DEVELOPMENT OF A CORRELATION BETWEEN
ROTARY DRILL PERFORMANCE AND
CONTROLLED BLASTING POWDER FACTORS

by

JOHN CHARLES LEIGHTON
B.A.Sc., The University of British Columbia, 1978

A THESIS SUBMITTED IN PARTIAL FULFILMENT OF
THE REQUIREMENTS FOR THE DEGREE OF
MASTER OF APPLIED SCIENCE

in

THE FACULTY OF GRADUATE STUDIES
Department of Mining and Mineral Process Engineering

We accept this thesis as conforming
to the required standard

THE UNIVERSITY OF BRITISH COLUMBIA
August 1982

© John Charles Leighton, 1982
In presenting this thesis in partial fulfilment of the requirements for an advanced degree at the University of British Columbia, I agree that the Library shall make it freely available for reference and study. I further agree that permission for extensive copying of this thesis for scholarly purposes may be granted by the head of my department or by his or her representatives. It is understood that copying or publication of this thesis for financial gain shall not be allowed without my written permission.

John Charles Leighton

Department of Mining and Mineral Process Engineering

The University of British Columbia
1956 Main Mall
Vancouver, Canada
V6T 1Y3

Date October 2, 1982
Despite the availability of established, sophisticated methods for planning and designing stable slopes in rock, comparatively little attention is usually paid to the problems of carrying out the excavation. Blasting should be carefully planned to obtain optimum fragmentation as well as steep, stable pit walls for a minimum stripping ratio. The principal difficulty facing a blast designer is the lack of prior information about the many critical blasting characteristics of the rock mass. The common practice of trial-and-error blasting will eventually lead to a suitable design, but this must be repeated time after time in variable geology. This frequently results in many blast damaged slopes with decreased stability and increased safety hazards.

For this research project, an extensive study was undertaken to develop a concise background knowledge on state-of-the-art blasting technology. A field research program at Afton Mine examined the relationship between characteristic rock mass features and blast performance for application in optimal blasting design. Due to the complex inter-relationships of the many rock mass properties, the development of a comprehensive rock blasting model is not feasible. A practical approach to the problem was achieved by classifying each rock type with a single Rock Quality Index value which can be obtained from monitoring the performance of a rotary blasthole drill. A series of controlled blasting tests revealed a strong correlation between the Rock Quality Index and Powder Factor values over a broad range of geological conditions. The correlation was found to be sufficiently reliable to enable the prediction of optimum Powder Factors for perimeter blasts in previously untested rock types.

This Rock Quality Index and Powder Factor correlation provides a practical
approach to solving the problems of site specific blasting design. Without
the costs of additional equipment or specially trained personnel, the drilling
can provide a continual supply of data, reflecting changes in the rock mass
and permitting the selection of an economical Powder Factor.

The ultimate goals of this simple correlation system are optimization
of fragmentation, elimination of unacceptable blast damage, preservation of
inherent rock strength, and maximization of slope stability.
# TABLE OF CONTENTS

<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>ABSTRACT</td>
<td>ii</td>
</tr>
<tr>
<td>TABLE OF CONTENTS</td>
<td>iv</td>
</tr>
<tr>
<td>LIST OF TABLES</td>
<td>viii</td>
</tr>
<tr>
<td>LIST OF FIGURES</td>
<td>ix</td>
</tr>
<tr>
<td>LIST OF PLATES</td>
<td>xi</td>
</tr>
<tr>
<td>ACKNOWLEDGEMENTS</td>
<td>xiii</td>
</tr>
<tr>
<td>CHAPTER 1 INTRODUCTION</td>
<td>1</td>
</tr>
<tr>
<td>1.1 References</td>
<td>12</td>
</tr>
<tr>
<td>CHAPTER 2 EXPLOSIVES AND BLASTING MATERIALS</td>
<td>13</td>
</tr>
<tr>
<td>2.1 Properties of Explosives</td>
<td>19</td>
</tr>
<tr>
<td>2.1.1 Strength</td>
<td>19</td>
</tr>
<tr>
<td>2.1.2 Velocity of Detonation</td>
<td>20</td>
</tr>
<tr>
<td>2.1.3 Power</td>
<td>22</td>
</tr>
<tr>
<td>2.1.4 Brisance</td>
<td>23</td>
</tr>
<tr>
<td>2.1.5 Sensitivity</td>
<td>23</td>
</tr>
<tr>
<td>2.1.6 Density</td>
<td>24</td>
</tr>
<tr>
<td>2.1.7 Detonation Pressure</td>
<td>26</td>
</tr>
<tr>
<td>2.1.8 Water Resistance</td>
<td>29</td>
</tr>
<tr>
<td>2.1.9 Considerations for Selecting an Explosive</td>
<td>29</td>
</tr>
<tr>
<td>2.2 Explosives Used on this Research Project</td>
<td>32</td>
</tr>
<tr>
<td>2.2.1 Ammonium Nitrate / Fuel Oil (AN/FO)</td>
<td>32</td>
</tr>
<tr>
<td>2.2.2 Slurries</td>
<td>38</td>
</tr>
<tr>
<td>2.2.3 Primers</td>
<td>39</td>
</tr>
<tr>
<td>2.2.4 Detonating Cord</td>
<td>40</td>
</tr>
<tr>
<td>2.2.5 Electric Blasting Caps</td>
<td>41</td>
</tr>
<tr>
<td>2.2.6 Blasting Machine</td>
<td>41</td>
</tr>
</tbody>
</table>
2.2.7 Plastic Borehole Liner 41

2.3 References 43

CHAPTER 3 ROCK MASS DETONICS 44

3.1 The Breakage Process in Homogeneous Rock 48

3.1.1 Bench Blasting 48

3.1.2 Crater Blasting 59

3.1.3 Pre-Shear Blasting 60

3.2 The Influence of Rock Mass Properties on the Rock Breakage Process 66

3.2.1 In-Situ Dynamic Rock Strength 67

3.2.2 Presence of Structural Features 69

3.2.3 Poisson's Ratio 74

3.2.4 Young's Modulus 76

3.2.5 Internal Friction 76

3.2.6 Rock Density 77

3.2.7 Seismic Wave Velocity 77

3.2.8 Water Content 79

3.2.9 In-Situ Stress 79

3.3 References 82

CHAPTER 4 THE INFLUENCE OF BLASTING ON SLOPE STABILITY 84

4.1 Effect of Physical Blasting Forces 87

4.2 Effect of Ground Vibrations 93

4.3 References 103

CHAPTER 5 DESIGN CONSIDERATIONS FOR CONTROLLED BLASTING 104

5.1 Selecting a Controlled Blasting Technique 107

5.1.1 Pre-Shear Blasting 108

5.1.2 Cushion Blasting 111

5.1.3 Buffer Blasting 113
<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>5.2 Blasting Design Parameters</td>
<td></td>
</tr>
<tr>
<td>5.2.1 Blasthole Diameter</td>
<td>115</td>
</tr>
<tr>
<td>5.2.2 Explosive Type</td>
<td>118</td>
</tr>
<tr>
<td>5.2.3 Sub-Drill Depth</td>
<td>118</td>
</tr>
<tr>
<td>5.2.4 Stemming</td>
<td>119</td>
</tr>
<tr>
<td>5.2.5 Optimum Charge</td>
<td>121</td>
</tr>
<tr>
<td>5.2.6 Powder Factor and Burden Volume</td>
<td>122</td>
</tr>
<tr>
<td>5.2.7 Blasthole Pattern</td>
<td>122</td>
</tr>
<tr>
<td>5.2.8 Front Row Considerations</td>
<td>123</td>
</tr>
<tr>
<td>5.2.9 Initiating and Firing Sequence</td>
<td>125</td>
</tr>
<tr>
<td>5.3 Blast Evaluation and Optimization</td>
<td></td>
</tr>
<tr>
<td>5.3.1 During Detonation</td>
<td>129</td>
</tr>
<tr>
<td>5.3.2 After Detonation</td>
<td>129</td>
</tr>
<tr>
<td>5.4 References</td>
<td>133</td>
</tr>
<tr>
<td>CHAPTER 6 EXPERIMENTAL STUDIES AND FIELD WORK</td>
<td></td>
</tr>
<tr>
<td>6.1 Background Information on Afton Mine</td>
<td></td>
</tr>
<tr>
<td>6.1.1 Regional Information</td>
<td>138</td>
</tr>
<tr>
<td>6.1.2 Historical Summary</td>
<td>141</td>
</tr>
<tr>
<td>6.1.3 Summary of Site Geology</td>
<td>143</td>
</tr>
<tr>
<td>6.2 Development of the Rock Quality Index</td>
<td></td>
</tr>
<tr>
<td>6.2.1 Mechanical Theory of Drill Performance</td>
<td>150</td>
</tr>
<tr>
<td>6.2.2 Establishing Rock Quality Index Values</td>
<td>154</td>
</tr>
<tr>
<td>6.3 Development of an Improved Perimeter Blast Design</td>
<td></td>
</tr>
<tr>
<td>6.3.1 Former Blasting Methods</td>
<td>162</td>
</tr>
<tr>
<td>6.3.2 Design of an Improved Perimeter Blasting Method</td>
<td>164</td>
</tr>
<tr>
<td>6.4 Development of the Powder Factor Correlation</td>
<td></td>
</tr>
<tr>
<td>6.4.1 Establishing Powder Factor Values</td>
<td>172</td>
</tr>
<tr>
<td>6.4.2 Data Analysis and Results</td>
<td>176</td>
</tr>
</tbody>
</table>
### LIST OF TABLES

<table>
<thead>
<tr>
<th>Table</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Blasthole Diameter, Volume and Cost</td>
<td>117</td>
</tr>
<tr>
<td>2</td>
<td>Climatic Data, Kamloops Airport</td>
<td>140</td>
</tr>
<tr>
<td>3</td>
<td>Description of Domains</td>
<td>148</td>
</tr>
<tr>
<td>4</td>
<td>Hardness of Rock Types Using Jennings' Hardness Classification</td>
<td>149</td>
</tr>
<tr>
<td>5</td>
<td>Drilling Machine Specifications</td>
<td>157</td>
</tr>
<tr>
<td>6</td>
<td>Rock Quality Index Values</td>
<td>160</td>
</tr>
<tr>
<td>7</td>
<td>Summary of RQI/Powder Factor Correlation</td>
<td>179</td>
</tr>
</tbody>
</table>
# LIST OF FIGURES

<table>
<thead>
<tr>
<th>Figure</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0-1</td>
<td>Effect of Pit Slope Angle on Volumes of Waste and Ore in a Hypothetical Open Pit</td>
<td>3</td>
</tr>
<tr>
<td>1.0-2</td>
<td>Effect of Pit Slope Angle on Stripping Ratio for the Hypothetical Open Pit of Figure 1.0-1</td>
<td>4</td>
</tr>
<tr>
<td>1.0-3</td>
<td>Effect of Fragmentation on Cost of Mining</td>
<td>6</td>
</tr>
<tr>
<td>2.0-1</td>
<td>General Classifications of Blasting Compounds</td>
<td>15</td>
</tr>
<tr>
<td>2.1-1</td>
<td>Test Methods for Determining Confined and Unconfined Velocities of Detonation</td>
<td>21</td>
</tr>
<tr>
<td>2.1-2</td>
<td>Relationship of Sensitivity to Specific Gravity and Charge Diameter</td>
<td>25</td>
</tr>
<tr>
<td>2.1-3</td>
<td>Detonation Pressure as a Function of Specific Gravity and Detonation Velocity</td>
<td>27</td>
</tr>
<tr>
<td>2.1-4</td>
<td>Factor N vs. Specific Gravity for Borehole Pressure Estimation</td>
<td>28</td>
</tr>
<tr>
<td>2.1-5</td>
<td>Some Relative Ingredients and Properties of Explosives</td>
<td>30</td>
</tr>
<tr>
<td>2.2-1</td>
<td>Confined Detonation Velocity of AN/FO vs. Borehole Diameter</td>
<td>34</td>
</tr>
<tr>
<td>2.2-2</td>
<td>Effect of Water Content on the Detonation Velocity of AN/FO</td>
<td>37</td>
</tr>
<tr>
<td>3.1-1</td>
<td>Comparison of Blasthole Configurations in Crater and Bench Blasting</td>
<td>49</td>
</tr>
<tr>
<td>3.1-2</td>
<td>Initial Radial Fracturing Mechanism on an Element of Rock</td>
<td>51</td>
</tr>
<tr>
<td>3.1-3</td>
<td>Plan View of Stage 1 of Rock Breakage Process</td>
<td>52</td>
</tr>
<tr>
<td>3.1-4</td>
<td>Blasthole Cross-Section in the Plane of a Radial Crack During Stage 1 of the Rock Breakage Process</td>
<td>53</td>
</tr>
<tr>
<td>3.1-5</td>
<td>Plan View of Stage 2 of Rock Breakage Process</td>
<td>56</td>
</tr>
<tr>
<td>3.1-6</td>
<td>Plan View of Stage 3 of Rock Breakage Process</td>
<td>58</td>
</tr>
<tr>
<td>3.1-7</td>
<td>Comparison Between Removed Volumes in Bench Blasting and Crater Blasting With Identical Charges and Burdens</td>
<td>61</td>
</tr>
<tr>
<td>3.1-8</td>
<td>Initiation of Pre-Shear Fracturing Mechanism on an Element of Rock Between Two Simultaneously Detonated Blastholes</td>
<td>63</td>
</tr>
<tr>
<td>3.1-9</td>
<td>Propagation of Pre-Shear Fracture Due to the Interaction of Two Shock Waves</td>
<td>64</td>
</tr>
<tr>
<td>3.2-1</td>
<td>Empirical Relationship Between Powder Factor, Fracture Frequency and Joint Shear Strength at Bougainville Copper</td>
<td>75</td>
</tr>
<tr>
<td>Figure</td>
<td>Description</td>
<td>Page</td>
</tr>
<tr>
<td>--------</td>
<td>-----------------------------------------------------------------------------</td>
<td>------</td>
</tr>
<tr>
<td>3.2-2</td>
<td>Empirical Relationship Between Powder Factor and In-Situ Seismic Velocity at Kennecott Copper</td>
<td>78</td>
</tr>
<tr>
<td>3.2-3</td>
<td>The Influence of an In-Situ Stress Field on a Pre-Shear Line With Various Blasthole Spacings</td>
<td>81</td>
</tr>
<tr>
<td>4.2-1</td>
<td>Peak Particle Velocity versus Scaled Distance</td>
<td>96</td>
</tr>
<tr>
<td>4.2-2</td>
<td>Particle Velocities and Damage Induced at Given Distances by Particular Charges</td>
<td>99</td>
</tr>
<tr>
<td>4.2-3</td>
<td>The Effect of Increasing the Number of Delays for Reducing Blasting Vibrations at Da Ye Mine, China</td>
<td>100</td>
</tr>
<tr>
<td>5.2-1</td>
<td>Bench Blasting Terminology and Design Parameters</td>
<td>116</td>
</tr>
<tr>
<td>5.2-2</td>
<td>Rock Breakage at the Bottom of a Blasthole due to Sub-Drilling</td>
<td>120</td>
</tr>
<tr>
<td>5.2-3</td>
<td>Various Patterns Commonly Used in Perimeter Blasting</td>
<td>124</td>
</tr>
<tr>
<td>5.2-4</td>
<td>Design Parameters for the Critical Front Row Burden</td>
<td>126</td>
</tr>
<tr>
<td>5.2-5</td>
<td>Various Firing Sequences Commonly Used in Perimeter Blasting</td>
<td>127</td>
</tr>
<tr>
<td>5.3-1</td>
<td>Characteristic Features of a Successful Perimeter Blast</td>
<td>130</td>
</tr>
<tr>
<td>6.1-1</td>
<td>Location Map for Afton Mine</td>
<td>139</td>
</tr>
<tr>
<td>6.1-2</td>
<td>Regional Geology at Afton Mine</td>
<td>142</td>
</tr>
<tr>
<td>6.1-3</td>
<td>Pit Plan Showing Domain Boundaries at Afton Mine (August 1981)</td>
<td>147</td>
</tr>
<tr>
<td>6.2-1</td>
<td>Rock Quality Index Values for Each Domain Ranked in Order of Increasing Quality</td>
<td>161</td>
</tr>
<tr>
<td>6.3-1</td>
<td>Former Perimeter Blast Pattern at Afton Mine</td>
<td>163</td>
</tr>
<tr>
<td>6.3-2</td>
<td>New Perimeter Blast Pattern at Afton Mine</td>
<td>166</td>
</tr>
<tr>
<td>6.3-3</td>
<td>Comparison of Drillhole Densities on Old and New Perimeter Blast Patterns</td>
<td>167</td>
</tr>
<tr>
<td>6.3-4</td>
<td>Plan of New Perimeter Blast Firing Sequence</td>
<td>169</td>
</tr>
<tr>
<td>6.4-1</td>
<td>Proposed Correlation Between Rock Quality Index and Powder Factor</td>
<td>178</td>
</tr>
</tbody>
</table>
LIST OF PLATES

Except where noted, all photographs were taken by the author.

<table>
<thead>
<tr>
<th>Plate</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Underblasting results in blocky, poorly fragmented rock and tight digging conditions. Broken shovel teeth, strained shovel cables, truck tire damage and increased loading times add significant delays and costs to the mining operation.</td>
<td>7</td>
</tr>
<tr>
<td>2</td>
<td>Overblasting can severely weaken rock slopes, requiring expensive and time consuming stabilization measures. Rockbolts, shotcrete, steel straps and mesh were used on the upper rock face in this photo, contrasting dramatically with the clean, well-blasted face below.</td>
<td>8</td>
</tr>
<tr>
<td>3</td>
<td>The C.I.L. truck transports and delivers bulk AN/FO safely and easily. On site, it mixes the fuel oil with the ammonium nitrate prills and ejects the finished mixture pneumatically through the black hose at the rear of the truck.</td>
<td>35</td>
</tr>
<tr>
<td>4</td>
<td>Loading AN/FO into a blasthole is a fast and easy procedure. This particular hole contained water, was pumped out and had a plastic liner inserted. Two primers are used to ensure detonation in wet holes, as indicated by the two Primacord downlines.</td>
<td>36</td>
</tr>
<tr>
<td>5</td>
<td>Plan view of a blasthole showing Stage 1 fracturing mechanisms in a homogeneous rock block as illustrated in Figure 3.1-3. Note the distorted blasthole surrounded by an intensely crushed zone and the radial fracture pattern.</td>
<td>54</td>
</tr>
<tr>
<td>6</td>
<td>View of fracture plane parallel to a single blasthole in a homogeneous rock block. Surface texture clearly indicates this to be a tensile fracture. Note radial nature of tensile fracture pattern as illustrated in Figure 3.1-4. Excessive fracture at toe of hole is probably due to location of the primer charge.</td>
<td>55</td>
</tr>
<tr>
<td>7</td>
<td>Pre-Shear blasting created a single fracture plane to produce this clean, smooth rock face along a highway. The presence of intact half-blastholes indicates a minimum of damage to the rock behind the pre-shear plane.</td>
<td>65</td>
</tr>
<tr>
<td>8</td>
<td>This exposed blasthole clearly reveals the effect of an intersecting joint. Note the shape of the crushed zone and the distinctly open joint caused by the pneumatic wedging of escaping high pressure explosion gases.</td>
<td>70</td>
</tr>
<tr>
<td>9</td>
<td>The effect of a major joint parallel and adjacent to the blasthole can be seen at the centre of this photo. The reflected tensile wave interacted with the joint to cause overbreak along the well defined joint plane. Premature gas release into the joint reduced the blasthole pressure such that a substantial portion of rock along the blasthole failed to break out.</td>
<td>72</td>
</tr>
</tbody>
</table>
The presence of non-shattered half-blastholes and several smooth shear faces indicate that this rock face was not overblasted. This photo illustrates that steep, outward dipping rock structure will ultimately control the final appearance of a rock face. In this case, the primary concern in the blast design is to keep such overbreak to a minimum.

Tension cracks outside the pit perimeter reveal the extent of the large pit wall failure initiated by careless blasting. Although triggered by gas pressure damage, continued movement was caused by a combination of high groundwater levels and blasting vibrations.

Cushion blasting was used to create this clean, low maintenance rock face in the very weak and highly fractured Kamloops Volcanic formation.

An aerial view of the Afton Mine site, November 1980.

View across Afton's pit looking west. The various geological units within the pit are clearly seen by the different coloured zones.

One of the two Bucyrus Erie 40-R drills used at Afton Mine.

All blastholes were drilled with 230 mm. (9 inch) diameter tri-cone bits with chisel-shaped tungsten carbide inserts.

The new perimeter blast design during detonation. As the pattern fires from left to right, the varying dust plume heights illustrate the effect of the delay sequence.

The test blast of Plate 17 produced this muck pile which illustrates the features of a successful blast. Note the evenness of throw, indicating a good front row charge.

A close up view of the muck pile toe area. Note the relatively uniform fragmentation and the lack of rock debris beyond the toe. The slope of this muck face is ideal.

A close up view on top of the muck pile. Note the slight rise of the evenly fragmented crest and the slight dip along the back of the berm. The area appears uniform without craters or humps.

A handful of free-flowing AN/FO. Safe handling is one of the important characteristics of a blasting agent.

A 25 kilogram bag of Hydromex T-3. This will be hand loaded into wet blastholes because of its excellent water resistance.

A Procore III Primer tied onto the end of a Reinforced Primacord downline is ready for lowering into a blasthole.
ACKNOWLEDGEMENTS

The author wishes to express his gratitude to his research supervisor, Mr. C.O. Brawner, for his encouragement, comments and guidance throughout this research project. Thanks are also extended to the other faculty members of the Department of Mining and Mineral Process Engineering for their general support and encouragement.

At Afton Operating Corporation, the author wishes to express his appreciation to Mike Lipkewich, Mine Manager, for his interest and willingness in permitting this research to be done at Afton Mine. Fred Mason, Pit Superintendent, and Hans Geertsema, General Pit Foreman, deserve special thanks for their comments and guidance throughout the development and testing of the new blasting designs. Valuable assistance in the geological work was provided by Doug Stewart, Chief Engineer; Allan Reed, Teck Exploration Engineer; and Ian Oliver, Pit Geologist.

The interest and patience of all the blasting crew members was greatly appreciated during the experimental stages. Their input with respect to practical problems encountered in the field contributed to the value of this thesis as a useful, working document. Thanks to: Mike Shields, Pat Stallard, Bill Kaczanowski, Dean Blanchard, Michael Godard, Barry McDonald, and Jim Seibal.

Financial support from the National Science and Engineering Research Council of Canada and from the Frederick Armand McDiarmid Scholarships was greatly appreciated.
CHAPTER ONE

INTRODUCTION
1.0 INTRODUCTION

Achieving a satisfactory level of slope stability is the most important consideration for a geotechnical engineer designing large scale surface excavations in rock. Unfortunately, rock is far from being an ideal engineering material, possessing highly complex characteristics which often change over very short distances. Different rock types can span a broad range of strengths, be highly fractured or broken, and can be intermixed with other substances such as clay gouge or hard infilling. Even the local environment can add further complications by the presence of water, local seismic activity, and weathering upon exposure.

The design of a rock slope must take into account the degree of stability that is necessary. In the field of civil engineering, rock cuts demand a high degree of stability along transportation corridors, in habitated areas, or in high cost installations such as hydro-power projects. Minor failures or rockfalls cannot be tolerated for the safety of the installations or the people working and travelling beneath the slopes. For this reason, a conservative approach is generally accepted in rock slopes designed for civil projects where the consequences of minor instability can justify the higher capital costs of construction.

However, in the open pit mining industry, there is the additional economic incentive to maintain the steepest stable slope angle for the minimum stripping ratio. Every degree of reduced pit wall angle represents a substantial amount of lost profit. This is more true than ever now that lower grade deposits are being developed and open pits are excavated to greater depths. See Figures 1.0-1 and 1.0-2. For these reasons, the mining industry cannot afford the higher costs of absolute slope stability. Pit slope design has evolved over the years to the point where a certain amount of instability (i.e. localized bench crest failure) is actually allowed for in order to achieve the maximum possible overall slope angle.
HYPOTHETICAL CIRCULAR OPEN PIT AND ORE BODY

FIGURE 1.0-1: EFFECT OF PIT SLOPE ANGLE ON VOLUMES OF WASTE AND ORE IN A HYPOTHETICAL OPEN PIT
(after Stewart and Kennedy)
FIGURE 1.0-2: EFFECT OF PIT SLOPE ANGLE ON STRIPPING RATIO FOR THE HYPOTHETICAL OPEN PIT OF FIGURE 1.0-1

(after Stewart and Kennedy)
Such design practices require detailed knowledge of the rock mass properties to be able to design within these closer tolerances. The geotechnical engineer will have to spend considerable effort to produce a safe, feasible and economic design. Much time will usually be spent on the field mapping, structural geologic studies, surface and subsurface hydrology, determination of strengths and effective friction angles of discontinuities, and finally, checking stability with the aid of mathematical and analytical models. Unfortunately, the engineer often fails to recognize the critical importance of problems involved in translating the design "on paper" to a slope "in rock". The great care taken for the design must be continued into the excavation stage if the optimum stable slope is to be realized.

Construction in rock usually requires blasting which, by its very nature, is a highly destructive force demanding careful control. Since blasting is the first step in the mining process, achieving successful fragmentation has an important effect on all the downstream ore handling and processing activities. Rock which is underblasted can add significant headaches and costs to a mining operation. See Figure 1.0-3 and Plate 1. At the same time, the final rock slope, having been designed to fairly close tolerances, must not be subjected to excessive forces which could alter the rock mass strength properties and lead to serious instability. Thus, overblasting will also cause extra problems and often result in very expensive and time consuming solutions to regain slope stability. See Plate 2. This problem of satisfying two apparently opposite tasks simultaneously is the dilemma faced by a blast designer. He must always strive to find the optimal point between fragmentation and stability where the operating costs are minimized.

Over the past 20 years, considerable progress has been made in the study of explosives, rock mass detonics, and blasting techniques. Much of this information was reviewed for this thesis and has been summarized in Chapters 2 through 5 to provide the background necessary for understanding rock blast-
FIGURE I.0-3: EFFECT OF FRAGMENTATION ON COST OF MINING

(after Hoek & Bray²)
Underblasting results in blocky, poorly fragmented rock and tight digging conditions. Broken shovel teeth, strained shovel cables, truck tire damage and increased loading times add significant delays and costs to the mining operation.
Overblasting can severely weaken rock slopes, requiring expensive and time consuming stabilization measures. Rockbolts, shotcrete, steel straps and mesh were used on the upper rock face in this photo, contrasting dramatically with the clean, well-blasted face below.
ing principles. Despite the availability of a good body of literature, much of the mining industry still tends to regard blasting as more of an art than a science. Consequently, there are many cases where a good pit slope design has been ruined by poor blasting practices, resulting in costly re-stabilization programs.

More and more mines are gradually realizing the importance of applying up-to-date blasting technology. Good references and experienced personnel are now available for designing the geometric blasting parameters (hole depth, subgrade, burden, spacing, collar), choosing the right explosive type, and selecting a delayed firing sequence. Yet, even in operations using recognized controlled blasting techniques, there are numerous cases where the blasting has damaged the rock slopes. This is due to one significant unknown factor:

What is the optimum amount of explosive to place in each blasthole which will provide good fragmentation and leave the final wall intact?

This parameter is commonly related to the powder factor.

Typically, in open pit mining, the explosive type, the perimeter blast geometry and the firing sequence are fixed as standard practice throughout the pit, leaving the powder factor as the only remaining design variable. Although it has long been recognized that the optimum powder factor is largely dependent on the nature of the rock mass, this critical relationship is extremely complex, incorporating many of the rock mass properties. Consequently, it has remained poorly understood and rarely documented.

In practice, the optimum powder factor is usually arrived at by a trial-and-error procedure. Due to fears of delaying the mining operation by under-blasting, this procedure tends to promote the use of excessive powder factors to ensure necessary fragmentation. Obviously, considerable damage can be inflicted to the slope before a satisfactory value is arrived at, resulting
in faces with high ravelling potential or sometimes triggering more massive slope failures. Due to the sequence of pit excavation, this tends to leave the most unstable faces at the top of the cut slope. When the excavation encounters variations in the rock mass, or moves into a different geologic unit, the problems of trial-and-error blasting begin again.

There is clearly a need to establish a reliable, working correlation between the properties of a rock mass and an optimum powder factor if one hopes to attain satisfactory and economical blasting results. This correlation should be relatively simple so that the non-geotechnical man-in-the-field could use it in regular blasting practice. Ideally, the system should be able to account for all the key rock mass properties which affect blasting and should be able to provide sufficient data for a hole-by-hole explosive design.

To date there has been limited success in achieving such a correlation system between powder factors and rock mass conditions. Two authors, Broadbent and Ashby, have established relationships between powder factors and single rock mass features, but these tend to be site specific. Due to the large number of key rock mass properties involved, a qualitative type of rock mass index, similar to those now in use by tunnel design engineers, would best be able to provide an overall value for correlation with powder factors.

This thesis contains the results of research work on this subject as part of a Masters of Applied Science degree program which commenced in September 1980. Following a year of background investigation, an experimental program of field testing was carried out over 6 months in 1981 at an open pit copper mine operated by Afton Operating Corporation near Kamloops, B.C. Afton Mine proved to be an excellent testing facility due to its broad range of geologic conditions.

As a result of the field research program, a good correlation was estab-
lished between controlled blasting powder factors and rock mass properties based on the performance of rotary blasthole drills. The ultimate goals of this simple correlation system are optimization of fragmentation, elimination of unacceptable blast damage, preservation of inherent rock strength, and maximization of slope stability.
1.1 REFERENCES


CHAPTER TWO

EXPLOSIVES AND BLASTING MATERIALS
2.0 EXPLOSIVES AND BLASTING MATERIALS

Explosives are perhaps the most thoroughly studied of all compounds and are a vast science unto themselves. A chemical explosive can be defined as a compound or a mixture of compounds which, when initiated by heat, impact, friction, or shock, undergoes a very rapid, self-propagating, exothermic decomposition.\(^1\) This decomposition produces more stable products, usually gases, which occupy a much larger volume than the explosive in its original configuration. The intense heat generated by the explosive reaction continues to rapidly expand the gaseous products to such an extent that they exert enormous pressure on their surroundings. The work done by an explosive depends primarily on the amount of heat given off during the explosion.

As shown in Figure 2.0-1, explosives are broadly classified as mechanical, chemical or nuclear. Since the mining and construction industries use chemical explosives exclusively, this will be the only group considered in this thesis. Chemical explosives can be sub-divided into high explosives or low explosives. High explosives "detonate", indicating that the reaction is moving through the explosive faster than the speed of sound in the unreacted explosive. Low explosives "deflagrate", a reaction slower than the speed of sound. All commercial explosives available today, except black powder, are high explosives.

High explosives are further sub-divided into primary and secondary explosives. Primary explosives are reliably detonated by spark, flame, and impact, but are too sensitive for use in conventional industrial applications. Characterized by high density (s.g. of 3 to 5), these extremely powerful products are generally used as military explosives and as explosive base ingredients in secondary explosive formulations. Secondary explosives are still very powerful but are less sensitive, requiring a detonating wave of considerable magnitude for successful initiation. Virtually all commercially available explosives belong in this secondary high explosive category.
FIGURE 2.0-1: GENERAL CLASSIFICATIONS OF BLASTING COMPOUNDS
Despite the large variety of formulations and brand names, secondary explosives can be split into the two general groupings of dynamites and gelatins. Originally, straight dynamites consisted of nitroglycerin and kieselguhr, a porous and inert variety of silica. Later, sodium nitrate was mixed with nitroglycerin to increase the power of the explosive. Their use has greatly diminished in recent years due to high costs and high sensitivity to shock and friction. The most widely used cartridged explosives in the field today are ammonia dynamos, sometimes referred to as extra dynamites. Ammonia dynamites are similar in composition to straight dynamites except that ammonium nitrate replaces a large portion of nitroglycerin and sodium nitrate. Manufactured in both high and low densities, ammonia dynamites are less expensive and considerably less sensitive to shock and friction.

Gelatins are tough, rubbery or plastic-textured compositions with the distinguishing feature of good water resistance. As with dynamites, gelatins have a variety of formulations. Blasting Gelatin is the most powerful explosive commercially available, composed of nitrocellulose, also known as gun-cotton, added to a very high percentage of nitroglycerin. Straight gelatin has a greatly reduced nitroglycerin content and is the equivalent of a straight dynamite in the dynamite category. Ammonium nitrate is used to replace some of the nitroglycerin and sodium nitrate to produce ammonia gelatin and semi-gelatin, which are respectively comparable to high and low density ammonia dynamites.

Although the wide variety of commercially available explosives have different chemical compositions, they all contain five basic functional ingredients:

1) Explosive bases
2) Combustibles
3) Oxygen carriers
4) Antacids

5) Absorbants

An explosive base is a solid or liquid which, upon the application of sufficient heat or shock, breaks down into gaseous products with an accompanying release of heat energy.\(^1\) Combustibles and oxygen carriers are added to an explosive to achieve oxygen balance. A combustible combines with excess oxygen in an explosive mixture to prevent the formation of nitrogen oxides. An oxygen carrier assures complete oxidation of the carbon in the explosive mixture to prevent the formation of carbon monoxide. An antacid is added to an explosive to increase stability in storage and an absorbant is used, when needed, to absorb liquid explosive bases.\(^1\)

Achieving complete oxygen balance during detonation is an important, but rarely stressed, aspect in the optimal performance of an explosive. The formation of nitrogen oxides or carbon monoxide, in addition to being undesirable because of their fumes, results in a lower heat of explosion than does the formation of carbon dioxide and nitrogen. A lower heat of explosion means a lower energy output, reducing the efficiency of the explosive and the blasting. Except for nitroglycerin and ammonium nitrate, virtually all explosives are oxygen-deficient. The sensitivity, strength and power of explosives reaches a maximum at perfect oxygen balance. The following two examples, each using an explosive formulated with ammonium nitrate and trinitrotoluene (T.N.T.), illustrate the effect of oxygen balance.

One explosive, 80/20 Amatol, detonates according to the following reaction:\(^2\)

\[
21\text{NH}_4\text{NO}_3 + 2\text{CH}_3\text{C}_6\text{H}_2(\text{NO}_2)_3 \rightarrow 14\text{CO}_2 + 47\text{H}_2\text{O} + 24\text{N}_2
\]

This explosive contains enough oxygen for complete oxidation and the reaction has a heat of explosion of approximately 1000 cal./gram of original explosive.
For 60/40 Amatol, the detonation reaction is:

$$17\text{NH}_4\text{NO}_3 + 2\text{CH}_3\text{C}_6\text{H}_2(\text{NO}_2)_3 \rightarrow 17\text{CO} + 11\text{CO}_2 + 36\text{H}_2\text{O} + 8\text{H}_2 + 23\text{N}_2$$

This explosive is 25% deficient in oxygen, producing a significant quantity of carbon monoxide and with a resulting heat of explosion of approximately 880 cal./gram. Thus, it is evident that a lack of oxygen results in a decrease in the heat of explosion, reducing the efficiency of the explosive.

In general practice, the term "explosive" is usually used as a collective term to include any substance used in blasting. This terminology has led to some misunderstanding when dealing with blasting agents. Technically, these widely used blasting compounds are not explosives, belonging to a completely separate classification as represented in Figure 2.0-1. Some references have added to the confusion by classifying blasting agents as tertiary high explosives, presumably because they require a powerful shock wave for initiating their detonation. This is completely incorrect. A blasting agent is any material or mixture, consisting of a fuel and oxidizer, intended for blasting, not otherwise classified as an explosive and in which none of the ingredients is classified as an explosive, provided that the finished product cannot be detonated by means of a No. 8 blasting cap.¹ A blasting agent consists primarily of inorganic nitrates and carbonaceous fuels and may contain additional substances such as powdered aluminum or ferrosilicon. However, the addition of an explosive ingredient changes the classification of the mixture from a blasting agent to an explosive.

Due to the complexity of modern explosives and blasting agents, further detailed discussion of their chemical and physical components and reactions is unwarranted within this thesis. Instead, it is more important to examine those performance properties of explosives which are critical to the understanding of an explosive's behaviour and its interaction with a rock mass.
2.1 PROPERTIES OF EXPLOSIVES

2.1.1 Strength:

The strength of an explosive may be defined as its ability to displace the confining medium, or as the amount of energy released by the explosion. Strength is synonymous with work and is measured by the ballistic mortar test. This is an empirical test for comparing explosives, measuring the ability of 10 grams of explosive to deflect a heavy steel mortar. Throughout this thesis, the strength of an explosive or blasting agent is expressed as a percentage of the strength of gravity-loaded ammonium nitrate/fuel oil (AN/FO). It should be noted that some rating systems use the strength of T.N.T. as the standard value for comparative purposes.

The two ratings commonly used are Relative Weight Strength (RWS), which compares explosives on a weight basis, and Relative Bulk Strength (RBS), which compares explosives on a volume basis. The weight strength and bulk strength are related by specific gravity or density.

\[
\text{RWS of Explosive} = \frac{\text{Weight of AN/FO} \times 100\%}{\text{Equivalent Weight of Explosive}}
\]

\[
\text{RBS of Explosive} = \frac{\text{Volume of AN/FO} \times 100\%}{\text{Equivalent Volume of Explosive}}
\]

\[
\frac{\text{RBS}}{\text{RWS}} \quad = \quad \frac{\text{Specific Gravity of Explosive}}{\text{Specific Gravity of AN/FO (0.84)}}
\]

The term "strength" has been traditionally used by explosives manufacturers to describe various grades of explosives. Although it is now out-dated, the term is still so common in industry that an understanding of strength ratings is important for anyone working in the field of blasting.

When dynamites were only a mixture of nitroglycerin and an inert filler such as kieselguhr, a 60% dynamite (containing 60% nitroglycerin by weight) was 3 times as strong as a 20% dynamite. However, present day dynamites contain active ingredients instead of inert substances, adding substantially to the
energy of the explosive. Consequently, a 60% dynamite still contains 60% nitroglycerin, but it is much less than 3 times the strength of 20% dynamite. Nowadays, the persistence of the archaic "strength" term causes considerable confusion and misunderstanding. It must be remembered that the "percent" labels on dynamites refer only to the nitroglycerin content and no longer signify any relative strength values.

Another drawback to the strength labels is due to the nature of the ballistic mortar test. This test takes no account of other important explosive properties such as the velocity of detonation.

Therefore, the term "strength" is inaccurate and misleading, bearing little relation to the capability of an explosive to break rock. It seems to have persisted over the years only by its simplicity of application. In general, strength ratings will only apply when comparing two explosives with similar velocities of detonation and should otherwise be avoided.

2.1.2 Velocity of Detonation:

Mentioned in the discussion of strength ratings, the velocity of detonation may be the most important single property to consider when rating an explosive. This may be quoted as either a confined or unconfined value. The confined detonation velocity is a measure of the speed at which the detonation wave travels through a column of explosive within a borehole or other confined space. This is usually measured with high speed photography, recording the time intervals for the light-emitting detonation wave front to pass between regularly spaced holes in a steel tube. The unconfined velocity indicates the detonation speed when the explosive is detonated in the open or unconfined state. This is measured by detonating the explosive from either end while it rests on a steel plate, and later recording the position of a notch formed in the steel where the opposing detonation waves collide. See Figure 2.1-1.

Since explosives are usually under some degree of confinement, the confined
MEASUREMENT OF THE CONFINED VELOCITY OF DETONATION

UNCONFINED VELOCITY OF DETONATION = \( V_c \left( \frac{L_2 - L_1}{x_1 - x_2} \right) \)

FIGURE 2.1-1: TEST METHODS FOR DETERMINING CONFINED AND UNCONFINED VELOCITIES OF DETONATION
value is the more significant. However, manufacturers usually quote the unconfined value because it is an easier test to perform and there is no standard borehole diameter to use for confined tests. Since unconfined velocities are generally 70 to 80 percent of confined velocities, it is important to know the conditions under which the manufacturer made his velocity measurement.

The detonation wave velocity of an explosive is dependent on:

1) the density of the explosive
2) the ingredients in the explosive
3) the particle size of the ingredients
4) the charge diameter
5) the degree of confinement

Decreased particle size, increased charge diameter, and increased confinement all tend to increase the detonation velocity. Velocity will decrease with charge diameter until, at the explosive's critical diameter, propagation is no longer assured and misfires are likely. With cartridge explosives the confined velocity is seldom attained because complete confinement is usually impossible.

The velocity of detonation is a very significant factor in assessing the ability of an explosive to break or move rock. A high velocity explosive has greater brisance or shattering ability and is preferrable in hard or brittle rock. A low velocity explosive relies more on its expanding gas pressures for a "heaving" action, providing satisfactory results in a soft or highly fractured rock at lower costs.

2.1.3 Power:

The strict definition of power is the rate of doing work. For an explosive, power depends on both strength and detonating velocity - the amount of energy released and the speed of release. In general terms, "power" is used to indicate the effectiveness of detonation of an explosive, such as
its potential to penetrate or shatter.

2.1.4 **Brisance:**

Brisance, a unique characteristic of explosives, is the extreme shattering effect resulting from almost instantaneous decomposition. Although there is no precise definition, the brisance of an explosive is generally accepted as being proportional to the product of its load density, reaction zone pressure, and detonating velocity. Explosive decomposition proceeds as a self-sustaining wave, travelling at high velocity, which is enveloped by extreme pressures. On contact with surrounding material, the wave delivers momentary shocks of terrific intensity, which induce the shattering effect.

2.1.5 **Sensitivity:**

Sensitivity is a measure of the impulse magnitude required to start an explosive reaction. Explosives will be sensitive to varying degrees of impact, friction, heat, or spark. Primary high explosives are very sensitive, requiring little energy to propagate an explosive reaction. Secondary high explosives are less sensitive while blasting agents are the least sensitive.

The sensitivity of an explosive determines the method by which a charge may be detonated, the minimum charge diameter, and the safety with which the explosives may be handled. While sensitive explosives can be reliably detonated by a No. 6 blasting cap, relatively insensitive compounds such as blasting agents must be initiated by a powerful, sensitive primer charge containing a high explosive such as T.N.T. or P.E.T.N. (Pentaerythritoltetranitrate). The shock wave travels through the explosive creating local friction between the grains, initiating secondary detonation.

As an explosive is compressed to higher densities to develop greater bulk strength, the explosive becomes less sensitive until it becomes totally insensitive or "dead-pressed". In cases where the charge diameter is less than the hole diameter, a phenomenon known as the "channel effect" can de-
sensitize an explosive and lead to mis-fires. The high pressure gas wave travelling up the air space ahead of the explosive's detonation wave front may compress the explosive to its dead-pressed state, extinguishing the detonation.

Highly sensitive explosives will detonate when used in small diameter charges. As the sensitivity of the explosive is reduced, the diameter of the charge must be increased. The relationship of sensitivity to density is illustrated in Figure 2.1-2.

2.1.6 Density:

The density of an explosive is usually expressed in terms of specific gravity, the ratio of the density of the explosive to the density of water under standard conditions. Occasionally, it is expressed in terms of a cartridge count which represents the number of $1\frac{1}{4}$ by 8 inch cartridges in a 50 pound box. This term is archaic and is clearly of no value when dealing with bulk explosives or blasting agents. For these products, the density is often expressed as the kilograms of explosive per metre of charge length in a given size borehole.

With only a few exceptions, the denser explosives generally give the higher detonation velocities and pressures required in hard, dense rock. A dense explosive also permits maximum utilization of the borehole drilled which is an important economic factor with the high costs of hard rock drilling.

In easily fragmented rock where little explosive energy is needed, a low density explosive will suffice. Not only are the low density explosives generally less expensive, but they will fill more of the hole, providing a better pressure distribution upon detonation.

The density of an explosive is important if the blast site will be in wet or high groundwater conditions. An explosive with a specific gravity less than 1.0 or a cartridge count greater than 140 will float in the borehole.
FIGURE 2.1-2: RELATIONSHIP OF SENSITIVITY TO SPECIFIC GRAVITY AND CHARGE DIAMETER
2.1.7 Detonation Pressure:

Detonation pressure is an important characteristic of an explosive but is seldom mentioned in technical data sheets or blasters' handbooks. The detonation pressure is a measure of the pressure in the detonation wave. This directly controls the amplitude of the stress pulse produced in the rock by an explosive. The reflection of this stress pulse at a free face is an important mechanism in rock breakage. Detonation pressure is a function of the detonation velocity and density of an explosive but their relationship is complex and depends on the ingredients of the explosive. The following approximation, illustrated in Figure 2.1-3, is one of several that can be made:

\[
p = \frac{4.5 \times 10^{-4} \times D \times C^2}{1.0 + 0.8 \times D}
\]

where: 
- \( P \) = detonation pressure in MPa
- \( D \) = specific gravity
- \( C \) = detonation velocity in m./sec.

Sometimes it is preferrable to use the borehole pressure for blasting studies. The borehole pressure is the peak effective pressure acting behind the detonation head on the cylindrical surface area of the borehole. It is approximately one half of the detonation pressure and can be estimated by:

\[
BP = \frac{N \times D \times C^2}{13470}
\]

where: 
- \( BP \) = borehole pressure in MPa
- \( D \) = specific gravity
- \( C \) = detonation velocity in m./sec.
- \( N \) = factor obtained from Figure 2.1-4

As can be seen, the detonation and borehole pressures are much more dependent on detonation velocity than on specific gravity.
FIGURE 2.1-3: DETONATION PRESSURE AS A FUNCTION OF SPECIFIC GRAVITY AND DETONATION VELOCITY
FIGURE 2.1-4: FACTOR N vs. SPECIFIC GRAVITY FOR BOREHOLE PRESSURE ESTIMATION
(after Calder⁷)
2.1.8 Water Resistance:

The water resistance of an explosive is a measure of its ability to withstand exposure to water without deteriorating or losing sensitivity. Water resistance ratings range from "Nil", for use in dry holes, to "Excellent", used when groundwater is percolating through the borehole or when the explosive is exposed to standing water for prolonged periods. In general, gelatins offer the best water resistance with low-density dynamites and dry blasting agents having little or none. Emission of brown nitrogen-oxide fumes from a blast often means that the explosive has deteriorated from exposure to water and indicates that a change should be made in the choice of explosive or blasting procedures.

2.1.9 Considerations For Selecting An Explosive:

When selecting an explosive for a specific blasting project, all of the above characteristics will have to be assessed to differing degrees. Predicting the behaviour of an explosive demands an appreciation of the interplay of its strength, detonating velocity, power and brisance. Such factors as safety and cost are obviously important, but for effectiveness alone, an explosive's velocity of detonation is the governing factor. The schematic diagram presented in Figure 2.1-5 summarizes the inter-relationships of the most important explosive properties.

There are a variety of trade-offs which must be considered when selecting an explosive, but for every given blasting job there is an explosive or blasting agent that will perform best. To select the most suitable explosive, the blaster must define the physical site conditions such as:

1) the hardness and density of the rock
2) geological and geophysical characteristics
3) moisture conditions
### Figure 2.1.5: Some Relative Ingredients and Properties of Explosives

(after Dick)

<table>
<thead>
<tr>
<th>GELATINOUS</th>
<th>NONGELATINOUS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Blasting Gelatin</td>
<td>Nitroglycerin</td>
</tr>
<tr>
<td>Straight Gelatin</td>
<td>Straight Dynamite</td>
</tr>
<tr>
<td>Ammonia Gelatin</td>
<td>High density Dynamite</td>
</tr>
<tr>
<td>Semigelatin</td>
<td>Ammonia Dynamite</td>
</tr>
<tr>
<td>Slurries</td>
<td>Dry blasting agents</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Decreasing water resistance</th>
<th>Increasing water resistance</th>
</tr>
</thead>
<tbody>
<tr>
<td>Decreasing detonation velocity</td>
<td>Increasing ammonium nitrate content</td>
</tr>
<tr>
<td>Decreasing density</td>
<td>Decreasing cost</td>
</tr>
<tr>
<td>Decreasing nitroglycerin content</td>
<td>Increasing cost</td>
</tr>
</tbody>
</table>

Decreasing water resistance

Decreasing detonation velocity

Decreasing density

Decreasing nitroglycerin content
4) available ventilation

and must also decide on what the optimum results are, including:

5) degree of fragmentation
6) height and displacement of muck pile
7) condition of final face

Knowing these factors, the blaster can decide which explosive properties are important in his particular situation.
2.2 **EXPLOSIVES USED ON THIS RESEARCH PROJECT**

Three different types of blasting compounds, all manufactured by Canadian Industries Limited (C.I.L.), were used during the field experiments carried out at Afton Mine. The majority of test blasts used bulk-loaded AN/FO, a common dry blasting agent. On occasions when groundwater conditions prevented the use of AN/FO, it was necessary to go to a slurry explosive for its improved water resistance. Most commonly, pre-packaged Hydromex T-3 was used, but bulk-loaded Powergel A was tried in two of the blasts.

The purpose of this section is to provide greater technical detail for the explosives and detonating materials used throughout the field experiments.

### 2.2.1 Ammonium Nitrate / Fuel Oil (AN/FO):

Dry blasting agents were patented in Sweden in 1867 but their full potential was not realized until the development of AN/FO in the mid-1950's. It quickly replaced dynamos and gelatins in many applications. Although much of the blasting field is becoming dominated by slurries, AN/FO is still the most widely used blasting agent in softer rock formations.

AN/FO consists of ammonium nitrate prills and fuel oil and, when the mixture is properly balanced, detonates with the following chemical reaction:

\[
3\text{NH}_4\text{NO}_3 + \text{CH}_2 \rightarrow 7\text{H}_2\text{O} + \text{CO}_2 + 3\text{N}_2
\]

From the left side of the equation one can calculate that an oxygen balanced AN/FO contains 94.5% ammonium nitrate and 5.5% fuel oil. These proportions are usually 94% and 6% in practice, with the extra fuel oil ensuring intimate combination with the ammonium nitrate.

Not being cap-sensitive, AN/FO must be initiated by a high explosive primer such as 75% ammonia gelatin, composition B (a mixture of T.N.T. and R.D.X.), or pentolite (a mixture of equal parts of T.N.T. and P.E.T.N.). Since initiation sensitivity increases with confinement and decreases with
greater charge diameters, the size of the primer must be increased appropriately in large diameter holes.

Depending on the density and particle size of the ammonium nitrate, the specific gravity of AN/FO varies from 0.75 to 0.95. Densities in the borehole can be increased with ejector type pneumatic loaders, but AN/FO reaches a "dead-pressed" state at a specific gravity of 1.2 and will not detonate. The velocity of detonation of AN/FO increases with borehole diameter to an upper limit of about 4200 m./sec. Figure 2.2-1 shows the relationship between the confined detonation velocity and the borehole diameter at two specific gravities of AN/FO.

The principal advantages of AN/FO are its safety in transportation, storage and handling, as well as the ease of loading and low price compared to other explosives. See Plates 3 and 4. Bulk-loaded AN/FO has the further advantage of completely filling the borehole, leaving no voids. This direct coupling to the borehole walls assures a more efficient use of explosive energy than can be obtained with cartridgeed explosives.

AN/FO is an ideal blasting material for weak or highly fractured rock masses where more heaving action and less brisance is required. Since weaker rock usually requires relatively low explosive loads, the low density of AN/FO means that more of the borehole is filled and the heaving forces are better distributed.

The most serious disadvantage with AN/FO is its complete lack of water resistance other than that supplied by external protection. As shown in Figure 2.2-2, AN/FO will fail to detonate with a moisture content as low as 9%. When a wet borehole is encountered, water is usually pumped from the hole, a plastic sleeve liner inserted, and the AN/FO loaded into the liner. This procedure is tedious and slow but is the only alternative without switching to a slurry or other water resistant explosive.

Although AN/FO will withstand prolonged periods of storage, it can deter-
FIGURE 2.2-1: CONFINED DETONATION VELOCITY OF AN/FO vs. BOREHOLE DIAMETER
(after Dick')
The C.I.L. truck transports and delivers bulk AN/FO safely and easily. On site, it mixes the fuel oil with the ammonium nitrate prills and ejects the finished mixture pneumatically through the black hose at the rear of the truck.
Loading AN/FO into a blasthole is a fast and easy procedure. This particular hole contained water, was pumped out and had a plastic liner inserted. Two primers are used to ensure detonation in wet holes, as indicated by the two Primacord downlines.
FIGURE 2.2-2: EFFECT OF WATER CONTENT ON THE DETONATION VELOCITY OF AN/FO
iorate under certain conditions. In a warm and humid environment, the ammonium nitrate prills will eventually become lumpy and mushy, leading to caking and a loss in sensitivity. Another cause of deterioration is the "cycling effect" which occurs when the AN/FO is cycled through 0°C or 32°C more than once. This causes the prills to swell and contract and eventually crumble. 8

The basic properties of AN/FO, as used at Afton Mine, are summarized in Appendix II.

2.2.2 Slurries:

Slurries are the most recent development in explosive formulations and trends indicate that they will eventually become the most widely used of all blasting materials. Sometimes called water gels, slurries contain high proportions of ammonium nitrate, part of which is in an aqueous solution. Slurry explosives contain cap-sensitive ingredients and the mixture itself may be cap-sensitive. The ammonium nitrate and sensitizer mixture is thickened and gelled with a gum to give considerable water resistance.

Developed in 1958, Hydromex T-3 was the first commercial slurry available in Canada. It is a mixture of ammonium nitrate, T.N.T., water and minor amounts of other ingredients. 11 It has a low sensitivity to shock, friction and impact, making it safe in handling and use. Only high strength primers can ensure consistent initiation. Unlike AN/FO, Hydromex T-3 has a high density and a high velocity of detonation. The consistency of the slurry ranges from fluid near 38°C to rigid at freezing temperatures.

At Afton Mine, Hydromex T-3 was used instead of AN/FO only when it was impossible to pump out a wet blasthole or when the hole recharged with water too rapidly. Since it was generally not required in large, bulk quantities, the Hydromex was supplied in 25 kg., 203 mm. (8 inch) diameter bags which were individually hand loaded.

As more wet blastholes were encountered in some areas of the open pit
during summer 1981, the laborious hand-loading of Hydromex T-3 caused a significant drop in blast production. This indicated that a bulk-loaded slurry may be needed as the pit deepens and encounters more groundwater. Two trial blasts were made with Powergel A, a bulk, truck-loaded slurry explosive.

This explosive is a metallized, T.N.T.-based slurry explosive with a lower density and lower velocity of detonation than Hydromex T-3. Powergel A is cap-sensitive yet is still safe to handle. It has the advantages of fast, easy loading and, like AN/FO, provides direct borehole coupling for more efficient use of the explosive energy.¹²

Neither Hydromex T-3 nor Powergel A can be considered as equivalent substitutes for AN/FO due to their higher densities and detonation velocities. Both explosives are much more expensive than AN/FO and were selected strictly for their water resistant properties.

The basic properties of Hydromex T-3 and Powergel A, as used at Afton Mine, are summarized in Appendix II.

2.2.3 Primers:

A primer is any high power, high velocity explosive compound capable of initiating detonation of low sensitivity blasting agents or explosives. The primers themselves can be detonated directly by blasting caps or detonating cord.

Procore III primers, which are exceptionally powerful, are used at Afton. Each primer weighs 450 grams (1 lb.) and consists of T.N.T. cast around a Pentolite core.¹³ Two holes are provided in each primer to allow the unit to be threaded onto a downline of detonating cord. The Procore primers provide very reliable initiation of both AN/FO and Hydromex T-3, being especially designed for large diameter blastholes.

These primers are designed for safe handling, being much less sensitive to shock, friction or impact in normal use than conventional high explosives.
They are completely waterproof and will not deteriorate during extended storage. Their performance is unaffected by temperature changes or by immersion in properly formulated blasting agent compositions, even for an extended period.

Procore primers are very high explosives and should be treated with appropriate care during handling and loading operations. Any pieces accidentally broken off these primers or powdered fragments should be cleaned up and promptly destroyed. If left on the ground, these pieces could become admixed with grit, making them more sensitive and a possible hazard to the blasting operation. See Appendix II.

2.2.4 Detonating Cord:

Primacord is a detonating cord consisting of a high explosive core of P.E.T.N. contained within a reinforced waterproof covering. It has a very high velocity of detonation and is relatively insensitive to detonation by friction or ordinary shock. It is unaffected by stray currents or other forms of extraneous electricity, and is an extremely safe method of initiating high explosives or blasting agents.

Although light in weight, it possesses good tensile strength and is extremely easy to handle and connect up in a blast. It absorbs water very slowly, even from a cut end, but should the cord become soaked with water, it will still detonate satisfactorily if initiated from a dry end. The violence and speed of its explosion is sufficient to ensure detonation of high explosives of normal sensitivity in contact with it.

Primacord comes in several types, each possessing the same basic detonating properties, but with differing degrees of strength and toughness. Afton Mine uses the heavy, yellow, reinforced Primacord for all trunklines and downlines within the blasting pattern where the cord may be subjected to abrasion. A lower cost detonating cord, red E-Cord, is used for the long trunkline from the blasting cap to the blast pattern. See Appendix II.
2.2.5 Electric Blasting Caps:

An electric blasting cap usually consists of a metal shell with two wires leading in from one end. The ends of these leg wires are connected within the device by a very fine bridge wire of high resistance. This heats to incandescence on the passage of an electric current of sufficient intensity, igniting a heat sensitive loose charge. All C.I.L. electric detonators are supplied with aluminum foil or polythene shunts as a protection against stray currents. A "static short" is also incorporated into their make-up as a protection against static charges.⁴

Although caps are available with a variety of short or long delay elements, Afton Mine uses the instantaneous electric blasting caps. See Appendix II.

2.2.6 Blasting Machines:

Blasting machines are small, portable units to provide current for firing blasts electrically where alternate power sources are not readily available. Afton Mine uses the C.I.L. 10-Shot Blasting Machine, designed for use where only a few caps are to be fired in a particular blast. It is powered by two standard "D" size flashlight batteries which charge heavy-duty metallized paper capacitors to a potential of approximately 200 volts.⁴ Its rated capacity is 10 E.B. Caps connected in a single series with a maximum circuit resistance (caps plus lead wires) of 40 ohms. See Appendix II.

2.2.6 Plastic Borehole Liner:

When wet boreholes are encountered, they have to be pumped dry if AN/FO is to be loaded. These holes must then be lined with a tubular plastic sleeve to provide a clean, dry contact in the borehole.

Plastic liners are recommended primarily for use in vertical holes in open-cut work where the blasting agent can be poured into the borehole. When pneumatic loading is used, the plastic liner increases the hazard of static electricity build-up. More toxic fumes are also generated when liners are
A variety of liners are available, but few are completely free of defects or pin-holes which may leak water into the AN/FO. Thus, even when liners are used, it is best to load and fire within one shift instead of leaving holes loaded for several days.

The plastic liner used at Afton Mine is a continuously extruded sleeve of 10 mil. plastic, coming on a 183 m. (600 ft.) roll. With a lay-flat dimension of 394 mm. (15-1/2 inches), it has a 250 mm. (9-7/8 inches) diameter bore which is slightly larger than the 230 mm. (9 inch) borehole diameter. The end of the sleeve is folded and tied shut after placing a small amount of ballast into the tube to ensure that it will sink to the bottom of the hole. Although the folded and tied end may not be as water-tight as the more expensive heat-sealed method, it provides sufficient protection for the duration of one shift. Despite the high groundwater conditions at Afton, no mis-fires were recorded during the field experiments. See Appendix II.
2.3 REFERENCES


7) CALDER, P.: Pit Slope Manual, Chapter 7 - Perimeter Blasting. CANMET (Canada Centre for Mineral and Energy Technology), May 1977.


All the following references are brochures prepared by Canadian Industries Ltd., Explosives Technical Marketing Services:

9) Information Report No. 117: Prilled Ammonium Nitrate

10) Information Report No. 150: Bulk AN/FO


12) Information Report No. 151: Bulk Slurry Explosives


CHAPTER THREE

ROCK MASS DETONICS
3.0 ROCK MASS DETONICS

As described in Chapter Two, explosives have several specific properties, available in a variety of combinations, which are carefully created by precise chemical formulations. Through decades of testing and analysis in controlled laboratory conditions, their detonating characteristics and behaviour in cylindrical holes have become well understood and documented. However, the mechanisms of rock mass fragmentation initiated by explosives have remained relatively obscure. Even though rock blasting comprises the most important industrial use of explosives, there has yet to be developed a comprehensive theory that completely explains how explosives break rock.¹

The poor understanding of fragmenting mechanisms can be attributed to several factors. First, the detailed physical processes leading to the end result are highly complex and occur in such a short time that observation becomes difficult. Second, the opaqueness of rock masks the internal fragmenting action, permitting only surface observations. Third, the large destructive forces involved in a blast make sensitive instrumentation almost impossible. Fourth, the geological make-up of rock masses varies greatly, resulting in different fragmentation characteristics from one blast site to another. Furthermore, each rock mass in itself is highly heterogeneous with included discontinuities which influence the distribution of transient stresses created by explosive detonation.

Clearly, the study of rock mass detonics is a complicated, specialized field of study, prompting Ulf Langefors, an eminent Swedish blasting expert, to state: "Fragmentation in rock blasting is one of the most important of the remaining problems in the sphere of technics."²

In an effort to reduce the complexity of the overall problem to a manageable working level, most of the experimental and theoretical work done on rock mass detonics has dealt with rock as a homogeneous and isotropic material.
Although this does not represent the heterogeneous real world, further progress in understanding blasting mechanisms will only be possible if and when the mechanisms in homogeneous materials are understood.\(^3\)

Many different materials have been used for model blasting experiments but plexiglass is the most popular. In addition to its uniform characteristics, plexiglass is transparent, permitting observation of fragmenting processes within the specimen. Although plexiglass is more ductile than most rocks, the fracture pattern at high rates of loading appears to be identical with that in rock.\(^4\) Test blast patterns can be drilled into plexiglass blocks and small explosive charges placed in the holes. The detonation is then filmed with the aid of high speed cine-cameras capable of up to 100,000 frames per second. A frame by frame study reveals the details of detonation, the movement of shock waves, and the fracturing sequences. Plexiglass model blasting has been performed by many researchers, with some particularly good photographs published by Lange for s and Kihlstrom\(^2\), Kutter and Fairhurst\(^4\), Cook\(^5\), and Johansson and Persson\(^6\).

Other blasting models have included underwater blasts for shock wave studies and the use of lead blocks for measuring volumetric displacement. Since none of these materials have the same properties or modulii as rock, some blasting tests have also been done with large blocks of homogeneous rock. Notable results have been published by Noren\(^7\) and by Bergmann et al\(^8\)\(\text{--}\)\(^10\) with large intact blocks of granite.

From such testing work, some ideas have emerged about key rock fragmenting mechanisms. At this time, there have been three major theories put forward, each developed by independent research groups. One is known as the "shock wave theory", championed primarily by the U.S. Bureau of Mines and described in literature by Hino\(^11\). This theory stresses impedance mismatch, release wave scabbing or fragmentation, and other shock wave concepts developed from fundamental work on impact loading of metals and compressibility of solids.
at extremely high pressures.

Another concept, known as the "energy theory", is widely accepted in
the Soviet Union and best described in publications by Cook. This describes
rock breakage as a stress relief following the initial transfer of the blast
energy into potential energy by powerful compression under the sustained
pressure of the detonation products and the great inertia of the burden.

The third theory of rock blasting is the "radial cracking theory" of
the Swedish Detonics Research group which has some overlap with both the "shock
wave" and "energy" theories. It stresses the importance of shear wave fracture
during the early stages of shock wave propagation, radial cracking thereafter
and finally concentric stress relief.

This third theory is the one with which this author is most familiar
and which formed the basis of any field blasting experiments. The next sec-
tion of this chapter outlines the fragmentation mechanics of homogeneous,
isotropic rock as explained by the Swedish theory. Once these basic principles
have been covered, the influences of inhomogeneous rock properties will be
discussed.
3.1 THE BREAKAGE PROCESS IN HOMOGENEOUS ROCK

In surface blasting, there are basically two different blasting configurations. In one case, the blasthole is completely surrounded by rock with the mouth of the blasthole at the only free surface. This configuration is known as crater blasting and is used for sinking cuts or ramp cuts. At Afton Mine, it was also used for most production blasts in the central portion of the pit.

In the other case, the rock mass has an additional free surface near the blasthole and parallel to it. Referred to as bench blasting or free face blasting, it is used at Afton for all pit perimeter blasts.

These two configurations are compared in Figure 3.1-1. Although the same detonics principles are involved in each case, the behaviour of the blasts and the end results are quite different.

3.1.1 Bench Blasting:

This is the most important surface blasting mode, forming the basis of all precise, controlled blasting methods. Any reference book on blasting techniques will stress the importance of blasting to a free face parallel to the blasthole. To better understand the mechanics involved, it is desirable to consider a single explosive charge which is fully coupled with the blasthole. Such a configuration may be likened to a thick-walled cylinder.

When the explosive is detonated, it changes extremely rapidly into very hot, high pressure gases occupying the same volume as the original explosive (i.e. the blasthole volume). This sudden application of the sustained explosion pressure causes the blasthole itself to expand and thus generate an intense strain in the surrounding rock. Measurements indicate the strain rate to be between 250 mm./sec. and 500 mm./sec.\textsuperscript{13}

The cylindrically expanding strain wave often exceeds the elastic limit of the softer rocks. As the volume is compressed, fracture occurs through
CRATER BLAST CONFIGURATION

Nearest free surface is perpendicular to blasthole axis.

BENCH BLAST CONFIGURATION

Nearest free surface is parallel to blasthole axis.

FIGURE 3.1-1: COMPARISON OF BLASTHOLE CONFIGURATIONS IN CRATER AND BENCH BLASTING
the collapse of intercrystalline and intergrain structure, completely pulverizing the rock immediately surrounding the blasthole. The extent of this crushed zone increases with both the detonation pressure of the charge and the degree of coupling in the hole.

As the outgoing strain wave extends into the rock, travelling at 3000 to 5000 metres per second, it sets up tangential stresses which create fine cracks extending radially and parallel to the blasthole axis. See Figure 3.1-2. The first of these radial cracks develops in 1 to 2 milliseconds. The actions during this first stage of the breakage process are illustrated in Figures 3.1-3 and 3.1-4 and in Plates 5 and 6.

During the first stage, the pressure associated with the outgoing shock wave is positive. When the shock wave reaches the free face of the bench it will reflect, causing the pressure to fall rapidly to negative values, creating a tension wave. This wave travels back into the rock and, because this material is less resistant to tension than to compression, further failure cracks will develop under the tensile stress.

If the reflected wave is sufficiently intense, it may cause scabbing or spalling at the free face. See Figure 3.1-5. However, this scabbing effect is considered to be of minor importance in rock breaking. It has been calculated that the explosive load would have to be eight times that normally required in order to cause failure of the rock by reflected shock wave alone.

The principal interaction between the radial crack system of the first stage and the reflected tensile wave was studied by Field and Ladegaard-Pedersen. They noticed that two primary radial cracks would extend to a greater distance from the blasthole than all other cracks and had a direction of about 45° to the free face. The material in front of the blasthole would tend to break out along these major cracks. In a series of model scale tests in plexiglass, they noticed that the reflected tensile wave increases the tensile stress at the tip of those Stage 1 cracks tangential to the curved
FIGURE 3.1-2: INITIAL RADIAL FRACTURING MECHANISM ON AN ELEMENT OF ROCK

(after Mercer\textsuperscript{13})
Expanding Shock Wave

Initial Radial Cracks

Original Blasthole

Pulverized Zone

Free Face

**Figure 3.1-3:** Plan View of Stage 1 of Rock Breakage Process

(after Lang and Favreau)
Expanding Shock Wave
Crushed Zone
Ridge-like
Traces of Tensile Fracture Path
Point of Detonation Initiation

Note:
Shape of shock wave front will vary slightly depending on relative values of Detonation Velocity and Shock Wave Velocity.

FIGURE 3.1-4: BLASTHOLE CROSS-SECTION IN THE PLANE OF A RADIAL CRACK DURING STAGE 1 OF THE ROCK BREAKAGE PROCESS
Plan view of a blasthole showing Stage 1 fracturing mechanisms in a homogeneous rock block as illustrated in Figure 3.1-3. Note the distorted blasthole surrounded by an intensely crushed zone and the radial fracture pattern.
View of fracture plane parallel to a single blasthole in a homogeneous rock block. Surface texture clearly indicates this to be a tensile fracture. Note radial nature of tensile fracture pattern as illustrated in Figure 3.1-4. Excessive fracture at toe of hole probably due to location of primer charge.
FIGURE 3.1-5: PLAN VIEW OF STAGE 2 OF ROCK BREAKAGE PROCESS
wave front. See Figure 3.1-5. Therefore, those cracks forming an angle of 40° to 80° to the normal of the free face are given a greater propagation velocity. They will travel ahead of the other cracks and, by relaxation of the surrounding material, will reduce the velocity of these. This theory actually provides a plausible explanation for earlier observations that the break-out angle increases with increasing burden and with increasing bench height as reported by Persson, Ladegaard-Pedersen and Kihlstrom\textsuperscript{16}.

Throughout the first and second stages of the breakage process, the function of the shock wave energy is to induce numerous small fractures around the blasthole. However, the shock wave energy theoretically amounts to only 5 to 15% of the total explosive energy.\textsuperscript{3} This strongly suggests that the shock wave is not directly responsible for any significant amount of rock breakage, but merely provides the basic conditioning for the third stage of the process.

In this last stage, the actual rock breakage is a slower action. Under the influence of the exceedingly high quasi-static pressure of the explosion gases, the primary radial cracks are rapidly enlarged under the combined effects of tensile stress induced by radial compression and by pneumatic wedging. As gases escape through the radial cracks and stemming, and as the mass in front of the blasthole yields and moves forward, the high compressive stresses within the rock unload in much the same way as a compressed coil spring being suddenly released.\textsuperscript{14} The effect of unloading induces high tension stresses within the mass which complete the breakage process. The rock is then accelerated forwards and upwards by the action of the explosive gases. See Figure 3.1-6. The heaving pressure of these gases acting on the blasthole burden causes the fragmented rock mass to be displaced into a muckpile with a shape depending on the properties of the explosive and the rock.

Experimental measurements have shown that the time interval between detonation and the start of motion of the free face is between 5 and 10 times
FIGURE 3.1-6: PLAN VIEW OF STAGE 3 OF ROCK BREAKAGE PROCESS

(after Lang and Favreau\textsuperscript{14})
the shock wave travel time from the blasthole to the free surface. After this time interval there is a rapid acceleration to the final breakout velocity. This phenomenon was first noticed by Noren in full scale granite blasts where the time interval between detonation and free face motion was found to be 7 times the shock travel time. This was a point of great concern due to its implications in the sequencing of multi-rowed delayed blast patterns. The processes involved were later explained in a mathematical model developed by Johanssen which accounted for expansion of the gases into specific crack volumes created by the initial high compressive stresses. This model also predicted the same delay interval of 7 times the shock travel time between detonation and the start of face movement.

From the knowledge of rock and explosive behaviour, better models of the blasting process can be postulated. As indicated by Just, the application of scientific principles may allow an engineer to design a blast which will produce a desired fragmentation and shape of muckpile to suit the particular loading equipment to be used.

3.1.2 Crater Blasting:

The important effect of a free surface on the fragmentation process is quite clear in the case of bench blasting. In crater blasting, with the only free surface being perpendicular to the blasthole axis, the effect is less significant.

Around the blasthole, the fragmenting process cannot go beyond the Stage 1 phase discussed for bench blasting. Without a free face to provide lateral reflection, the initial compressive stress wave continues to expand out into the rock until it dissipates at some considerable distance from the blasthole. This results in a radial pattern of extensive cracks parallel to the blasthole axis.

The compressive stress wave only gets reflected back into the rock near
the mouth of the hole at the upper free surface. However, this tensile wave has much less effect than in the case of bench blasting because the wave front is perpendicular to the radial cracks formed in Stage 1. Even though the cracks extend to the surface, they are not instrumental in breaking loose much rock.

Consequently, crater blasting results in a small amount of fragmented material and a large volume of rock mass which has been cracked in all directions. This latter point is of critical concern when considering blasting effects on the ultimate stability of an excavated face. Therefore, crater blasting must always be avoided when trying to establish a final wall or slope in rock.

In general, crater blasting is a highly inefficient use of explosives and borehole length. In test blasts with identical charge sizes and the same distance to the nearest free face, bench blasting removed 10 times the volume as did crater blasting.\(^3\) See Figure 3.1-7. It's only practical application is for sinking cuts or ramp cuts.

3.1.3 Pre-Shear Blasting:

Falling into neither of the two basic configurations examined so far, this important blasting technique deserves a separate discussion.

Blastholes for a pre-shear line are like those in a crater blast with the only free surface at the mouth of the blasthole. However, pre-shear blasting requires the blastholes to be much closer together for maximum control of the rock fracturing action and to enable the use of lighter explosive charges.

Up to this point, the discussion of rock mass detonics has dealt with the modes of action around single blastholes. The unique mechanics of pre-shear blasting involves the interaction of multiple blastholes detonated simultaneously to produce a single, clean fracture along an entire line of
FIGURE 3.1-7: COMPARISON BETWEEN REMOVED VOLUMES IN BENCH BLASTING AND CRATER BLASTING WITH IDENTICAL CHARGES AND BURDENS

(after Persson, Lundborg & Johansson)
pre-shear holes. Clearly, the necessary tensile forces must be created in some way other than reflection from a free face. For ease of illustration, the interaction between two pre-shear holes is examined.

When the two shock waves simultaneously travel out from the blastholes, the rock at the mid-point between the holes is subjected to their combined compressive forces. As described by the laws of elasticity, this very high compression sets up tangential shear stresses which cause an initial fracture to form on the pre-shear line. See Figure 3.1-8.

As the two shock waves meet, their compressive components parallel to the pre-shear line oppose each other while the normal components combine to form a tensile force perpendicular to the pre-shear line. See Figure 3.1-9. This tensile force acts on the initial fracture and causes it to rapidly propagate in both directions, intercepting the original blastholes.

Since the fracture process employs the summation of forces from two compressive shock waves, the force in each wave can be below that required for the threshold of rock fracture. Thus, small charges in each blasthole are sufficient to produce the single pre-shear fracture while not shattering the final rock face behind the intended line of excavation. See Plate 7.

For this reason, pre-shear blasting is ideal for producing clean stable faces in many rock types when used in conjunction with controlled bench blasting techniques.
FIGURE 3.1-8: INITIATION OF PRE-SHEAR FRACTURING MECHANISM ON AN ELEMENT OF ROCK BETWEEN TWO SIMULTANEOUSLY DETONATED BLASTHOLES
FIGURE 3.1-9: PROPAGATION OF PRE-SHEAR FRACTURE DUE TO THE INTERACTION OF TWO SHOCK WAVES
Pre-Shear blasting created a single fracture plane to produce this clean, smooth rock face along a highway. The presence of intact half-blastholes indicates a minimum of damage to the rock behind the pre-shear plane.
3.2 THE INFLUENCE OF ROCK MASS PROPERTIES ON THE ROCK BREAKAGE PROCESS

The important processes in rock blasting are reasonably well understood and are relatively simple within homogeneous materials such as plexiglass or large rock blocks. However, in real rock blasting, the situation is much more complicated and multi-facetted. There are many different types of rock blasting operations in which any one of the different simple mechanisms may predominate.

A real rock mass is usually strongly inhomogeneous and anisotropic with regions of high and low strength within the fracturing range of one blasthole. The rock normally contains a complicated network of both large and fine cracks and fissures. Although most of these will be naturally occurring structures in the rock, some may have been produced by a previous blasting round or the neighbouring charge in the same round. A rock mass may have planes of easy fracture oriented in any conceivable way relative to the free face and the axis of the blasthole. The free face itself will normally be quite irregular. A further consideration is the combined effect and interaction of charges in several holes, fired simultaneously or at delayed time intervals.

Clearly, the basic knowledge presented on blasting mechanisms up to this point is not sufficient for a complete and detailed understanding of detonics in a real rock mass. Although the effects of individual rock mass features are becoming better understood, it is doubtful whether a comprehensive and interactive rock mass detonics model will ever be developed. Perhaps this is why blasting has been regarded as an art rather than a science for so long. Undoubtedly, many of the old "powder monkeys" were artists in their own right, subconsciously taking every advantage of known joints and headings within their individual quarries.

To date, the most successful scientific efforts have been in empirical rather than theoretical relations, based on real rock blasting experiments.
and incorporating those major features of the process known to be important. Through this type of work, the following rock mass properties have been identified as those having significant effects on blasting mechanisms.

3.2.1 In-Situ Dynamic Rock Strength:

Although the testing of rock strength has been done as standard procedure for many decades, this heading points out two very important qualifications in determining strength values which will be meaningful for blasting.

Upon detonation of the explosive charge, the rock at the blasthole wall is subjected to the intense detonation pressure over a period of about 0.05 milliseconds. Since the strength of rock increases with the rate of loading, it is important to realize that neither standard Uniaxial Compressive Strength tests nor Triaxial tests can possibly produce a useful strength value for rock under such a dynamic loading rate. Unfortunately, a simple correlation is not available since the rate at which strength increases with loading rate varies with rock type. Weaker rocks and those with lower tensile strengths show a greater relative strength increase with increasing loading rate. To date, strength/loading rate relationships have not been accurately defined, especially for loading periods less than 1 millisecond. One of the major problems in blasting optimization lies in the uncertainty of the properties and behaviour of rocks under the extremely dynamic conditions associated with explosive attack.

A sophisticated technique for obtaining dynamic rock strength has been developed in Switzerland by Young and Dubugnon. Their investigations used a reflected shear wave to study the dynamic failure of limestone under controlled laboratory conditions with careful instrumentation.

However, any valid rock strength testing should be made in-situ rather than in the laboratory. It is absolutely impossible for a reasonably sized laboratory specimen to properly represent the volume of rock within a blast-
hole's zone of influence. By making in-situ measurements, the effects of weathering or of any observed structure on the rock strength can be assessed. If rock specimens were transferred to a laboratory, the weakest portions would fall apart so that only the strongest parts of the rock would get tested.21

The in-situ dynamic strength of rock depends on more than just the strength of the intact rock. A strong rock type can be significantly weakened by weathering, groundwater, alteration, the presence of structural discontinuities, or fractures due to previous blasting.

The in-situ testing of rock types should consist of determining both the dynamic tensile strength and the dynamic compressive strength. The dynamic compressive strength is determined by setting off charges having various borehole pressures ranging upwards from the static compressive strength of the rock. The burden/hole diameter ratio should be roughly the same as that to be used for the designed blast, since the apparent compressive strength of the rock mass varies with the burden/hole diameter ratio. The in-situ dynamic compressive strength of the rock mass will be equal to the maximum borehole pressure which does not cause crushing around the blasthole.

Since tensile fracturing usually accounts for the greater part of fragmentation, the dynamic tensile strength is also important, especially for pre-shearing calculations. Its value is determined by drilling off several sets of holes at various spacings. The largest spacing which still allows a good pre-shear crack to form between the holes can determine an accurate figure for the dynamic tensile strength by substituting into the following equation:21

\[
T = \frac{r \times BP}{500 \times S - r}
\]

where:
- \(T\) = Dynamic Tensile Strength (MPa)
- \(r\) = Borehole radius (mm.)
- \(BP\) = Borehole pressure (MPa)
- \(S\) = Borehole spacing (metres)
3.2.2 Presence of Structural Features:

One of the most obvious features of a rock mass is the presence of such structural discontinuities as faults, joints and bedding. When these features intercept a blasthole or pass near to one, they will radically influence the sequence of events outlined in Section 3.1. The influence of discontinuities will primarily depend on their spatial density, their continuity through the rock mass, and whether they are tightly closed or open. The type of infilling, if any is present, will also have some effect.

Upon detonation, a joint passing through the blasthole will tend to act as an extension of the hole. The initial high pressure gases will flow into the joint, creating considerable crushing at the blasthole wall and causing pneumatic wedging to preferentially expand the joint. See Plate 8. If the joint is very continuous, the wedging effect can cause disruption for a great distance in all directions. At the same time, gas leakage into the joint will reduce the pressure available in the blasthole for the normal fragmentation process.

When bedding or nearly horizontal joints occur in bench blasting, this pneumatic wedging effect is often responsible for unexpectedly large distances of throw. Due to the pre-existing planes of preferential gas travel, the rock will separate along these planes resulting in blocky fragmentation. Finer fissures through the blasthole wall will not necessarily cause such effects as they may close up under the large axial and tangential pressures in the early stages of shock propagation.

Large, open joints parallel to the blasthole may cause reflection of the shock wave before it reaches the free surface. This will result in more intense fragmentation of the material closer to the blasthole and even internal scabbing at the joint. The radial cracks will not cut through the joint opening. Instead, the gas pressure will expand into the joint and cause unexpectedly large break-out boulders which may reach far beyond the intended excava-
This exposed blasthole clearly reveals the effect of an intersecting joint. Note the shape of the crushed zone and the distinctly open joint caused by the pneumatic wedging of escaping high pressure explosion gases.
tion limit.

Narrow or tight joints parallel to the borehole may be too thin to cause reflection of the shock wave but will separate under the influence of the returning tensile wave. This may reduce the interaction with the established radial cracks and can prevent the rock from breaking out around the blasthole.³ See Plate 9.

If a joint has been infilled with different materials, it provides an acoustic impedance discontinuity.¹ The more nearly the impedance of the infill material matches that of the adjacent rock, the greater the energy transmitted and the lesser refracted or reflected.

The orientation of structure relative to the bench face or pit slope greatly affects the influence of the blast on the final wall. When the structures are horizontal or parallel to the final wall, a clean smooth face can readily be achieved. Problems usually arise when the structures are undercut by the excavation. Even in good blasts which create clean faces and no excess fractures, structures dipping steeply out of the wall will ultimately control the appearance of the final face. See Plate 10. For this reason, it is important to scale a final face immediately upon completion of muck excavation. Such problems of undercutting structures should be avoided by proper design of the pit slope.

Although many open pit mines may have large portions of the pit within one rock type, the blasting characteristics will change around the pit perimeter as the orientation of the wall shifts relative to the principal discontinuities. This was highlighted in a study at the Smallwood Mine where the blasting engineer worked in conjunction with the pit geologist in a series of blasting tests.²²

The joint spacing or density of natural fractures is another significant factor in a blast. Closely spaced joints in a hard rock type will cause it to behave like a low strength material. Within some orebodies and highly
The effect of a major joint parallel and adjacent to the blasthole can be seen at the centre of this photo. The reflected tensile wave interacted with the joint to cause overbreak along the well defined joint plane. Premature gas release into the joint reduced the blasthole pressure such that a substantial portion of rock along the blasthole failed to break out.
The presence of non-shattered half-blastholes and several smooth shear faces indicate that this rock face was not overblasted. This photo illustrates that steep, outward dipping rock structure will ultimately control the final appearance of a rock face. In this case, the primary concern in the blast design is to keep such overbreak to a minimum.
sheared fault zones, the rock is already dissected to an acceptable fragmentation gradation requiring only the gas pressure action of the blast to loosen the rock.

Much of the rock at Bougainville Copper Mine falls into this description. Since the spatial density of the jointing appeared to be a major variable in blasting around the pit, a relationship was established by John Ashby between fracture frequency observed in core and blasting powder factors. See Figure 3.2-1. It must be noted that such a correlation is site specific since it involves only one of the many rock properties. A separate set of correlation curves would be required for each different rock type. Although there is no consideration for the relative orientation of structural features to the pit wall, the density of natural fragmentation is such that this aspect becomes inconsequential.

Ashby's approach in establishing an empirical correlation with a dominant rock mass feature may be one of the most effective means currently available to overcome the limited understanding of complex structural influences on rock mass detonics.

3.2.3 Poisson's Ratio:

The nature of failure induced by blasting changes with the elastic constant of the rock. Rocks possessing the lowest Poisson's Ratio (σ) fail directly by a brittle fracture process while those having a high value fail mainly by plastic means.

Dividing rocks into those that normally exhibit plastic or brittle behaviour may be misleading because the nature of failure depends upon the loading rate. At a high loading rate, brittle failure is possible even in materials that normally deform plastically. Both the static and dynamic values of σ increase with confining pressure with the dynamic value being the lower of the two.
POWDER FACTOR
Gravity Loaded ANFO
Blasting Gelatine

<table>
<thead>
<tr>
<th>kg/m³</th>
<th>kg/tonne</th>
<th>100 lb/ton</th>
<th>kg/m³</th>
<th>lb/yard³</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.5</td>
<td>0.8</td>
<td>1.3</td>
<td>0.6</td>
<td>1.0</td>
</tr>
<tr>
<td>0.6</td>
<td>1.0</td>
<td>1.7</td>
<td>0.5</td>
<td>0.8</td>
</tr>
<tr>
<td>0.7</td>
<td>1.1</td>
<td>1.1</td>
<td>0.4</td>
<td>0.6</td>
</tr>
<tr>
<td>0.8</td>
<td>0.9</td>
<td>0.9</td>
<td>0.3</td>
<td>0.4</td>
</tr>
<tr>
<td>0.9</td>
<td>0.8</td>
<td>0.8</td>
<td>0.3</td>
<td>0.3</td>
</tr>
</tbody>
</table>

*\(Y = 2.5\) tonnes/m³

Fractures/meter

\[ \text{Powder Factor} = \frac{0.56Y \tan (\phi + I)}{\text{Fracture Frequency}} \]

where \(Y\) = insitu density of rock
\(\phi\) = basic friction angle
\(I\) = roughness angle

Fractures/foot

**FIGURE 3.2-1** : EMPIRICAL RELATIONSHIP BETWEEN POWDER FACTOR, FRACTURE FREQUENCY AND JOINT SHEAR STRENGTH AT BOUGAINVILLE COPPER

(after Ashby²³)
For this reason, the velocity of detonation of the explosive and the peak blasthole pressure should be increased as Poisson's Ratio decreases in the 0.5 to 0.2 range to achieve acceptable fragmentation. The approximate range for $\sigma$ for most common rock types is 0.2 to 0.3.$^{19}$

3.2.4 Young's Modulus:

Both the static and dynamic Young's Modulus increases with confining pressure. The dynamic elastic modulus for some rocks could be up to 20% greater than that of the static case, even under moderate confining pressures. The effective Young's Modulus of a rock mass with open joints or cavities is less than its intrinsic Young's Modulus.

3.2.5 Internal Friction:

As the explosion-generated strain wave radiates out into the rock, some of its mechanical energy is converted into heat due to wave transmission resistance within the rock mass. This internal friction gives an indication of a rock's ability to attenuate strain waves. In massive rock bodies, fragmentation decreases with an increase in internal friction. In rock which exhibits a dense network of natural cracks and planes of weakness, the wastage of strain energy has been demonstrated by Lang$^{24}$ where fragmentation depends almost entirely upon the gas pressure action of the explosive.

Internal friction varies considerably with rock type, increasing with porosity, permeability and jointing of the rock.$^{19}$ Attenuation increases considerably across a fault or shear zone. Internal friction parallel to the bedding can be half that normal to the bedding. If pores are filled with water, the maximum pore pressure rise during passage of the strain wave is several times greater than if they contain gas.

Unfortunately, most internal friction values have been determined in the laboratory using rock cores. In-situ determination of strain wave attenuation would provide a much more accurate indication of a rock's susceptibility
to strain wave fracturing mechanisms.

3.2.6 Rock Density:

The densities and strengths of rocks can usually be well correlated. In general, low density rocks can be most easily deformed and require relatively low powder factors for successful fragmentation. Dense rocks usually require fully coupled powerful explosives, especially if such rocks have a massive structure.

Even if the rock is satisfactorily fractured, the broken mass requires a minimum displacement for good diggability. Denser rocks will cause a decrease in muck displacement simply due to the increased load of the burden. This can be compensated for by increasing blasthole diameter, reducing the pattern, or changing to an explosive with a higher borehole pressure.

For vertical blastholes which are deep and/or in very dense rock, the weight of overlying rock will impede muck displacement and may cause problems at grade level such as unbroken toes and high floors.

3.2.7 Seismic Wave Velocity:

The seismic wave velocity represents the speed at which the shock wave travels through the rock mass. It is usually used in-situ on geophysical surveys to determine depth of overburden or broken rock.

However, the seismic wave velocity also reflects some of the other important rock mass properties. Rocks which have a higher seismic velocity are generally observed to be stronger, while those which are weakened by weathering, alteration, fracturing due to dense jointing or previous blasting have a lower velocity.

In research work at Kennecott Copper Corporation, Carl Broadbent\textsuperscript{25} developed a relationship between seismic wave velocities and powder factors required to achieve acceptable fragmentation. See Figure 3.2-2. This is similar to the empirical approach used by Ashby\textsuperscript{23}, but seismic velocity may be the better
FIGURE 3.2-2: EMPIRICAL RELATIONSHIP BETWEEN POWDER FACTOR AND IN-SITU SEISMIC VELOCITY AT KENNECOTT COPPER

(after Broadbent)
rock mass property for correlation because it is affected by several other properties. This relationship is probably not universal, but it should apply within geologically consistent areas.²⁵

3.2.8 Water Content:

In most rocks, water saturation markedly increases the velocity of propagation of elastic waves owing to the filling of pores and fissures with water. Thus stresses and water content are inherent characteristics of a rock mass which reduce the effects of jointing on the elastic properties of the solid rock.

Fluids in a porous rock decrease friction between grain surfaces, decreasing the yield strength. Increased water content appears to reduce energy absorption and thus makes breakage by explosive energy easier. Unfortunately, the attenuation of the explosion-generated strain wave by the water annulus around the charge tends to eliminate the benefits of water within the strata.¹⁹ The advantages of water within the rock mass will only be realized with fully coupled explosives having specific gravities greater than 1.0.

3.2.9 In-Situ Stress:

Although this feature is most commonly encountered in underground blasting, it has been significant in some surface blasting operations.

The presence of pre-existing stress cracks crossing the blasthole is one effect of high static stresses. These influence the blast performance in the same way as other structural discontinuities.

The second effect is due to the stress concentration itself. In the presence of a gravitational and/or tectonic stress field, the fracture pattern is influenced by the non-uniform stress condition around the blasthole. Some of the cracks which start to grow in radial directions may eventually curve off into the direction of the static stress field. This phenomenon is most visible when attempting to establish a straight pre-shear fracture as invest-
igated by Nicholls and Duvall\textsuperscript{26}. See Figure 3.2-3. When the stress field acts in a direction normal to pre-existing cracks, it can be sufficiently intense to prevent extension of these cracks, but at the same time, assist in forming long new cracks in the direction of the stress field.

It should be appreciated that such ground stresses can exist only well within the rock mass. There will be a surface layer of fractured, largely stress relieved rock over the tops and faces of all benches. Front row burdens will probably not be influenced by even high stress fields but the effect must be expected to increase towards the back row of the blast pattern.
FIGURE 3.2-3: THE INFLUENCE OF AN IN-SITU STRESS FIELD ON A PRE-SHEAR LINE WITH VARIOUS BLASTHOLE SPACINGS

(after Nicholls and Duvall)
3.3 REFERENCES


CHAPTER FOUR

THE INFLUENCE OF BLASTING ON SLOPE STABILITY
4.0 THE INFLUENCE OF BLASTING ON SLOPE STABILITY

Designing for optimum slope stability is one of the most important applications of rock mechanics to surface excavations. During the past two decades, rock slope engineers have been able to design stable, economical excavations of ever-increasing scale and complexity due to advances in geologic data gathering, core recovery and orientation, shear strength testing of discontinuities, and monitoring of pore water pressures. A few engineers have also attempted to use statistical means for incorporating uncertainty into slope stability analysis. Through the improved capacity and availability of desktop microcomputers, it is now possible to easily handle large volumes of geological and hydrological data, to produce, with confidence, slope angles of optimum stability within ranges of a few degrees.

Many of the rock mass properties incorporated into a good slope design are those which have been determined from careful laboratory testing of intact rock specimens. If a rock slope engineer is to produce a truly optimal slope design, then he clearly must depend on these rock mass properties being present within the final rock slope. Any subsequent changes in these properties contributing to an overall decrease in the competency of the rock mass will result in a less stable rock slope.

Unfortunately, most engineers involved in rock slope design are not familiar with blasting mechanics and fail to realize the important influence of blasting on the stability of their designed slopes. Even good blasting can affect the inherent rock properties and may lead to minor changes in anticipated slope stability. At best, the blast designer can only hope to contain the blast damage within the mass of rock to be excavated and to minimize any deleterious effects in the surrounding rock. At this point in time, the ability of engineers to design stable rock slopes is more advanced than the ability to construct them when blasting is required.
There are two principal means by which blasting can adversely influence slope stability. The most obvious is through insufficient control of the physical explosive forces and the expanding gases, creating excessive disruption of the rock mass. The other effect is that of substantial ground vibrations caused by detonating a large quantity of explosive in a single instant. These two types of blast induced damage are discussed separately.
4.1 EFFECT OF PHYSICAL BLASTING FORCES

Highly destructive forces are necessary when attempting to fragment and displace rock, but they must be constrained or controlled through the application of special blasting techniques in order to minimize their effect beyond the perimeter of the desired excavation.

In the previous chapter, the behaviour of blasting forces and their physical effects were discussed with respect to their ability to produce well fragmented muck, particularly near a free face. This section examines the effect of those same forces on the final rock slope and their influence on ultimate slope stability. Although the entire perimeter blast pattern must be correctly designed and fired to produce minimal wall damage, attention will be focussed on the row of line holes which form the critical interface between the muck and the final wall.

One of the difficulties in achieving both good muck and an intact face is the fact that initial fracturing mechanisms extend radially in all directions from the blasthole. Since sufficient explosive is required to break the line hole burden, it is inevitable that some cracking of the final wall will take place adjacent to the blasthole positions. This increase in fragmentation of the rock slope surface reduces the competancy of the final face and can lead to instability. The degree of damage will depend on the amount of explosive in the blasthole, its distribution, and its fracturing ability (i.e. velocity of detonation).

If properly controlled, this fracturing does not usually extend for significant distances into the face, resulting in a ravelling type of failure as opposed to large scale mass failure. Although ravelling may not represent a major loss of overall wall angle, it still presents a serious safety hazard to men and equipment working below. Even a small piece of rock can gain sufficient speed on a high pit slope to do serious damage. For this reason, it is good practice to scale any final rock faces to remove weakened material
before it becomes a hazard.

If the blasthole contains a large amount of explosive, the cracks can easily extend far into the rock face to intersect existing natural discontinuities. This deeper penetration produces more blocky failures which cannot be removed by scaling, presenting a more serious problem. Such damage is likely to cause the loss of bench crests, reducing the effective capacity of the bench to catch material falling from above.

A concentration of excessive cracking in the rock slope surface not only weakens the face from a structural aspect, but leaves it open to further degradation by the subsequent weathering actions of water and ice. Water from both precipitation and subsurface seepage collecting in open fractures is well known for its ability to seriously reduce slope stability. In Canada, and other countries with cold winter climates, a freezing front can penetrate into the rock for considerable distances. The water freezing in the near-surface fractures causes "ice-jacking" and can result in a significant amount of spalling and rockfall. Over a period of several years, this phenomenon can cause large volumes of rock to spall, gradually reducing a well-benched pit wall into a relatively continuous rubble slope and reducing the overall slope angle.

Although some rock fracturing is unavoidable simply due to the charge proximity, several steps can be taken in the blasting design to minimize this effect. A common approach is to increase the spatial dispersion of the explosive along the line hole row. By doubling the number of line holes, each blasthole will contain only half the usual amount of explosive for the same powder factor. This principle is most clearly seen in pre-shear blasting where very close hole spacing is used for maximum face control. The explosives user must ensure that the spatial distribution of the explosive is proper for the soundness and smoothness of the final surface that is desired.²

Minimizing rock face damage can also be achieved through selecting an
explosive with the lowest velocity of detonation capable of producing the desired fragmentation. The higher velocity explosives are only necessary where rock structure is massive and more fracturing is required.

Further improvements can be made by de-coupling the charge from the blasthole wall. By leaving an annulus of air between the explosive and the rock, the detonation wave is partially attenuated before it is transmitted into the rock, reducing its shattering effect. On this research project, some experiments were made on fully coupling the charge to the front or muck side of the blasthole, thereby increasing the degree of de-coupling from the back of the blasthole. This work is detailed in Section 6.3.2.

Immediately following these fracturing mechanisms around the blasthole, the rapidly expanding high pressure gases provide the other major physical blasting force. These pressures are usually the more serious problem, even if the face fracturing is under control, because they can disrupt the rock mass for considerable distances into the final wall.

The effect of the expanding gases is most serious in a rock mass with a high natural fracture frequency and very continuous joint planes. These discontinuities provide planes of easy travel for the gas pressures, causing pneumatic wedging to preferentially open and extend the joints. This action causes rock block separation and can result in a total loss of cohesion along the joints, an important factor in maintaining slope stability. Since cohesion is a difficult value to assess, rock slope engineers may be prudent to assume a zero value for bench design when they cannot be confident about the quality of the blasting operations.

Another critical stability factor, the angle of frictional sliding resistance ($\phi$), can also be adversely affected by the disruptive forces of the expanding gases. Inter-block shearing may cause $\phi$ to shift from its peak toward its residual value, or $\phi$ could be virtually eliminated through direct loss of inter-block contact at shallow depths.
Due to the larger zone of influence of the penetrating gas pressures, rock slope damage can occur well within the final wall, causing large scale and moderately deep-seated pit slope failures. If such damage weakens an area at the toe of an existing high pit wall, subsequent instability could involve the full height of the rock slope. As mentioned in the case of face fracturing above, newly expanded joints may permit groundwater and seepage forces to further contribute to a reduction in stability.

Such a situation occurred at the start of the research work at Afton Mine while the old blasting practices were still in use. A large pit wall failure was triggered by a single blast when entrapped gas pressures travelled over 40 metres along a major shear plane, opening it up, breaking cohesion, and causing strength loss by planar shearing. Subsequent inflow of seeping groundwater worsened the situation and accelerated the slide mass which eventually grew to involve a 35 metre high portion of the pit wall and extended far beyond the intended pit perimeter. See Plate 11. The extra cost of monitoring, re-stabilizing and extra waste stripping was ample justification for embarking on a blast improvement program.

The key to preventing rock slope damage from expanding gases is to allow release of the excess pressures as soon after detonation as possible, before they can travel into the rock mass for any significant distance. This is best achieved by making sure that blast pattern burdens, especially on the line hole row, are small enough to prevent the blast from becoming "choked". By providing full release of the rock burden on a hole, the excess gases will naturally vent forward into the muck and not back into the final slope. This reinforces the importance of always blasting to a free face instead of crater blasting, especially on perimeter patterns.

Further improvements can be made by lightening or eliminating the stemming in the blasthole, thereby providing full gas pressure venting immediately after the necessary strain pulse has been transmitted into the rock. This
Tension cracks outside the pit perimeter reveal the extent of the large pit wall failure initiated by careless blasting. Although triggered by gas pressure damage, continued movement was caused by a combination of high groundwater levels and blasting vibrations.
may leave slightly tighter muck in front of the final face, but it should still be diggable and provide a cleaner final rock slope.

Through understanding the geological site conditions important for maintaining slope stability, it should be possible to plan and execute a good perimeter blast pattern where the physical blasting forces will have a minimal effect on the final rock face. Unfortunately, very little practical work has been done on this problem to date, primarily due to the complex influences of geologic features. The importance of tailoring blast designs to in-situ rock mass conditions for optimizing rock slope stability forms the central theme of the field research and this thesis.
4.2 EFFECT OF GROUND VIBRATIONS

In recent years, the problems of physical blasting forces have been overshadowed by the amount of attention paid to studies on the vibrational effects of blasting. This is largely due to the fact that wave phenomenon can be scientifically monitored and understood through the known laws of physics. Unlike physical blasting effects, vibrational wave effects are somewhat independent of the frequency and orientation of discontinuities, making it slightly easier to model their influence on slope stability within any single rock type.

In the case of the physical forces, the concern for rock mass damage focussed mainly on the rock immediately adjacent to the blast which was to become the final slope. The effects of ground vibrations cover a much larger area around the entire blast zone because of the shock wave's ability to travel over great distances through a solid rock mass while maintaining much of its source energy. Thus, a relatively minor blast may adversely affect the nearby existing slopes and possibly lead to problems if they are already in a state of marginal stability. Furthermore, the convoluted geology common to the ore zones of many open pits can affect the complex reflections and refractions of waves such that they can focus in unexpected locations.

After the primary shock front has passed beyond the shattered and fractured zone of rock around the blasthole, it travels in the form of elastic waves and vibrations. This energy takes on different forms which travel at different velocities and cause different types of deformation to occur in the rock. The fastest wave, a Primary or P wave, is a compressive wave deforming the rock in a radial direction. The slower Secondary or S wave is a shear wave causing deformation of the rock at right angles to the direction of wave travel. The P and S waves are both known as body waves because they travel within the rock mass and are responsible for any internal vibrational damage.

When the body waves arrive at ground surface, new waves are generated
including a group which travels along the surface. These surface waves, which are slower than either of the body waves, are known as Rayleigh or R waves. Their motion is quite different from that of the body waves, being characterized by larger amplitudes, lower frequencies, and a lower propagation velocity. The R waves are very important since they propagate along the surface of the earth and because their amplitude decays more slowly with distance travelled than either the P or S waves. This is especially true if the groundwater table is close to surface and the rock mass is saturated. The R waves contain significantly more energy than the body waves and are of concern for vibration effects on surface slope stability.

The wide variation in geometrical and geological site conditions preclude the solution of ground vibration problems by means of elastodynamic equations. The most reliable predictions are given by empirical relationships based on the results of several monitored blasts. An early study of this nature was made in 1957 by Edwards and Northwood during construction of the St. Lawrence Seaway. Of all the variables examined, they found that the peak particle velocity of the ground motion correlated best to building damage. Peak particle velocity is the maximum velocity of particle motion during the passage of the seismic wave beneath the particle.

This discovery prompted the U.S. Bureau of Mines to review its own records, resulting in the same conclusion and in the development of the following scaling law for particle velocity versus scaled distance between the blast and the point of interest:

\[ V = K \times \left( \frac{d}{W^{\frac{1}{3}}} \right)^x \]

where:  
- \( V \) = peak particle velocity (in./sec.)
- \( d \) = distance from shot to observation point (ft.)
- \( W \) = charge weight per delay (lbs.)
- \( K \) = constant depending on charge distribution and the material type (varies from 45 to 450)
and where: $x$ = variable depending on material type and whether the longitudinal, vertical or transverse component is being measured.

As a general rule, the longitudinal and vertical components are approximately equal and the transverse component is considerably smaller than the other two, so that usually the longitudinal or vertical velocities are used. According to the U.S.B.M., the exponent $x$ varies from -1.5 to -1.9 for these components with -1.7 being a good average.$^5$

If the charges were spherical, charge weight would vary as the cube of the radius of the sphere, theoretically dictating the use of cube root scaling. However, in most bench blasts the loaded blastholes are more representative of cylindrical charges with the charge weight varying as the square of the cylinder radius. Thus, it has been found convenient to use square root scaling for prediction of the widest range of blasting conditions.

Based on his analysis of several hundred thousand blasting vibrations, Oriard$^2$ has recently published a graph showing peak particle velocity versus scaled distance. See Figure 4.2-1. Expressed mathematically, this relationship is almost identical to the earlier U.S.B.M. version:

$$V = H x \left( \frac{D}{W^2} \right)^{-1.6} x k_1, k_2, k_3$$

where: $V$ = peak particle velocity (in./sec.)

$H$ = velocity intercept at unity scaled distance

$D$ = distance from shot to observation point (ft.)

$W$ = charge weight per delay (lbs.)

$k$ = factors representing the variations in explosives, confinement, spatial distribution, geology and other parameters of interest

The slope of -1.6 represents the attenuation. Although it is not the same at all sites nor the same for all wave types, this slope shows surprising accuracy for most situations.$^2$

With increasing spatial dispersion, this plot shows a line which has
FIGURE 4.2-1: PEAK PARTICLE VELOCITY VERSUS SCALED DISTANCE

(after Oriard²)
a flatter slope and a lower intercept value. If the charge quantity is distributed over a large area, then the closest portion of the elastic zone can be adjacent to only a portion of the total charge per delay and the effects at close distances cannot reach predicted levels.\(^6\)

The effect of charge confinement is also noted on the graph. If a charge is buried too deeply to break out to surface, the explosive charge will produce maximum vibration. If the charge is too close to the surface, energy release into the atmosphere produces less energy in the form of elastic waves. This further reinforces the importance for blasting to a free face and using minimal burdens in order to avoid backslope damage.

Beyond these general relations, it is an unfortunate fact that the plot of Figure 4.2-1 is still inadequate for predicting peak particle velocities for blast planning purposes. Although based on an unusually large amount of field data, it cannot account for the highly site specific conditions existing on any individual blasting problem. The complex geological setting which acts as the wave transmitting medium has a strong influence on the attenuation, frequency and displacement characteristics of a seismic wave. For example, hard massive rock will be characterized by smaller displacements and higher frequencies, whereas soil will be characterized by larger displacements and lower frequencies.\(^6\) Site conditions can also affect seismic wave attenuation by geometric spreading, selective scattering, absorption and dispersion.\(^2\) Both the attenuation and the wave form characteristics are influenced by such geological factors as layering, jointing and water content, as well as the small scale elastic properties of the medium. In certain regions underlain by prominent horizontal layers of sedimentary rock, it has been noted that surface waves appear to be more prominent and persist to greater distances than is typical for regions that are more heterogeneous and geometrically complex. For these reasons, the peak particle velocities induced by blasting cannot be realistically predicted and should be measured by on-site monitoring.
to obtain the true values.

A further difficulty arises from interpreting the practical implications of peak particle velocity values. Although it has long been acknowledged that they generally correlate to rock mass damage, there is very little hard data dealing with any specific relationships. Perhaps the best work done in this respect is the oft-quoted research by Langefors and Kihlstrom who produced a general indication of damage thresholds in relation to charge weight per delay and distance from the blast. See Figure 4.2-2. This plot was developed from the U.S.B.M. equation using values of $K=200$ and $x=-1.5$ based on guidelines by Oriard. It must be noted that this figure should only be used for very general guidance. When vibration damage is a serious problem for a particular mine site, proper values of $K$ and $x$ must be determined from field measured peak particle velocity data.

However, the general relation of Figure 4.2-2 is sufficient to illustrate one of the easiest and most efficient ways to control excessive blasting vibrations. This figure indicates that the detonation of 200 kilograms of explosive per delay will cause rock breakage up to 20 metres from the blasthole while the detonation of 2000 kilograms per delay will extend this distance to approximately 70 metres. Clearly, the use of an increased number of delays to limit the amount of explosive detonated at any one time is an extremely important and simple method of limiting damage to the final rock slope.

The important inter-relationships between the number of delays, the level of vibration, and the movements of an unstable slope were well demonstrated in an extensive 3 year study at the Da Ye Mine, one of the largest open pit operations in the People's Republic of China. One of the several experimental methods examined for stability control involved increasing the number of delays within a consistent perimeter blast pattern. The results, shown in Figure 4.2-3, clearly indicate the effectiveness of decreasing the charge per delay. Through repeated testing, peak particle velocity measurements at the unstable slope
FIGURE 4.2-2: PARTICLE VELOCITIES AND DAMAGE INDUCED AT GIVEN DISTANCES BY PARTICULAR CHARGES

(after Langefor's & Kihstrom)
FIGURE 4.2-3: THE EFFECT OF INCREASING THE NUMBER OF DELAYS FOR REDUCING BLASTING VIBRATIONS AT DA YE MINE, CHINA
(after Liao & Qin)
indicated the critical threshold of movement and determined the maximum permissible charge per delay.

Delays are such important elements in controlled surface blasting that they are now readily available in a broad variety of periods and types. Despite their increased availability, many mining operations still fail to take advantage of them, largely due to concerns about their extra cost and the additional time required to tie them into a blasting pattern. It is also important to select the correct type of delay for the particular site conditions. In many cases where sufficient delays are being installed, inappropriate periods are being used due to lack of attention to rock motion and displacement characteristics. The full benefits of a well delayed blast can only become apparent after being properly applied into a well designed blasting pattern.

A good case study at Kennecott Copper Corporation examines the shifting from a time-consuming and expensive pre-shear blasting system to a multiple delayed buffer blast with economical savings.9 After application of the new system, the cost of delays was found to be minimal compared to pre-shearing and the same minimal wall damage was achieved by reducing the vibrational shock energy transmitted into the surrounding slopes.

A further example can be found at the Adams Mine where the introduction of a delayed pattern became an essential part of mining.10 Preserving stability and steep walls with reduced overbreak and reduced stripping paid for the system cost many times over.

As discussed in this section, many site specific variables have complex influences on blast induced vibration levels, making it difficult to fully understand the kinematics involved. However, a detailed scientific understanding is not really necessary in order to control the vibrational effects which adversely influence slope stability. When used in combination with an appropriate controlled blasting technique, to be discussed in the next chapter, the availability of a wide variety of delay devices permits a good blaster
to eliminate such stability problems by simple lowering the maximum charge per delay.
4.3 REFERENCES


CHAPTER FIVE

DESIGN CONSIDERATIONS FOR CONTROLLED BLASTING
5.0 DESIGN CONSIDERATIONS FOR CONTROLLED BLASTING

The previous chapters have presented a fairly complete summary of the different fields forming the basic components in the science of blasting. With a thorough understanding of all the aspects discussed, and an appreciation of their inter-relationships, a blasting engineer has all the technical background necessary to produce a good, optimal blasting design. However, the principal challenge has been, and will probably continue to be, the tailoring of the blast design to in-situ geological conditions when provided with very limited pertinent site data. It is this aspect which introduces the "art" of blasting into the overall design. Combining both the "science" and the "art" of rock blasting can be a complex procedure and is generally performed best by those blasting engineers with the broadest field experience.

Apart from the site specific problems, a blast design is composed of several parameters and variables, many of which are inter-dependent. These are discussed in this chapter. Although they are presented in a certain sequence, this is not to suggest that this is the only way to undertake a blast design. Various blasters will have their own approach to the problem and the sequencing of this chapter simply reflects this author's preference. It would be difficult to develop a universal formula approach due to the way in which site variations alter the importance of different design parameters. No matter which approach is used to arrive at an initial scheme, blast designing is an iterative process where constant adjustments can be expected as field conditions and site plans change. The key is to get as close as possible to the optimal design at the start so that any subsequent changes are minor.

The sequence of blast design steps presented here is that used by the author when approaching a new site with little or no information available on the blastibility characteristics. The premise of this system is to calculate the maximum possible charge per blasthole, and then to plan the blasthole
pattern based on a powder factor value for the rock mass. This approach generally produces the most economical blast design through full utilization of each blasthole and by keeping the drilling costs to a minimum. However, the success of this approach depends upon some prior knowledge of a suitable powder factor for the particular rock mass in question. To date this has been the main stumbling block to producing an optimal blast design and is why it has been selected as the focus of this research project. If the powder factor is known for a particular rock mass, the blast design procedure becomes an economic exercise in optimizing the balance between drilling costs and explosive costs.
5.1 SELECTING A CONTROLLED BLASTING TECHNIQUE

In order to minimize the damage to the final wall and to achieve optimum slope stability, it is essential that some form of controlled blasting be used for pit perimeter blasts. However, arbitrarily implementing any controlled blasting technique will not guarantee good results since each of the recognized methods has special, unique characteristics. It is important to carefully select the technique which is most appropriate for the site conditions and will produce the best results at the greatest economy.

Of the several factors which must be considered when selecting a blasting technique, perhaps the most important is the nature of the rock mass to be blasted. This would include the degree of jointing (from massive to highly fractured), the continuity of the joints, and the condition of the joints (closed, opened or infilled). The strength and elastic properties of the intact rock must also be assessed since hard, brittle rocks demand different blasting techniques from those employed in soft, friable rocks.

It is also important to consider the type of results which are desirable and most economic. Some of the controlled blasting techniques are capable of producing precise, sculptured rock faces, but require a lot of carefully drilled blastholes, adding significant cost to the project. While this degree of control may be necessary in some civil engineering applications, it can rarely be justified in a mining operation where a modified approach or a less precise technique may be more appropriate.

The type and size of drilling equipment available is another major consideration in the selection process. Certain techniques require fairly small blasthole diameters and the ability to drill the holes at precise angles. Drilling accuracy and speed may also affect the amount of rock which can be drilled off per shift. This may require the use of a different drill type from that used in the large scale production blasts, increasing the initial capital expenditure on equipment.
Depending on site location, other factors for consideration could include the presence of groundwater or seepage, the types of explosives that can be supplied, and the skill or experience of available blasting personnel. High groundwater levels or seepage rates will necessitate the use of blasthole liners or high density slurry explosives. More remote mining sites may have difficulty getting the special explosive products necessary for some controlled blasting methods, while a lack of trained personnel may make it difficult to implement a more complex blasting program. These factors can stringently limit the techniques which can be considered.

Each of the above mentioned factors has differing degrees of importance in the various blasting methods, but all must be considered together when making a selection. This section will now discuss the principal recognized controlled blasting techniques and will outline the conditions under which each of them can be most favourably applied.

5.1.1 Pre-Shear Blasting:

Pre-shearing or pre-splitting is a technique which is used very extensively and successfully in civil engineering excavations in competent rock and less commonly in open pit mining.

This technique involves drilling a line of small diameter holes to conform with the desired line of break. These holes are usually 65 to 125 mm. diameter, using tracked percussive equipment capable of drilling at the desired face angle. Generally, the recommended hole spacing is approximately 12 times the hole diameter, but closer spacing may have to be used where complex rock structure requires greater control.¹

It is also recommended that the explosive diameter be half the hole diameter and the load distributed so that only half the hole length is loaded. The charge should be decoupled from the rock by leaving an air space between the charge and the blasthole wall. This is usually achieved by placing the
charge in a tube of smaller diameter than the hole and centering this tube in the blasthole with some form of spacer.\(^2\)

The row of pre-shear holes is fired before the main charge and the reinforcing effect of the closely spaced holes, together with an essentially infinite burden, results in the formation of a clean fracture running from one hole to the next. The detonics principles of this action have already been discussed in Section 3.1.3. The main charge is then fired using an appropriate number of delays to prevent vibrational damage to the rock slope. Upon excavation, a good pre-shear face is characterized by a clean face running across the parallel half-blastholes as illustrated in Plate 7.

The extra costs involved in the denser drilling pattern of pre-shearing can be offset by the reduced excavation costs brought about by the elimination of overbreak. This also leads to the advantages of increased safety, savings on scaling operations and realization of the steepest stable slope angle.

In a mining operation, the requirement for small diameter angled holes may mean that additional drills have to be purchased just for the pre-shear holes. Although this is necessary for the best results, some mines are loathe to take on the costs of extra drilling equipment. If only the large diameter production blasthole rigs are available, a rough form of pre-shearing can still be carried out but some experimentation may be required to achieve an acceptable result. The main problem involves getting proper distribution of the light explosive charge within the larger hole. This usually involves the time consuming process of placing the explosive in smaller cardboard tubes which are centred in the blasthole. In general, pre-shearing with large diameter holes can only be attempted in competent rock and the characteristic pattern of half-blastholes is rarely visible on the final face.

For a number of years it was commonly thought that the pre-shear fracture protected the rock behind it from vibrations induced by the main blast. This is now known not to be the case since these vibrations are induced by compres-
sive strain waves which are unable to notice a pre-shear fracture unless it is greater than 10 mm. across. However, this fracture plane can provide a vent path for the expanding explosive gases, preventing radial cracks from propagating across the pre-shear line.

When pre-shear blasting is successful, the results can be spectacular. However, this has tended to result in an overselling of the technique into rock mass conditions which are not appropriate. Pre-shearing can actually cause more harm than good if applied in highly fractured rock masses, especially where the joints are open, continuous, and inclined to the pre-shear line. This is because the pre-shear blast is completely "choked" such that the expanding gases are trapped and will readily travel out into the joints. This weakens the final slope and can result in extensive overbreak because fracturing will follow the joints rather than the pre-shear line.

Generally, pre-shear blasting works well in any rock type as long as the structure is fairly massive with tight, well spaced joints. It should not be used in a rock mass with a high natural fracture frequency. Results can be equally good in both hard and soft rock if the proper explosive is used but the smoothness of the final wall actually depends on the drilling accuracy as much as any other factor.

Many mines presently employing pre-shear blasting are doing so with their full size rotary drills on reduced hole spacings. Although the end result is usually acceptable, much better control can be achieved with smaller diameter drills such as those used at La Cananea Mine in Mexico. Unfortunately, many mines look only at the drilling costs and not the slope stability benefits, rejecting the use of a small drill for pre-shearing. However, on sites where small diameter drills are in use, they are found to be valuable for a wide range of odd jobs in addition to their pre-shear drilling.
5.1.2 **Cushion Blasting:**

Cushion blasting, sometimes referred to as slashing or trimming, requires exactly the same blasthole pattern as for pre-shearing with closely spaced holes drilled along the final breakage plane. The only difference between the two methods is the detonation sequence.

In cushion blasting, the main portion of the perimeter blast is usually fired and excavated first. This leaves a greatly reduced burden on the final row of closely spaced holes, permitting the use of light, decoupled charges and rapid release of blasting energy. The firing of this final row trims off the remaining rock, producing a clean, low maintenance face very similar to that of a pre-shear blast. See Plate 12.

Cushion blasting can be applied in cases where the site is geologically unsuitable for pre-shearing but where good face control is required. It can be used in weak rock masses with a high intensity of natural fracturing because the excess explosive gases easily vent out into the light burden without damaging the backslope. As a further advantage, larger sized production drilling equipment can be used with reasonable success since it is not necessary to achieve the unique rock fracturing action of pre-shearing between the holes.

The principal disadvantage to cushion blasting in a mining operation is the difficulty caused in sequencing the drilling, blasting and excavating operations. The final row of holes often have to be drilled after the front portion of the blast has been fired, especially in weak or highly fractured rock, otherwise they would tend to cave or become cut-off. Also, sequencing for the shovel can be awkward since it must excavate most of the blast at one stage and then return a few days later to excavate the final trimming.

For these reasons, many mines have adopted a modified form of cushion blasting where the cushion hole row is detonated on the last delay of the main portion of the blast. However, if this approach is used, it is important to use a long enough final delay to allow a quasi-free face to develop on
Cushion blasting was used to create this clean, low maintenance rock face in the very weak and highly fractured Kamloops Volcanic formation.
the cushion row. Otherwise, this modified method will produce a similar result to good quality buffer blasting, but at lesser economy.

Thus, the scheduling headaches in an efficiently operated open pit with high equipment utilization will often disqualify true cushion blasting as a practical solution. However, this method will continue to be very valuable in the field of civil engineering and should not be overlooked when faced with isolated trouble spots of bad rock within an open pit operation.

5.1.3 Buffer Blasting:

Buffer blasting is the most simple and economical method of controlled blasting, accounting for its use throughout the mining industry. It does not require the closely spaced, angled, small diameter blastholes as in pre-shear or cushion blasting, thereby reducing drilling costs and speeding up the entire operation.

This method simply requires a modification to the burden, spacing and explosive load on the last row in the perimeter blast pattern. In this row, full size production holes are drilled vertically at some offset distance from the final digline. The aim is to reduce groundshock from the blast, but clearly there is a limit to the size of the offset distance before unacceptable digging conditions are created at the final digline. The burden and spacing for the buffer row are simply reduced to 0.5 to 0.8 times that of the adjacent production row. The final buffer row is fired with the rest of the perimeter blast, using fairly long delay elements to ensure that it fires last and towards a free face.

Essentially, buffer blasting is a form of controlled backbreak along the final row. The smaller burden and reduced spacing permits lighter charges for better control of the fracturing and little time is required before the optimum offset distance becomes apparent. Typically, the buffer holes are placed with their bases at or near the projected toe of the final slope.
The principal advantages of buffer blasting include the ability to drill vertical holes with the main production drills and the firing of the entire pattern in one stage instead of two. While the smoothness of the final face cannot be compared with that obtained by pre-shear or cushion blasting, this is usually compensated for by savings in drilling and blasting procedures. It may produce minor crest fracturing or backbreak, but the amount of damage is much less than would be produced by the main production blast if no controlled blasting was used at all.

Buffer blasting works well over a broad range of rock mass conditions, but extra care in design is required when working with hard or massive rock in order to avoid coarse fragmentation, high pit floors and excessive backbreak. Similarly, care is required when dealing with weak, incompetent rock which is susceptible to extensive backbreak and stability problems. When dealing with these extremes in rock mass conditions, buffer blasting may not be economical or will probably have to be used in conjunction with some other controlled blasting technique such as pre-shearing.

Generally, the flexibility in the design of a buffer blast method as well as its ease of implementation has made it popular throughout the world. Case studies have described its successful application at the Bong Mine in Liberia and some further development work has been done at the Adams Mine in Ontario where experiments have attempted to refine the technique.
5.2 BLASTING DESIGN PARAMETERS

A blast design is made up of several parameters which work together to produce the final desired results. However, changing any one of the parameters can drastically alter the outcome of the blast. This fact makes it difficult to truly optimize each parameter since the number of possible variations and combinations are far too numerous for actual field testing in each of the different rock zones.

This section will examine each of the blast design parameters in relation to their influence upon the blast behaviour and their influence upon the amount of damage to the surrounding, intact rock. These parameters are presented in a sequence which might be used when undertaking a blast design at a new site where no previous blastibility data is available. Refer to Figure 5.2-1 for an illustration of the terms and parameters discussed.

5.2.1 Blasthole Diameter:

In recent years there has been a tendency in open pit work to drill blastholes of increasing diameter. This tendency has been facilitated by development of large electric rotary machines, improvements in bit technology and extensive use of bulk, site-mixed explosives. The cost benefits to be gained by increasing the diameter of the blasthole are shown in Table 1. Although the cost per litre suggests that there is further benefit to be gained by going to very large holes, there is an upper limit. Since effective charge utilization is achieved with a burden at 40 times the hole diameter, large holes will result in a burden with the same dimensions as the bench height. This proportion is very inefficient and Persson suggests that the blasthole diameter be limited by the following relation:

\[
\text{Blasthole Diameter} < \frac{\text{Bench Height}}{40}
\]

Large diameter blastholes also contain too high an explosive concentration for proper damage control, leading to excessive fracturing of the remaining
FIGURE 5.2-1: BENCH BLASTING TERMINOLOGY AND DESIGN PARAMETERS
# TABLE I

## BLASTHOLE DIAMETER, VOLUME AND COST

<table>
<thead>
<tr>
<th>Hole Diameter</th>
<th>Hole Volume</th>
<th>Hole Cost* (in granite)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inches</td>
<td>Millimetres</td>
<td>litres/m.</td>
</tr>
<tr>
<td>1</td>
<td>25.4</td>
<td>0.51</td>
</tr>
<tr>
<td>2</td>
<td>50.8</td>
<td>2.03</td>
</tr>
<tr>
<td>3</td>
<td>76.2</td>
<td>4.56</td>
</tr>
<tr>
<td>4</td>
<td>101.6</td>
<td>8.11</td>
</tr>
<tr>
<td>6</td>
<td>152.4</td>
<td>18.2</td>
</tr>
<tr>
<td>10</td>
<td>254.0</td>
<td>50.7</td>
</tr>
<tr>
<td>15</td>
<td>381.0</td>
<td>114.0</td>
</tr>
<tr>
<td>20</td>
<td>508.0</td>
<td>203.0</td>
</tr>
</tbody>
</table>

* 1975 costs, after Persson"
rock. As the explosive impact intensifies, airblast problems are increased and flyrock has been known to travel up to 1 kilometre.\(^9\)

A blasting engineer has to compromise between the apparent cost savings of large blastholes and the problems resulting from them. The engineer would be well advised to investigate the matter extremely thoroughly and to discuss his recommendations with blasting engineers at other mines with similar rock types and production requirements before committing management to the high capital cost investment in something as important as a drill rig.\(^2\)

5.2.2 Explosive Type:

The considerations for the selection of an appropriate explosive type have already been detailed in Section 2.1.9.

For large scale operations, selections should be limited to bulk explosives for easy storage, transport, handling and loading. As discussed in the chapter on explosives, compatible detonating materials and blasting accessories must also be chosen to develop a complete blasting system. Explosive manufacturers and suppliers are best able to advise on some of these details.

In general, rock which is already weakened by a high intensity of natural fracturing requires more heaving action than breaking action, making AN/FO a suitable choice. As the rock becomes harder and more massive, explosives with higher densities and detonation velocities will provide the required fracturing power.

5.2.3 Sub-Drill Depth:

It is necessary to extend the blastholes to a certain depth below the intended floor of the excavation in order to completely break the rock between the blastholes down to the required level. Poor fragmentation at bench grade can lead to very expensive shovel operation due to delays and can leave a hummocky surface which is very hard on haulage truck suspension and steering systems.
Breakage usually projects from the base of the bottom charge in the form of an inverted cone with the sides inclined at an angle of 15° to 25° to the horizontal. See Figure 5.2-2. The breakage angle will vary with rock strength and orientation of dominant structural features. In horizontally bedded or well jointed rock it is often unnecessary to employ heavy toe loads or subgrade drilling since the rock has a large radius of rupture under these conditions and will break out more easily. In multi-row blasting, the breakage cones intersect to give a reasonably even transition from broken to undamaged rock. Experience has shown that a sub-drill depth of 0.2 to 0.3 times the distance to the adjacent blasthole is usually adequate to ensure effective digging at bench grade.

In the pit perimeter blasts, certain blastholes will be on top of the underlying bench crest position. In these holes the sub-drill depth should be reduced or even eliminated to avoid damaging and destabilizing the future bench. Similarly, sub-drilling should be minimized in blastholes along the final digline to avoid creating a weakened toe zone on the bench above.

In areas with known seepage or ponding problems, especially adjacent to haulage ramps, slightly increased sub-drill depths can help to improve drainage.

A further reason for over-estimating sub-drill depth is to allow for minor blasthole caving which inevitably follows the removal of the drill stem. Although the amount of caving will vary with geologic and hydraulic conditions, a shortened hole can lead to poor blast performance and a rough bench grade. It is much easier to bring an overdrilled hole back to its proper depth with a few scoops of drill cuttings than it is to try blowing out a shortened hole.

5.2.4 Stemming:

The upper unloaded portion of the blasthole, known as the collar, is usually backfilled with some form of granular material known as stemming.
FIGURE 5.2-2: ROCK BREAKAGE AT THE BOTTOM OF A BLASTHOLE DUE TO SUB-DRILLING

(after Hoek & Bray^2)
The use of stemming is a generally accepted procedure for containing the initial blast energy and directing the explosive effort into the rock. Too little stemming will allow the explosion gases to vent prematurely and will generate flyrock and air blast problems as well as reduce blast effectiveness.\(^9\) Too much stemming means the explosive load is too low in the hole and will produce coarse, blocky muck on the upper layer of the blast.

The stemming height varies from 12 times the charge diameter in hard competent rock with static compressive strength > 210 MPa to 22 times the charge diameter in softer rock with a static compressive strength of approximately 100 MPa. The stemming can be increased up to 30 diameters for soft incompetent rock with a static compressive strength of 35 MPa.\(^4\) Frequent open joints will require a larger collar since this type of rock is more apt to crater at the surface.

Although stemming should consist of well graded granular material (10 to 15 mm. crushed rock), drill cuttings are almost universally used because they are the most convenient and cheapest material available.

5.2.5 Optimum Charge:

The optimum charge is simply the amount of explosive which will attain maximum utilization of the blasthole. Once the above parameters are known, the optimum charge may be calculated by multiplying the available hole capacity by the explosive density:

\[
\text{Optimum Charge} = \gamma \times \pi \times \frac{D^2}{4} \times (H + SD - C)
\]

where:  
\(\gamma\) = explosive density (kg./m.\(^3\))  
\(D\) = blasthole diameter (m.)  
\(H\) = bench height (m.)  
\(SD\) = sub-drill depth (m.)  
\(C\) = collar height (m.)

The higher the density or bulk strength, the greater the explosive energy
that can be contained within the blasthole.

In addition to calculating the optimum charge, it is advisable to estimate maximum and minimum charges from minimum and maximum stemming requirements.

5.2.6 **Powder Factor and Burden Volume:**

Powder factor is a very useful ratio of the quantity of explosive required to fragment a certain volume of rock and is usually expressed as kg./m.\(^3\) or kg./tonne. However, if the blast designer is faced with a site where no previous blasting tests have been done, then it is likely that he will not have any idea of an appropriate powder factor for the geological conditions present. Experienced blasters are often able to estimate a fairly close initial value, but even they may have to make a few test blasts before closing in on the correct value.

To date, it has been difficult to guess a correct initial powder factor due to the complex influences of so many rock mass properties. This research project has concentrated on improving powder factor predictability before a lot of rock slope damage is caused by overloaded test blasts. The results are detailed in Chapter 6.

Once a powder factor value has been selected, the volume of rock burden on each blasthole can be simply calculated by dividing the powder factor into the optimum charge:

\[
\text{Burden Volume} = \frac{\text{Optimum Charge}}{\text{Powder Factor}}
\]

It is often convenient to convert the blasthole burden from a volume to a surface area value by dividing it by the bench height.

5.2.7 **Blasthole Pattern:**

Establishing a blasthole pattern requires the determination of hole spacings along each row and the burdens in front of the rows. If the surface area of the blasthole burden is known, then the required values of burden
and spacing can easily be calculated after selecting an appropriate burden to spacing ratio.

Although there are many possible patterns, there are a few basic configurations which are most accepted. See Figure 5.2-3. With a burden to spacing ratio of 1:1, the square pattern is the simplest to lay out. However, blasting tests carried out in both the field and the laboratory have shown that it is important to reduce the burden and to let the spacing increase.\textsuperscript{11,12} In practice, many mines use rectangular patterns with burden to spacing ratios of 1:2.

A better charge distribution is achieved by the use of the staggered pattern where the holes are located at the apices of equilateral triangles. This type of pattern is known in crystallography as the hexagonal close packed system and represents the most perfect distribution of points within a mass. This staggered pattern has a burden to spacing ratio of 1:1.15 and is most appropriate in blasting weak or highly jointed rock.

In hard competent rock, better fragmentation can be achieved by using the "Swedish" pattern. Burden to spacing ratios ranging from 1:4 up to 1:8 have been used successfully.\textsuperscript{8}

It should be noted that the burden and spacing values as laid out on a grid basis of rows and columns are known as the effective burden and effective spacing. This is because the actual burden and spacing values during detonation may be altered due to the choice of delayed firing sequence.

Once the effective burden and spacing values have been determined, the previously calculated sub-drill depths should be checked to make sure they will be adequate. It is better to iterate while in the design stage than engaging in costly blasting tests.

5.2.8 Front Row Considerations:

Successful movement of the front row burden is essential for the success of the overall blast and deserves special attention. Excessive charge leads
FIGURE 5.2-3: VARIOUS PATTERNS COMMONLY USED IN PERIMETER BLASTING

(after Hoek & Bray\textsuperscript{2})
to blow outs and flyrock at the free face, resulting in a hazard to equipment and a wastage of explosive. Too small a charge will not initiate the necessary motion of the burden, resulting in a choking of the blast on subsequent rows. A correct front row charge is a key to economic blasting.

However, the irregularity of the front burden and bench face makes accurate charge calculations difficult. If vertical blastholes are being used and the bench face is inclined as a result of the shovel's digging angle, the front row burden will vary with depth as illustrated in Figure 5.2-4. For this reason, the mean burden is the correct value to use in front row charge design calculations.

The slope of the bench face can also be dealt with by using a higher energy toe load in the front holes to ensure release at the bench toe. Alternatively, the blastholes could be inclined to provide a more uniform burden.

If the free face is uneven with larger burdens at certain points, extra holes known as easer holes can be placed in the excess material. However, this is a time consuming practice and should be avoided by providing very clear dig limit lines for the shovel operators.

5.2.9 Initiating and Firing Sequence:

The selection of a good initiation and firing sequence is a most vital step in designing a successful blast. By strategically locating a number of delay elements within the blast pattern, the blaster will not only reduce the vibrational damage discussed in Section 4.2, but will improve the efficiency of the blast by ensuring each blasthole will detonate towards a freshly created free face. Although there are several standard firing sequences as shown in Figure 5.2-5, a great deal of freedom and originality can be used, especially when designing for specific blast site configurations.

Traditionally, blasts were fired row-by-row, but as bench blast patterns became longer, the charge weight per delay became excessive. As a result,
FIGURE 5.2-4: DESIGN PARAMETERS FOR THE CRITICAL FRONT ROW BURDEN
FIGURE 5.2-5: VARIOUS FIRING SEQUENCES COMMONLY USED IN PERIMETER BLASTING

(after Hoek & Bray²)
row-by-row firing was modified to along-the-row firing where each row is broken into shorter segments. However, the number of rows fired per blast should not exceed 4 to 6 in order to prevent the final rows from becoming choked.

An alternative method is to fire the blast en echelon, thereby reducing the maximum number of holes per delay to twice the number of rows in the pattern. It should be noted that the burden to spacing ratio created by this firing method is considerably less than the effective burden to spacing ratio. For example, a square pattern with an effective ratio of 1:1 will have a ratio of 1:2 upon detonation. This can provide better fragmentation without the cost of extra drilling. En echelon firing can be initiated from a free end of the bench or by starting with a "Vee" cut in the centre of the pattern.

The length of delay intervals is usually dictated by local ground conditions and their behaviour during the blast. For a surface delayed system, intervals from 2 to 6 milliseconds per metre of burden are typical.\(^1\) Generally, the longest delay element that can be used between firing lines without leading to cutoffs in undetonated holes will produce the best results. This longer time ensures that a good quasi-free face has had time to develop in front of each succeeding detonation. In weak fragmented rock where hole cutoffs can be a problem, down-the-hole delay systems may provide greater security and permit inter-row delays of 50 to 100 milliseconds.

The initiating of all charges is critical to the success of the entire blast. Care must be taken when planning the surface layout of detonating cord so that there is more than one possible firing path to each hole. A great variety of initiating and firing methods have been tested\(^5\) since they are a key factor in achieving optimum fragmentation with minimal damage to the surrounding rock.
5.3 BLAST EVALUATION AND OPTIMIZATION

Blast evaluation is a very important step in the overall design process. It is only by careful study and analysis of the blasting results that it is possible to make proper adjustments for working towards blast optimization. Observations made both during and after detonation are valuable in assessing the blast behaviour.

5.3.1 During Detonation:

Since the actual detonation happens so quickly, the use of high speed cine cameras are really necessary to be able to do any detailed study of the blast motion.\(^{16}\) Unfortunately, such studies are expensive and are reserved for special projects.

However, a lot of useful information can be gleaned by an experienced observer. The onset of movement of the front burden can usually be seen and the observer should watch for an evenness of lift and the presence of any flyrock. Lack of clear motion may indicate inadequate front row charges, resulting in a choked blast. Also, the rhythmic sound of the rapidly firing delayed charges provides an audible check for any misfires in the pattern.

The appearance of orange-brown nitrogen dioxide fumes indicates excess oxidizer or deficient fuel. Nitrogen oxides are sometimes emitted by ammonium nitrate compounds in broken or wet ground. This could mean that liner bags will be necessary to protect the explosive. Black smoke is indicative of an explosive too rich in fuel or deficient in oxidizer. The appearance of abnormal smoke should be followed by a check into the mixing of the explosives and the implacement in the ground.

5.3.2 After Detonation:

When the dust and fumes have dispersed, the blasting engineer can assess the success of a blast by walking over and around the blasting area to closely inspect the shape and details of the muck pile. See Figure 5.3-1.
FIGURE 5.3-1: CHARACTERISTIC FEATURES OF A SUCCESSFUL PERIMETER BLAST
The front row should have moved out evenly, but not too far. Excessive throw is not necessary and only adds to the amount of costly cleanup time required before the excavating equipment can move in. The fragmentation should appear to be even and consistent across the face of the muck pile.

There should be a slight and even lift along the crest of the muck pile. Since most bench heights are designed for efficient shovel operation, low muck piles represent low productivity and high cost cleanup.

The upper surface of the muck should look evenly fragmented and not hummocky. Flat areas are clear signs of misfires. This could be caused by poor surface tie-ins or a result of long delays leading to blasthole cutoffs.

Isolated surface craters will indicate excessive hole charge or insufficient stemming. An area of several surface craters or humped up ground is usually a sign that the area became choked during the firing sequence. This could be caused by inadequate charges or by short delays which do not permit free faces to form or by too many holes firing together simultaneously.

The back area of the muck should be characterized by a slight drop, indicating a good forward movement to the free face. Clear tension cracks should be visible in front of the final digline. Any visible cracking on surface behind the final digline definitely indicates damage to the backslope and a wastage of explosive. This could be due to oversize hole charges, excessive stemming, or excessive burden on the critical final row.

In general, an effective forward movement and rotation of the blasted rock mass towards a free face is the best indication of a satisfactory blast.

Observations during the excavating stage are also important. Fragmentation should be reasonably even throughout the blast without the need for any secondary drilling and blasting. The muck should not be too tight in order to permit an economical digging rate. Coarse or blocky ground in the upper layer usually indicates oversized collars and excessive stemming, although well defined horizontal bedding can produce similar results. Coarse or blocky
ground at the bottom of the blast may indicate insufficient sub-drilling or
the need for higher energy toe charges in the blastholes.

Apart from the general characteristics seen throughout the blast area,
any unusual isolated features which may seem inexplicable can often be connected
to localized geological changes. The presence of weak shear zones or hard
intrusive stringers which were invisible before excavating should be noted
and their presence expected on the next bench below.

Upon completion of the excavation, the final face should be closely exam­
ined for any damage such as backbreak or open fractures. Also, long term
observations of any ravelling tendency may indicate the degree of damage
inflicted on the rock slope.

Once the blast evaluation is complete, the blasting engineer must decide
whether the results are satisfactory. If not, it may be necessary to embark
on a series of modifications and blasting tests in order to arrive at an optimum
design. Wherever possible, only one variable should be changed at a time and
the blast results carefully documented with notes and photographs for comparison
with other test blasts.

With the large number of design parameters available for adjustment,
the optimization procedure can become complex. The best way to modify the
blast is to select the parameter which is the easiest to adjust, such as the
powder factor. By holding the geometric parameters constant and varying only
the powder factor, it should be possible to optimize each blast on a site-specific basis since powder factor varies most directly with the rock mass
properties.

The critical evaluation of the muck pile shape, fragmentation and digging
conditions of each blast is an essential part of every blasting operation.
Although this evaluation is time consuming and therefore expensive, the cost
of this evaluation is usually justified in the development of an efficient,
optimized blasting system.²
5.4 REFERENCES


CHAPTER SIX

EXPERIMENTAL STUDIES AND FIELD WORK
6.0 EXPERIMENTAL STUDIES AND FIELD WORK

As pointed out in the previous chapters, the powder factor is one of the most critical but least understood parameters in controlled blasting design. Often, it is the major design variable since many mining operations prefer to keep their perimeter blast geometry and firing sequence constant throughout the pit for ease of planning, surveying and drilling. While it has been realized for many years that the powder factor value principally depends on the condition and properties of the rock mass, the relationship is uncertain due to the number of extremely complex inter-relationships involved, as discussed in Section 3.2.

An optimum powder factor is usually found after a series of trial-and-error blasts, but this procedure is not economical since the desire to ensure adequate fragmentation usually leads to excessive powder factor values and consequently damages the ultimate pit slope. A reliable correlation between the rock mass properties and a powder factor value would lead to great savings by optimizing the blasting and by reducing the costs involved with slope stability problems. Since open pits usually encompass several different geological zones or domains, such a correlation would permit continued monitoring of changes in the rock mass and blast optimization in each domain.

The first practical attempt to answer this problem was made by Carl Broadbent at Kennecott Copper Corporation where he developed a relationship between seismic wave velocities and powder factors. See Section 3.2.7. More recently, John Ashby developed a relationship at Bougainville Copper between natural fracture frequency and powder factors. See Section 3.2.2. While both their efforts are steps in the right direction, each of these correlations involve only one of the many rock mass properties, making their application very site specific. A separate set of correlation curves would have to be established in each different rock type.

For this research project, it was decided that the approach of establish-
ing empirical correlations with dominant rock mass features was probably the most effective way to overcome the limited understanding of the complex relationships between rock mass properties and powder factors. However, in continuing with this research, it was decided not to select another single rock mass property since this would also produce a correlation too specific in site applicability. Instead, it was decided to attempt to establish a powder factor correlation with some measure of rock quality, thereby taking account of a large number of the key rock mass properties involved.

This approach was inspired by the techniques used by tunnel design engineers. The problems they face are highly analogous to those faced by blasting engineers, since each is attempting to gain knowledge about rock conditions and properties concealed within the mass to be excavated. To aid in overcoming this critical problem, tunnelling engineers have devised several different systems of rock mass classification, used to indicate the type of tunnelling conditions or support requirements which may be encountered. These classification systems, based on drilling results and the analysis of recovered core, produce a quality rating which can be assessed in light of past tunnelling experience. To date, Bieniawski's C.S.I.R. Geomechanics Classification and Barton's N.G.I. Tunnelling Quality Index have been successfully applied in a wide variety of rock mass conditions around the world, clearly indicating their ability to overcome the problem of site specific restrictions. Based on the success of these systems, it was felt that some similar sort of rock quality classification could be equally well applied to the problems of open pit blast design. Ultimately, it was hoped that such a rock quality indicator could be correlated to optimum powder factors for economical, controlled perimeter blasting.

Clearly, the research for this project could not be carried out in anything less than a full scale, operating open pit mine. An agreement was made with Afton Operating Corporation, permitting the research program to take
place at Afton Mine which was in need of an improved blasting program.

Afton Mine proved to be an ideal facility for this research project for several reasons. First, the pit encompassed a broad range of geological conditions from very weak sediments to hard and massive volcanics, making it possible to obtain rock quality measurements from one extreme to the other. Second, the actively mined area of the pit was small enough to ensure that a good number of perimeter blasts, in all rock types, could take place during the period of field research. Third, the mine had lacked a Drilling and Blasting Foreman for some time, permitting the author to take an active and direct role in planning and executing the blasting operations.

The field research was carried out at the mine site from the beginning of May to the end of October 1981. In order to achieve the final objective, the work was done in five separate phases:

1) Establishing a Rock Quality Index (RQI) value for each geological domain in the pit.
2) Designing a new and proper perimeter blasting method.
3) Executing test blasts to determine the optimum powder factor values.
4) Deriving a correlation between RQI values and powder factor values.
5) Testing the proposed correlation with further test blasts.

The remainder of this chapter discusses each of these phases in detail. Upon completion of the work at the mine site, an Open Pit Blasting Manual was prepared for Afton Operating Corporation.\textsuperscript{5}
6.1 BACKGROUND INFORMATION ON AFTON MINE

6.1.1 Regional Information:

Afton Mine is located alongside the Trans-Canada Highway, 420 km. by road from Vancouver and only 13 km. west of Kamloops, a city of 60,000 people in the south-central interior of British Columbia. See Figure 6.1-1. The area is centrally located at the confluence of the North and South Thompson rivers, and is served by three major highway arteries, by both Canadian Pacific and Canadian National Railways, and by Pacific Western Airlines.

The regional economy has mining, forestry, agriculture and tourism as its main industries. As a result, Kamloops is a vital commercial and industrial centre, well able to serve the neighbouring mining industry. The local prosperity and conveniences of city living result in low staff turnover which is an important factor to the operation of Afton Mine.

The area is in the centre of the interior plateau, bounded by the Coast Mountains to the west and the Monashee Mountains to the east. The climate is part of the semi-arid, cold steppe formation and supports extensive natural growth of ponderosa pine and bunchgrass. The major climatic statistics, as recorded at the Kamloops airport weather station, are listed in Table 2. At an elevation of 640 m. above sea level, Afton Mine is about 300 m. higher than the airport, but differences between the weather at the two locations are minor. Afton may receive slightly less precipitation, but the proportion occurring as snow would be greater. Winter temperatures at the mine can be 5° to 10°C cooler than the airport, permitting a fairly deep frost penetration over the months of sustained sub-freezing conditions. As a result, late February and March are the times of most critical slope stability as meltwater builds up behind the frozen pit faces.

Geologically, Afton Mine is located in the Quesnel Trough, a 30 to 60 km. wide belt of Lower Mesozoic volcanics, enclosed between older rocks and much
FIGURE 6.1-1: LOCATION MAP FOR AFTON MINE
### TABLE 2

**CLIMATIC DATA, KAMLOOPS AIRPORT**

*(Based on 30 years of record)*

<table>
<thead>
<tr>
<th>MONTH</th>
<th>MEAN TEMPERATURE</th>
<th>MEAN PRECIPITATION(*)</th>
<th>MEAN NUMBER OF DEGREE-DAYS BELOW 0°C</th>
</tr>
</thead>
<tbody>
<tr>
<td>January</td>
<td>-6.1 °C</td>
<td>31.6 mm.</td>
<td>217.6</td>
</tr>
<tr>
<td>February</td>
<td>-1.3</td>
<td>16.0</td>
<td>81.5</td>
</tr>
<tr>
<td>March</td>
<td>3.5</td>
<td>9.7</td>
<td>15.9</td>
</tr>
<tr>
<td>April</td>
<td>9.1</td>
<td>10.4</td>
<td>0.0</td>
</tr>
<tr>
<td>May</td>
<td>14.1</td>
<td>18.0</td>
<td>0.0</td>
</tr>
<tr>
<td>June</td>
<td>18.0</td>
<td>29.9</td>
<td>0.0</td>
</tr>
<tr>
<td>July</td>
<td>20.8</td>
<td>22.5</td>
<td>0.0</td>
</tr>
<tr>
<td>August</td>
<td>19.8</td>
<td>27.5</td>
<td>0.0</td>
</tr>
<tr>
<td>September</td>
<td>14.9</td>
<td>24.4</td>
<td>0.0</td>
</tr>
<tr>
<td>October</td>
<td>8.4</td>
<td>15.2</td>
<td>0.9</td>
</tr>
<tr>
<td>November</td>
<td>1.6</td>
<td>22.0</td>
<td>34.4</td>
</tr>
<tr>
<td>December</td>
<td>-2.8</td>
<td>32.3</td>
<td>127.5</td>
</tr>
</tbody>
</table>

Annual Totals: - 259.5 mm. 477.8 deg.-days

(*) Rain plus Snow Equivalent

Data Obtained From: Environment Canada, Climate Information Services, 1200 West 73rd Avenue, Vancouver, B.C.
invaded by batholiths and lesser intrusions. Afton lies in the southern part of the trough, known as the Nicola Belt, which continues nearly 200 km. southward to its termination at the U.S. border and contains the important copper mines of Highland Valley, Craigmont, Copper Mountain and the former Hedley gold camp.

In the vicinity of Afton, the Iron Mask district is part of a major structure extending northwestward across the Nicola Belt. See Figure 6.1-2. In the southern portion of this structure, the 18 km. long Iron Mask Pluton and the smaller Cherry Creek Pluton are important features. Significant known copper occurrences in the district are all in the plutons, mainly close to the plutonic margins.

6.1.2 Historical Summary: (from Carr and Reed)

Except in an old prospect pit near its east end, the Afton orebody was hidden by Tertiary and Pleistocene cover up to 27 metres thick and by a salt pond. Numerous small mines occur in the district and nearly 200,000 tonnes of material was mined between 1891 and 1928. Old workings on the Afton property date from this early period, and the orebody itself lies partly on a Crown-granted claim staked in 1904.

Axel Berglund first staked the Afton claims in 1949. Subsequent drilling programs explored the area of the old Pothook shaft, where modest reserves of 0.6% copper were indicated. Testing the property widely with scattered holes, DDH 70-4 encountered persistent low-grade copper mineralization near the old prospect pit by the highway. In 1971, when C.F. Millar resumed exploration for Afton Mines Ltd., he percussion drilled on a grid around this hole and so discovered the Afton orebody. Development of the orebody was begun in 1972 with extensive drilling continuing to the end of 1974. Decision for production was announced in 1975.

The Stage 1 Pit contained 31 million tonnes of ore grading 1.0% copper,
FIGURE 6.1-2: REGIONAL GEOLOGY AT AFTON MINE

(from Carr and Reed)
0.58 ppm. gold and 4.19 ppm. silver at a cutoff of 0.25% copper and a waste-to-ore ratio of 4.2:1. Excavation of the Stage 1 Pit was completed at the end of June 1981 and the expanded Stage 2 Pit, 70 metres deep at completion of the field research program, will ultimately reach a depth of 280 metres at a waste-to-ore ratio of 7:1. Open pit ore reserves are sufficient to maintain mill production until 1988.

The ore is milled at 8500 tonnes per day, providing a metallic concentrate grading about 97% copper and a flotation concentrate grading about 50% copper. About 87% of the copper is recovered in milling. A unique feature of Afton Mine is the on-site top blown rotary converter (T.B.R.C.) smelter which produces blister copper exceeding 99% in purity.

Plate 13 is an airphoto mosaic made in November 1980, providing a detailed view of the entire Afton Mine site.

6.1.3 Summary of Site Geology:

Much of the material in this section was prepared by Doug Stewart, Afton's Chief Engineer, for a paper previously written on this research project. The Afton deposit is about 520 metres long and is essentially tabular, striking N70°W and dipping at 55°S. The orebody is divided into two distinctly different zones: a bright red supergene zone which comprises 80% of the ore, and a pale grey hypogene zone forming the remainder. The supergene zone extends to a depth of 400 metres and contains metallic copper and chalcocite. The underlying hypogene material contains bornite and chalcopyrite.

The Afton orebody occurs at the northwestern extremity of the Iron Mask Pluton, a volcanic dioritic mass which was intruded into the older Nicola Volcanics. Diorite is the predominant rock type and, with the volcanics, is flanked to the north by a deep graben structure which is occupied by younger (Tertiary) sediments and volcanics. The pit transgresses the east-west striking graben structure so that the north wall is within the younger sediments
PLATE 13

An aerial view of the Afton Mine site, November 1980.
and volcanics while the remaining pit is within diorite and older volcanics. Minor occurrences of magnetite veins, andesite, serpentine, and porphyry latites are found in the diorites and Nicola Volcanics. The widely variable geological zones can be clearly seen in the panoramic view of Plate 14.

Variety in the characteristics of these rock masses is provided not only by their lithology, but by intense alteration, jointing, and faulting. The degree of alteration has been such that diorites and volcanics are often macroscopically difficult to identify. Near-surface alteration of feldspars to sericite and kaolin, abundant chlorite, calcite, epidote, saussuritization of feldspars, and pisolitic alteration to chlorite in basalts all indicate that alteration has been significant. Within the orebody, the pervasive development of supergene ore with scales, granules, plates and dendrites of native copper indicates that the processes of alteration have long been active.

Jointing is well developed and most of the rocks within the open pit break readily into blocks smaller than 600 mm. (2 ft.) across. Strike faults are strongly developed and, together with oblique faults and lithological changes, have served to form the well established boundaries of the pit structural domains. See Figure 6.1-3 and Table 3.

Joint and fault conditions vary throughout the pit with friction angles of about 30° for joints and 21° for faults and shears. However, friction angles as low as 13° have been determined for some discontinuities with clay gouge present on their surfaces. Back analyses of failures suggest that cohesion on the failure plane was initially about 24 kPa. (500 psf.). The unconfined compressive strengths vary from about 7 MPa. (1000 psi.) in the Tertiary Sediments to over 207 MPa. (30,000 psi.) in the dacite and are dependent on the degree of alteration of the rock. See Table 4.

The geology at Afton has been described in detail by Carr and Reed.
View across Afton's pit looking west. The various geological units within the pit are clearly seen by the different coloured zones.
NOTE: All elevations shown are in feet above mean sea level.

FIGURE 6.1-3: PIT PLAN SHOWING DOMAIN BOUNDARIES AT AFTON MINE (August 1981)
## TABLE 3

### DESCRIPTION OF DOMAINS

<table>
<thead>
<tr>
<th>DOMAIN</th>
<th>ROCK TYPES</th>
<th>DESCRIPTION</th>
</tr>
</thead>
<tbody>
<tr>
<td>I</td>
<td>Iron Mask Diorite</td>
<td>Heavily sheared, epidote, chlorite, hematite</td>
</tr>
<tr>
<td>II</td>
<td>Nicola Volcanics</td>
<td>Magnetite dykes, epidote, serpentine, calcite</td>
</tr>
<tr>
<td>III</td>
<td>Iron Mask Diorite</td>
<td>Intensely sheared, clay gouge, breccia</td>
</tr>
<tr>
<td>IV A &amp; B</td>
<td>Iron Mask Diorite</td>
<td>Minor dykes, minor alteration, chlorite, calcite</td>
</tr>
<tr>
<td>IV A - Py</td>
<td>Iron Mask Diorite</td>
<td>Pyrite mineralization, moderately altered, epidote</td>
</tr>
<tr>
<td>V</td>
<td>Iron Mask Diorite</td>
<td>Some volcanics, epidote, hematite, chlorite</td>
</tr>
<tr>
<td>VI</td>
<td>Grey Diorite</td>
<td>Minor alteration, some chlorite, epidote</td>
</tr>
<tr>
<td>VII</td>
<td>Tertiary Volcanics</td>
<td>Conformable and unconformable, andesite, latite</td>
</tr>
<tr>
<td>VIII</td>
<td>Tertiary Sediments</td>
<td>Mudstone, shale, arkose, clay gouge</td>
</tr>
<tr>
<td>VIII A</td>
<td>Tertiary Volcanics</td>
<td>Dacite as lopoliths, dykes, sills, very hard</td>
</tr>
</tbody>
</table>
## TABLE 4

### HARDNESS OF ROCK TYPES USING JENNINGS' HARDNESS CLASSIFICATION

<table>
<thead>
<tr>
<th>ROCK TYPE</th>
<th>SCALE OF HARDNESS</th>
<th>APPROXIMATE RANGE OF UNCONFINED COMPRESSIVE STRENGTH</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>(p.s.i.)</td>
</tr>
<tr>
<td>Mudstone</td>
<td>R1 - 2</td>
<td>500 - 1,500</td>
</tr>
<tr>
<td>Shale</td>
<td>R2</td>
<td>1,000 - 4,000</td>
</tr>
<tr>
<td>Dacite</td>
<td>R5</td>
<td>16,000 - 32,000</td>
</tr>
<tr>
<td>Nicola Volcanics</td>
<td>R4</td>
<td>8,000 - 16,000</td>
</tr>
<tr>
<td>Grey Diorite</td>
<td>R2 - 3</td>
<td>3,000 - 7,000</td>
</tr>
<tr>
<td>Hematite</td>
<td>R3</td>
<td>4,000 - 8,000</td>
</tr>
<tr>
<td>Pyritic Diorite</td>
<td>R3</td>
<td>4,000 - 7,000</td>
</tr>
</tbody>
</table>
6.2 DEVELOPMENT OF THE ROCK QUALITY INDEX

In an open pit operation, an ideal measure of in-situ rock quality would be one which could be easily obtained without additional costs and personnel, and one which would provide a complete, unbiased coverage of the entire pit area. None of the in-situ rock mass classification methods currently available fulfill all of these requirements.

This research project set out to satisfy as many of these criteria as possible. The approach involved the study of rotary drill performance in the broad variety of rock types at Afton, and the development of index values based on the drilling data. If the resulting rock mass index values appeared to be reasonable, then work would continue with the blasting tests in order to determine optimum powder factors for correlation purposes.

The use of rotary drills as the principal monitoring tools was decided upon for several reasons. First, data collected from production blasthole drilling provides inexpensive, comprehensive coverage of the pit because all rock mined must be drilled before being blasted. Since most mines already keep blasthole drilling records, a rock quality measure can be based on the data already available and no additional personnel or equipment are required. Second, the blasthole wall represents the most critical point of interaction between the explosive and the rock. For this reason, the blasthole position is the most valuable location from which to obtain some measure of rock quality.

Third, a large body of literature is available on the performance mechanics of large rotary drills, suggesting that they will reflect many of the rock mass properties important to blasting. This is partly because the cutting action of a drill bit is a dynamic rock fracturing action in itself.

In order to understand the derivation of a rock quality measurement from rotary drill performance, it is necessary to briefly examine some of the theory of drilling mechanics.
6.2.1 Mechanical Theory of Drill Performance:

As presented by Teale\textsuperscript{12}, the rotary drilling action is accomplished by a combination of two distinct, separate actions. First, rock is broken by indentation where the cutting edges of the bit are continuously pushed into the rock by the weight on the bit. Second, a combination of brittle shear failure and crushing of the rock is caused by lateral movement of the bit as the cones roll over the bottom of the hole during bit rotation. The efficiency of this cutting and fracturing action is not only a function of bit wear, but depends to a large degree on the design and geometry of the original bit\textsuperscript{13}.

Operationally speaking, the drill penetration rate is a prime consideration since it will govern the economics of drilling. This plays an important part in blast designing where an optimal balance must be found between the drilling and explosive costs. Thus, a low penetration rate may lead to fewer blastholes, necessitating more explosive per hole and resulting in the undesirable aspects of excessive ground rupture, airblast and flyrock as discussed in Chapter 5.

The importance of the penetration rate has prompted many authors to study the rotary drilling action and to develop mathematical models for prediction purposes. This is a complex problem with many different variables involved, but the various authors, including Fish\textsuperscript{14}, Tsoutrelis\textsuperscript{15}, Bauer\textsuperscript{16} and Markman\textsuperscript{17}, generally agree that penetration rate (R) is related to the following factors:

1) Axial thrust or weight (W) on the bit. This is also related to the hydraulic down pressure (P).
2) Rotation speed (RPM).
3) Diameter (D) of the drillhole and bit.
4) Uniaxial compressive strength ($\sigma_C$) of the rock.
5) Shear strength ($\tau$) of the rock.
6) Abrasiveness (a) of the rock.
7) Geometry of the bit.
8) Flushing at the bit.

Using these parameters, the same authors proposed the following experimental equations:

Fish\textsuperscript{14}: \( R = f(\text{RPM}, \tau, \text{bit hardness}, a) \times \frac{W}{\sigma_c} \)

where \( f \) is a function not explicitly defined.

Tsoutrelis\textsuperscript{15}: \( R = \text{RPM} \times (W - W_0) \times \frac{A}{\sigma_c - B} \)

where \( A, B \) and \( W_0 \) are experimental constants.

Bauer\textsuperscript{16}: \( R = \text{RPM} \times \frac{W}{D} \times \frac{61 - 28 \log \sigma_c}{300} \)

Markman\textsuperscript{17}: \(
\frac{1}{R} = \left( \frac{13790 \times \sigma_c \times \tau}{\text{RPM} \times \sqrt{W^3}} + \frac{12 \times \sigma_c \times \tau}{\sqrt{W^3}} + 17.8 \right) \times \\
(1 + .0055 a) \times K_d \times K_b
\)

where \( K_d \) and \( K_b \) are bit geometry coefficients.

Unfortunately, most of these equations require the use of various experimental constants in order to include parameters such as wear resistance of the bit, rod design, flushing conditions, and rock structural properties. This makes these equations impractical for any general use.

With the exception of Markman's equation, which is non-linear, the penetration rate (\( R \)) is shown to be proportional to the axial pressure on the bit (\( W \)) and to the rotation speed (\( \text{RPM} \)), and is shown to decrease as the uniaxial compressive strength of the rock (\( \sigma_c \)) increases. Rock abrasiveness and bit geometry are also included explicitly in Markman's equation.

In 1975, Mathis\textsuperscript{18} examined all four of the above equations, noted their general similarities, and derived a simplified general equation for penetration rate:
\[ R = \text{RPM} \times W \times f(\sigma_c) \times K \]

where: \( f(\sigma_c) \) decreases as \( \sigma_c \) increases.

\( K \) is a constant that depends on experimental conditions and the drilling equipment used.

With this simplified equation, Mathis further noted that by keeping the drilling procedure and RPM constant, as they often are in practice, the only remaining measurable variables were penetration rate (\( R \)) and weight (\( W \)) on the bit. Putting both these variables on one side of the equation indicated that the ratio of weight to penetration rate would reflect variations in rock quality. This ratio was proposed as the Rock Quality Index as follows:

\[ \text{RQI} = \frac{\text{Weight on Drill Bit (W)}}{\text{Penetration Rate (R)}} \]

Since both of the required parameters are measured easily during drilling of individual blastholes, it should be possible to obtain a comprehensive assessment of rock mass quality.

Drill operators at most mines already keep records of drilling parameters for the evaluation of drill bit life and performance. Procedures vary from mine to mine, but typical parameters recorded for each blasthole include depth (\( d \)), drilling time (\( t \)), hydraulic down pressure (\( P \)), and RPM. The hydraulic down pressure (\( P \)) is proportional to the weight on the bit (\( W \)), differing only by the factor of bit area in contact with the rock. Therefore, for a given size and type of bit, and taking \( \frac{d}{t} \) as penetration rate, the RQI can be more practically calculated as:

\[ \text{RQI} = \frac{\text{Hydraulic Down Pressure}}{\text{Penetration Rate}} \]

or

\[ \text{RQI} = \frac{P \times t}{d} \]
This measure of rock mass quality appeared to meet all the requirements outlined at the beginning of this section and its ease of calculation would permit the tabulation of a very large number of blastholes within the limited field time available. As a result, the RQI was selected as the basis for rock mass classification and was ultimately used for correlation to powder factors.

6.2.2 Establishing Rock Quality Index Values:

Shortly after the original idea of RQI was proposed, it was assessed during the summer of 1975 in a research project by Little. He calculated several RQI values, plotted and contoured them on pit plans, and attempted to correlate the maps to lithology, structural geology and rock strength for the purpose of open pit slope design. However, he found the index to be unreliable, principally due to poor recording techniques by the drillers, and the lack of sensitivity to changes over small areas made it difficult to determine domain boundaries. Thus, for the purpose of slope design and stability evaluation, Little felt that the use of the RQI system could not be justified since geological mapping provided him with better information. After this study, no other researchers did any further work with the RQI concept.

Despite this unfavourable report, it was decided to revive the RQI concept for the current research into powder factor correlation, chiefly because of its simplicity and ease of practical application. However, learning from Little's findings, some important changes were made:

1) Concerted effort was put into improving the 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 mapping.
3) The individual areas assessed for RQI values were kept large enough to minimize effects of bit wear, shift changes, etc.

All the drilling at Afton Mine was carried out by two electric-powered, track-mounted Bucyrus Erie 40-R drills. See Plate 15 and Table 5. These drills are equipped with the maximum permissible size of 230 mm. (9 inch) diameter tri-cone bits with chisel-shaped tungsten carbide inserts. See Plate 16.

Fortunately, Afton had good drill records going back for several years as well as good geological and blasthole plans. This provided a large amount of data to compile the preliminary RQI values. Detailed geologic mapping in the early production stages had established the well-defined geological and structural domains as shown in Figure 6.1-3. As mining progressed, the positions of the domain boundaries were continually updated on the master pit plans through daily mappings by the pit geologist. These domain boundaries were retained for the RQI assessment and blasting tests in order to develop a correlation system compatible with existing pit planning work. The same rock mass features which determined these slope design domains, including slope face orientation, are those which affect the blasting behaviour of the rock. Also, each domain provided a large enough area to obtain reasonable, average RQI values.

Although not technically difficult, establishing the RQI values for each domain was a long, tedious procedure requiring about a month of data tabulation. For each bench level, the detailed plans showing all the blasthole numbers and positions were laid over the geological plans. Within each domain, every blasthole number was located among the original drillers' log sheets to determine the hydraulic down pressure (P), the hole depth (d), and the drilling time (t) for that particular blasthole. The task was complicated by the fact that the drillers seldom drilled the holes in the same sequence as they had been numbered by the surveyors. This tabulation process was done for each
One of the two Bucyrus Erie 40-R drills used at Afton Mine.
**TABLE 5**

**DRILLING MACHINE SPECIFICATIONS**

<table>
<thead>
<tr>
<th>Specification</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drill Type</td>
<td>Bucyrus Erie 40-R, track-mounted</td>
</tr>
<tr>
<td>Power Source</td>
<td>Electric</td>
</tr>
<tr>
<td>Hole Diameter Range</td>
<td>171 - 230 mm. (6 3/4 - 9 inches)</td>
</tr>
<tr>
<td>Rotation Speed Range</td>
<td>0 - 77 RPM</td>
</tr>
<tr>
<td>Maximum Torque</td>
<td>7,186 Nm. (5300 ft.-lbs.)</td>
</tr>
<tr>
<td>Maximum Pull-down Force</td>
<td>222 kN (25 tons)</td>
</tr>
<tr>
<td>Maximum Feed Rate</td>
<td>1.5 m./min. (5 ft./min.)</td>
</tr>
<tr>
<td>Maximum Extraction Rate</td>
<td>31.2 m./min. (102 ft./min.)</td>
</tr>
</tbody>
</table>
All blastholes were drilled with 230 mm. (9 inch) diameter tri-cone bits with chisel-shaped tungsten carbide inserts.
domain and repeated over bench numbers 2100, 2070, 2040 and 2010. In general, RQI values were found to be consistent from one bench level to the next within each domain. In all, data was collected and RQI values calculated for some 6000 blastholes.

In order to select a single determinative RQI value for each domain, the values from all four bench levels were combined and plotted to examine the nature of their distribution. See Appendix III. Generally, each plot showed a distinctive central peak flanked by a smaller number of higher and lower values. This was felt to be sufficient justification for simply selecting the mean or average RQI value to represent each domain.

The final results are shown in Table 6 and Figure 6.2-1. The relative position of these values agreed with the ranking estimated by site geologists, providing sufficient confidence to continue onto the next phase of the research project.
### TABLE 6

**ROCK QUALITY INDEX VALUES**

<table>
<thead>
<tr>
<th>DOMAIN</th>
<th>ROCK QUALITY INDEX</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>psi-min./ft.</td>
</tr>
<tr>
<td>I</td>
<td>345</td>
</tr>
<tr>
<td>II</td>
<td>360</td>
</tr>
<tr>
<td>IV A</td>
<td>245</td>
</tr>
<tr>
<td>IV A - Py</td>
<td>290</td>
</tr>
<tr>
<td>IV B</td>
<td>325</td>
</tr>
<tr>
<td>V</td>
<td>290</td>
</tr>
<tr>
<td>VI</td>
<td>270</td>
</tr>
<tr>
<td>VII</td>
<td>380</td>
</tr>
<tr>
<td>VIII</td>
<td>150</td>
</tr>
<tr>
<td>VIII A</td>
<td>380</td>
</tr>
</tbody>
</table>
FIGURE 6.2-1: ROCK QUALITY INDEX VALUES FOR EACH DOMAIN RANKED IN ORDER OF INCREASING QUALITY
6.3 DEVELOPMENT OF AN IMPROVED PERIMETER BLAST PATTERN

At the time when field research commenced at Afton, the blasting methods in use did not conform to acceptable practice. Before any of the important powder factor test blasts could begin, a new design was required for Afton's perimeter blasts because controlled blasting techniques were not previously used.

6.3.1 Former Blasting Methods:

Due to the confined conditions of mining the "doughnut" shape of the expanded Stage 2 pit, it was often difficult to maintain a sufficient amount of broken muck far enough ahead of the shovels. Consequently, the main pit production blasts were almost always completely choked due to the lack of an available free face. Apart from being extremely inefficient, these heavily loaded choked blasts were usually fired with a minimum of delays and the entrapped gas pressures were forced to expand out into Afton's highly fractured rock for great distances. As mentioned in Section 4.1, visible rock mass disruption propagated from a production blast, through the perimeter blast zone, and into the final wall, triggering a large scale slope failure. See Plate 11.

This problem was partly because the width of the perimeter blast zone was too narrow, permitting the heavy production blasts to take place far too close to the pit wall. The extensive backbreak was often such that the shovels were able to dig well into the perimeter zone without it having been blasted.

The original perimeter pattern consisted of one row of buffer line holes and one row of regularly spaced production holes. See Figure 6.3-1. Each row had a full production burden of 6.1 m. (20 ft.) which was too heavy to permit proper movement and rotation of the rock during the blast, impeding venting of gas pressures. If explosive loads of sufficient quantity to achieve full burden movement were used, they would have inflicted serious damage to
Figure 6.3-1: Former perimeter blast pattern at Afton Mine
the final wall and probably produced flyrock. Furthermore, the first production row was directly over the position of a future bench crest. The sub-drill depth of 1.2 m. (4 ft.) was resulting in the fracturing and weakening of these future crests, reducing the ultimate effectiveness of the benches.

This perimeter pattern was usually fired by the simple row-by-row method with no delays along the rows. The excessive charges per delay occurring on longer bench patterns set up significant vibrations which rattled windows in the engineering offices half a kilometre away from the nearest rim of the pit.

All of these factors contributed to overbreak and severe damage to the rock structure in several areas of the final pit. Continued monitoring of the large wall failure, mentioned above, showed an acceleration of movement in the failure mass when further blasting took place in that quadrant of the pit. As part of the geotechnical program at the mine, contours were established around the failure showing the maximum possible charge per delay. Much of this work was done by Brian Hill, a summer student who developed this topic for his undergraduate thesis.21

6.3.2 Design of an Improved Perimeter Blast Method:

Due to the tight mining layout, it was impossible to improve the situation with respect to the choked production blasts. However, a significant increase in the number of delays on these blasts kept the vibrational problems under better control. All new blast design work was confined to the perimeter blasts, following the guidelines presented in Chapter 5. An example of the blast design calculation is presented in Appendix IV.

Pre-shearing was ruled out due to Afton's highly fractured, weak rock and the fact that only the large diameter drills were available. Cushion blasting would have produced the best results but the sequencing of the drilling, blasting and excavation was too complicated. Furthermore, Bucyrus Erie
40-R drills are incapable of drilling the angled holes necessary for creating the bench face angle. Instead, it was felt that buffer blasting would yield good results in the highly fractured rock, as long as very light explosive loads were used adjacent to the final wall. The new buffer blast design is shown in Figure 6.3-2.

The first priority of the new design was to increase the overall width of the perimeter blast zone to better protect the final wall from the main production blasts. This was accomplished by adding another row to the perimeter blast pattern and by increasing the distance out to the nearest row of the main blast, taking advantage of the free-digging in the considerable overbreak caused by the choked blasts. It was greatly hoped that the digging could continue for another 3 metres, thereby reducing the mean burden on the front row to 4.8 m. (16 ft.). However, a crest burden of 6.1 m. (20 ft.) had to be maintained to satisfy the drillers while they manoeuvred along the second production row.

Also, the former burden-to-spacing ratio of 1:1 was reduced to 1:1.33 by shrinking the burden on both the buffer line holes and the first production row to 4.6 m. (15 ft.). This is a much more favourable ratio, permitting better movement and rotation of the burden at lower charges. A staggered pattern would have been desirable, but the surveyors and drillers were not prepared to accept a departure from the grid system.

Although this new pattern appears to require a lot of extra drilling, the blasthole density within the perimeter zone is actually reduced by 9% from 4.04 holes/100 m² on the old pattern to 3.67 holes/100 m² on the new pattern. This is largely due to the extra 2.7 m. (9 ft.) between the second production row and the last row of the main production blast. See Figure 6.3-3.

With the reduced burdens on the buffer line holes, the required charges of bulk-loaded AN/FO would just fill the subgrade portion of the hole. To get better explosive distribution up the hole and to protect the final wall,
FIGURE 6.3-2: NEW PERIMETER BLAST PATTERN AT AFTON MINE
Drillhole Density = 4.04 holes/100 sq.m.

OLD PATTERN

Drillhole Density = 3.67 holes/100 sq.m.

NEW PATTERN

FIGURE 6.3-3: COMPARISON OF DRILLHOLE DENSITIES ON OLD AND NEW PERIMETER BLAST PATTERNS
the explosive was de-coupled from the back of the hole and fully coupled to the front of the hole. A new and inexpensive technique was devised for de-coupling by using conventional plastic blasthole liners manufactured to half the blasthole diameter. These turned out to be one third the cost of the waxed, cardboard tubes normally used. See Appendix IV. Furthermore, the use of these liners did not upset the routine of the blasting crew since they involved the same installation procedures as regular, full-sized liners. The only difficulty with the smaller diameter liners was the increased blow-back pressure caused by the pneumatic AN/FO loader. However, by reducing this pressure on the pumper truck, the loading routine was eventually worked out.

The most important change for improving blast performance and reducing pit slope damage was the new firing sequence with its increased number of delays. This minimized the amount of explosive detonating at any instant, reducing harmful vibration waves, especially in areas of sensitive slope stability. In contrast to the former blasting method, the engineering office personnel were usually unable to tell when the blast was fired. The firing sequence is designed to be easy to lay out and can be continued along any length of berm. See Figure 6.3-4.

The system involves a variation of en echelon firing since most of Afton's perimeter blasts had a free end as well as a free front face. This firing pattern contributes to greater drilling economy and blast efficiency since the drill-pattern burden of 4.6 m. is reduced to a detonating burden of 2.5 m. Expressed another way, the effective burden-to-spacing ratio of 1:1.33 is reduced to 1:2.2.

As a further benefit of this firing sequence, the oblique direction of muck heave is preferrable to throwing it out into the shovel area. When combined with a properly controlled front row charge, this factor eliminates costly clean-up time before moving the shovel back to the digging face.

The success of this new perimeter blast design was proved in its first
FIGURE 6.3-4: PLAN OF NEW PERIMETER BLAST FIRING SEQUENCE
test blast on June 26th. From that point on it was established as the standard perimeter blast design at Afton Mine. See Plate 17. This left the powder factor as the only remaining blast design variable to be selected for each different rock group. For the convenience of the blasting crew, a set of tables was produced, relating the required blasthole loads to powder factor values for each of the three explosives used at Afton. See Appendix V.

Once the new perimeter blast design was in use, it was possible to proceed to the next phase of test blasts to determine optimum powder factors in each geological domain.
PLATE 17

The new perimeter blast design during detonation. As the pattern fires from left to right, the varying dust plume heights illustrate the effect of the delay sequence.
6.4 DEVELOPMENT OF THE POWDER FACTOR CORRELATION

6.4.1 Establishing Powder Factor Values:

In order to decide upon the optimum powder factors for each rock group, it was necessary to carry out a program of test blasts with the new pattern in each of the pit domains. Only in those blasts yielding good results could the powder factor used be accepted as the correct value for that particular domain. Clearly, many of the early tests contributed no final values, but each one helped to focus more rapidly towards an optimal powder factor on subsequent blasts.

The criteria used for identifying a satisfactory blast were discussed in detail in Section 5.3. Before a powder factor could be selected as the optimum value for its particular rock type, all of these criteria had to be met:

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 buffer line holes.
5) No broken ground beyond the final digline.
6) No digging problems.
7) Uniform fragmentation.
8) A clean final wall with minimum ravelling potential.

The final test blast in the Tertiary Volcanics of Domain VII, shown during detonation in Plate 17, was a particularly good example of all the features of a successful blast. This is illustrated in Plates 18, 19 and 20.

Based on a qualitative evaluation process, the assessment of a successful blast is a highly subjective procedure. Unlike the determination of RQI values, the final selection of an optimum powder factor was based on the author's
The test blast of Plate 17 produced this muck pile which illustrates the features of a successful blast. Note the evenness of throw, indicating a good front row charge.
A close up view of the muck pile toe area. Note the relatively uniform fragmentation and the lack of rock debris beyond the toe. The slope of this muck face is ideal.
PLATE 20

A close up view on top of the muck pile. Note the slight rise of the evenly fragmented crest and the slight dip along the back of the berm. The area appears uniform without craters or humps.
own judgement and experience after having witnessed many blasts over the course of the field research program. Test blasts continued as long as possible in order to obtain the maximum number of data points in the limited time available.

6.4.2 Data Analysis and Results:

Towards the end of the field research period, final powder factor values had been collected from five of the domains. Although further blasting tests were still required in the remaining domains, it was felt that sufficient data was available for an initial attempt to correlate the powder factors with their respective RQI values.

The final values for powder factor and RQI were plotted and a simple regression analysis was made using one of the library routines of the Hewlett-Packard 9845 computer. After attempts to fit different types of curves to the plot, the analysis indicated the best fit to be a natural logarithmic curve with a coefficient of determination of 0.98. This strong correlation is described by the following equation:

\[ \ln(\text{Powder Factor}) = \frac{\text{RQI} - 24.9}{7.1} \]

where: Powder Factor is in kg./tonne
RQI is in MPa-min./metre

By converting the units (see Appendix VI), the correlation becomes:

\[ \ln(\text{Powder Factor}) = \frac{\text{RQI} - 885}{315} \]

where: Powder Factor is in lbs./s.tbn
RQI is in psi-min./ft.

In using this correlation, it must be remembered that the powder factor quoted is for AN/FO and the RQI is based on a Bucyrus Erie 40-R drill with a 230 mm. (9 inch) diameter carbide tipped tri-cone bit.
With the remaining time in the field program rapidly drawing to a close, this correlation was used as a guide to predict likely powder factors for the blast tests in the remaining domains. This proved to be the first test of the RQI/Powder Factor correlation and the subsequent test blasts were successful. The final powder factor data points served to reinforce the accuracy of the correlation curve as shown in Figure 6.4-1.

At the end of the field program, a new blasting manual was produced for Afton Mine. This included a large master pit plan showing all the geological domains with their corresponding RQI and powder factor values. This data is summarized in Table 7. Since the plan positions of the different rock zones will shift with deepening excavation, the pit geologist must keep this plan up to date along with his regular geological sheets. Should the rock quality conditions change significantly with depth, RQI values tabulated from the daily drill logs should reflect this, indicating a corresponding change in powder factor to maintain optimal blast performance.

6.4.3 Testing the RQI/Powder Factor Correlation:

The use of the preliminary curve to successfully predict powder factors in the last domains already provided a good initial test for the proposed correlation. However, two more challenging tests of the RQI system occurred in the Fall of 1981.

In September, two production blasts in Domain II failed to break down to bench grade and had large boulders mixed in with the muck. It was impossible for the shovel to excavate this area, causing major production delays and necessitating secondary drilling and blasting. At Afton's request, the author returned to the mine site to examine the situation.

Upon closer inspection, the problem was attributed to several hard veins of magnetite erratically located within the Nicola Volcanics of Domain II. It was suspected that the same conditions could be encountered in the adjoining
FIGURE 6.4-1: PROPOSED CORRELATION BETWEEN ROCK QUALITY INDEX AND POWDER FACTOR

S.I.:
\[
\ln(\text{powder factor}) = \frac{RQI - 24.9}{7.1}
\]

Imperial:
\[
\ln(\text{powder factor}) = \frac{RQI - 885}{315}
\]
## TABLE 7

**SUMMARY OF POWDER FACTOR/RQI CORRELATION**

<table>
<thead>
<tr>
<th>DOMAIN</th>
<th>POWDER FACTOR</th>
<th>ROCK QUALITY INDEX</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>lbs./s. ton</td>
<td>kg./tonne</td>
</tr>
<tr>
<td></td>
<td>psi-min./ft.</td>
<td>MPa-min./m.</td>
</tr>
<tr>
<td>I</td>
<td>0.18</td>
<td>0.090</td>
</tr>
<tr>
<td>II</td>
<td>0.19</td>
<td>0.095</td>
</tr>
<tr>
<td>IV A</td>
<td>0.13</td>
<td>0.065</td>
</tr>
<tr>
<td>IV A - Py</td>
<td>0.15</td>
<td>0.075</td>
</tr>
<tr>
<td>IV B</td>
<td>0.17</td>
<td>0.085</td>
</tr>
<tr>
<td>V</td>
<td>0.15</td>
<td>0.075</td>
</tr>
<tr>
<td>VI</td>
<td>0.14</td>
<td>0.070</td>
</tr>
<tr>
<td>VII</td>
<td>0.20</td>
<td>0.100</td>
</tr>
<tr>
<td>VIII</td>
<td>0.10</td>
<td>0.050</td>
</tr>
<tr>
<td>VIII A</td>
<td>0.20</td>
<td>0.100</td>
</tr>
<tr>
<td>IX</td>
<td>0.25</td>
<td>0.125</td>
</tr>
</tbody>
</table>

**Imperial Units:**  $\ln (\text{Powder Factor}) = \frac{\text{RQI} - 885}{315}$

**S. I. Units:**  $\ln (\text{Powder Factor}) = \frac{\text{RQI} - 24.9}{7.1}$
blast pattern. Examination of the drill logs for this blast site revealed several holes with significantly higher RQI values, clearly reflecting the presence of further magnetite veins. By using higher explosive charges in those particular blastholes, as dictated by the RQI/Powder Factor correlation, the blast was successful and no hard toes remained.

This event demonstrated the ability of the drills and the RQI to indicate isolated changes within the rock mass. From this, it is conceivable that improved techniques in the recording of drill performance data may permit the selection of powder factors on a hole-by-hole basis. After the field work was completed, it was learned that an automatic drill data recording device is available from Totco Mining and Industrial Instrumentation which may be able to provide the necessary accuracy for more refined RQI measurements.

The most critical test of the RQI concept came in October when a large zone of dacite, labelled Domain IX, appeared at the base of the large, unstable slope failure referred to earlier. Completely unlike any other rock type in Afton's pit, the dacite was very hard and quite massive, requiring a fairly heavy explosive load for proper fragmentation. In addition to the problem of cleanly excavating the dacite, there were grave concerns about the effects of the heavier impact on the wall failure above. However, it was felt that the large number of delays within the new perimeter blast pattern would be sufficient to minimize any disturbance of the slide.

There was no clear data on an appropriate powder factor for the dacite since the only blast previously attempted did not succeed in completely fragmenting the massive structure. Having little else to go on, it was decided that this situation could provide a crucial test of the RQI/Powder Factor Correlation and the chance to see if the curve could be extrapolated into higher RQI values. Although there were only 43 blastholes within the dacite, a rough RQI of 10.2 MPa-min./m. (450 psi-min./ft.) was calculated and the corresponding powder factor of 0.125 kg./tonne (0.25 lbs./s.ton) determined.
See Table 7. A few boulders were shaken loose on the already weakened upper slope, but no significant movement was recorded on the failure mass and the dacite was blasted out successfully. The results of this blast provided a further data point for the proposed correlation and indicated that it was possible to extend the existing curve into the realm of more competent rock masses.

Since this last occasion, there have been no further tests on the limitations or applicability of the proposed RQI/Powder Factor correlation.

6.4.4 Suggested Further Research:

Before it will be possible to confidently apply this correlation outside Afton's open pit, a more rigorous series of tests should be carried out. The ideal testing program would involve repeating this entire research project in other mines with identical drilling equipment and bit size. A comparison of the results from each mine would be needed to provide definitive evidence as to the validity of the existing correlation on a universal basis. Should such a program prove successful, then it would be worthwhile to begin establishing similar correlations for other drill type and bit combinations. At this time, such an extensive testing program is beyond the limits of the current research project, but it is hoped that work will be continued in this area by future researchers.

Smaller scale research projects could be done on further refinement of the Rock Quality Index. By drilling a large number of holes within a single, uniform rock type, it should be possible to develop better statistical parameters on RQI value distributions and to assess the significance of bit wear effects as the number of holes drilled increases.

A separate drilling program should be carried out with an automatic drill data recording device, such as the one mentioned above, to test its ability to improve data quality. This could then be followed by a series of carefully
controlled blasting tests to assess the feasibility of hole-by-hole powder factor selection.

Some preliminary experiments were carried out at Afton in an attempt to establish a correlation between the RQI values and the seismic velocities in each domain. Due to poor equipment and lack of time, these experiments were not completed. A much more extensive and properly controlled field program is necessary to test this hypothesis, but such a correlation could provide an interesting cross-check with the early work done by Broadbent.¹

In general, it is hoped that the ideas presented in this research project will prove to be of practical value in operating mines, thereby providing a considerable amount of further data for improvements and refinements to the presently proposed RQI/Powder Factor correlation.
6.5 PRACTICAL APPLICATIONS OF RESULTS

From the outset, the author was determined to approach this research project in a practical manner such that any results could be readily applied by practising engineers and thus be of maximum benefit. It is felt that this basic objective has been met in establishing the Rock Quality Index classification and in the proposed RQI/Powder Factor correlation.

The concept of the RQI can be easily implemented, taking advantage of existing drill performance data and without the need for extra equipment or specially trained personnel. This system is particularly valuable in pits with several domains of widely varying lithology, permitting a constant monitoring of the rock mass conditions. In non-glaciated parts of the world, where open pits are excavated through extensive zones of weathered rock, the RQI would be an excellent indicator of changing and improving rock quality conditions as the pit approached the lower level of the weathering horizon. Due to the simplicity of the RQI system, the author is optimistic that it will be of practical value in third world nations.

By extending the RQI concept to a correlation with powder factors, a highly practical method is available for overcoming one of the major problems in controlled blasting design. Although further testing will be necessary to determine whether the proposed correlation is universally applicable, the important result of this research is to show that a comprehensive program can certainly be developed within a single open pit operation. With prior knowledge of rock mass conditions obtained from drilling data, it is now possible to determine the optimal explosive charge which will produce the desired fragmentation with minimal pit slope damage. This has been a major objective of rock slope engineers for many years. As the importance of this system becomes apparent, the author hopes that the present drill data recording techniques will improve, permitting a greater refinement of RQI calculations and
ultimately leading to a hole-by-hole powder factor design. The potential economic benefits could be substantial.

Although this research project concentrated mainly on the aspect of blast optimization, there was some evidence that a good correlation may exist between RQI and ore milling rates. This possibility was examined briefly, but a major difficulty was presented by the variations in ore storage time on the stockpile between excavating and milling. However, the variations in milling rate were of sufficient importance to the mill operators that some form of RQI correlation would be very valuable.

It is also hoped that the background knowledge accumulated on rock blasting and discussed in the four previous chapters has been presented in a manner which will make practical contributions to the understanding and application of modern blasting techniques.
6.6 REFERENCES


CHAPTER SEVEN

CONCLUSIONS
7.0 CONCLUSIONS

Rock blasting is the vital first step in any excavating or mining project. In the past, achieving good fragmentation was the sole consideration in blast design, but in recent years rock slope engineers have drawn attention to other key factors, such as slope stability, which are important to the overall safety and economy of a blasting operation. The principal requirement is to confine the destructive blast energy to the desired excavation while inhibiting or preventing damage to the surrounding rock of the final pit slope. Although this objective has been greatly facilitated by the development of various controlled blasting techniques, there are still certain areas in blast design which are poorly understood yet critical to achieving the desired results. The most important area of uncertainty is that of the inter-relationship between blast performance and the properties of the rock mass being blasted.

It has long been realized that the many geological features characterizing a particular rock mass have direct influences on the blasting results. Although the contribution of certain features is partially known, it is their inter-action together which controls the rock fracturing process. Since the complexity of problem prohibits the development of a comprehensive rock blasting theory, this study examined the possibility of applying a single rock mass classification which would take account of those rock mass properties most important to blasting. Based on the performance of a rotary drill, this classification, known as the Rock Quality Index, has been found to bear a strong correlation to the powder factor, a major variable in site specific blasting design.

As a result of the background study and field work on this research project, the following conclusions have been made:

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

3) The Rock Quality Index is sensitive to changing rock mass conditions from one blasthole to the next. Improved monitoring methods may permit indexing on a hole-by-hole basis.

4) The practice of choked blasting in rock with a high frequency of continuous, natural fractures will result in extensive ground rupture for considerable distances beyond the intended excavation limits.

5) Pit slope movements caused by excessive, blast-induced ground vibration can be minimized or eliminated by using multiple delays within a blasting pattern to reduce the maximum detonating charge per delay.

6) The careful design of the burden and charge weight along the front row of a blast pattern is a key element in successful perimeter blasting.

7) Provided a well planned controlled blasting design is used, the geometric parameters can be held constant and the powder factor kept as the only design variable, even when blasting in a broad variety of geologic conditions.

8) A strong correlation exists between the powder factor and the rock mass conditions as described by the Rock Quality Index (RQI) at Afton Mine. For the particular drilling equipment and explosive type used, this correlation can be defined as follows:

\[
\ln (\text{Powder Factor}) = \frac{\text{RQI} - 24.9}{7.1}
\]

where:  Powder Factor is in kg./tonne
        RQI is in MPa-min./metre
9) A good blast evaluation program is essential in order to continually maintain optimal blasting performance as site conditions or blasting configurations change.

10) By understanding and applying the basic principles of controlled rock blasting, it is possible to produce the desired fragmentation and, at the same time, preserve the inherent rock mass properties important to the stability of the final rock slope. Ultimately, this achieves the greatest overall economy for the construction and maintenance of excavations in rock.
BIBLIOGRAPHY


43) DUPONT: Four Major Methods of Controlled Blasting. Training Notes, Copyright 1964.


APPENDIX I

GLOSSARY OF BLASTING TERMINOLOGY
GLOSSARY OF BLASTING TERMINOLOGY

Absorbant: a material used to absorb liquid explosive bases.

Acoustic Impedance Discontinuity: a break in the continuum of the rock mass (i.e. an open joint or fault) which causes a drop in the shock wave energy.

Airblast: a strong vibrational shock wave propagating through the air as a result of a large, inadequately confined explosion.

Along-the-Row Firing: a method for detonating perimeter patterns where delay elements are used along the row in addition to the inter-row delays for blast vibration reduction.

AN/FO: Standard abbreviation for Ammonium Nitrate/Fuel Oil. This is a common bulk blasting agent consisting of 94% ammonium nitrate prills and 6% diesel fuel. Inexpensive and safe to handle.

Antacid: an ingredient added to an explosive formulation to increase stability in storage.

Attenuation: the reduction of shock wave amplitude and energy as the wave propagates outward from the source point.

Backbreak: distance of broken or fractured rock beyond the intended line of excavation. See also Overbreak.

Ballistic Mortar: an empirical test for comparing explosive strengths, measuring the ability of 10 grams of explosive to deflect a heavy steel mortar.

Bench Blasting: blasting to two or more free surfaces.

Bench Grade: usually taken to be the elevation at the toe of the bench face.
Blasthole Cutoff: the shearing off of an undetonated blasthole, preventing detonation. Usually caused by damage from neighbouring blastholes or shearing along failure planes.

Blasthole Liner: a tubular plastic sleeve used to protect explosives of low water resistance when the blasthole is wet.

Blasting Agent: a blasting material, consisting of a fuel and an oxidizer, not classified as an explosive and in which none of the ingredients is an explosive. Cannot be detonated with a No. 8 cap.

Blasting Machine: a small, portable device to provide current for firing blasts electrically where alternate power sources are not readily available.

Blasting Records: records of blasthole pattern, charge weights, firing sequence, and post-blast site conditions for a blast evaluation program.

Borehole Pressure: the peak effective pressure acting behind the detonation head on the cylindrical surface area of the borehole.

Breakout Velocity: velocity imparted to the fragmented rock on the bench face as a result of the blast forces.

Brisance: the extreme shattering effect resulting from almost instantaneous decomposition of the explosive upon detonation.

Buffer Blasting: the most simple and economical method of controlled blasting, it requires a row of lightly loaded holes slightly offset from the final digline for reducing backbreak.

Bulk Explosive: an explosive which is produced and handled in large batch quantities and is usually free-flowing. Typically used on mine sites where large volumes of explosive are required.
Burden to Spacing Ratio: the ratio of effective burden divided by the effective spacing, usually quoted to characterize a blast pattern.

Burden Volume: The volume of rock in front of a single blasthole = burden x spacing x bench height.

Cap Sensitivity: describes the sensitivity of an explosive to being initiated by a single blasting cap.

Caving: refers to the collapse of an undetonated blasthole. Usually caused by drill stem removal or natural degradation as opposed to Cutoffs.

Channel Effect: occurs when the explosive column is narrower than the blasthole diameter. A wave of air pressure travels ahead of the detonating front, compressing the explosive to its dead-pressed state and preventing complete detonation of the hole.

Charge Diameter: the diameter of the explosive column. For bulk explosives, this is the same as the blasthole diameter.

Charge Weight: the quantity of explosive in a single blasthole.

Choked Blast: a blast which is excessively confined, preventing proper rock mass motion and release of the burden.

Collar: the unloaded upper portion of the blasthole.

Cohesion: refers to the cohesive component of inter-block shearing resistance within a rock mass.

Controlled Blasting: methods of blasting used to reduce the damage to surrounding rock masses and structures.
Combustible: an explosive ingredient which combines with excess oxygen to prevent the formation of nitrogen oxides.

Coupling: the extent to which an explosive charge is in direct contact with the rock wall of the blasthole.

Crater Blasting: blasting with the only free surface being normal to and at the mouth of the blasthole.

Crest Burden: minimum distance measured on the bench surface from the front blasthole to the crest of the bench face.

Cushion Blasting: a controlled blasting method where the final row is fired after the main portion of the perimeter pattern has been fired and excavated.

Cycling Effect: when AN/FO is cycled through 0°C or 32°C more than once, the prills swell and contract, eventually crumbling.

Dead-Pressed: refers to an explosive which has been compacted to a density where it will no longer detonate.

Delay: a device consisting of a slow burning composition which provides a short break in the detonating sequence of a blast pattern.

Density: referring to the density of an explosive, usually quoted as a specific gravity.

Detonating Cord: a cord used for firing blast patterns non-electrically. The cord usually consists of a core of PETN wrapped within a reinforced waterproof covering.

Detonation Burden: the burden on a blasthole during the blast pattern detonation. May be different from Effective Burden due to firing sequence.
Detonation Pressure: the pressure in the detonation wave of an explosive in a blasthole.

Detonation Spacing: the spacing on a group of blastholes during the blast pattern detonation. May be different from Effective Spacing due to firing sequence.

Detonics: the study of explosive detonation.

Digline: the intended line of ultimate excavation at the back of a perimeter blast pattern.

Discontinuity: any break in the continuum of a rock mass. Includes faults, joints, bedding, shear zones, or fissures.

Downline: the length of detonating cord extending from the surface blast pattern down the blasthole to the primer charge.

Down-the-Hole Delay System: a blasting technique where all the delay elements are in the blastholes adjacent to the primers. Used in situations where Cutoffs may occur.

Drill Cuttings: the conic pile of finely ground rock produced by the drill when excavating a blasthole.

Dynamic Rock Strength: the strength of rock under very rapid loading conditions such as those encountered in blasting.

Dynamites: the original formulation of high explosives, consisting primarily of nitroglycerin and an absorbant.

Easer Hole: an extra hole placed between the front row and the bench crest in locations where bench face irregularity results in excessive front row burden.
Effective Burden: the burden on a row of blastholes measured perpendicular to the row as on a blasthole plan.

Effective Spacing: the spacing between blastholes along a row as measured on a blasthole plan.

Electric Blasting Cap: a metal shell with two wires leading into one end used to initiate the entire blast firing sequence.

En Echelon Firing: a firing sequence where the blastholes are fired in groups at some angle to the bench face.

Explosive Base: a solid or liquid which breaks down into gaseous products with an accompanying release of heat energy. Forms the basic ingredient of explosive formulations.

Firing Sequence: the order in which a blasthole pattern is fired, usually employing a number of delay elements.

Flyrock: rock which is thrown through the air as a result of excessive blasting forces.

Fragmentation: a qualitative term referring to the size to which rock is broken by blasting. There is no generally accepted size or gradation.

Free Face Blasting: see Bench Blasting.

Friction Angle: the dip angle to which inter-block friction is able to resist inter-block shearing. Usually denoted by $\phi$.

Gelatins: tough, rubbery or plastic-textured explosive compositions which possess good water resistance. Usually are high density.
Geological Domain: a zone in which a rock mass has consistent properties and structural features and has a slope with relatively constant orientation.

Geometric Blasting Parameters: those blast design parameters which are usually calculated by geometric means and remain constant throughout much of the pit. Includes burden, spacing, sub-drill and bench height.

High Explosives: explosives which detonate, indicating that the reaction moves through the explosive faster than the speed of sound.

High Toes: refers to portions of the rock mass which failed to break out down to intended bench grade, producing a hummocky floor.

Hydraulic Down Pressure: the hydraulic pressure applied to a drill stem forcing the bit against the bottom of the blasthole.

Ice-Jacking: a weathering process where freezing water expands in near-surface cracks and fissures, forcing the rock to spall off the face.

Initiating Trunkline: the long line of detonating cord extending from the blasting cap to the blast pattern.

Initiation: the commencement of explosive detonation.

Internal Friction: the resistance to wave transmission within a rock mass, producing heat from the shock wave's mechanical energy.

Leg Wires: the two fine wires which lead into the metal shell of an electric blasting cap.

Low Explosive: explosives which deflagrate, indicating that the reaction moves through the explosive slower than the speed of sound.
Maximum Instantaneous Charge: the largest amount of explosive which detonates at any instant within a multi-delayed firing sequence.

Mean Burden: average distance from a front row blasthole to the nearest free face. Usually used for front row burden calculations.

Muck: rock which has been fragmented by blasting.

Optimum Charge: the maximum amount of explosive which can be placed in a blasthole, allowing for the necessary height of stemming.

Optimum Powder Factor: the lowest powder factor which will achieve desired fragmentation and produce minimal damage to the surrounding rock mass.

Overblasting: using excessive quantities of explosive without regard for the resulting damage to the surrounding rock.

Overbreak: the additional volume of rock which must be excavated due to severe Backbreak beyond the final digline.

Oxygen Carrier: an explosive ingredient to ensure complete oxidation of the carbon in the explosive mixture, preventing the formation of carbon monoxide.

Peak Particle Velocity: the maximum velocity of particle motion during the passage of the seismic wave beneath the particle.

Perimeter Blast: the blast which takes place in the zone immediately adjacent to the final pit slope.

Pneumatic Wedging: the action which causes loss of inter-block contact due to the rapidly expanding explosive gases penetrating into
pre-existing discontinuities.

Poisson's Ratio: an elastic constant usually denoted by $\sigma$

Powder Factor: a convenient ratio relating the required amount of explosive to a unit mass or volume of rock to be blasted.

Power: used to indicate the potential of an explosive to penetrate or shatter.

Pre-Shear Blasting: a controlled blasting method where the line hole row is fired before the main portion of the blast, creating a well-defined fracture plane along the final face.

Primary Explosive: the most sensitive category of High Explosives, reliably detonated by spark, flame or impact.

Primary Wave: the fastest of the shock waves, it is a compressive wave which deforms the rock in a radial direction.

Primer: any high power, high velocity explosive compound capable of initiating detonation of low sensitivity blasting agents or explosives.

Production Blast: a large scale blast within the pit operation which produces the muck on the required production scale.

Propagation Velocity: the speed with which a shock wave travels through a material.

Quasi-Free Face: refers to the imaginary 'free' planes which occur upon succeeding rows of a multi-delayed blast, providing relief for the next burden.

Quasi-Static Pressure: the high gas pressure which remains relatively constant within the blasthole over the very brief duration
of the detonic reaction.

Ravelling: the slow but steady degradation of a rock slope surface, producing talus material at the base of the slope.

Rayleigh Wave: the slow seismic waves which travel along ground surface.

Relative Bulk Strength: a strength comparison between identical volumes of an explosive and a standard explosive such as AN/FO.

Relative Weight Strength: a strength comparison between identical weights of an explosive and a standard explosive such as AN/FO.

Rock Quality Index (RQI): an index value which can reflect rock mass quality, based on rotary drill performance.

Rock Slope Engineering: the practice of site investigation, design and analysis for producing stable surface excavations in rock.

Row-by-Row Firing: a firing sequence where each row of the pattern is fired one at a time with the only delays between the rows.

Rupture Radius: the distance around a blasthole which is broken or disturbed by the detonation of an explosive charge in that blasthole.

Scaling: removing weakened or loosened rock from a face to avoid a safety hazard.

Secondary Blasting: the re-drilling and re-firing of rock which was not properly fragmented by the first blast attempt.

Secondary Explosive: the less sensitive category of High Explosives.
Secondary Wave: the seismic shear wave, causing deformation of the rock at right angles to its direction of travel.

Seismic Wave Velocity: the speed with which the primary seismic wave travels through the rock mass.

Sensitivity: a measure of the impulse magnitude required to start an explosive reaction.

Site Specific Design: the adapting or modifying of a blast design to suit the particular rock mass conditions at the blast site.

Slurry: an ammonium nitrate based explosive in an aqueous solution, gelled with a gum to give considerable water resistance.

Spalling: a flaking or scabbing action on the surface of the rock caused by the reflection of the compressive shock wave at the free face.

Square Pattern: a blast pattern with a Burden to Spacing Ratio of 1:1.

Staggered Pattern: a blast pattern with holes placed at optimum density within the rock mass for a Burden to Spacing Ratio of 1:1.15.

Stemming: the granular material used to backfill the unloaded collar of a blasthole. Drill Cuttings are usually used.

Stripping Ratio: the ratio of stripping (waste) volume to the volume of ore recovered in an open pit mine.

Structural Orientation: the orientation of a discontinuity plane in space. Usually defined by strike and dip, or by dip and dip direction.

Sub-Drill Depth: the depth of the blasthole below final bench grade to prevent the formation of High Toes.
Surface Delay System: a blasting method where all the delay elements are tied into the surface trunklines of the blast pattern.

Swedish Pattern: a blasting pattern for hard rock where the Burden to Spacing Ratio can be 1:4 or less.

Swell: the increase in rock volume from its pre-blasted state to its fragmented condition.

Toe Burden: the distance from the front row blastholes to the nearest free face measured at bench grade elevation.

Trim Blasting: see Cushion Blasting.

Trunklines: the surface pattern of detonating cord which links all the blastholes together within the blast area.

Underblasting: using insufficient explosive, resulting in coarse fragmentation and possibly leading to secondary blasting.

Velocity of Detonation: the speed with which the detonating wave propagates through the explosive. The confined value is greater than the unconfined value.

Water Gels: see Slurry.

Water Resistance: a rating of an explosive's ability to withstand exposure to water without deteriorating.

Young's Modulus: the elastic modulus of a material, usually denoted by E.
APPENDIX II

BASIC PROPERTIES OF EXPLOSIVES AND BLASTING MATERIALS

USED AT AFTON OPERATING CORPORATION
AMMONIUM NITRATE / FUEL OIL (AN/FO)

AN/FO is a dry blasting agent consisting of 94% prilled ammonium nitrate and 6% diesel fuel oil. It was the principal explosive used at Afton Mine and was supplied in bulk for easy transport and loading. See Plate 21.

Basic Properties:

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Specific Gravity</td>
<td>0.84</td>
</tr>
<tr>
<td>Density</td>
<td>840 kg./m.³</td>
</tr>
<tr>
<td>Relative Weight Strength</td>
<td>100</td>
</tr>
<tr>
<td>Relative Bulk Strength</td>
<td>100</td>
</tr>
<tr>
<td>Unconfined Detonation Velocity</td>
<td>2700 m./sec.</td>
</tr>
<tr>
<td>Water Resistance</td>
<td>Nil</td>
</tr>
</tbody>
</table>

Properties in a 230 mm. (9 inch) Diameter Borehole:

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density</td>
<td>34.5 kg./m. of hole</td>
</tr>
<tr>
<td>Estimated Detonation Velocity</td>
<td>4000 m./sec.</td>
</tr>
<tr>
<td>Estimated Detonation Pressure</td>
<td>3600 MPa.</td>
</tr>
<tr>
<td>Estimated Borehole Pressure</td>
<td>2400 MPa.</td>
</tr>
</tbody>
</table>

Cost at Afton Mine (August 1981) = $ 33.09 / 100 kg.
A handful of free-flowing AN/FO. Safe handling is one of the important characteristics of a blasting agent.
HYDROMEX T-3

Hydromex T-3 is a slurry explosive principally composed of ammonium nitrate, T.N.T. and water. At Afton Mine, it was stocked in 203 mm. (8 inch) diameter bags, each containing 25 kg. of the explosive. See Plate 22.

Basic Properties:

- Specific Gravity: 1.46
- Density: 1460 kg./m.$^3$
- Relative Weight Strength: 89
- Relative Bulk Strength: 155
- Unconfined Detonation Velocity: 4600 m./sec.
- Water Resistance: Excellent

Properties in a 230 mm. (9 inch) Diameter Borehole:

- Density: 60 kg./m. of hole
- Estimated Detonation Velocity: 5700 m./sec.
- Estimated Detonation Pressure: 9850 MPa.
- Estimated Borehole Pressure: 5100 MPa.

Cost at Afton Mine (August 1981) = $ 180.00 / 100 kg.
A 25 kilogram bag of Hydromex T-3. This will be hand loaded into wet blastholes because of its excellent water resistance.
POWERGEL A

Powergel A is a slurry explosive similar to Hydromex T-3. At Afton Mine it was supplied in bulk for use on blast patterns where most of the holes contained groundwater.

Basic Properties:

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Specific Gravity</td>
<td>1.25</td>
</tr>
<tr>
<td>Density</td>
<td>1250 kg./m.³</td>
</tr>
<tr>
<td>Relative Weight Strength</td>
<td>82</td>
</tr>
<tr>
<td>Relative Bulk Strength</td>
<td>122</td>
</tr>
<tr>
<td>Unconfined Detonation Velocity</td>
<td>4500 m./sec.</td>
</tr>
<tr>
<td>Water Resistance</td>
<td>Excellent</td>
</tr>
</tbody>
</table>

Properties in a 230 mm. (9 inch) Diameter Borehole:

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density</td>
<td>51 kg./m. of hole</td>
</tr>
<tr>
<td>Estimated Detonation Velocity</td>
<td>5600 m./sec.</td>
</tr>
<tr>
<td>Estimated Detonation Pressure</td>
<td>8800 MPa.</td>
</tr>
<tr>
<td>Estimated Borehole Pressure</td>
<td>5000 MPa.</td>
</tr>
</tbody>
</table>

Cost at Afton Mine (August 1981) = $ 92.60 / 100 kg.
DETONATING MATERIALS

Procore III Primers: (See Plate 23)

<table>
<thead>
<tr>
<th>Weight</th>
<th>450 grams each</th>
</tr>
</thead>
<tbody>
<tr>
<td>Water Resistance</td>
<td>Good</td>
</tr>
<tr>
<td>Cost at Afton Mine (August 1981)</td>
<td>$ 2.95 each</td>
</tr>
</tbody>
</table>

Reinforced Primacord: (See Plate 23)

<table>
<thead>
<tr>
<th>Weight</th>
<th>25.3 gm./m.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diameter</td>
<td>5.1 mm.</td>
</tr>
<tr>
<td>Tensile Strength</td>
<td>1.11 kN.</td>
</tr>
<tr>
<td>Detonation Velocity</td>
<td>6200 m./sec.</td>
</tr>
<tr>
<td>Cost at Afton Mine (August 1981)</td>
<td>$ 32.30 / 100 m.</td>
</tr>
</tbody>
</table>

E-Cord:

<table>
<thead>
<tr>
<th>Weight</th>
<th>17.9 gm./m.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diameter</td>
<td>4.1 mm.</td>
</tr>
<tr>
<td>Tensile Strength</td>
<td>1.0 kN.</td>
</tr>
<tr>
<td>Detonation Velocity</td>
<td>6200 m./sec.</td>
</tr>
<tr>
<td>Cost at Afton Mine (August 1981)</td>
<td>$ 25.10 / 100 m.</td>
</tr>
</tbody>
</table>

Instantaneous Electric Blasting Caps:

<table>
<thead>
<tr>
<th>Leg Wire Length</th>
<th>3.0 m.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Leg Wire Diameter</td>
<td>0.53 mm.</td>
</tr>
<tr>
<td>Resistance (Cap + Leg Wires)</td>
<td>1.55 Ohms</td>
</tr>
<tr>
<td>Cost at Afton Mine (August 1981)</td>
<td>$ 73.30 / 100 caps</td>
</tr>
</tbody>
</table>
A Procore III Primer tied onto the end of a Reinforced Primacord downline is ready for lowering into a blasthole.
DETONATING MATERIALS  continued

Plastic Blasthole Liner:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Layflat Dimension</td>
<td>394 mm.</td>
</tr>
<tr>
<td>Bore Diameter</td>
<td>250 mm.</td>
</tr>
<tr>
<td>Wall Thickness</td>
<td>10 mil.</td>
</tr>
<tr>
<td>Cost at Afton Mine</td>
<td>$ 63.25 / 183 m. roll</td>
</tr>
</tbody>
</table>
APPENDIX III

DATA LOGS AND PLOTS USED TO DETERMINE

ROCK QUALITY INDEX VALUES
DATA LOGS AND PLOTS USED TO DETERMINE ROCK QUALITY INDEX VALUES

This appendix contains an example of the type of drillers' log sheets used at Afton Mine for recording the drill performance parameters.

This is followed by the frequency distribution plots of the RQI values, used to investigate the nature of their distribution within each domain. The shape of these plots provided sufficient justification for using mean RQI values to represent each domain.
<table>
<thead>
<tr>
<th>DATE</th>
<th>SHIFT</th>
<th>HOLE/RUN NUMBER</th>
<th>FEET DRILLED</th>
<th>DRILLING TIME</th>
<th>HYD. DOWN PRESSURE</th>
<th>RPM</th>
<th>AIR (PSI)</th>
<th>ROCK TYPE</th>
<th>COMMENTS</th>
</tr>
</thead>
<tbody>
<tr>
<td>3/10/77</td>
<td>1</td>
<td>4203</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>4204</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>4205</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>4206</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>4207</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>4208</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>7</td>
<td>4209</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>8</td>
<td>4210</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9</td>
<td>4211</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>4212</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>11</td>
<td>4213</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>12</td>
<td>4214</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>13</td>
<td>4215</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>14</td>
<td>4216</td>
<td>20</td>
<td>30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>13/10/77</td>
<td>0</td>
<td>4217</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>1</td>
<td>4218</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>4219</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>4220</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>4221</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>4222</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>6</td>
<td>4223</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>7</td>
<td>4224</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>8</td>
<td>4225</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9</td>
<td>4226</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>4227</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>11</td>
<td>4228</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>12</td>
<td>4229</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>13</td>
<td>4230</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>14</td>
<td>4231</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>15</td>
<td>4232</td>
<td>20</td>
<td>25</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**TOTAL:** 3560

**DATE OFE:**

**TYPE OF FAILURE:**
- Bearings
- Cones
- Shirttail

**DRILLER:**

**COMMENTS:**
DOMAIN 1

No. of readings = 567

Mean RQI = $345 \frac{\text{psi-min.}}{\text{ft.}}$

= $7.8 \frac{\text{MPa-min.}}{\text{m.}}$
DOMAIN II

No. of readings = 342

Mean RQI = 360 \frac{\text{psi-min.}}{\text{ft.}}

= 8.2 \frac{\text{MPa-min.}}{\text{m.}}
DOMAIN IV A

No. of readings = 396

Mean RQI = $245 \frac{\text{psi-min.}}{\text{ft.}}$

$= 5.5 \frac{\text{MPa-min.}}{\text{m.}}$
DOMAIN IV A-Py

No. of readings = 336

Mean RQI = \(290 \frac{\text{psi-min.}}{\text{ft.}}\)

\[= 6.5 \frac{\text{MPa-min.}}{\text{m.}}\]
DOMAIN IV B

No. of readings = 1197

Mean RQI = 325 \( \frac{\text{psi-min.}}{\text{ft.}} \)

= 7.4 \( \frac{\text{MPa-min.}}{\text{m.}} \)
DOMAINE V

No. of readings = 1065

Mean RQI = 290 \( \frac{\text{psi-min.}}{\text{ft.}} \)

= 6.5 \( \frac{\text{MPa-min.}}{\text{m.}} \)
DOMAIN VI

No. of readings = 411

Mean $RQI = 270 \ \frac{psi-min.}{ft.} \ \ \ \ \ \ \ \ \ \ \ \ \ = 6.1 \ \frac{MPa-min.}{m.}$
DOMAIN VII

No. of readings = 351

Mean $RQI = 380 \frac{\text{psi-min.}}{\text{ft.}}$

$= 8.6 \frac{\text{MPa-min.}}{\text{m.}}$
DOMAIN VIII

No. of readings = 1023

Mean RQI = 150 \( \frac{\text{psi-min.}}{\text{ft.}} \)

= 3.4 \( \frac{\text{MPa-min.}}{\text{m.}} \)
DOMAIN VIII A

No. of readings = 380

Mean RQI = 380 \( \frac{\text{psi-min.}}{\text{ft.}} \)

= 8.6 \( \frac{\text{MPa-min.}}{\text{m.}} \)
APPENDIX IV

DESIGN EXAMPLE FOR THE NEW PERIMETER BLAST

and

COMPARISON OF DE-COUPLING METHODS
DESIGN EXAMPLE FOR THE NEW PERIMETER BLAST

A detailed discussion of all the considerations for a controlled blast design are presented in Chapter Five. These guidelines were used in the following design for Afton's new perimeter blast, although some external constraints limited the flexibility of certain design aspects.

The Field Parameters:

- **Bench Height** = \( H = 9.1 \text{ m. (30 ft.)} \)
- **Blasthole Diameter** = \( D = 230 \text{ mm. (9 in.)} \)
- **Rock Density** = \( \gamma_r = 0.37 \text{ m}^3/\text{tonne (12 ft}^3/\text{S.Ton)} \)
- **Powder Factor** = \( \text{PF} = 0.085 \text{ kg./tonne (0.17 lbs./S.Ton)} \)
- **Length of Blast** = \( L = 61 \text{ m. (200 ft.)} \)

Design Constraints:

- Minimum hole spacing of 3.05 m. (10 ft.) along buffer line row.
- Minimum crest burden of 6.1 m. (20 ft.).
- Minimum row burdens of 3.7 m. (12 ft.).
- Staggered pattern not acceptable.

Blasthole Pattern:

To maintain a grid pattern, use hole spacings twice that of the buffer line row. \( \therefore S = 6.1 \text{ m. (20 ft.)} \)

Buffer line burden should be 20 times the blasthole diameter, or between 0.6 and 0.8 times the production row burden:

\[
20 \times D = 20 \times 230 \text{ mm.} = 4.6 \text{ m. (15 ft.)}
\]

\[
0.6 \times B = 0.6 \times 6.1 \text{ m.} = 3.7 \text{ m. (12 ft.)}
\]

\[
0.8 \times B = 0.8 \times 6.1 \text{ m.} = 4.9 \text{ m. (16 ft.)}
\]

Therefore, select burdens for both rows to be = 4.6 m. (15 ft.)

**Burden to Spacing Ratio** = 1:1.33
Sub-Drill Depths:

For buffer line holes and first production row holes, select a minimal sub-drill depth of 0.61 m. (2 ft.) to avoid damage to the future bench below. Shallow sub-drill should be alright due to large radius of rupture in Afton's highly fragmented rock.

For second production row, assume a toe cratering angle of 20°. Sub-drill depth will be the greater of:

\[ SD = (4.6 - (0.61/\tan 20°)) \times \tan 20° = 1.07 \text{ m. (3.5 ft.)} \]

or

\[ SD = (8.8 - (1.83/\tan 20°)) \times \tan 20° = 1.4 \text{ m. (4.6 ft.)} \]

Therefore, select sub-drill depth = SD = 1.5 m. (5 ft.)

Extra hole length will allow for usual blasthole caving.

AN/FO Charge Design:  (refer to tables in Appendix V for quick reference)

In Buffer Line Holes:

Charge Weight = \( (B \times S \times H \times PF)/\gamma_r \) = \( (4.6 \times 3.05 \times 9.1 \times 0.085)/0.37 \)

= 29 kg. (64 lbs.) per hole

In First Production Holes:

Charge Weight = \( (4.6 \times 6.1 \times 9.1 \times 0.085)/0.37 \)

= 59 kg. (130 lbs.) per hole

In Second Production Holes:

Charge Weight = \( (8.8 \times 6.1 \times 9.1 \times 0.085)/0.37 \)

= 112 kg. (250 lbs.) per hole

Firing Sequence:

As shown in Figure 6.3-4 of Chapter Six.
Blast Inventory Summary:

Number of Buffer Line Holes = \((L/3.05) + 1\) = 21

Number of First Production Holes = \((L/6.1) + 1\) = 11

Number of Second Production Holes = \((L/6.1) + 1\) = 11

Total Explosive Quantity Required = 2.49 tonnes (2.76 S.Tons)

Number of 50 msec. delays = \(((L/6.1) \times 4) + 3\) = 43

Number of 100 msec. delays = \((L/6.1)\) = 10

Duration of Detonation = \(((L/3.05) + 2) \times 0.05\) = 1.1 seconds
COMPARISON OF METHODS FOR DE-COUPLING BUFFER LINE CHARGES

Cardboard Liner Tubes:

Supplied by Sono-Co Ltd., 877 Cliveden, Annacis Island.

As currently supplied to Brenda Mines:

Main tubes: 20 ft. x 4-3/4" (4.714" i.d.) x 1/8" wall.
Waxed inner and outer ply plus water resistant sealer.
$ 3362.31 / 1000 tubes + $ 66.58 set-up charge.

Couplers: 12" x 5-1/4" (5" i.d.) x 1/8" wall.
$ 366.25 / 1000 couplers + $ 66.58 set-up charge.

Total Charge = $ 3861.72 / 1000 tubes = 19.3¢ / ft. = 63.32¢ / m.

Plastic Pipe:

Supplied by International Plastics, 12180 Vickers Way, Richmond.

As supplied to Kidd Creek Mine:

"Big 0" corrugated drain pipe (the cheapest pipe available)
4.61" o.d., 4.0" i.d., 74 lbs./250 ft. coil

Cost = $ 85.00 / 250 ft. coil = 34¢ / ft. = 116¢ / m.

Plastic Borehole Liner:

Supplied by Layfield Plastics, 12104 - 121A St., Edmonton.

As supplied to Afton Mine:

7-1/2" layflat dimension, 4-3/4" bore, 8 mil. wall thickness.

Cost = $31.00 / 500 ft. roll = 6.2¢ / ft. = 20.3¢ / m.
APPENDIX V

EXPLOSIVE LOAD TABLES FOR PERIMETER BLASTS
AN/FO REQUIREMENTS IN LBS./HOLE
SECOND PRODUCTION ROW
5 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10</td>
</tr>
<tr>
<td>29</td>
<td>120</td>
</tr>
<tr>
<td>30</td>
<td>125</td>
</tr>
<tr>
<td>31</td>
<td>130</td>
</tr>
<tr>
<td>32</td>
<td>135</td>
</tr>
<tr>
<td>33</td>
<td>140</td>
</tr>
<tr>
<td>34</td>
<td>145</td>
</tr>
<tr>
<td>35</td>
<td>150</td>
</tr>
<tr>
<td>36</td>
<td>155</td>
</tr>
<tr>
<td>37</td>
<td>160</td>
</tr>
<tr>
<td>38</td>
<td>165</td>
</tr>
<tr>
<td>39</td>
<td>170</td>
</tr>
<tr>
<td>40</td>
<td>175</td>
</tr>
<tr>
<td>41</td>
<td>180</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
### AN/FO Requirements in LBS./HOLE

#### First Production Row

2 Foot Subgrade

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10</td>
</tr>
<tr>
<td>26</td>
<td>60</td>
</tr>
<tr>
<td>27</td>
<td>63</td>
</tr>
<tr>
<td>28</td>
<td>65</td>
</tr>
<tr>
<td>29</td>
<td>68</td>
</tr>
<tr>
<td>30</td>
<td>70</td>
</tr>
<tr>
<td>31</td>
<td>73</td>
</tr>
<tr>
<td>32</td>
<td>75</td>
</tr>
<tr>
<td>33</td>
<td>78</td>
</tr>
<tr>
<td>34</td>
<td>80</td>
</tr>
<tr>
<td>35</td>
<td>83</td>
</tr>
<tr>
<td>36</td>
<td>85</td>
</tr>
<tr>
<td>37</td>
<td>88</td>
</tr>
<tr>
<td>38</td>
<td>-</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
AN/FO REQUIREMENTS IN LBS./HOLE

BUFFER LINE ROW

2 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10</td>
</tr>
<tr>
<td>26</td>
<td>30</td>
</tr>
<tr>
<td>27</td>
<td>31</td>
</tr>
<tr>
<td>28</td>
<td>33</td>
</tr>
<tr>
<td>29</td>
<td>34</td>
</tr>
<tr>
<td>30</td>
<td>35</td>
</tr>
<tr>
<td>31</td>
<td>36</td>
</tr>
<tr>
<td>32</td>
<td>38</td>
</tr>
<tr>
<td>33</td>
<td>39</td>
</tr>
<tr>
<td>34</td>
<td>40</td>
</tr>
<tr>
<td>35</td>
<td>41</td>
</tr>
<tr>
<td>36</td>
<td>43</td>
</tr>
<tr>
<td>37</td>
<td>44</td>
</tr>
<tr>
<td>38</td>
<td>45</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
BAGS OF HYDROMEX T-3 / HOLE
SECOND PRODUCTION ROW
5 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>AN/FO Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10 0.11 0.12 0.13 0.14 0.15 0.16 0.17 0.18 0.19 0.20 0.21 0.25</td>
</tr>
<tr>
<td>29</td>
<td>2.5 2.5 3.0 3.0 3.5 3.5 4.0 4.0 4.5 4.5 5.0 5.0 6.0</td>
</tr>
<tr>
<td>30</td>
<td>2.5 3.0 3.0 3.5 3.5 4.0 4.0 4.5 4.5 5.0 5.0 5.5 6.5</td>
</tr>
<tr>
<td>31</td>
<td>2.5 3.0 3.0 3.5 3.5 4.0 4.0 4.5 5.0 5.0 5.5 5.5 6.5</td>
</tr>
<tr>
<td>32</td>
<td>3.0 3.0 3.5 3.5 4.0 4.0 4.5 4.5 5.0 5.0 5.5 6.0 7.0</td>
</tr>
<tr>
<td>33</td>
<td>3.0 3.0 3.5 3.5 4.0 4.5 4.5 5.0 5.0 5.5 5.5 6.0 7.0</td>
</tr>
<tr>
<td>34</td>
<td>3.0 3.5 3.5 4.0 4.0 4.5 4.5 5.0 5.5 5.5 6.0 6.0 7.5</td>
</tr>
<tr>
<td>35</td>
<td>3.0 3.5 3.5 4.0 4.5 4.5 5.0 5.0 5.5 6.0 6.0 6.5 7.5</td>
</tr>
<tr>
<td>36</td>
<td>3.0 3.5 4.0 4.0 4.5 4.5 5.0 5.5 5.5 6.0 6.5 6.5 8.0</td>
</tr>
<tr>
<td>37</td>
<td>3.5 3.5 4.0 4.0 4.5 5.0 5.5 6.0 6.0 6.5 7.0 8.0</td>
</tr>
<tr>
<td>38</td>
<td>3.5 3.5 4.0 4.5 4.5 5.0 5.5 5.5 6.0 6.5 7.0 8.5</td>
</tr>
<tr>
<td>39</td>
<td>3.5 4.0 4.0 4.5 5.0 5.0 5.5 6.0 6.5 7.0 7.5 8.5</td>
</tr>
<tr>
<td>40</td>
<td>3.5 4.0 4.5 4.5 5.0 5.5 5.5 6.0 6.5 7.0 7.0 7.5 9.0</td>
</tr>
<tr>
<td>41</td>
<td>3.5 4.0 4.5 5.0 5.0 5.5 6.0 6.0 6.5 7.0 7.5 7.5 9.0</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.$^3$/S. ton
BAGS OF HYDROMEX T-3 / HOLE

FIRST PRODUCTION ROW

2 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>AN/FO Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10 0.11 0.12 0.13 0.14 0.15 0.16 0.17 0.18 0.19 0.20 0.21 0.25</td>
</tr>
<tr>
<td>26</td>
<td>1.0 1.5 1.5 1.5 1.5 2.0 2.0 2.0 2.0 2.5 2.5 2.5 3.0</td>
</tr>
<tr>
<td>27</td>
<td>1.5 1.5 1.5 1.5 2.0 2.0 2.0 2.0 2.5 2.5 2.5 2.5 3.0</td>
</tr>
<tr>
<td>28</td>
<td>1.5 1.5 1.5 1.5 2.0 2.0 2.0 2.5 2.5 2.5 2.5 3.0 3.5</td>
</tr>
<tr>
<td>29</td>
<td>1.5 1.5 1.5 2.0 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.5</td>
</tr>
<tr>
<td>30</td>
<td>1.5 1.5 1.5 2.0 2.0 2.0 2.5 2.5 2.5 2.5 3.0 3.0 3.5</td>
</tr>
<tr>
<td>31</td>
<td>1.5 1.5 2.0 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5</td>
</tr>
<tr>
<td>32</td>
<td>1.5 1.5 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.0 4.0</td>
</tr>
<tr>
<td>33</td>
<td>1.5 1.5 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5 4.0</td>
</tr>
<tr>
<td>34</td>
<td>1.5 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5 3.5 4.0</td>
</tr>
<tr>
<td>35</td>
<td>1.5 2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5 3.5 4.0</td>
</tr>
<tr>
<td>36</td>
<td>1.5 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5 3.5 3.5 4.5</td>
</tr>
<tr>
<td>37</td>
<td>2.0 2.0 2.0 2.5 2.5 2.5 3.0 3.0 3.0 3.5 3.5 3.5 4.5</td>
</tr>
<tr>
<td>38</td>
<td>2.0 2.0 2.0 2.5 2.5 3.0 3.0 3.0 3.5 3.5 3.5 3.5 4.0</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
BAGS OF HYDROMEX T-3 / HOLE

BUFFER LINE ROW

2 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>AN/FO Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10 0.11 0.12 0.13 0.14 0.15 0.16 0.17 0.18 0.19 0.20 0.21 0.25</td>
</tr>
<tr>
<td>26</td>
<td>0.5 0.5 0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5</td>
</tr>
<tr>
<td>27</td>
<td>0.5 0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5</td>
</tr>
<tr>
<td>28</td>
<td>0.5 0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5</td>
</tr>
<tr>
<td>29</td>
<td>0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5</td>
</tr>
<tr>
<td>30</td>
<td>0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5</td>
</tr>
<tr>
<td>31</td>
<td>0.5 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 2.0</td>
</tr>
<tr>
<td>32</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 2.0</td>
</tr>
<tr>
<td>33</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 2.0</td>
</tr>
<tr>
<td>34</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 2.0</td>
</tr>
<tr>
<td>35</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 1.5 1.5 2.0 2.0</td>
</tr>
<tr>
<td>36</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 1.5 1.5 2.0 2.0</td>
</tr>
<tr>
<td>37</td>
<td>1.0 1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 1.5 1.5 2.0 2.0</td>
</tr>
<tr>
<td>38</td>
<td>1.0 1.0 1.0 1.0 1.0 1.5 1.5 1.5 1.5 1.5 1.5 2.0 2.0 2.5</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
COLLAR HEIGHTS FOR POWERGEL A

SECOND PRODUCTION ROW

5 FOOT SUBGRADE

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>0.10</th>
<th>0.11</th>
<th>0.12</th>
<th>0.13</th>
<th>0.14</th>
<th>0.15</th>
<th>0.16</th>
<th>0.17</th>
<th>0.18</th>
<th>0.19</th>
<th>0.20</th>
<th>0.21</th>
<th>0.25</th>
</tr>
</thead>
<tbody>
<tr>
<td>29</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>23</td>
<td>23</td>
<td>23</td>
<td>22</td>
<td>21</td>
<td>21</td>
<td>21</td>
<td>20</td>
<td>18</td>
<td></td>
</tr>
<tr>
<td>30</td>
<td>26</td>
<td>25</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>23</td>
<td>23</td>
<td>22</td>
<td>22</td>
<td>22</td>
<td>21</td>
<td>19</td>
<td></td>
</tr>
<tr>
<td>31</td>
<td>26</td>
<td>26</td>
<td>25</td>
<td>25</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>23</td>
<td>23</td>
<td>22</td>
<td>22</td>
<td>21</td>
<td>19</td>
</tr>
<tr>
<td>32</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>26</td>
<td>25</td>
<td>25</td>
<td>24</td>
<td>23</td>
<td>23</td>
<td>22</td>
<td>22</td>
<td>20</td>
<td></td>
</tr>
<tr>
<td>33</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>26</td>
<td>25</td>
<td>24</td>
<td>23</td>
<td>24</td>
<td>23</td>
<td>21</td>
<td></td>
</tr>
<tr>
<td>34</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>25</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>23</td>
<td>21</td>
<td></td>
</tr>
<tr>
<td>35</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>26</td>
<td>25</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>22</td>
<td></td>
</tr>
<tr>
<td>36</td>
<td>31</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>26</td>
<td>25</td>
<td>24</td>
<td>24</td>
<td>22</td>
<td></td>
</tr>
<tr>
<td>37</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>27</td>
<td>26</td>
<td>25</td>
<td>24</td>
<td>23</td>
<td></td>
</tr>
<tr>
<td>38</td>
<td>32</td>
<td>32</td>
<td>31</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>27</td>
<td>26</td>
<td>25</td>
<td>23</td>
<td></td>
</tr>
<tr>
<td>39</td>
<td>33</td>
<td>32</td>
<td>32</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>27</td>
<td>26</td>
<td>24</td>
<td></td>
</tr>
<tr>
<td>40</td>
<td>34</td>
<td>33</td>
<td>33</td>
<td>32</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>27</td>
<td>25</td>
<td></td>
</tr>
<tr>
<td>41</td>
<td>35</td>
<td>34</td>
<td>33</td>
<td>33</td>
<td>32</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>25</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
**COLLAR HEIGHTS FOR POWERGEL A**

**FIRST PRODUCTION ROW**

**2 FOOT SUBGRADE**

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>AN/FO Powder Factor (lbs./S. ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.10 0.11 0.12 0.13 0.14 0.15 0.16 0.17 0.18 0.19 0.20 0.21 0.25</td>
</tr>
<tr>
<td>26</td>
<td>24  24  23  23  23  22  22  22  22  22  22  22  22</td>
</tr>
<tr>
<td>27</td>
<td>25  25  24  24  24  23  23  23  23  23  23  23  23</td>
</tr>
<tr>
<td>30</td>
<td>28  27  27  27  27  26  26  26  26  26  26  26  24</td>
</tr>
<tr>
<td>31</td>
<td>28  28  28  28  27  27  27  27  26  26  26  26  26</td>
</tr>
<tr>
<td>32</td>
<td>29  29  29  29  28  28  28  28  27  27  27  27  26</td>
</tr>
<tr>
<td>33</td>
<td>30  30  30  29  29  29  29  28  28  28  28  27  26</td>
</tr>
<tr>
<td>34</td>
<td>31  31  31  30  30  29  29  29  29  28  28  28  27</td>
</tr>
<tr>
<td>35</td>
<td>32  32  31  31  31  30  30  30  29  29  29  29  28</td>
</tr>
<tr>
<td>36</td>
<td>33  33  32  32  32  31  31  31  31  30  30  30  28</td>
</tr>
<tr>
<td>37</td>
<td>34  34  33  33  33  32  32  32  32  31  31  31  30</td>
</tr>
<tr>
<td>38</td>
<td>35  34  34  34  34  33  33  33  33  32  32  32  31</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.$^3$/S. ton
### COLLAR HEIGHTS FOR POWERGEL A

**BUFFER LINE ROW**

**2 FOOT SUBGRADE**

<table>
<thead>
<tr>
<th>Hole Depth (Feet)</th>
<th>0.10</th>
<th>0.11</th>
<th>0.12</th>
<th>0.13</th>
<th>0.14</th>
<th>0.15</th>
<th>0.16</th>
<th>0.17</th>
<th>0.18</th>
<th>0.19</th>
<th>0.20</th>
<th>0.21</th>
<th>0.25</th>
</tr>
</thead>
<tbody>
<tr>
<td>28</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>26</td>
<td>25</td>
</tr>
<tr>
<td>29</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>27</td>
<td>26</td>
<td>26</td>
</tr>
<tr>
<td>30</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>28</td>
<td>27</td>
<td>27</td>
</tr>
<tr>
<td>31</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>28</td>
<td>28</td>
<td>28</td>
</tr>
<tr>
<td>32</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
<td>29</td>
</tr>
<tr>
<td>33</td>
<td>32</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
<td>30</td>
</tr>
<tr>
<td>34</td>
<td>33</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>31</td>
<td>30</td>
</tr>
<tr>
<td>35</td>
<td>34</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>32</td>
<td>31</td>
</tr>
<tr>
<td>36</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>33</td>
<td>32</td>
</tr>
<tr>
<td>37</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>33</td>
</tr>
<tr>
<td>38</td>
<td>36</td>
<td>36</td>
<td>36</td>
<td>36</td>
<td>36</td>
<td>36</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>35</td>
<td>34</td>
</tr>
</tbody>
</table>

Rock Density = 12 ft.³/S. ton
APPENDIX VI

CONVERSION TABLES FOR IMPERIAL AND S.I. UNITS
<table>
<thead>
<tr>
<th>IMPERIAL UNITS</th>
<th>CONVERSION FACTORS</th>
<th>S.I. UNITS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lengths:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>mil</td>
<td>25.4</td>
<td>micrometre (micron)</td>
</tr>
<tr>
<td>inch</td>
<td>25.4</td>
<td>millimetre</td>
</tr>
<tr>
<td>foot</td>
<td>0.3048</td>
<td>metre</td>
</tr>
<tr>
<td>mile</td>
<td>1.609344</td>
<td>kilometre</td>
</tr>
<tr>
<td>Areas:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>square inches</td>
<td>645.16</td>
<td>sq. millimetres</td>
</tr>
<tr>
<td>square feet</td>
<td>0.09290304</td>
<td>sq. metres</td>
</tr>
<tr>
<td>square miles</td>
<td>2.589988110</td>
<td>sq. kilometres</td>
</tr>
<tr>
<td>square miles</td>
<td>258.999</td>
<td>hectares</td>
</tr>
<tr>
<td>acres</td>
<td>0.4046873</td>
<td>hectares</td>
</tr>
<tr>
<td>Volumes:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>cubic inches</td>
<td>16387.064</td>
<td>cu. millimetres</td>
</tr>
<tr>
<td>cubic feet</td>
<td>0.02831685</td>
<td>cu. metres</td>
</tr>
<tr>
<td>ounces (Imp.)</td>
<td>28.4131</td>
<td>cu. centimetres</td>
</tr>
<tr>
<td>gallons (Imp.)</td>
<td>4.5460905</td>
<td>litres</td>
</tr>
<tr>
<td>Mass:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>pound</td>
<td>0.45359237</td>
<td>kilogram</td>
</tr>
<tr>
<td>short ton</td>
<td>0.90718474</td>
<td>tonne</td>
</tr>
<tr>
<td>Force:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>ounce</td>
<td>27801.38510</td>
<td>dyne</td>
</tr>
<tr>
<td>pound</td>
<td>4.448221615</td>
<td>newton</td>
</tr>
<tr>
<td>Pressure:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>p.s.i.</td>
<td>6894.757293</td>
<td>pascal</td>
</tr>
<tr>
<td>inches of Hg</td>
<td>33.864</td>
<td>millibar</td>
</tr>
<tr>
<td>Density:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>lbs./cu. foot</td>
<td>16.0185</td>
<td>kg./cu. metre</td>
</tr>
<tr>
<td>cu. ft./S. ton</td>
<td>0.031214</td>
<td>cu. m./tonne</td>
</tr>
<tr>
<td>lbs./S. ton</td>
<td>0.5000</td>
<td>kg./tonne</td>
</tr>
<tr>
<td>R.Q.I.:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>p.s.i.-min./ft.</td>
<td>0.022620596</td>
<td>MPa-min./m.</td>
</tr>
</tbody>
</table>

* Converting from Imperial to S.I., multiply by the conversion factor.

* Converting from S.I. to Imperial, divide by the conversion factor.
EPILOGUE
EPILOGUE

You can convince some of the people all of the time.

You can convince all of the people some of the time.

But you can't convince all of the people all of the time!

Photo by: Mark Stoakes