CHAPTER I
INTRODUCTION

1.1 Research Background

Limestone is the main substance in cement production. The massive and hard nature of limestone requires its mining to involve a blasting process in breaking down or dismantling rocks from their rock mass. Rock blasting is one of the most important stages in mining operations. Blasting aims to break up material from its parent rock so that the resulting fragmentation size can facilitate subsequent mining activities (Putri, 2018). There are several parameters that determine the success of the blasting stage, such as ground vibration, air blast, but the main parameter is the fragmentation results (optimal recovery rate), which will influence the *harga pokok produksi* (HPP) incurred to break up one ton of limestone (Monica & Yulhendra, 2021).

Fragmentation is one of the main parameters used to evaluate blasting activities. Off-the-mark blasting fragmentation will result in the formation of large rock chunks (boulders), which can be quite a challenge in succeeding operational processes, such as difficulties in transporting and in post-processing (Monica & Yulhendra, 2021). Several factors, mainly the surrounding geological conditions, explosive specifications, and blasting geometry, influence the fragmentation quality of blasted rocks (Sukmara et al., 2020).

PT. Semen Tonasa is a company engaged in the cement industry, with its main mining commodities being Limestone. PT. Semen Tonasa carries out its mining in Biring Ere Village, Bungoro District, Pangkajene and Kepulauan Regency, South Sulawesi Province. PT. Semen Tonasa employs the open-cut/open-cast mining method in its limestone mining operations, where mining is carried out by cutting the side of the hilltop down following the contour of the land with relatively shallow excavations, forming terraces (benches) in the excavation process. The rock mass are sufficiently strong, conventional equipment is incapable of excavating them. Consequently, PT. Semen Tonasa implements blasting (Rabbani, 2020).

Blasting at PT. Semen Tonasa is supported by two primary contractors, namely PT. United Tractor Semen Gresik (UTSG) as the provider of blasthole drilling services, and PT. Dahana as the main provider, processor, and executor in blasting activities, accompanied by the blasting crew of PT. Semen Tonasa. During the observation process, blasting activities at the limestone mine of PT. Semen Tonasa were concentrated on area B8, B9, and B12. The explosives used are Ammonium Nitrate Fuel Oil (ANFO) and Dahana Bulk Emulsion Explosives (DABEX) (Adam Rhisky et al., 2021). Both of these explosives, ANFO and DABEX, have different specifications and characteristics, resulting in varied outcomes for each explosive. There are many differences in terms of specifications and characteristics, ranging from the water resistance properties of DABEX and ANFO to the relative weight strength and relative bulk strength values of DABEX which are different from ANFO. These numerous differences can lead to varying performances, especially in terms of fragmentation results (optimal recovery rate), which significantly the cost per ton of blasted rock (Boy et al., 2021)

To address the problem described above, it is deemed necessary to conduct research related to each explosive material entitled "Comparative Study Of The Performance Of Dabex And Anfo Explosives Based On Fragmentation And Its Influence On Blasting Costs" with the objective of evaluating the performance of each explosives and gather other valuable data that may assist future strategic choice-making.

1.2 Problem Formulation

PT. Semen Tonasa carries out blasting activities as a method of dismantling limestone. This blasting process involves two types of explosives, namely Ammonium Nitrate Fuel Oil (ANFO) and Dahana Bulk Emulsion Explosives (DABEX). These two types of explosives have different specifications and characteristics, resulting in varied outcomes for each explosive, which can directly influence the HPP. The problem formulation for this study, as indicated by the described problem, are as follow:

1. How do ANFO and DABEX explosives perform in terms of the quality of fragmentation and mine HPP (cost/ton of diggable material).

2. How does fragmentation (optimal recovery rate) influence HPP (cost/ton of diggable material).
3. What are the ideal blasting conditions (best practices) that can be achieved by each explosive on each area based on the aspects of HPP (cost/ton of diggable material).

1.3 Research Objectives

Based on the formulation of the problems that have been described, the objectives of this study are as follows:

1. To evaluate the performance of ANFO and DABEX explosives in terms of the quality of fragmentation and HPP (cost/ton of diggable material).
2. To analyze the influence of fragmentation (optimal recovery rate) on HPP (cost/ton of diggable material).
3. To analyze the most ideal blasting condition (best practice) that can be achieved by each explosive on each area based on the aspect of HPP (cost/ton of diggable material).

1.4 Research Benefits

Based on the previously outlined research objectives, this study aims to shed light on how ANFO and DABEX perform in terms of fragmentation quality (optimal recovery rate), a factor that significantly influence HPP. Additionally, it also aims to analysis the optimal conditions in terms of HPP that each explosive can achieve in each area, while taking its geological conditions into account. This information could aid future strategic decision-making.

1.5 Scope of Research

This research was conducted in Biring Ere village, Bungoro District, Pangkajene and Kepulauan Regency, South Sulawesi Province. This research was conducted for 4 months from March 2024 to June 2024 by collecting primary data such as rock factors, and blasting fragmentation, while secondary data was in the form of average blasting geometry, and the prices of material components and accessories used in blasting activities. This research focuses on a comparative study between two types of explosives to determine the performance of each explosive in terms of fragmentation quality, which will determine the percentage of material that can be excavated (diggable material) and through that the HPP (cost/ton of diggable material) will be obtained. In caculating the best practice or the most ideal outcome that can be achieved by each explosive in each area in terms of fragmentation quality and HPP (cost/ton of diggable material), rock factor data will be used, which describes the geological conditions of each area. By utilizing this data the best practice or the most ideal outcome that can be achieved by each explosive in each area can be predicted. The collected data is then processed and analyzed using Wipfrag software to determine the fragmentation quality of each explosive and then processed again in Excel to determine the optimal recovery rate and HPP (cost/ton of diggable material) for each explosive.

CHAPTER II
LITERATUR REVIEW

2.1 Drilling

Blasting is a very efficient and often employed technique for breaking down or disassembling materials. This method aims to dismantle or break up the material into smaller pieces, commonly referred to as fragments. In order to achieve good fragmentation, a drill hole needs to be made that will create a void that will then be filled with explosives as needed. The hole drilled to fill the explosives in this way is referred to as blast hole. The act of drilling holes in this manner is referred to as blast hole drilling. The tool used to drill these holes is called a blast hole drill or often simply called a drill, here are the basic concepts related to drilling for blasting and the types of drill bits (Gokhale, 2011).

2.1.1 Basic principles of drilling and drill bits

In practice, drilling for blasting is done using a drill bit or drill bit connected to the bottom of the drill rod. The purpose of this drill bit is to destroy the rock at the tip of the drill by utilizing the continuous energy supply from the drill rod, so that the drill bit can continue to dig rocks (Hustrulid, 1999). When the drill bit breaks fresh rocks, also known as cutting rocks, the rocks are propelled to the surface using liquid or fluid that is expelled from the drill rod through the drill bit. This allows the drill bit to continue drilling fresh rocks (Gokhale, 2011). There are several types of drill bits, namely (Suhascaryo, 2020):

1. Drag bit

   This type has no rotating parts. Drag bit consists of three wing blades that are used to drill soft formations by digging into the formation surface. Drilling fluid is channeled directly to the wings to effectively clean the excavation. There are several disadvantages associated with the use of drag bits. First, it is difficult to achieve a straight borehole. Second, there is a large scratch between the borehole and the formation, which causes rapid wear on the bit and shrinkage of the borehole. Third, there is a risk of bit balling due to lack of proper hole cleaning. A Figure of a drag bit can be seen in Figure 1.

Figure 1. Drag bit (Suhascaryo, 2020)

2. Roller-Cone bit

Roller-cone bits have cones that can rotate so they can destroy the rock they penetrate. The advantages of this type of drill bit compared to drag bits are:

a. Less torque
b. Finer powder
c. Resulted drill hole does not immediatly shrink

A cross-sectional illustration depicting a roller-cone drill bit can be seen in Figure 2.

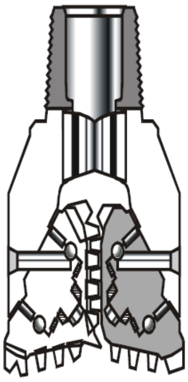

Figure 2. Roller-cone bit (Suhascaryo, 2020).

3. Diamond bit

Diamond bit is a chisel that uses diamond as the chisel tip. The reason diamond is used is because diamond is a mineral with the highest hardness (10 Mohs), the hardness of diamond itself can be around four to five times harder than tungsten carbide. The advantage of using diamond bits is their durability, these drill bits can stay in the hole for a very long time, with drilling times that can reach 300 hours. A cross-sectional schematic of the diamond bit can be seen in Figure 3.

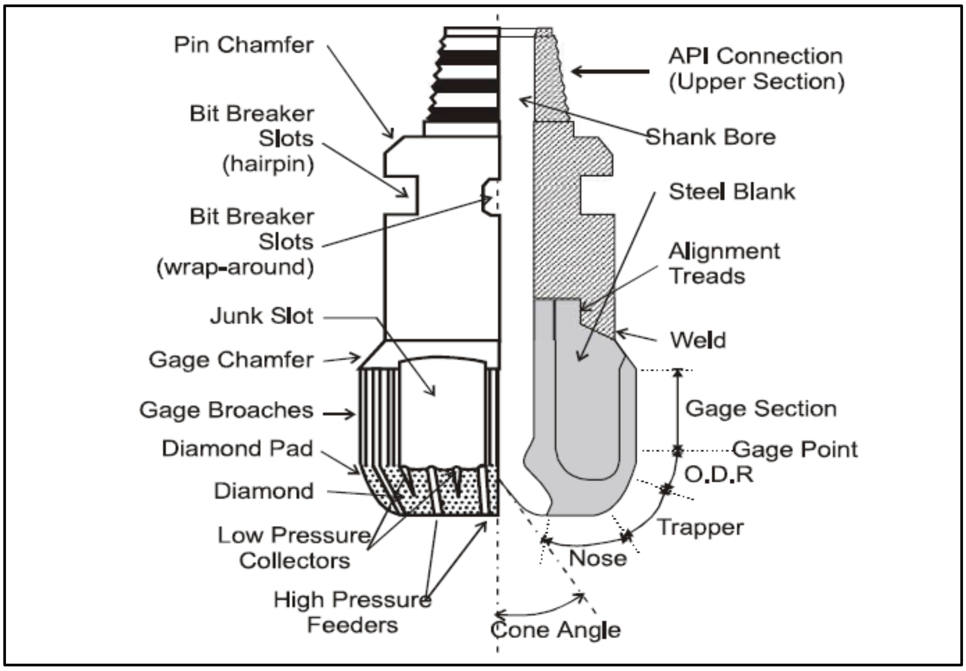

Figure 3. Cross-sectional schematic of the diamond bit (Suhascaryo, 2020).

Based on the method or principle of operation, there are several types of mechanical drilling systems, the following is an explanation regarding each mechanical drilling system (Gokhale, 2011):

1. Percussion drill

   In percussive drilling techniques, the energy produced by the drilling machine is channeled through the drill rod and drill tip to destroy rock. The main part of this drilling machine is the piston which is responsible for pushing and pulling the middle part of the drill rod, which is called the leg. In the percussive method, the process of destroying the rock surface occurs by the drill tip. For example, a hammer drill is a type of drilling tool that uses this principle.

2. Rotatory-percussion drill

   The rotary-percussive drilling method involves the combination of rotating movement and impact action of the drill bit, resulting in a process of crushing and grinding of the rock surface. This technique can be applied to various types of rock.

2.1.2 Drill pattern

One of the parameters that needs to be considered for successful blasting is the drilling and blasting pattern used, which includes the layout of the blast holes used. Proper drilling patterns are a key factor in achieving desired results in mining operations (Minara & Yulhendra, 2020). Drilling pattern A drilling pattern is a pattern used in the drilling process to determine the location of blast holes in an organized manner. There are two types of drilling patterns that are commonly used, namely (Sari et al., 2020):

1. Parallel drilling pattern

   In a parallel drilling pattern, blast holes are placed in consecutive and parallel rows at a certain distance. This means that the blast holes are arranged in a straight, parallel line. This drilling pattern is often used when the main goal is to achieve regular and uniform rockfall in a predetermined direction. An illustration of this pattern can be seen in Figure 4.

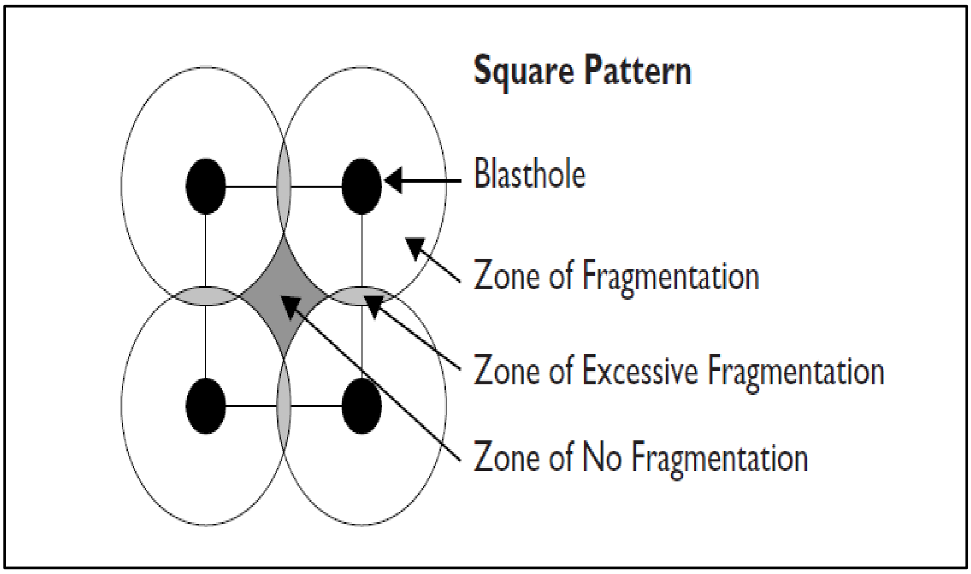

Figure 4. Parallel drilling pattern (Gokhale, 2011)

2. Staggered drilling pattern

   Staggered drilling pattern involves placing blast holes that interact criss-cross or alternately in each column. The blast holes in one column are placed between the blast holes in the previous column. This design is employed to enhance fragmentation efficiency and prevent the buildup of rock particles around the blast hole. This pattern is often used when the main objective is to maximize rock fragmentation to suit operational needs. An illustration of this pattern can be seen in Figure 5.

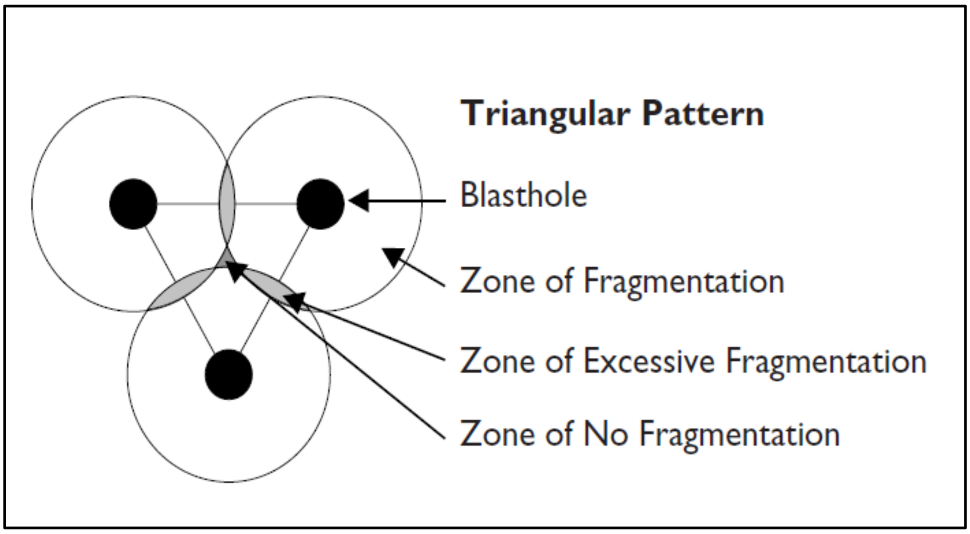

Figure 5. Staggered drilling pattern (Gokhale, 2011)

2.2 Mine Surace Blasting

In the mining industry, especially in the production stage, blasting is a procedure that entails the extraction of rock from its rock mass or parent rock through the use of explosives. The main purpose of blasting is to remove large amounts of hard material efficiently with optimal results (Hustrulid, 1998). The optimal results can be observed in a variety of ways, including flyrock, ground vibration and air blast, but one of the main parameters in evaluating blasting is fragmentation. The fundamental concept of rock fracture or the mechanism of rock devastation must be comprehended before the impact of fragmentation on optimal recovery rates can be assessed (Khademian, 2024).

2.1.1 Mechanism of rock breakage

There are numerous theories that have been proposed regarding the mechanism by which rocks are destroyed during blasting. However Konya (1990) stated that the rock disintegration mechanism induced by blasting is comprised of four primary stages:

1. In general, the process of rock destruction due to blasting activities starts from a complex and dynamic series. The first phase commences with the emergence of a pressure wave or stress wave upon detonation. These waves move at speeds close to the speed of sound, compressing the rocks around the explosion area with great force. The impact is the creation of microfractures or small cracks in the rock structure, especially around the drill wall which is the focal point of the explosion, this process can be seen in Figure 6.

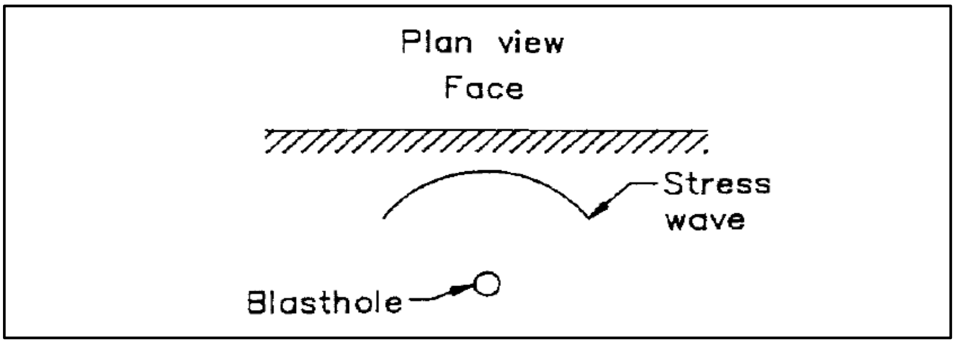

Figure 6. Mechanism of rock breakage, stage 1 (Konya, Calvin & Walter, Edward, 1990)

2. Pressurization or an increase in pressure will occur in the blast hole as a result of the detonation gas consistently filling the blast hole after the pressure wave has passed. The result of this increase in pressure will cause radial cracks or fractures that are perpendicular to the surface of the drill hole forming a circle. These stages can be seen in Figure 7.

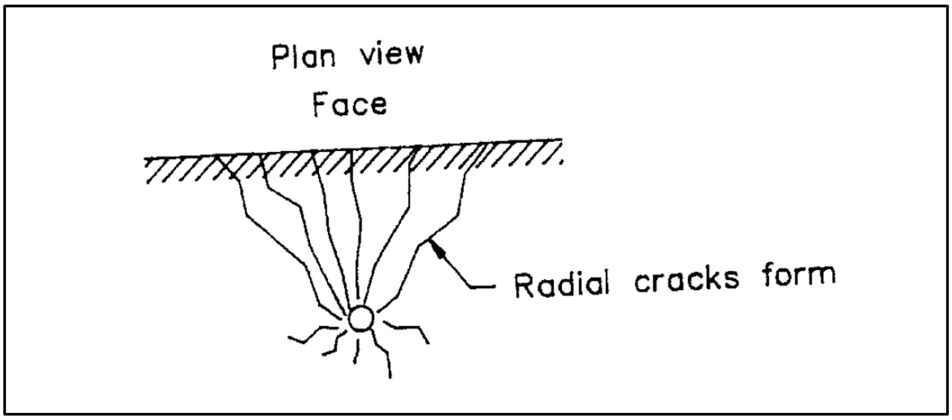

Figure 7. Radial crack formation (Konya, Calvin & Walter, Edward, 1990).

3. High pressure gas will begin to penetrate or intrude the fractures created by the radial crack until the fracture is fully compressed to approximately 60% of the total distance between the drill hole and the free face before finally there is movement from the free face side. This stage can be seen in Figure 8.

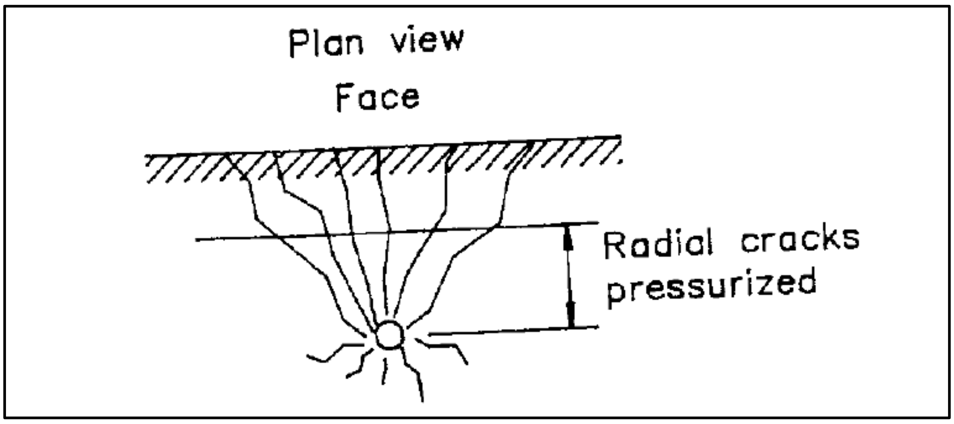

Figure 8. An increased stress on radial cracks (Konya, Calvin & Walter, Edward, 1990)

4. Movement begins to occur on the free face side and flexural failure occurs or failure caused by the inability of an object to withstand the deformation it receives, which is caused by the high pressure gas forcing the rock mass out into a curvature or bend. This stage can be seen in Figure 9.

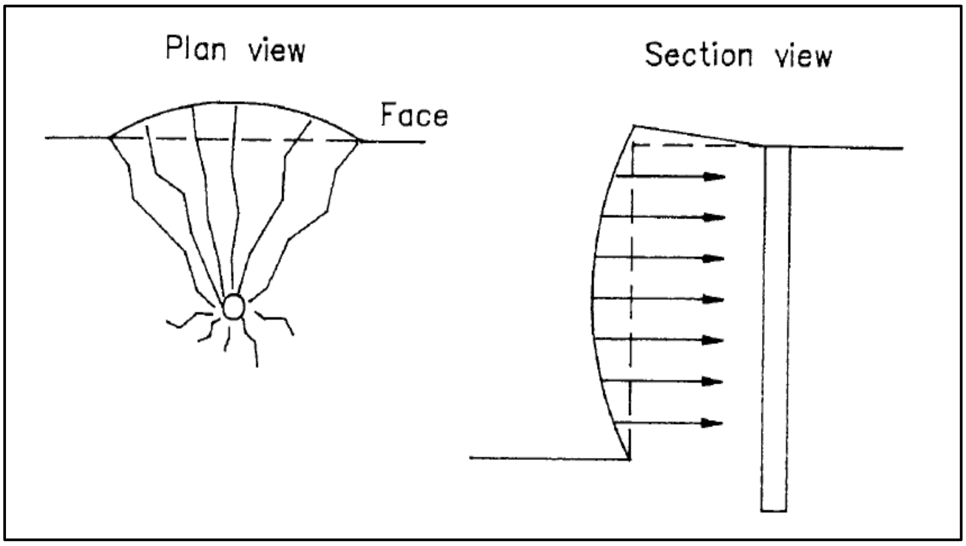

Figure 9. The emergences of flexural failure (Konya, Calvin & Walter, Edward, 1990)

2.2.1 Blasting pattern

A critical component of the rock blasting industry is the blasting pattern, which is related to the sequence of blast times between blast holes in one row and blast holes in the next line, as well as between blast holes with each other. This blasting pattern plays a significant role in determining the intended blasting outcome, and in practice, is frequently associated to the effectiveness and safety of the operation (Hustrulid, 1999). The blasting pattern is established by determining the sequence of explosion durations and the anticipated direction of material collapse under specific circumstances. The utilization of delay periods in blasting systems has several identifiable advantages, including (Permana & Heriyadi, 2019):

1. Reduced vibration

   The vibrations generated during the firing procedure can be mitigated by employing the appropriate delay time. This is crucial as excessive vibration can damage surrounding structures and pose a danger to the environment.

2. Reduced overbreak and flyrock

   A delay time that is effectively managed can prevent overbreak, which is a phenomenon that arises when detonation generates an excessive number of rock fragments. Furthermore, it can reduce the risk of fly rock, which can pose a serious threat to workers and equipment.

3. Reduced airblast

   Establishing a sufficient delay period can assist in the reduction of disruptions brought on by airblasts and the sound (noise) that is created by blasting.

4. Improved fragmentation quality

   By setting the appropriate delay time, the blasting pattern can be designed to produce the appropriate size of rock fragmentation as required, which is an important factor in mining and construction operations.

Based on the direction of collapse, blasting patterns are classified as follows (Gokhale, 2011):

1. V-*Cut* or Chevron

   This pattern is typically executed in parallel drilling patterns, which are either square or rectangular and resemble the letter V. This pattern is also commonly referred to as the Chevron pattern, with the main advantage being that it reduces the possibility of flyrock and can help reduce the size of the rock after blasting. This is due to the fact that boulders colliding with each other will reduce their respective sizes. This pattern can be seen in Figure 10.

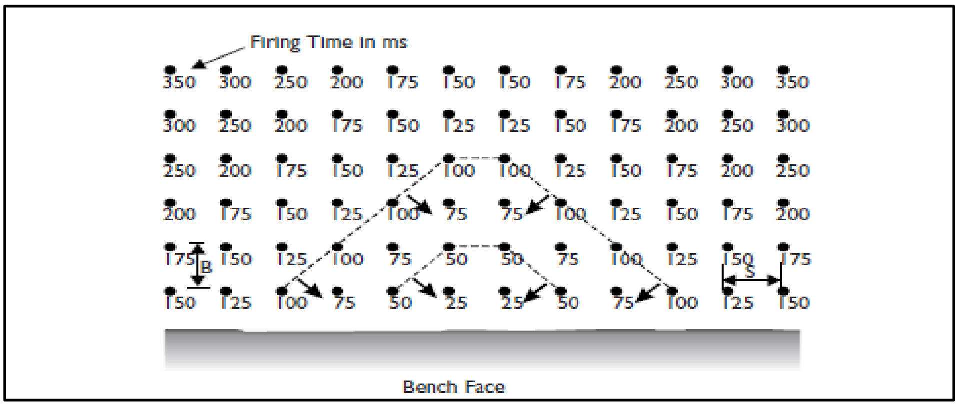

Figure 10. Chevron blasting pattern (Gokhale, 2011).

2. Echelon pattern

   The echelon or corner cut pattern is a blasting pattern that is suitable for use when we have two free fields. This pattern has good fragmentation when compared to other patterns. The distribution of rocks from this pattern is not even but is well distributed at the base of the slope or level, which enables more optimal loading for equipment such as wheel loaders. This pattern can be seen in Figure 11.

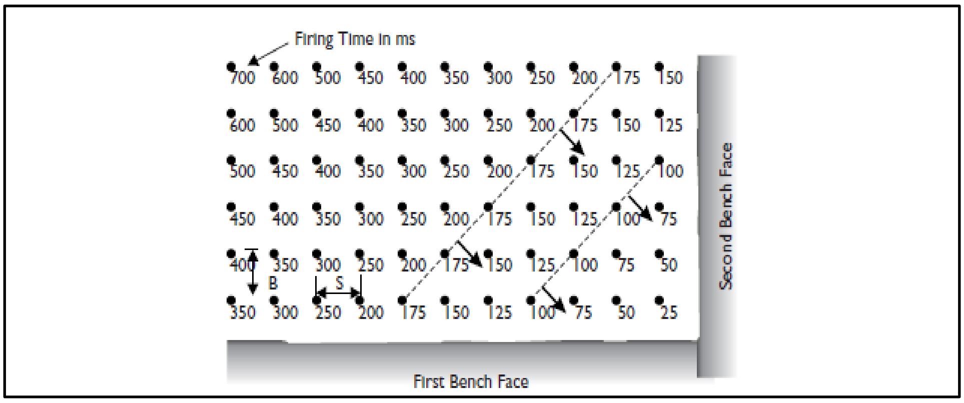

Figure 11. Echelon blasting pattern (Gokhale, 2011)

2.3 Blasting Geometry

Blasting geometry is crucial to ensure that blasting can produce optimal fragmentation and can determine the desired bench or bench shape. The main components in blasting geometry include burden, spacing, stemming, subdrilling, blast hole depth, explosive column length, and bench height. The following is the method for determining blasting geometry according to R.L. Ash (Putri, 2018):

1. Burden (B)

   The distance between drill holes that are confronting the adjacent freeface is referred to as the burden. Burden is determined by considering the density of the explosives used, for example using (Dahana Bulk Emulsion Explosives) DABEX whose density is 1.15 gr/cc. Determining the burden also depends on the diameter of the blast hole and the fragmentation required. If the goal is to achieve good fragmentation, the burden should be reduced, as a smaller burden distance results in better fragmentation. Burden can be determined using the equation 1, 2, 3, and 4.

$$\mathrm{Af1} = \frac{(\mathrm{Dstd})^{\frac{1}{3}}}{D} \qquad (1)$$

$$\mathrm{Af2} = \left(\frac{\mathrm{SG} \times (Ve)^2}{SGstd \times (Vestd)^2}\right)^{\frac{1}{3}} \qquad (2)$$

$$\mathrm{CBC} = \mathrm{SBC} \times Af\,1 \times Af\,2 \qquad (3)$$

$$\mathrm{B} = \frac{Kb \times De}{39.3}\ \mathrm{m} \qquad (4)$$

   Where:

   Af 1 = Rock adjustment factor

   Af 2 = Explosives adjustment factor

   De = Diameter of drill hole (inch).

   Dstd = Standard rock density

   D = Rock density

   CBC = Corrected burden coefficient

   SBC = Standard burden coefficient (30).

   SG = Explosives specific gravity.

   V = Velocity of detonation of the current explosives (ft/s).

   SGstd = Standard specific gravity for explosives (1,20 gr/cc).

Vestd = Standard velocity of detonation for explosives (12000 Ft/s).

2. Spacing

Spacing is the distance between blast holes horizontally from the freeface. Spacing can be determined using the equation 5.

$$S = Ks \times B \tag{5}$$

Where:

Ks = Spacing ratio (1.0-2.0)

B = Burden (m)

3. Stemming

Stemming or plug is the length of an explosion hole that is deliberately left and not filled by explosives to withstand the energy pressure from the explosive material underneath, usually using material from drilling. The first thing needed to calculate stemming is to determine the stemming ratio (Kt), namely the ratio between stemming length and burden. Usually the standard Kt used is 0.70 and this is sufficient to control airblast, flyrock and stress balance. Equation 6 can be utilized to calculate stemming length.

$$T = Kt \times B \tag{6}$$

Where:

T = Stemming (m)

Kt = Stemming ratio

B = Burden (m)

3. Subdrilling (J)

Subdrilling is the length of the blast hole which is deliberately slightly longer than the bench’s heigh to optimize fragmentation and prevent any irregularities on the blasting area's floor. The subdiling length is influenced by the geological structure, bench’s height, and the pitch of the blast hole. The subdrilling length is obtained by determining the subdrilling ratio (Kj) which is above 0.20. In massive rocks, a Kj of 0.3 is usually used. The relationship between Kj and burden is stated as follows:

$$J = Kj \times B \tag{7}$$

Where:

J = Subdrilling (m)

Kj = Subdrilling ratio (0,2 – 0,4)

B = Burden (m)

5. Depth of drill hole (L)

The depth of the blast hole is the sum of the length of the stemming plus the length of the fill column plus the subdrilling. According to R.L. Ash, the depth of the blast hole is based on the hole depth ratio (Kh) which ranges from 1.5-4.0. The relationship between blast hole depth and burden is as follows:

$$H = Kh \times B \tag{8}$$

Where:

H = Depth of drill hole(m)

Kh = Depth of drill hole ratio (1,5 – 4)

B = Burden (m)

6. Collumn charge (CC)

The length of the column charge is the result of subtracting the depth of the blast hole from the stemming length. The length of the input column can be determined using the following equation:

$$PC = H\text{-}T \tag{9}$$

Where:

PC = Depth of collumn charge (m)

H = Depth of drill hole (m)

T = Stemming (m)

2.4 Rock fragmentation

The most critical aspect of blasting results is fragmentation, which is also referred to as "rock that has been scattered”. This is due to the fact that rock fragmentation is the direct impact of blasting results and can influence subsequent activities (Khademian, 2024). The Kuznetsov and Rossin-Ramler methods can be employed to make fragmentation predictions based on explosive specifications and geometric data. The Kuznetsov equation is used to determine the average fragmentation size, while the Rossin-Ramler equation is used to determine the distribution or percentage of material retained in sieve holes of a certain size. The Kuznetsov equation for determining the average fragmentation size is as follows (Kuznetsov, 1973):

$$X = A \times (\frac{V}{Q})^{0.8} \times Q^{0.17} \times (\frac{E}{115})^{-0.63} \tag{10}$$

Where:

X = Average fragmentation size (cm)

A = Rock factor

V = Total volume of blasted rock

Q = Amount of explosives (kg)

E = Relative Weight Strength for the current explosives (ANFO = 100; TNT = 115, Dabex = 77)

A critical element in fragmentation system optimization is the development of practical methods for determining the degree of fragmentation. In general, the term degree of fragmentation refers to the average particle size and the distribution of particles around that mean. Both direct and indirect (photographic) methods are available for determining the fragmentation. There are several weaknesses or errors that are very likely to occur when using the photographic method, starting from low image resolution which will limit the software's ability to read the size of rock fragments, image distortion, and the biggest and most important weakness, namely that this method only pays attention to the surface of the muckpile (Konya, Calvin & Walter, Edward, 1990).

Muckpiles or piles of rock fragments resulting from blasting have different fragmentation qualities in each area. This is mainly due to various factors that influence the blasting process. One of the main factors is the geometry of the blasted area. Complex or irregular geometries can cause uneven distribution of blast energy. This unevenly distributed energy can result in variable fragmentation in different parts of the muckpile. Apart from that, the presence of a top cap or area that is not affected by stemming also affects the quality of fragmentation. This area tends to have coarser fragmentation compared to other parts. Another factor that plays a role is excessive burden, namely a condition where there is excessive burden on certain parts of the slope. As a result, fragmentation in this area is less than optimal and different from areas that have a lighter load. An illustration of differences in fragmentation quality due to the factors above can be seen in Figure 12 (Konya, Calvin & Walter, Edward, 1990).

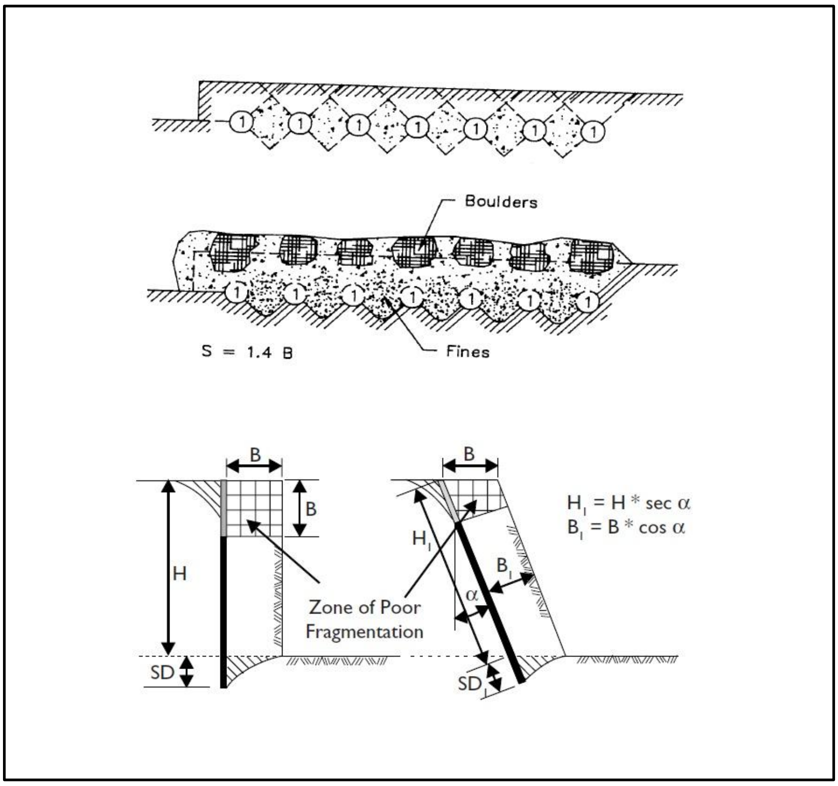

Figure 12. Fragmentation profile through rock muckpile from a surface blast (Gokhale, 2011).

2.4.1 Rock factor

One parameter that is very significant in predicting the size of rock fragmentation is the rock factor, which is obtained through Blastibility Index (BI) analysis. Previous research shows that inputs for blasting capabilities are generally based on estimates from operators and consultants handling blasting tasks. One of the information required by the Kuz-Ram model is the rock factor obtained from the Blastibility Index (BI). BI systems enables engineers to estimate the characteristics of rock masses in the field, thereby providing the opportunity to predict the response of rock masses to fracturing energy (Lilly, 1986).

Blasting performance is significantly influenced by four primary parameters. These parameters include the structural properties of the rock mass, the distance and orientation of weak areas, the density of the material in question and its hardness. Determination of the BI value is obtained from the sum of the weights of five parameters, namely (Hamdi & du Mouza, 2005; Lilly, 1986) :

1. Rock Mass Description (RMD)

   This RMD parameter is used to indicate the quality of the rock mass in the field by observing the rock structure using RQD (Rock Quality Designation). According to Hamdi (2005), RQD data measurements using a scanline can be carried out at 100 cm intervals with the scanline drawn horizontally and vertically against the slope. Determination of rock condition based on the RQD value can be seen in Table 1.

Table 1. Determination of the quality of rock masses (Hamdi & du Mouza, 2005)

Rock condition	RQD (%)
Hard intact	99-100
Hard stratified, or schisetose	95-99
Massive, moderately jointed	85-95
Moderately bloky and seamy	75-85
Very blocky and seamy	30-75
Completely crushed but chemically intact	3-30
Sand and gravel	0-3

The discontinuity frequency obtained from scanline sampling can also be used to determine RQD. For various discontinuity spacing distribution forms, correlations between linear discontinuity frequency and RQD have been established (Sen & Kazi, 1984). For instance, the relationship between linear discontinuity frequency and RQD is illustrated below for a negative exponential distribution of discontinuity spacings (Priest & Hudson, 1976):

$$RQD = 100e^{-0.1\lambda}(0.1\lambda + 1) \quad (11)$$

Where:

λ = Discontinuity frequency

Based on the RQD value and the results of the interpretation of the rock condition, the rock mass description category can be determined, which can be seen in Table 2.

Table 2. Rock mass Description (Lilly, 1986)

Rock mass description (RMD)	Rating
Powder/friable	10
Blocky	20
Totally Massive	50

2. Joint plane spacing (JPS*)*

JPS is the perpendicular distance to joints or weak planes which is measured starting from the first weak plane in sequence towards the last weak plane on a predetermined scanline. Determination of the rating for JPS can be seen in Table 3.

Table 3. Joint Plane Spacing (Lilly, 1986)

Joint Plane Spacing	Rating
Close (<0.1 m)	10
Intermediate (0.1-1 m)	20
Wide (>1 m)	50

3. Joint plane orientation (JPO)

The orientation of the joint plane is important for determining the correct freeface direction or blasting direction. The data needed to determine Joint plane orientation is joint orientation data. Determination of ratings for JPO can be seen in Table 4.

Table 4. Joint Plane Orientation (Lilly, 1986)

Joint Plane Spacing	Rating
Horizontal	10
Dip out of face	20
Strike normal to face	30
Dip into face	40

4. Specific gravity influence (SGI)

Specific gravity influence is a rock property that is related to the specific gravity and porosity of the rock. To measure specific gravity influence, equation 12 is used.

$$SGI = (25 \times SG) - 50 \quad (12)$$

Where:

SGI = Specific Gravity Influence

SG = Specific Gravity

5. Hardness

   Rock hardness is an critical component of rock weighting as it can have a direct impact on the boulders in the field, which are classified as soft, medium, hard, and very hard. Rock hardness can be determined by laboratory testing to test the mechanical properties of the rock. These mechanical properties are related to the uniaxial compressive strength and hardness of the rock. This relationship can be seen in equation 13.

$$\text{Hardness (H)} = 1.36 \times \ln(UCS) - 0.84 \quad (13)$$

Where:

UCS = Unconfined compressive strength.

After obtaining the five parameters from the blastibility index, the BI value can be obtained using equation 14.

$$BI = 0.5\,(RMD + JPO + JPS + SGI + H) \quad (14)$$

Where:

BI = Blastibility index

RMD = Rock mass description

JPO = Joint plane orientation

SGI = Specific gravity influence

H = Hardness (Mohs).

After obtaining the BI value, the rock factor (A) can be determined by utilizing equations 15.

$$\text{Rock factor} = 0.12\ x\ \text{blastibility index} \quad (15)$$

The size distribution of rock can be obtained by utlizing the Rossin-Ramler (1933) equation or equations 16 (Yilmaz, 2023):

$$Xc = \frac{X}{0.693^{1/n}} \quad (16)$$

$$Rx = e^{-\left(\frac{X}{Xc}\right)^{n}}$$

Where:

Rx = Percentage of retained rock mass with size X (cm)

Xc = Rock size characteristics (cm)

X = Sieve size(cm)

n = Uniformity index

Meanwhile, determining the amount of n (uniformity index) is obtained using the following equation (Cunningham, 1987):

$$n = (2.2 - 1.4\frac{B}{De}) \times (1 - \frac{W}{B}) \times (1 + \frac{A-1}{2}) \times (\frac{PC}{L}) \quad (17)$$

Where:

B = Burden

d = Diameter of drill hole (mm)

W = Standard deviation of drilling accuracy (m)

L = Collumn charge (m)

H = Bench height (m)

C (n) = Correction factor

The n value determines the level of uniformity of the size distribution of fragmentation resulting from blasting. The n value usually ranges from 0.8 to 2.2, where the greater the n value, the more uniform the fragmentation size, conversely, a low n value indicates a non-uniform fragmentation size (Suparno et al., 2022).

2.5 Principals of Blasting Economy and Recovery

Blasting economics is method of assessing the efficiency of blasting costs including other activities related to blasting, blasting costs as one of the cost components for the company. The blasting costs include the cost of explosives, the cost of blasting equipment and supplies. In blasting economic analysis, companies conduct a thorough evaluation of their expenses related to blasting. This includes monitoring and controlling the cost of explosives used, investing in efficient blasting equipment, as well as managing human resources involved in the blasting process. By understanding and optimizing these cost components, companies can increase their operational efficiency and optimize the results of their blasting activities (Agnesty et al., 2018)

Cost is a loss of economic resources, quantified in monetary units, in order to acquire products or services that are anticipated to generate profits or benefits in the present or future. From this opinion, it can be concluded that costs are a sacrifice of economic resources to achieve certain goals that are beneficial now or in the future. The amount of costs that need to be incurred to produce one unit of product is called the cost of production (HPP). There are two methods for determining HPP, namely (Purwanto, 2020):

1. Full costing

   Full costing is a method of determining production costs which consist of raw material costs, direct labor costs, and factory overhead costs, both variable and fixed. The full costing calculation is performed by accumulating all cost components without regard for whether or not the products have been sold.

2. Variable costing

   Variable costing is a production cost determination method that exclusively considers production costs that are variable, including basic material costs, direct labor costs, and variable factory administrative costs. In variable costing there is the term period cost which refers to fixed costs that are used even though the product has not been sold

Recovery Blasting is one of the parameters that determines the success of blasting results. A blasting activity is said to be successful when the planned volume of material matches the volume if the material is transported. The blasting recovery value is in the form of a percentage (%) which shows the effectiveness of the rock exposed in a blasting activity. Blasting recovery can be calculated using the following equation 17 (Monica & Yulhendra, 2021):

$$\text{Recovery} = \frac{\text{Volume of transported rock}}{\text{Volume of blasted rock}} \times 100\% \tag{17}$$