AN INVESTIGATION TO EVALUATE THE RELATIONSHIP,
BETWEEN ROCK QUALITY INDEX (RQI) AND POWDER
FACTOR FOR SURFACE MINING

by

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ABSTRACT

The determination of the blastability of the rock mass is a problem that faces blasting engineers on a daily basis. The costly trial and error approach may eventually lead to a suitable design, although constant variations in the rock mass properties presume that this approach is constantly in progress. Shortly after the publication, at UBC, of the Leighton report on the correlation between the Rock Quality Index (RQI) and the controlled powder factor defined at Afton Mine, this research project was established in order to determine the extent of such correlation. The Rock Quality Index is obtained from rotary blasthole drill performance data. Defined as a single numerical estimation of the rock mass properties, the RQI reflects variations in the rock quality. It is a simple and practical approach to blast design based on an empirical relationship.

At Equity Silver, Lornex and Greenhills Mines, data were gathered and evaluated in order to confirm the relationship and determine the factors that influence or control its definition. It was found that many factors actually affect the determination of the relationship. They are easily divided in two categories, the factors related to the drilling mechanism or to the blasting process. The accuracy of the drilling data was of concern to Leighton and a drill performance recorder was used at two of the locations in order to obtain accurate data.
This research project permits the conclusion that the basic idea of a correlation between Rock Quality Index and the rock mass blastability is right. The correlation is, at this time, site specific due to the difference in the RQI values obtained from various drilling equipment and also due to the specific blasting conditions. Since many variables are found to affect both the drilling and blasting mechanisms, basic guidelines are proposed in order to assure the success of further research on the Rock Quality Index.

Optimum blasting is the key to improved slope stability, higher stripping ratio, high productivity and reduced maintenance cost. However, optimum blasting is achieved only with a well established rock mass characterization program.
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CHAPTER 1
1.0 **INTRODUCTION**

Open pit slope stability is governed by three major factors: structural geology, groundwater and blasting. They differ in the sense that mine operators have better control over blasting than they do with the first two. However, when careful engineering is used when designing a pit slope, very little is generally available during the excavation of the same slope. Although blasting has became more a science than an art, many mining operations are still breaking the rock inefficiently. In some cases, the willingness of the blasting engineer to minimize production downtime leads him to the utilization of a high powder (energy) factor, therefore causing damage to the rock mass outside the pit perimeter with a resulting flatter slope angle. On the other hand, when urged to keep the direct blasting costs to a minimum, the blasting engineer will often underload the blastholes and choked blasts will occur. In addition to the pit wall damage, tough digging conditions reduce productivity and increase equipment maintenance.

Therefore, there is a major economic incentive toward the achievement of optimum blasting practice. Steeper and stable pit walls and well fragmented muck piles represent reduced stripping ratio and operating costs. Many mine operators have realized the situation and understood the fact that a more profitable balance sheet does not generally coincide with the minimum blasting costs. Consequently, they are planning blast
optimization programs.

Blasting optimization starts with the characterization of the rock mass. The blasting engineer must determine the blastability of the rock and allocate to it a certain powder (energy) factor. However, because of the inhomogeneous nature of the rock, the determination of blastability is a very difficult task. Consequently, blast optimization programs are generally performed as costly trial and error processes. Nevertheless, several methods have been proposed in order to determine rock mass blastability and therefore reduce the extent of the trial and error approach. All are handicapped by major drawbacks such as the method is costly, it requires skilled personnel or special equipment; or it may be inaccurate, too general or site specific; or finally, the method is just not practical in a mining operation.

A research project, directed by Professor C.O. Brawner of the Mining and Mineral Process Engineering Department of University of British Columbia, has been directed toward the determination of a simple and practical method of assessing the rock mass blastability. The proposed method is not based on any particular rock mass property but rather utilizes the tricone rotary bit as a testing tool to evaluate a measure of the strength of the rock mass, and therefore takes into account all rock mass properties as a whole. It is believed that the large body of data available from the blasthole drill logs could be interpreted to yield a useful indication of the rock mass
blastability. These data are already collected for other purposes by the driller and consequently does not require any additional expense. It consists of the penetration rate (hole depth and drilling time) and the weight on the bit (down pressure). From these data, the Rock Quality Index (RQI) is defined as:

\[
RQI = \frac{W}{PR} = \frac{Wt}{d} = \frac{Pt}{d}
\]

where \( W \) = weight on the bit
\( PR \) = penetration rate
\( t \) = drilling time
\( d \) = hole depth
\( P \) = hydraulic down pressure

and with \( W = f(P) + Cte \)

Very good results were obtained by John C. Leighton at Afton Mine when correlating RQI and optimum design powder factor for perimeter blasts.
1.1 THE 1980 TO 1982 RESEARCH PROJECT

This section briefly summarizes the reports published by Leighton in 19821-3.

The first field program was performed at Afton Mine during the summer of 1981. In a period of over six months, Leighton standardized and optimized the controlled blasting procedure to the extent where the powder (energy) factor was the only design variable left while monitoring the drill performance in nine geological domains of the pit (Figure 1.1-1). The statistical analysis of the RQI distribution within each domain showed that the average value was representative of the distribution. The graph shown in Figure 1.1-2 was then derived, representing the optimum design powder (energy) factor for perimeter blasting in each domain versus the RQI allocated to the particular domain at Afton Mine.

The following is a quotation of the first two conclusions reported by Leighton1:

"1) The action of a rotary drill is affected by major rock mass properties and structural geology, causing it to reflect the competency of the rock on a qualitative basis."

"2) Provided that the hydraulic down pressure and penetration rate are carefully monitored when drilling each blasthole, the Rock Quality Index can be calculated and will serve as a reliable indicator of the rock mass condition."
FIGURE 1.1-1: ROCK QUALITY INDEX VALUES FOR EACH DOMAIN RANKED IN ORDER OF INCREASING QUALITY. (after Leighton¹)
FIGURE 1.1-2: PROPOSED CORRELATION BETWEEN ROCK QUALITY INDEX AND POWDER FACTOR AT AFTON MINE. (after Leighton)
It was also specified that the relationship was, at this time, only valid for the particular bit design and drilling equipment (chisel shape WC, BE 40R). Finally, Leighton left the discussion open on topics related to the effect of bit wear on RQI, the use of a drill performance recorder as a means of obtaining more accurate data and the possibility of using the RQI to predict grinding rate. Leighton considered the relationship may be site specific and it should be evaluated at each mining project.
1.2 THE 1982 TO 1984 RESEARCH PROJECT

This research project was undertaken in September 1982 following the publication of Leighton's thesis report¹.

The objectives were the following:

1) evaluate the validity of the correlation
2) define the limits of applicability (if any)
3) investigate the influence of major parameters
   i) the level of accuracy of the drilling data
   ii) the different drilling equipments
   iii) the different blast designs
   iv) the rock mass behaviour during drilling & blasting

Backed by an extensive literature review, the field program was performed during the summer of 1983 in three open pit operations in British Columbia.

The author obtained, free of charge, a drill performance recorder from the Geolograph Pioneer Company. The three mines where field testing was performed were:

1) Equity Silver Mine (5 weeks)

The Equity Silver Mine is a silver-antimony-gold property located in northern British Columbia. They are using 200 mm (7 7/8 inch) and 229 mm (9 inch) diameter bits on Bucyrus Erie 40-R drills. Benches are 5 m (16.4 ft) high. The intensity of in-situ fracturing is highly variable. The blast results were generally rated from fair to good. Simple modifications of the
blast design were implemented during the research project. A drill performance recorder was installed on the 229 mm (9 inch) diameter drill.

2) Lornex Mine (5 weeks)

The Lornex Mine is a large copper producer located in the central part of the province. They are using 250 mm (9 7/8 inch) diameter bits on five BE 45-R drills. At this site, although fracturing is intense, the degree of alteration also modifies the strength of the rock mass. The blast designs were not optimum and modifications were proposed. The drill performance recorder was also used at this site.

3) Greenhills Mine (3 weeks)

The Westar Mining Greenhills property is a new coal mine located in southeastern British Columbia. They are using 270 mm (10 5/8 inch) diameter bits on Reed SK-60II drills. The characteristics of the different beds of sedimentary rocks between the coal seams suggest that different powder (energy) factors could be used from seam to seam. Nevertheless, the well managed blasting procedure was designed to produce a fine muck pile for the medium scale hydraulic shovels. Calibration problems ruled out the installation of the drill performance recorder.
Data from approximately 1200 blastholes were reviewed from each site.

The first part of this thesis contains three chapters. Each covers a particular aspect of the literature review that was undertaken. Chapter Two is directed toward the understanding of the rock mass failure behaviour and analyses the different parameters that influence fracture development. The different failure criteria are also reviewed. Chapter Three is concerned with the drilling process. The principal objective being to review all the factors that influence the penetration rate, from the rock mass properties through the operating parameters. The relationship between the theoretical analysis and the practical drilling process is always present. The last chapter of this first part deals with the blasting process. The factors that influence the explosive consumption are presented in detail. The rock mass behaviour during the blasting process is also covered.

The second part of this thesis presents the procedures, results, recommendations and conclusions of the field research program.
1.3 REFERENCES


3. LEIGHTON, J.C.; Predicting Powder Factors from Rotary Drill Performance for Controlled Blasting, 14th Canadian Rock Mechanics Symposium, Vancouver, 1982
CHAPTER 2
2.0 ROCK FRAGMENTATION

The fracture of rock is a prime consideration in mining\(^1\)\(^-\)\(^6\). While the design of underground openings in rock attempt to eliminate fractures, other mining processes like drilling, blasting and grinding are directed toward the failure of the rock mass.

It is found appropriate to review the principal theories of failure of brittle rock, the influence of the rock mass properties on the different fracture processes and the similitude between them prior to the detailed analysis of drilling and blasting action on the rock mass. First, brittle failure should be defined. Quoting Hoek\(^7\): "Brittle failure is said to occur when the ability of the rock to resist load decreases with increasing deformation." However, the deformation is usually minimum, thus leading to sudden failure. Within the published theories of failure, Mohr's approach is the more accepted and practical. He predicts failure on any plane when the shear stress exceeds the combined effect of cohesion and friction (effective normal stress times the tangent of the friction angle) on that particular plane. The Griffith concept bridges the gap between theoretical and real strength of the rock mass. It is later reviewed in order to obtain a better explanation of actual results. Finally, Hoek and Brown developed an empirical failure criterion that predicts the fracture behaviour of both the rock mass and intact rock
specimen.

The structural characteristics of the rock on both large and small scale are expected to produce local anomalies in the stress distribution. The stress concentration is a function of the normal stress applied, the frictional resistance and the cohesion of the plane of weakness.
2.1 REVIEW OF ROCK FAILURE MECHANISMS

2.1.1 Rock Mass Characteristics

Under stress, rock behaviour differs from other engineering materials. This is due to the fact that the rock is usually neither homogeneous nor isotropic. Heterogeneity occurs on both small and large scale. Fracture propagation is a function of rock mass properties such as the position of the discontinuities in space, the boundaries between different rock types, the variation in the magnitude and orientation of in-situ stress field, etc. On a smaller scale, crystal structure variations, voids, inclusions, grain boundaries and preferred orientation of mineral constituents will influence the stability or continuity of fracture growth. Geological history also affects rock mass properties and thus its behaviour, and makes each site different. (Ex. Weathering)

The brittleness of the rock is mainly caused by the variations of the physical properties of its constituents and internal structural imperfections. Brittleness and weakness of the rock are generally associated with its heterogeneity.

2.1.2 The Coulomb-Navier Theory of Failure

The Coulomb-Navier theory predicts that failure will occur in a material when the maximum shear stress at one point reaches a critical value, i.e., the shear strength of the material.
Moreover, it is predicted that the failure plane will bisect the angle between the maximum and minimum stresses. This latter statement is not verified by experiment. It is also possible to demonstrate from this theory that the shear strength in tension equals the shear strength in compression, a situation that never occurs in brittle material. Navier proposed to modify this theory. He postulates that the normal stress acting on the failure plane increases the shear strength of the rock proportionally to its magnitude. He introduces the coefficient of internal friction. The Coulomb-Navier theory predicts that the compressive strength is larger than the tensile strength, but at a ratio lower than indicated by experiment.

2.1.3 The Mohr Theory of Failure

The Mohr theory predicts that a material will fracture when the shear stress on the failure plane has reached a limiting value, a function of the normal stress on this plane, or if the largest tensile principal stress equals the tensile strength of the material. The relationship between the shear stress and the normal stress has to be determined experimentally and is represented by the envelope of failure. The Coulomb-Navier failure criterion is represented on a Mohr diagram with a straight line envelope, and is a special case of the Mohr theory. Note that sedimentary rocks like limestone and sandstone are illustrated with a curved Mohr envelope whereas
more brittle rocks like granite and quartzite possess straight envelopes. The Mohr criterion also predicts the direction of the failure plane and the state of stress in the material at failure. (Figure 2.1-1)

2.1.4 The Griffith Theory of Failure

The Coulomb-Navier and the Mohr theories consider the failure on a macroscopic scale, whereas the Griffith theory approaches the failure on a microscopic scale to explain rock mass behaviour. In 1921, Griffith published a theory that is later used to explain the low strength of the rocks compared to the theoretical strength of perfect crystals. He hypothesized that fracture of brittle materials is due to high tensile stresses induced at the extremities of small, randomly oriented microscopic cracks. The cracks create a stress concentration that overcome the theoretical strength of the material even though the applied stress is much lower. Griffith postulates that no force is carried across the cracks. He also assumed that cracks were equally oriented in space. Failure occurred when the crack having the worst orientation reached a stress value larger than the strength of the material or a limiting value of local strain energy. However, the Griffith theory fails to explain some experimental results.

First, the criterion predicts an increase of the triaxial compressive strength with an increased confining pressure, but
FIGURE 2.1-1. GRAPHICAL REPRESENTATION OF MOHR THEORY OF STRESS CONDITIONS FOR FAILURE OF INTACT ROCK.

(after Hoek and Brown⁷)
many scientists observed a larger augmentation of strength than
predicted. Also, with the presence of pore pressure, the
Griffith criterion predicts that the strength of the rock will
increase. Again, the theory underestimates the real increase.
Finally, the Griffith theory indicates that the compressive
strength should be eight times the tensile-strength of the rock.
Experiments reported values of compressive strength to tensile
strength ratio between 5 and 22. Even when anisotropic
distribution of the Griffith cracks is considered, the theory
does not seem to be in accordance with experiments.

McClintock and Walsh\textsuperscript{8} developed the idea that it is
possible for the cracks to close and thus carry normal and shear
stresses due to friction. These stresses will increase the
strength of the rock by reducing the stress concentration at the
ends of the cracks. Similarly, fluid under pressure in the
pores will modify the stress distribution in the rock. This
modified theory predicts a compressive to tensile strength ratio
of 10.

Fracture propagation explained by this theory is described
by Walsh and Brace\textsuperscript{9}. The frictional force resisting shear
between crack surfaces is proportional to the normal force
transmitted across these surfaces. Non-uniform stress in the
rock develops local stresses at microcrack tips that eventually
exceed the strength of the surrounding materials. The crack
begins to grow, but only to a length in the order of magnitude
of the grain size, and stops when it intersects other cracks.
The direction of propagation is generally parallel to the maximum compressive stress. If the load is increased, other less favorably oriented microcracks will be initiated or re-initiated. However, no single crack grows catastrophically and produces a macroscopic fracture, but under sufficient stress, regions of high crack density will create a macroscopic shear plane.

The Griffith failure criterion may be represented by a parabolic Mohr envelop. As modified by McClintock and Walsh, the Griffith theory of failure is represented by a straight envelope on a Mohr diagram.

2.1.5 Hoek and Brown Empirical Failure Criterion

Hoek studied crack initiation and propagation and concluded that both the original and modified Griffith theories were satisfactory when dealing with the rock mass, but less accurate when describing the failure of intact laboratory specimens\(^1\) (Figure 2.1-2). Nevertheless these theories were the starting point in the development of this empirical failure criterion.

Hoek and Brown derived their failure criterion by trial and error. They experimented with curve fitting in order to correlate both the original Griffith theory, under tensile stress, and the failure data produced under compressive stress situations. In spite of the absence of a fundamental relationship between the empirical constants included in the
FIGURE 2.1-2: MOHR CIRCLES FOR FAILURE OF SPECIMENS OF QUARTZITE TESTED BY HOEK. ENVELOPES INCLUDED IN THE FIGURE ARE CALCULATED BY MEANS OF THE ORIGINAL AND MODIFIED GRIFFITH THEORIES OF BRITTLE FRACTURE. (after Hoek)
criterion and any physical characteristic of the rock, the criterion adequately predicts rock fracture behaviour. The criterion defines the magnitude of the major principal stress at failure as a function of the minor principal stress, the uniaxial compressive strength of intact specimens of the rock mass and two empirical constants. (Figure 2.1-3)

It is assumed that the intermediate principal stress has no influence on the failure mechanism and that the major and minor principal stresses are always the effective stresses, obtained by considering the pore or joint pressure in the rock. The empirical failure criterion of the rock mass is related with both CSIR and NGI rock mass classification systems, thus permitting the evaluation of the empirical constants. (See Table 1)
FIGURE 2.13: GRAPHICAL REPRESENTATION OF HOEK AND
BROWN FAILURE CRITERION.

(after Hoek10)
### Table 1

**Approximate Relationship between Rock Mass Quality and Material Constant (after Hoek and Brown)**

<table>
<thead>
<tr>
<th>Empirical failure criterion</th>
<th>Carbonate rocks with well-developed crystal cleavage</th>
<th>Lithified argillaceous rocks, sandstone, shale and slate cleavage</th>
<th>Arsenic rocks with strong crystals and poorly developed cleavage</th>
<th>Fine-grained polymetallic igneous crystalline rocks, dolomite and marble</th>
<th>Coarse-grained polymetallic igneous and metamorphic crystalline rocks, amphibolite, gneiss, granite, schist, diorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Empirical failure criterion</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$g_1 = g_2 + \sqrt{g_3 g_4 - s g_5}$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$t = g_6 (g_7/ g_8 - T) B$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>$T = \sqrt{m - \sqrt{n^2 + k}}$</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>INTACT ROCK SAMPLES</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Laboratory size specimens</td>
<td>$m = 7.0$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>free from joints</td>
<td>$s = 1.0$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 100</td>
<td>$A = 0.816$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 500</td>
<td>$B = 0.658$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>VERY GOOD QUALITY ROCK MASS</strong></td>
<td>$m = 3.5$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Tightly interlocking undisturbed rock with unwethered joints at 1.3m.</td>
<td>$s = 0.1$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 85</td>
<td>$A = 0.651$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 100</td>
<td>$B = 0.679$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>GOOD QUALITY ROCK MASS</strong></td>
<td>$m = 0.7$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Fresh to slightly weathered rock, slightly disturbed with joints at 1 to 3m.</td>
<td>$s = 0.04$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 65</td>
<td>$A = 0.369$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 10</td>
<td>$B = 0.669$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>FAIR QUALITY ROCK MASS</strong></td>
<td>$m = 0.14$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Several sets of moderately weathered joints spaced at 0.3 to 1m.</td>
<td>$s = 0.0001$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 44</td>
<td>$A = 0.198$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 1.0</td>
<td>$B = 0.662$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>POOR QUALITY ROCK MASS</strong></td>
<td>$m = 0.04$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Numerous weathered joints at 30 to 50mm with some gouge - clean waste rock.</td>
<td>$s = 0.00001$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 23</td>
<td>$A = 0.115$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 0.1</td>
<td>$B = 0.646$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>VERY POOR QUALITY ROCK MASS</strong></td>
<td>$m = 0.007$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Numerous heavily weathered joints spaced &lt; 50mm with gouge - waste with fines.</td>
<td>$s = 0$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CSIR rating 3</td>
<td>$A = 0.042$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>NGI rating 0.01</td>
<td>$B = 0.534$</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
2.2 DRILLING, BLASTING AND GRINDING

In the majority of mining operations, the reduction of the rock mass to a suitable size for loading and handling is done through drilling and blasting. At the plant, the size reduction of the rock fragments below a certain limit facilitates extraction of valuable components or chemical action. In all cases, a uniform size of the product is the best. However, drilling, blasting and grinding produce a range of fragment sizes. This is related to the fact that energy is transmitted to the rock by compression under concentrated load. This concentrated load produces a non-homogeneous stress field and energy distribution which result in a non-efficient fragmentation process and a non-uniform size of fragments.

In the zones of high stress, failure of the rock under compression produces a very fine, powder-like material. At low-stress levels, the failure is of a brittle type, resulting in larger fragments. The size of the fragments is a function of the degree of heterogeneity on a small scale and of the intensity of jointing on a larger scale.

The energy consumption in the fragmentation process is a function of the relative amount of fines produced and their degree of fineness\textsuperscript{11-13}. It should be realised that the relative energy used in drilling rocks exceeds the energy used to blast the same rock (Figure 2.2-1). This statement is well understood when one imagines how the size of the rock specimen
FIGURE 2.2-1: COMPARISON OF ENERGY REQUIREMENTS FOR VARIOUS FRAGMENTATION PROCESSES.

(after Page^{12})
to be broken influences its strength. As the size is reduced, the imperfections in the structure are reduced and the material increases in strength and, therefore, needs more energy to continue the size reduction. Nevertheless, it is important to note that the energy consumption in all the rock breakage processes is also a function of a number of other parameters such as the indentor's geometry in drilling and the depth of the charge or presence of a free face in blasting.

The optimum conditions of drilling and blasting are the ones that need the least amount of energy to produce a given set of results. The following chapters will deal with the analysis of the fragmentation processes in drilling and in blasting.
2.3 REFERENCES

1. CLAUSING, D.P.; Comparision of Griffith's Theory with Mohr's Failure Criteria, 3rd Symposium on Rock Mechanics, Quarterly of the Colorado School of Mines, N3, July 1959


3. FAIRHURST, C.; CORNET, F.; Rock Fracture and Fragmentation, 22nd U.S. Symposium on Rock Mechanics, Massachusetts Institute of Technology, Cambridge, Massachusetts, July 1981

4. RATIGAN, J.L.; An Examination of the Tensile Strength of Brittle Rock, 23rd U.S. Symposium on Rock Mechanics, Goodman editor, AIME, 1982


6. MORRISON, R.G.K.; A Philosophy of Ground Control, Dept. of Mining and Metallurgical Engineering, McGill University, Montreal, 1976


9. WALSH, J.B.; BRACE, W.F.; Mechanics of Rock Deformation,


12. PAGE, C.H.; Blasting and Comminution, unpublished report, Steffen, Robertson and Kirsten

13. YOUNG, C.; Rock Fragmentation - Needs and Possibilities, Rock fragmentation session of the 17th U.S. Symposium on Rock Mechanics, University of Utah, Utah Engineering Experiment Station, 1976
CHAPTER 3
3.0 THE DRILLING PROCESS

Drilling is the initial operating sequence in open pit mining. It is also one of the highest operating costs involved and consequently the drilling process is optimized accordingly to the site conditions. Different combinations of bit designs and operating parameters are tested in order to define the one that minimizes the overall costs and maximizes the penetration rate.

The rotary drills face the undisturbed rock mass and, with the knowledge of the principles of tricone rotary drilling, useful rock mass data can be derived from drill performance. The use of a tricone rotary bit as a valid rock testing tool is not new, although until recently, scientists were restricting the field of application to the prediction of the drillability or abrasiveness. The first part of this chapter is therefore an overview of the tricone bit design considerations. These design considerations are not basically a pure mechanical problem controlled by the size and strength of the different bit components and the stresses applied on them. The rock mass failure behaviour influences the bit design and the overall drilling process, whereas the stresses are very large.

In order to understand the action of a tricone rotary bit on the rock mass, it is necessary to review the theoretical mechanisms of rock penetration by a single tooth. Different models have been proposed and are presented in the second
section of this chapter. A discussion on the concept of specific energy in drilling follows.

In addition, the study of the factors that influence rock mass drillability is of prime importance in this research project. Fortunately, a large number of scientists have been concerned with the determination of the rock mass drillability and a wide body of literature is available. These factors are of two types: the mechanical properties of the rock mass being drilled and the factors related to the particular drilling tool used and the operating parameters.

The drillability section is followed by a review of the tricone rotary drilling in practice. This fifth section also covers drilling formulas. Finally, the chapter concludes with a discussion on the Rock Quality Index.
3.1 TRICONE ROTARY BITS

Howard R. Hughes patented the first rotary rock bit on August 10, 1909. This invention was later improved to become one of the most economical rock penetrating tools. (Figure 3.1-1) Roller cone bit improvements were steady over the past twenty years because of significant design optimization of the cutting structure materials and of the bearings. Today, carbide insert bits with increased rate of penetration and longer life have largely replaced milled-tooth bits (Figure 3.1-2).

The principal limitation in the design of a roller cone bit is the space. The size of every piece is determined by the diameter of the hole and the size of other components. In most cases, one part can only be made larger if another part (or parts) is reduced in size. The selection of the optimum dimensions results in a series of compromises until the various elements are properly balanced to give the best overall performance. Manufacturers claim that the bit design is mainly based on past experience. The fundamentals of mechanical design are seldom used. The severe conditions in which rock bits work force all parts to withstand larger stresses, in relation to the ultimate strength of the material, than the ones commonly encountered in other mechanical designs. Also, the simplicity of the design is the key to reliability. Figure 3.1-3 shows a tricone rotary bit used in the blasthole industry. The importance of each major component, related to the rate of
IMPORTANCE OF DEVELOPMENT (arbitrary scale)

- 1909: FIRST ROTARY ROCK BIT
- 1925: SELF-CLEANING CONES
- 1935: UNITIZED BIT BODY
- 1940: FIRST TRI-CONE BITS
- 1947: CONE OFFSET AND SPECIFIC TYPES
- 1951: HARD ROCK BITS
- 1954: SOFT FORMATION BITS
- 1959: TUNGSTEN CARBIDE INSERTS
- 1961: JET BITS
- 1964: SEALED BEARINGS
- 1969: SHAPED INSERTS AVAILABLE
- 1971: SOFT INSERTS, JOURNAL BEARINGS

(after Estes)
FIGURE 3.1-2: ROLLER BIT IMPROVEMENT (after Brown et al.²)
FIGURE 3.1-3: NOMENCLATURE OF ROTARY BIT. (after Dresser Industries)
penetration and the life of the bit, is described in the following sections.

3.1.1 **Journal Angle and Skew Angle**

Bottom hole action is a function of the position of the journal on the arm. The position of the journal in relation to the horizontal plane defines the journal angle. The variation of the journal position with the axis of the bit is the skew angle or offset.

The most basic design element of a tricone rock bit is the journal angle also termed pin angle. For any given bit diameter, the cone size is a function of the journal angle. The cone diameter decreases with an increase in the journal angle. Soft rock bits generally have a low journal angle.

Another variation of the journal position is the offset. It is defined as the horizontal distance between a vertical plane through the axis of the journal and the bit axis. (Figure 3.1-4) The skew angle is the angle subtended by the offset and a pre-selected point on the journal axis. This offset is designed for soft to medium rock bits to provide a gouging and scraping action. This slight sliding movement of the cone increases the drilling rate or, reduces the thrust requirement for a given drilling rate. However, the amount of offset is a function of the rock formation because skewed cones result in accelerated carbide wear in abrasive formations and introduce side load to
FIGURE 3.1-4: TRICONE BIT OFFSET. (after Hughes Tool Co.)
3.1.2 Teeth and Carbide Inserts

Tungsten carbide insert bits were introduced in 1951 to drill extremely hard and abrasive formations that were very costly to drill with conventional steel tooth bits. The cutting structure of a tungsten carbide insert bit resists abrasion and high compressive loads. Components of cylindrical sintered tungsten carbide pressed into machined holes in alloy steel cones form the teeth of this type of bit. The cost differential between a tungsten carbide bit and a steel-toothed bit is offset by the increased footage. Nevertheless, after labour, drill bits constitute the most significant cost component of large diameter blasthole drilling.

The first carbide inserts utilized were hemispherical ended because of the inherent strength of this shape. This shape is still the best for hardest formations. Information gained by testing a large number of bits permit manufacturers to upgrade their design. Insert shape, spacing and positioning are functions of the rock type to be drilled. Very profitable blasthole bit performance has been achieved with shaped carbide inserts (Figure 3.1-5). Tungsten carbide bits are now used in a wide range of rock formations. In Appendix 1, different manufacturers models are listed as a function of the rock formation.
FIGURE 3.1-5: VARIOUS INSERT SHAPES COMMONLY USED IN BLAST HOLE ROCK BITS. (after Toll\textsuperscript{5})
Hard rock bit:
As previously specified, hemispherically shaped end compacts were found the most efficient and durable in hard rock drilling. The compact nose projection over the cone surface is small and permits the compact to resist high impact load without breaking. There is no offset on these bits so the cones rotate on a true rolling movement and thus, abrasive wear is minimized. The compacts are closely spaced on the cones and penetration is obtained by a crushing-chipping action on the rock. The rock fails under pure impact and indentation of the cutter elements.

Medium-hard rock bit:
The tungsten carbide inserts in this type of bit possess a bullet shaped conical end with more compact projection. The rate of penetration of these bits is faster than the ball nosed compact bits in the medium-hard formation.

Medium-soft rock bit:
In this case, the chisel shaped insert bit was found the most successful design in the softer plastic formations. The cones of these bits are slightly skewed to increase the rate of penetration and the scraping-gouging action slowly replaces the crushing-chipping type of rock breakage found with harder formation rock bits.

Soft rock bit:
The teeth of a soft formation tricone rotary bit are broad and thin, deep and widely spaced. In 1933, the development
of rollercone bits with inter-fitting teeth made possible the use of longer teeth without a corresponding reduction of the cone size. Thus, soft formation bits can efficiently utilize longer carbide on the cones, allowing for more wear. Also, since the teeth penetrate the rock deeper, skew is needed to dislodge the rock more easily. Provided with long lasting bearings, the tungsten carbide compact bits have been used with success in the softest formations.

In summary, the spacing and the length of the compacts increase with an increased softness of the formation. The number of compacts increases as the formation hardness increases. Even if the total number of compacts has to be high so each tooth load is not too large, the number must be few enough to make sure that when the teeth become worn, a sufficient unit load will still be applied to the formation and provoke failure. Finally, the nose shape increases in roundness as the formation becomes harder.

3.1.3 Bearings

The bearings are important parts in a tricone rotary bit. They transmit the weight to the inserts at the rock contact. Tough drilling conditions will rapidly shorten the bearing life, and thus, lead to the failure of the entire bit. The bearing life is a function of the intensity of the unit loads transmitted by the bit and the nature of the bottom hole
environment. Roller cone bits are occasionally subjected to large dynamic loading which lower the bit life substantially. However, the environment was found to have a much greater influence on the life of the bearings than the dynamic loading does. In some mining operations, water is injected into the air circulation line to avoid the formation of dust particles. Medlock demonstrated the effect of water on bearing life and his results are shown in Figure 3.1-6. He found that clean, dry air was the best circulation fluid for maximum bearing life and also, that the life increases with the rate of air flow.

The different bearing segments are shown in Figure 3.1-3. They have specific functions, but again, compromises must be made until all elements are well balanced. The radial loading caused by the weight on the bit is taken by the roller bearings and by the nose friction bearing. The number and size of rollers is a function of the bearing pin roller race. A maximum number of rollers reduces the unit loading and thus, the spalling and wear. On the other hand, the diameter of the rollers must be large enough to support the load.

The same design considerations hold for the ball bearings, but in this case, another constraint has to be fulfilled. The thickness of the sides of the ball race must be of an appropriate thickness to resist breakage. This is due to the fact that even if the ball bearings carry some of the radial load, especially if some wear has occurred on the end of the pilot pin, their most important function is to hold the cones on
FIGURE 3.1-6: RESULTS OF ROLLER BEARING TESTS.

(after Medlock\textsuperscript{8})
the arm journals.

The nose bearing which resists outward thrust is made of a pilot pin, hard faced with a special alloy and ground smooth, fitted into a bushing since no space is available for another row of rollers. The bushing is an antifriction metal lining made of stainless steel. In order to keep the unit load to a minimum on this arrangement of friction bearing, a large diameter of adequate length is required.

3.1.4 Gage Design and Shirttail Protection

In the recent years, gage design has been improved in order to minimize premature wear of the tricone bit. The influence of gage wear on the bit performance is important. The dullness or breakage of gage compacts makes rock drilling along the wall of the hole inefficient. This causes the three cones to encounter inward pressures as shown on figure 3.1-7. The pressure comes from the weight applied on the bit in combination with the rounded gage. The resulting force is directed toward the center of the bit. However, bit bearings are designed to withstand outward thrust produced under normal drilling conditions. The presence of inward thrust on the bit creates large pressures on the inner bearings and speeds up wear until complete failure.

The abrasive wear that occurs on gage inserts also affects the bit shirttail. Tungsten carbide inserts are used to delay rapid wear of the lower part of the shirttail. Without heavy
FIGURE 3.1-7: EFFECT OF SIDE LOADS ON BIT BEARINGS.

(after Dresser Industries⁴)
protection the chances are that the roller bearings could eventually be exposed and allowed to drop out, breaking the bit.

3.1.5 Aircourses

Aircourses are passageways for the circulating air. Although approximately twenty percent of the total compressed air is diverted through the bearings for cooling and cleaning purposes, the air is first used to flush the cuttings from the hole bottom to the surface. The importance of cutting removal on drilling performance will be described in section 3.5.1. At this point, only the circulation of air through the bit is considered.

Tricone rotary bits are differentiated into two types as far as the air circulation is concerned:

1. Conventional bits which direct the air right above the cones.
2. Jet bits which direct the air onto the hole bottom, between the cones.

Conventional bits have long been standard, but they are now replaced by the jet bits. The principal disadvantages relate to the fact that the flushing medium is passed around the cones before it reaches the hole bottom, thus reducing its ability to move the cuttings and, in many cases, causing severe erosion of the cone shell and teeth at high velocity. Jet bits have three nozzles on the periphery of the bit body through which the air
runs directly to the hole bottom. This design gives superior cleaning of the chips and minimizes the sandblasting effect on the cutting structure.

Figure 3.1-8 shows a cross sectional view of the air courses of a jet bit. Through the bearings, the air first runs through screens or filters at the entrances to the bearing diversion. This prevents the bearings from being clogged with debris carried in the air. Then the air is discharged in the nose of the cone and in the ball race, cooling the bearings and keeping them free of foreign material. Finally the air escapes through the cone openings, behind the gage. The major part of the air volume flows through the nozzles that are of adjustable diameter.
FIGURE 3.1-8: CROSS SECTIONAL VIEW OF A JET BIT.

(after Steinke⁹)
3.2 PENETRATION OF BRITTLE ROCK

Drilling with tricone rotary bits causes the rock to break by the combination of crushing-chipping actions. If the bit is loaded under sufficient weight, compressive stress developed at the end of the tungsten carbide inserts overcomes the material's compressive strength. The rock directly below the insert is crushed in a fine powder-like material, whereas the stress concentration around the crushed zone ultimately develops discrete fractures. The propagation of these fractures to the intersection of the hole bottom forms the drilling chips. This later mode of failure is associated with the tensile or shear mode.

Even though these stresses are dynamic in nature, it has been demonstrated that they are applied slowly enough to be simulated by static loading conditions. Thus, tricone rotary drilling can be simulated as an indentation process. The knowledge of the rock penetration process has been developed through experimental and mathematical research. The factors that affect the mechanics of rock penetration are the applied force, the strength properties of the rock and the geometry of the tooth[10]. Different penetration models has been proposed. Although they included some simplifications, they contribute to the understanding of the drilling process.

3.2.1 Stresses Beneath the Indentor
The force applied by the insert creates a complex stress field beneath it that is essentially compressive. The stress distribution is a function of the design of the indentor and the rock conditions (smooth vs. irregular). Tandanand proposed a simplified mathematical approach to define the stress condition under a spherical indentor (Figure 3.2-1).

At the surface, the radial, tangential and vertical stresses are in compression. However, the radial stresses change into tension at the boundary between the insert and the rock and is the largest tensile stress (top right). The largest compressive stress is the vertical stress, induced at the center of the contact surface and equal to the load on the insert. Here, the rock fails in the crushing mode (i.e., tri-axial compression). The maximum shear stress is found on the z axis at a point dependent on the value of Poisson's ratio of the indented material \((z/a = 0.5)\).

Sikarskie has published a description of a wedge penetrating the rock in quasi-static conditions and the formation of chips:

i) Initially, formation of crushed material at wedge tip with increasing applied force.

ii) Primary fracture forms and extends into virgin material with increasing applied force.

iii) During the crushing phase, stresses are increasing in the surrounding material mass, until, at some point, secondary fractures are initiated.
FIGURE 3.2-1: STRESS PROFILE ON CONTACT SURFACE AND ON AXIS OF SYMMETRY.

(after Tandanand)
iv) These fractures grow in a stable manner with increasing load until crack instability is reached, whereupon rapid crack growth (with little or no increase in force) extends the fracture to the free surface. A chip is formed.

Two types of failure are said to occur beneath an indentor. Immediately around the tip of the insert, the rock is crushed in tri-axial compression. This type of failure produces powder-like material and does not optimize the use of energy available (i.e., volume of broken material/unit of energy). The second type of failure is the tensile failure. Away from the zone of tri-axial compression, the confinement decreases and the rock possesses a more brittle behaviour. If the tensile stress is large enough, crack initiation occurs.

The volume of material submitted to tri-axial stress can be reduced by modifying the indentor geometry or by reducing the lateral confinement.

3.2.1.1 The Indentor Geometry

Figure 3.2-2 shows the graphical representation of a model developed by Reichmuth. On the basis of purely elastic analysis, he has derived the following relationship:

\[
\tan \beta > \frac{(\pi - 2U)}{(2 + \pi U)}
\]

\[
\tan \beta \leq \frac{(\pi - 2U)}{(2 + \pi U)}
\]

Where \( \beta \) = semi-included wedge angle.

\( U \) = coefficient of friction between the tooth and the rock.
FIGURE 3.2-2: MEDIA AFTER SECONDARY FAILURE HAS OCCURRED. (after Reichmuth\textsuperscript{13})
In the case 2a, the stress distribution is effectively localized in a region below the tooth, causing considerable crushing and compaction of the rock and inhibiting the formation of chips. The intensity of the compressive stress, transmitted from the insert to the intact rock through the crushed zone, decreases as the wedge angle increases. In addition, the preferential growth orientation of the secondary cracks is steeper and the frictional forces tend to exert greater restraint on the material under a wider wedge angle. Conversely, with a sharper wedge, as in case 2b, secondary fractures are produced closer to the free surface and chips are formed. In this case, very little compaction and crushing of the rock is developed in the tooth's nose area.

From the above, it is concluded that the force required to obtain a given penetration depth increases rapidly with an increase in the wedge angle. Experiments have confirmed this conclusion (figure 3.2-3). Moreover, these experiments also illustrated that there was a change in the failure behaviour of the rock when wedge angle were increased over 75 to 90 degrees, in accordance with the Reichmuth model. With a sharp wedge, rock is believed to behave in a brittle manner whereas with a larger wedge angle, the rock behaviour is more ductile. As a result, insert shape has a much greater effect on drilling rate for drills which are load limited (roller-bit) because a large insert will not penetrate as deep as a sharp one, resulting in a
FIGURE 3.2-3: CHARACTERISTIC FORCE-PENETRATION CURVES FOR CHARCOAL GRAY GRANITE UNDER STATIC BIT LOADING.

(after Reichmuth)
reduced drilling rate\textsuperscript{16}. Although small wedges are more efficient, larger wedge angles are used to keep wear within reasonable limits.

3.2.1.2 Indexing

The interaction of a bit tooth with a previously formed crater is referred to as indexing. When an insert is loaded adjacent to a previously formed crater, the tensile fracture generally progresses toward its direction and the chip breaks into the crater. Typical force-displacement curves are shown in figure 3.2-4 for indexed tooth penetration. In general, the average slope of a force-displacement curve decreases with decreasing indexing distance. Indexing is more efficient because of the reduced proportion of crushed material formed per unit volume of broken rock. The optimum indexing distance may be as great as five times the penetration depth. In a research project conducted by Gnirk and Cheatham\textsuperscript{18}, the optimum indexing distance was related to the width of the insert and the penetration depth. This is of importance in determining the optimum spacing of individual insert on a roller-bit and for the selection of the tooth angle for various rock types.

3.2.2 Tool Penetration Models

Theoretical analysis of brittle rock penetration by a rigid tool, although simplified, are the basis of our understanding of
FIGURE 3.2-4: AVERAGE FORCE PENETRATION CURVES FOR CARTHAGE MARBLE AS OBTAINED WITH 30 AND 60 DEGREE SHARP BIT-TEETH AT ATMOSPHERIC PRESSURE FOR VARIOUS INDEXING DISTANCES (after Cheatham and Gnirk[17])
the drilling process. This paragraph will review four of them.

3.2.2.1 Paul and Sikarskie Penetration Model

The rock penetration model proposed by Paul and Sikarskie\textsuperscript{19} predicts the gross behaviour of rock under a loaded insert at shallow depth. In this model, it is assumed that:

i) The Mohr-Coulomb criterion is satisfied everywhere on the failure plane at the instant of failure.

ii) The fractures occur along planes extending from the insert tip to the free surface at an inclination (to the surface) of:

$$
\psi = \frac{\pi}{4} - (\beta + \phi)/2
$$

with $\beta =$ half included wedge angle

$\phi =$ angle of internal friction

iii) The force penetration curves are linear during the crushing phase of penetration.

iv) The rock is considered as a material which exhibits both crushing near the insert tip and the formation of chips on both sides of the penetrating wedge, although the model analysis omits the crushed zone.

The force penetration curves predicted by the model are shown in figure 3.2-5 for both constant load and constant rate. The actual loading conditions on a wedge are somewhere between these two extreme conditions. The chips are formed when the crushing curves intersect the failure line. The crushing curves
FIGURE 3.2-5: THEORETICAL FORCE-PENETRATION CURVE FOR BRITTLE CRATER MODEL. (after Paul and Sikarskie)
slope \( k \) is experimentally determined whereas the failure line slope is given by:

\[
K = 2 S_0 \sin \beta \left(1 - \sin \phi \right) / \left(1 - \sin(\beta + \phi) \right)
\]

where \( S_0 \) = Uniaxial compressive strength
\( \beta \) = Semi-included wedge angle
\( \phi \) = Angle of internal friction

The assumptions of planar chip surface and simultaneous fracture can be viewed as an upper bound type of solution and therefore, the penetration depth predicted by this model is the minimal depth that can be obtained at a given load. This model is conservative. Moreover, the crushing phase of the penetration process is not an exact brittle fracture problem. The pulverized nature of the material and the high hydrostatic stresses in the vicinity of the wedge tip make the material behave more or less plastically.

The model predicts the cessation of chip formation when the semi-included tooth angle exceeds \( \left( \pi/2 - \phi \right) \). Cheatham and Gnirk\(^{17}\) noted that since most rocks exhibit an angle of friction of approximatively 30 degrees, the above expression implies that only crushing will occur for a tooth angle of approximatively 120 degrees. However, the observed range of value for numerous varieties of rock is 75 to 90 degrees.

3.2.2.2 Bauer and Calder Penetration Model
The rock penetration model proposed by Bauer and Calder\(^2\) is based on the study of indentor penetration in rock. They concluded from a series of indentation tests that, for indentors shapes commonly employed in drilling, the force-penetration relation is either linear or becomes linear as penetration proceeds. In addition, it was apparent that the indentors "seat" themselves on a shell of very compacted material as the applied load increases. It results that the insert-rock contact area also becomes constant as penetration proceeds. Therefore,

\[ \frac{dF}{dh} = Ka \]

where \( F \) = total applied force on indentor

\( K \) = rock penetration constant

\( A \) = horizontal projection of indentor area at depth \( h \)

\( K \) is simply the slope of the force-penetration curve when it becomes linear, divided by the contact area (i.e., this constant is not equal to \( K \) or \( k \) of the Paul & Sikarskie model). Figure 3.2-6 shows a plot of the uniaxial compressive strength versus \( K \). The correlation is excellent. The value of \( K \) for different shapes of indentor is given by the following equations:

i) constant area indentors: \( K = \frac{F}{hA} \)

ii) wedges: \( K = \frac{F}{(L \tan(a/2)h^2)} \)

where \( L \) = length of wedge

\( a \) = included angle of wedge (2\( \beta \))
FIGURE 3.2-6  ROCK PENETRATION CONSTANT VS UNIAXIAL COMPRESSIVE STRENGTH. (after Bauer and Calder$^{20}$)
iii) hemispheres: \( K = \frac{F}{\pi h^2(r - h/3)} \) for \( h \leq r \)

iv) cones: \( K = \frac{3F}{\pi h \tan^2(a/2)} \)

The total penetration obtained when an indentor is forced into rock in rotary drilling is made of two parts. Firstly, a portion caused by the indentor penetration, usually less than 10% (termed \( h \)), and secondly a portion caused by the stress fracture beneath the indentor tip (termed \( d \)). The value of \( d \) is obtained from the following equations:

i) line load (cylindrical envelope of diameter \( d \) caused by a wedge)
\[
d = \frac{F(1 - \sin \phi)(\omega - 1)}{\pi \text{Sc} \sin \phi}
\]

ii) point load (spherical envelope of diameter \( d \) caused by a cone or a sphere)
\[
d = \frac{3F(1 - \sin \phi)(\omega - 1)}{4\pi \text{Sc} \sin \phi}
\]

where \( F \) = applied load

\( \text{Sc} \) = uniaxial compressive strength

\( \phi \) = angle of internal friction

\( \omega = \tan^2 (45 + (\phi/2)) \)

In practice, however, the depth of rock failure due to the stress field beneath the indentor could be as low as 0.4d due to indentor geometry or indexing effects.

3.2.2.3 Pariseau and Fairhurst Penetration Models

The rock penetration models proposed by Pariseau and
Fairhurst\(^2\) are derived from the application of plasticity analysis in wedge penetration of rock. The most practical approach is the one that considers the existence of a false-nose composed of finely crushed material at the apex of the wedge (see Figure 3.2-7). Penetration tests confirm the existence of such a zone. The force-displacement characteristic for this condition is given by the following equation:

\[
\frac{F}{hbSo} = \frac{Tan\beta}{(Tan\phi Tan\mu)} \left[exp(\pi Tan\phi) - Tan^2\mu\right]
\]

and thus

\[
h = \frac{F}{MbSo}
\]

where \(F\) = applied force

- \(b\) = bit contact edge (= cte = unity)
- \(So\) = uniaxial compressive strength
- \(M\) = \(\frac{Tan\beta}{(Tan\phi Tan\mu)} \left[exp(\pi Tan\phi) - Tan^2\mu\right]\)
- \(\beta\) = semi-included wedge angle
- \(\phi\) = angle of internal friction
- \(\mu = \frac{\pi}{4} - \frac{\phi}{2}\)

In this model, however, the plastic region extends from the bit face to the rock surface. It is likely that in practice, the plastic region is of finite dimension and does not extend to the rock surface. Then brittle failure as well as plastic failure accompanies penetration. They approached this problem by assuming a non-linear failure criterion. Because the Coulomb envelope is for many rocks a linear approximation of what is in fact a non-linear failure envelope, they examined a parabolic
FIGURE 3.2-7: ASSUMED STRESS FIELD FOR 'FALSE NOSE' SITUATION.

(after Pariseau and Fairhurst$^{21}$)
type of failure criterion. They derived the following relationship:

\[ \frac{F}{h b S_0} = \frac{\sin \phi_0 \tan \beta}{\sin \phi} \left[ \frac{1}{2} \frac{\sin (2 \theta + \beta)}{\sin \beta} \right] + \frac{(T_0 \tan \beta)}{(2 S_0 \sin \phi_0)} \]

and thus, \( h = \frac{F}{M' b S_0} \)

where \( F = \) applied force
\( b = \) bit contact edge (=cte=unity)
\( S_0 = \) uniaxial compressive strength
\( T_0 = \) uniaxial tensile strength \( T_0 \leq 0 \)
\( \phi_0 = \) inclination of the yield envelop to the S-axis at the point it touches the uniaxial compressive strength stress circle
\( \phi = \) variable = \( f(\sigma_m) \)
\( \sigma_m = \frac{1}{2} (\sigma_1 + \sigma_3) \)
\( \sigma_1 = \) major principal stress
\( \sigma_3 = \) minor principal stress
\( \beta = \) semi-included wedge angle
\( M' = \left[ \frac{(\sin \phi_0 \tan \beta)}{\sin \phi} \right] \left[ \frac{1}{2} \frac{\sin (2 \theta + \beta)}{\sin \beta} \right] + \frac{(T_0 \tan \beta)}{(2 S_0 \sin \phi_0)} \)
\( \theta = \frac{\pi}{2} + \frac{1}{2}(\phi_0 - \phi) + \frac{1}{2}(\cot \phi_0 - \cot \phi) \)

The problem with this equation is the fact that \( \phi \) is a function of the major and minor principal stresses encountered in the area of plastic behaviour (ie: beneath the indentor). This value is obtained by knowing the equation of the failure envelop and the magnitude of the stresses.
3.3 SPECIFIC ENERGY

The concept of specific energy can be applied to all fragmentation processes as shown in figure 2.2-1, it is defined as the energy required to fracture a certain volume of rock. The amount of energy required to fracture the rock is dependent on the fragments size and the fragmentation process. In drilling, it is a quantity that depends not only on the strength of the rock, but also on drilling parameters such as the geometry of the bit, the size of the hole, the cutting removal method, etc. The specific energy in drilling considers all the modes of failure occurring at the rock-bit interface and the total energy losses. During the drilling process, the rock is initially broken and chips are formed, but because perfect cleaning condition never occurs, the rock is reground. Such effect implies that a certain amount of energy is also consumed in a comminution process and therefore, the specific energy in drilling is a collective measure of the specific energies required for crushing, chipping and grinding the rock \(^2\).

The concept of specific energy in drilling has been intensively studied by Teale\(^2\). He postulated that, for a given tool, to excavate a given volume of rock, a certain theoretical minimum quantity of energy will be required. This amount of energy will be a function of the nature of the rock mass, and the difference between the theoretical requirements and the real process would be a measure of the work dissipated in the
communion process, in friction between the tool and the rock and/or in mechanical losses in the system. In other words, this difference is a measure of the efficiency of the process.

In tricone rotary drilling, the specific energy reaches very high values at low thrusts. This is caused by the fact that below a certain value, the thrust will be inadequate to force the tool into the rock. As the volume of rock excavated will tend toward zero, a finite amount of work will still be done to overcome friction. But as the thrust increases, the size of particles broken also increases and the friction loss will constitute a rapidly decreasing portion of the total energy input. Therefore, the specific energy will decrease. Nevertheless, at high thrust values, the tool is heavily pushed into the rock and clogs. The reduction in the efficiency of the process will cause the specific energy to rise again. The lowest value obtained is a measure of the maximum mechanical efficiency of that particular tool in the particular operating conditions into a given rock. This quantity is in the order of magnitude of the uniaxial compressive strength of the rock. Tests performed by Teale\textsuperscript{23} showed that the ratio of minimum specific energy to uniaxial compressive strength ranges between 0.8 and 1.6.

Although Teale noted that the uniaxial compressive strength is not an absolute measure of the rock strength nor possesses physical similarity with the drilling process, he suggested that there must be some kind of relationship between the two.
However Mellor\textsuperscript{24} published the results of research in which he examined the validity of the use of the uniaxial compressive strength as a normalizing factor for the specific energy in fragmentation process. He pointed out that the uniaxial compressive strength is actually proportional to the specific energy in breaking the rock in uniaxial compression (ie., $S.E.\text{ucs} = \sigma \varepsilon * 10^{-3}$) and therefore can be used as a normalizing factor.

Even though, according to Teale, the ratio of minimum specific energy/uniaxial compressive strength is around one for optimum operating conditions, in practice, the values range from 0.3 to 3.
Drillability can be defined as the resistance of the rock to penetration. Unfortunately, there are as many drillability indexes as there are drillability tests. This results in a wide divergence on norms and in confusion. Drillability indexes are used to predict the drilling performance in a given rock mass. However, on this research project, we are using the drilling performance to define an index of the rock mass failure properties. For this reason, the review of the factors controlling drillability has been undertaken and presented in this paragraph.

Two general methods exist to determine the rock mass drillability:
1) Drilling the rock
2) Analysis of the mechanical properties of the rock

3.4.1 Drilling Tests

This method has been used by many scientists. The procedure is generally close to the following. A typical drill-bit arrangement is selected. The rock is drilled under strict conditions (thrust, rpm) up-side down to obtain perfect cleaning conditions. The bit wear, drilling time or energy input are monitored for a given drilled depth and eventually result as a drillability index. In some occasions, the index is related, with different degrees of success, with physical properties of
the rock. Drillability indexes derived from this method depend not only on the rock properties, but also on the equipment used and the operating conditions. Although the procedure is logical, the experimental conditions have to closely simulate the field conditions to result in a reliable drillability scale.

In open pit drilling, drillability tests are generally performed by bit manufacturers. Even though some companies promote the indentation test others are using a more empirical approach called the microbit test. Details of the test procedure are given in the literature\textsuperscript{26}. The main factor in this approach is that the results of this test are correlated with other microbit drillability results of rock formations on which full size bits have been used. Figure 3.4-1 is a chart showing an empirical correlation of known formations with microbit data. The drilling rate obtained from this chart is modified by experimental factors (0.5 to 0.7) to reflect the average drilling rate for the life of the bit. In general, the results of this test are conservative. As a matter of fact, it should be noted that the results are as good as the degree of representativity of the tested specimen over the whole rock mass. In addition, no allowance is made for the abrasiveness of the rock.

3.4.2 Analysis of the Mechanical Properties of the Rock

One of the first drillability indexes was the Moh's scale. According to Protodyakonov\textsuperscript{25}, a drillability scale based on
FIGURE 3.4-1 ESTIMATED DRILLING RATE OF NEW HUGHES TRICONE ROCK BITS AT 60 rev/min. AS DETERMINED BY MICROBIT DRILLING RATE TESTS.

(after Rollow26)
standardized method of measuring the mechanical properties of rocks is a more rational approach. Different techniques exist to measure the strength of the rock:

1) Resistance to destruction by elementary stresses (compression, tension, shear)
2) Resistance to local destruction (hardness, indentation)
3) Resistance to breakage (grindability)

The uniaxial compressive strength of the rock shows a significant correlation with percussive drillability although it is not always reliable in tricone rotary drilling. Rollow gave the example of Virginia dolomite and Gray granite, both at 214 MPa (31000 psi), where the drilling rate, under the same load, is three times greater in dolomite than in granite. Although no close relationship exists between compressive strength and drilling rate, a relation exists between the weight on the bit (W) and the compressive strength. Tandanand suggested that the poor correlation is due to the fact that the magnitude of strain involved in the failure process in drilling is not proportional with the failure strain caused in uniaxial compression because the two modes of failure are different. It implies that the elastic properties of the rock are to be considered too. He proposed the shore scleroscope hardness test to measure the amount of elastic and inelastic deformations of the rock. Others suggested indentation tests in order to obtain a quantitative hardness classification that relates to the
resistance of the rock to bit tooth penetration. Results from this test can be used with a theoretical expression of the drilling rate (see section 3.5).

The drillability is also dependent on the structural characteristics of the rock mass. A coarse-grained structure is easier to drill and causes less wear than a fine-grained structure. In addition, the degree and type of fracturing, alteration, cementing material and the angle at which the discontinuity planes are drilled affect drill performance. Figure 3.4-2 shows the increase in penetration rate in relation to the number of weakness planes for a percussive drill. One of the rock mass properties that reflects the mechanical behaviour and the intensity of fracturing is the sonic velocity. Gstalder and Raynal published a relationship between the drilling penetration rate, the Young's Modulus and the sonic velocity (Fig 3.4-3). The good correlation suggests that the drill performance can reflect the failure behaviour of the rock. Finally, a drillability index was proposed over thirty years ago and was defined as:

\[ K = \frac{dS}{d(W/D^2)} \]

where \( K \) = drillability

- \( S \) = penetration rate (in/min)
- \( W \) = weight (lb)
- \( D \) = diameter (in)

It is interesting to note the similarity between \( K \) and the RQI.
FIGURE 3.4-2: ESTIMATED RELATIVE INCREASE IN PENETRATION RATE VERSUS NUMBER OF WEAKNESS PLANES PER METRE DRILL-HOLE (after Blindheim\textsuperscript{30})
FIGURE 3.4-3: CORRELATION OF DRILLING PERFORMANCE WITH ROCK CHARACTERISTICS. (after Gstalder and Raynal)
3.5 TRICONE ROTARY DRILLING

In tricone rotary drilling, the energy is transmitted to the bit-rock interface via a loaded rotating pipe. Flushing air is supplied through the drill pipe to remove cuttings at the hole bottom and transport them to the surface. The major parameters in this process are the weight or thrust on the bit (W), the rotary speed (RPM), the torque (T) and the air pressure and volume. These parameters are inter-related and it is not possible to define the optimum operating condition for a given size and type of bit in a given rock type. Trial tests must be performed. Nevertheless, the understanding of the influence of one parameter over the others or its effects on the penetration rate or drilling costs would certainly reduce the extent of the tests.

3.5.1 Drilling Parameters

3.5.1.1 The Weight on the Bit

According to Fish\textsuperscript{32}, the thrust is one of the most important variables in the drilling process. It is applied on the bit by hydraulic pressure thru cylinders or by other mechanical processes. Consequently, the hydraulic pressure reading is the pressure in the system and since every manufacturer has a different pressure to weight ratio, it is recommended to use the weight rather than the down pressure when
comparing drill performances. Appendix II gives the
gerelationships between pressure reading and weight on the bit for
several drill rig models.

The intensity of the loading at the bit-rock interface
defines four types of drilling conditions as shown in figure
3.5-1 and described in section 3.3:

1) abrasive phase: the formation is worn away

2) fatigue phase: the formation is loaded several time before
rock failure occurs

3) spalling phase: sufficient amount of weight causes the
formation to fracture readily

4) founder phase: excess weight buries the carbides to full
depth

Although the rate of penetration increases proportionally
to $W^{**a}$ (where $1 \leq a \leq 2$), it should be noted that the
relationship is only valid during optimum conditions. In the
founder phase or when the hole is improperly cleaned, increasing
the weight will not increase the penetration rate. In practice,
the weights used on tungsten carbide bits range from 0.7 to 1.4
tonnes per centimeter (2 to 4 tons per inch) of bit diameter.
The stronger the rock mass, the higher the load. If sufficient
load is needed to overcome the compressive strength of the rock,
excessive weight can cause insert breakage and reduce bearing
life although bearings are larger and stronger in hard formation
tricone bits. In a study performed at a mine site in Labrador,
it was recognised that the thrust on the bit was the most
FIGURE 3.5-1 DRILLING CONDITIONS (after Dresser Industries)
important parameter in tricone rotary drilling.

3.5.1.2 The Rotary Speed

The rate of penetration of a rock bit is generally increased, when rotating speed is increased, by a ratio slightly less than 1:1 up to the limits of proper hole cleaning\(^5\). The factors that determine the maximum rotary speed are the bearing's wear, the insert breakage and the drill rig limitations. Bearing life is inversely proportional to rotary speed. Although tungsten carbide is a very hard and wear resistant material, it is also brittle and inserts can be subjected to excessive breakage at high rotary speed. In hard formations, the practice is to decrease the rotary speed as the weight is increased. The opposite is also true (Figure 3.5-2). In practice, rotary speed in almost constant at a given mine site or within the domains.

3.5.1.3 The Air Pressure and Volume

The controllable factor of air is air pressure, modified by changing the jet nozzles of the bit. The situation is the following: by using small size nozzles, the air pressure increases, forcing more air to flow through the bearings. This procedure will generally extend the bearings life. However, the air volume is reduced and poor bailing velocity would result. The poor bailing velocity will cause poor bottom hole
SELECTING THE RIGHT BIT

FIGURE 3.5-2: RELATIONSHIP BETWEEN ROCK COMPRESSIVE STRENGTH, PENETRATION RATE, WEIGHT AND RPM (after Steinke)
conditions, therefore excessive wear of the bit components and
decrease in the drilling efficiency at a given thrust. On the
other hand, high bailing velocity can cause cone erosion and
extreme wear too⁴ (sand blasting). The optimum bailing velocity
is a function of the hole conditions (rough or smooth wall), the
rock density, the type of cuttings (shape, wet or dry), etc. In
any case, with high penetration rate, the bailing velocity has
to be increased in order to clear the generated cuttings.

3.5.1.4 The Torque

The torque is not exactly an operating parameter. It is
the application of the thrust and the rotation of the tool on
the rock mass that produce the torque in the system. In tricone
rotary drilling, with true rolling bits, very little torque is
developed. However, a high torque may be developed by a bit
with intense wear of the gage structure³. In soft formations,
the rock is drilled by the scraping-gauging effect of tricone
bit teeth and the intensity of this action is related with the
torque developed³⁸. Normally, drills are designed to sustain
torque between 1.6 to 3.2 joules per 10 Kg (10 to 20 ft-lbs per
100 lbs) of thrust, although this amount is seldom necessary⁶.
The torque developed is directly proportional to the penetration
rate.

3.5.2 Drilling Equations
Many theoretical and empirical drilling equations have been published in the past thirty years. Many others will probably be published in the future. In this paragraph, some of the proposed drilling equations are reviewed in order to summarize the drilling process.

The simplest drilling equations are:

Maurer\textsuperscript{16}

\[ PR = \frac{(dV/dt)}{A} \]

where \( PR \) = penetration rate,

\( V \) = volume,

\( t \) = time,

\( A \) = area of the hole

Morris\textsuperscript{10}

\[ PR = Np \]

where \( N \) = rotary speed,

\( p \) = penetration per revolution

Although those equations are simple and theoretically true, their applicability require a more detailed evaluation of the drilling process. The theoretical approach is based on the stress concentration and failure of the rock beneath individual loaded inserts. Unfortunately, the complexity of the equations increases quickly due to the number of variables that have to be considered.

Maurer\textsuperscript{16}

\[ PR = K(NW^2/A)^s \]
where \( s = 1 = \) good cleaning
\( s = 1/2 = \) poor cleaning
\( K = \) rock mass constant
\( W = \) weight on the bit
\( N = \) rotary speed
\( A = \) area of the hole

The following equations assume perfect cleaning conditions.

Morris\(^{10}\)

\[
PR = Nb \left( \frac{p'}{E} \right) \left( \frac{W}{0.08C} \right)
\]

where \( b \approx 1.8 \) and converts bit rotary speed to cone rotary speed.
\( C = \) total number of inserts per bit.
\( \left( \frac{p'}{E} \right) = \) rock penetration factor from indentation hardness tests.

Gnirk and Cheatham\(^{18}\)

\[
PR = 0.156mNl \left( \frac{w}{D} \right)^2 \left( \frac{(W/wnt1op)}{2} - 75.69 \right)
\]

where \( m = \) number of penetrations per revolution
\( N = \) rotary speed
\( l = \) length of cutting edge of a single bit-tooth (inch)
\( W = \) applied weight (lbs)
\( D = \) bit diameter (inch)
\( nt = \) number of bit-teeth effectively in contact with the rock at the bottom of the drill hole at any instant
\( w = \) bit-tooth flat width (inch)
\( \sigma_p = \text{compressive strength of the rock at differential pressure } p \) (psi)

Some scientists have preferred a more practical approach and derived empirical drilling equations. These are usually a more useful tool in engineering estimate:

Bauer and Calder\(^\text{20}\)

\[
PR = ((61 - 28\log_{10} Sc) \ W \ N) / (250 \ D)
\]

where \( Sc \) = uniaxial compressive strength

\( W \) = applied weight

\( N \) = rotary speed

\( D \) = hole diameter

Cunningham\(^\text{26}\)

\[
PR = N (W^{a}) / 0.424 \ \sigma_d^{1.5}
\]

where \( \sigma_d \) = drilling strength

\( a = f(\sigma_d) \) and \( > 1.1 \)

\( N \) = rotary speed

\( W \) = applied weight

The derivation of the Cunningham equation is reviewed in detail in the future section because of its similarity with the Rock Quality Index.
3.6 **ROCK QUALITY INDEX**

It has been shown, in this chapter that the penetration rate is a function of the operating parameters, the bit design and the rock properties. The general relationship is expressed as:

\[ \text{PR} = W^{**a} \times (\text{RPM})^{**b} \times f(\text{rock mass properties, drilling procedures}) \]

However, at a given mine site, the drilling procedure is generally constant. In addition, the variation in rotary speed are limited, when compared with the variation in the applied weight on the bit, the latest being the principal operating parameter. The general relationship can therefore be expressed as:

\[ \text{PR} = W^{**a} \times f(\text{rock mass properties}) \]

and

\[ f(\text{rock mass properties}) = (W^{**a})/\text{PR} \]

This was the conclusion derived by Mathis. He defined the Rock Quality Index as:

\[ \text{RQI} = W/\text{PR} \]

In a research project conducted by Cunningham and directed toward the expression of a simple relationship between the drilling parameters, it was found that an empirical relationship was a more workable approach than a theoretical one. Although there is no indication that Cunningham reviewed
the work of Mathis, the conclusions are quite similar. The
relationship expressed by Cunningham was given in section 3.5
and is repeated here:

\[ PR = K(W^{**a})N \]

where \( N \) = rotary speed
\( W \) = weight
\( a, K \) = constant
\( PR \) = penetration rate

and if \( N \) is kept constant:

\[ PR = K(W^{**a}) \]

Then, based on the general relationship between the
compressive strength and the weight required to drill a
formation at a specific rate, Cunningham constructed a family of
lines relating the penetration rate and the applied weight
(Figure 3.6-1). Those lines are termed lines of equal drilling
strength (\( \sigma d \) expressed in thousand of PSI). The drilling
strength is treated as a single physical property of the
formation drilled and has little meaning in any other context.
It is a numerical expression of the rock mass behaviour during
the drilling process, with tricone bits. Cunningham then
expressed \( K \) and \( a \) as functions of \( \sigma d \), and the relationship
became:

\[ PR = W^{**a} / 0.424(\sigma d^{**1.5}) \]

with \( a = (0.178254 \ln(\sigma d) + 1.09793) \quad 1.1 <= a <= 1.9 \)

In addition, the drilling strength, which is derived from
FIGURE 3.6-1: DRILLING RATE VS WEIGHT PER INCH OF BIT DIAMETER. (after Cunnihaham²⁸)
drilling tests, can be approximated by the uniaxial compressive strength when the tests are not performed.

The work by Cunningham therefore confirms the simple relationship proposed by Mathis. The W/PR ratio quantitatively expresses the variation in rock quality.

The practical advantages of the Rock Quality Index are numerous. The index is quickly and easily obtained without additional costs or personnel and provide an unbiased coverage of the entire pit area, since all rock must be drilled before being blasted. The mathematical determination of the RQI is also very simple and easy to understand because of its empirical nature. This is an important consideration when dealing with the practical application of the index in the field.
3.7 **SUMMARY**

Tricone rotary drilling is a complex rock breakage process. It is influenced by the tool design, operating parameters mainly $W$ and rock mass properties. The theoretical optimum drilling condition corresponds to the least amount of energy per unit volume of broken rock. However, in practice, the optimum drilling condition is determined by a cost evaluation of the process.

$$DC = ROC + B/L$$

where $DC = $ drilling costs ($/ft$)

$B = $ bit cost ($$

$L = $ life of bit (ft)$

$ROC = $ drill operating cost ($/ft$)

with $ROC = f($machine cost, operator wages, effective drilling rate$)$

It is possible that the practical optimum drilling condition does not coincide with the theoretical optimum.

Nevertheless, the rock mass properties are reflected on the drill performance and the establishment of the Rock Quality Index provide a useful method of rock mass characterization.
3.8 REFERENCES


3. Rotary Drilling Bits, published by Hughes Tool company, Houston, Texas

4. Blasthole Bit Technology for the 80's, published by Dresser Industries Inc.


8. MEDLOCK, J.D.; Laboratory Testing of Rotary Rock Bits, Drilling and Blasting Symposium, Quaterly of Colorado School of Mines, V56, N1, January 1961


10. MORRIS, R.I.; Rock Drillability Related to a Rollercone
Bit, Dresser Industries Inc.


14. GNIRK, P.F.; CHEATHAM, J.B.; An Experimental Study of Single Bit-Tooth Penetration into Dry Rock at Confining Pressures 0 to 5000 psi, Society of Petroleum Engineers Journal, June 1965


16. MAURER, W.C.; The State of Rock Mechanics Knowledge in Drilling, Failure and Breakage of Rocks, Ch. 15, AIME, 1967

17. CHEATHAM, J.B.; GNIRK, P.F.; The Mechanics of Rock Failure Associated with Drilling at Depth, Failure and Breakage of Rocks, Ch. 17, AIME, 1967

18. GNIRK, P.F.; CHEATHAM, J.B.; A Theoretical Description of


22. TANDANAND, S.; UNGER, F.H.; Drillability Determination - a Drillability Index for Percussion Drills, USBM, report of investigation 8073, 1975


26. ROLLOW, A.G.; Estimating Drillability in the Laboratory,

27. WHITE, C.G.; A Rock Drillability Index, Quarterly of the Colorado School of Mines, V 64, N 2, 1969


30. BLINDHEIM, O.T.; Drillability Predictions in Hard Rock Tunnelling, Tunnelling '79, IMM, 1979


32. FISH, B.G.; The Basic Variables in Rotary Drilling, Mine and Quarry Engineering, V 27, N 1 and N 2, 1961

33. WILLIAMSON, T.N.; Rotary Drilling, Surface Mining, Sect. 6.3, Pfleider editor, AIME, 1972

34. Automatic Recorder of Drilling Parameters, General Presentation, Published by Jean Lutz, S.A., France

35. MATHIS, C.; Proposal of a Report on Rock Quality Index Based on Rotary Drill Performances, Unpublished paper, University Of Alberta, March 1975
4.0 THE BLASTING PROCESS

The quality of the blasting results influences the productivity of the mining operation. Ideally, the blasted rock should be well fragmented and heaved in order to maximize the production capabilities of the loading and hauling equipment, the crusher and the mill (autogenous).

The following quotation is from Hagan and Mercer:

"Mine management often concentrated on minimizing the cost of individual operating function without considering the effects of such cost reduction on dependent operations. All too often, explosives costs appear as a single entry in the mining cost statement, and undue significance is attached to them. Explosives cost, blasting costs or even total breaking costs must not be considered in isolation. Management should use only total production costs as a basis for determining the cost-effectiveness of an individual operation such as blasting." (Figure 4.0-1)

In addition, the rock slope angle has to be evaluated.

A shovel digging a well fragmented and displaced rock mass is twice as productive as a shovel digging a poorly fragmented and tight muck pile (Figure 4.0-2). The truck productivity is also increased, wear and tear on every piece of equipment is reduced. In appendix III is a list of possible benefits of improved fragmentation. Mine operators should acknowledge that large equipment are designed to handle larger volume of
FIGURE 4.0-1: EFFECT OF FRAGMENTATION ON COST OF MINING.

(after Hoek & Bray2)
FIGURE 4.0-2: OUTPUT FOR DIFFERENT SHOVEL TRUCK COMBINATIONS SINGLE CRUSHER, GOOD CONDITIONS AND POOR DIGGING. (after Bauer³)
materials, not larger size of fragments.

Nevertheless, optimum blasting results are obtained only when the blasting engineer understands the relationships between the rock mass properties and the well-balanced ratios of blasthole distances, bench heights, subdrilling and stemming, charge lengths and explosive strengths. A more scientific approach to the blasting process can result in an increase in productivity and therefore profitability.

In the first part of this research project, published by Leighton, the blasting process has been approached from the theoretical standpoint. His literature search has resulted in two chapters on the rock mass detonic and on the design considerations for controlled blasting. The objectives of the present report are such that to a certain extend, repetition will be avoided. Fortunately, the field of blasting can still be approached with different concepts. Because of the importance of the rock mass fragmentation of the mining operations, the author has decided to review the blasting process by relating the principal parameters to the their effects on the degree of fragmentation and indirectly, on the energy requirement.

In this chapter, two blasting theories will be reviewed in order to demonstrate how controversial this subject is. The core of the chapter will be concerned with the major factors affecting fragmentation. The design powder factor and its influence on the mining operation is a subject of prime
importance. However, in some instances, the mining operations dictate variations in the powder factor. A particular paragraph will deal with the design powder factor approach. Finally, a review of the rock mass blastability index completes this chapter.
4.1 THEORY OF BLASTING

The perfect explanation of the rock fragmentation process during blasting, supported by an accurate mathematical model that predicts results for any conditions, is still to be achieved. This field is confused and contradictory literature is abundant.

The blasting theories were grouped in two classes as a function of the relative importance accorded to the role of the stress waves or the expanding gases in the fragmentation process. All of them were obtained when working in homogenous materials, plexiglass or in rock specifically chosen to be as free as possible of major discontinuities. However, in most large scale production blasts, massive and joint free rocks are very seldom encountered.

4.1.1 Gas Expansion Theory

This theory is well described in the literature by Langefors and Kihlstrom. The chemical energy of the explosive is quickly liberated and transformed into high pressure gases. The gases cause a very rapid expansion of the borehole volume. This sudden expansion is eventually stopped by the elastic resistance of the rock, but it has nonetheless generated a strong compressive strain wave, which then travels through the rock mass, causing limited damage. The wave velocity is a function of the density and the geostructural properties of the
rock mass. It ranges between 3000 and 4500 meters per second (10,000 - 15,000 fps), being faster in dense and massive rock.

As the compressive wave travels through the medium, from the blasthole, it sets up tangential tensile stresses that create radial cracking. These cracks travel at a speed of about 0.4 times the velocity of the compressive wave. At the free face, the compressive wave is reflected back towards the borehole as a tensile stress wave. Slabbing of the rock surface may occur.

Those first two stages of rock breaking process are affected by the shock wave. But Langefords and Kihlstrom believe that the shock wave energy accounts only for 5 to 15% of the total theoretical explosive energy and is distributed all around the blasthole. It supposes that only 1/3 of the wave energy is actually used to break a definite burden. Thus, the shock wave, while not being responsible for the actual breakage of the medium, sets up the stage for the expanding gases.

The final part of the breaking process is done by the pressurized gases rushing into the radial cracks, wedging them open. The rock in front of the borehole then yields and moves forward. However, as the frontal surface moves forward, the pressure is reduced but tension is maintained and the radial cracks continue to grow. When the burden is optimum, several cracks will reach the free face and the complete loosening of the rock mass occurs. Blasting generally uses about 30% of the total explosive energy just in physically moving the rock.
4.1.2 Stress Wave Theory

This theory was first developed by the USBM in the 1950's. Later, it was demonstrated that a crater can be formed solely by strain wave breakage. However, during the last 15 years, this theory was second to the expanding gases theory. Recent work by Winzer and Ritter may soon lead to an improved strain wave theory of rock breakage. They observed that even as a small percentage of the explosive energy, the strain wave, in conjunction with the rock mass structural defects, plays a considerably greater role than previously though.

First, the tangential tensile strains caused by the compressive wave produce the radial cracks. Also, as the compressive wave reaches the free face with sufficient energy, it is reflected toward the blasthole as a tensile stress wave. The tensile stresses can initiate or re-initiate cracks. According to the Griffith theory, the tensile stress will have its maximum effect when tangential to the tip of the crack. This is equally valid for the microcracks or the radial cracks. Figure 4.1-1 shows that the angle subtended by the cracks tangential to the reflected strain wave is 138.2°. Blasting experiments have shown craters averaging 135° and confirm, to a certain extent, the Griffith theory during the blasting process. Also, it has been observed that the first visible cracks in the rock generally appeared not directly in front of the blasthole, but to one or both sides of it. They also noted that the
Mirror image of charge

Path of radial strain wave propagating at sound velocity of rock

Angle subtended by crater, 138.2°

Crack propagating at 0.38 of sound velocity of rock.

FIGURE 4.1-1: INTERACTION OF STRAIN WAVE WITH PROPAGATING CRACK.

(after Harries\textsuperscript{10})
fragments in motion from the free face continue to break-up, due to stress waves trapped in the block as it is detached from the face.

4.1.3 Practical Blasting Theory

At the present time, the blasting process is explained by both theories. Kutter and Fairhurst\textsuperscript{11} have demonstrated that optimum results are obtained only when the two mechanisms are present. They have well described each of them by using laboratory methods. But in the field, the highly destructive nature of blasting and the short interval during which it occurs make accurate measurement a technological challenge. Fortunately, as more sophisticated instruments are being developed, scientists will acquire more quantitative data on the blasting process. This will eventually improve our knowledge of this fragmentation mechanism.
4.2 THE MAJOR FACTORS AFFECTING FRAGMENTATION

Fragmentation of the rock mass by explosives is achieved by the optimization of a considerable number of design parameters. However, the rock mass is generally fractured and this important property cannot be controlled by the blasting engineer. Thus, his design has to be planned in conjunction with the structural properties of the rock mass.

This section will examine in detail the principal design parameters and rock mass properties in relation with their influence on the fragmentation process. Although the author would prefer to avoid the overlapping with part one of this research project, some of these factors have to be reviewed in order to highlight their importance on the blasting process and their influence on the results of this project.

4.2.1 Rock Mass Properties

Many authors have discussed the rock mass properties and their relationship with the fragmentation during blasting. Throughout this research project, two of them have appeared to overshadow the others. There are the geo-structural characteristics and failure behaviour of the rock mass.

4.2.1.1 Structural Geology

The effects of the rock structure on the blasting process...
and the fragmentation can be grouped in two categories: micro-scale and macro-scale effects. The micro-fissures have been studied by Dally et al\textsuperscript{12} in laboratory tests. They concluded that flaws improved fragmentation in two different ways. First, there are acting as crack initiation sites by the passage of the reflected tensile wave. Secondly, the flaws enhance branching by reducing the energy requirement to drive the cracks. Micro-fissures also influence the elastic behaviour of the rock mass. Preferentially oriented micro-structures induce large variations in the rock properties that can be revealed by wave velocity anisotropy.

Nevertheless, macro-fissures are the most important blasting variable as degree of fragmentation is concerned. They override any of the physical and mechanical properties of the rock\textsuperscript{13}. The existing bedding and jointing planes, in conjunction with the fracture pattern resulting from the previous blasts, have already defined the fragmentation limits. The blasthole, in many cases, is surrounded by a network of cracks that will serve as the least resistance path for the development and propagation of tensile or shear failure. During the blasting process, these cracks are extended to great length, and the formation of new cracks in their immediate vicinity is suppressed because of the absence of tensile stress development\textsuperscript{14}. Even where a new radial crack is developed, it terminates prematurely where it intersects a pre-existing crack. This latter statement is especially true when these pre-existing
cracks have been widened by gases from blastholes on previous delay periods. In those cases, the blast loosens the blocks and throws them into the muck pile with very little improvement in the fragmentation.

Another effect of the macro-structure is the gas expansion in major joints which intersect the blasthole wall. This leads to premature venting of the gases and flyrocks or stability problems, depending on the direction of the wedging mechanism. Despite the constraints brought over the fragmentation control by the rock mass discontinuities, the blasting engineer must be capable to produce the most economical and practical design. This is generally done by the selection of the orientation of the effective faces relative to dominant joint planes, bedding planes, etc'.

The easiest direction to blast is along the strike of the major set of discontinuities. The fragmentation is not generally as fine as it would be in other directions, but it allows the use of a reduced powder (energy) factor. The resulting crater shape is influenced by the structural discontinuities and areas of poor fragmented rock will likely occur when the spacing between the blastholes is too large (Figure 4.2-1). Smaller diameter and reduced spacing should correct this problem.

When the direction of blasting is at right angles to the major rock joints, it generally requires more energy to move the rock but the fragmentation will improve'. In some instances, the burden would need to be reduced in order to
FIGURE 4.2-1: ILLUSTRATIONS OF THE EFFECT OF ROCK STRUCTURE ON CRATER FORMATION. (after Bauer$^3$)
assure the total breakage of each slab. Blasting down-dip will often result in excessive back break and large toe burden, especially when the dip ranges from 40 to 60 degrees. The development of the face at an angle greater or equal at 45 degree to the dip direction will reduce the backbreak. However, when the dip is greater than 70 degrees, careful blasting produces stable and steep slopes. Blasting up-dip will be easier because of a reduced toe burden. But in the case where the toe burden should be equal, blasting down-dip is easier than blasting up-dip (Figure 4.2-2).

When dealing with quasi-horizontal bedding planes, attention should be paid to the collar area where poor fragmentation may occur although bottom detonation permits a helpful build-up of the shock wave toward the collar. Pocket charges are sometime necessary and will increase dramatically the blasting costs. Pocket charges would also be needed in the situation where hard rock beds lie between soft rock seams, within the height of the bench. If column charges are used, premature venting through the soft beds will result in poor fragmentation of the hard bands (Figure 4.2-3).

Coates and Gyenge have attempted a review of the relationships between the blasthole spacing S, joint spacing Sj and oversize specification M. Table 2 shows how there are inter-related. Reviewing these situations, cases 2, 3 and 5 are unlikely to occur as the spacing of the blastholes is less than the maximum specified oversize. The following discussion is
REDUCTION OF BURDEN

PREDOMINANT FRACTURING AT RIGHT ANGLES TO DIRECTION OF BLASTING

PLAN

LESS DIFFICULT DUE TO SMALLER TOE BURDEN

EASIER DIRECTION

SECTION

FIGURE 4.2-2: ILLUSTRATIONS OF THE EFFECT OF ROCK STRUCTURE ON CRATER FORMATION. (after Bauer)
FIGURE 4.2-3: ILLUSTRATIONS OF THE EFFECT OF ROCK STRUCTURE ON CRATER FORMATION.

(after Bauer³)
### TABLE 2

**EFFECT ON OVERSIZE FRAGMENTATION OF BLASTHOLE SPACING, S, JOINT SPACING, S_j, AND OVERSIZE SPECIFICATION, M**

(over Coates and Gyenge\textsuperscript{18})

<table>
<thead>
<tr>
<th>Case</th>
<th>S_j:S</th>
<th>S_j:M</th>
<th>S:M</th>
<th>Fragmentation Sensitive to Powder Factor?</th>
<th>% Oversize</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>S_j &gt; S</td>
<td>S_j &gt; M</td>
<td>S &gt; M</td>
<td>Yes</td>
<td>Medium</td>
</tr>
<tr>
<td>2</td>
<td>S_j &gt; S</td>
<td>S_j &gt; M</td>
<td>S &lt; M</td>
<td>Yes</td>
<td>Low</td>
</tr>
<tr>
<td>3</td>
<td>S_j &gt; S</td>
<td>S_j &lt; M</td>
<td>S &lt; M</td>
<td>Yes</td>
<td>Low</td>
</tr>
<tr>
<td>4</td>
<td>S_j &lt; S</td>
<td>S_j &gt; M</td>
<td>S &gt; M</td>
<td>No</td>
<td>High</td>
</tr>
<tr>
<td>5</td>
<td>S_j &lt; S</td>
<td>S_j &lt; M</td>
<td>S &lt; M</td>
<td>No</td>
<td>Low</td>
</tr>
<tr>
<td>6</td>
<td>S_j &lt; S</td>
<td>S_j &lt; M</td>
<td>S &gt; M</td>
<td>No</td>
<td>Low</td>
</tr>
</tbody>
</table>
quoted from Coates and Gyenge\textsuperscript{18}.

"Of the three likely cases, case 1 is of low probability as the joint spacing $S_j$, is greater than the spacing of the blastholes, $S$. However, if it occurred, the problem of obtaining fragmentation could be easily resolved with either the conventional use of larger diameter blastholes, correspondingly large spacing and above average powder factor. In case 4, where the representative joint spacing, $S_j$, is less than that of the blastholes, $S$, but greater than the maximum size of muck, $M$, the problem of a large percentage of oversize fragments is not usually solvable by increasing the powder factor. Reduced size of holes and spacing at the same powder factor can be effective, although it might be more expensive than accepting the cost of secondary breakage with the same conventional pattern. Case 6, which represents the ideal situation of the representative joint spacing $S_j$, being less than the maximum specified size of muck $M$, permit the use of larger diameter blastholes, large spacing and low powder factors."

The authors are not without specifying that this table can be used only as a means of quick evaluation of the site situation. Other variables like the variation of joint spacing and their degree of cementation are also important in relation to the production of oversize fragments.

Finally, as the frequency of discontinuities and planes of weakness decreases, a higher energy factor will be used in order
to create a larger number of fractures. This is generally done by generating a more powerful strain wave, as produced by high density slurries.

4.2.1.2 Failure Behaviour

The failure behaviour and energy absorption property of the rock mass largely influences the fragmentation process. The physical properties of the material that relate to the ability of the material to absorb, store and release energy are of the utmost importance. The modulus of Elasticity is a measure of the brittleness of the rock. Generally, rocks which show a high modulus of Elasticity also have a high compressive strength and so are harder to break. In addition, their deformation at failure is minimum.

A better evaluation of the brittleness is given by the analysis of the stress-strain curve of the rock in uniaxial compression to failure. Figure 4.2-4 shows such curves from brittle rock to creeping rocks.

It was found by Lang that the elastic brittle behaviour was associated with the shock wave type of failure of the rock during blasting. The more brittle the rock mass, the higher is the fracturing effectiveness of the strain wave. In this type of failure mode, the rock stores a large amount of energy and releases it violently. The fragmentation is generally good.

In less brittle rocks, however, plastic deformation occurs and a considerable amount of energy will be absorbed during this
### TYPICAL STRAIN BEFORE FRACTURE OR FAULTING (PERCENT)

<table>
<thead>
<tr>
<th></th>
<th>&lt; 1</th>
<th>1-5</th>
<th>2-8</th>
<th>5-10</th>
<th>&gt; 10</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>COMPRESSION</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>(\sigma_1 \geq \sigma_2 = \sigma_3)</td>
<td><img src="image1.png" alt="Diagram" /></td>
<td><img src="image2.png" alt="Diagram" /></td>
<td><img src="image3.png" alt="Diagram" /></td>
<td><img src="image4.png" alt="Diagram" /></td>
<td><img src="image5.png" alt="Diagram" /></td>
</tr>
<tr>
<td><strong>EXTENSION</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>(\sigma_3 \leq \sigma_1 = \sigma_2)</td>
<td><img src="image6.png" alt="Diagram" /></td>
<td><img src="image7.png" alt="Diagram" /></td>
<td><img src="image8.png" alt="Diagram" /></td>
<td><img src="image9.png" alt="Diagram" /></td>
<td><img src="image10.png" alt="Diagram" /></td>
</tr>
</tbody>
</table>

### TYPICAL STRESS-STRAIN CURVES

#### EXAMPLES

- **TYPE I: ELASTIC**
  - basalt
  - ![Diagram](image11.png)

- **TYPE III: PLASTIC ELASTIC**
  - sandstone
  - ![Diagram](image12.png)

- **TYPE V: PLASTIC ELASTIC**
  - siltstone
  - ![Diagram](image13.png)

- **TYPE II: ELASTIC PLASTIC**
  - rock-salt
  - ![Diagram](image14.png)

- **TYPE V: ELASTIC PLASTIC CREEP**
  - schist
  - ![Diagram](image15.png)

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**FIGURE 4.2-4: SPECTRUM OF ROCK BEHAVIOUR** (after Hendron)
stage\textsuperscript{2}\textsuperscript{2} instead of being used for the fragmentation of the rock mass. Failure of these types of rock, during blasting, is associated with the shearing action of the expanding gases pushing the burden and causing outward deformation. The result of a shear type failure is a coarse fragmentation with large blocks and slabs. This situation often prompts mine operators to declare that a given rock type within their pit is easy to drill but uneasy to blast. This may generally be explained when looking to the drilling practice in soft to medium formations, where the long projection inserts or teeth scrape and shave the rock rather than making it fail under concentrated stress fields like in hard rocks (see chapter 3). When dealing with those types of rock, the blasting engineer must consider that the rock mass will experiment little, if any, of the pre-conditioning effect of the strain wave on the burden. Because the fragmentation is therefore entirely done by the expanding gases, the optimization of gas retention by the use of an adequate length and type of stemming material\textsuperscript{2}\textsuperscript{3}, the careful planning of the initiation pattern so that any two freefaces are equidistant from the blasthole and by making sure that the rate of yeild at the point of minimum burden is not too great, should result in improved fragmentation in those "uneasy to break" rocks.\textsuperscript{1}

The attenuation of the strain wave energy, within the burden distance, largely reduces the explosive potential to create tensile stresses. The attenuation of the strain wave varies inversely with the distance to the power of 1.4 in strong
elastic rocks whereas values up to 2.5 are used in less brittle rocks\textsuperscript{24}. Attenuation of the strain wave also occurs in the jointed rock mass although the presence of groundwater will largely reduce this effect. Nevertheless, as the intensity of fracturing of the rock mass increases, the less brittle is the failure behaviour of the rock mass.

4.2.2 Design Parameters

This paragraph examines the importance of the charge distribution on the degree of fragmentation. The geometric relationships between the different design parameters have to be balanced in order to allocate the explosive energy, across the pattern, according to the work to be done. This is, after the structural properties of the rock mass, the second most important factor in determining fragmentation.

The design parameters that will be discussed in the following pages have been classified by the author in a system that emphasises the degree of control the blasting engineer experienced over these parameters.

CATEGORY 1: BLASTHOLE DIAMETER AND BENCH HEIGHT

The parameters of category one are the blasthole diameter and the bench height. Generally, the blasting engineer does not control these parameters. There have been fixed by the type of equipment bought to match mine design, production requirements,
regulations and/or dilution constraints. Design proposals that include a modification of the blasthole diameter or bench height would not be received with enthusiasm at upper management level. However, the blasting engineer must understand the influence of these parameters on the fragmentation process and on operating costs.

Drilling costs are reduced when the blasthole diameter is increased. However, there is an upper limit in the blasthole size. This limit is reached when the effective burden equals 40 times the blasthole diameter or equals the bench height. In such cases, the fragmentation is generally coarser because of the poor distribution of the explosive across the pattern and the large collar rock volume. Thus, at a given site, an increase in the energy factor must come with an increase in the blasthole diameter in order to maintain the degree of fragmentation. The use of large diameter blastholes also results in an uneconomical use of the blasthole volume. In addition, pit wall instability is more likely to occur when large punctual charges are used.

Nevertheless, the influence of the blasthole diameter on the degree of fragmentation is subordinated to the structural characteristics of the rock mass. Where pronounced joint planes divide the burden into large blocks, good fragmentation will be achieved only when each block is intercepted by a blasthole. In the intensely fractured rock mass, fragmentation is structurally controlled and the use of larger diameter blastholes causes
relatively small reduction in the degree of fragmentation.

The geometric relationships between the blasthole diameter, bench height and (effective) burden are well documented. Persson$^{26}$ and Bergmann$^{27}$ suggest a burden/blasthole diameter ratio of 30 to obtain good results within a cost effective design.

Although these may be slightly more expensive, medium size diameter blastholes give better breakage since the charge is brought up higher in the blasthole. Smith and Ash$^{28}$ have studied the geometric relationship between blasting parameters. They showed that the fragmentation was improved when bench height was increased from one to values over two times the (effective) burden, everything else being the same (Figure 4.2-5). This is explained by the bending conditions developed within the burden during the blasting process. They observed coarse fragmentation, backbreak and toe formation in situations where $H/Be$ was equal to one.

**CATEGORY 2: BURDEN, SPACING AND EXPLOSIVE PROPERTIES**

The parameters of category two are the spacing/burden ratio and the explosive properties. The blasting engineer has more control over these variables although economic constraints eventually narrow the range of alternatives. The burden and spacing are often considered as the principal blast design variables. There are important, indeed, but the initiation
FIGURE 4.2-5: TRENDS OF FRAGMENTATION INDEX, $F_c$, WITH L/B AND S/B RATIOS. (after Smith and Ash\textsuperscript{28})
sequence can totally alter the drilling pattern. Consequently, burden and spacing are referred to as effective burden (Be) and effective spacing (Se) when related to the blasting process. The effective burden is defined as the shortest distance between the blasthole and the effective free face at the instant the charge detonates\(^2\). There is an optimum effective burden for which the volume of well fragmented rock is maximum (Figure 4.2-6). When the effective burden is too large, the attenuation of the strain energy results in less developed radial cracks and uneven pre-conditioning of the rock mass. The expanding gases vent into a network of cracks inadequate for efficient fragmentation and generally escape by ejecting the stemming material. The subsequent holes will thus face an aggravated situation (chain reaction). Therefore, the first row burden should be carefully designed. When the effective burden is less than optimum, strain wave fracturing increases. This usually results in premature venting of the expanding gases through the burden and problems such as flyrocks and airblasts. However, mine operators are generally dealing with too large rather than too small effective burden.

The spacing dimension is a function of the burden and Bauer\(^3\) suggests Se/Be ratio between 2 and 5. This geometric relationship exercises a dominant control over radial cracking and the optimum is therefore depending upon the rock mass structure.

Finally, on the practical aspect of drilling and blasting,
Burden

a) Completely contained, only failure is pulverisation near the charge and radial tensile failure running out from it.

b) Start of surface failure burden not broken. Some doming of the surface.

c) Surface and subsurface failure almost meet. There will be a shelf of unbroken rock between the two. Doming or surface bulging.

d) Full crater, burden completely broken out. Surface and subsurface failures run through to the surface.

e) Full crater, lower volume than optimum fine fragmentation noise, flyrock, bowl shaped crater.

Figure 4.2-6: Schematic of the effect of decreasing the burden on similar charges fired in rock. (after Bauer)
mine operators should emphasize the following drilling practice:
in case of cave-in, the driller should attempt his new hole on
the spacing rather than the burden dimensions. This is not
always done and the result is that the adjacent holes become out
of balance during the detonation process.

The selection of the explosive is a function of the
relative cost alternatives, the explosive properties, the rock
mass properties and the expected results. Explosives are
generally bulk-delivered on the pattern to minimize
transportation and handling costs. In such a form, slurries are
about twice as expensive as ANFO. Utilization of pre-pack
slurries or custom products has to be kept to a minimum. They
are very expensive products. In addition, hole pumping and
lining procedures can permit the use of ANFO in wet holes and
reduce the total blasting cost. The reduction in strength of
ANFO with increasing water content is shown in Figure 4.2-7. In
the field, wet ANFO will be eventually indicated by orange-
yellow smoke during blasting. It is important to note that,
given the importance of blasting results on downstream
operations, the reduction of only 20% of the ANFO efficiency
makes slurries an economical alternative. The explosive
selection process also has to consider the rock mass properties.
Massive rock formation will require a maximum of strain energy
as provided by high density slurries whereas highly fissured
rock mass will need a maximum of bubble energy from the
expanding gases as provided by ANFO. Practically, mine


**Figure 4.2-7**: Effect of water content on the detonation velocity of AN/FO.

(after Leighton$^4$)
operators are using ANFO as much as possible until serious problems and/or detailed cost studies demonstrate the advantages of slurry explosives. Complete discussion of explosive selection is given in the literature. The blasting engineer should acknowledge that for each blasting situation, there is an optimum explosive.

**CATEGORY 3: SUBDRILLING AND COLLAR HEIGHT**

The parameters of category three are the subdrilling and collar height. These two parameters are usually defined as a function of the effective burden (Figure 4.2-8). Page suggests a subdrilling height ranging between 0.0 and 0.5 Be, Buchta suggests an interval from 0.1 to 0.4 Be whereas Hoek suggests 0.2 to 0.3 Be. This wide range of design criteria signifies to the blasting engineer the importance of performing his own tests in the different domains in the pit. The role of subdrilling is to break the rock at grade between the blastholes. Thus according to Hagan optimum subdrilling is function of the structural and density characteristics of the rock, the type of explosive (i.e., energy per foot of blasthole), the blasthole diameter, the effective burden, etc. Where the time interval between the drilling and the blasting of the blasthole is long, it is a good practice to drill a certain extra distance for sludge or drill cuttings which may accumulate in the bottom of the blasthole. However, over-drilling should
FIGURE 4.2-8: RATIO AT BENCHES (after Buchta36)
be minimized. It may cost up to $750,000/year in large operations. In such cases, blastholes must be backfilled to proper depth to avoid adverse effects like excessive ground vibrations and poor collar load. On the other hand, insufficient subdrilling causes a high pit floor and blasting in those conditions eventually results in more toes on each successive shot. Some operators increase the subdrilling on the front row of multi-row blast to improve displacement. This practice severely damages the next bench and careful design of the front row burden is a better alternative. Finally, in perimeter blasts or in highly fractured rock mass, the subdrilling can be reduced to nothing.

The collar height should be kept between 0.7 and 1.0 Be. The stemming length has to be long enough to resist the premature venting of the expanding gases and force them to push out the burden. Premature venting through the stemming column results in a waste of explosive energy and associated problems such as poor fragmentation, toes and absence of a free face for the subsequent blasthole, etc. On the other hand, a too long collar height may create blocky fragmentation in the top part of the bench. Even though drill cuttings are not the best stemming material, there are generally used because there are the cheapest material available.

CATEGORY 4: DELAY INTERVAL, INITIATION SEQUENCE AND DESIGN

POWDER FACTOR
The parameters of category four are the delay interval, the initiation sequence and the design powder factor. These are the parameters on which the blasting engineer exerts total control. Variations in the initiation sequence have no significant effect on the drilling and blasting costs, but, with its influence on the degree of fragmentation, can reduce the total production cost. The originality of the delay pattern design is only limited by the quality of the blasting crew.

The degree of fragmentation, in open pit bench blasting, is strongly dependent on the availability of effective free face and on the Se/Be ratio. Both are controlled by the delay interval and initiation sequence. The effective free face is created, in multi-row blasts, by allowing sufficient movement of the burden to take place prior to the initiation of the charges in the next row. If the delay interval is too short, the following row is overloaded by the previous burden. The results are poor fragmentation, tight muck pile, toe problems, backbreaks and flyrocks caused by cratering. The shot becomes choked. If the delay interval is too long, cut-offs are likely to occur, unless a long period down the hole initiation system is used. Thus, the best delay interval is the longest that can be used without cut-off problems. It is generally dependant upon the rock mass characteristics, highly fissured rock mass being more prone to line disruption. In experiments carried by Bergmann et al\textsuperscript{37}, different delay intervals between adjacent holes along one row were tested in regard to fragmentation.
optimization (Figure 4.2-9). It suggests that delay interval between 3 and 6 millisecond per meter (1 and 2 ms/f) of effective burden, this would be a good starting point when designing a row per row blast with no down the hole initiation system. The blasting engineer must consider that simultaneous initiation of holes (along one row) produces a more uniform stress distribution in the rock, resulting in a smaller number of well defined fractures and therefore poorer fragmentation. Optimum fragmentation depends on the full development of the crack network around each hole before the charge in the next hole is detonated. For a sequential blast, Andrews\textsuperscript{38} suggests delay ratio of 9 ms/m (3 ms/f) of Be between holes along the row, with an upper limit of 15 ms/m (5 ms/f) of Be in massive or infrequently jointed rock; and a delay interval between the rows equal to two or three times the delay ratio used between the holes in a row.

The effective spacing/effective burden ratio also influences fragmentation, as stated previously. Many mines use a square drilling pattern because it is easy to lay-out and simple to drill. Therefore, V1 initiation pattern should be considered (Figure 4.2-10). However, it is often more simple to tie-up the square pattern in V rather than in V1. In all cases, the V and V1 patterns give better fragmentation than in-line patterns because of the modification of the bench height/effective burden ratio. These two patterns also cause additional breakage with in-flight collision\textsuperscript{39}. 
FIGURE 4.2-9: EFFECT OF DELAY TIME BETWEEN SHOTHOLES ON AVERAGE FRAGMENT SIZE. THE SAME POWDER FACTOR WAS USED FOR ALL SHOTS. (after Bergman & all\textsuperscript{37})
FIGURE 4.2-10: BLASTHOLE/INITIATION PATTERNS WITH B=S FOR SHOTS FIRE TO AN OPEN FACE. (after Hagan39)
Delay interval and initiation sequence also influence the pit slope stability by reducing the amount of explosive fired simultaneously and by directing the blast toward a free face. Figure 4.2-11 shows the important relationship between the instantaneous charge initiation and damages at distance from the blast. On this figure, values of $K = 200$ and $B = -1.5$ have been used to solve the following equation:

$$V = K \left(\frac{R}{W^{**-2}}\right)^{**B} \text{ (USBM)}$$

With $V = \text{particle velocity}$

$W = \text{weight of explosive detonated per delay}$

$R = \text{radial distance from the point of detonation}$

$K, B = \text{constants (site specific)}$

This figure should be used as a general tool to determine the maximum charge per delay as the blast pattern moves toward the pit limits. The blasting engineer must remember that the values of $K$ and $B$ are site specific and that only average values are given here. Those constants are easily determined in the field by using seismograph.

Because of its intrinsic importance in the blasting process and in the field of this investigation, the design powder factor is covered in a particular paragraph.
FIGURE 4.2-11 PLOT OF PARTICLE VELOCITIES INDUCED AT GIVEN DISTANCES BY PARTICULAR CHARGES (after Hoek and Bray)
4.2.3 The Design Powder Factor

The powder factor is the most widespread and yet most misunderstood statistic in the field of blasting. Although many operators consider the powder factor as an accounting tool, numbers of them are now using it as an efficient design quantity. It permits a good understanding of the blasting process, consistent results and controlled product. The fact that different explosives possess different energy content leads to the normalization of the powder factor. This is done by multiplying the weight of explosive used by its relative weight strength (energy) factor. The weight strength factor of straight ANFO is arbitrarily fixed as 100.

The blasting engineer therefore rationalizes the loading procedure by determining the quantity and type of explosive to be poured in each hole in order to keep the energy distribution as uniform as possible. The design powder factor does take into account only the burden volume of the blasthole and thus, is generally higher than the accounting powder factor that include free digging and overbreak. However, the blasting engineer is still facing the problem of determining the optimum powder factor for each different blasting domain in the pit. The scope of this research project is to simplify the trial and error procedure by defining a reliable relationship between the powder factor and the Rock Quality Index.

There is an optimum powder factor for each rock mass and
blasting conditions. With everything kept constant, fragmentation improves as the powder factor is increased until a maximum is reached (Figure 4.2-12). Increasing the powder factor further then eventually causes the fragmentation to deteriorate and is a waste of explosive energy. However, it has been shown that if the powder factor is reduced by about 20% from the optimum value, in a given open pit, the ground vibration level can be increased by a factor of 2 or $3^{3/8}$ (Figure 4.2-13). This reduction in the powder factor can be obtained without intention when ANFO is side-initiated by a primacord downline. In this case the energy output is reduced. However, in perimeter blasting, the general rule is that the powder factor is reduced in order to maintain the integrity of the final wall. Therefore the blasting engineer shall design the shot by using the maximum advantage of down-the-hole delays systems in order to create effective freefaces and counter balance this increase in the ground vibration level. The reduction, in the powder factor, on a trim perimeter shot is in the range of 35 to 50 percent.

A wide practice in open pit mining is the choked blasting situation. The shot is said to be choked when it deals with the firing of rock into previously-shot muck which lies directly against the vertical "freeface". There is a number of advantages and disadvantages related to this practice. Some operators are concerned with the cost of moving equipment in and out because of flyrock hazards. Others want to minimize the
FIGURE 4.2-12: SHOVEL PRODUCTION ON A YEARLY BASIS AT ONE OPERATION VERSUS EXPLOSIVE CONSUMPTION IN ROCK OF 20,000 PSI COMPRESSIVE STRENGTH. (after Bauer$^3$)
FIGURE 4.2-13: A PLOT OF GROUND VIBRATION VS POWDER FACTOR MEASURED FROM A SERIES OF PRODUCTION BLASTS AT A LARGE OPEN PIT MINE SHOWING THE ABRUPT INCREASE OF THE LEVEL OF GROUND VIBRATION AS THE POWDER FACTOR IS DECREASED. (after Andrews\textsuperscript{38})
movement of drilling equipment or to accumulate blasted reserves in the pit. Further advantage is the increase in safety when drilling close to the crest of the bench. Mine operators may also practice choked blasting because of the width of the orebody in relation to the production requirements. It can also be the result of blending requirements or simply inadequate mine planning. In any case, operators will face the following disadvantages. First, if there is no natural parting at the pit floor to assist rock displacement at the toe region, it may result in an uneven pit floor. The second disadvantages is the higher ground vibration level caused by choked blasts and its effect on slope stability. Finally, chocked shots necessitate an increased powder factor in order to keep the degree of fragmentation constant. Bauer suggests a factor of 1.1 while Hagan acknowledges an overall fragmentation reduction and the need for an increase in energy requirement but he does not propose any numerical value. In his opinion, the fracturing mechanisms which rely on the strain wave are not influenced by the buffer rocks. Only the breakage mechanisms that require displacement are altered. Lang proposes longer delay intervals in order to maintain the fragmentation and obtain satisfying results.
4.3 ROCK MASS BLASTABILITY

The characterization of the rock mass in regard with blasting and the determination of the optimum powder factor under specific blasting conditions are the main subjects of this project. The evaluation of the blastability of the rock mass has been attempted by many scientists during the last 20 years. This section will review and critique typical relationships developed recently.

The determination of rock mass blastability can be obtained by the following methods:

1) visual determination of the blastability.
2) geophysics, by using special boreholes or drilled production blastholes.
3) correlation with one or more rock properties, measured in laboratory, in-situ or both.
4) blastability index derived from small scale crater tests.
5) rock mass characterization with production rotary drills.

4.3.1 Visual Determination of the Rock Mass Blastability

This method is far from optimum and requires a certain amount of site specific experience or a great deal of experience in a wide range of rock mass. This is the method generally used in Western Canada. By trial and error, the blasting engineer hopes that he will eventually define the optimum powder factor
in each and every domains on the property. It is a slow process. Trial shots have to be carefully monitored and evaluated. Several shots of every tested design must be performed before alteration of any parameters. This method asks for a detailed filing system and the use of photographs are strongly recommended. However, when the variability in the rock mass properties is high, the operation is continually performing test blasts. The cost of such procedure on the downstream unit operations are very high. Moreover, the damage caused to the final wall and/or the design of a conservative slope angle are even more costly. This is the situation we want to optimize.

4.3.2 Characterization of the Rock Mass By Geophysics Methods

These methods are well known of exploration geologists and geophysists. Using special equipment and skill technicians, it is possible to obtain data on the rock mass to be blasted. Those data are presented in the form of logs that show variations in rock mass properties such as sonic velocity, density, resistivity, etc. This type of information can be useful, in identifying softer or harder layers of rocks along the blasthole length. Based on this information, the blasting engineer can determine the location of the charge, booster and primer and optimize fragmentation or reduce wastage of explosive energy. These data can hardly be used as blastability index because of their relationship with only one, rarely a few, rock
mass properties, unless several different logs are used. The principal consideration in these methods is cost. They are expensive. In addition, as the variability of the material increases, in the strike and/or dip direction, the amount of data needed for each daily shot will hardly be balanced by the cost efficient blast design that would result. The use of these methods can not be justified in open pit production blasting.

4.3.3 Correlation of the Blastability with One or More Rock Mass Properties, Measured In Laboratory or In-Situ

This is the field where most of the work has been done. Various attempts have been made to relate the blastability of the rock mass with properties such as Young's Modulus, uniaxial compressive strength, tensile strength, etc. One of the first blastability index was the uniaxial compressive strength/tensile strength ratio. A low value indicates a plastic behaviour whereas a high value indicates a brittle behaviour. Munoz-Casayus has published a relationship between the rock uniaxial compressive strength and the powder factor (Figure 4.3-1). Those types of relationship are generally poor due to the fact that rock properties were measured in static laboratory tests and, in addition do not consider the structural properties of the rock mass.

Other experiments were carried on the site. Rock mass properties such as sonic velocity were related with the powder
FIGURE 4.3-1: UNIAXIAL COMPRESSIVE STRENGTH VS POWDER FACTOR. (after Munoz-Casayus)
factor. Velocities tend to decrease as the number of fractures in the rock mass increase. The velocity is also reduced in rock mass showing a plastic behaviour, accordingly with a reduction in the modulus of elasticity. Figure 4.3-2 shows the relationships developed by Heinen and Dimock\textsuperscript{3} of Kennecott Copper Corporation and by Broadbent\textsuperscript{4}. Because both velocity and blastability are functions of the same rock mass characteristics, there are directly related. This method has few disadvantages:

1. Soft material shows velocities 600 m/s (2000 fps) higher than normal when frozen.

2. Shooting seismic lines must be done when the equipment in the area is not operating due to background noise.

3. The presence of hard anomalies below the mining level surveyed could result in velocities higher than normal.

Heinen and Dimock claimed that the cost of the seismograph and the seismic surveys performed are insignificant compared to the cost savings to the operation. However, if only one seismic line survey per 120 meters (400 ft) is sufficient in areas of uniform structure, 4 times more are needed in areas of high variability.

When the fracture process is believed to be structurally controlled, relationships between the intensity of fracturation and powder factor (Figure 4.3-3) or blastability index
**FIGURE 4.3-2: ACOUSTIC VELOCITY VS POWDER FACTOR.**

(after Heinen and Dimock^{43})
FIGURE 4.3-3: FRACTURE FREQUENCY VS POWDER FACTOR. (after Ashby$^{30}$)
(Figure 4.3-4) have been developed.

Finally, an original approach has been taken by Christensen and Olsen\textsuperscript{46} to evaluate the resistance to blasting in tunnelling. In the field they evaluate the blasting results by measuring the advance in percentage of drilled depth as a function of a given normalized charge. Then, using regression analysis, they related the rock's sonic velocity, tensile strength and density, measured in the laboratory and the jointing intensity to the field blastability index. They claimed a correlation coefficient of 0.9367. This example was reviewed to point out how a rock mass classification system could be developed to evaluate the rock mass blastability.

4.3.4 Determination of the Blastability by Small Scale Crater Tests

The most logical method of determining the blastability of the rock mass is certainly by blasting it. Experiments were carried out by Bauer et al\textsuperscript{47} at I.O.C. They developed a rock mass blastability index based on the optimum depth ratio and the strain energy factor computed from crater tests.

In crater tests, a spherical charge, approximated by a length to diameter ratio less than 6, is detonated at different depths and the scaled crater volume is plotted as a function of the scaled depth of embedment (Figure 4.3-5). The maximum volume occurs at the optimum depth of embedment \( z_0 \). The
1.6
1.5
1.4
1.3
1.2
1.1
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0
10
20
30
40
50
60
70
80
90
100

CORRECTION FACTORS FOR ESTIMATING JOINT STRENGTH

ESTIMATION OF QUALITY ALTERATION FACTOR

STRONG
MEDIUM
WEAK
VERY WEAK

1.00
0.90
0.80
0.70

ERQD = RQD × ALTERATION FACTOR

FIGURE 4.3-4: BLASTABILITY FACTOR VS EQUIVALENT RQD.

(after Borquez 45)
FIGURE 4.3-5: VARIATION OF BROKEN ROCK VOLUME WITH DEPTH OF EMBEDMENT FOR A CONCENTRATED CHARGE.

(after Coates and Gyenge)
explosive is completely choked and does not produce any crater at the critical depth, Zc. These relations are expressed by the following equations:

\[ Zc = E(W)^{1/3} \]
\[ Zo = \Delta o E(W)^{1/3} \]

with \( Zc \) = critical depth
\( Zo \) = optimum depth
\( E \) = strain energy factor (rock mass constant)
\( W \) = charge weight
\( \Delta o \) = optimum depth ratio = \( Zo/Zc \)

On this basis, they defined the blastability as \( \Delta o E \).

During this research, they also confirmed the existence of two types of failure during blasting: brittle and shear (plastic). Figure 4.3-6a shows how the strain energy factor (E) can be used to predict the type of failure. Moving to the left, rocks get softer (plastic) and conversely going to the right they get harder (brittle). The most interesting result, however, is the relationship between the blastability index and the strain energy factor (Figure 4.3-6b). This figure shows that at both extreme values of E, the material shows very low blastability and thus requires more energy per ton of rock. Finally, they developed a relationship between the powder factor and the strain energy factor (Figure 4.3-6c). That relationship is taking into account the two major characteristics that influence the blasting results, the structural properties and the failure
FIGURE 4.3-6a: OPTIMUM DEPTH RATIO, $\Delta_0$ VS STRAIN ENERGY FACTOR, $E$ (after Bauer et al\textsuperscript{47})

FIGURE 4.3-6b: BLASTABILITY FACTOR, $\phi E$, VS STRAIN ENERGY FACTOR, $E$ (after Bauer et al\textsuperscript{47})

FIGURE 4.3-6c: POWDER FACTOR VS STRAIN ENERGY FACTOR, $E$ (after Bauer et al\textsuperscript{47})
behaviour. However, the determination of the strain energy factor is a long and costly process.

Although the rock characterization methods, proposed up to now in this paragraph, numerically evaluate the blastability of the rock mass, their accuracy is as variable as the degree of homogeneity of the rock. In addition, all require additional operations such as rock testing, seismic velocity measurement, structural mapping or crater tests.

4.3.5 Characterization of the Rock Mass from the Performance of Production Rotary Drills

Since every hole has to be drilled before being blasted, the blasting engineer should take advantage of all the information available at this stage to optimize his design. The costs are minimum and the data are numerous, so there is no drawback in rock mass that present high degree of inhomogeneity.

In hard brittle rock, there is a similitude between the failure in drilling (roller-bit, percussive) and in blasting. In both cases, the rock fails under concentrated, non-uniform stresses. Close to the vicinity of the applied force, the rock compressive strength is overcome and the rock is finely crushed. Away from that zone, the rock fails in a brittle manner where the tensile stress reaches the tensile strength. The influence of the structural properties on drilling and blasting is more complex. Micro-scale discontinuities influence drilling
performance by acting as weakness planes such as macro-scale discontinuities influence blasting. In both cases, the failure mechanism is improved. However, the influence of macro-discontinuities on drilling performance and the overall effect of micro-fissures on blastability need a more detailed investigation. It is believed that, from a certain level of intensity, macro-scale discontinuities improve drilling performance. Micro-flaws are said to improve fragmentation.

The first attempt to correlate the Rock Quality Index with the rock mass behavior was performed by T.E. Little in 1975. On this research project, the characterization of the rock mass using blasthole drills performance was directed toward the design of stable pit slope. The idea was that the RQI would provide useful data on the future behaviour of the rock slope, when being correlated with lithology, structural geology and rock strength. In such case, the geotechnical engineering department, by monitoring the drill performance, would obtain a quantitative measure of the rock quality, determine critical areas and implement remedial measures. The study was performed in three open pit mines in British Columbia.

However, the report concluded that the Rock Quality Index was not reliable enough for predicting the slope behaviour. First, the accuracy of the data input was defined as not sufficient enough to permit valuable interpretation of the rock mass behaviour. Little also found that the RQI was primarily related to the rock strength but not to actual structure.
Higher values of RQI were encountered in hard competent rocks while lower values were correlated with fault zones. Unfortunately, hard dyke rocks, at one site, showed a high RQI even though the fracturing was intense. The report stated that there was no direct relationship between RQI values and geology. Finally, different RQI values were obtained in adjacent blastholes when drilled with different drill models equipped with different bit sizes, although, at one site, different diameter holes yielded the same RQI when drilled by the same drill model. Moreover, it was found that steel tooth bits produce higher RQI than tungsten carbide insert bits, as predicted by the theory. A detailed report of the conclusions of this research project are given in Appendix IV.

Little suggested that, because only major trends are detected by the RQI, its application to slope design was limited. The use of a drill performance recorder would produce better quality input data, but at higher cost and increased complexity. He concluded that basic geotechnical data gathering methods therefore remain the best alternative. However, Little believed that the Rock Quality Index would be a useful tool in predicting energy requirements in drilling, blasting and grinding.

No other research was done on the Rock Quality Index until Leighton revived this concept in the current research project. Based on Little's conclusions, three major modifications were derived:
1) Improve quality and accuracy of drill performance records.

2) Rather than using the RQI to determine domain boundaries, it was simply used to classify the rock quality within the domain boundaries already established by conventional methods.

3) Keep domain areas large enough to minimize effect of bit wear, shift changes, driller bias, etc.

Leighton then developed a relationship between the Rock Quality Index and the controlled powder factor at Afton Mine (Figure 4.3-7). His study showed that the performance of the blasthole drills reflects the rock mass properties and structural geology; that, when carefully monitored, the hydraulic down pressure and penetration rate would produce an index of the rock mass condition and that the average RQI value within each domain was representative of the distribution.

The very encouraging results have prompted the development of a continuity of the research project. The report of the present investigation is produced in the following chapters.
FIGURE 4.3-7: PROPOSED CORRELATION BETWEEN ROCK QUALITY INDEX AND POWDER FACTOR AT AFTON MINE. (after Leighton 4)
4.4 SUMMARY

The author would like to emphasize the fact that the blasting procedure directly influences three other aspects of any open pit mining operation: the slope stability, the productivity and the equipment maintenance. The direct blasting cost is very small compared with the actual total operation costs. Therefore, a slight increase in the blasting cost can be justified by overall savings that will result.

The optimum energy factor to be used in a given domain is a function of a wide range of parameters. This is the reason why the design powder factor approach is successful. Mine operators always appreciate standard practices but few of them apply it to blasting. The design powder factor approach results in uniform explosive distribution within the pattern and/or the domains when the blast pattern extends over the domain boundaries. It is also important to consider the weight strength of the explosive utilized and express the powder factor consistently on a relative energy basis in the industry (i.e., relative to ANFO, weight strength = 100).

Good records of blast procedures are very important. They permit one to review and appreciate the design modifications. The use of a polaroid camera to obtain a set of black and white photographs of the blast results can be justified. The blasting file also contains:

1) - plan view of blast pattern (bench plan)
- type of explosives
- explosive load and design powder factor
  -- front row
  -- production rows
  -- buffer row(s)
- net powder factor (total blast volume including free digging)
- delay pattern and initiation sequence
- drilling records
- choked or free faced

2) - location or domain
- short description of the geological features
  -- rock type, fracture intensity, alteration
  -- majors discontinuities in relation with the direction of blasting
  -- groundwater (number of wet holes, pumped holes, slurry holes, etc.)

3) - blast evaluation form and photographs

<table>
<thead>
<tr>
<th></th>
<th>Excessive</th>
<th>Good</th>
<th>Fair</th>
<th>Poor</th>
<th>Nil</th>
</tr>
</thead>
<tbody>
<tr>
<td>Throw</td>
<td>----</td>
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<td>Lift(heave)</td>
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<tr>
<td>Fragmentation</td>
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<td></td>
<td>Present</td>
<td>Occasional</td>
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<td>Cratering</td>
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<td>Toe(s)</td>
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<td>Oversizes</td>
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<td>Shotguns</td>
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<tr>
<td>Smoke</td>
<td></td>
<td>Color</td>
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</tr>
<tr>
<td>Misfire Number</td>
<td></td>
<td>Explanation</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Very Good | Fair | Tough

4) - Pit wall conditions 1 to 5 (see Table 3)

5) - Remarks

Since good blasting results seems to vary from one site to the next, from one evaluator to the other, rating has to be defined relative to a certain standard. Figure 4.4-1 shows the features of a satisfactory production blast. The following criteria must be met:
TABLE 3

LEVELS OF BLASTING DAMAGES

COMMONLY OBSERVED ON PIT

WALLS (after Ashby30)

<table>
<thead>
<tr>
<th>Observed Conditions of the Wall</th>
<th>Dip Angle</th>
<th>Digging Condition at Face (Electric Shovel)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Arbitrary Damage Level</strong></td>
<td><strong>Joints &amp; Blocks</strong></td>
<td><strong>Appearance and Condition of Face</strong></td>
</tr>
<tr>
<td><strong>1 Slight</strong></td>
<td>Joints closed, infilling still welded.</td>
<td>&gt;75° circular sections of wall control holes seen.</td>
</tr>
<tr>
<td><strong>2 Moderate</strong></td>
<td>Weak joint infilling is broken, occasional blocks and joints slightly displaced.</td>
<td>&gt;65° Face is smooth, some hole sections seen.</td>
</tr>
<tr>
<td><strong>3 Heavy</strong></td>
<td>Some joints dislocated and displaced.</td>
<td>&gt;65° Minor spalls from face.</td>
</tr>
<tr>
<td><strong>4 Severe</strong></td>
<td>Face shattered, joints dislocated. Some blocks</td>
<td>&gt;55° Face irregular, some spalls, some backbreak cracks.</td>
</tr>
<tr>
<td><strong>5 Extreme</strong></td>
<td>Blocks dislocated and disoriented. Blast-induced fines or crushing observed.</td>
<td>55°&gt;37° Face highly irregular, heavy spalling from face.</td>
</tr>
</tbody>
</table>
FIGURE 4.4.1: FEATURES OF A SATISFACTORY BLAST. (after Hoek and Bray<sup>2</sup>)
1) Uniform, moderate movement of the toe burden without flyrock rubble.
2) A slight rise along the muck pile crest.
3) No surface cratering or flat areas.
4) A slight drop along the last row of holes (buffer holes in controlled blast)
5) No broken ground beyond the final digline.
6) No digging problems.
7) Uniform fragmentation.
8) A clean wall with minimum ravelling potential.

Appendix V lists a simple blast modification procedure.
4.5 REFERENCES

1. MERCER, J.K.; HAGAN, T.N.; Progress Towards Optimum Blasting - a Key to Increased Productivity and Profitability, 11th Commonwealth Mining and Metallurgical Congress, Hong Kong, 1978


3. BAUER, A.; Drilling - Blasting Course Notes, Queen's University, Kingston, unpublished


8. STARFIELD, A.M.; Strain Wave Theory in Rock Blasting, Proceedings of the 8th Symposium on Rock Mechanics, C. Fairhurst, Editor, University of Minnesota, AIME, 1967

9. WINZER, S.R.; RITTER, A.P.; The Role of Stress Waves and
Discontinuities in Rock Fragmentation: a Study of Fragmentation in Large Limestone Blocks, 21st U.S. Symposium on Rock Mechanics, University of Missouri-Rolla, 1980


15. BELLAND, J.M.; Structure As A Control in Rock Fragmentation,
16. COHEN, C.J.; Bench Blasting Related to Joint Design,
   World Mining, April 1980

17. PAGE, C.H.; Blasting and Comminution, Unpublished report,
    Steffen, Robertson and Kirsten

18. COATES, D.F.; GYENGE, M.; Incremental Design in Rock
    Mechanics, Mine Branch Monograph 880 (1973),
    Dept. Energy, Mines and Resources, Ottawa, Canada

19. CALDER, P.N.; Pit Slope Manual Chapter 7 - Perimeter
    Blasting, CANMET, report 77-14, pp. 82, May 1971

20. HENDRON, A.J. Jr.; Rock Mechanics in Engineering Practice,
    Stagg & Zienkiewicz Editors, John Wiley and Sons,
    1968

21. LANG, L.C.; New Permafrost Blasting Methods Developed at
    Asbestos Hill, Canadian Mining Journal, March 1976

22. PEARSE, G.E.; Rock Blasting: Some Aspect of the Theory
    and Practice, Mine and Quarry Engineering, January
    1955

23. KONYA, C.J.; DAVIS, J.; The Effect of Stemming Consist on
    Retention in Blastholes, Proceeding of the 4th
    conference on blasting and explosive techniques,
    Society of Explosives Engineers, Louisiana, 1978

24. COATES, D.F.; Rock Mechanics Principles, Mine Branch
    Monograph 874 (revised 1970), Dept. of Energy, Mines
    and Resources, Ottawa, Canada

25. Le Manuel des Explosifs, E.I. Dupont de Nemours and Co
26. PERSSON, P.A.; Bench Drilling in Open Pit Mines, World Mining, August 1975

27. Seminar on Applied Explosive Technology (Dupont), Presented to Lornex Mining Corp. June 1983


29. HAGAN, T.N.; The Effects of Blast Geometry and Initiation Sequence on Blasting Results, Chapter 6 of the Australian Mineral Foundation Workshop Course Manual, Drilling and Blasting Technology, Adelaide, 1977

30. ASHBY, J.P.; Production Blasting and Development of Open Pit Slopes, Proceedings, 3rd International Conference on Stability in Surface Mining, Brawner Editor, AIME, 1982

31. DICK, R.A.; Factors in Selecting and Applying Commercial Explosives and Blasting Agents, USBM 1968 (inf. circ. 8405)

32. MANON, J.J.; How to Select an Explosive or Blasting Agent for a Specific Job, Engineering and Mining Journal, May, 1977

33. THORNBY, G.M.; FUNK, A.G.; Aluminized Blasting Agents, Proceedings of the 7th conference on blasting and
explosive techniques, Society of Explosives Engineers, Phoenix, 1981


35. DRURY, F.C.; WESTMAAS, D.J.; *Considerations Affecting the Selection and Use of Modern Chemical Explosive*, Proceedings of the 4th conference on blasting and explosive techniques, Society of Explosives Engineers, Louisiana, 1978

36. BUCHTA, L.; *Open Pit Blasting - How to Improve Results*, World Mining, June 1982


40. BAUER, A.; *Personal Communication*, 1983
41. LANG, L.C.; Buffer Blasting Techniques in Open Pit Mines, Second Open-pit Operators Conference and Third Annual Meeting of CIM District Six, October 1978

42. MUNOZ-CASAYUS, J.M.; Approche Technique de la Determination des Parametres de Sautage dans les Mines a Ciel Ouvert, 3eme Session d'Etude sur les Techniques de Sautage, Laval University, 1980

43. HEINEN, R.H.; DIMOCK, R.R.; The Use of Seismic Measurements to Determine the Blastability of Rock, Proceedings of the 2nd conference on explosive and blasting technique, Society of Explosives Engineers, Louisville, 1976

44. BROADBENT, C.; Predictable Blasting with In-Situ Seismic Surveys, Mining Engineering, SME, April 1974


CHAPTER 5
5.0 FIELD RESEARCH PROJECT

During the summer of 1983, the correlation between the Rock Quality Index and the powder factor was tested in three mine sites of British Columbia (Figure 5.0-1). This chapter, which reports the findings of the field work, is divided into six majors sections. The first section reviews the investigation performed with the drill performance recorder on improved data quality. The next three sections present the results of the research project at the three mine sites. The last two cover the discussion, inferences and further research.

At the sites, the Rock Quality Index was compiled for every blasthole, within the blasting domains, from the bit performance records (driller logs) and/or from the recorded charts. According to the work of Leighton¹, the mean or average value was taken as representative of the RQI distribution of the domains. Efforts were made to keep the blasting domains large enough, although this was not always possible.

The author did not restrict the determination of the rock mass blastability only to perimeter blasting. According to Ashby², controlled blasting starts with production blasting. As a matter of fact, many operators consider that they are doing wall control blasting when they shoot a four row pattern, along the perimeter, with down-the-hole delays. In the majority of the cases, the damage to the final wall were already created by the last production blasts in the area, including the ones on
FIGURE 5.0-1: LOCATIONS OF THE MINES WHERE THE RQI STUDIES WERE CARRIED OUT.
the previous benches. Consequently, it was felt that the RQI approach should also be applied to production blasting. Moreover, the extent of the correlation was tested over a wide range of rock mass with various combinations of drill model/bit size. Blast performances were rated as underloaded, optimum and overloaded.
5.1 DRILL PERFORMANCE RECORDERS IN THE MINING INDUSTRY

Drill performance recorders are being used in the mining industry for many years and are highly reliable. The installation of a recorder to monitor blasthole drill performance is a cost-saving investment in mining operations. Introduced as a drilling tool, there are generally well accepted by the personnel. A drill performance recorder provides the mine operator with a permanent record of the drilling procedure and thus permits comprehensive drilling optimization programs. The principal drilling parameters that can be monitored are the bit depth, pulldown thrust or weight, rotary speed, developed torque and penetration rate. In addition, other parameters such as the vertical acceleration of the drill pipe can also be monitored. The principal advantages of drill performance recorders are:

1) to assist the driller to control the operation
2) to be used as an instructional tool for the training of new operators
3) to permit assessment of the drilling crews
4) to provide early warning of mechanical failure
5) to avoid over-drilling
6) to characterize the rock mass being drill for geological logging, blast optimization or grinding rate evaluation purposes

The first four advantages are well covered in the
literature\(^3\) and will not be reviewed here. The fifth point is easy to understand. In major open pit operations, the cost of over-drilling is in excess of the cost of installation of a drill performance recorder in each drill rig. The application of a drill performance recorder to the drilling characterization of the rock mass is more recent.

Hagan\(^6\) suggested the use of drill performance recorders in surface coal mining where the overburden strata consists of adjacent beds of highly variable strength. He proposed the monitoring of the penetration rate and/or the torque along the blasthole depth to optimize the blast engineering. Although he acknowledged the fact that variations in the penetration rate reflect the relative ease with which the various blocks can be fragmented by blasting and the optimum distribution of explosives and stemming materials within the blasthole, he considered that a comparison of penetration rates will not usually reflect the relative ease of blasting rocks which exhibit equal rock substance strength but dissimilar mean spacings between discontinuities (Figure 5.1-1)

Other methods are also available to characterize the rock mass. The Societe Nationale des Petroles d'Aquitaine (France) has developed a dynamic theory based on the creation, by the tricone bit, of longitudinal vibrations and the transmission of these vibrations through the drill stem\(^5\). By analysing the vibrations at the top of the drill pipe, it is possible to obtain data on the rock being drilled. The information is
FIGURE 5.1-1: EFFECT OF MEAN DISCONTINUITY SPACING ON
A) EASE OF BLASTING
B) PENETRATION RATE
(after Hagan$^4$)
related to the hardness characteristics of the rock. This logging method was able to pick-up thin, hard streaks that are obscured in the averaging process involved in the sonic log. The application of this technique to the blasthole industry to permit the prediction of explosive consumption has not yet been demonstrated. However, the instrumentation is available in North America and ready to test.

In this research project, although the characterization of the rock mass is based on parameters that are normally collected by the mine management, it was decided to investigate the effect of the accuracy of the data input on the RQI. Leighton has suggested that the quality of the data input would be improved when using a drill performance recorder.

At two of the three mining sites, it was possible to attach a "Geolograph" pen recorder model MOR III to one of the Bucyrus Erie drills. The instrument recorded, on a chart, the variations of the applied weight on the bit and the bit depth as a function of time (Figure 5.1-2). The obtained data were used to compute the Rock Quality Index; then termed RQI-RECORDER and to compare it with the RQI-LOGS computed from the driller logs.

Many interesting observations are derived from this investigation.

A. From a practical point of view, the interpretation of the charts is lengthy. The drilling time is generally well recorded and simple to compile. The applied weight, principal variable in the drilling process, produces a
FIGURE 5.1-2: RECORDED CHART (Lornex)
graph that is less easy to interpret. Estimation of an average value introduces a bias. The determination of the drilled depth, from the charts, is very difficult and consequently, it was preferred to use the depth recorded by the drillers on the bit performance logs. When the drilling procedure is not continuous within a given blasthole, the graph produced is highly irregular. Moreover, if the driller moved the drill pipe without bringing the drill performance recorder to a stand-by position, the drilled depth was recorded in the upward direction too. Caving situations also produced a wide variation in the applied weight. In these conditions, the combined effects made the data useless.

B. Technically, the drill performance recorder is a very useful piece of equipment on a drill rig. Many drillers have appreciated the presence of the depth meter and the availability of a graphical representation of their drilling practice. Concerning the Rock Quality Index, comparison of the compiled data shows that the RQI-LOGS are slightly higher than the RQI-RECORDER. RQI values compiled from the drill performance logs overestimate the toughness of the rock. This is due to the fact that most of the drillers recorded the larger down pressure applied while, in reality, the average down pressure over a given blasthole is generally lower. In addition, the drilling time is rounded to the nearest five minutes by the
majority of the drillers. The relationship between RQI-LOGS and RQI-RECORDER is not a simple one. Figure 5.1-3 is a plot of the difference between RQI-LOGS and RQI-RECORDER as a function of RQI-LOGS. It seems that the average difference increase with the value of RQI-LOGS up to a peak and then seems to be reduced slightly. The relationship is better represented on the histogram of figure 5.1-4. It suggests that the overestimation, or bias, from the driller logs is a function of the rock quality itself. However, analysis of the results from Equity Silver mine shows that the ratio RQI-LOGS/RQI-RECORDER is constant from a domain to the other, although the four domains considered were allocated RQI values ranging from 36 to 44. Drill performance recorders accurately monitor the drilling procedure. However, from the author's experience, the use of a drill performance recorder in relation with the gathering of valuable data for the determination of the Rock Quality Index should be simplified to a point where the manual compilation of the data becomes superfluous. In addition, the system should combine an automatic control device that will bring the recorder on stand-by when the drill pipe is moved up. For example, a computerized monitoring system that integrates RQI from instantaneous penetration rate and weight data collected at regular time intervals during the drilling process (i.e., Δt=15, 30, 60 sec). The data could be stored
Figure 5.1-3: Computer plot of the difference between RQI-LOGS and RQI-REC as a function of RQI-LOGS.
FIGURE 5.1-4: HISTOGRAM OF THE AVERAGE DIFFERENCE BETWEEN RQI-LOG AND RQI-REC AS A FUNCTION OF RQI-LOG.
on diskettes or tapes in a format that is compatible with an office micro-computer.
5.2 **EQUITY SILVER MINE**

5.2.1 **Summary of the Geology**

The Equity Silver Mine is located 35Km (22 miles) southeast of the town of Houston, B.C., and approximately 580 air Km (360 miles) north-northeast of Vancouver. The elevation of the mine site is approximately 1300 m (4265 ft). The deposit is divided into two zones, the Main zone and the Southern Tail zone.

Mining of the Southern Tail orebody commenced in April 1980 with a millfeed rate of 5500 tonne/day (6060 S.ton/day). It was scheduled to be mined out by the end of 1983.

The geology has been described by Cyr et al. The deposit occurs in an inlier of sedimentary, pyroclastic and volcanic rocks believed to be of cretaceous age. Strata strike 010 degrees and dip 45 degrees west. The sedimentary rocks are intruded by a quartz monzonite stock on the west side and a gabbro-monzonite complex on the east side. Copper-silver-antimony-gold mineralization forms an elongate west-dipping tabular zone between the stocks. Post mineralization andesitic and quartz latitic dykes crosscut the cretaceous rocks and the gabbro-monzonite complex. Figure 5.2-1 shows the regional geology of the area as described above. The pyroclastic unit overlies the classic unit and has a maximum thickness of 975 m (3200 ft). The rocks consist of intercalated subaerial tuffs, breccias and reworked pyroclastic debris all showing wide grain
1. JURASSIC - LOWER CRETACEOUS

North

Skeena Arch

Bowser Basin

Topley Intrusions

Deposition of Marine Clastics (Clastic Division)

Zone of Crustal Weakness

North

South

Nechako Arch

Nechako Trough

Deposition of "Goosly Sequence"

Volcanism

Deposition of Sedimentary-Volcanic

Volcanic Flow

Pyroclastics

Clastic

2. UPPER CRETACEOUS

North

Nechako Trough

South

Volcanism

Deposition of "Goosly Sequence"

Sedimentary-Volcanic

Pyroclastics

Clastic

3. TERTIARY

West

Folding and Intrusion

Quartz Monzonite Stock

Surface

Goosly Sequence

Gabbro-Monzonite Complex

Ore Emplacement

East

0 50 100
Km

4. PRESENT

West

Goosly Lake Monzonite Volcanics Stock

Ore

Goosly Sequence

Gabbro-Monzonite Complex

East

0 3 6
Km

FIGURE 5.2-1: SCHEMATIC HISTORICAL GEOLOGY, EQUITY SILVER MINE (after Pease)
size variation. The distinction between dust, ash and lapilli tuff is based on the maximum particle diameters of 0.5mm, 5.0mm and 50mm (0.02, 0.20, 2.0 inch) respectively. Dramatic particle size variations over relatively short distance has been observed.

There is an occurrence of dykes of several different compositions on this property. They are classed into two major groups, fine grained andesite and feldspar porphyry. In the Southern Tail deposit, feldspar porphyry dykes strike north-south and dip slightly steeper than the ore horizon. Width ranges from 6 to 10 m (20 to 30 ft). There are also many andesite dykes throughout the Southern Tail deposit and they are generally narrow and erratic. The dykes are not mineralized.

The structure of the deposit is relatively simple since the region has not been exposed to intense deformations. There are no large scale faults on the property. Tabular in form with a strike of 025 degrees, the deposit dips 30 to 50 degrees westerly. The strike length of the ore zone is about 750 metres (2460 ft) with a down dip extension of more than 300m (980 ft). The ore zone thickness varies from 15 to 30 m (50 to 100 ft). The pyroclastic division, including the dust tuffs, is characterized by a reticulate fracture pattern. The fracture intensity varies greatly but it generally increases closer to the mineralized areas. The ash tuffs are less fractured than the dust tuffs (Table 4).

The mineralization of the Southern Tail deposit is
<table>
<thead>
<tr>
<th>ROCK TYPE</th>
<th>HARDNESS</th>
<th>APPROXIMATE RANGE OF UNIAXIAL COMRESSIVE STRENGTH (MPa)</th>
<th>DENSITY (g/cc)</th>
<th>RQD</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quartz Latite Dyke</td>
<td>R4-R5</td>
<td>89-166</td>
<td>13000-24000</td>
<td>2.58</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dust Tuff</td>
<td>R4</td>
<td>55-111</td>
<td>8000-16000</td>
<td>2.78</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ash Tuff</td>
<td>R3</td>
<td>27-55</td>
<td>4000-8000</td>
<td>2.78</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mineralized Tuff</td>
<td>R2-R3</td>
<td>20-62</td>
<td>3000-9000</td>
<td>3.00</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
structurally controlled. Sulfides were deposited as open space fillings and their abundance increased with the intensity of fracturing. Pyrite and arsenopyrite are the most common minerals in the deposit. However, the principal copper mineral is chalcopyrite while tetrahedrite is the principal silver mineral. The ore occurs as veins and dissemination of massive sulfides present as local pods. The orebody is narrow and the mineralization is erratic, leading to wide variations in grade over short distances. In addition, the western wall of the pit was more subjected to alteration than the eastern wall.
5.2.2 The Blasting Procedures at Equity Silver

At Equity Silver, the mining equipment was selected to operate with 10 meter (33 ft) benches. However, due to dilution constraints, the bench height was reduced to 5 metres (16 ft) excluding the 1.5m (5 ft) subdrill depth. In addition, as the pit floor became closer to the designed pit limit, the number of working faces was reduced and choked blasts became more frequent. During this research project, many of the blasts were oriented east-west (across the pit), with the blasting taking place in the strike direction. In such conditions, controlled blasting is not performing well, especially in the highly broken rock mass. Moreover, designed in this configuration, a given blast pattern will include many different rock types (see Figure 5.2-2).

The blast patterns were 5 m by 5 m (16 ft by 16 ft) with the 229 mm (9 inch) diameter blastholes and 4 m by 5 m (13 ft by 16 ft) with the 200 mm (7 7/8 inch) diameter holes. The blast geometry was therefore very disadvantageous. Because the loading practice was based on the collar height, which, in return, was function of the hole diameter, the powder factor, within a given pattern, was highly variable as the rock density differs from one rock type to the next. In addition, the variations in hole depth due to caving, caused by the high degree of fracturing of the rock mass and the groundwater, were affecting the energy distribution within a given rock type. The
FIGURE 5.2-2: EQUITY SILVER MINE 1260m BENCH

GEOLOGY, scale 1:5000
groundwater has also a more direct influence on energy distribution. The utilization of bulk ANFO was sometimes restricted by the amount of groundwater. When the blastholes cannot be properly lined with plastic sleeves, slurry explosives were used. Consequently, energy distribution was seldom uniform within the blast patterns.

In order to minimize the possible damage to the pit walls, the row per row initiation system was modified to a hole per hole delay sequence on a V1 pattern. Therefore, the amount of explosive detonating on the same delay was reduced, the geometric ratios were improved as well as the displacement and fragmentation of the muck pile. Nevertheless, the results were still non-uniform.

5.2.3 Development of the Correlation Between RQI and Powder Factor

The different blasting domains within the pit were differentiated by the following characteristics:

1. Rock type: dust tuff, ash tuff, quartz latite dyke, mineralized ash tuff, mineralized dust tuff, quartz porphyry dyke.

2. Degree of alteration: the tuffs were also divided into east and west wall zones, according to the degree of alteration.

At Equity, some domains did not cover very large areas due to the width of the pit at the 1260 m (4134 ft) elevation.
Consequently, data were limited, especially in the northern end of the pit (ash tuff). The RQI values were compiled, in a given domain, for the different bit diameter and for the recorded data (Table 5, Figure 5.2-3). In the Southern Tail pit, two BE 40-R drills were operating. One was drilling 229 mm (9 inches) diameter blastholes whereas the other one was drilling 200 mm (7 7/8 inches) diameter blastholes. Although they were generally operating on different blast patterns, it occurred, in a few occasions that both were drilling side by side on the same pattern. It was observed that, for a given set of rock conditions, the RQI compiled from the smaller size blastholes was, in the average, 0.60 times the RQI compiled from the 229 mm (9 inch) diameter blastholes. This observation has to be analysed with Little's¹³ findings on the effect of bit size and drill model on RQI.

Since the blasting procedures were variables, only the choked production blasts performed along the strike direction provided sufficient data to permit analysis. However, the conditions within this group of blasts were not constant, changes in the delay and initiation sequence being tested. Blast result analysis and compilation of the powder factors were performed, taking into account the different rock types/domains within the pattern. Table 6 summarizes the results. The optimum powder factors were then compared with the Rock Quality Index allocated to each domain (Figure 5.2-4).
<table>
<thead>
<tr>
<th>DOMAINS</th>
<th>RQI (229mm-9 inch)</th>
<th>RQI (200mm-7 7/8 inch)</th>
<th>RQI-REC&lt;sup&gt;(1)&lt;/sup&gt; (229mm-9 inch)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>kg-min/m (*1000)</td>
<td>lbs-min/ft (*1000)</td>
<td>kg-min/m (*1000) lbs-min/ft (*1000) kg-min/m (*1000) lbs-min/ft (*1000)</td>
</tr>
<tr>
<td>I</td>
<td>Dust Tuff, east</td>
<td>61.0</td>
<td>40.9</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>II</td>
<td>Dust Tuff, west</td>
<td>55.8</td>
<td>37.5</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>III</td>
<td>Ash Tuff, east</td>
<td>75.3</td>
<td>50.6</td>
</tr>
<tr>
<td>IV</td>
<td>Ash Tuff, west</td>
<td>66.1</td>
<td>44.4</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>V</td>
<td>Quartz Latite Dyke</td>
<td>65.6</td>
<td>44.1</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>VI</td>
<td>Mineralized DT, east</td>
<td>54.8</td>
<td>36.8</td>
</tr>
<tr>
<td>VII</td>
<td>Mineralized DT, west</td>
<td></td>
<td></td>
</tr>
<tr>
<td>VIII</td>
<td>Mineralized AT, east</td>
<td>77.7</td>
<td>52.2</td>
</tr>
<tr>
<td>IX</td>
<td>Mineralized AT, west</td>
<td></td>
<td></td>
</tr>
<tr>
<td>X</td>
<td>Quartz Porphyry Zone</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<sup>(1)</sup> Recorder  
<sup>(2)</sup> One pattern only  
<sup>(3)</sup> 12 data points only
FIGURE 5.2-3: ROCK QUALITY INDEX VALUES FOR EACH DOMAIN RANKED IN ORDER OF DECREASING ORDER, EQUITY SILVER.
TABLE 6

CORRELATION BETWEEN RQI AND POWDER FACTORS AT EQUITY SILVER MINE

<table>
<thead>
<tr>
<th>DOMAINS</th>
<th>ROCK QUALITY INDEX (229mm - 9 inch)</th>
<th>POWDER FACTOR (choked)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>kg-min/m (*1000)</td>
<td>lbs-min/ft (*1000)</td>
</tr>
<tr>
<td>I</td>
<td>61.0</td>
<td>40.9</td>
</tr>
<tr>
<td>II</td>
<td>55.8</td>
<td>37.5</td>
</tr>
<tr>
<td>III</td>
<td>75.3</td>
<td>50.6</td>
</tr>
<tr>
<td>IV</td>
<td>66.1</td>
<td>44.4</td>
</tr>
<tr>
<td>V</td>
<td>65.6</td>
<td>44.1</td>
</tr>
<tr>
<td>VI</td>
<td>54.8</td>
<td>36.8</td>
</tr>
<tr>
<td>VII</td>
<td>53.6</td>
<td>(36.0)</td>
</tr>
<tr>
<td>VIII</td>
<td>77.7</td>
<td>52.2</td>
</tr>
<tr>
<td>IX</td>
<td>72.6</td>
<td>48.8</td>
</tr>
<tr>
<td>X</td>
<td>54.9</td>
<td>(36.9)</td>
</tr>
</tbody>
</table>

no optimum powder factor allocated to domains III, VIII, IX
FIGURE 5.2-4: PROPOSED CORRELATION BETWEEN ROCK QUALITY INDEX AND POWDER FACTOR AT EQUITY SILVER MINE.
5.2.4 Analysis of the Results

No real optimization of the powder factor was performed at Equity Silver. The optimum powder factors listed were determined from the evaluation of the blast results and calculated from the blast reports. Due to the variations in the blasting procedure, the evaluation of the results and the comparison between the powder factors was difficult. Consequently, it was decided that only the production choked blasts would be considered for this research project. Other types of blast performed at Equity Silver during the data gathering period were perimeter, ramp construction and cut sinking blasts. The few perimeter blasts performed were very good. In general, the blasting procedure in regard to the slope stability was improved as design guidelines were specified.

One of the peculiarities of the blasting procedure at Equity Silver, is the fact that the blasting results are very dependent on the collar/charge ratio. With 6.5 m (21.3 ft) holes on 5 m (16 ft) benches, a slight variation in the charge length makes the ratio change very rapidly. Increasing the powder factor by increasing the charge (length) will cause the stemming column to be reduced and therefore, to be more easily ejected, thus reducing the explosive confinement and efficiency.

The examination of the RQI values allocated to each domain shows that the eastern side of the pit, corresponding to the less altered rocks at bench elevation, possesses higher RQI than the
western domains. This suggests that the RQI is sensitive to the degree of alteration. However, it was observed that groundwater levels were higher close to the east wall.

Due to the absence of sufficient blast result observations, no relationship between the RQI and the powder factor were obtained in the northern part of the pit. In addition, the RQI values allocated to those domains are derived from a smaller group of data, and should not be considered as accurate as the ones obtained in the southern and central part of the pit where two benches were mined out during the data gathering period. The interpretation of the data related to these domains yields Figure 5.2-4. It shows that there is a trend between the rock mass blastability and the Rock Quality Index as predicted by Leighton\(^1\).
5.3 **LORNEX MINE**

5.3.1 **Summary of the Geology**

The Lornex copper-molybdenum deposit is situated in the Highland Valley of British Columbia, 74 Km (46 miles) southwest of Kamloops. The elevation of the mine site is approximately 1060 m (3478 ft). Because of the low grade, the Lornex orebody has to be mined on a large scale to be profitable. Full production stated in 1972 and during the summer of 1983, the designed mill feed rate was 75300 tonne/day (83000 s.ton/day). The stripping ratio is around 2.2 : 1.0.

The geology has been described by Waldner et al. The deposit is entirely within the Skeena Quartz Diorite (SQD) zone, a slightly porphyric, medium to coarse-grained rock. The ore zone is approximately 2100 m (6890 ft) long and 750 m (2460 ft) wide. The deposit is believed to plunge 30 to 40 degrees toward the northwest to a depth in excess of 750 m (2460 ft). The orebody is truncated on the west side by the Lornex fault striking northerly with variable westward dip. On the southeast end of the property, a pre-mineral quartz porphyry dyke trends northwesterly. In some area, contacts of the dyke are not well defined because of the alteration of adjoining Skeena Quartz Diorite. The west part of the pit is within the Bethsaida Gronodiorite Zone (Figure 5.3-1).

The predominant sulphide minerals are chalcopyrite,
FIGURE 5.3-1: LORNEX OPEN PIT MINE. scale 3/4"=1000'
bornite, molybdenite and pyrite. They occur primarily as fracture fillings with quartz and as fracture coatings, veins average 5 to 15 millimeters (0.2 inches to 0.6 inches) in width, ranging from a hairline to more than a meter. Strike length may exceed 200 meters (656 ft). Mineralization at Lornex is controlled by fracture density and distribution. Structural mapping has produced over 11000 data points plotted on Figure 5.3-2. The three major attitudes for copper-molybdenum veins are dominant in distinct zones of the orebody, although, in the central and western zones, there is an overlap of all three vein attitudes, resulting in higher grades. In addition, two post-mineral fracture systems have been recognized in the pit.

Four types of hydrothermal alteration associated with sulphide mineralization occur in this deposit. The most important one is the argillic alteration from which results the presence of sericite, kaolinite, montmorillonite and chlorite. In general, grade increases with an increasing intensity of argillic alteration and original rock strength decreases with an increasing degree of any type of hydrothermal alteration. The porphyry dyke is less affected by alteration.

The following statements on the Lornex deposit are quoted from Waldner et al.10

"1. The principal sulphides form a concentric pattern with bornite in the center, chalcopyrite outside bornite and molybdenite zone overlapping portions of the bornite and chalcopyrite zones. Pyrite forms a halo around the ore
FIGURE 5.3-2: LOWER HEMISPHERE, EQUAL AREA STEREOGRAHIC PROJECTIONS OF STRUCTURES MAPPED IN THE LORNEX OPEN PIT (after Waldner et al.)
Zone.

2. Copper grades and total sulphide content decrease outward from the core of the orebody to its periphery.

3. Sulphide and alteration zones are deep in the north and shallow in the southern portions of the orebody, indicating a 30 to 40 degree northwest plunge.

4. Zones of moderate to intense argillic alteration correspond to higher grades."

5.3.2 Blasting Procedures at Lornex

At Lornex, a vast quantity of material is moved every day. It is one of the largest truck and shovel operations in Canada and therefore numerous large size units of equipment are used. At the time the research project was taking place, the mining method was straightforward. The stage one pit having been excavated, they were pushing back the east and south walls. Mining was accomplished on several benches to obtain optimum blending capacity and to maintain the stripping ratio. Production blasts were choked most of the time while the perimeter blasts free-faced only on the eastern wall. In the ore zone of the southern part of the pit, all the blasts were choked, even at wall contact.

Blast patterns were ranging from 8.5 m by 8.5 m (28 ft by 28 ft) in the waste rock on the east wall to 11 m by 11 m (36 ft by 36 ft) in the soft ore zones. Subdrilling, on the 12.2 m
(40ft) benches, was also variable, being nonexistent in the soft ore zones. The blast results were generally ranging from poor to fair. The powder factor calculation did not consider the weight strength of the explosives used (ANFO, AL-ANFO, Slurries). Therefore, the energy distribution within the pattern was not uniform. In addition, drilling was not accurate and variations of the burden, spacing and depth dimensions were frequently noted. Finally, yellow/orange smoke was common, indicating wet ANFO or non-optimum mix. The production blasts, shot in-line or on a V pattern were generally choked, but when they were free-faced, it was observed that they choked themselves due to the low energy factor and/or the short 25 ms delay period between the rows. During the research project, it was shown that the practice of using two downlines of reinforced primacord in a 250 mm (9 7/8 inches) blasthole was producing side initiation of the explosive column and consequently was a very ineffective use of the explosive energy available.

The design of the wall control blasts was poor. The last 6 m (20 ft) of the pattern, closest to the wall, were allocated a higher energy factor than the rest of the pattern. The initiation was performed by a down-the-hole system permitting longer delays between the rows of the V pattern. However, because of the general underloaded conditions of the rock mass, the displacement was not sufficient and the blasts were choking themselves. Vibrations and crack propagation toward the rock slopes were damaging. The walls did not look good. Design
modifications concerning energy distribution were suggested.

It should be noted that three Hercudet test blasts were performed at Lornex during this research project. The very good results obtained were partly due to the system, but also to the well engineered design.

5.3.3 Development of the Correlation Between RQI and Powder Factor

It was not possible to derive a correlation between the Rock Quality Index and the powder factor at Lornex. The main reason being the lack of useful blasting data. Approximately half of the blasts monitored were side initiated. On the east wall, blast results were improving as mining progressed toward the south end, but the satisfactory results described in chapter four were not reached. Blasts were underloaded.

The compilation of the RQI values was done from the driller logs and the recorded charts. Of the five Bucyrus 45-R drills, three were generally operating on any given shift. Although some steel tooth bits were used in the soft ore zones, no dramatic variation in the Rock Quality Index from the WC insert bits were observed. However, it is a potential source of interpretation error. The domains were defined from

a) Rock types: Skeena Quartz Diorite (SQD) or Quartz Porphyry (QPP)

b) Degree of alteration of the Skeena Quartz Diorite
Some of the domains, especially on the east wall, were limited in surface to one or two blast patterns. In the southern end of the pit, the areas were larger. Because no other drilling data from the previous months were analysed, the quantity of data is a function of the domain area. Although no correlation was derived with the powder factor, there is a good correlation between the Rock Quality Index and the degree of alteration of the Skeena Quartz Diorite (Figure 5.3-3, Table 7).

In addition, in the ore zones of the southern end of the pit, a correlation was established between the RQI and the grindability of the rock (Figure 5.3-4). A plan of the previous bench was superimposed over the blasthole plan and the grinding rate allocated to an ore block on the preceding bench was directly compared with the average RQI on the block's projection, 12.2 m (40 ft) deeper. The same exercise was also performed on a larger scale.
FIGURE 5.3-3: ROCK QUALITY INDEX IN DIFFERENT DOMAINS OF LORNEX MINE.
<table>
<thead>
<tr>
<th>DOMAINS</th>
<th>ROCK QUALITY INDEX (250mm-9 7/8 inch)</th>
<th>ESTIMATED UNIAXIAL COMPRESSION STRENGTH (MPa)</th>
<th>(psi)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Driller logs Recorder</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>kg-min/m lbs-min/ft kg-min/m lbs-min/ft</td>
<td></td>
<td></td>
</tr>
<tr>
<td>I</td>
<td>SQD massive to weakly altered</td>
<td>49.3 33.1 40.6 27.3</td>
<td>261</td>
</tr>
<tr>
<td>II</td>
<td>SQD altered and fractured</td>
<td>33.6 22.6 29.8 20.0</td>
<td>111</td>
</tr>
<tr>
<td>III</td>
<td>SQD moderate argillic alteration</td>
<td>29.6 19.9 24.6 16.5</td>
<td></td>
</tr>
<tr>
<td>IV</td>
<td>SQD weak to intense argillic alt.</td>
<td>26.3 17.7 24.1 16.2</td>
<td>179</td>
</tr>
<tr>
<td>V</td>
<td>QPP weakly altered</td>
<td>24.9 16.7 - -</td>
<td>-</td>
</tr>
<tr>
<td>VI</td>
<td>SQD Intense argillic alteration</td>
<td>20.4 13.7 - -</td>
<td>-</td>
</tr>
<tr>
<td>VII</td>
<td>SQD moderate to intense argillic alt.</td>
<td>24.0 16.1 - -</td>
<td>-</td>
</tr>
<tr>
<td>VIII</td>
<td>QPP weakly altered</td>
<td>28.4 19.1 - -</td>
<td>132</td>
</tr>
<tr>
<td>IX</td>
<td>SQD moderate to intense argillic alt. (1) from point-load tests</td>
<td>26.3 17.7 - -</td>
<td>50</td>
</tr>
</tbody>
</table>
FIGURE 5.3-4: RELATIONSHIP BETWEEN ROCK QUALITY INDEX AND GRINDING RATE AT LÖRNEX.
5.3.4 Analysis of the Results

It is impossible to derive a correlation when one side of the relationship is missing. Nevertheless, the rock mass in domain I requires more explosives than the rock mass in the southern ore zones, in accordance with the general relationship.

The fact that the Rock Quality Index is sensitive to the variations of the strength properties of the rock mass is confirmed by the correlation between the RQI and the degree of alteration. The more altered the rock mass, the lesser the RQI. This could be of interest in rock slope engineering, in order to monitor the variations in rock quality with depth.

The second interesting relationship, although very approximative at this time, is the variation of the grindability with the Rock Quality Index. It is not a coincidence, both being fragmentation processes. In this research project, the grindability rating was obtained from the previous bench and transferred directly one bench down. The mine geologist will investigate this subject in more detail.
5.4 GREENHILLS MINE

5.4.1 Summary of the Geology

The Westar Mining Greenhills Mine is located 35km (22 miles) north of the town of Sparwood, in the east Kootenay region of southeastern British Columbia. The pit elevation is presently 2000m (6562 ft) although the coal loadout facilities, shops and offices are at lower elevations. The property is divided in several zones or pits, along the ridge, in order to expose a sufficient tonnage of coal. The first shipment was sent in August 1982 and the project life is estimated at 20 years. The actual yearly production range between 2.8 and 3.0 M tonnes (2.2 to 3.3 M s.ton) of clean coal although it can be expanded to 4.0 M tonnes/year (4.4 M s.ton/year). Out of the 29 seams identified on the property, four of them, 1, 7, 10 and 16 are major and generally continuous throughout the property.

According to Britch¹¹, the major structure is a broad open syncline which plunges gently to the north. The limbs of the syncline dip between 20 to 40 degrees on the west limb and from 20 to 60 degrees on the east limb. Regional and local faulting are also present. A general outlook of the stratigraphy is given in Figure 5.4-1. During the research project, data gathering has been concentrated in the Cougar pit where seams 16 to 29 are exposed on the west limb (Figure 5.4-2). One of the major features of the Cougar pit is the presence of crossfaults.
FIGURE 5.4-1: GREENHILLS MINE STATIGRAPHY (after Britch)

- SANDSTONE
- SILTSTONE
- MUDSTONE
- COAL

ELK MEMBER
SANDSTONE SILTSTONE MUDSTONE 60 m+
COAL BEARING MEMBER
SANDSTONE SILTSTONE MUDSTONE COAL 550 m
MOOSE MOUNTAIN MEMBER
CHERT SANDSTONE 12-25 m
JURASSIC FERNIE FORMATION
SANDSTONE SILTSTONE SHALE 245 m+

SEAM 29
SEAM 28
SEAM 27
SEAM 25-26
SEAM 22
SEAM 20
SEAM 19
SEAM 18
SEAM 16-16 L
SEAM 13
SEAM 11
SEAM 10-10 L
SEAM 9
SEAM 8-1
SEAM 8-2
SEAM 7
SEAM 5
SEAM 3
SEAM 1
SEAM M
MOOSE MTN MEMBER

500 m
600 m

FIGURE 5.4-1: GREENHILLS MINE STATIGRAPHY (after Britch)
FIGURE 5.4-2: GREENHILLS COUGAR PIT scale 1:5000
striking easterly and dipping 70° to the south. The faulted areas exhibit oxidized materials of very low strength. In addition, the sedimentary sequence is slightly altered and the rock mass is more fractured. In the northern part of the Cougar pit, the rock shows a more competent behaviour. Table 8 shows the different strength parameters of the rocks.
# Table 8

**Rock Properties Greenhills Mine**

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Hardness</th>
<th>Approximate Range of Uniaxial Compressive Strength (MPa)</th>
<th>Density (g/cc)</th>
<th>Shear Strength of Discontinuity</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>R4</td>
<td>115-155</td>
<td>2.70</td>
<td>28°</td>
</tr>
<tr>
<td>Siltstone</td>
<td>R3-R4</td>
<td>40-152</td>
<td>2.70</td>
<td>33.5°</td>
</tr>
<tr>
<td>Mudstone</td>
<td>R2-R3</td>
<td>60-99</td>
<td>2.70</td>
<td>31°</td>
</tr>
</tbody>
</table>
5.4.2 The Blasting Procedures at Greenhills

At Greenhills mine, blasts are designed to produce a fine and evenly fragmented muck pile for optimum loading conditions and high productivity. Because of the nature of the operation and the geology of the mine, all blasts are directed toward a free face. However, as the dip angle of the beds is reduced, additional row(s) of holes are required at the toe location to improve fragmentation and displacement. At the present time, no perimeter blast has been shot, the operation being at the top of the ridge and therefore far away from the final wall.

Benches are designed 14 metres (46 ft) high. Blastholes 270 mm (10 5/8 inches) in diameter are drilled to a depth of 15.5 metres (51 ft) on a staggered 7.0 by 8.1 m (23 ft by 26.5 ft) pattern laid-out by the surveyors. Caving occurs at bench crest areas and blastholes are then redrilled. Even though the design powder factor approach is used, the real powder factor may be increased by the presence of filler holes added, when needed, within the pattern and/or by the presence of blastholes deeper than 15.5 metres (51 ft). In both cases, there is no burden volume allocated to the extra explosive load. However, with free digging, the overall powder factor is generally slightly less than the net design powder factor.

The explosive used is straight ANFO. Initiation is by the NONEL system which eliminates side initiation. Delays between the rows are kept to 50 ms to avoid the probability of
disruption of the explosive column along the well defined parting planes between the different sedimentary beds. The design powder factor is evaluated by the drill and blast foreman based on the geology, his experience on the previous benches, the drill performance and the cuttings. In the hard zones, such as the competent sanstone beds of the 16 and 20 upper seams hanging walls, in the northern part of Cougar pit, the powder factor is presently fixed to 0.55 kg/m$^3$ (0.41 lbs/s.ton) down from the 0.59 kg/m$^3$ (0.44 lbs/s.ton) used during the 1983 summer. During the same blast optimization program, the powder factor allocated to the soft areas, such as the faulted zones in the southern Cougar pit, was reduced to 0.47 kg/m$^3$ (0.35 lbs/s.ton). Blast results are still very good.

5.4.3 Development of the Correlation Between RQI and Powder Factor

At Greenhills, coal is mined on benches in a sequence that starts from the east (seam 29) westward. The minable seams in Cougar Pit are seams 29, 28, 27, 25-26, 22, 20 upper, 20 lower, 18 and 16. The waste rock that lies between the seams are called according to the number of the following seam (i.e., the 16 seam hanging wall is the waste that has to be moved to gain access to the 16 seam). The blasting domains are therefore very different from one to the next, as a function of the rock types and degree of fracturing of the sedimentary sequence. The
nature of the domains remains constant from one bench to the next. However, the presence of faults in the southern part of the pit slightly disturbs the continuity of the coal seams in the striking direction, and increases the intensity of fracturing and alteration of the rocks. Therefore, a differentiation is done, when necessary, between the northern and southern domains. Areas of the domains were generally large. Also, as the recorder was not operating, drilling data from the previous bench were analysed; the drilling and blasting records permitted easy interpretation. The allocation of the RQI to the different domains is summarized in the following pages (Table 9, Figure 5.4-3).

Since the blasting procedures at Greenhills are kept constant and the powder factor is the only variable, evaluation of blast results was made simple. As the opposite of Lornex Mine, Greenhills blasts are designed to produce a evenly fragmented, well displaced muck pile that optimizes the productivity of the loading equipment. Therefore, no blast results were considered underloaded. Table 10 and Figure 5.4-4 describe the relationship between RQI and energy factor derived from the data gathered. It is interesting to note that the powder factor has been reduced at Greenhills since the research project took place. Since Greenhills expresses the explosive consumption in kg/m³, this unit is also included in the table.
### TABLE 9

**SUMMARY OF RQI VALUES AT GREENHILLS MINE**

<table>
<thead>
<tr>
<th>DOMAINS</th>
<th>ROCK QUALITY INDEX (270mm-10 5/8 inch)</th>
<th>kg-min/m (*1000)</th>
<th>lbs-min/ft (*1000)</th>
</tr>
</thead>
<tbody>
<tr>
<td>I Seam 28 HW</td>
<td>32.3</td>
<td>21.7</td>
<td></td>
</tr>
<tr>
<td>II Seam 27 HW</td>
<td>33.6</td>
<td>22.6</td>
<td></td>
</tr>
<tr>
<td>III Seam 25-26 HW</td>
<td>37.4</td>
<td>25.1</td>
<td></td>
</tr>
<tr>
<td>IV Seam 22 HW</td>
<td>29.2</td>
<td>19.6</td>
<td></td>
</tr>
<tr>
<td>V Seam 20up HW north</td>
<td>62.9</td>
<td>42.3</td>
<td></td>
</tr>
<tr>
<td>VI Seam 20up HW south</td>
<td>39.7</td>
<td>26.7</td>
<td></td>
</tr>
<tr>
<td>VII Seam 20lw HW north</td>
<td>39.1</td>
<td>26.3</td>
<td></td>
</tr>
<tr>
<td>VIII Seam 11w HW south</td>
<td>25.9</td>
<td>17.4</td>
<td></td>
</tr>
<tr>
<td>IX Seam 21 HW north</td>
<td>48.7</td>
<td>32.7</td>
<td></td>
</tr>
<tr>
<td>X Seam 16 HW south</td>
<td>42.1</td>
<td>28.3</td>
<td></td>
</tr>
</tbody>
</table>
FIGURE 5.4-3: ROCK QUALITY INDEX VALUES FOR EACH DOMAIN RANKED IN ORDER OF DECREASING QUALITY, GREENHILLS.
TABLE 10
CORRELATION BETWEEN RQI AND POWDER FACTORS AT GREENHILLS MINE

<table>
<thead>
<tr>
<th>DOMAINS</th>
<th>ROCK QUALITY INDEX (270mm-10 5/8inch)</th>
<th>POWDER FACTOR</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>kg-min/m (*1000)</td>
<td>lbs-min/ft (*1000)</td>
</tr>
<tr>
<td>I</td>
<td>32.3</td>
<td>21.7</td>
</tr>
<tr>
<td>II</td>
<td>33.6</td>
<td>22.6</td>
</tr>
<tr>
<td>III</td>
<td>37.4</td>
<td>25.1</td>
</tr>
<tr>
<td>IV</td>
<td>29.2</td>
<td>19.6</td>
</tr>
<tr>
<td>V</td>
<td>62.9</td>
<td>42.3</td>
</tr>
<tr>
<td>VI</td>
<td>39.7</td>
<td>26.7</td>
</tr>
<tr>
<td>VII</td>
<td>39.1</td>
<td>26.3</td>
</tr>
<tr>
<td>VIII</td>
<td>25.9</td>
<td>17.4</td>
</tr>
<tr>
<td>IX</td>
<td>48.7</td>
<td>32.7</td>
</tr>
<tr>
<td>X</td>
<td>42.1</td>
<td>28.3</td>
</tr>
</tbody>
</table>
ROCK QUALITY INDEX
(270mm-10 5/8 inch)

FIGURE 5.4-4: PROPOSED CORRELATION BETWEEN ROCK QUALITY INDEX AND POWDER FACTOR AT GREENHILLS MINE.
5.4.4 Analysis of the Results

The determination of a good correlation at Greenhills is due to three factors: the domains boundary were well defined, the drilling data were numerous enough within each of the domains and the blasting procedure was kept constant. However, as for the other sites, the powder factors were not really optimized. The values allocated to each domain are the minimum powder factor that still yields good blasting results. Therefore, it is not known if the powder factors can be further reduced.

The two domains, described by the drill and blast foreman as the toughest to blast showed the highest Rock Quality Index. The domains are described as competent sandstone and competent siltstone with sandstone. The faulted end of these two domains, easier to blast, possess lower RQI.

The relationship developed at this site between the Rock Quality Index and the powder factor is still at an early stage of development. Further testing and data evaluation are suggested in order to fine-tune the correlation. The results however are very encouraging.
5.5 DISCUSSION AND INFERENCES

5.5.1 Accuracy of the Input Data

In previous studies on Rock Quality Index, the accuracy of the input data was considered a major factor in the establishment of a valid rock mass characterization index. At Afton Mine, the determination of domain boundaries by conventional methods and the evaluation of the Rock Quality Index within these domain has yielded good results. The domain areas were large and the drilling data numerous. However, Leighton was concerned by the high degree of variation in the RQI in adjacent blastholes. In this research project, the author observed maximum variations ranging between 50 to 200% with a common departure ranging between 25 to 140%. This high degree of variability suggests that a considerable amount of data is needed in order to make the average or peak value representative of the distribution. These are not always available. The variations are not only caused by the inhomogeneous nature of the rock mass, but also by the drilling practice and the dedication of the drilling crew in reporting the operational parameters. Generally, the drilling time is reported to the nearest five minutes, sometimes generously rounded-up to the upper limit. The down pressure is usually specified as the maximum reading (ie., 4.1 MPa (600 psi) while most of the hole was drilled at 3.4 MPa (500 psi)) or as a range
3.4 MPa to 4.1 MPa (500 - 600 psi). The hole depth is accurate most of the time although some drillers prefer to drill a few more feet to be on the safe side (especially in caving ground). If the Rock Quality Index approach to rock mass characterization is to be implemented at one site, the drilling crew must be aware of the influence of their reports on the decision making process that follows. The author believes that no real problems would be encountered. In the present research project, it was believed that the drill performance recorder would provide very accurate data so that the quantity needed in every domain would be dramatically reduced. The RQI values compiled at Equity Silver and Lornex were computed only from the data gathered during the period of time the author spent at each site, considering that the RQI obtained from the recorder would back-up the establishment of the correlation. At Greenhills, the RQI was compiled from drilling data obtained from records up to six months old and covering two benches in almost every domain of the Cougar pit. Accurate and extensive data are important. It seems that the quantity compensates for the wide variations.

5.5.2 Drill Performance Recorders

The use of a drill performance recorder partly reduces the degree of variation of the drilling data. The RQI values compiled from the recorded charts do not include the uncertainties carried by the driller's method of reporting the
operating parameters. However, there are still some external factors, reducing the accuracy of the RQI, that are recorded. For example, the bit wear, the difference in bailing velocity and/or pressure between the drill, slight variations in the drilling practice, etc. In addition, the use of a chart recorder may add a new source of bias. When the applied weight on the bit varies constantly along the blasthole depth, the estimation of the (average) applied weight for that particular blasthole includes an estimator bias. In the worst cases, the data were not included in the data file. One other problem that was experienced is the determination of the blasthole number corresponding to the recorded data. On the charts, the drillers were required to register the first and last blasthole number of the shift. Nevertheless, it was sometimes difficult to relate all the holes drilled with the records available. Finally, the drillers were asked to write the blasthole number besides the recorded data. The author feels that there are many incentives to the installation of a drill performance recorder on blasthole drill rigs. However, if the installation of such a device is made with the idea of gathering rock mass characterization data, a computerized system that would compile, store and release the information should be developed. In the mean time, mine operators can provide the drilling crew with a $10.00 pocket calculator and teach them how to compute the Rock Quality Index. In the office, data would be easily filed on a computer by technicians. The investigation of the use of a drill
performance recorder should be continued in order to permit the development of the optimum system. They are definitively useful in eliminating the driller's bias. However, if the quality of the data is important, the quantity of valuable data is as important when characterizing the rock mass.

5.5.3 Drilling Procedures

Drilling efficiency influences the Rock Quality Index. It is possible, within a given operation, that the efficiency of the various drilling units varies due to the driller and/or the conditions of the equipment. The effect of the drilling crews has already been covered and therefore, this paragraph will review the basic mechanical factors.

The condition of the drills, such as the age or wear of the components is an other important factor in the establishment of correlations with the Rock Quality Index. Many operations pay much more attention to the maintenance of trucks and shovels than to the drill maintenance. Compressors and rotary motors should be checked and compared in order to determine if the drills are operating at the same level of efficiency. Generally, drills of the same model bought at different times will not perform equally. The same is true with different manufacturer units. The comparison of the Rock Quality Index in units of weight rather than pressure eliminates the differences in the relationships between hydraulic system pressure and
applied weight which are variables from one drill model (manufacturer) to the other. It is also suggested to calibrate the pressure gage in the cabin to ensure accurate readings when no drill performance recorder is used.

A very important question was left open by Leighton \(^1\) in his report. He wondered how the wear of tricone bits does influence the penetration rate, other variables kept constant. This question was investigated by the author. Although very approximative, Figure 5.5-1 illustrates the three situations that may arise. The variations of the penetration rate are a function of the type of wear. Bit manufacturers claim that under normal drilling conditions, carbide insert bits will not show any reduction in the penetration rate until the very last hours. Nevertheless, the graph suggests that some variations will occur. Therefore, the compilation of numerous drilling data in every domain is recommended to smooth off these variations in RQI.

Finally, the comparison of the RQI values at Equity Silver Mine, obtained from the different diameter blastholes and a review of the Little's \(^1\) findings prompt the following question: would a given rock mass yield to the same RQI when drilled with different diameter bits if the degree of drilling efficiency is kept constant. This is a very complex problem related to the determination of the level of drilling efficiency and optimum drilling conditions. The drilling guidelines provided by the bit manufacturers and concerning the applied weight per
Figure 5.5-1: Influence of tricone rotary bit wear on penetration rate (after Paquette\textsuperscript{12}).

1. Cutter shape changes (more blunt)
2. Cutter shape stays constant with time; cutter and cone matrix wear equally (normal)
3. Cone matrix wears faster than cutters
inch of bit diameter are based on experience. According to Burke\textsuperscript{14}, they yield a constant penetration rate, although the penetration rate may increase as the diameter is reduced. He added: "With the exception of very hard formations, the penetration rate is generally limited only by the bailing capacity of the drilling equipment." As the diameter of the blasthole is reduced, the bailing velocity increases and the cutting removal process is generally improved. Therefore, as the bit diameter is reduced, the Rock Quality Index decreases accordingly (Table 11). Further research is suggested in order to obtain a relationship that would permit conversion of RQI values obtained from different bit diameters. The involvement of a bit manufacturer company and its research facilities would be appreciated.

5.5.4 Blasting Procedures

It has been shown, in chapter four, how the variations in blast design influence the optimum design powder factor. This field research project has demonstrated this fact. The establishment of a correlation between RQI and optimum design powder factors is only possible when the blast design remains constant. Variations in the design parameters influence the results and make impossible the evaluation of the blast performance on a constant basis. Nevertheless, it is believed that a correlation can be obtained for any blast design as long
<table>
<thead>
<tr>
<th>BIT DIAMETER</th>
<th>APPLIED WEIGHT</th>
<th>PENETRATION RATE</th>
<th>ROCK QUALITY INDEX</th>
</tr>
</thead>
<tbody>
<tr>
<td>311mm - 12 1/4 inches</td>
<td>reduced</td>
<td>constant or slight increase</td>
<td>reduced</td>
</tr>
<tr>
<td>152mm - 6 inches</td>
<td>reduced</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
as they are kept constant within a given domain.

Choked blasting should be avoided. It is inefficient and damages the walls. There are very few occasions where it becomes the only solution. The design powder factor concept should be understood and implemented. Wall control blasting procedure starts with the production blasts. Even if the present walls are considered interim pit walls, stripping expenses can be postponed by producing more stable and steeper walls. In addition, it permits the development and testing of good wall control blasting procedures.

5.5.5 Rock Mass Conditions and Properties

The Rock Quality Index reflects rock mass properties and its failure behaviour. A review of the work on RQI shows that the strength of the rock correlates with the RQI. In addition, the RQI values are reduced in highly fractured rock masses. Within a predetermined blasting domain, the Rock Quality Index can be correlated with the optimum design powder factor.

It would be of interest to store RQI data in a geological model. Then, by using geostatistical routines, such as variograms or averaging methods, a more complete picture of the relationships between the Rock Quality Index and the rock mass properties could be obtained.

In this research project, two rock mass conditions have been noticed for their effect on RQI. In intensely fractured
rock masses, the drilling time may increase slightly due to caving ground and therefore increase the RQI values while the explosive requirements reduce with the increase in the degree of fracturing. The second factor is groundwater. It affects the RQI in two different ways. In highly broken rocks, the groundwater increases the intensity of caving. Finally, groundwater affects the bailing capacity of the drill when a column of water stands over the bit at the bottom of the hole. In both cases, the groundwater changes the penetration rate and increases the Rock Quality Index.

5.5.6 Correlations with the Rock Quality Index

A relationship between RQI and the design powder factor exists. It can be established in any operation if the basic guidelines are followed. At this time, the correlations are site specific in a sense that it is not possible to compare the RQI values of different operations nor the powder factors applied to distinct blast designs. As more data are obtained, a range of relationships should develop.

A correlation was also established at Lornex Mine between the RQI and the grinding rate of the ore. However, it appears that the accuracy of the relationship is also a function of the domain area. This constraint has to be evaluated with the degree of precision required by the milling operation.

Finally, variations in the Rock Quality Index in adjacent
holes are large enough that it suggests that a hole per hole loading approach may not be practical. Nevertheless, a comparison of the blast average RQI with the Rock Quality Index allocated to the domain could lead to a better loading practice on a day to day basis. However it should be remembered that many drilling factors influence the RQI, therefore care must be used when the data are few.
5.6 SUGGESTED FURTHER RESEARCH

The results obtained in the research project established in the Mining and Mineral Process Engineering of U.B.C. are interesting enough to permit the continuation of the research on Rock Quality Index. In this section, the author wishes to summarize the basic guidelines that would permit the definition of a fruitful correlation between the Rock Quality Index and the design powder factor. Also included are the points that require further investigation. It should never be forgotten that the Rock Quality Index approach to rock mass characterization has to remain simple and practical.

Guidelines for the establishment of a correlation between RQI and design powder factors

1) The factors that influence the determination of the Rock Quality Index and the optimum design powder factor have to be well understood. The author believes that this thesis report reviewed these factors and their influence on the drilling and blasting mechanisms in detail.

- The RQI is affected by
  a) the operator skill and honesty in producing accurate reports
  b) the drilling parameters
     i) weight, rotary speed and compressor capabilities,
although the last two should eventually remain almost constant.

ii) the bit design and wear.

C) the rock mass properties

i) structural: micro and macro scale.

ii) failure behaviour: uniaxial compressive strength, Young's Modulus and Poisson ratio.

D) the quantity and quality of the data.

- The optimum energy factor is influenced by

A) the mining method

i) direction of blasting in relation to the major discontinuities.

ii) presence or absence of free faces.

B) the blast design parameters

i) ratios of hole diameter, bench height, burden, spacing, subdrilling.

ii) delays and initiation sequence.

iii) explosive properties.

C) the rock mass properties

i) structural: micro and macro scale.

ii) failure behaviour: uniaxial compressive strength, Young's Modulus and Poisson ratio.

2) In the field, the following procedure is suggested:

A) Define blasting domains.
b) Standardize blasting procedures and define the optimum design powder factor.

c) Average RQI data for the last three benches in every domain. Consider changes in drilling practice and equipment. Encourage drillers to pay attention to their reports.

d) Plot RQI vs powder factor. Look at the possibility of further optimization of the powder factor according to the relationship.

3) Mine operators that possess good drilling and blasting records have an advantage. They should understand the influence of optimum blasting practice on the operation and slope stability. Their familiarity with the design powder factor approach is a good asset.

Further investigation on the methods of comparison of RQI values obtained from different operations is needed in order to establish a basic or general relationship. The development of a computerized drill performance recorder and their use on blasthole drill rigs would simplify the establishment of a large data base. If such a data base is already available, it would be interesting to perform simple geostatistic routines in order to obtain other correlations between the Rock Quality Index and the rock mass properties. Moreover, on a long term basis, the use of moving average techniques could permit to evaluation of
the trends in Rock Quality Index and predict the optimum design powder factor that would be required on future benches.
REFERENCES


2. ASHBY, J.P.; Production Blasting and the Development of Open Pit Slopes, Proceedings, 3rd International Conference on Stability in Surface Mining, Brawner editor, AIME, 1982

3. SCARTACCINI, T.E.; Blast Hole Drill Recorder Utilization, Proceedings, Fall Meeting of the Society of Mining Engineers of AIME, St. Louis, Missouri, 1970

4. HAGAN, T.N.; Reid, I.W.; Performance Monitoring of Production Blasthole Drills - A Means of Increasing Blasting Efficiency, Proceedings, 2nd Surface Mining and Quarrying Symposium, Bristol, England, October 1983


6. Manuel d'Information: Vibralog, Published by Jean Lutz, S.A. (France). Available (in English) from Solroc Consultants Inc, Saint-Laurent, P.Q. H4T 1E3

7. CYR, J.B.; et al; Geology and Mineralization at Equity Silver Mine, Houston, B.C., Proceedings, 8th CIM District Six Meeting, Smithers, B.C., 1983

9. ROTHERHAM, D.C.; Geology and Ore Reserves of the Sam Goosly Silver-Copper Deposits Volume I, 1979, unpublished report

10. WALDNER, G.D.; et al; Lornex, Porphyry Deposits of the Canadian Cordillera, paper 13, CIM special volume no. 15, 1976


12. PAQUETTE, C.; Personal Communication, August, 1983


14. BURKE, J.R.; Personal Communication, June, 1984

15. RAYMOND, G.; Ore Estimation Problems in an Erratically Mineralized Orebody, CIM Bulletin, June, 1979
6.0 CONCLUSIONS

The excavation of a rock slope using drilling and blasting methods is an engineering task. The influence of the blast results on the slope stability and the total operating costs is such that efforts have to be made to reach the optimum blast design. However, no rock blasting theory can take into account all the variables included in the design process of an open pit blast. Consequently, the optimization process is a very difficult task. The Rock Quality Index is a simple rock mass classification system that reflects the rock mass properties as a whole. Obtained from the blasthole rotary drills, it is practical and not expensive.

Correlation between the Rock Quality Index and the design powder factor can be established easily when the right procedure is followed. First, the powder factor has to be the only blast design variable left, the others being held constant within the pit. Secondly, since the RQI is also dependent upon the drilling practice and the drilling equipment, these factors must be maintained constant. Practically, this is not possible and therefore, a quantity of accurate drilling data is needed and averaged. At this point, the only parameters left are the rock mass properties. As the interaction of the rock mass properties influence the blasting results, the same properties affect the drill performance. Consequently, an empirical relationship can be defined. It also appears that a correlation between Rock
Quality Index and grindability can also be determined. Based on two years of research on the subject, the following conclusions are presented:

1) A correlation between the Rock Quality Index and the optimum design powder factor exists. At this point, correlations are defined for the different drilling and blasting practices, as long as there are maintained constant throughout the pit.

2) It appears that a correlation can be established with the rock mass grindability although the grinding rate is also a function of the size of the fragments in the muck pile.

3) Accurate monitoring of the drilling parameters is needed. The downpressure readings should be recorded in units of applied weight.

4) Considerable data are needed in order to be representative of the rock mass properties within each domain.

5) Groundwater and caving blastholes influence the Rock Quality Index.

6) Rock Quality Index data could be stored within the geological computer model to facilitate analysis.

7) Drill performance recorders are a useful drilling tool. However, when used in conjunction with RQI data gathering programs, recorded charts should be replaced by digital information.
8) Even with improved monitoring, a hole per hole loading approach may not be practical.

9) Blasting procedures:
   i) The RQI approach to rock mass characterization has to be made in conjunction with the design powder factor approach to blast engineering.
   ii) Controlled blasting commences with production blasts.
   iii) Sequential blasting techniques improve slope stability.
   iv) Choked blasts decrease the slope stability and the blast performance.
   v) Blast evaluations are also part of the blast engineer's task.
BIBLIOGRAPHY
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2. ASHBY, J.P.; Production Blasting and Development of Open Pit Slopes, Proceedings, 3rd International Conference on Stability in Surface Mining, Brawner Editor, AIME, 1982


4. Automatic Recorder of Drilling Parameters, General Presentation, Published by Jean Lutz, S.A., France

5. BAUER, A.; Drilling and Blasting Courses Notes, Queen's University, Kingston, Unpublished


7. BAUER, A.; Personal Communication, 1983


10. BELLAND, J.M.; Structure as a Control in Rock Fragmentation, C.I.M. Bulletin, March 1966

12. Blasthole Bit Technology for the 80's, Published by Dresser Industries Inc.

13. BLINDHEIM, O.T.; Drillability Predictions in Hard Rock Tunnelling, Tunnelling '79, IMM, 1979


16. BROADBENT, C.; Predictable Blasting with In-Situ Seismic Surveys, Mining Engineering, SME, April 1974

17. BUCHTA, L.; Open Pit Blasting - How to Improve results, World Mining, June 1982

18. BURKE, J.R.; Personal Communication, June 1984

19. CALDER, P.N.; Pit Slope Manual Chapter 7 - Perimeter Blasting, CANMET, Report 77-14, pp. 82, May 1971

20. CHEATHAM, J.B.; GNIRK, P.F.; The Mechanics of Rock Failure Associated with Drilling at Depth, Failure and Breakage of Rocks, Ch. 17, AIME, 1967

Int. Soc. Rock Mech., Belgrade, 1970

22. CLAUSING, D.P.; Comparison of Griffith's Theory with Mohr's Failure Criteria, 3rd Symposium on Rock Mechanics, Quarterly of the Colorado School of Mines, N3, July 1959


25. COHEN, C.J.; Bench Blasting Related to Joint Design, World Mining, April 1980


27. CYR, J.B.; et al; Geology and Mineralization at Equity Silver Mine, Houston, B.C., Proceedings, 8th CIM District Six Meeting, Smithers, B.C., 1983


29. DAREING, D.W.; RADZIMOUSKY, E.I.; Effect of Dynamic Bit Forces on Bit Bearing Life, Society of Petroleum
Engineering Journal, December 1965

30. DICK, R.A.; Factors in Selecting and Applying Commercial Explosives and Blasting Agents, Inf. Circ. 8405, USMB, 1968

31. DRURY, F.C.; WESTMAAS, D.J.; Considerations Affecting the Selection and Use of Modern Chemical Explosive, Proceedings of the 4th conference on blasting and explosive techniques, Society of Explosives Engineers, Louisiana, 1978


33. FAIRHURST, C.; CORNET, F.; Rock Fracture and Fragmentation, 22nd U.S. Symposium on Rock Mechanics, Massachusetts Institute of Technology, Cambridge, Massachusetts, July 1981


35. FISH, B.G.; The Basic Variables in Rotary Drilling, Mine and Quarry Engineering, V27, N1 and N2, 1961

36. GNIRK, P.F.; CHEATHAM, J.B.; An Experimental Study of Single Bit-Tooth Penetration into Dry Rock at Confining Pressures 0 to 5000 psi, Society of Petroleum Engineers Journal, June 1965

37. GNIRK, P.F.; CHEATHAM, J.B.; A Theoretical Description of


40. HAGAN, T.N.; The Effects of Blast Geometry and Initiation Sequence on Blasting Results, Chapter 6 of the Australian Mineral Foundation Workshop Course Manual, Drilling and Blasting Technology, Adelaide, 1977

41. HAGAN, T.N.; HARRIES, G.; The Effects of Rock Properties on Blasting Results, Chapter 4 of the Australian Mineral Foundation Workshop Course Manual, Drilling and Blasting Technology, Adelaide, 1977

42. HAGAN, T.N.; REID, I.W.; Performance Monitoring of Production Blasthole Drills - A Means of Increasing Blasting Efficiency, Proceedings, 2nd Surface Mining and Quarrying Symposium, Bristol, England, October 1983

44. HEINEN, R.H.; DIMOCK, R.R.; The Use of Seismic Measurements to Determine the Blastability of Rock, Proceedings of the 2nd Conference on Explosive and Blasting Technique, Society of Explosives Engineers, Louisville, 1976


49. KHALAF, F.; ABOUZEID, A.Z.M.; Analogy Between Indentation and Blasting Tests on Brittle Rocks, 21st U.S. Symposium on Rock Mechanics, University of Missouri-Rolla, May 1980

50. KONYA, C.J.; DAVIS, J.; The Effect of Stemming Consist on Retention in Blastholes, Proceedings of the 4th conference on blasting and explosive techniques, Society of Explosives Engineers, Louisiana, 1978


52. LANG, L.C.; New Permafrost Blasting Methods Developed at
53. LANG, L.C.; **Buffer Blasting Techniques in Open Pit Mines**, 2nd Open-Pit Operators Conference and Third Annual Meeting of CIM District Six, October 1978


57. LEIGHTON, J.C.; **Predicting Powder Factors from Rotary Drill Performance for Controlled Blasting**, 14th Canadian Rock Mechanics Symposium, Vancouver, 1982


61. MANON, J.J.; **How to Select an Explosive or Blasting Agent**
for a Specific Job, Engineering and Mining Journal, May 1977

62. Manuel d'Information: Vibralog, Published by Jean Lutz, S.A. (France). Available (in English) from Solroc Consultants Inc, Saint-Laurent, P.Q. H4T 1E3

63. MATHIS, C.; Proposal of a Report on Rock Quality Index Based on Rotary Drill Performances, Unpublished paper, University of Alberta, March 1975

64. MAURER, W.C.; The State of Rock Mechanics Knowledge in Drilling, Failure and Breakage of Rocks, Ch. 15, AIME, 1967


66. MEDLOCK, J.D.; Laboratory Testing of Rotary Rock Bits, Drilling and Blasting Symposium, Quarterly of Colorado School of Mines, V56, N1, January 1961


68. MERCER, J.K.; HAGAN, T.N.; Progress Towards Optimum Blasting - A Key to Increased Productivity and Profitability, Commonwealth Mining and Metallurgical Congress, Hong Kong, 1978


71. MORRELL, R.S.; UNGER, H.F.; Drilling Machines, Surface, SME Handbook Sect. 11-4, Vol. 1, Cummins and Given, editors, AIME, 1973

72. MORRIS, R.I.; Rock Drillability Related to a Rollercone Bit, Dresser Industries Inc.

73. MORRISON, R.G.K.; A Philosophy of Ground Control, Dept. of Mining and Metallurgical Engineering, McGill University, Montreal, 1976

74. MUNOZ-CASAYUS, J.M.; Approche Technique de la Determination des les Mines a Ciel Ouvert, 3eme Session d'Etude sur les Techniques de Sautage, Laval University, 1980


76. PAGE, C.H.; Blasting and Comminution, Unpublished report, Steffen, Robertson and Kirsten

77. PAQUETTE, C.; Personal Communication, August, 1983


80. PEASE, R.B.; Geological mapping, 1982 Field Season, Equity Silver Mine Property, Unpublished report

81. PEARSE, G.E.; Rock Blasting, Some Aspect of the Theory and Practice, Mine and Quarry Engineering, January 1955

82. PEARSSON, P.A.; Bench Drilling in Open Pit Mines, World Mining, August 1975


84. RATIGAN, J.L.; An Examination of the Tensile Strength of Brittle Rock, 23rd U.S. Symposium on Rock Mechanics, Goodman editor, AIME, 1982

85. RAYMOND, G.; Ore Estimation Problems in an Erratically Mineralized Orebody, CIM Bulletin, June 1979


89. _Rotary Drilling Bits_, Published by Hughes Tool company, Houston, Texas

90. ROTHERHAM, D.C.; _Geology and Ore Reserves of the Sam Goosly Silver-Copper Deposits Volume 1_, unpublished report, 1979

91. SCARTACCINI, T.E.; _Blast Hole Drill Recorder Utilization_, Proceedings, Fall Meeting of the Society of Mining Engineers of AIME, St. Louis, Missouri, 1970

92. _Seminar on Applied Explosive Technology (Dupont)_ , Presented to Lornex Mining Corp., June 1983


96. STEINKE, S.C.; _Blasthole Bit Evaluation_, Published by Smith-Gruner Company

98. TANDANAND, S.; UNGER, F.H.; *Drillability Determination - A Drillability Index for Percussion Drills*, USBM, report 8073, 1975


102. WALDNER, G.D.; et al; *Lornex*, Porphyry Deposits of the Canadian Cordillera, paper 13, CIM Special Volume no. 15, 1975


104. WHITE, C.G.; *A Rock Drillability Index*, Quarterly of the Colorado School of Mines, V 64, N 2, 1969

105. WILLIAMSON, T.N.; *Rotary Drilling*, Surface Mining, Sect. 6.3, Pfleider editor, AIME, 1972
106. WINZER, S.R.; RITTER, A.P.; The Role of Stress Waves and Discontinuities in Rock Fragmentation: A Study of Fragmentation in Large Limestone Blocks, 21st U.S. Symposium on Rock Mechanics, University of Missouri-Rolla, 1980

107. YOUNG, C.; Rock Fragmentation - Needs and Possibilities, Rock Fragmentation session of the 17th U.S. Symposium on Rock Mechanics, University of Utah, Utah Engineering Experiment Station, 1976
APPENDIX I
## APPENDIX I

### BIT COMPARISON

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<tr>
<th>Soft rock</th>
<th>Hughes</th>
<th>HH33</th>
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<td>Security</td>
<td>S8M</td>
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<td>Smith</td>
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<td>M62</td>
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<th>Medium-hard rock</th>
<th>Hughes</th>
<th>HH77</th>
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<td></td>
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<td>M73 - M74</td>
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<td></td>
<td>Security</td>
<td>H8M</td>
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<tr>
<td></td>
<td>Smith</td>
<td>Q7</td>
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<td>Varel</td>
<td>QMCS</td>
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<th>Hard rock</th>
<th>Hughes</th>
<th>HH99</th>
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<td>M83</td>
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<td>H10M</td>
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<td></td>
<td>Smith</td>
<td>Q9</td>
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<tr>
<td></td>
<td>Varel</td>
<td>QMCH</td>
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APPENDIX II
### APPENDIX II

#### RELATIONSHIP BETWEEN DOWN PRESSURE AND APPLIED WEIGHT

<table>
<thead>
<tr>
<th>MANUFACTURER AND MODEL</th>
<th>MAXIMUM PULLDOWN HYDRAULIC PRESSURE</th>
<th>WEIGHT RATIO FACTOR</th>
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<tbody>
<tr>
<td>Bucyrus Erie 30-R</td>
<td>30,000 lbs @ 1,000 lbs. gage pressure</td>
<td>Series 45 motor 21 x gage pressure + 3,800 lbs.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Series 60 motor 27 x gage pressure + 3,800 lbs.</td>
</tr>
<tr>
<td>Bucyrus Erie 40-R</td>
<td>50,000 lbs.</td>
<td>Series 45 motor 57 x gage pressure + 7,000 lbs.</td>
</tr>
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<td></td>
<td></td>
<td>Series 60 motor 72 x gage pressure + 7,000 lbs.</td>
</tr>
<tr>
<td>Bucyrus Erie 45-R</td>
<td>70,000 lbs. @ 1,300 lbs. gage pressure</td>
<td>45 x gage pressure + 10,000 lbs.</td>
</tr>
<tr>
<td>Bucyrus Erie 50-4</td>
<td>75,000 lbs.</td>
<td>Series 45 motor 49 x gage pressure + 9,000 lbs., 60 motor 62 x gage pressure + 9,000 lbs., 75 motor 82 x gage pressure + 9,000 lbs.</td>
</tr>
<tr>
<td>Bucyrus Erie 60-R &amp; Bucyrus Erie 61-R</td>
<td>110,000 lbs. @ 1,200 lbs. gage pressure</td>
<td>80 x gage pressure + 14,000 lbs.</td>
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<td>Gardner-Denver GD-60</td>
<td>60,000 lbs.</td>
<td>The actual weight on the bit can be read from the hydraulic gage on the GD-60, GD-80, GD-120 and the GD-130.</td>
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<tr>
<td>Gardner-Denver GD-80</td>
<td>80,000 lbs.</td>
<td>Same as GD-60</td>
</tr>
<tr>
<td>Gardner-Denver GD-120</td>
<td>120,000 lbs.</td>
<td>Same as GD-60</td>
</tr>
<tr>
<td>Gardner-Denver GD-130</td>
<td>130,000 lbs.</td>
<td>Same as GD-60</td>
</tr>
<tr>
<td>Machine</td>
<td>Weight @ Pressure</td>
<td>Hydraulic Gage Indication</td>
</tr>
<tr>
<td>-------------</td>
<td>-------------------</td>
<td>---------------------------</td>
</tr>
<tr>
<td>Marion M-4</td>
<td>105,000 lbs. @ 3,500 lbs. gage pressure</td>
<td>24.3 x gage pressure + 20,000 lbs. Hydraulic gage indicates actual weight on the bit</td>
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<tr>
<td>Marion M-5</td>
<td>120,000 lbs. @ 3,500 lbs gage pressure</td>
<td>28.6 x gage pressure + 20,000 lbs. Hydraulic gage indicates actual weight on the bit</td>
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<td>Chicago Pneumatic T-650</td>
<td>30,000 lbs. @ 2,300 lbs. gage pressure</td>
<td>13. x gage pressure</td>
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<tr>
<td>Chicago Pneumatic T-750</td>
<td>50,000 lbs. @ 2,300 lbs. gage pressure</td>
<td>21.75 x gage pressure</td>
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<td>Chicago Pneumatic C-850</td>
<td>60,000 lbs. @ 2,300 lbs. gage pressure</td>
<td>26 x gage pressure</td>
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<tr>
<td>Chicago Pneumatic C-950</td>
<td>90,000 lbs. @ 2,700 lbs. gage pressure</td>
<td>33 x gage pressure</td>
</tr>
<tr>
<td>Chicago Pneumatic C-975</td>
<td>100,000 lbs. @ 3,200 lbs. gage pressure</td>
<td>31.25 x gage pressure</td>
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<td>Robbins RR 10-S</td>
<td>65,000 lbs.</td>
<td>30.6 x gage pressure + 6,000 lbs.</td>
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<tr>
<td>Robbins RR 11</td>
<td>70,000 lbs.</td>
<td>30.6 x gage pressure + 7,500 lbs.</td>
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<tr>
<td>Robbins RRT-50</td>
<td>50,000 lbs.</td>
<td>30.6 x gage pressure + 5,000 lbs.</td>
</tr>
<tr>
<td>Robbins RRT-60</td>
<td>60,000 lbs.</td>
<td>30.6 x gage pressure + 6,000 lbs.</td>
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<tr>
<td>Robbins RRT-70</td>
<td>70,000 lbs.</td>
<td>30.6 x gage pressure + 8,000 lbs.</td>
</tr>
<tr>
<td>Equipment</td>
<td>Weight (lbs) @ Pressure (gage)</td>
<td>Multiplication Factor</td>
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<tr>
<td>-----------------</td>
<td>--------------------------------</td>
<td>-----------------------</td>
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<tr>
<td>Robbins H100</td>
<td>80,000 lbs.</td>
<td>41.28 x gage pressure + 3,000 lbs.</td>
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<tr>
<td>Schramm T985H</td>
<td>38,000 lbs. @ 2,000 lbs. gage pressure</td>
<td>19. x gage pressure</td>
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<tr>
<td>Schramm T64HB</td>
<td>30,000 lbs. @ 2,000 lbs. gage pressure</td>
<td>15. x gage pressure</td>
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<td>Schramm C985H</td>
<td>50,000 lbs. @ 2,500 lbs. gage pressure</td>
<td>20 x gage pressure</td>
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<tr>
<td>Schramm C9120</td>
<td>50,000 lbs. @ 2,500 lbs. gage pressure</td>
<td>20 x gage pressure</td>
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</table>
APPENDIX III
APPENDIX III
POSSIBLE BENEFITS FROM IMPROVED FRAGMENTATION

- Increased shovel capacity
- Reduced shovel maintenance:
  -- partly due to longer bucket and teeth life because the shovel would not be abused in digging oversized boulders on hard bottoms
- Increased truck capacity (reduced loading time)
- Reduced truck maintenance:
  -- partly due to finer material flows better
  -- partly due to large oversize boulders not striking the body
- Increased crusher capacity
- Reduced crusher maintenance
- Increased mill capacity
- Decreased or eliminated secondary breaking cost
- Decreased pit clean-up costs
- Decreased truck tire wear:
  -- partly because finer materials provide better travel surfaces
- Decreased vehicle cost from better pit floors (pick-ups, maintenance trucks, etc.)
APPENDIX IV
APPENDIX IV

REVIEW OF T.E. LITTLE REPORT ON RQI

Shortly after being proposed, the concept of Rock Quality Index was investigated in regard with its applicability to the design of open pit slope by Little. RQI values were contoured and attempts were made to correlate contour patterns with mechanical and physical properties of the rock. The studies were carried out at Endako, Kaiser and Gibraltar mines during the summer of 1975. It was found that several independent variables can have an influence on RQI values.

1. Conclusions of the Endako Mine field work:

i) Recorded drilling data:
Little observed that the accuracy of calculated values of RQI is totally dependant on the accuracy of recorded data on drill bit performance records (driller logs). It was noted that the RQI contours tend to relate with the movements of the drills. Drill bit performance records show that for many work shifts, the same drilling parameters have been allocated to all holes drilled in one shift. In addition, drilling time was recorded to the nearest five minutes and the hydraulic down pressure to the nearest 50 to 100 psi.

ii) Drill type and size:
At Endako, Bucyrus BE 40-R drilling 9 inches diameter holes and Marion M-4 drilling 9 7/8 inches diameter holes were used.
A comparison of adjacent holes drilled with the different drilling equipment shows that the RQI (9") was equal to 0.6 - 0.7 RQI (9 7/8"). It was concluded that the RQI varies significantly with the type of drill used.

iii) Lithology and structural geology:

Generally, higher RQI values occurred in the vicinity of dyke characterized as a highly fractured rock mass, but have a high compressive strength. Lower values were encountered in a fault zone area where the rock is generally well fractured, altered and has a low compressive strength. It was concluded that the RQI appeared to be related to the rock strength which may or may not correlate with the lithology or structure. Little also tried to relate rock hardness and degree of fracturing with RQI without success.

II. Conclusion of the Kaiser Mine field work:

i) Recorded drilling data:

It was observed that high variations in the hydraulic down pressure and rotary speed are due to the presence of alternating beds of sandstone and siltstone above the coal seam. In addition, the drilling time for holes deeper than 40 feet included the time required to add a second piece of drill steel. This practice reduces the real penetration rate and therefore increased the RQI.
ii) Bit design and size:

Extremely high values of RQI were obtained in areas where steel tooth bits were used. The RQI for steel tooth bits ranged from 1.5 to 8.0 times the RQI for insert bits. Little also observed that, even though all blastholes were drilled by the same model of drill, the RQI values obtained for the 9 7/8 inch diameter bits did not differ significantly from those obtained with the 12 1/4 inches diameter bits.

iii) Lithology and structural geology:

Areas of high RQI were correlated with zones of hard sandstones. In addition, a general trend of the RQI subparallel with the structural geological strike was noted. Low RQI values were also encountered in faulted areas.

III. Conclusions from the Gibraltar Mine field work

i) Recorded drilling data:

At Gabraltar, the hydraulic down pressure was recorded as a standard range (ie., 500 - 100 psi) while the actual value was probably around 650 psi.

ii) Drill type and size:

As for Endako Mine, two models of drill were used: BE 45R (9 7/8") and Marion M4 (12 1/4") Comparison of RQI values showed that the RQI (9 7/8) was equal to 0.5 to 0.7 times the RQI (12 1/4).
iii) Lithology and structural geology:

High RQI values were recorded in areas of low percentage of broken and faulted rock. In areas where the intensity of fracturation increased, the RQI values were generally lower, indicating a relation between RQI and the degree of fracturing of the rock.

Summary of Little's Conclusions

1 - interpretations are limited by the reliability of the data input.

- some times, the RQI is related to drill movement more than to any other factors.

- averaging techniques compensated for the inaccurate data, but these techniques masked the real changes in rock quality.

2 - RQI is primarily related to rock strength, however, faulted areas and intensely fractured rocks showed a low RQI.

- there is no direct relationship between RQI and geology.

3 - larger bits on larger drills produce higher RQI (note that the drill models were different).

- different size bits on the same type of drill produced similar RQI values.

- steel tooth bits produced RQI values up to 8.0 times as high as those for tungsten carbide insert bits.
APPENDIX V
MODIFICATION OF BLASING METHODS

When it is evident that unsatisfactory results are being obtained from a particular blasting method and that the method should be modified, the blasting engineer may have to embark on a series of trails in order to arrive at an optimum design. As with any trials, careful documentation of each blast is essential and, whenever possible, only one variable at a time should be changed. The following sequence of test work is an illustration of the type of experiment which would be carried out to evaluate the cost effectiveness of using a higher energy explosive. Similar test sequences could be carried for each of the other factors which are relevant in a particular situation.

Rationalization:

a. Document present powder factors on an equivalent energy basis using the weight strengths of various explosives compared to that of the explosive in current use. Weight strength data should be obtained from the explosives manufacturer if these are not already available.

Evaluation:

b. For a blast with the explosive currently in use, document the behaviour of the blast during initiation and the condition of the resulting muck pile.
c. Document rate and conditions of digging.

d. Document fragmentation based upon the ratio of oversized material requiring secondary blasting to the total blast tonnage.

e. Document drilling and blasting costs.

Experimentation:

f. Select a similar area of ground and carry out a blast with a higher powder factor which is obtained by using a higher energy explosive e.g. by increasing the aluminum content of a slurry.

Evaluation:

g. Document the results as for steps b to e.

h. Carry out a cost-benefit study.

i. Repeat the experiment before preparing a statement to management suggesting a modification or a retention of the existing method.

It should also be noted that it is now possible to simulate blast performance on a computer. This approach eliminates a long trial and error process and permits one to narrow the field of possible modifications in regard to a target objective (i.e., reduction of flyrocks, toes, etc.). The most publicized system in North America is available through R.F. Favreau of the Royal Military College in St.-Jean, Quebec.