000 cycles/sec
Transverse sensitivity	5% maximum

Recording Equipment

9. The particle velocity data is normally recorded by a light pen oscillograph. Peak reading voltmeters have recently been used for vibration monitoring. This device consists of a simple voltmeter plus additional electronic circuitry for holding the maximum voltage on the read-out scale until the instrument is reset. It is useful when information is immediately required, but does not provide a permanent record such as a chart print-out. Peak-reading voltmeters are used to complement existing seismographs and are usually provided as options.

10. The recording system is calibrated vertically by superimposing a pulse of known value on the recording paper. For horizontal calibration, there are timing lines every 0.02 seconds.

11. A price list including the cost of a basic Sprengnether VS-1200 recording system, plus options and accessories, is given in Table D-3.

Table D-3: Cost of Sprengnether seismic unit

Model	Description	Price
VS-1200	Engineering and research seismograph	$4,775.00
	Options	
	Fourth signal channel	$ 270.00
Total - complete system		$5,045.00

All prices are F.O.B. St. Louis, discounts are not included.

Setting Up

12. The recording unit of the seismograph should be shock mounted in a dry area. It can be located remote from the seismometer in the event that large vibrations are expected in that region.

13. The seismometer can be weighed down with a rock or heavy object. If the peak particle velocity is greater than 5 in./sec (13 cm/sec), the seismometer should be bolted down.

14. The sensitivity of the instrument must be switch-selected before a blast. The sensitivity must be low enough that the three waveforms recorded from the blast are not superimposed on one another, yet are large enough to be accurately measured.

15. The instrument should be calibrated manually before each blast. Calibration will also be applied automatically one second after the start of recording.

Reducing the Data

16. After the blast, the paper with the wave traces is removed from the seismograph. The date,

blast number, location of the seimometer, instrument sensitivities, etc, should all be recorded.

17. The peak particle velocity is determined from the wave record. The distance from the seismometer to the centre of the closest production row that is nearly perpendicular to the blast-to-seismometer direction is determined. This is divided by the square root of the maximum charge weight per delay for that blast, to obtain a $D/\sqrt{W}$ value for the measured peak particle velocity.

GLOSSARY

BACKBREAK

Fragmentation and fracturing of rock by blasting beyond the intended line of break.

BERM

A horizontal shelf or ledge built into an embankment or sloping wall of an open pit or quarry to break the continuity of an otherwise long slope for the purpose of strengthening and increasing stability of the slope or to catch or arrest slough material.

BOREHOLE PRESSURE

The peak effective pressure caused by expanding gases that acts behind the detonation head on the cylindrical surface area of the borehole during an explosion; approximately equal to one half of the detonation pressure.

BUFFER BLASTING

A control blasting technique employed during the main production blast where the last row of boreholes has a reduced burden, spacing and explosives load.

BUFFER ROW

A row of explosives with reduced spacing and explosives load; in buffer blasting, the buffer row is the last row adjacent to the planned excavation limit.

BULK STRENGTH

A measure of performance of an explosive based on the number of energy units per unit volume of explosives relative to 100 energy units/unit volume of AN/FO at a known density.

BURDEN

The distance between the explosive charge and the free face of the material to be blasted.

COLLAR

The unloaded portion of a blasthole extending from the surface down to the top of the explosives column.

COMPRESSIVE ROCK STRENGTH

The amount of compressive stress that a rock can withstand under uniaxial loading without failing.

CONTROL BLASTING

Various techniques used to limit the amount of backbreak developed during the blasting phase of the excavation cycle by reducing the level of ground shock vibrations.

COUPLING RATIO

The square root of the ratio of the volume of the borehole (excluding the volume of the collar) divided by the volume of explosive material.

CRUSHED ZONE

A region immediately surrounding a blasthole where the compressive ground stress due to an explosion has exceeded the dynamic compressive strength of the rock.

CUSHION BLASTING

A control blasting technique employed after the main production blast where the rock slope is trimmed to the planned excavation limit.

DECOUPLED CHARGE

A charge which has a smaller diameter than the blasthole in which it is loaded; the coupling ratio is less than one.

DETONATION PRESSURE

The pressure exerted by gases as they are first produced at the detonation head.

DRIVING FORCE

Those forces in a system which tend to cause failure.

DYNAMIC ROCK STRENGTH

The amount of stress that a rock can withstand without failing, under changing loading conditions.

FACTOR OF SAFETY
The ratio of the forces tending to resist failure to those forces tending to cause failure.

FAULT
A fracture or a fracture zone along which there has been displacement of the two sides relative to one another parallel to the fracture; the displacement may be a few inches or many miles.

FOLIATION
A crystalline segregation of certain minerals in a rock in a dominant plane due to metamorphism; schistosity, flow cleavage and fracture cleavage are considered as types of foliation.

FRACTURE
A break in the continuity of a body not attended by a movement on one side or the other and not oriented in a regular system.

FRACTURE RADIUS
See radius of rupture.

FREQUENCY RESPONSE
The range of frequencies that can be sensed (within certain acceptable limits or error) by a device.

GELATIN
A type of explosive which has a water-resistant "gel" base; this makes it a waterproof, cohesive, plastic product.

GRANODIORITE
A plutonic rock consisting of quartz, plagioclase, (andesine or calcic oligoclase) and orthoclase, with minor biotite, hornblende, or pyroxene; intermediate between quartz monzonite and quartz diorite; contains at least twice as much plagioclase as orthoclase.

HERTZ
Cycles per second.

IN SITU ROCK STRENGTH
In place rock strength as opposed to in the laboratory.

JOINT
A crack, fracture, or fused crack in rock along which there has been very little or no movement parallel to the crack.

LINE DRILLING
A control blasting technique in which a row of closely spaced holes is drilled at the planned excavation limit; the holes form a plane of weakness to which the final production row is designed to break.

LONGITUDINAL WAVE VELOCITY
Speed of a wave travelling parallel to the direction of propagation.

LOW DENSITY EXPLOSIVES
Explosives which have less breaking power due to their lower density; density may be decreased by loose packing, by altering the coarseness of the components, or by adding space consuming materials such as gas, woodmeal or microballoons; AN/FO is a commonly used low density explosive.

MICROBALLOONS
Plastic or glass spheres used to decrease the density of AN/FO.

OVERBREAK
See backbreak.

PARTICLE VELOCITY
The speed of a rock particle, acquired as a result of shock wave disturbance transmitted through rock.

POISSON'S RATIO
The absolute value of the ratio of the transverse strain to the corresponding axial (longitudinal) strain in a body subjected to uniaxial loading.

POST-SPLITTING
See cushion blasting.

PRE-SHEARING
See pre-splitting.

PRE-SLOTTING
See pre-splitting.

PRE-SPLITTING
A control blasting technique employed before the main production blast where a row of closely spaced, lightly loaded holes are detonated so that a continuous open fracture is formed along the planned excavation limit.

RADIAL STRESS
Stress normal to the tangent to the boundary of any opening.

RADIUS OF RUPTURE
Distance from the centre of a blasthole to the limit of radial cracking produced by a charge detonated in that blasthole.

RESISTING FORCE
Those forces in a system which tend to resist failure.

SCALED ACCELERATION
Acceleration multiplied by the square root of the explosive weight of a cylindrical charge; acceleration multiplied by the cube root of the explosive weight of a spherical charge.

SCALED DISTANCE
Distance from some point to a blast divided by the square root of the explosive weight of a cylindrical charge; distance from some point to a blast, divided by the cube root of the explosive weight of a spherical charge.

SCALING
The removal of loose rocks from the surface of a pit wall.

SCALLOP
An undesirable remnant of rock remaining at the toe of a bench due to blasting inefficiency.

SEISMOGRAPH
An instrument which detects and records earth vibration (seismic) waves.

SEMI-GELATIN
A type of explosive that partially resembles a gelatin but is more economical.

SHEAR FACTOR
The weight of explosives required to produce one square foot of pre-split surface area.

SHEAR ZONE
A portion of a rock mass traversed by closely spaced surfaces along which shearing has occurred; rock may be crushed and brecciated.

SLABBING
See cushion blasting.

SLASHING
See cushion blasting.

SLURRY
An explosive consisting of ammonium nitrate, water, thickeners, and a high energy sensitizer such as T.N.T.; has high bulk strength and good water resistance.

SMOOTH BLASTING
See cushion blasting.

SPACING
The distance between adjacent holes in a row of blastholes.

STEMMING
Material which is placed in boreholes on top of or around the explosive column to contain the detonation products and to improve blasting efficiency; usually sand, drill cuttings or fine crushed stone.

STATIC ROCK STRENGTH

The amount of stress that a rock mass can withstand from a stationary load without failing.

STRESS

The force per unit area as the area approaches zero acting within a body.

STRESS RELIEVING

See pre-splitting.

SUBGRADE DRILLING

That part of blasthole drilling in which the depth of the hole is extended past the planned surface of the underlying bench.

TANGENTIAL STRESS

Stress parallel to the tangent to the boundary of any opening.

TECTONIC STRESS

Stress caused by deformation of the earth's crust; this stress may occur near the surface and may greatly exceed the stress in the rock due to gravity.

TENSILE ROCK STRENGTH

The amount of tensile stress that a rock can withstand without failing.

THERMOCHEMICAL PRESSURE

The pressure which theoretically should be created when an explosive is detonated; calculated from thermochemical properties of the explosive.

TOE

The base of a bank, bench, or slope in a quarry or open pit mine.

TRANSDUCER

An instrument which converts an applied force into an electrical signal, the magnitude of the signal being proportional to the size of the applied force being measured.

TRIM BLASTING

See cushion blasting.

UNDERCUT BEDDING

A plane along which failure and sliding may occur because the slope of free face is greater that that of the bedding.

UNDERCUT JOINT

See undercut bedding.

UNIAXIAL COMPRESSION

Compression in only one direction; no forces act in other directions.

UNIAXIAL COMPRESSIVE STRENGTH

The strength of a rock sample under uniform compressive stress on one axis only.

VIBRATION SENSITIVITY

The size of electrical signal generated by a transducer for each unit of vibration; usually expressed as millivolts per inch per second.

WEDGES

Wedge-shaped blocks of rock whose boundaries are joint or fault surfaces.

YOUNG'S MODULUS OF ELASTICITY

The stress required to produce unit linear strain.

SYMBOLS

		UNITS
C	percentage of hole length (excluding collar) that contains explosives	none
c_ℓ	longitudinal wave velocity	ft/sec, m/sec
C.R.	coupling ratio	none
D	detonation velocity of a confined charge	ft/sec, m/sec
D_{BUF}	distance from buffer row to line of pre-split of line drilled holes	ft, m
D_{VERT}	vertical distance from bottom of buffer hole to bedding plane	ft, m
E	Young's Modulus of Elasticity	psi, Pa
F.R.	amount of measurable back-break from the last row of a production blast	ft, m
k_2	blasting site factor for peak particle velocity	none
m_2	blasting site attenuation factor for peak particle velocity from a cylindrical charge	none
N	constant for borehole pressure equation	none
n_2	blasting site attenuation factor for peak particle velocity from a spherical charge	none
P_b	borehole pressure	psi, Pa

		UNITS
$(P_b)_c$	borehole pressure for a coupled charge	psi, Pa
$(P_b)_{dc}$	borehole pressure for a decoupled charge	psi, Pa
$P_{b_{BUFF}}$	borehole pressure for a charge in the buffer row	psi, Pa
$P_{b_{PROD}}$	borehole pressure for a charge in the production row	psi, Pa
r	blasthole radius	in., cm
r_{BUF}	radius of blasthole in the buffer row	in., cm
r_c	charge radius	in., cm
r_h	blasthole radius	in., cm
r_{PROD}	radius of blasthole in the production row	in., cm
S	hole spacing	in., cm
S_T	tensile strength of rock	psi, Pa
S.B.D.	safe blasting distance	ft, m
S.S.D.	safe scaled distance for cylindrical charges	ft/(lb per delay)$^{\frac{1}{2}}$, (m/kg per delay)$^{\frac{1}{2}}$
T	dynamic tensile rock strength for use in pre-splitting	psi, Pa
v	peak particle velocity	in./sec, cm/sec

		UNITS
W	charge weight per delay	lb per delay, kg per delay
x	absolute value of site attenuation factor, m_2	none
ρ	specific gravity of explosive	none
ρ_m	mass density of rock	$lb.sec^2/ft^4$, kg/m^3