WALL CONTROL BLASTING AT THE ORPHAN BOY MINE – BUTTE, MONTANA

Jewel Sakyi

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Abstract

Blasting is often conducted to produce desirable fragment sizes to minimize production cost and energy consumption in downstream processes, including loading, hauling, and crushing. However, the venting of explosive energy in the rock formation sometimes causes unwanted damage beyond the desired perimeter of the blast area. Control blasting has mainly been used in surface mining operations to minimalize blast damage. To explore the applicability of control blasting in underground mining operations, protect the safety of students, faculty, and staff, prevent overbreak, and to ensure the stability of openings and workings, four experimental control blasts, comprising of three smooth blasts and one presplit blast were conducted at the Orphan Boy Mine (an underground mine at Montana Tech). Based on the half-barrels that resulted, the success of the three smooth blasting was less than 20%. The presplit blasting resulted in extensive fracturing of the adjoining rock mass. Fracturing, weathering, and jointing of the rock mass were observed as factors that limited the success of the control blasts. Future work could focus on establishing the effects of these geological conditions on controlled blasting at the Orphan Boy.

Keywords: Control Blasting, Mining, Rock Fragmentation, Presplitting, Blast Damage
Dedication

Dedicated to the memory of my dad, Mr. Rexford Sakyi Addo. You will always be an inspiration.
Acknowledgements

I am most grateful to God for His protection and guidance throughout my studies. My deepest appreciation to the Montana Tech Mining Engineering Department for funding my Master’s Degree.

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<tr>
<td>ABS</td>
<td>Absolute Bulk Strength</td>
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<td>AWS</td>
<td>Absolute Weight Strength</td>
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<tr>
<td>RBS</td>
<td>Relative Bulk Strength</td>
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<tr>
<td>RWS</td>
<td>Relative Weight Strength</td>
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<tr>
<td>g/cm³</td>
<td>gram per cubic centimeter</td>
</tr>
<tr>
<td>lb/ft³</td>
<td>pound per cubic foot</td>
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<tr>
<td>MPa</td>
<td>Megapascals</td>
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<td>g</td>
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<td>ms</td>
<td>Millisecond</td>
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<td>psi</td>
<td>pounds per square inch</td>
</tr>
<tr>
<td>ANFO</td>
<td>Ammonium Nitrate Fuel Oil</td>
</tr>
<tr>
<td>kg/m</td>
<td>kilogram/meter</td>
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## Glossary of Terms

<table>
<thead>
<tr>
<th>Term</th>
<th>Definition</th>
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<tbody>
<tr>
<td>ABS</td>
<td>The measure of explosive energy reported per unit of volume or the heat of reaction available in each cm$^3$ of explosive.</td>
</tr>
<tr>
<td>AWS</td>
<td>The measure of explosive unit per unit weight.</td>
</tr>
<tr>
<td>RBS</td>
<td>Ratio of the explosive ABS to the ABS of ANFO.</td>
</tr>
<tr>
<td>RWS</td>
<td>Ratio of the explosive AWS to the AWS of ANFO.</td>
</tr>
<tr>
<td>ANFO</td>
<td>A blasting product with approximately 94.5% industrial-grade ammonium nitrate and 5.5% No. 2 grade diesel fuel oil for a nearly oxygen-balanced mix.</td>
</tr>
<tr>
<td>Explosive</td>
<td>Chemical compound, mixture or device that can undergo a rapid chemical reaction, producing an explosion; a cap sensitive mixture.</td>
</tr>
<tr>
<td>Half-barrel</td>
<td>Part of perimeter holes that are left intact after blasting.</td>
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1. Introduction

Mining is an important sector for the development of modern civilization. It is the source of many metals and nonmetals that are used in an infinite number of technologies; in medicine, agriculture, security, manufacturing, etc. Thus, mining is a significant contributor to the development of present day high technology that supports the lives of many people. These metals and nonmetals are obtained from the earth through rock excavation techniques, including blasting. Blasting is presently the most commonly applied technique to extract materials of interest from the earth, and it is used in mining (surface and underground), construction, and quarry operations. Blasting involves loading explosives into drill holes and initiating them to fragment the rock into desirable sizes.

Although blasting is carried out to produce fragment size distributions designed to minimize production costs and energy consumption in downstream processes like loading, hauling, and crushing (Kecojevic and Komljenovic, 2006), unwanted damage beyond the desired perimeter of excavation does occur during the venting of the explosive energy in the rock formation. To minimize this damage, controlled blasting such as presplitting and smooth blasting have been employed in both surface and underground operations. Literature, however, suggests that wall control blasting is employed in surface mining more than in underground mining. This is due to the fact that two separate blasts are required for presplitting which is stressful to schedule, and although smooth blasting does not have that problem, it often does not produce effective results as does presplitting (Van Eeckhout, 1987).

This research investigated how wall control blasting could be adopted at the Orphan Boy Mine (red marked in Figure 1), located on the west side of Montana Tech’s campus in Butte, Montana with the goal to protect the safety of students and faculty, prevent overbreak and to
ensure the stability of underground openings or workings. The results of this research will help improve the existing blast practices at the Orphan Boy Mine and introduce an appropriate wall control blasting technique as a means of mitigating rock overbreak, improving wall stability and reducing hazards associated with rock falls. This work discusses the research procedure and results, with conclusions and recommendations for future work.

Figure 1: The Orphan Boy Mine (Google Maps)
1.1. **Research Objectives**

This research aimed to:

- Study and propose the best parameters for which effective wall control blasting can be conducted at the Orphan Boy;
- Limit rock over break in underground blasting; and,
- Reduce and or eliminate rock falls by creating stable walls.
2. Blasting

Blasting is a common practice in most mines, and offers the most economical and flexible means of accessing the orebody in a hard rock mine (Sellers, 2011). In hard rock mining activities, blasting is conducted after drilling in the ground fragmentation process, and involves the detonation of explosives to break rocks into desirable sizes. Explosives (chemical substances) are placed in drill holes, and detonated to fragment the rock or formation into smaller sizes for appropriate end use. An efficient blast produces the desired tonnage and particle size distribution for easy loading, hauling and processing or disposal.

In underground mining, blasting is used to either provide access to an orebody from openings such as main haulage ways, main levels, ramps, crosscuts, sublevels, etc. or to mine out a rock of value. The structural geology of the underground area determines the type of blast design that will be used to create openings or operate the mine, and the use of the opening determines the width, height, cross-sectional shape and length of the blast (Iverson et al., 2013; ISEE, 2011). Underground blasting is generally carried out in a confined space unique to the underground environment compared to the surface environment. The diverse rock characteristics have a variable effect on how the rock reacts when it is blasted and how stable the created opening will be after the blast (ISEE, 2011).

Three types of blasting; namely, production blasting, secondary blasting and controlled blasting are commonly used to fragment rock, and protect the final perimeter in mining operations. Bender (1999) defined production blasting as the type of blasting that is designed to fragment a predetermined volume of rock; widely spaced holes are drilled and fired in a controlled delay sequence. After a production blast, a secondary blast may be required to reblast a portion of the rock to reduce it to a size fit for handling by the machines and equipment.
5

available (Balasubramanian, 2017). Controlled blasting is used to control blast-induced effects such as overbreak, fractures within rock walls and ground vibration.

2.1. Blasting Theory

According to Khoshrou (1996), the sudden release of energy, and the reaction products at high pressure by the rapid chemical reaction in an explosive contained in a drill hole gives rise to compression waves in the explosive and in the surrounding rock material. These waves play a central role in the functioning of the explosive and in the fragmentation of the rock mass. From the point of initiation, the drill hole expands by the crushing of the walls immediately surrounding the explosive due to the development of high pressure, followed by a development of radial cracks. Compressive stress waves emanate or radiate in all directions from the drill hole with a velocity equal to the sonic wave velocity in the rock mass. New cracks are developed while pre-existing ones are enlarged. When these compressive stress waves reflect against a free face in the rock mass, they cause tensile stresses in the rock mass between the drill hole and the free face. If the tensile strength of the rock is exceeded in the process, then the rock breaks in the burden area, which should be the case in a correctly designed blast, otherwise the compressive stress waves will just disappear in the rock mass and the rock does not break. Large volumes of gases are produced under very high pressures which are dissipated into the cracks formed, and expand the cracks. If the distance between the drill hole and free face is correctly calculated, the rock mass between the drill hole and the free face will give way or yield and be thrown forward, thus fragmenting the rock. Figure 1 shows the fracturing of a rock mass.
2.2. Blast Design

The purpose of a blast design is to distribute explosive energy in such a way that certain fragmentation and muck pile displacement will be achieved. The design of any blast must encompass the fundamental concepts of ideal blast design, which are then modified when necessary to account for local geologic conditions (Konya and Walter, 1990). In order to achieve optimum results from the explosive energy, the blast must be so designed as to balance all parameters that contribute to the desired fragmentation; blast holes must be arranged in the desired manner with the correct depth, the right quantity of explosive placed into the holes and the appropriate initiating technique used to effect the detonation. Blasts should always be designed to optimize breakage and minimize overbreak (ISEE, 2011).
Factors that inform the design of a blast include the rock structure, porosity, rock density, bench geometry, explosive characteristics, charge distribution, arrangement of delays used and method of initiation. These factors are generally in two groups; controllable and uncontrollable.

2.2.1. Controllable Factors

Factors that a blaster has control over when designing a blast according to (Mirabelli, 2005) are:

- Explosives: The type of explosive and its properties affects a blast design. The following are properties and characteristics of explosives that should be considered for a blast;
  - Density of the explosive: The density of an explosive controls how much of the product can be loaded into a blast hole, and influences its sensitivity. It is defined as the mass of an explosive per unit volume usually expressed as g/cm$^3$ or lb/ft$^3$ (ISEE, 2011).
  - Velocity of detonation (VOD): This is the speed at which the detonation wave travels through an explosive, and may be expressed for either confined or unconfined conditions. An explosive’s VOD depends on the density, ingredients, particle size, charge diameter and degree of confinement (USACE, 1972).
  - Energy: An explosive’s energy can be expressed based on the weight or volume. It is expressed in either absolute or relative terms as AWS, ABS, RWS and RBS (Khoshrou, 1996).
  - Water resistance: The ability of an explosive to withstand exposure to water without suffering negative effects in performance. Some explosives can be directly pumped into blast holes filled with water while some cannot. The
presence of water can dissolve or leach some of the ingredients in an explosive, or cool the reaction to such a degree that the ideal products of detonation will not form even though the product is oxygen-balanced (Konya and Walter, 1990).

- **Form**: This refers to the way in which an explosive exists physically. Explosives either come as packaged or in bulk.

- **Confinement**: This controllable variable deals with how the explosive energy is confined so that it can fragment the rock mass. It includes:

  - **Distance of drill holes relative to one another**;

    - **Burden**: This is the distance of the explosive charge to the nearest free face. The ideal burden is the distance of radial cracking produced by the detonation to produce the desired fragmentation or movement. A burden less than the radial cracking radius causes energy venting and excessive movement from the face while a burden greater than the cracking radius causes insufficient fragmentation and movement (ISEE, 2011).

    - **Spacing**: Spacing is the distance between blast holes in a row, and measured perpendicular to the burden. (ISEE, 2011).

  - **Type and amount of stemming**: To confine the explosive energy in the blast hole, the top portion of the hole is filled with inert material. Drill cuttings are common although other materials can be used too (Bender, 1999).

  - **Distance of drill holes from a free face**, and

    - **Burden**.

  - **Amount of material surrounding the explosive in the drill hole**.
• Distribution: Explosive energy is distributed through a rock mass. The following factors determine how the energy is distributed;

  o Diameter of the drill hole: This is the measurement through the center of the drill hole opening, from one side to the other. The size of a drill hole must allow required energy level through loading density (ISEE, 2011).

  o Diameter of the explosive to be used: The diameter of an explosive is the cross-sectional width of an explosive cartridge or charge. This influences its VOD up to a certain diameter. The minimum critical diameter, which is the smallest charge diameter at which the detonation process, once it is initiated, will support itself is known as its critical diameter (ISEE, 2011).

  o Depth of drill hole: Total length of the hole drilled in a blast face.

  o Depth of charged column: This is equal to the depth of the drill hole minus the stemming length (Figure 2).

  o Spacing of the drill holes: Spacing is the distance between blast holes in a row, and measured perpendicular to the burden. (ISEE, 2011).
2.2.2 Uncontrollable Factors

Uncontrollable factors are factors that the blaster has little or no control over such as geology, rock characteristics, regulations and specifications as well as distance to the nearest structure. The blast design is therefore determined by the properties and behavior of the rock mass to be blasted (Konya and Walter, 1990). The uncontrollable variables include (ISEE, 2011):

- Rock structure: According to Bhandari (1997), differences in a rock structure are due to the origin of formation and structural features such as joints, bedding and faults and the influence of these structural features on the response of the rock mass to applied loads cannot be ignored when designing a blast.
o Jointing: A discontinuity plane of natural origin along which no significant shear displacement has occurred (Palmström, 2015).

o Bedding: This marks a sudden change in depositional conditions and possible major changes in rock properties. Beddings separate sedimentary rock layers that differ considerably from other beds (ISEE, 2011).

o Faults: Fractures in rocks where there has been relative displacement of the two sides. Actual displacement is often measured in a vertical plane perpendicular to the strike of the fracture (ISEE, 2011).

- Rock geotechnical properties: It is important to know and understand the engineering properties that characterize specific rock types for blast design and modelling purposes. These properties are often specified by these properties (ISEE, 2011):
  
  o Rock density: Rock density is defined as the rock’s mass per unit volume (ISEE, 2011). Bhandari (1997) wrote that the ease or difficulty in breaking a rock is determined by its density, and indicates the energy needed to deform and displace the rock.

  o Rock porosity: Rock porosity is a measure of the void space within a rock. A highly porous rock has a high percentage of voids or pore spaces. These voids or open spaces can increase the potential for a rock to take in and possibly hold water. Extreme porosity with vesicular basalt can effectively reduce explosive energy confinement (ISEE, 2011).

  o Rock strength: The strength of a rock is measured as the force under which rock fails or break. Rocks can fail under compression, tension or shear force (ISEE, 2011) and, according to Khoshrou (1996), this largely depends on the nature of
the mineral composition and can only be defined when factors like intensity and duration of load, size of rock samples, pressure and temperature, pore-water pressure, and failure criteria are known. Rocks are generally strongest in compression, so blast designs should place the rock in tension for breakage and in shear for creating smooth surfaces (ISEE, 2011). Table I shows rock classifications based on the uniaxial compressive strength (UCS).

Table I: Rock Classification Based on UCS (Rinehart, 1965)

<table>
<thead>
<tr>
<th>Description</th>
<th>UCS (MPa)</th>
<th>Rock Type</th>
</tr>
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<tbody>
<tr>
<td>Very high strength</td>
<td>&gt; 220</td>
<td>Quartzite, diabase and dense basalt</td>
</tr>
<tr>
<td>High strength</td>
<td>110 to 220</td>
<td>Majority of igneous rocks, weakly cemented sandstone, strong metamorphic rocks, hard shale, majority of limestone, dolomite</td>
</tr>
<tr>
<td>Medium strength</td>
<td>55 to 110</td>
<td>Many shale, porous sandstone and limestone, schistose varieties of metamorphic rocks</td>
</tr>
<tr>
<td>Low strength</td>
<td>28 to 55</td>
<td>Porous low density rocks, friable sandstone, tuff, clay shales, weathered and chemically altered rocks of any lithology</td>
</tr>
<tr>
<td>Very low strength</td>
<td>&lt; 28</td>
<td></td>
</tr>
</tbody>
</table>

- Rock velocity: This is the velocity at which the rock transmits a shock wave. Typically, stronger and denser rocks have higher velocities than weak, softer rocks. Some theories recommend the explosive VOD exceed the rock velocity (ISEE, 2011).

- Rock resilience: The rock’s ability to store the elastic energy of strain is its resilience. The most common measure of rock resilience are (ISEE, 2011);
- **Modulus of elasticity**: The modulus of elasticity is the ratio of stress to strain in simple compression or tension. If a body is compressed equally from all directions, its original volume will be decreased. Weathered and fractured rocks have a low modulus of elasticity while rocks with a higher modulus of elasticity are stronger (Khoshrou, 1996).

- **Poisson’s ratio**: This is the ratio of the lateral unit strain to the longitudinal unit strain of a rock that has been stressed longitudinally within its elastic limit (ISEE, 2011).

### 2.3 Blast Damage

Though blasting is an efficient method of fragmenting rock, it causes cracks and fractures which manifests as blast damage. Blast damage refers to the change in rock properties which reduces its performance and behavior (Singh, 1992). According to Ramulu (2012), blast damage can be grouped as fabric due to fracturing, structural damage exploiting discontinuities and shears, and lithological damage causing parting between two different rock units or lithological boundaries between similar rock types.

Iverson, et al. (2013), stated that the damage results in overbreak or rock that was not designed to be removed, loose rock to be scaled, and permanent damage to the remaining perimeter. Damage to rock walls at the perimeter of excavation are often not desired and can result in safety problems and additional costs due to (McKown, 1986):

- Extra mucking;
- Extra material to backfill overbreak;
- Additional maintenance of rock walls;
- Additional rock reinforcement; and,
• Additional pumping or grouting that may be required.

Blast damage around an underground excavation has often been assessed as overbreak rather than assessing the actual damage (Saiang, 2008). Figure 3 shows the distinction between an overbreak and blast damage. Overbreak often results in these safety problems (Iverson et al., 2013):

• Wider spans that require extra ground support and increased probability of failure if not properly assessed;
• Creation of rough and undulating back and wall surfaces that increase the hazards associated with scaling and the installation of bolts and support fittings;
• Perimeter damage to the extent of requiring more scaling, and the development of more potential loose rock; and,
• Creation of flat-arched backs which will need extra bolting where a rounded arch will naturally aid in supporting the back.

Figure 4: Blast damage (Warneke et al., 2007)
Ramulu et. al. (2009) categorized blast induced damage as:

- Near-field damage due to high frequency and critical vibrations; and,
- Far-field damage due to repeated low frequency and sub-critical vibrations. This was correlated with shear wave velocity of rock mass to develop an equation with a correlation coefficient of 0.76.

\[ D_{\text{max}} = 322.5(V_s)^{0.61}, \text{ (m)} \]  
(1)

where \( D_{\text{max}} \) is the maximum extent of rock mass damage due to repeated vibrations, and \( V_s \) is the S-wave velocity in meters per second.

Yu and Vongpaisal (1996) proposed a Blast Damage Index (Dib) to estimate the type of damage due to blasting, and concluded that damage resulting from blasting is a function of blast induced stress and rock mass resistance to damage. The Blast Index is the ratio of the blast induced stress to the resistance offered against damage.

### 2.3.1 Factors that Influence Blast Damage

Typically, problems from blasting occur because of poor blast designs; poor drilling performance, explosives loading, and firing sequencing but according to Singh (2018) and McKown (1986), blast damage is often attributed to the rock mass. Rathore and Bhandari (2007) noted that blast damage in a rock mass is related to the condition of the rock mass and the stress level experienced; the type of damage and its extent is a function of the blast design and rock mass characteristics. Factors that affect blast damage can be grouped into three categories (Singh, 2018):

- Rock Mass Quality: Blasting performance and blast damage are mainly controlled by the following rock mass characteristics;
- Number of joints,
- Rock quality designation (RQD),
- Joint filling and conditions,
- Aperture, frequency and orientation of joints,
- Weathering of rocks,
- Rock mass rating (RMR),
- Hydro-geological conditions, and
- Layers of foliation.

- Explosive Characteristics and Distribution: The following parameters should be considered during the selection and loading of explosives;
  - Bore hole pressure,
  - Velocity of detonation (VOD), and
  - Powder factor.

- Blast Design and Execution: When a blast is properly designed, drilled, loaded and fired, then blast damage can be minimized. These factors can have a major impact on blast damage;
  - Drill hole deviation,
  - Blast hole parameters,
  - Cut design and blasting,
  - Buffer holes, and
  - Perimeter hole pattern and amount of explosive.
2.4. Blast Damage Control

Geologic structures, as planes of weakness, have a tendency to fracture beyond the perimeter, and if the rock is weak, no blasting technique can possibly create a smoother, more solid face than tolerable by the rock formation. Best results occur in homogeneous formations having minimal geologic structures and seams. The common approaches to control overbreak at the perimeter of an excavation are (ISEE, 2011):

- Reduce charge weights in the final line of blast holes;
- Drill extra holes to compensate for the reduced charge weight and provide a preferential plane of weakness to demarcate the final excavation perimeter; and,
- Arrange the initiation of the boreholes defining the perimeter to get maximum relief away from the perimeter, and create a preferential splitting mechanism.

To control overbreak, the creation of loose rocks, and produce a competent final wall, several blasting techniques have been developed which together, are known as wall control blasting techniques. According to Sharma (2011), wall control techniques include change in explosive type, altering blast hole diameter, decoupling explosives, deck charging, and changing drill parameters such as burden and spacing. These techniques are adopted to control the extent to which rocks are broken; thus, it ensures that rocks are not broken beyond the excavation limit, and to protect personnel and equipment from back break and loose rock falls. Though a large amount of drilling is required in control blasting, it is preferred over conventional blasting in underground operations because of these advantages (Ramulu, 2012):

- The shape of the opening maintains a smooth profile;
- Stability of the opening and the stand-up-time of the tunnel are improved;
• Support requirement is reduced;
• Overbreak is reduced to minimize unwanted excavations and filling to reduce cost and cycle time; and,
• Ventilation improves due to a smoother profile.

Various wall control blasting techniques that are implemented to protect final rock faces in both surface and underground blasting include presplitting, line drilling, cushion blasting (trim blasting), and smooth blasting. If back breaks are not controlled, it eventually decreases the overall slope angle, with major economic consequences such as decreased recoverable ore reserves and increased ore to waste ratios (Singh et al., 2014). Thus, the best approach is to control the effects of blasting so that the inherent strength of the walls is not destroyed. The ultimate application of a wall control blasting technique depends on the rock characteristics, rock density, and geology (Tose, 2006). The application of wall control blasting techniques is an important practice in most mining operations with pre-splitting being the most frequently used technique.

2.4.1. Presplitting

Presplitting forms a fracture plane beyond which the radial cracks from blasting cannot travel. Presplitting involves the use of lightly loaded, closely spaced drill holes (Figure 4) which are often fired before the production blast (Konya and Walter, 1991). Presplit holes are fired instantaneously or with millisecond delays. According to Singh et al. (2014), the concept of pre-splitting is that radial cracks from lightly shot boreholes either join an adjacent borehole or other radial cracks from an adjacent hole to form a plane of broken rock between the boreholes. This is a technique for protecting the final wall from being damaged by production blasting.
2.4.2. Line Drilling

Line drilling is a simple but expensive technique that involves the drilling of closely spaced small diameter holes at the perimeter of the excavation. The drill holes are not loaded with explosives (Figure 5), but form a discontinuity at the excavation limit (Sharma, 2011). Line drilling may act as stress concentrators causing the fracture to form between the line drill holes during the production blasting cycle and is more commonly used together with either presplitting or trim blasting instead of being used alone. Due to high cost of drilling and difficulty in maintaining hole orientation because of very close spacing, this technique is not usually used except in situations where it is feared that even a light loading of the perimeter hole may cause damage beyond the perimeter (McKown, 1986).
2.4.3. Cushion Blasting

Cushion blasting is also known as trim blasting. Konya and Walter (1991) defined cushion blasting as a control technique that reduces the rate of energy release against the final wall, and used to clean up a final wall after the detonation of production holes. In cushion blasting, a single row of holes is drilled at the perimeter of the excavation and loaded lightly with explosive as shown in Figure 6. The explosive charge is decked with inert material, stemmed throughout the entire column and initiated with detonating cord or millisecond delay to minimize the delay between holes (Khoshrou, 1996). This technique offers no protection to the final wall,
but rather reduces the rate of energy release against the final wall. Trim holes are fired after production blasting.

![Figure 7: Schematic for cushion blasting (Modified after McKown, 1986)](image)

**2.4.4. Smooth Blasting**

Smooth blasting, similar to pre-splitting involves drilling a number of closely spaced holes around the final excavation (Figure 7). The holes are loaded with light, well distributed charges and fired after the main production holes or the last delay (Khoshrou, 1996). Upon detonation of the charged boreholes, the rock is cracked smoothly along the plane coincident with the axes of the holes. The method reduces drilling costs to a minimum and provides improved results over the previous presplitting and other known methods. Balasubramanian
(2017) stated that smooth blasting is the most widely accepted technique for overbreak control in underground headings and stopes. In smooth blasting, delay intervals between perimeter holes and the nearest production hole are kept high to facilitate the complete movement of material before the perimeter holes detonate so that gas expansion in the perimeter holes occur towards the opening (Ramulu, 2012).

![Figure 8: Typical layout for smooth blasting (Modified after McKown, 1986)](image)

2.5. Factors that Affect Perimeter Control

In perimeter control blasting, four primary factors affect the smoothness and soundness of the remaining rock walls. These factors are drilling accuracy, spacing and loading of perimeter holes, treatment of first-row-in holes, and geology (McKown, 1986).

2.5.1. Drilling Accuracy

According to McKown (1986), drilling accuracy is very important in achieving good perimeter control results but is often overlooked. Incorrectly collared and misaligned perimeter
holes can result in overbreak. Correct hole alignment is essential for perimeter holes to lie in the same plane. Deviation of more than 4 to 6-inches can adversely affect the results. Perimeter and first-row-in holes should be kept parallel so that the burden remains constant from the top to the bottom of the adjacent holes. Iverson et. al. (2013) stated that jackleg drills cannot easily achieve precision drilling compared to drill jumbos, which have precision control.

2.5.2. Spacing and Loading of Perimeter Holes

Except for massive, homogenous rock formations, perimeter hole loadings and spacings should be at the lower limit of those recommended (Table II), or results will be poor. It is essential that results of perimeter control work be evaluated regularly, regardless of the procedure used. Test holes should be fired in advance of production work to verify an initial loading and spacing, or initial rounds should be considered as test rounds so that evaluation and required changes can be made (McKown, 1986).
### Table II: Proposed Parameters for Control Blasting (DuPont Blasters Handbook)

<table>
<thead>
<tr>
<th>Blast Type</th>
<th>Hole Diameter (in)</th>
<th>Spacing (ft)</th>
<th>Burden (ft)</th>
<th>Explosive Charge (lb/ft)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Cushion Blasting</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>2 - 2½</td>
<td>2</td>
<td>4</td>
<td>0.08 - 0.25</td>
</tr>
<tr>
<td></td>
<td>3 - 3½</td>
<td>4</td>
<td>5</td>
<td>0.13 - 0.50</td>
</tr>
<tr>
<td></td>
<td>4 - 4½</td>
<td>5</td>
<td>6</td>
<td>0.25 - 0.75</td>
</tr>
<tr>
<td></td>
<td>5 - 5½</td>
<td>6</td>
<td>7</td>
<td>0.75 - 1.00</td>
</tr>
<tr>
<td></td>
<td>6 - 6½</td>
<td>7</td>
<td>9</td>
<td>1.00 - 1.50</td>
</tr>
<tr>
<td><strong>Smooth Blasting</strong></td>
<td>1½ - 1⅓/4</td>
<td>2</td>
<td>3</td>
<td>0.12 - 0.25</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>2½</td>
<td>3½</td>
<td>0.12 - 0.25</td>
</tr>
<tr>
<td><strong>Presplitting</strong></td>
<td>1½ - 1⅓/4</td>
<td>1 - 1½</td>
<td></td>
<td>0.08 - 0.25</td>
</tr>
<tr>
<td></td>
<td>2 - 2½</td>
<td>1½ - 2</td>
<td></td>
<td>0.08 - 0.25</td>
</tr>
<tr>
<td></td>
<td>3 - 3½</td>
<td>1½ - 3</td>
<td></td>
<td>0.13 - 0.50</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>2 - 4</td>
<td></td>
<td>0.25 - 0.75</td>
</tr>
</tbody>
</table>

---

### 2.5.3. Treatment of First-Row-In Holes

Carefully drilling and loading the row of holes adjacent to the perimeter holes is very important, but often ignored. By overcharging these holes or positioning them too close to the perimeter holes, it is easy to cause backbreak beyond the perimeter line with resulting overbreak and damage to the final rock surface. First-row-in holes should be drilled parallel to the perimeter holes, and have a burden, spacing and loading about ½ to ¾ that of the production holes. It is important for first-row-in holes to provide adequate relief and uniform burden for subsequent perimeter hole detonations (McKown, 1986).
An adequate geotechnical investigation is the first step in assessing the potential problems caused by the geology. There are several important geologic factors which influence results of perimeter control blasting, and may be divided into intact rock properties and properties of the rock mass as (McKown, 1986):

2.5.4.1. Intact Rock Properties

- Intact strength and soundness: The intact tensile and compressive strength is an indication of how easily the rock will break. The tensile strength is the most relevant, and can be determined using a direct tensile strength test or the simplified Brazilian tensile strength test. The compressive strength, although not as appropriate for blasting work, is commonly known and measured using an unconfined compression test or a point load strength index test (McKown, 1986).

- Degree of weathering: Severely weathered rock can be effectively reduced to a soil with only fragments of strong rock remaining. In such cases, blasting may not be required for excavation. Where blasting is utilized in weathered rock, spacing and loading of perimeter holes must be reduced to levels well below those conventionally used or excessive rock and overbreak will result (McKown, 1986).

2.5.4.2. Rock Mass Properties

- Degree of fracturing: The degree of fracturing of a rock mass can be best expressed in an NX or larger core boring using RQD. RQD is a modified core recovery percentage in which all the pieces of sound core over 4 in in length are counted as recovery. The smaller pieces are considered to be a result of close
jointing, shears, faults, or weathering in the rock mass and are not counted (McKown, 1986).

- Spacing of discontinuities: This usually refers to the average spacing of a set of parallel joints, and is best measured at rock outcrops or in existing rock cuts. Close spacing of joints (less than 8 in.) is also an indicator of a highly fractured rock mass in which close spacing and very light loading of the perimeter holes may be required to get satisfactory results (McKown, 1986).

- Joint orientation: In perimeter control, the orientation of discontinuities relative to the final rock surface can have a great effect on the results. However, with discontinuities oriented at angles as low as 20 to 30 to the proposed perimeter line, good perimeter control results can be achieved (McKown, 1986).

- Joint condition: Aspects of joints that can affect perimeter control results include continuity, amount of separation, and the degree of weathering of the walls of the joint. Continuous joints which intersect at a perimeter hole and a separation greater than 1 mm can result in venting of explosive gases unless the hole is fully stemmed. Overbreak can also result from breakage back into the wall along the joint. Weathered joint walls are more susceptible to cracking with resulting wall damage (McKown, 1986).
3. Research Methodology

Experimental blasts were conducted at the east heading (Figure 9) of the Orphan Boy Mine to control overbreak that results from blasting. The Orphan Boy Mine, began as an underground mine in the 1880s. Montana Tech inherited this old mine in 2010 and developed it into the Underground Mine Education Center (UMEC). In 2012, a decline was driven by contractors in order to reach the 100 foot level.

Figure 9: The Orphan Boy area with east heading (in red)
The UMEC serves as a hands-on educational environment for teaching, learning, and research. Students, under the guidance of faculty, are mainly responsible for drilling, blasting, mucking, and hauling activities in the mine as part of the Practical Underground Mining Course. This is to give mining engineering students a hands-on experience with hardrock underground mining activities. Ground support at the UMEC is generally bolt and mesh, as shown in Figure 9.

Drilling at the Orphan Boy Mine uses pneumatic jackleg drills with a 6 ft (1.8 m) drill steel and 1-5/8 in (36 mm) carbide insert cross drill bits operating at 95 psi. Typical blasts in the mine are designed to be 10 ft (3.05 m) high by 10 ft wide, with 6 ft advance. One or two holes in the center of the design are often used as burn holes to allow free movement during a blast. Drill
holes are primed with 8 g boosters, loaded with ANFO at a density of 1 g/cm³, and initiated with long period (LP) non-electric millisecond delays. Figure 10 shows students drilling a blast round at the mine.

3.1. Blast Design

The geology of the rock found at the UMEC was considered in designing the blast. Rose (2017) described the rock as granitic with a density of 2.6 g/cm³, UCS of 14.27 MPa (2,070 psi),
Poisson’s ratio of 0.22, Rock Mass Rating (RMR) of 69, and Young’s modulus of 36.13 x 10^3 (5.24 x 10^6 psi). Different researchers proposed different factors for designing a smooth blast. According to Singh (2018), the optimum spacing and burden for perimeter holes depend on the rock type and drill hole diameter (Equations 2 and 3). The design factors are given in Table III.

\[
S = S_f \times d \quad \text{(m)}
\]

\[
B = B_f \times S
\]

where \(S\) is the spacing between the perimeter holes, \(S_f\) is the spacing factor, \(d\) is the drill hole diameter, \(B\) is the burden for the perimeter holes, and \(B_f\) is the burden factor.

### Table III: RMR and Perimeter Hole Pattern Design Factors (Singh, 2018)

<table>
<thead>
<tr>
<th>RMR Value</th>
<th>Spacing factor ((S_f))</th>
<th>Burden factor ((B_f))</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;45</td>
<td>13 - 14</td>
<td>1.25</td>
</tr>
<tr>
<td>45 - 55</td>
<td>14 - 15</td>
<td>1.2</td>
</tr>
<tr>
<td>55 - 65</td>
<td>15 - 16</td>
<td>1.15</td>
</tr>
<tr>
<td>65 - 75</td>
<td>16 - 17</td>
<td>1.1</td>
</tr>
<tr>
<td>&gt;75</td>
<td>17 - 17.5</td>
<td>1.05</td>
</tr>
</tbody>
</table>

Holmberg (1982) also proposed the following design parameters for perimeter hole spacing, burden to spacing ratio, and linear charge concentration for smooth blasting:

\[
S_{dc} = 16 \times d_b, \quad \text{m}
\]

\[
m_{dc} = 1.25
\]

\[
q_{lcc} = 90 \times (d_b)^2, \quad \text{kg/m}
\]

where \(S_{dc}\) is the spacing of perimeter holes while drilling, \(m_{dc}\) is the burden to spacing ratio of the perimeter holes while drilling, \(q_{lcc}\) is the linear charge concentration in the perimeter holes, and \(d_b\) is the diameter of blast holes.
Table IV shows the blast design details proposed by Olofsson (1988) for smooth blasting.

Table IV: Design Details for Smooth Blasting

<table>
<thead>
<tr>
<th>Hole Diameter (mm)</th>
<th>Burden (m)</th>
<th>Spacing (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>25 - 32</td>
<td>0.30 - 0.45</td>
<td>0.25 - 0.35</td>
</tr>
<tr>
<td>25 - 48</td>
<td>0.70 - 0.90</td>
<td>0.50 - 0.70</td>
</tr>
<tr>
<td>51 - 64</td>
<td>1.00 - 1.10</td>
<td>0.80 - 0.90</td>
</tr>
</tbody>
</table>

3.1.1. Explosive Properties

The explosive used for the perimeter control was the 22 mm diameter, continuous packaged Orica Detagel™ Presplit product which contains a continuous length of detonating cord running through the entire length of the package. The buffer and production holes were charged with Amex™ ANFO, 8g Pentex™ SB Cast Boosters, and Exel™ LP detonators (periods and corresponding delays shown in Appendix A). The explosive properties are listed in Table V.

Table V: Properties of Explosives used for the Research

<table>
<thead>
<tr>
<th>Explosive</th>
<th>VOD</th>
<th>RBS</th>
<th>RWS</th>
<th>Density</th>
<th>Fume Class</th>
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</thead>
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<tr>
<td>Detagel™ Presplit</td>
<td>4.2</td>
<td>138</td>
<td>103</td>
<td>1.05 - 1.15</td>
<td>1</td>
</tr>
<tr>
<td>Amex™</td>
<td>3.3</td>
<td>100</td>
<td>119</td>
<td>1</td>
<td>1</td>
</tr>
</tbody>
</table>
4. Experimental Blasts and Results

Four experimental blasts were conducted to determine which blast design works best for the Orphan Boy Mine. Figure 11 shows a typical experimental setup. The drill holes were 6 - 7 ft long (1.8 - 2.1 m), and 36 mm in diameter. Based on the spacing range suggested by Olofsson (1988), and equations (2) and (4), the spacing of the perimeter holes for the experimental blasts was determined to be 0.5 to 0.7 m (20 in to 28 in). A total of fifty-one perimeter holes were blasted in these four experiments; thirty-eight in smooth blasting, and thirteen in presplitting.

![Figure 12: Typical experimental Setup (Modified after Anon, 2015)](image)

4.1. Experiment 1

A total of thirty holes were blasted; eleven perimeter holes, three lifter holes, two burn cuts and fourteen stoping holes. The lifter and stoping holes were bottom primed (Figure 12) and charged with ANFO, and the perimeter holes were bottom primed with five sticks of Orica DetageⅠPresplit each. The first experimental smooth blasting was conducted at the east heading of the Orphan Boy. The blast design was 2.6 m (8.5 ft) high and 2.6 m wide with a 1.8 m
(6ft) advance. Figure 13 shows the blast layout after drilling. The perimeter holes were spaced at 0.7 m (28 in) and delayed for 1,000 ms after the lifter holes detonated. The timing sequence for this blast is shown in Appendix B.

Figure 13: Schematic of bottom-hole priming (Modified after Connolly, 2018)

Figure 14: Experiment 1 design layout after drilling
4.1.1. Result and Discussion

Out of eleven perimeter holes that were blasted in this experiment, only one hole resulted in a half barrel as expected (Figure 14). It was deduced that, most of the explosive used in charging the perimeter holes came out due to the pulling force from the other holes because of the tie-up and bottom priming of the hole. Also the holes were not spaced closely enough to split as expected due to so much energy being detonated at a time. Based on the number of perimeter holes blasted and the half barrels that resulted, the success of this experiment was less than 10%.

Figure 15: Result of experiment 1
4.2. Experiment 2

Based on observations from the first experiment, the perimeter holes for the second experimental blast in the east heading were top primed (Figure 15) with four sticks of Orica Detagel™ Presplit each. The number of holes blasted in this experiment were thirty-two; twelve stoping holes, two cut holes, four lifter holes, and fourteen perimeter holes. The blast design after drilling is shown in Figure 16, and the firing sequence is shown in Appendix B. The opening for this blast was 2.4 m (8 ft) wide and 2.7 m (9 ft) high with 1.8 m (6 ft) advance. The spacing for the perimeter holes for this blast was reduced to 0.6 m (24 in).

Figure 16: Schematic of top-hole priming (Modified after Connolly, 2018)
4.2.1. Result and Discussion

Perimeter holes in this experiment were spaced closer than in the first experiment. Three perimeter holes out of fourteen blasted in this experiment produced half-barrels, which was an improvement on the first experiment. The charging of the perimeter holes was changed to top priming, and the number of Orica Detagel™ Presplit reduced from five to four. Eleven of the perimeter holes did not result in half-barrels which represents 79% of the perimeter holes. Top priming resulted in some perimeter holes not firing as the detonators were pulled from the explosives during initiation. The success of this experiment was 21% based on the number of resulting half-barrels out of fourteen perimeter holes blasted.
4.3. Experiment 3

Thirteen perimeter holes were drilled with a spacing of 0.5 m, closer than spacing used in experiments 1 and 2. The perimeter holes were bottom primed with four sticks of Orica Detagel™ Presplit each based on the increased success of the second experiment and to ensure all perimeter holes were initiated. A total of twenty-nine holes were blasted in this experiment. The blast design was 2.6 m (8.5 ft) high and 2.6 m wide with a 1.8 m (6ft) advance. Figure 18 shows the design layout for this blast after drilling. The perimeter holes for this experiment were blasted together with the lifter holes, 1,000 ms after the last buffer hole detonated (Appendix B).
4.3.1. Result and Discussion

The perimeter holes for this experiment were top primed like in experiment 2, but closer spaced than in experiment 2. Out of thirteen perimeter holes blasted in this experiment, two half-barrels resulted (Figure 19). Though the spacing of 0.5 m for this experiment was based on observations from experiment 2, the resulting half barrels were fewer. The success of this experiment was 15% based on the number of half barrels produced out of thirteen perimeter holes blasted.
4.4. Experiment 4

Thirteen perimeter holes were blasted in this experiment, and spaced at 0.5 m (Figure 20). The technique adopted for this experiment was presplitting thus, only the perimeter holes were blasted. The holes were top primed with four sticks of Orica Detagel™ Presplit and fired with 5,500 millisecond delays to produce planes of fracture and serve as buffer for the main blast.
4.4.1. Result and Discussion

Planes of fracture were visible after the thirteen holes in this experiment were blasted. However, the energy exerted from the explosives resulted in excessive shattering of the final rock face (Figure 21), resulting in loose rocks which had to be scaled (Appendix D).
Figure 22: Result of experiment 4
5. Conclusions

Four experimental control blasts were conducted at the Orphan Boy to explore the applicability of controlled blasting. Application of controlled blasting would protect the safety of students, faculty, and staff, prevent overbreak, and to ensure the stability of openings and workings. Specifically, three smooth blasts and one presplit blast were conducted to analyze their effectiveness. Based on the resulting half-barrels, the success of the three smooth blasting was less than 20% and the presplit blasting resulted in extensive fracturing of the adjoining rock mass.

Fracturing, weathering, and jointing of the rock mass were observed as factors that limited the success of the control blasts. The accuracy of drill holes in this research could not be ascertained as drilling was done using pneumatic jackleg drills. Openings at the Orphan Boy are small when compared to successful works done by published researchers, thus the buffer row design concept as proposed by Iverson et. al (2013) and Ramulu (2012) was not adopted for blasts in this research.

The loading density of the Orica Detagel™ Presplit (0.8 lb/ft using the equation below) was well above the range proposed for these experiments, which could have had an impact on the performance of the blasts conducted in these experiments.

\[
D_l = 0.3404 \times D_e^2 \times d_e
\]  

(7)

where \(D_l\) is the explosives loading density (lb/ft), \(D_e\) is the diameter of the hole (in), and \(d_e\) is the density of the explosive (g/cm³).
6. Recommendations

Based on the results of this research, the following recommendations are made:

- Several additional experimental smooth and presplit blasts with different designs should be conducted and its progress monitored for future use at the Orphan Boy.

- Future controlled blasts should consider the use of the buffer row concept, and its impact assessed.

- A thorough research into the effects of the geological conditions of rock mass found at the Orphan Boy on controlled blasting should be conducted, and applicable design parameters proposed to aid in the success of future controlled blasts.

- The number of Orica Detagel™ Presplit sticks used in the perimeter holes should be reduced to two sticks per hole for smooth blasts, and one stick per hole for presplits. Thus, the loading density per 6 ft hole for smooth blasts will be 1.6 lb and 0.8 lb for presplit blasts. This will reduce the amount of explosive energy used in fragmenting the perimeter of excavation.

- Holes should be surveyed for precision after drilling so as to inform decisions that would be carried out regarding perimeter holes.

- Smooth blasting with dummy holes should be considered in subsequent blasts. This approach will help reduce the impact on the final rock face after blasting.
7. References


<table>
<thead>
<tr>
<th>Period</th>
<th>Time (m/s)</th>
</tr>
</thead>
<tbody>
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<td>1</td>
<td>400</td>
</tr>
<tr>
<td>2</td>
<td>800</td>
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<td>3</td>
<td>1200</td>
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<tr>
<td>15</td>
<td>6000</td>
</tr>
<tr>
<td>16</td>
<td>6500</td>
</tr>
</tbody>
</table>
9. Appendix B: Firing Sequence of Blasts

Figure 23: Experiment 1 firing sequence

- Burn cut, uncharged
Figure 24: Experiment 2 firing sequence
Figure 25: Experiment 3 firing sequence
10. Appendix C: Tie-in for Blasts

Figure 26: Experiment 2 tie-in
Figure 27: Experiment 3 tie-in
11. Appendix D: Experiment 4 Results

Figure 29: Final face after blasting
Figure 30: Final face after scaling
Figure 31: Back and face after production blast
Figure 32: Resulting oversized rock after production blast
SIGNATURE PAGE

This is to certify that the thesis prepared by Jewel Sakyi entitled “Wall Control Blasting at the Orphan Boy Mine – Butte, Montana” has been examined and approved for acceptance by the Department of Mining Engineering, Montana Technological University, on this 22nd day of April, 2019.

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Member, Examination Committee

Diane Wolfgram, PhD, P.E., Professor
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