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Rock movement due to blasting and its impact on ore grade control in Nevada open pit gold mines

Zhang, Songlin, M.S.

University of Nevada, Reno, 1994
UNIVERSITY OF NEVADA
RENO

Rock Movement due to Blasting and its Impact on
Ore Grade Control in Nevada Open Pit Gold Mines

A thesis submitted in partial fulfillment of
the requirements for the degree of
Master of Science in Mining Engineering

By
Songlin Zhang

Pierre Mousset-Jones, Thesis Advisor

May 1994
The thesis of Songlin Zhang is approved

Thesis Advisor

Chairman of Mining Engineering Department

Dean, Graduate School

University of Nevada,

Reno

May, 1994
ACKNOWLEDGEMENTS

The research project was initiated by Professor Pierre Mousset-Jones, who contributed greatly to the success of the project. His assistance with revising the thesis is very much appreciated. Many thanks are due for the considerable support and advice from Professors Jaak Daemen and Robert Watters. The general manager Mr. Bob Martinez, chief engineer Mr. Paul Martin and senior geologist, Mr. Jon Hurley of the Coeur Rochester mine, are specially thanked for their support. The research project was partially supported by the Newmont Gold Company and the Generic Mineral Technology Center in Mine System Design and Ground Control, office of Mineral Institutes, US Bureau of Mines Cooperation Grant No. G1125151. The chief mine engineer Mr. Ross Oliver, mine engineer Mr. Mark Wood, mine superintendent Mr. Ross Maclean, mine engineer Mr. David Hamre are gratefully acknowledged for their assistance and cooperation at the Rain mine. Finally, I would like to thank my wife, Cunying Ju, for her support and encouragement.
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ABSTRACT

This thesis presents the results of a research project into blast induced rock movement and its impact on ore grade control at the Rain Mine of the Newmont Gold Company and the Coeur Rochester Mine. A total of twelve blasts were monitored between the two mines using additional holes loaded with marker bags and a Quarryman laser profiler to locate the bags and survey the surface rock movement. An average of 4.5-6.0 feet horizontal movement occurred at the Rain Mine with a powder factor of 0.3-0.4 lb ANFO/ton. An average of 10-45 feet horizontal rock movement occurred at the Coeur Rochester Mine with a powder factor of 0.57-0.76 lb Heavy ANFO/ton. This rock movement resulted in a theoretical dilution from three percent up to eighty percent depending on the amount of rock movement and the size of polygon. A method to adjust the pre-blast polygon according to the rock movement is discussed in order to decrease dilution when digging. It is concluded that dilution induced by blasting can be minimized, if the position of a digging polygon is modified as a result of systematic measurements of rock movement.
CHAPTER 1 INTRODUCTION

1. DRILLING, BLASTING AND ORE GRADE CONTROL

Most western United States open pit gold mines are disseminated deposits (Geological Survey of Nevada, 1987). The mineralization is often quite irregular, and because there is no obvious visual distinction between geological contacts, it is difficult to define the ore/waste boundary by observation. Rather, blast hole samples are used to estimate the continuity of mineralization in a bench. Typically, two dimensional digging polygons are located on the muck pile based on the pre-blast location of the blast holes. Rock movement caused by blasting can be of major concern at these mines. Blast movement can mix and dilate the ore and move it beyond the flagged digging polygons established on the blasted rock according to their pre-blast locations. A recently completed survey on drilling/blasting and ore grade control practices in Nevada open pit gold mines, discussed in chapter 2, indicates that some mines realize that rock movement induced by blasting is one significant factor which impacts on the accuracy of mining ore and waste. A number of mines are not concerned about blast movement since it appears to be quite small, or it is too complicated to consider its impact on grade control.
The primary goal of ore grade control is to provide timely and accurate information to the production crew as to the location of ore and waste boundaries. The results from the survey indicate that the blast holes are usually laid out by surveyors, and are numbered in a logical sequence. Blast hole samples are collected and assayed with guidance from the block/mineralization model of the deposit and historic blasthole and production data from the preceding benches. The pre-blast polygons are interpolated using the blast hole assay results. This is carried out either by a traditional or geostatistical interpolation method. After the blast, flags or wood stakes are used to indicate polygonal contacts on the muck pile surface, and show digging limits for the loader operators. Geologists are involved in ore grade control practices, by mapping the pit geology, providing geological information to grade control engineers and/or assisting the loader operators adjusting the digging polygons in the pit to suit local conditions.

Currently, digging plans assume that limited horizontal movement occurs during blasting. This approach assumes that the digging polygon or mining block remains in place after blasting. However, this rarely happens during production blasting. In-fact, some horizontal movement always occurs since shock waves and gas pressure from an explosive will always produce some rock movement. This makes it difficult to find the right balance between fragmentation and movement i.e.
"break it real well, but don't move it."

Some mines try to use a small powder factor to constrain rock movement in order to improve grade control. This often results in poor fragmentation which can affect digging efficiency and normal production. Some mines try to use a higher powder factor in order to achieve a good fragmentation for higher production. In this case, the rock movement can cause significant dilution. Therefore, for most open pit gold mines, a compromise is needed between rock movement and fragmentation.

2. POTENTIAL PROBLEMS

The discussion above raises the following questions concerning ore grade control at operating gold mines, 1) Using the current ore grade control practice, what is the extent of the dilution due to rock movement caused by blasting? 2) Can the pre-blast polygons be used as the post-blast digging polygons by accepting some dilution caused by rock movement? 3) Is the rock movement due to blasting measurable and predictable? If rock movement can be predicted both in magnitude and direction, the pre-blast polygons can be modified so that dilution can be minimized without compromising rock fragmentation. Blast effect is an important component for an entire mining process since it can affect the loading, hauling, crushing, conveying and...
processing at a mine. Therefore, blast-induced dilution can reduce total gold production over the life of a mine. It appears from the survey that current ore grade control practices used in Nevada open pit gold mines leave room for improvement when considering dilution caused by blast movement. This research project was initiated with the expectation that, if successful, the results will be of considerable benefit to operating gold mines.

3. LITERATURE REVIEW

A number of papers, (Cherry, 1967; Langefors and Kihlstrom, 1979; Favreau, 1980; McHugh, 1983; Cunningham, 1983; Schamaun, 1983; Hyde and Favreau, 1986; Harries, 1987; Mohanty, Tidman and Jorgenson, 1989; Yang and Kavetsky, 1989 and 1990; Lucas and Nies 1990), discuss rock movement and rock fragmentation and muck pile formation. Several blast prediction models have been developed (Cherry, 1967; Langefors and Kihlstrom, 1979; Favreau, 1980; McHugh, 1983; Cunningham, C. 1983; Schamaun, 1983; Hyde and Favreau, 1986; Harries, 1987; ICI Explosives, 1988; Yang and Kavetsky, 1989 and 1990). Modelling blasting processes is no longer a new technology. However, little has been written on the relative success of these models to consistently and adequately predict the effect of a blast in the field. A blasting model is any relation between blast-related
parameters such as blast layout geometry, geological information, explosive properties and initiation system, and blast results (i.e. rock fragmentation, rock movement and muck pile shape). To date, only a few papers, (Yang and Kavetsky, 1989 and 1990; Lucas and Nies 1990), consider rock movement and its impact on ore grade control.

In 1988 Yang and Kavetsky developed a two-dimensional model to predict muck pile shape in bench blasting. In the following year the same authors presented a three-dimensional kinematic model of muck pile formation and grade boundary movement in bench blasting. Yang and Kavetsky's models, which are known as "mathematical" models, include the geometry of blast design, initiation pattern, boundary confinement, and explosive energy. Three assumptions are incorporated in the models. Firstly, when the rock begins to move it is assumed to have been already fragmented, either by a stress wave and/or from geological jointing. Secondly, it is assumed that the rock fragments move as projectiles with an initial velocity. Thirdly, it is assumed that the initial velocity of each fragment is determined by the energy imparted to the fragment from the energy source and the amount of resistance to movement received from the surrounding fragmented blocks. Limited data from a case study were used to calibrate the models. Further work is needed to test the capability of the model to accurately predict blast movement for a variety of geology.
rock strength, geometry of blast layout, initiation pattern, and explosive properties.

Lucas and Nies raise a question concerning rock fragmentation and ore grade control, and discuss a procedure to improve blast fragmentation and minimize the ore grade control problem at Homestake's McLaughlin Mine in California. The paper mentions that blast movement can cause a major problem in ore grade control in disseminated open pit gold mines, if better fragmentation is required to maintain production. No field measurements of rock movement due to blasting are given to support this deduction.

In 1987, ICI Explosives developed the Sabrex model (Scientific Approach to Breaking Rock with Explosives), which predicts rock fragmentation and muck pile profile. The model considers three essential elements: detonation properties, strength and elastic properties of rock, and blast geometry. Sabrex can accommodate the behavior of commercial explosives not only in an ideal or stable chemical reaction, but also in a non-ideal or unstable chemical reaction situation. The model considers the mode of initiation and the confinement provided by the surrounding rock. The properties of rock normally used as input to the model are: 1) density; 2) Poisson's ratio; 3) Young's modulus; 4) unconfined static compressive strength; 5) dynamic tensile strength and; 6) the shock attenuation coefficient. The inputs on blast
geometry are, 1) blasthole diameter; 2) blasting pattern; 3) blasthole inclination; 4) bench height; 5) depth of subdrilling; 6) collar distance; 7) initiation pattern and delay times. The model predictions include:

1. fragmentation size distribution
2. muck pile profile
3. grade level fragmentation
4. collar block fragmentation
5. flyrock control
6. backbreaking, including damage below subgrade
7. cost per unit volume of rock broken

Sabrex has been used very little for blast design analysis in surface gold mines, since the model is primarily designed for free face blasting, which is rarely the case in gold mining. In addition, little field verification of Sabrex in western US gold mines has been undertaken. The average rock movement can be estimated using muck pile profiles from Sabrex. A test of Sabrex in 1993 at the Coeur Rochester Mine, which has free face blasting, showed that the model significantly underestimated rock movement.
A literature review found no publication mentioning any program of systematic measurement relating to the study of rock movement caused by blasting, for bench production blasting at a gold mine. Such a program is essential to understand rock movement and its impact on ore grade control. Only after sufficient data are obtained from such an investigation can a blast prediction model be developed and calibrated. This will enable a gold mine to accurately predict and adjust for blast movement allowing it to achieve minimum dilution during the mining process.

• This thesis presents the results of internal and surface rock movement measurements at the Ram Mine of the Newmont Gold Company, and the Coeur Rochester Mine. A total of twelve blasts have been monitored at the two mines. An evaluation of current ore grade control practices is given. Conclusions and recommendations are relevant for operating gold mines to increase mining accuracy and minimize dilution.
CHAPTER 2  SURVEY OF BLASTING PRACTICES USED IN NEVADA OPEN PIT GOLD MINES

In order to investigate drilling/blasting and ore grade control practices used by Nevada open pit gold mines, a questionnaire was designed and distributed to 35 mines in Nevada. Twenty-seven companies participated in this survey and returned questionnaires. The list of participants is shown in Appendix A. Since some companies requested not to be identified with the data, a number is used for each mine.

1. CHARACTERISTICS OF GOLD DEPOSITS IN NEVADA

Table 2-1 shows a summary of deposit characteristics of the participating mines. The deposits can be classified into three major types: 18 are sediment hosted, 7 are related to volcanic activity, 4 are skarns, while 1 is sinter and intrusive hosted respectively. Disseminated gold deposits dominate, and constitute 21 out of the total, with the remainder showing vein and stock work characteristics. Over half of the mines surveyed classified their orebodies as disseminated sedimentary deposits.
Disseminated gold deposits exhibit very poor visible distinction between ore and waste boundaries. Two thirds of the mines consider it either impossible or difficult to visually distinguish boundaries between ore and waste, see Table 2-2. The 'possible' category means that error can occur in locating ore and waste boundaries, without any guidance from the blast hole assay results. Even when boundaries between ore and waste might possibly be determined from geological contacts, assay results from blast holes are commonly used to determine ore and waste polygonal positions.

<table>
<thead>
<tr>
<th>Boundary Visibility</th>
<th>Impossible</th>
<th>Difficult</th>
<th>Possible</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine's ID#</td>
<td>2, 7, 8, 13, ISA, ISB, 25, 26</td>
<td>1, 9, 10, 11, 12, 16, 19, 20, 24, 27</td>
<td>2, 3, 4, 5, 6, 17, 18, 21, 22, 23</td>
</tr>
<tr>
<td>Sum</td>
<td>8</td>
<td>10</td>
<td>9</td>
</tr>
</tbody>
</table>

Table 2-2: Boundary Visibility of Ore and Waste
The survey shows that a number of different rocks are encountered, dominated by two major types, i.e. sedimentary and igneous/volcanic. Table 2-3 shows that the sedimentary rocks are composed of shale/argillite, limestone, siltstone, sediments, clay and conglomerate. Igneous rocks are composed of tuff, andesite, volcaniclastic, basalt, and quartz. Most rocks from both groups are metamorphosed and weathered, and become argillized, silicified and decalcified. The ability to blast sedimentary rocks is reported to be from easy to medium. Three mines report that, occasionally, it is difficult to blast limestone, conglomerate and shale, while volcanic hosted deposits are commonly considered to be in the range from medium to difficult.
<table>
<thead>
<tr>
<th>Rock Types</th>
<th>Sediment Hosted</th>
<th>Volcanic Hosted</th>
<th>Skarn Hosted</th>
<th>Other*</th>
<th>Ability to be Blast</th>
</tr>
</thead>
<tbody>
<tr>
<td>Alluvium</td>
<td>20</td>
<td></td>
<td></td>
<td></td>
<td>Easy</td>
</tr>
<tr>
<td>Sediments</td>
<td>1, 12, 15, 27</td>
<td>1</td>
<td>1</td>
<td></td>
<td>Easy-Medium</td>
</tr>
<tr>
<td>Limestone</td>
<td>5, 7, 10, 11, 14, 15A, 25, 26, 27</td>
<td>1, 5, 14</td>
<td></td>
<td></td>
<td>Easy-Difficult</td>
</tr>
<tr>
<td>Sandstone</td>
<td>5, 15A</td>
<td>1</td>
<td></td>
<td>5</td>
<td>Easy-Medium</td>
</tr>
<tr>
<td>Siltstone</td>
<td>7, 17, 19, 20, 22, 25, 26</td>
<td></td>
<td></td>
<td></td>
<td>Easy-Medium</td>
</tr>
<tr>
<td>Shale</td>
<td>3, 5, 7, 10, 14, 17, 20</td>
<td>5</td>
<td></td>
<td></td>
<td>Easy-Difficult</td>
</tr>
<tr>
<td>Argillite</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Clay</td>
<td>14, 19</td>
<td>1</td>
<td></td>
<td></td>
<td>Easy</td>
</tr>
<tr>
<td>Conglomerate</td>
<td>16</td>
<td>1</td>
<td>16</td>
<td></td>
<td>Medium-Diff</td>
</tr>
<tr>
<td>Ash or Volcaniclastic</td>
<td>11</td>
<td>2, 8, 11</td>
<td>1</td>
<td>1</td>
<td>Medium-Diff</td>
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<tr>
<td>Tuff</td>
<td>11</td>
<td>8, 11, 12, 13, 24</td>
<td></td>
<td></td>
<td>Medium-Diff</td>
</tr>
<tr>
<td>Andesite</td>
<td>6, 9, 11, 3, 18</td>
<td>-</td>
<td></td>
<td></td>
<td>Medium-Diff</td>
</tr>
<tr>
<td>Basalt</td>
<td>15A, 20</td>
<td>4</td>
<td></td>
<td></td>
<td>Medium-Diff</td>
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<tr>
<td>Quartz</td>
<td>18</td>
<td></td>
<td></td>
<td></td>
<td>Difficult</td>
</tr>
<tr>
<td>Rhyolite</td>
<td>4, 6</td>
<td></td>
<td></td>
<td></td>
<td>Difficult</td>
</tr>
</tbody>
</table>

* Including sinter and intrusive hosted types

Table 2-3: Rock Types and Their Ability to be Blasted
BLASTING PRACTICES

1) Blast Size

Due to the geological complexity of gold deposits, a 20 foot bench height is widely used, as is shown in Table 2-4. Some mines, i.e. #20, #21, #23, and #27 use a double bench height in waste areas. The size of the blasts ranges from 10,000 to 117,000 ft² in area, which is equivalent to 100x100 to 345x345 ft. The tonnage of material is in the range of 4,000-260,000 tons, while most individual blasts are less than 100,000 tons. The number of blast holes per blast varies from 30 to 650. The amount of explosive weight charged in each blast varies from 2,500 to 167,000 lbs, with over 90 percent less than 100,000 lbs. Mine #8 ranks highest in amount of explosives used per blast in an ore zone, and mine #11 for waste, represented by 167,000 and 280,000 lbs respectively, see Table 2-4. Generally, at each operating mine, there is a tendency to use small blasts in ore and large blasts in waste.
<table>
<thead>
<tr>
<th>Item</th>
<th>Range</th>
<th>Mine's Tons (Ore)</th>
<th>Sum</th>
<th>Mine's ID (Waste)</th>
<th>Sum</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Bench Height (FT)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>10a</td>
<td>It</td>
<td>1</td>
<td></td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>15±</td>
<td>1.3,13,17,21</td>
<td>5</td>
<td>1, 5, 13</td>
<td>3</td>
<td></td>
</tr>
<tr>
<td>20±</td>
<td>3.4,6,7,9, 90,11,14, 13A, 16, 19,20, 22,23,24, 26,27</td>
<td>17</td>
<td>3, 4, 6, 7,9,10,14, 15A, 16, 17, 19, 22, 23, 24</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>2±</td>
<td>2,3, 25</td>
<td>3</td>
<td>2, 3</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td>30t/35/40t</td>
<td>12, 19B.23</td>
<td>3</td>
<td>11, 12, 19B.20,21, 23, 27</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td><strong>Typical Areas One Blast (1000ft²)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>-20</td>
<td>1, 4, 5, It</td>
<td>4</td>
<td>2, 4, 3, It</td>
<td>4</td>
<td></td>
</tr>
<tr>
<td>20-30</td>
<td>1.3,5,12,13B, 17,21,22, 24</td>
<td>1</td>
<td>1, 5,12B, 17,22,24</td>
<td>6</td>
<td></td>
</tr>
<tr>
<td>30-100</td>
<td>7,9, 11, 14, 15A, 16, 19, 23, 26,27</td>
<td>10</td>
<td>7,9, 11, 12, 14, 15A, 16, 19,23,26,27</td>
<td>11</td>
<td></td>
</tr>
<tr>
<td>Over 100</td>
<td>6, 7</td>
<td>2</td>
<td>6, 7, 25</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td><strong>Total Material Blasted per Blast (1000oz3con)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>-30</td>
<td>2.3,4,5,9,17,11B,22,27</td>
<td>9</td>
<td>2, 3,4, 5, 9, 17, 11B</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>30-100</td>
<td>1.7, 10,12,13A, 15A, 19, 22,23,24,26</td>
<td>11</td>
<td>1.7, 10, 13A, 15A, 19, 22,23,24,26</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>100-200</td>
<td>6, 11, 14, 15B, 9.6,23,27</td>
<td>7</td>
<td>6, 11, 12, 14, 15B, 9.6,23,27</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>Over 200</td>
<td>t</td>
<td>1</td>
<td>t, 20,21,27</td>
<td>4</td>
<td></td>
</tr>
<tr>
<td><strong>Typical number of Blasts/Hole One Blast</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>-100</td>
<td>2.3,4, 5, 9,17,20</td>
<td>7</td>
<td>2.3,4, 5, 9,17,20</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>100-230</td>
<td>1.7, 10, 12, 15A, 15B, 11B, 21,22,23, 24,23,26,27</td>
<td>14</td>
<td>1.7, 10, 12, 15A, 15B, 11B, 21,22,23,24,23, 26,27</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>230-430</td>
<td>6, 11,13,14, 16, 19,23</td>
<td>7</td>
<td>6, 11, 13, 14, 16, 19, 23</td>
<td>7</td>
<td></td>
</tr>
<tr>
<td>430-630</td>
<td>t</td>
<td>1</td>
<td>t, 11</td>
<td>2</td>
<td></td>
</tr>
<tr>
<td><strong>Total Width of Explosive One Blast (1000oz3con)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>-30</td>
<td>2.3,4, 3, 9, 10, 13, 17, 11B, 20,21,24</td>
<td>12</td>
<td>2.3,4, 3, 9, 10, 13, 17, 11B, 20,21,24</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>30-100</td>
<td>1.6,7, 11, 12, 14, 15A, 15B, 16, 22, 23,26,27</td>
<td>13</td>
<td>1.6,7, 12, 14, 15A, 15B, 16, 20,21,22, 25,26,27</td>
<td>14</td>
<td></td>
</tr>
<tr>
<td>100-200</td>
<td>t</td>
<td>1</td>
<td>t</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>Over 200</td>
<td>0</td>
<td></td>
<td>11</td>
<td>1</td>
<td></td>
</tr>
</tbody>
</table>

Table 2-4: Blast Size, Total Material and Explosive Charge
2) Blast Hole Layout

In order to obtain evenly distributed samples, 20 mines use a square pattern, as shown in Table 2-5, 9 mines use a staggered layout. The survey indicates that the same spacing and burden are used by all mines using staggered blast hole layout. Four mines use rectangular patterns.

<table>
<thead>
<tr>
<th>Pattern</th>
<th>Square</th>
<th>Staggered</th>
<th>Rectangular</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine’s ID#</td>
<td>1, 2, 4, 5, 8, 10, 12, 14, 15A, 15B, 17, 19, 20, 21, 22, 23A, 24, 25, 26, 27</td>
<td>3, 4, 9, 11, 13, 18, 21A, 23A, 24</td>
<td>6, 7, 16, 24, 25, 26</td>
</tr>
<tr>
<td>Slim</td>
<td>20</td>
<td>9</td>
<td>6</td>
</tr>
</tbody>
</table>

- Only ore
- Only waste

Table 2-5: Summary of Blast Hole Patterns

3) Burden and Spacing

Table 2-6 summarizes the burden and spacing used at the mines. It indicates that three patterns, namely 14x14, 15x15 and 16x16 ft, are the most common. Tables 2-5 and 2-6 indicate that square patterns are more common so that an even distribution of samples can be obtained with a square layout. This might lead to better estimation of the ore/waste boundary and achieve more accurate modeling for grade.
control purpose.

The results from the survey indicate that some mines use a short bench with a small burden/spacing when blasting ore. When blasting waste, higher bench with larger burden/spacing or double bench techniques are used. For example, Mine #17 uses a 15 foot bench in ore and 20 foot in waste, with the same burden and spacing. Mine #20 uses a 14x14 foot pattern in ore with a bench height of 20 feet, and a double bench height of 40 feet and 28x28 foot pattern in waste, see Table 2-6.

<table>
<thead>
<tr>
<th>Patterns Burden x Spacing (ft)</th>
<th>Mine's ID# Corresponding to Bench Height</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>15 ft</td>
</tr>
<tr>
<td>12x12</td>
<td>3, 11°</td>
</tr>
<tr>
<td>14x14 or 12x16</td>
<td>5, 21°</td>
</tr>
<tr>
<td>15x15 or 14x16</td>
<td>13°, 16°</td>
</tr>
<tr>
<td>16x16</td>
<td>1</td>
</tr>
<tr>
<td>18x18</td>
<td></td>
</tr>
<tr>
<td>Over 20x20</td>
<td></td>
</tr>
</tbody>
</table>

* Only in ore zone.
* Only in waste zone.

The rectangular patterns will cover about the same area as square layout corresponding to 14x14 ft or 15x15 ft.

Table 2-6: Summary of Burden and Spacing
4) Blast Hole Diameter

A larger diameter blast hole is usually matched with a higher bench. However, in the case of geologically complicated deposits, the higher the bench, the bigger the vertical sampling interval, and the higher the dilution. Table 2-7 illustrates that most mines use a combination of 20 feet bench and 6¼", 6½", 6¾" blast hole diameters. The larger diameters of blast holes, 7¾" - 9¾", are used either in the situation of handling over 50 million tons of material per year, or with a bench height of 30 and 35 feet, i.e. Mines #11, #12 and #27.

<table>
<thead>
<tr>
<th>Bench Height (ft)</th>
<th>&lt; 6 Inch Diameter</th>
<th>6¼, 6½ and 6¾ Inch Diameter</th>
<th>7¾─9¾ Inch Diameter</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Mine's ID#</td>
<td>Sum</td>
<td>Mine's ID#</td>
</tr>
<tr>
<td>10</td>
<td>18</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>15</td>
<td>1, 13, 17°, 21°</td>
<td>4</td>
<td></td>
</tr>
<tr>
<td>20</td>
<td>9</td>
<td>1</td>
<td>3, 4, 6, 7º, 9, 10, 11º, 14, 15º, 17º, 19, 20º, 22º, 23º, 24º</td>
</tr>
<tr>
<td>25</td>
<td>2, 8</td>
<td></td>
<td></td>
</tr>
<tr>
<td>30</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>35</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>40</td>
<td>15, 20º, 21º, 23º</td>
<td>4</td>
<td></td>
</tr>
</tbody>
</table>

o ow

The combination of Bench height and blast hole diameter is only used in ore zone.
The combination of bench height and blast hole diameter is only used in waste zone.

Table 2-7: The Combination of Bench Height and Blast Hole Diameter
5) Explosives

For most of the surveyed mines, since over 95 percent of the blast holes are dry, water is not a major concern. This is primarily due to the prevailing drought in Nevada which has lowered the water table, which is impacted also by dewatering, a method used to improve slope stability in several deep mines. However, if rainfall increases in the future, this situation may change. ANFO is the major explosive used, since it is the cheapest commercial explosive. Table 2-8 indicates that 21 mines load ANFO in dry holes. Heavy ANFO, Emulsion, and Slurry are usually charged in wet holes. It appears that only ANFO is used if at least 99 percent of the holes are dry, in order to simplify explosive loading. Three mines, #1, #8 and #11, use Heavy ANFO in both dry and wet hole conditions due to either reported difficulty in Blastibality of rock or wet holes.

<table>
<thead>
<tr>
<th>Explosives</th>
<th>Percent of Dry Blast holes</th>
<th>Sum</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Greater Than 99%</td>
<td>99% - 95%</td>
</tr>
<tr>
<td>ANFO</td>
<td>2, 4, 5, 6, 15A, 15B, 16, 17, 19, 21, 23, 25</td>
<td>3, 10, 12, 13, 22, 24</td>
</tr>
<tr>
<td>Heavy ANFO</td>
<td>1, 15A, 21</td>
<td>3, 22, 24</td>
</tr>
<tr>
<td>Emulsion</td>
<td></td>
<td>14, 20, 26, 27</td>
</tr>
<tr>
<td>Slurry</td>
<td>12</td>
<td>9, 11,</td>
</tr>
<tr>
<td>Sum</td>
<td>13</td>
<td>6</td>
</tr>
</tbody>
</table>

Table 2-8: Types of Explosives Used at Nevada Gold Open Pit Mines
6) Powder Factors

The energy released per unit of a explosive is different from one explosive to another. In order to compare the powder factor for the same explosive, a unit conversion is needed based on the energy released from an explosive. For example, if it is assumed that the unit energy released from ANFO is 100, Heavy ANFO will be 130 with an emulsion and ANFO blend ratio of 30:70. 1.27 lbs Heavy ANFO/yard\(^3\) is equivalent to 1.65 lbs ANFO/yard\(^3\) in standard energy output. Table 2-9 summarizes powder factors and their relation to overall Blastibality. The powder factor varies over a wide range because of different geology, rock type and strength, and rock ability to be blasted. 0.59-1.65 lbs ANFO/yard\(^3\) is used in ore zone blasting, 0.36-2.34 lbs ANFO/yard\(^3\) in waste. It appears that powder factors for blastibality ranging from easy to medium are usually less than 1.0 lbs ANFO/yard\(^3\), and rock of difficult blastibality requires a powder factor of up to 1.5 lbs ANFO/yard\(^3\), i.e. mines #8 and #11. Mine #8 uses the highest powder factor in the ore zone, and mine #11 is the highest in waste.
Blastibility Powder Factor (p) (lbs ANFO/yard')

<table>
<thead>
<tr>
<th>Blastibility</th>
<th>Powder Factor (p) (lbs ANFO/yard')</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>p ≤ 0.6</td>
</tr>
<tr>
<td>Easy Ore</td>
<td>ISB</td>
</tr>
<tr>
<td></td>
<td>Waste</td>
</tr>
<tr>
<td>Medium Ore</td>
<td>3, 17</td>
</tr>
<tr>
<td>Waste</td>
<td>3, 17, 20, 25</td>
</tr>
<tr>
<td>Difficult Ore</td>
<td>16, 27</td>
</tr>
<tr>
<td>Waste</td>
<td>24</td>
</tr>
</tbody>
</table>

The units of other type of explosives were converted into equivalent ANFO unit based on the energy index released per unit weight.

Table 2-9: Summary of Powder Factors

7) Initiation System

Three patterns are used for an initiation sequence as shown in Table 2-10:

1. Surface Delay Pattern.
2. Down Hole Delay Pattern.
3. Surface and Down Hole Delay Pattern.
The surface delay is normally completed by putting surface delay caps between designated rows. Detonating cord or an instant detonator is directly connected to a down hole primer, and there is no down hole delay time. Several mines use the same period detonators to initiate down hole primers. The same initiation delay for a down hole primer is achieved by down hole delay detonators. Actual delay is achieved by surface delay caps. This case is considered as another form of surface-delay pattern, which is discussed later. The down hole delay is obtained by using different period detonators to connect down hole primers. Each blast hole is hooked up by detonating cord. Sometimes, bridge delay caps on the surface might be used between different initiation blocks in order to extend the number of delay periods. Down hole and surface delay patterns use surface and down hole delays. The actual delay between two rows or holes is achieved by both surface and down hole delays, and this always results in a hole-by-hole initiation sequence. Table 2-10 illustrates that more companies prefer to use a surface delay pattern. Generally, delay time between rows or holes is in the range of 25-100 ms, and 65 ms is mostly used as the surface delay time.

Hole-by-hole initiation is reported by six mines. Seventeen mines use row-by-row in the form of a V or half wing of a V. Half of the mines initiate a blast from the free face, half of them fire against a buffer. One mine, #17, blasts against solid rock
when in ore.

<table>
<thead>
<tr>
<th>Initiation Patterns</th>
<th>Mine's ID#—Surface Delay Time / Down Hole Delay Time</th>
<th>Sum</th>
</tr>
</thead>
<tbody>
<tr>
<td>Surface Delay</td>
<td>3—25/0, 7—100/0, 8—-65/0, 11—-65/0, 14—-65/0, 16—-(25-100)/0, 17—-17/0, 19—-65/0, 22—-100/0, 23—-42/0, 25—-65/0, 26—-100/0, 27—-(17-25)/0</td>
<td>10 25%</td>
</tr>
<tr>
<td>Down Hole Delay</td>
<td>5—0/25, 6—0/25, 9—0/30, 10—-0/25, 13—-0/50</td>
<td>5 23%</td>
</tr>
<tr>
<td>Surface and Down Delay</td>
<td>1—-65/100, 2—-65/25, 11—-65/100, 12—-50/150, 15A—-35/50, 15B—-35/50, 20—-(65 &amp; 100)/50, 21—-100/100</td>
<td>7 32%</td>
</tr>
</tbody>
</table>

Table 2-10: Summary of Initiation Patterns and Delay Time

3 FRAGMENTATION RESULTS

Table 2-11 presents a summary of fragmentation results. Most mines report good fragmentation in waste blasts. Nearly half of the mines report that poor fragmentation is sometimes encountered in an ore blast. The survey indicates that rock types and structure are considered to be the main causes of poor fragmentation. Half of the mines consider that this is the result of caved and/or short holes. Only a few mines indicate blast design and/or water as a reason for poor fragmentation.

Most mines report that oversize seldomly occurs in waste blasting. 9 mines
indicated that oversize sometimes appears in ore block blasting. The oversize is handled by either secondary blasting or a mechanical impact method.

<table>
<thead>
<tr>
<th>Frequency of Poor Fragmentation</th>
<th>Ore: Mine's ID#</th>
<th>Sum</th>
<th>Waste: Mine's ID#</th>
<th>Sum</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sometimes</td>
<td>1, 2, 9, 10, 12, 13, 16, 17, 18, 19, 22, 23, 24, 25</td>
<td>14</td>
<td>1, 2, 9, 10, 11, 14, 18, 22</td>
<td>8</td>
</tr>
<tr>
<td>Rarely</td>
<td>3, 4, 5, 6, 7, 8, 11, 14, 15A, 15B, 20, 21, 26, 27</td>
<td>14</td>
<td>3, 4, 5, 6, 7, 8, 12, 13, 15A, 15B, 16, 17, 19, 20, 21, 23, 24, 25, 26, 27</td>
<td>20</td>
</tr>
</tbody>
</table>

Table 2-11: Summary of Fragmentation

4 ROCK DISPLACEMENT CAUSED BY BLASTING

Five mines, #4, #6, #11, #15B and #24, carried out limited measurements of rock displacement caused by blasting, but none of them provided any quantitative results. Several methods are reported being used to measure rock movement, e.g. bags as markers in the blast holes, chalking blast holes, and surveying blast hole location on the muck pile surface after blast. Two mines think that rock movement sometimes causes a problem in ore grade control. It is reported that excessive rock movement occurs quite frequently at mine #24, which results in a serious grade control problem at this mine.
The survey indicates that the blast holes are usually laid out by surveyors, and are numbered in a logical sequence. Blast hole samples are collected by the drillers using through the deck or pie pan samplers. Typically, samples are split and assayed in a laboratory. Only one mine splits samples in the field. Half of the mines use through the deck sampling, and half use a pie pan. Most mines carry out error and bias analysis in order to check sampling procedures and representativeness of assay results. Three mines sample subdrilling, one mine takes two samples from each blast hole.

Five mines conduct limited measurements of rock movement. However, only one occasionally adjusts a pre-blast polygon for the blast direction. One mine blasts in the direction of the strike of the vein. Most mines do not have any information on rock movement, one mine reported that it considered rock movement but decided that it probably resulted in less than 10 percent dilution.
CHAPTER 3 INSTRUMENTATION AND METHODOLOGY

1. RESEARCH OBJECTIVES

Excessive rock movement might be one of the major factors which impact on ore grade control in Nevada open pit mines. The amount of rock movement is determined by multiple variables such as rock strength, powder factor, initiation pattern and etc.

• The following were the research objectives:

1) To measure surface and interior rock movement of a bench due to blasting.

2) To develop a practical method for a mine to modify polygon positions on the blasted muck pile surface in order to compensate for rock movement due to blasting.

3) To develop improved digging plans in order to reduce dilution.

4) To improve blast design practices in order to minimize the rock movement without compromising rock fragmentation and loading efficiency.
2. INSTRUMENTATION

The research projects were carried out at both the Coeur Rochester Mine, Lovelock, Pershing county, and the Rain Mine, Newmont Gold Company, Carlin, Elko county. The research activities involved the use of several survey instruments to measure and monitor bench surface and internal rock movement caused by blasting. The following measuring equipment was used:

1) MDL Quarryman – a rock face laser surveying system.
2) Wild T2000 and Topcon total stations.

Figure 3-1: Quarryman Laser Profiler System
1) Figure 3-1 shows the Quarryman laser profiler instrument. The system employs a laser system which reflects a pulsed beam of laser energy off a rock face. The Quarryman measures distances of up to 400 meters (1200 ft) from a rock surface. The multiple observations are automatically stored in a data collector, and the data can be transferred to a PC computer. Special software then converts the data into coordinates for survey points, which makes it possible to calculate the distance and direction according to pre- and post-blast measurements at a particular location. A unique advantage of the laser profiler is that it allows a surveyor to measure a point without using a rod man. This avoids unsafe locations such as a loading site and working slope, and minimizes interruption of the loading operation.

The time interval when collecting two different points in the field is less than one second. Because of this high data collecting speed and the long range of laser beam reflection, it is very efficient to use the Quarryman to profile a muck pile or slope in an open pit mine. Three muck piles were profiled using the Quarryman. The results are discussed in Chapter 4.

2) Some of the rock movement data at the Coeur Rochester Mine were obtained using a Wild T2000 total station. A Topcon total station was used at the Rain Mine to transfer the station coordinates for the Quarryman in the pit. The accuracy of the
Wild T2000 and Topcon is up to one inch per 20,000 feet. Therefore, the Wild T2000 was used to check the accuracy of the Quarryman at the Coeur Rochester Mine. The coordinates of two points were surveyed by both the Wild T2000 and the Quarryman. The horizontal distance between the two points was 784 feet, and 203 feet vertically.

Table 3-1 illustrates the coordinate differences for the measurements. Because the accuracy of the total station is so high, it can be assumed that the difference between the coordinates of the two instruments is a measure of the error in the Quarryman survey system. The results in Table 3-1 indicates that the vertical accuracy of the laser profiler system is lower than the horizontal.

<table>
<thead>
<tr>
<th>Difference (Inch)</th>
<th>Northing</th>
<th>Easting</th>
<th>Elevation</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.2</td>
<td>1.5</td>
<td>6.5</td>
<td></td>
</tr>
</tbody>
</table>

Table 3-1: Accuracy Test of Quarryman Profiler

Table 3-1 illustrates that the Quarryman laser profile system cannot be used when a highly accurate survey is required. The system can produce a one inch error in the horizontal measurement and six inches in the vertical, over a survey range of up to 700 feet (200 meters).

Two aspects were considered during the surveying in order to minimize the
survey error by the laser profiler. Firstly, the Quarryman was never set up at a station more than 300 feet away from the surveyed points. Secondly, the station for the Quarryman set up was surveyed by either the Wild T2000 or the Topcon total station. This ensured that the error from the Quarryman's measurements was less than one inch for Easting/Northing, and three inches in elevation.

3 RESEARCH PROCEDURES

Measuring rock displacements induced by blasting was the primary field activity of this research project. The field measurements were only carried out in daylight hours. In order to measure internal rock movement, additional drill holes, which were not loaded with explosives, were used in the test blasts. These holes were usually placed in the middle of a regular blast hole pattern, and loaded with internal movement markers (or bags), as shown in Figure 3-2. A group of additional holes was used to ensure adequate bag coverage in order to obtain enough rock movement information.

The following was the typical research procedure used in the field:

1) The positions of additional holes were determined before blasting, according to information on blast design, bench geology and grade control.
Figure 3-2: Positions of Additional Holes

2) Three feet long by five inch diameter bags were painted in stripes with different colors, and filled with fine rock. These bags were used as down hole markers to measure the interior rock movement. For example, an additional hole was loaded with four or five bags which were painted from one to five stripes of the same color, and a different color was assigned to each hole. This provided a unique identification for each individual bag within one group of internal rock movement bags.

3) After the holes were drilled, the colored and coded bags were loaded into the additional holes in five feet vertical intervals, starting from lower bench grade
level. Drill hole cuttings were placed in between each bag, see Figure 3-3. Bench heights of 20 and 25 feet were used by the Rain and Coeur Rochester mines respectively, therefore, four bags were loaded in each additional hole at the Rain, and five at the Coeur Rochester Mine.

![Figure 3-3: Additional Hole Loading Configuration](image)

(A) Rain Mine of Newmont  
(B) Coeur Rochester Mine

4) After the blast holes were loaded, numbered wooden stakes were placed in the collars of selected holes in order to measure surface movement.
5) Before blasting, the positions of the additional holes and surface markers were located using either the total station or the Quarryman.

6) After blasting, the post-blast positions of the surface markers were surveyed by the total station or the Quarryman. The post-blast position of a surface stake was counted only when the stake was found, and it remained in the collar of the blast hole, or there was enough evidence to ensure that it was in a post-blast position, such as the presence of drill hole cuttings, or a tube of a downhole detonator.

7) When a shovel was loading, the Quarryman was set up in a safe position. The bag location was surveyed when it was exposed on the slope of a working face. The number of bag stripes and color was read through the Quarryman's telescope, and confirmed by the loader operator.

8) Pre- and post-blast surveying data were input into a computer, and the distance and direction of rock movement were calculated.

9) Horizontal and cross-sectional rock movement maps were generated by AutoCAD. The maps show the amount and direction of rock movement,
which makes it possible to relate rock movement to the blast layout, initiation sequence and digging polygons. Some internal rock movement bags were not recovered due to the following reasons:

(1) The area containing the internal movement bags was excavated during night and weekend shifts.

(2) When the bucket of a shovel hit a bag on the bottom of a bench, the loader operator could not see the bag under the bucket, and as the bucket left the working face, loose material fell down and covered the bag. Therefore, it was difficult to find bags #4 and #5 even during day shift.

(3) Sometimes a bag from the upper area of a working face was dislocated, and there was no evidence to indicate the original post-blast location. In this case the bag was ignored.

To improve bag recovery, more effort is needed to coordinate the loading operation and bag movement monitoring in order to recover more bags, and accurately locate their post-blast positions.
CHAPTER 4 ROCK MOVEMENT STUDY AT
THE RAIN MINE

1 GEOLOGY AND ROCK TYPES

The Rain Mine is a disseminated gold deposit located at the southern end of the Carlin Trend, 13 miles south of Carlin, Nevada. Gold is found in rock from the Webb formation of Mississippian Age. The deposit is along a major fault in the North-South direction. Rocks in the ore zone are sandstone, siltstone and mudstone, and waste includes barite, dolomite and limestone. Rock strength changes from soft to hard depending on the extent to which it has been silicified, argillized and calcified. Generally the rocks in ore are soft, and are quite easy to blast. The rock Blastibility in waste ranges from medium to moderately hard.

2 BLASTING PRACTICES

The blast layout at the Rain Mine, for a 20 foot high bench, consists of 6¾" diameter blast holes on a 14' x 14' center pattern with 1-2 feet subdrilling. The powder factor is around 0.3-0.4 lb ANFO/Ton. Typically, detonation cord is directly connected to a primer. A 65 millisecond (ms) delay between the rows is achieved by
using surface delay detonators. Two initiation patterns are used. One initiates from the edge of the blast, which is closer to a free face or blasted rock. The other initiates the holes from the middle of the blast. Of the eight blasts monitored at the Rain Mine, three were initiated at the edge and three from the middle. Two down hole delay blasts, the same practice used at the Gold Quarry Mine, were also monitored at the Rain Mine in order to compare results. 250 ms down hole delays and 50 ms surface delays were used in the down hole delay pattern. The initiation pattern was an echelon V.

3. FIELD MEASUREMENTS AND RESULTS

Six blasts with three different initiation patterns were monitored in August, 1992. Two blasts were profiled in October using a Quarryman laser profiler. Table 4-1 summarizes the six blasts shown in Figures B-1 to B-6, Appendix B. The blast layouts and bench geology are also illustrated in these figures. Figures B-7 to B-12 in Appendix B illustrate the surface marker positions, horizontal movement vectors, and initiation patterns.
### Table 4-1: Summary for the Six Blasts Monitored at the Rain Mine

<table>
<thead>
<tr>
<th>Blast Number</th>
<th>#1</th>
<th>#2</th>
<th>#3</th>
<th>#4</th>
<th>#5</th>
<th>#6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of Holes</td>
<td>157</td>
<td>75</td>
<td>52</td>
<td>153</td>
<td>90</td>
<td>108</td>
</tr>
<tr>
<td>Stemming (ft)</td>
<td>157</td>
<td>152</td>
<td>149</td>
<td>156</td>
<td>155</td>
<td>16</td>
</tr>
<tr>
<td>Explosive Length (ft)</td>
<td>73</td>
<td>78</td>
<td>81</td>
<td>74</td>
<td>75</td>
<td>70</td>
</tr>
<tr>
<td>Weight of explosive per hole (lbs of ANFO)</td>
<td>10W</td>
<td>107</td>
<td>111</td>
<td>102</td>
<td>104</td>
<td>96</td>
</tr>
<tr>
<td>Powder Factor (lb/ton)</td>
<td>0.33</td>
<td>0.35</td>
<td>0.37</td>
<td>0.34</td>
<td>0.34</td>
<td>0.32</td>
</tr>
<tr>
<td>Maximum Charge per Delay (lbs of ANFO)</td>
<td>3300</td>
<td>3070</td>
<td>1554</td>
<td>1632</td>
<td>2040</td>
<td>1440</td>
</tr>
<tr>
<td>Rock Types</td>
<td>Silstone Mudstone</td>
<td>Silicious Mudstone</td>
<td>Bane Mudstone</td>
<td>Siltstone Sandstone</td>
<td>Limestone Mudstone</td>
<td>Limestone Sandstone</td>
</tr>
<tr>
<td>Rock Strength</td>
<td>Compact, Moderate Hard</td>
<td>Compact, Moderate Hard</td>
<td>Soft</td>
<td>Soft to Moderate hard</td>
<td>Compact, Hard</td>
<td>Compact, Moderate hard</td>
</tr>
<tr>
<td>Down hole Delay (ms)</td>
<td>0-iv</td>
<td>250</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>250</td>
</tr>
<tr>
<td>Row-to-Row Delay (ms)</td>
<td>65</td>
<td>50</td>
<td>65</td>
<td>65</td>
<td>65</td>
<td>50</td>
</tr>
</tbody>
</table>

The plots only show those blast holes which had surface markers. The lines which connect the holes represent detonating cord. A number between the lines indicates the initiation sequence or delay time. An arrow shows the magnitude and direction of surface rock movement. When a hole is identified in the figure, but there is no arrow, this indicates the marker could not be found after the blast. A circled arrow represents the direction and the amount of interior rock movement. The number beside the arrow gives internal movement bag number. Bag numbers 1 through 4 represent their positions from the top to bottom of a bench, see Figure 3-2(a). Figures B-13 to B-18 are cross-sectional maps, showing internal rock movement for
Examination of Figures B-7 to B-18 yields the following observations:

1. Material moves either towards the initiation position (Figures B-7, B-8, B-11, B-12) or towards the free face (Figure B-10). If the blast was initiated from the edge of the bench, then the movement was towards the blasted rock (Figures B-7, B-8 and B-9). Figures B-10 and B-11 show that when initiation is in the middle of the blast, the rock at the bottom of the bench moves towards the initiation point, while the rock at the top of the bench moves away. Blast #6 did not yield much information because of a low bag recovery.

2. Rock moves in a direction almost parallel to the initiation direction (Figures B-7, B-8, B-10, B-11, B-12).

3. Rock closest to the free face moves further than rock some distance from the free face. Blast #4 was initiated from the center and there was a free face on the east side of the blast (Figure B-10). The surface movement was 10.5 feet for the second and third rows from the free face, while 3.5 feet movement occurred five rows from the free face.
4. The average movement of bags #1, #2, #3 and #4 is listed in Table 4-2. If the blast was initiated from the edge, as shown in Figures B-13 (blast #1), B-14 (blast #2), and B-15 (blast #3), material at the bottom of a bench moved further than that near the top of a bench.

Table 4-2: Average Movement of the Bags (feet)

<table>
<thead>
<tr>
<th></th>
<th>Bag #1 (Top)</th>
<th>Bag #2</th>
<th>Bag #3</th>
<th>Bag #4 (Bottom)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Blast #1*</td>
<td>3.5</td>
<td>5.6</td>
<td>7.2</td>
<td>3.7</td>
</tr>
<tr>
<td>Blast #2**</td>
<td>2.8</td>
<td>~ 5.2</td>
<td>6.0</td>
<td>N/A</td>
</tr>
<tr>
<td>Blast #3*</td>
<td>3.9</td>
<td>4.2</td>
<td>5.7</td>
<td>7</td>
</tr>
<tr>
<td>Blast #4***</td>
<td>4.3</td>
<td>3.5</td>
<td>2.5</td>
<td>1.7</td>
</tr>
<tr>
<td>Blast #5****</td>
<td>4.2</td>
<td>4.3</td>
<td>3</td>
<td>0.8</td>
</tr>
<tr>
<td>Blast #6*****</td>
<td>3.5</td>
<td>5</td>
<td>1.5</td>
<td>0.8</td>
</tr>
</tbody>
</table>

Note:  
* — Blasts initiated from the edge.  
** — Blasts initiated from the center.  
*** — An echelon V pattern was used to initiate the blasts.
4 CALCULATION OF AVERAGE ROCK MOVEMENT

Blasted rock movement typically depends on the powder factor, initiation pattern, geometry of the blast layout and rock mass geological and mechanical properties. The current mining practice interpolates the pre-blast polygon as a block with vertical contacts on a spatial boundary (shown in Figure 4-1A). During the blast the block is moved. The material in the middle of a bench appears to move further than that near to the top or bottom of a bench. The rock movement characteristics in the vertical direction also need to be considered when calculating the average polygonal block movement.

The average horizontal rock movement is the weighted average displacement of all the polygon layers weighted by layer thickness. This average rock movement can serve as a correction factor for locating the post-blast polygon position with reference to the pre-blast location. If a good estimate of the location of the post-blast polygon can be made, then by flagging it rather than the pre-blast polygon location, there is the potential to minimize dilution during rock excavation.
The average rock movement is a function of the rock displacement at different heights in a bench, and its value varies between the maximum and minimum movement of the bench. In order to simplify the calculation of the average rock movement, it is approximated to be equal to the horizontal displacement from the center of shaded area, O', to the pre-blast boundary, AB (shown in Figure 4-1B). Therefore, the following equation can be used to express the calculation of the average rock movement for a monitored point according to the rock displacement at different heights in a bench.

Figure 4-1: Polygonal Block and the Rock Movement (no scale)
Where:

\[ D_j = \sum K_j d_j \]

- \( D_j \) --- Average movement of a particular point (or a hole).
- \( d_j \) --- Rock movement at different heights in a bench.
- \( K_j \) --- Weighting factor for the rock movement at different heights in a bench \((\sum K_j = 1)\).
- \( j \) --- Number to identify the rock movement at different heights in a bench \((j = 1, 2, 3...\).

Since many factors affect the rock movement, and monitored points (or holes loaded with bags) are not placed exactly at the boundary contact, the average block movement of a polygon, which will be used to modify the pre-blast polygon, needs to be estimated according to the average rock movement of the monitored points inside or close to the polygon. This estimated block movement can be expressed as:

\[ D = \sum W_j D_j \]

- \( D_j \) --- Average movement of a particular point, calculated from \(4-1\).
- \( W_j \) --- Weighting factor to reconcile the influence of the rock movement for different monitored points. A point close to the polygonal boundary has more influence on the estimated block movement than does a point...
farthest away from the polygonal boundary ($x^W_i = 1$).

Monitored point number or additional hole number which is loaded with bags.

If every down hole marker is located after blasting, the rock movement in the vertical direction can be characterized, as is shown in Figure 4-1B.

AB is a pre-blast vertical line. Its post-blast position from A' to B' could be a curve. If it is assumed that the post-blast line A' to B' is straight, then the post-blast position for a point inside AB falls along the line A'B'. The average movement for the line AB can be represented as the rock movement in the middle of the bench or the average movement of four down-hole markers in a hole, i.e. $D_i = \frac{\sum D_j}{4}$.

If N holes are used to monitor the blast movement, the average movement for these holes is represented as $D_1, D_2, ... D_N$. The mean value of the average movement for these holes is assumed to be the average movement for the block boundaries, i.e. $D = \frac{\sum D_i}{N}$.

Overall, the bag recovery was 95% for the first and second bags, 70% for the third bags and only 22% for the bottom bags. Therefore, there was not enough data
to estimate the rock movement of a point for a polygonal block calculation, the arithmetic mean of the #2 and #3 bag displacement was used to represent the average movement, using the procedures discussed above. When bags #2 and #3 were found, the average movement of a blast or a particular area was calculated to be the arithmetic mean of the average movement of the additional holes. For example, Table 4-3 lists the horizontal movement of bags for blast #1.

Table 4-3: Down Hole Bag Movement for Blast #1 (feet)

<table>
<thead>
<tr>
<th>Horizontal Movement</th>
<th>Hole #1</th>
<th>Hole #2</th>
<th>Hole #3</th>
<th>Hole #4</th>
</tr>
</thead>
<tbody>
<tr>
<td>Down hole bag #2</td>
<td>5.6</td>
<td>7.7</td>
<td>3.9</td>
<td>6.7</td>
</tr>
<tr>
<td>Down hole bag #3</td>
<td>^ 5.5</td>
<td>10.1</td>
<td>8.2</td>
<td>4.7</td>
</tr>
<tr>
<td>Average Movement of #2 and #3</td>
<td>5.6</td>
<td>9.1</td>
<td>6.1</td>
<td>5.8</td>
</tr>
</tbody>
</table>

The average rock movement of hole #1 is \((5.6 + 5.5)/2\) or 5.6. The weighting factor for bags #2 and #3 is 0.5 respectively. The estimated movement for blast #1 will be expressed as \((5.6+9.1+6.1+5.8)/4\) or 6.6. The weighting factors, \(W\), are 0.25.

Following the above procedures, the average rock movement for the six blasts was calculated and is listed in Table 4-4:
Table 4-4: Average Horizontal Movement for the Six Blasts

<table>
<thead>
<tr>
<th>Blast Number</th>
<th>#1</th>
<th>#2</th>
<th>#3</th>
<th>#4</th>
<th>#5</th>
<th>#6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average Horizontal Movement (ft)</td>
<td>6.6</td>
<td>4.9</td>
<td>4.6</td>
<td>4.0</td>
<td>5.4</td>
<td>3.8</td>
</tr>
</tbody>
</table>

The average horizontal movement of an edge initiation pattern was 5.6 feet, for a middle initiation pattern 4.7 feet, and for an echelon V pattern 4.4 feet.

5 BLAST-INDUCED DILUTION

1 Grade Dilution

Dilution refers to the proportion by which the grade of ore is diluted. This can be expressed by the following equation:

\[ \delta_j = 1 - \frac{G_i}{G_j} \]  

Where:
- \( \delta_j \) --- Dilution
- \( G_i \) --- Average Grade Actually Mined From a Pre-blast Digging Polygon After Blasting.
- \( G_j \) --- Average Grade Estimated for a Polygon Before Blasting
In addition to the digging direction and minimum mining width, blast movement is one of the significant factors which may cause considerable dilution. Figure 4-2 illustrates blast induced dilution.

After a blast, typically, a digging polygon is located according to the pre-blast polygon determined by interpolation from the blast hole sampling and assay results obtained prior to blasting. This approach assumes that only vertical movement occurs during blasting and that there is no horizontal displacement.

Figure 4-2: Diagram of Blast-Induced Rock Movement (no scale)

Figure 4-2 illustrates the movement of a digging polygon. The material inside the pre-blast polygon is moved to the position of the post-blast polygon after blasting.
but the actual digging polygon is flagged at the location of the pre-blast polygon. Therefore, the material in the area S, (which could be waste or low grade ore, depending on the material surrounding the ore polygon), is mined as ore, and the ore in the area Sj could be mined as waste. Only the material in the area Sore is mined correctly. This impacts on the economics of the mine in three ways. Firstly, potential waste or low grade ore (S1) is treated as ore, which results in unnecessary processing costs in the mill. Secondly, transporting waste or low grade ore to the mill results in reducing throughput of mill grade ore. The results are higher costs and a decrease in gold production. Thirdly, the ore in area Sj could be mined as waste, which leads to a reduction in total gold produced.

2 Weight Dilution

If the polygon is a geological ore boundary, i.e. there is no mineral of value outside the ore block, the dilution (ti) can be expressed as the weight (W1) in the area S, (which is waste), divided by the total weight (W) mined from the digging polygon areas (S1 + Sore). This is illustrated in Figure 4-2.

Therefore: \[ \eta = \frac{W_1}{W} \]
Field measurements indicate that blasted rock moves in a direction which is almost perpendicular to the detonating cord (initiation line). If the row-by-row initiation method is used, the rock will move ideally in one direction except for the rock close to the boundaries. In the case of an echelon V initiation pattern, the rock on each side of the echelon V moves towards the center of the echelon V, Figure 4-3. It has been shown earlier in this chapter that the rock in the middle of the bench tends to move further than that at the top or bottom, but the amount of movement at the same height appears to be similar. From these rock movement characteristics, the following reasonable assumptions can be made concerning blasted rock movement:

Figure 4-3: Rock Moves Towards the Echelon-V-Center
1. The rock moves in one direction perpendicular to the initiation line for the row-by-row initiation pattern, or in a direction towards the echelon V center when located to the side of the V.

2. At the same height in a blast, the rock will move in the same direction, but the amount of movement will differ for each height.

Figure 4-4 illustrates three dimensional rock movement. The rock at different heights has different amounts of movement. The bench can be divided into levels. The post-blast ore block boundaries at different levels are adjusted according to the rock movement at each level. The dilution of each level can be calculated, and the actual dilution of the block can be expressed as the mean of the dilution occurring at each level.

![Figure 4-4: Rock Movement at Different Levels](image-url)
Figure 4-4 illustrates five levels for a bench. The horizontal movement in the middle of a level is used as the average horizontal movement for that level. For example, $d_1$ is the average horizontal movement of level 1, it is the horizontal movement of AB's middle point. Level 1 post-blast position is located based on the value of $d_1$, see Figure 4-5. The corresponding dilution $n_1$ can be calculated. The levels 2 through 5 can also be adjusted according their displacement, $d_2$, $d_3$, $d_4$, and $d_5$, and the corresponding dilutions, $n_2$, $n_3$, $n_4$, and $n_5$ can be obtained. The mean of $n_1$, $n_2$, $n_3$, $n_4$, and $n_5$ is used as the average block dilution:

$$h = \frac{\eta_1 + \eta_2 + \eta_3 + \eta_4 + \eta_5}{5}$$

![Figure 4-5: Post-blast Polygon Adjustment](image)
In order to simplify the calculation, the average block movement $D$, expressed as the distance $d_0$ discussed earlier in this chapter, see Figure 4-1, is used to adjust the post-blast block position. This procedure only needs to adjust the post-blast block location once instead of five times. The block dilution can be obtained according to the average block movement, since the dilution is a function of the rock displacement for the same block. The dilution obtained from the average block movement, $\frac{d_0}{H_B}$, is between the minimum and maximum dilution occurring at each level $n_i$ ($i = 1, 2, ..., n$), or:

$$\text{Min} (n_i, i=1,2,...n) \leq \frac{d_0}{H_B} \leq \text{Max} (n_i, i=1,2,...n)$$

If it is assumed that the rock at the same level moves the same amount in a particular direction, then the post-blast block boundaries representing the area will be translated by the same amount of displacement in the same direction with respect to the pre-blast boundaries. This means that the areas of pre- and post-blast blocks will be the same, Figure 4-2. The volume change, $\Delta V$, can be written as:

$$\Delta V = H'(S_{ore} + S_i) - H(S_{ore} + S_j)$$

Where:

- $H'$ --- Post-blast Bench Height
- $H$ --- Pre-blast Bench Height
- $S_{ore}$ --- Mined ore area
- $S_i$ --- Waste area
- $S_j$ --- Area of ore moved from pre-blast polygon, treated as waste
Since \( S_1 = S_j \), then \( S_{\text{are}} + S_1 = S_{\text{are}} + S_j = S \). \( S \) represents the digging polygon area.

\[
\Delta W = (H' - H)S = \Delta HS
\]

Where: \( S \rightarrow \text{Polygon area} \)

Equation 4-6 is derived according to assumptions discussed previously, that the volume change of the block, due to swelling, only occurs in the vertical direction. This is not the case in those areas close to the blast boundaries or an echelon V initiation center, since a change in the horizontal dimension does occur for these areas. However, if it is assumed that the swelling only occurs in the vertical direction, even though the height of a given block in the blast will be different before and after the blast, the weight of the given block will be the same, since the areas are identical.

According to Figures 4-4 and 4-5, the weight dilution can be written as:

\[
\eta = \frac{W_1}{W} = \frac{S_p H_p}{(S_{\text{are}} + S_j) H' p'}
\]

Where:

- \( W_1 \) --- Weight of Waste
- \( W \) --- Weight of a block
- \( p \) --- Pre-blast unit weight
- \( p' \) --- Post-blast unit weight

The weight of \( S_{\text{are}} \) can be written in terms of the post-blast height \( H' \) and unit
weight \( p' \), i.e. \( S_{w2} H \ p = S_{o2} H' p' \). Therefore, equation 4-7 can be written as:

\[
\eta = \frac{S_c}{(S_{w2} + S_2)} = \frac{S_c}{S}
\] 4-8

Typically, the muck pile is not level after a blast because of different swelling at different locations. The lower the swell factor, the tighter the blasted material, and the lower the muckpile height.

\( \eta \) calculated by formula 4-8 is a maximum dilution when the pre-blast polygons are geological boundaries (i.e. there is no mineral value outside of the boundaries). If the polygon position is determined according to an economic cutoff, (i.e. there is some mineral value outside boundaries, but the value is lower than the economic cutoff grade, and considered as waste), \( \eta \), calculated according to formula 4-8, can be explained as a weight dilution, and it represents a maximum estimation of grade dilution.

The calculation of real grade dilution must consider in three dimensions both the grade distribution outside the pre-blast polygon and the rock movement. The accuracy of the result is not fully known because there are several assumptions and judgements involved in all geostatistical models. In order to simplify the calculation, a weight dilution concept is introduced, and formula 4-8 is used to calculate the weight dilution, which is a maximum estimate of grade dilution.
However, it is reasonable to assume that if blast movement can be measured and estimated in three dimensions, then geostatistics can be used to interpolate three dimensional digging polygons, the locations of which can be suitably adjusted for blast movement.

3 Assumptions

In order to estimate the blast-induced dilution, first the pre-blast blocks have to be adjusted according to the rock movement characteristics. Based on the observations presented previously, the following assumptions are made:

1) Rock moves in a direction which is perpendicular to the initiation line (or detonating cord) in the area which is two rows (30 feet) away from the echelon-V-initiation center line.

2) Rock moves by converging to the center line of an echelon V, if it is within two rows from the center line.

Assumptions 1 and 2 apply to an echelon-V initiation pattern, rock on both sides of the V moves towards the echelon-V center, as is shown in Figure 4-3. Therefore, the height of the muck pile in the echelon-V center area is higher than the rest of the muckpile.
3) Rock moves towards the free face for the area within three rows from the free face.

4) A pre-existing blasted rock buffer does not change rock movement direction, but may reduce the amount of rock movement.

Assumptions 3 and 4 represent the boundary influence on rock movement. Generally, rock moves in a direction where less restriction exists. Rock, within two rows of a free face, may move towards that face. The rock moves perpendicular to the initiation line, since considerable restriction exists due to the blasted rock present from the preceding rows.

5) The amount of rock movement measured in specific locations is used to estimate the average interior rock movement throughout the whole blast area. This assumption means the amount of rock movement is the same in the entire blast except at the boundaries of the blast.

6) Dilution can be calculated using equation 4-8.

Based on these assumptions, the pre-blast polygon can be adjusted.
4 Dilution Induced by Blasting

The estimated dilution for each of the six blasts is listed in Table 4-5. Assuming the area of the pre-blast polygon is equal to the area of the post-blast polygon, then \( S_j = S_1 \), which means that the amount of material, possibly treated as waste in area \( S_j \), is equal to the amount of waste or low grade ore moved into the post-blast polygon (or digging polygon). The calculated dilution in Table 4-5 assumes that the rock in area \( S_1 \) is waste. However, this could also be low grade material because the polygon boundary may not be a geological ore contact, but rather an economic cut off. In this case, the dilution would be less.

<table>
<thead>
<tr>
<th>Blast#</th>
<th>#1</th>
<th>#2</th>
<th>#3</th>
<th>#4</th>
<th>#5</th>
<th>#6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dilution</td>
<td>N/A</td>
<td>N/A</td>
<td>3%</td>
<td>10%</td>
<td>N/A</td>
<td>10%</td>
</tr>
</tbody>
</table>

Table 4-5: The Blast-Induced Dilution for the Six Blasts

Figure 4-6 illustrates the shape of a pre- and post-blast polygon for blast #6.

The calculations to determine the dilution are as follows:

\[ S_1 = S_2 = 184 \text{ ft}^2 \]

and

\[ S_\text{ore} + S_1 = 1807 \text{ ft}^2 \]

therefore,

\[ \text{Dilution} = S_1 / ( S_\text{ore} + S_1) = 184/1807 = 10\% \]
Figure 4-6: Pre- and Post-blast Polygon for Blast #6

The dilution for blast #3 is only 3% because ore is on three sides of the polygon (Figure 4-7). There is no dilution for blasts #1, #2 and #5 since only waste was blasted.

Figure 4-7: Pre- and Post-blast Polygon for Blast #3
The following examples are used to discuss the influence of polygon size and movement direction on dilution (Figure 4-8).

**Case 1:** Assume that there is a 28 x 28 foot square polygon inside blast #1 and the movement of 6.6' is in a direction parallel to one side of the square (shown in Figure 4-8A).

Then, 
\[ S_1 = 6.6' \times 28' = 185 \text{ ft}^2 \]
\[ S_1 + S_{ore} = 28' \times 28' = 784 \text{ ft}^2 \]

Therefore, 
\[ \text{Dilution} = \frac{185}{784} = 24\% \]

**Case 2:** If a 10 foot horizontal movement occurred at blast #1, the dilution will be 36%.

**Case 3:** Suppose that the polygon is a 56 x 56 foot square, and the movement direction is still parallel to one side of the polygon, and the horizontal movement is 6.6'.

Where, 
\[ S_1 = 6.6' \times 56' = 370 \text{ ft}^2 \]
\[ S_1 + S_{ore} = 56' \times 56' = 3136 \text{ ft}^2 \]

\[ \text{Dilution} = \frac{370}{3136} = 12\% \]

**Case 3:** Assume the same conditions as in case 1, but the polygon is rotated 45°. Therefore, the rock movement is in the diagonal direction of the square (shown in Figure 4-8B).
Where, \[ S_1 = 240 \text{ ft}^2 \]

Dilution = \( \frac{240}{784} = 31\% \)

Figure 4-8: Assumed Polygon Shapes for Blast #1

From these examples, it can be concluded that the bigger the horizontal movement, the greater the dilution, and the bigger the polygon size, the smaller the dilution. If the rock movement is in the direction of the longest dimension of a polygon, there will be increased dilution.
In October, one blast ( #7 ) was profiled before and after blasting using the Quarryman laser profiler. 1125 and 1510 points were surveyed respectively for the pre- and post-blast profiles in blast #7. The location of each point was selected and this resulted in a reasonable but random coverage of the respective areas, see Figures B-19 and B-20 in Appendix B. The area to be blasted was bounded by free faces on the northeast and northwest sides, a blasted rock buffer on the southwest side and solid rock on the southeast side, as is shown in Figure B-21 in Appendix B. The layout for blast #7 consisted of 265 Wast holes, which are shown in Figure B-21. The blast was initiated from the side closest to the buffer, since it was intended to move the rock in the direction of the buffer. Delay time between rows was 65 MS and the powder factor was 0.25 lb ANFO/ton. The profiles were created using Surfer software and edited using AutoCAD.

Figure B-22 in Appendix B shows the post-blast muckpile contours for blast #7. The total volume in blast #7 was 1.26 million ft$^3$ (Figure B-21) or 97000 tons. The swell volume was 289000 ft$^3$, Figure B-23, giving a swell factor of 1.23. Since blast #7 was initiated from the buffer side rather than from the free face, either the buffer was pushed up, or the rock next to the buffer was thrown up onto it. Typically,
the surface elevation was increased approximately 5-7 feet in the buffer zone or in the area of blast #7 closest to the buffer. This is illustrated in the contour map of the difference between the pre- and post-blast surfaces, see Figure B-23 in Appendix B. Figures B-24 B-25 and B-26 in Appendix B are 3D profiles of the bench and muckpile surfaces and their respective differences. The plots are looking down 30 degrees in the southwest direction and give a graphic view of rock movement. More accurate information on rock movement can be obtained from the difference contour map in Figure B-23.

The initiation pattern intended to move the material in the direction of the buffer, rather than in the direction of the free faces. However, the material close to the free faces on the northeast and northwest sides was thrown in the free face direction because there was less constriction on these sides. The material in the area close to the bench crest was moved onto the haul road and the lower bench, which created a 5-9 feet muckpile, while the bench crest area went down 3-5 feet after blasting. This indicates that, in the bench crest area, the free faces have more influence on rock movement direction than the initiation pattern. The bench surface height was increased, on average, 1-3 feet in the middle area of the blast. Since there was no free face in any direction on the south end of the blast, the material had to move towards the initiation point, which resulted in a 9-feet high heave compared
with the original surface.

7 CONCLUSIONS FROM THE RAIN MINE MEASUREMENTS

1. During the blasting sequence, rock tends to move in a direction where there is less restriction such as a free face, blasted rock, or new space created by a neighboring row.

2. The material in the middle of the bench appears to move further than that near the top or the bottom of the bench.

3. Different initiation patterns only influence the direction of rock movement. There is no significant quantitative difference in the displacement magnitude between an echelon V initiation pattern and the patterns used at the Rain Mine.

4. A free face greatly affects the direction of rock movement in the area close to the free face. A free face does not have a significant influence on rock movement when approximately four rows or 50 feet away.

5. The larger the horizontal movement, the greater the dilution, and the smaller
the polygonal dimension, the larger the dilution. The dilution is related to the polygon shape and direction of rock movement. There will be higher dilution if the rock is moved along a shorter dimension of a polygon.

6. Surface profile measurements are feasible and can give effective information on overall rock movement, swell factor, relation between muck pile shape and explosives distribution.

7. Detailed analysis of the overall rock movement from profiles relative to the blast hole location, loading density, initiation pattern, rock properties etc. can give very useful information on blast efficiency and movement, and can help in determining a relationship between surface and internal rock movement.

8. If coordinates of royalty boundaries are known, the post blast volume of rock within a particular boundary can be calculated.
CHAPTER 5  ROCK MOVEMENT STUDY AT 
THE COEUR ROCHESTER MINE

1  GEOLOGY AND ROCK TYPES

The Rochester deposit is entirely contained by the upper volcanic and epiclastic rocks of the Permian-Triassic Koipato Group. The Koipato Group consists of the Limerick, Rochester, and Weaver Formations, in ascending order. Mineralization is hosted by the Weaver and upper Rochester Formations. Both formations are rhyolitic in composition and consist predominantly of ash-flow tuffs and flows.

Several hydrothermal and one metamorphic event have substantially altered the Weaver and Rochester formations. The rhyolite volcanics have been completely devitrified and recrystallized into a tough finegrained rock consisting of interlocking grains of quartz, feldspar, and sericite. Combined with the silicification associated with mineralization, the rock types at Rochester are extremely competent. Compressive strengths of 24,000 to 30,000 pounds per square inch are typical throughout the pit. Grinding and impact testing yield Bond indices that vary from 13 to 20.
The presence of structures, including veins and post-mineral faults, have a strong influence on the size and location of ore zones. Two primary structural trends control veins, one north-south and the other north-east. Where these structural systems intersect, especially near the Weaver-Rochester contact, ore mineralization broadens into large zones of stockworkings that represent the majority of ore at Rochester. The margins of the deposit are often characterized by singular vein trends with narrow ore blocks and minimal stockwork development.

Post-mineral faults, joint swarms, and underground workings create a complex structural environment at Rochester. The present day operation encompasses an area of underground mining activity from the early 1900's. Drifts and stopes are encountered on every bench. Many of these workings are filled with water. Faults are frequent, with east-west, north-south, and north-west trends being dominant. Conjugate joint swarms created by this faulting have fractured the rock throughout most areas in the pit. The distribution and orientation of these structural features strongly influence blast performance.
2 BLASTING PRACTICES

The blast layout for a 25 foot-high bench consists of 6 1/2" diameter blast holes on 15' x 15' square pattern with 3 feet subdrilling. The powder factor is around 0.57-0.76 lbs heavy ANFO/ton or equivalent to 0.68-0.91 lbs ANFO/ton. Typically, a 500 millisecond (ms) down hole detonator is connected to a one pound primer. The blast holes in the same row are linked by detonating cord. MS delay connectors on the surface are used between the rows to achieve the desired timing delay. An echelon V initiation pattern is used if one free face is present. An echelon initiation pattern is used when two or three free faces are present. The blast is initiated from the free face.

3 FIELD MEASUREMENTS AND RESULTS

Twelve blasts were monitored during the summer of 1992. Complete data for surface and internal rock movement were obtained from six blasts, which were used to analyze rock movement and its impact on ore grade control.

Table 5-1 summarizes the six blasts. An echelon V initiation pattern was used in blasts #4, #7, #9 and #17, and blasts #19 and #20 were initiated using a half wing
echelon V pattern. The row-to-row delay time was 65 milliseconds for blasts #4, #17, #19 and #20, 17 milliseconds for blast #7, and 35 milliseconds for blast #9.

Table 5-1: Summary for the Six Blasts Monitored at the Coeur Rochester

<table>
<thead>
<tr>
<th>Blast ID</th>
<th>6750</th>
<th>6750</th>
<th>6750</th>
<th>6775</th>
<th>6775</th>
<th>6775</th>
</tr>
</thead>
<tbody>
<tr>
<td>#4</td>
<td>565</td>
<td>650</td>
<td>541</td>
<td>470</td>
<td>585</td>
<td>800</td>
</tr>
<tr>
<td>#7</td>
<td>15x15</td>
<td>15x15</td>
<td>15x15</td>
<td>14x14</td>
<td>15x15</td>
<td>15x15</td>
</tr>
<tr>
<td>#9</td>
<td>10.6</td>
<td>10.6</td>
<td>10.6</td>
<td>9.9</td>
<td>10.6</td>
<td>10.6</td>
</tr>
<tr>
<td>#17</td>
<td>164.9</td>
<td>198.4</td>
<td>151.4</td>
<td>134.2</td>
<td>185.4</td>
<td>236.3</td>
</tr>
<tr>
<td>#19</td>
<td>249.2</td>
<td>286.7</td>
<td>238.6</td>
<td>180.5</td>
<td>258.0</td>
<td>352.8</td>
</tr>
<tr>
<td>#20</td>
<td>35</td>
<td>65</td>
<td>65</td>
<td>65</td>
<td>65</td>
<td>65</td>
</tr>
</tbody>
</table>

Figures C-1 to C-6 in Appendix C are the horizontal movement maps for these six blasts. The lines which are parallel to each other indicate detonating cord. Figures C-1 through C-6 illustrate that there are either one or two groups of parallel lines, which indicate two initiation patterns, 1) half wing of echelon V, and, 2) echelon V. An arrow shows the amount and direction of surface rock movement. A circled arrow represents the amount and direction of interior rock movement. A number beside a circled arrow indicates the down hole movement bag number. Bag
numbers 1 through 5, see Figure 3-2B in Chapter 3, represent positions from the top to the bottom of a bench. Figures C-7 to C-18 in Appendix C are cross-sectional movement maps, showing internal rock movement for each blast. The cross-sectional directions are perpendicular to the initiation line.

1 Rock Movement Direction

1) Internal

Examination of Figures C-1 to C-6 yields the following observations on rock movement direction:

a. Interior rock consistently moves in a direction almost perpendicular to the initiation line (or detonating cord), see Figures C-1, C-2, C-3, C-5, C-6, or towards the initiation point/center for an initiation echelon V (Figures C-1, C-2, and C-3). These are illustrated in figures 5-1 and 5-2.
Figure 5-1: Rock Movement Perpendicular to Initiation Line

Figure 5-2: Rock Movement Towards the Initiation Point or Center of an Echelon V Pattern
b. Rock moves along the center line of an echelon V if it is close to the echelon V center area, see Figure C-4. This is illustrated in Figure 5-3.

![Figure 5-3: Rock Movement Along the Center Line for an Echelon V](image)

2) Surface

(a) Figure C-1 indicates that surface rock appears to move in a random direction in the case of an echelon V initiation pattern.
(b) For a half wing echelon V pattern, the surface rock tends to move towards the initiation point or the free face, as is shown in Figure C-6.

3) Delay Time Influence

In order to compare the influence of delay time on rock movement, 17 and 35 ms delays were used for blasts #7 and #9 respectively, see Figures C-2 and C-3.

a. Blast #7 (Figure C-2) illustrates that the bags from additional holes #4 to #9 move in multi-directions.

This might result from the short surface delay time (17 ms). The rock in these areas is 5 rows away (or 75 feet) from the free face, and the preceding initiation row does not have enough time to move the material far enough to create adequate room for the following rows. In this case the rock moves in the direction of least restriction. 180 surface markers were used in this blast. After blasting a few of them were found, but none could be used to calculate displacement.
b. There is not enough data on interior movement to compare the rock movement of blast #9 with the others. Therefore, no conclusion can be made concerning the effect of using 35 ms surface delays on rock movement direction.

4) Influence of a Buffer and Free Face

a. In blast #7, movement bags from additional holes #2 and #3 still move in a direction which is almost perpendicular to the initiation line (detonating cord). These bags were 2 or 3 rows (30-45 feet) from the free face. This result appears to indicate that initiation direction has much more influence than the free face, see figure 5-4.
b. Two groups of internal movement bags were used in blast #4 in order to compare the impact of a buffer on rock movement, see figure C-1. One group was placed close to a free face, i.e. additional holes #1, #2 and #3, and the other was close to a buffer side, i.e. additional holes #8, #9 and #10. Both groups were about 70-100 feet away from a free face or blasted rock buffer. The results show that the buffer does not have any influence on rock movement direction when the rock is 75 feet away from the buffer. The rock close to the buffer side appears to move further than that on the free face side. This might result from the high powder factor in the local area of additional holes #8 to #10 for blast #4.
Amount of Rock Movement

1) Internal

The internal movement bags were placed at different heights in a bench, and numbered 1 to 5 from the top to the bottom in each additional hole. Bags #1 to #5 represent layers 1 to 5 respectively in a bench, see Figure 3-2B. The average movement for bags with the same number in each blast can be used to indicate the movement of the corresponding layer of the bench.

The bag recovery was 43 percent for the first bag, 37 percent for the second and third bags, 20 percent for the fourth bag and 9 percent for the bottom bag. Therefore, there is not enough data to estimate the rock movement by using the method discussed in Chapter 4. The average movement for the bags with the same number is calculated to be the arithmetic mean of each individual bag movement. The overall arithmetic mean for each bag number is used to represent the average movement for the blast. This is shown in Table 5-2.
<table>
<thead>
<tr>
<th>Blast</th>
<th>Bag</th>
<th>Bag</th>
<th>Bag</th>
<th>Bag</th>
<th>Bag</th>
<th>Average Interior Rock Movement of Each Blast (ft)</th>
</tr>
</thead>
<tbody>
<tr>
<td>CM</td>
<td>#1</td>
<td>#2</td>
<td>#3</td>
<td>#4</td>
<td>#5</td>
<td></td>
</tr>
<tr>
<td>6750-4</td>
<td>12.1</td>
<td>16.5</td>
<td>29.2</td>
<td>28.6</td>
<td>/</td>
<td>(12.1 + 16.5 + 29.2 + 28.6) / 4 = 21.6</td>
</tr>
<tr>
<td>6750-7</td>
<td>13.0</td>
<td>12.5</td>
<td>41.8</td>
<td>/</td>
<td>32.6</td>
<td>(13.0 + 12.5 + 41.8 + 32.6) / 4 = 25.0</td>
</tr>
<tr>
<td>6750-9</td>
<td>8.7</td>
<td>14.3</td>
<td>39.0</td>
<td>/</td>
<td>/</td>
<td>(8.7 + 14.3 + 39.0) / 3 = 17.7</td>
</tr>
<tr>
<td>6775-17</td>
<td>/</td>
<td>29.0</td>
<td>314</td>
<td>24.4</td>
<td>/</td>
<td>(29.0 + 314 + 24.4) / 3 = 25.7</td>
</tr>
<tr>
<td>6775-19</td>
<td>/</td>
<td>/</td>
<td>229</td>
<td>284</td>
<td>/</td>
<td>(22.9 + 284) / 2 = 25.7</td>
</tr>
<tr>
<td>6775-20</td>
<td>10.8</td>
<td>14.0</td>
<td>309</td>
<td>398</td>
<td>469</td>
<td>(10.8 + 14.0 + 309 + 398 + 469) / 5 = 28.5</td>
</tr>
</tbody>
</table>

Table 5-2: The Average Movement of the Bags

The following observations are made on the results shown in Table 5-2.

a. Generally, the average movement of bags #1 and #2 was less than 15 feet except for blast #17, bag #3 was in the range of 18.2 to 30.9 feet, bags #4 and #5 move from 24.2 to 46.9 feet.

b. It appeared that material at the bottom of a bench tended to move further than that near the top of a bench, which is different from the Rain mine results, and could be a result of the higher powder factor.

c. The average internal movement for the six blasts varied from 17.7 to 28.5 feet.
d. 17 and 35 ms surface delays were used for blasts #7 and #9 respectively, and 65 ms for the other blasts. Concerning the influence of delay time on the amount of rock movement, blasts #7 and #9 did not have enough data to compare with the other blasts.

2) Surface

The average surface movement was calculated according to the measurements of surface movement markers, see Table 5-3. 17 ms surface delays were used in blast #7. The muck pile surface was so rough that no surface marker could be used to measure the surface movement. Surface markers were not used in blast #9.

Table 5-3: The Average Surface Movement of Six Blasts

<table>
<thead>
<tr>
<th>Blast ID</th>
<th>#4</th>
<th>#7</th>
<th>#9</th>
<th>#17</th>
<th>#19</th>
<th>#20</th>
</tr>
</thead>
<tbody>
<tr>
<td>Surface Movement (ft)</td>
<td>8.9</td>
<td>N/A</td>
<td>20.4*</td>
<td>13.5</td>
<td>N/A</td>
<td>10.2</td>
</tr>
<tr>
<td>Interior Movement (ft)</td>
<td>21.6</td>
<td>25.0</td>
<td>17.7</td>
<td>28.3</td>
<td>25.7</td>
<td>28.5</td>
</tr>
</tbody>
</table>

These measurements, based on surface movement bags, are not reliable.

a. Surface stakes were inserted into the collar of the blast holes in blasts #4, #17.
and #20 as surface movement markers. The average surface rock movement was 8.9, 13.5 and 10.2 feet respectively, which is less than the internal rock movement.

b. In blast #9, 20 sampling bags, filled with drill cuttings, were placed on the bench surface in the center of four regular blast holes. They were numbered and identified with colored ribbon, and used to measure surface movement.

Figure C-3 indicates that the surface movement bag #8 was found on the nearby unblasted bench surface, which was not a post-blast muckpile surface position. The surface movement measurements using surface bags were unreliable for this blast. The surface movement bags may not move with the bench surface, but can be thrown from the surface during blasting, therefore, the post-blast position may not correspond to the pre-blast position. This indicates that surface bags are not suitable for surface markers. The surface movement data for blast #9 is considered to be invalid.

3) Comparison of Surface and Internal Movement

The amount of surface movement is far less than the internal rock movement.
see Table 5-3. The average interior rock movement is over two times the surface rock movement.

4 DILUTION CAUSED BY ROCK MOVEMENT

1. Assumptions

In order to estimate blast-induced dilution, pre-blast blocks have to be adjusted accurately according to the rock movement characteristics. The dilution can be calculated by comparing pre- and post-blast polygons. Based on the observations presented above, the following assumptions are made:

(1) Rock moves in a direction which is perpendicular to the initiation line (or detonating cord) in the area which is two rows (30 feet) away from the echelon-V initiation center line.

(2) According to blast #4, see Figure C-1, it is assumed that rock moves by converging to the center line of an echelon V, if it is within two rows from the center line. This is similar to that experienced at the Rain mine.
(3) **Rock moves towards the free face for the area within three rows from the free face.**

(4) A **pre-existing blasted rock buffer does not change rock movement direction**, but may **reduce the amount of rock movement.**

(5) The **amount of rock movement measured in specific locations is used to estimate the average rock movement throughout the entire blast area.** This assumption means that the amount of rock movement is the same for the whole blast area except at the **boundaries of the blast.**

(6) **Dilution can be calculated using equation 4-8.**

The validity of these assumptions is discussed in Chapter 4.

The pre-blast blocks for the six blasts were adjusted accordingly based on the above assumptions. This is shown in figures C-17 to C-22 in Appendix C.
2. **Blast-induced Dilution**

As discussed in Chapter 4, the theoretical dilution can be calculated according to the equation:

\[ \eta = \frac{W_i}{W} \]  

where:

- \( \eta \) — Weight dilution (or dilution)
- \( W_i \) — Weight of waste moved to digging block
- \( W \) — Weight of digging block

The above equation can also be expressed as:

\[ i_i = \frac{S_i}{S} \]  

where:

- \( S_i \) — Area of waste moved to polygon
- \( S \) — Polygon area

Table 5-4 lists dilution for each individual block in the six blasts. The term dilution refers to weight dilution.
Table 5-4: Theoretical Maximum Blast-Induced Dilution for Each Block

<table>
<thead>
<tr>
<th>Block No.</th>
<th>Blast ID</th>
<th>Average Movement ft</th>
<th>Block Area ft²</th>
<th>Diluted Area ft²</th>
<th>Dilution</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>#4 @6750</td>
<td>21.6</td>
<td>123400</td>
<td>3920</td>
<td>3.2%</td>
</tr>
<tr>
<td>2</td>
<td>#9 @6750</td>
<td>17.7</td>
<td>64740</td>
<td>6261</td>
<td>9.7%</td>
</tr>
<tr>
<td>3</td>
<td>#17 @6775</td>
<td>28.3</td>
<td>51600</td>
<td>8110</td>
<td>15.7%</td>
</tr>
<tr>
<td>4</td>
<td>#20 @6775</td>
<td>28.5</td>
<td>49500</td>
<td>6930</td>
<td>14%</td>
</tr>
<tr>
<td>5</td>
<td>#20 @6775</td>
<td>28.5</td>
<td>23260</td>
<td>7723</td>
<td>33.2%</td>
</tr>
<tr>
<td>6</td>
<td>#19 @6775</td>
<td>25.7</td>
<td>22900</td>
<td>5004</td>
<td>21.9%</td>
</tr>
<tr>
<td>7</td>
<td>#19 @6775</td>
<td>25.7</td>
<td>19910</td>
<td>4942</td>
<td>24.8%</td>
</tr>
<tr>
<td>8</td>
<td>#20 @6775</td>
<td>28.5</td>
<td>3740</td>
<td>2090</td>
<td>55.9%</td>
</tr>
<tr>
<td>9</td>
<td>#20 @6775'</td>
<td>28.5</td>
<td>3410</td>
<td>2478</td>
<td>72.7%</td>
</tr>
<tr>
<td>10</td>
<td>#7 @6750</td>
<td>25</td>
<td>2820</td>
<td>1961</td>
<td>69.5%</td>
</tr>
<tr>
<td>11</td>
<td>#19 @6775</td>
<td>25.7</td>
<td>2800</td>
<td>1400</td>
<td>50%</td>
</tr>
<tr>
<td>12</td>
<td>#7 @6750</td>
<td>25</td>
<td>1826</td>
<td>1136</td>
<td>61.9%</td>
</tr>
<tr>
<td>13</td>
<td>#7 @6750</td>
<td>25</td>
<td>1324</td>
<td>1004</td>
<td>75.8%</td>
</tr>
<tr>
<td>14</td>
<td>#7 @6750</td>
<td>25</td>
<td>1132</td>
<td>972</td>
<td>85.9%</td>
</tr>
</tbody>
</table>

The dilution values given in Table 5-4 are based on the assumption that the rock surrounding the block is assumed to be waste, which is not the case at the Coeur Rochester Mine. In practice, the pre-blast polygons at the mine were determined according to an economic cut-off grade. The surrounding rock was low grade material, and the actual dilution was much smaller than the values in Table 5-4. Indeed, these values can be interpreted as the maximum dilution for the blocks.
Table 5-4 yields the following observations:

1) The larger the mining block, the smaller the dilution, and vice versa. For example, the area of block #1 is 123,000 ft², the dilution caused by blasting is 3.2%. The area of block #14 is 1,132 ft², the dilution is 85.9%.

2) It appears that less than 10% dilution occurs if the mining block area is in the order of $10^6$ ft², 10-40% dilution for a block area of $10^5$ ft², and over 40% dilution for a $10^4$ ft² block area.

5 CONCLUSIONS FROM COEUR ROCHESTER MEASUREMENTS

1) Interior rock consistently moves in a direction which is almost perpendicular to the initiation line (detonating cord), or along the center line of an echelon V if it is close to the center. The buffer does not appear to have a significant influence on the rock movement direction.

2) Surface rock appears to move in a random direction in the case of an echelon V initiation pattern, and tends to move towards the initiation point for a half wing echelon V pattern or towards the free face. Surface movement bags are not suitable for surface movement markers.
3) A 17 ms delay time might be too short for a blast and results in rock movement in multi-directions. This produces a more complicated grade control problem.

4) The free face and buffer do not have much influence on movement direction if the rock is 30-45 feet away.

5) The average internal movement for these six blasts is in the range of 17.7 to 28.5 feet, and the average surface movement 8.9 to 13.5 feet. Interior rock moves further than that on the surface.

6) No information is available on the influence of the free face or buffer on the amount of rock movement.

7) The larger the mining block, the smaller the dilution, and vice versa. The theoretical upper limit of the dilution varies from three to eighty five percent.

8) It appears that less than 10% dilution occurs if the mined ore block area is in the order of $10^6$ ft$^2$, 10-40% dilution for $10^5$ ft$^2$, and over 40% dilution for $10^4$ ft$^2$. 
Reducing dilution and increasing recovery should be a major concern for a operating mine. It has been shown in earlier chapters that excessive blast movement may produce from three to seventy percent dilution. Methods to calculate average rock movement and dilution have been discussed. This chapter presents methods to reduce or compensate for blast induced dilution and improve grade control.

The blasting practice survey indicates that most open pit gold mines in Nevada do not carry out any field measurements to determine direction and amount of rock displacement caused by a blast. Nearly half the mines reported that the rock strength ranged from soft to medium, and ease of blast ranged from very easy to medium. The powder factor used at these mines was generally less than 0.4 lbs ANFO/ton. The data from the Rain mine showed that the average rock movement varied from 4.5 to 6 feet with the powder factor 0.3-0.4 lbs ANFO/ton, which resulted in an estimated dilution of 3 to 10 percent. This amount of dilution might be quite acceptable for an operating mine.

Chapters 4 and 5 presented the data and results obtained from the Coeur and Rain mines, and demonstrated that the digging polygons can be relocated according
to the direction and amount of rock movement. However, it is probably not necessary to carry out field measurements of blast induced rock movement at every operating mine. For example, blast induced dilution might be quite low at a mine with relatively soft rock using a small powder factor. Therefore, if no field measurements are obtained for rock movement due to blasting, the following methods can be considered for a mine which uses a small powder factor.

1) The rock movement is related to the amount of explosive used per unit weight or volume. The more explosive used per unit weight or volume, the larger the rock movement. Therefore, it is recommended to use the smallest possible powder factor while still maintaining acceptable fragmentation.

2) It was shown in Chapters 4 and 5, that the rock movement direction is dominated by the initiation sequence and direction. The rock moves in a direction which is almost perpendicular to the detonating cord or initiation line. The rock will move in multi-directions if different initiation directions are used, which results in more dilution. Therefore, using a single initiation direction in blasting will produce uniform movement generally in one direction. This will improve ore grade control practice.
3) For mines with vein mineralization or relatively small mining polygons (which depend on loading equipment), the blast induced rock movement can produce considerable dilution. The blast direction should be in the strike direction of the vein or along the longer dimension of the ore block, both of which can significantly reduce blast induced dilution.

The methods recommended above are considered suitable for mines using a relatively small powder factor, and which experience only a few feet of horizontal movement, for example the Rain mine uses powder factor 0.3-0.4 lbs ANFO/ton, which generates approximately 5 feet horizontal movement.

If a higher powder factor is used by mines with small mining blocks or vein mineralization, the horizontal rock movement and corresponding dilution will be a major concern. For example, at the Coeur Rochester mine, the powder factor varies from 0.7-0.9 lbs ANFO/ton which results in 25-30 feet of rock movement. Therefore, field measurements of blast induced rock movement are recommended in order to confirm whether or not there is excessive rock movement. Minimum dilution can be achieved by adjusting a digging polygon according to the amount and direction of rock movement. The following methods are recommended for these mines:
1) Carry out a series of systematic measurements of rock internal and surface movement to confirm the direction and amount of rock displacement.

2) Develop an empirical rock movement model based on the measurements, which considers the powder factor, blast hole layout, rock type, initiation pattern and rock internal movement characteristics.

3) Develop a method to adjust a pre-blast digging polygon in order to reduce dilution. The digging direction, rock movement and mineralization models for a specific ore block must be taken into account when adjusting a digging polygon.

The limited field measurements at both mines indicate that dilution caused by blasting impacts on the economics of the mine in two ways. Firstly, potential waste or low-grade ore is treated as ore, which results in unnecessary processing costs in the mill. Secondly, the ore moved beyond a digging line due to blasting is mined as waste, which leads to a reduction in total gold produced over the life of the mine. Therefore, it is very important to monitor more blasts in order to investigate rock movement caused by blasting, confirm the obtained data, and find a relationship between rock movement and rock types, rock strength, blast layout, powder factor and
explosive energy distribution and initiation pattern etc.

Interior rock movement markers can be used to characterize the rock movement over a bench height. Since the requirement for drilling additional holes impacts on mine production, it is necessary to find an alternative method to determine internal rock movement, which minimize any interruption with production and can efficiently locate internal movement markers.

Many blast fragmentation and rock movement models have been developed, Cherry, 1963; Langefors and Kihlstrom, 1979; Favreau, 1980; McHugh, 1983; Cunningham, C. 1983; Schamaun, 1983; Hyde and Favreau, 1986; Harries, 1987; ICI Explosives, 1988; Yang and Kavetsky, 1989 and 1990. Most of these models emphasize rock fragmentation, and do not consider rock movement and its impact on ore grade control. Several models consider muckpile formation and rock movement, ICI Explosives, 1988; Yang and Kavetsky, 1989 and 1990. A recent test using SABREX and input data from the Coeur Rochester Mine, significantly underestimated the movement measured in the field. In addition, the model assumes blasting to a free face, rather than to a rock buffer, which is the typical practice in gold mining operations. The model needs to be improved or calibrated according to the results from field measurements. Therefore, it is
necessary to either develop or test existing empirical models of blasted rock movement with respect to rock type/rock strength, blast layout, powder factor, etc., based on the information obtained from the measurements. Following this, a systematic procedure can be developed to predict blasted rock movement, and to project the shape and location of a post-blast 3D digging polygon using the combined model.

A theoretical prediction model can also be developed by using a discrete element analysis or the Yang and Kavetsky model to predict rock movement due to blasting, and can be calibrated with data obtained from field measurements.

The empirical and theoretical models can be used at selected mines in order to determine which one appears to more accurately and consistently predict rock movement from a blast, and then a procedure to relocate post-blast digging polygons for a typical blast at a mine can be developed in order to reduce blast induced dilution.
REFERENCES


BIBLIOGRAPHY


APPENDIX A  LIST OF COMPANIES WHO PARTICIPATED IN THE BLAST SURVEY.

BLASTING PRACTICES QUESTIONNAIRE

Robinson Mine, Magma Nevada Mining Co., P.O. Box 382, Ruth, NV 89319

Wind Mountain Mines, Inc., P.O. Box 160, Empire, NV 89405

Pinson Mining Company, P.O. Box 192, Winnemucca, NV 89445

Gold Bar Mine, P.O. Box 1030, Beatty, NV 89003

Battle Mountain Gold Co., Box 1627, Battle Mountain, NV 89820

Bond Gold Bullfrog, Box 519, Beatty, NV 89003

Independence Mining Co. Inc., Mountain City Star Route, Elko, NV 89801

Coeur Rochester Mine, P.O. Box 1057, Lovelock, NV 89419

Cominco American Resources Inc., P.O. Box 847, Carlin, NV 89822

Dee Gold Mining Co., P.O. Box 1189, Elko, NV 89801

McCoy Gold Mine, Echo Bay Co., Box 1658, Battle Mountain, NV 89820

Round Mountain Gold Co., P.O. Box 480, Round Mountain, NV 89045

Paradise Peak Mine, FMC Gold Co., P.O. Box 145, Gabbs, NV 89409

Getchell Mine, P.O. Box 220, Golconda, NV 89414

Chimney Creek Mine, P.O. Box 69, Golconda, NV 89414

Hycroft Resources & Development Inc., Box 3030, Winnemucca, NV 89446
USMX, Inc., P.O. Box 809, Ely, NV 89301

Aurora Mine, Nevada Goldfield Inc., P.O. Box 3070, Hawthorne, NV 89415

Western States Minerals, Box K, Lovelock, NV 89419

Rabbit Creek Mine, P.O. Box 552, Winnemucca, NV 89445

Cortez Gold Mines, Cortez, NV 89821

Bald Mountain Mine, Placer Dome Inc., Box 2706, Elko, NV 89801

Marigold Mining Co., P.O. Box 9, Valmy, NV 89438

Western Hog Ranch Mine, P.O. Box 9, Gerlach, NV 89412

Newmont Gold Co., P.O. Box 669, Carlin, NV 89042

Barrick Goldstrike Mines, P.O. Box 29, Elko, TW 89801

Lone Tree Mine, P.O. Box 388, Valmy, NV 89438
QUESTIONNAIRE ON DRILLING/BLASTING PRACTICES AND GRADE CONTROL IN NEVADA GOLD MINES

Name of Person Completing Questionnaire: ________________________

Position in Company: ________________________

Name of Company: ________________________

Address of Company: ________________________

Telephone Number: ________________________

Date of completion: ________________________

Return to: Professor Pierre Mousset-Jones
Department of Mining Engineering
Mackay School of Mines
University of Nevada, Reno
Reno, NV 89557-0139
Phone: 784-6959
Fax: 784-1766

General Information

Yearly production:
Ore: ___________ tons
Waste: ___________ tons

Daily production:
Ore: ___________ tons
Waste: ___________ tons

Number of shifts/day: Drilling
Ore: ___________
Waste: ___________

Blasting
Ore: ___________
Waste: ___________
**Geological Information related to blasting**

**Classification of deposit:**

1: Skarn hosted.  
2: Sinter hosted.  
3: Volcanic hosted.  
4: Intrusive hosted.  
5: Sediment hosted.  
6: Other.

What is type of gold occurrence?

1: Vein.  
2: Disseminated .  
3: Stock work.  

Is free gold present?  
Yes, ______  
No. ______

If Yes, Maximum size of free gold ______

What is/are the dominant rock type(s) encountered in the mine?

<table>
<thead>
<tr>
<th>Type(s)</th>
<th>Approximate Percentage</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore: 1:</td>
<td></td>
</tr>
<tr>
<td>2:</td>
<td></td>
</tr>
<tr>
<td>3:</td>
<td></td>
</tr>
<tr>
<td>4:</td>
<td></td>
</tr>
<tr>
<td>Waste: 1:</td>
<td></td>
</tr>
<tr>
<td>2:</td>
<td></td>
</tr>
<tr>
<td>3:</td>
<td></td>
</tr>
<tr>
<td>4:</td>
<td></td>
</tr>
</tbody>
</table>

Degree of weathering of rock to be blasted:

<table>
<thead>
<tr>
<th>Estimated Percentage</th>
<th>Weathered Condition</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore: 1:</td>
<td>Heavy.</td>
</tr>
<tr>
<td>2:</td>
<td>Medium.</td>
</tr>
<tr>
<td>3:</td>
<td>Low.</td>
</tr>
<tr>
<td>4:</td>
<td>Not weathered.</td>
</tr>
<tr>
<td>Waste: 1:</td>
<td>Heavy.</td>
</tr>
<tr>
<td>2:</td>
<td>Medium.</td>
</tr>
<tr>
<td>3:</td>
<td>Low.</td>
</tr>
<tr>
<td>4:</td>
<td>Not weathered.</td>
</tr>
</tbody>
</table>
Rock alteration is:

Ore: 1: Argillised.  
2: Silicified.  
3: Other.  

Waste: 1: Argillised.  
2: Silicified.  
3: Other.  

Rock mass jointing:

<table>
<thead>
<tr>
<th>Frequency</th>
<th>Jointing</th>
<th>Bedding</th>
<th>Low Angle Faults</th>
<th>High Angle Faults</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore:</td>
<td>1: high.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>2: Medium.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>3: Low.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Waste:</td>
<td>1: High</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>2: medium.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>3: Low.</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Orientation of jointing relative to beach fragmentation?

Ore: 1: Assists.  
2: Resists.  
3: Little affect.  

Waste: 1: Assists.  
2: Resists.  
3: Little affect.  

The overall blasting difficulty is considered:

Ore: 1: Easy.  
2: Medium.  
3: Difficult.  

Waste: 1: Easy.  
2: Medium.  
3: Difficult.


**Equipment Information**

Is drilling and blasting done by a contractor? Yes. No.

How much interaction takes place between the mine and the:

- Drill & Blast Contractor
  - (If Applicable)
  - Explosive Supplier

<table>
<thead>
<tr>
<th>High</th>
<th>Medium</th>
<th>Low</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

What type(s) of drill equipment is/are used at your mine?

<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Model No.</th>
<th>Number</th>
<th>Diameter</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Ore:
1: __________
2: __________
3: __________
4: __________

Waste:
1: __________
2: __________
3: __________
4: __________

Do you use an explosive truck to charge holes? Yes, No.

If Yes, please give information of explosive truck:

<table>
<thead>
<tr>
<th>Explosive Type For Truck</th>
<th>Load Capacity (lbs)</th>
<th>Numbers</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

1: __________
2: __________
3: __________

The average feet drilled per drill per shift:

- Ore: ________ft
- Waste: ________ft

Average number of holes drilled per drill per shift:

- Ore: __________
- Waste: __________

Do you monitor the drills? Yes, No.

(If Yes, please answer the questions 1-4)
1: Type of monitoring equipment.

Make: ___________________ Model No: ___________________

2: Which of following is monitored?

- Bit location
- Hole deviation
- Penetration rate
- RPM
- Thrust
- Torque

<table>
<thead>
<tr>
<th></th>
<th>Yes</th>
<th>No</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bit location</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hole deviation</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Penetration rate</td>
<td></td>
<td></td>
</tr>
<tr>
<td>RPM</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Thrust</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Torque</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

3: Data stored for each hole on drill and collected at end of shift:

- Yes, ______
- No, ______

4: Data transmitted continuously to computer? Yes, ______ No, ______

If Yes, what system is used?

Make: ___________________ Model No: ___________________

Do you use any computer software to design blasting?

- Yes, ______
- No, ______

If answer Yes, please answer the following questions:

1: Name of software: ______________________________

2: Name and address of company: ____________________________

3: Your blasting layout depends on computer software

- A: Completely. ______
- B: Partially. ______
- C: other. ____________________________

In your opinion, blasting layout

1: should be heavily based on computer design.
2: should use computer design as a guide.
3: should be completely based on the experiences of the blasting engineer/supervisor.
4: other ______________________________
Blasting Layout

The typical bench height is/are

Ore: ______ ft.
Waste: ______ ft.

The typical dimensions of a blast are

Ore: 1: Length ______ ft.
2: Width ______ ft.
3: Number of blast holes ______.
4: Total tons of rock to be blasted ________.
5: Total explosive used per blast ________ Lbs.
6: Maximum charge per delay ________ lbs.
7: Explosive charge per hole:
   (1): ________ lbs for diameter _____ in.
   (2): ________ lbs for diameter _____ in.
8: Swell factor used ________.

Waste: 1: Length ______ ft.
2: Width ______ ft.
3: Number of blast holes ______.
4: Total tons of rock to be blasted ________.
5: Total explosive used per blast ________ Lbs.
6: Maximum charge per delay ________ lbs.
7: Explosive charge per hole:
   (1): ________ lbs for diameter _____ in.
   (2): ________ lbs for diameter _____ in.
8: Swell factor used ________.

Comment on these parameters relative to rock fragmentation and grade control:


Blast hole pattern is

Ore: 1: Square. _____    ie: Staggered
2: Rectangular. _____
3: Staggered. _____
4: Other. _____

Waste: 1: Square. _____
2: Rectangular. _____
3: Staggered. _____
4: Other. _____
Blast design layout parameters:

Ore: 1: Burden
or horizontal distance between rows is _______ ft.
2: Hole spacing
or horizontal distance between holes is _______ ft.
3: Vertical subdrilling is _______ ft.
4: Hole diameter is _______ in.
5: Vertical or inclined, angle is _______ degree.

Waste: 1: Burden
or horizontal distance between rows is _______ ft.
2: Hole spacing
or horizontal distance between holes is _______ ft.
3: Vertical subdrilling is _______ ft.
4: Hole diameter is _______ in.
5: Vertical or inclined, angle is _______ degree.

The distance from toe of front row holes to the free face is

Ore: _______ ft.
Waste: _______ ft.

Do you use buffer blasting to minimize rock displacement?

YES, ______ NO. ______

If Yes, what are the typical dimension of the buffer rock pile?

Ore: Width _______ ft.
     Height _______ ft.

Waste: Width _______ ft.
     Height _______ ft.

The hole conditions are

Ore: 1: ______ % of holes with at least 2 ft standing water.
     2: ______ % of holes with less than 2 ft standing water.
     3: ______ % of dry holes.

Waste: 1: ______ % of holes with at least 2 ft standing water.
     2: ______ % of holes with less than 2 ft standing water.
     3: ______ % of dry holes.

Do you pump the wet blast holes? Yes, ______ No. ______

If Yes, typical percentage of holes which need pumping: ______ percent.

What type(s) of explosive is/are used in pumped holes?
<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Type</th>
<th>VOD (ft/sec)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3:</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

If No, What type(s) of explosive is/are used in wet holes?

<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Type</th>
<th>VOD (ft/sec)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3:</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

What type(s) of explosive are used in dry holes?

<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Type</th>
<th>VOD (ft/sec)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3:</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Do you measure charge emplaced in each hole? **Yes,** ___ **No.** ___

If **Yes,** how? By weight ___ or by column length ___

Please indicate charging configuration:

Do you use a stemming machine? **Yes,** _____ **No.** _____

What materials do you use for stemming?

- Ore. ________________________
- Waste. ________________________

The stemming height is

- ore. ______________ ft
- waste. ______________ ft

Do you use the same booster for all the holes in a blast? **Yes.** _____ **No.** _____

*If No, please explain why you use different boosters:*

____________________________________________________________

**What kind(s) caps is/are used for:**

<table>
<thead>
<tr>
<th>Manufacturer</th>
<th>Type</th>
<th>Total Period Numbers</th>
</tr>
</thead>
<tbody>
<tr>
<td>1: Down hole delay</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2: Surface delay</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3: Initiation</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Is the precision of the caps tested at the mine?

Yes,    No.

If Yes, please describe how you measure the precision:

Please use diagrams provided to indicate: 1: hole pattern, 2: down hole delay time, 3: surface delay time, 4: Igniting sequence. An example is shown for illustration.

Example: Typical Blast Design And Firing System

300 millisecond Down Hole Delay in All Blast Holes

X Represents Surface Delay Caps

Indicates Ignition/trigger

Line Between Holes And Rows Illustrates Igniting Surfaces
Example: Typical Blast Design And Firing System

300 millisecond Down Hole Delay in All Blast Holes

X Represents Surface Delay Caps

--- Indicates igniting Detonator

Line Between Holes And Rows Illustrates Detonating Cord
Use This Diagram To Describe The Echelon And/OR Staggered Hole Pattern Used At Your Mine

- n
- millisecond Overelay
- Igniting Detonator
- Detonating Cord
Use This Diagram to Describe The Square And/Or Rectangular Hole Pattern Used At Your Mine

- Millisecond Down Hole Delay
- Millisecond Surface Delay
- Igniting Detonator
- Detonating Cord
Please use diagram below to indicate hole charging configuration:

Example: Typical Charging Configuration Used For A Normal Blast

Please fill in the diagram to illustrate charging configuration used for a normal blast at your mine.
Is/are pre-splitting/cushion blasting/light charging used near the final pit slope?

Yes, ___  No. ___

If Yes, please answer the following questions:

1: Hole diameter is _____ in.
2: Hole spacing is _____ ft.
3: Inclined angle is _____ degree.
4: Weight of explosive per hole is _____ lbs.
5: Results of this/these method on stability of final slope is
   (1): Good. ___
   (2): Fair. ___
   (3): Poor. ___

6: Please use diagram provided to illustrate hole charging configuration for wall control:
Blast Monitoring And Grade Control

What method do you use to sample the blast hole cuttings?
1: Piepan. _____ Dimensions. ____________
2: Through the deck sampler ___________. Dimensions. ____________
3: Core. _____ Dimensions. ____________
4: Other. ____________________________ Dimensions. ____________

How many individual samples do you take at each drill hole? ___

Does the sample include sub-drill material? Yes, ____ No. ____

Who does the sampling? Driller._____
Other: _________________________________

Where is the sample reduced? Field. _____
Lab. _____

If field, please explain how?

_________________________________________

Have you carried out any statistical studies on potential for error and bias in your blast hole sampling procedure?
Yes, ____ No. ____

If Yes, please briefly describe:

_________________________________________

_________________________________________

Do you monitor vibration caused by blasting Yes, ____ No. ____

If Yes, please indicate instrument you used
Make: _______________________________
Type: _______________________________

How much rock movement and mixing from blasting is tolerated for grade control? Please describe:

_________________________________________

_________________________________________

_________________________________________
Typically how often does a blast from the grade control perspective cause excessive displacement of the ore?

1: Seldom. _______
2: Sometimes. _______
3: Frequently. _______

Is rock displacement due to the blasting measured?

Yes, _______ No. _______

If Yes, How and with what instrument?

______________________________
______________________________
______________________________

Do you layout the grade control flagging based on the position of the rock prior to blast?

Yes, _______ No. _______

If No, please explain method used:

______________________________
______________________________
______________________________

How many different kinds of flagging do you use for grade control?

______________________________

Is visual separation of ore/waste?

1: Impossible. _______
2: Difficult. _______
3: Possible. _______

Do you measure blast hole deviation over complete length of hole?

Yes, _______ No. _______

If Yes, How and with what instrument?

______________________________
Have you attempted to measure rock size distribution from a blast?

Yes, _____  No. ______

If Yes, Please describe method used and general results:

___________________________________________________________________________
___________________________________________________________________________
___________________________________________________________________________
___________________________________________________________________________

Average rock fragmentation due to blasting is

Ore:  1: Good. _____  Waste:  1: Good. _____
      2: Fair. _____     2: Good. _____
      3: Poor. _____    3: Poor. _____

How often do you get poor fragmentation?

Ore:  1: Rarely. _____  Waste:  1: Rarely. _____
      2: Sometimes. _____ 2: Sometime. _____
      3: Often. _____     3: Often. _____

What are the causes for poor fragmentation when it occurs?

Ore:  1: Rock type. _____
      2: Rock structure: joints and faults. _____
      3: Blast Design. _____
      4: Problem with delays. _____
      5: Water table. _____
      6: Caved holes. _____
      7: Short holes. _____
      8: Deviated holes. _____
      9: Other. __________________________

Waste:  1: Rock type. _____
       2: Rock structure: joints and faults. _____
       3: Blast design. _____
       4: Problem with delays. _____
       5: Water table. _____
       6: Caved holes. _____
       7: Short holes. _____
       8: Deviated holes. _____
       9: Other. __________________________

What is/are factor(s) affecting oversize dimension?

1: __________________________
2: __________________________
3: __________________________
What is the oversize dimension above which rehandling is needed?

Ore: __________ ft     Waste: __________ ft

What is typical percent of oversize rock in a blast?

Ore: ______ %     Waste: ______ %

What method(s) is/are used to rehandle the oversize rock?

1: Secondary blasting. ______  
2: Mechanical breaking. ______  
3: Other. ____________________________

Rehandling work is

Ore: 1: Seldom. ______  2: Sometimes. ______  3: Frequent. ______
Waste: 1: Seldom. ______  2: Sometimes. ______  3: Frequent. ______

Please describe any problems in the drilling and blasting functions at your mine which need further investigation:

_________________________________________________________________
_________________________________________________________________
_________________________________________________________________
_________________________________________________________________

PREPARED BY:

Mr. Songlin Zhang  
Graduate Student  
Dept. of Mining Engineering  
Mackay School Of Mines  
University of Nevada-Reno

July 1, 1991
**APPENDIX B  INDEX TO FIGURES FOR THE RAIN MINE**

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<tr>
<th>Figure B-I:</th>
<th>Blast Layout and Bench Geology of Blast #1</th>
<th>Page</th>
</tr>
</thead>
<tbody>
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<td>Blast Layout and Bench Geology of Blast #2</td>
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<td>Figure B-3:</td>
<td>Blast Layout and Bench Geology of Blast #3</td>
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<td>Figure B-4:</td>
<td>Blast Layout and Bench Geology of Blast #4</td>
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