

DOKUZ EYLUL UNIVERSITY
GRADUATE SCHOOL OF NATURAL AND APPLIED SCIENCES

**INVESTIGATION OF DECKING SYSTEM AND
APPLICATION OF WATER-DECK IN PRODUCTION
BLASTING AT OPEN CAST MINES**

by
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August, 2007
İZMİR

INVESTIGATION OF AIR-DECKING SYSTEM AND APPLICATION OF WATER-DECK IN PRODUCTION BLASTING AT OPEN CAST MINES

**A Thesis Submitted to the
Graduate School of Natural and Applied Sciences of Dokuz Eylul University
In Partial Fulfillment of the Requirements for the Degree of Master of Science in
Mining Engineering, Mining Operation Program**

**by
Oktay ERTEN**

**August, 2007
İZMİR**

M.Sc THESIS EXAMINATION RESULT FORM

We have read the thesis entitled **“INVESTIGATION OF DECKING SYSTEM AND APPLICATION OF WATER-DECK IN PRODUCTION BLASTING AT OPEN CAST MINES”** completed by **OKTAY ERTEN** under supervision of **ASSOCIATE PROFESSOR GURCAN KONAK** and we certify that in our opinion it is fully adequate, in scope and in quality, as a thesis for the degree of Master of Science.

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Oktaý ERTEN

INVESTIGATION OF DECKING SYSTEM AND APPLICATION OF WATER-DECK IN PRODUCTION BLASTING AT OPEN CAST MINES

ABSTRACT

Excavating the required raw material from an open cast mine is the process which is costly and creates bad environmental effects. Drilling and blasting is the most proper technique of excavating rock in order to minimize the costs. When the cost ratio of drilling and blasting in over all costs is concerned, it is understood that approximately 30% of the total costs is caused by drilling and blasting. Increasing demand for the raw material and decreasing the reserves require this operations to be conducted extremely economically. Additionally, blast induced environmental problems are also in question. These parameters should not give any damage to the residential areas in the vicinity of the mining sites. Especially, the major problems that the mining companies encounter is the blast induced ground vibrations and that these ground vibrations cause visible damage in the residents situated in the vicinity of the mine sites. As far as the world mining is concerned, the blasting operations conducted by conventional techniques are on the decrease. Moreover, the blasting operations have recently been based on the location of the mine and site geometry. Air Deck in other words creating an air volume within a blast hole, have recently been applied in coal open cast mines, is the example of such special operations. In the scope of this thesis, the Air Deck applications conducted at open cast mine were investigated and Water Deck which is a new method of decking was applied in production blasting at Cimentas limestone quarry. Moreover, a set of site survey was carried out in order to determine the safe charge amount for blasting at so called limestone quarry

Key words: Water deck, Air deck, Blasting, Open cast mine

DECKING SİSTEMİNİN ARAŞTIRILMASI VE WATER DECK'İN AÇIK İŞLETMELERDE ÜRETİM PATLATMASINDA UYGULANMASI

ÖZ

Açık işletmelerde doğadan gerekli hammaddenin kazanılması oldukça maliyetli ve olumsuz çevresel etkiler yaratan bir işlemdir. Maliyetin minimizasyonu için kullanılan en uygun kayaç kazma yöntemi delme patlatmadır. Toplam madencilik maliyetinde delme patlatma maliyetine bakıldığında yaklaşık %30'luk bir orana sahip olduğu görülür. Artan hammadde talebi ve rezervlerin azalması bu operasyonların son derece ekonomik düzeyde yapılmasını gerektirir. Ayrıca patlatma kaynaklı olumsuz çevresel parametrelerde söz konusudur. Bu parametrelerin çevre yerleşim birimlerine zarar vermemesi gerekmektedir. Özellikle yerleşim yerlerine yakın açık maden işletmelerinin karşılaştığı temel sorunlardan birisi patlatma kaynaklı yersarsıntılar ve bu sarsıntıların çevre yerleşim yerlerinde görünür hasarlara neden olmasıdır. Dünya madenciliğine baktığımızda, konvansiyonel patlatma yöntemlerinin azaldığı ve lokasyon ve ocak düzenine göre uygulanan patlatma operasyonlarının gerçekleştirildiği görülür. Son yıllarda özellikle kömür ocaklarında uygulanan Air-Deck bir başka deyişle patlayıcı deliği içerisinde hava hacmi oluşturulması söz konusu özel uygulamalara bir örnektir. Bu tez çalışmasında açık işletmelerde kullanılan Air-Deck uygulamasına yer verilmiş ve yeni bir uygulama olan Water Deck uygulaması Cimentas kalker ocağında yapılmıştır. Bunun yanında söz konusu ocakta güvenli patlayıcı miktarının belirlenmesine yönelik bir dizi saha çalışması gerçekleştirilmiştir.

Anahtar kelimeler: Water deck, Air deck, Patlatma, Açık işletme

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CHAPTER ONE

INTRODUCTION

1.1 Introduction to Topic

Drilling and blasting techniques have been the most common and economical technique of rock excavation and displacement in mining as well as in civil construction works. Drilling and blasting operations are of more importance in terms of production blasting in open cast mines. As far as the open cast mining operations are concerned, these operations make up about %30 of total cost of open cast mining. Therefore, the main objective of blasting operation conducted at an open cast mine is to fragment the rock at minimum cost. The fragmentation size of the muck pile is also of crucial importance for other mining operations such as downstream process and milling. Because of the increasing population of the world and production needs of other industries, mining companies have to increase their production capacity to meet the raw and ultimate material demanded by the industries in question. To be able to meet the increasing demands, there have been developments in explosive materials and blasting techniques from the early days of blasting with black powder.

Many investigators have carried out a number of researches in order for the mining companies to get the maximum profits from the drilling and blasting operations. These so-called researches are mostly related to developments in explosive materials, blasting design (computer aided blasting design methods), drill hole pattern (spacing, burden and drill hole diameter relations) and determining the surface of the geological structure (measuring while drilling methods). All these major developments in the drilling and blasting operations have made these operations more reliable and less expensive. However, as it has been mentioned above, the increasing population has caused more residential settlements to have been built in the vicinity of the mining areas particularly open cast mines. Therefore, the proximity between the residential settlements and the open cast mines has been of

critical importance in terms of the damage induced by blasting to the mentioned residential structures.

In this case, the mining companies are supposed to reduce this above mentioned damage as much as possible to prevent the undesirable effect of blasting operations on the residential structures. A number of new alternative blasting methods such as the use of chemical based explosives, non-electrical initiation system and delay timing, the use of explosives with low specific gravity and the controlled blasting techniques such as pre-splitting, smooth blasting, cushion blasting, rip rap blasting, decoupling, decking have been proposed in order to decrease the environmental effect of the blasting operations.

From the developments explained above, the technique called “decking” has been commonly used in production blasting for some time to diminish the intensity of blasting effect on the environment. Creating any kind of volume (such as solid, liquid, air) within the drill hole is called decking. The decking system that has been commonly used in production blasting for years is air-decking. In this technique, an air gap is introduced in an explosive column in the drill hole as a means of optimizing the rock breakage for a given charge length. The most crucial reason for air decking to be used in production blasting is the fact that this technique enables increased duration of shock wave action on the surrounding rock mass through repeated oscillations of the shock wave within an air gap. This effect results in increased degree of fracturing in the rock mass.

1.2 Objectives and Scopes

The main objective and scope of this thesis is to investigate the commonly used decking system as an alternative blasting method, and the new decking system, which uses the same principle as air decking, will be introduced. The new decking system contains the water column introduced in the middle of the explosive column to be detonated. Some expectations and the parameters to be evaluated in the scope of the field experiments are given as follows;

- The middle column water deck will be introduced in the explosive column
- Enhanced fragmentation because of the increased shock duration is one of the expectations of this field study
- To be able to make a reduction in the vibration and noise intensity induced by blasting is also one of the expectations of the so-called field study
- 15% to 35% savings in total explosive load.

Because of the fact that such technique is expected to make a reduction in the vibration intensity generated by blasting, the limestone quarry located in Izmir province in Türkiye has been chosen for the field experiments in question. The so called limestone quarry has been surrounded by the settlements for many years, and this situation made the blasting operation difficult to conduct. It was considered that the most convenient place to choose is this quarry to carry out the experiments at issue. All the experiments conducted and related analyses will be explained in the succeeding chapters.

1.3 Outline of Thesis

The remainder of this thesis is arranged into six chapters as follows:

Chapter 2 introduces the general blasting theories and fundamental blast designs. The concept of bore hole pressure and dynamic fracturing mechanism are also presented.

Chapter 3 highlights the process of Air-Decking introduced within a bore hole. The mechanism of Air-Deck is also dealt with. Moreover, it states the idea of Water-Decking based on a laboratory studies

Chapter 4 explains the general view of the environmental effects of blasting. Ground vibration waves and monitoring is also taken into consideration.

Chapter 5 introduces the image analyses technique used in determining the fragmentation size of the muckpile

Chapter 6 case study on Cimentas Quarry

Chapter 7 draws conclusions of the conducted studies and provides some recommendations for future academic research

1.4 Significance and Relevance to Mining Industry

Drilling and blasting operations are inevitable operations for most mining activities conducted both in underground and at open cast mines. However, because of the proximity of the residential areas to the mining areas has made these so-called operations even more difficult than ever to conduct. As far as the effects of blasting are concerned, the effects at issue cannot be eliminated entirely, but they can be diminished as much as possible. The technique called Air-Decking is one of the common techniques used for diminishing the effects of blasting on residential areas. In this technique, one or more air gab is introduced within the bore hole, and this increases the duration of the shock wave action. As a result of this, good fragmentation can be achieved despite the fact that less explosive amount is used. In this thesis, different application is considered. This is based on the similar concept of Air Decking; however, the Water Deck is considered to be used within the bore hole other than Air Deck. As known well, liquids can not be pressed under the pressure applied; therefore, liquids transmit the all effect of pressure to all directions. When the water deck is introduced within the bore hole, it will be pressed by the energy of detonated explosive amount by either side of the Water Deck. In other words good fragmentation can be achieved. When the Water Deck is introduced within the bore hole, the following outputs are expected in terms of the mining activities carried out;

- Reduction in the dust amount produced by blasting
- Reduction in the intensity of ground vibration
- Reduction in the explosive amount used within a bore hole

CHAPTER TWO

BLASTING THEORY AND FUNDAMENTALS OF BLAST DESIGN

2.1 Introduction

Blasting is the first step of the excavation process. Therefore, it is of crucial importance in terms of other mining activities such as downstream and milling process. The success of blasting is directly related to how well the shock energy generated by blasting is confined within a blast hole. Moreover, the explosive and rock type can be effective on blasting. Another important parameter related to blasting is the blast design. The proper blast design should be applied the blast site in order to minimize the effects arising from blasting. For example, blast hole diameter, blast hole length, geological structure, burden, spacing are some of the features that need to be taken into consideration while the blast is designed. The proper and well designed blast can maximize the profit. Therefore, blast design and blast theory should be figured out perfectly. Chapter 2 will introduce the blasting theory and the fundamentals of blast design.

2.2 Blast Mechanism

Quarries remove overburden from the mines to produce the mineral being mined for use in the industry. Holes are drilled in the rock to be broken and explosives charges are placed in the holes. The top portions of the holes are filled with inert materials (usually drill cuttings) for stemming purpose. When the explosive charge is detonated in a borehole, a great deal of energy in the form of gas is liberated in a very short period of time. The rapid expansion of this gas exerts a pressure energy on the borehole walls that propagates away from the boreholes. In the neighborhood of the boreholes, the stress induced by this energy exceeds the elastic limit of the rock, and fracture occurs, thereby filling the new and existing cracks in the rock. In a properly designed blast most of the energy is used in breaking the rock and some part is converted into ground vibrations. But in case of improper design, poor breakage

with most of the energy is being converted into vibrations, which is the main source of annoyance to the inhabitants residing nearby the mine workings. (Basu & Sen, 2005)

2.3 Rock Fracturing and The Concepts of Blast Hole Pressure

The study of fracture initiation and propagation in rocks due to explosive energy is of engineering importance. This knowledge has many practical applications for mining engineering, civil engineering and petroleum engineering. The knowledge in mining engineering is required for efficient rock breakage, limiting damage into host rock mass and effective destressing of a rockburst prone area in an underground mine. A system engineering approach is required for better understanding of phenomena governing dynamic fracture initiation and propagation study. Figure 2.1 illustrates three main systems involved in this approach: rock, explosive and boundary conditions. Each system subdivided into sub-systems, which in turn are broken down into sub-subsystems. While physically independent, these systems do interact with each other. A number of rock-explosive energy interaction studies have been reported in the literature. For example, the propagation of the shock wave energy (a sub-subsystem of the explosive energy system) depends on the spatial distribution of geological anomalies and the attenuation characteristics of the rock (both are sub-subsystems of the rock system) as well as the resonance effect (a subsystem of the boundary conditions system) created by the reflected waves from a boundary. (Saharan, Mitri & Jethwa, 2006)

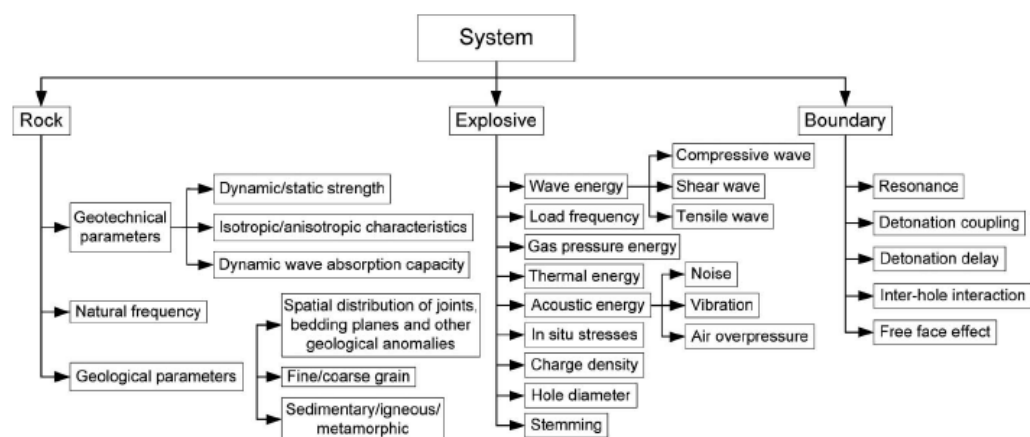


Figure 2.1 Engineering systems for rock blasting (Saharan, Mitri & Jethwa, 2006)

2.3.1 Rock Breakage Theory

When the explosives are detonated within the blast hole, a few or many cracks are driven from the blast hole into the rock. The most theory of breakage consists of two stages, first the shock wave causes radial cracks to form around the blast hole then the gases penetrate into cracks, widen them and make them longer (Langefors & Kihlström, 1973). Similar to the view of Langefors and Kihlström, Rasheef and Geomans (1992) have studied the energetic process of the fragmentation and proposed a model where the explosive energy is separated into different types of energies. According to their model, the released explosive energy is divided into two parts: the shock wave energy and the gas expansion energy. The different types of energy are sub-forms of these two main processes. Particularly, one part of the shock wave energy is spent in cracking, whereas another portion is consumed in opening-up cracks. (Hamdi, Mouza & Fleurisson, 2001). Another theory presented by Brinkmann (1987) suggests that the back damage is primarily controlled by shock and that the gas penetration is the mechanism controlling breakout of the burden. He did his experimental work using blast hole liners (Sarracino & Brinkmann, 1990). The liners consisted of stainless steel tubes that were placed inside the bore holes. The explosive was charged within the steel tubes, which worked like a chimney and vented the gases. He measured the pressure within the steel tubes. Brinkmann (1987) also mounted accelerometers both near the burden rock surface and behind the holes. He used two different explosives, an emulsion explosive with high VOD and dynamite with low VOD. His conclusions included:

- The emulsion had a faster pressure rise time and a faster pressure drop than the dynamite;
- Gas penetration is the dominant mechanism controlling fragment velocities;
- The emulsion caused more fracturing within the burden;
- Back fracturing from the emulsion is more intensive and extends further than that from the dynamite; and
- Break out is controlled by gas penetration and fragment sizes are governed by shock.

2.3.2 Concepts of Blast Hole Pressure

Blasting is an exceptionally dynamic process in an opaque and often anisotropic medium. This creates intrinsic difficulty in attempting to either understand the mechanisms or predict results with precision. However, engineer need to deliver specific blasting results and must harness whatever tools are available. The simplest tool is the concept of powder factor, which correctly links increased explosive usage with more energetic blasting results-increased fragmentation and movement in particular, but not necessarily back damage. Powder factor does not apply when considering the specific damage originating from individual blast holes, an important question when attempting to design for reduced damage with lower energy in the hole; probably the most common parameter used for such design is “blast hole pressure”, P_B . This, also known as the “explosion” pressure, is defined as the pressure exercised by the explosive after detonation, if the resultant gases are at rest and confined at the same density as the original explosive (Cunningham, 2006). What makes the concept particularly attractive, is its ease of derivation, from the detonation velocity and the explosive density, typically quoted (Cooper, 1996) as

$$P_B = \rho V_d^2 / 8000 \quad (2.1)$$

where P_{CJ} is sonic pressure, GPa; ρ is density, kg/m^3 ; D is the detonation velocity, km/s . This is often quoted with the related equation,

$$P_{CJ} = \rho V_d^2 / 4000 \quad (2.2)$$

The value 4000 in equation (2.2) above derives from

$$P_{CJ} = \rho V_d^2 / (\gamma + 1) \quad (2.3)$$

Assuming that, γ , the polytropic index at the sonic locus, is about 3. Thus if the explosive is ANFO with a density of 800 kg/m^3 and the measured detonation velocity

is 3.5 km/s, an estimation of pressure using equation (2.1) would be $P_B = 800 \times 3.5^2/8000 = 1.225$ GPa.

This pressure is then used as the starting point for calculations which ultimately match explosive pressure to rock strength so as to draw conclusions as to intensity of cracking in the rock. The equation used to estimate applied pressure is

$$P_R = P_b (v_b/v_h)^\gamma \quad (2.4)$$

Where P_R is pressure applied to the rock surface; v_b/v_h is the ratio of volume of explosive to volume of hole, and γ is an assumed value for decoupled charges; for example 1.2 as given by Hustrulid (Hustrulid, 1999), and Chiappetta (Chiappetta, 1990).

Chiappetta (1990) also provides a formula for the maximum spacing of decoupled holes in splitting designs:

$$S = \frac{dx(P_B + T)}{T} \quad (2.5)$$

where S is maximum hole spacing, inches; d is hole diameter, inches; P_B is bore hole pressure, psi; T is tensile strength (Brazilian test), psi.

2.3.3 Dynamic Rock Fracture Mechanism

Explanation of the detonation process can be found in several publications related to blasting engineering. (Bhandari, 1997; Clark, 1987; Fourney, 1993). According to these explanations, the detonation of an explosive charge, confined within a bore hole, initiates a very rapid chemical reaction at a velocity known as the velocity of detonation (VOD or V_d) with values of 2000-7000 m/s. such a chemical reaction is exothermic (i.e. the temperature is in the range from 3000 to 4000°K) and converts the solid mass of explosive into gaseous products. The very rapid conversion of the chemical energy into thermal and gaseous energy develops a detonation pressure (P_d)

with magnitude in the range from 0.5 to 50 GPa. The detonation pressure generates an explosion pressure (P_e) in the reacted portion of the explosive column as shown in Figure (2.2a). The interface between the detonated portion and the un-reacted explosive column, known as the Chapman-Jouguet Plane or CJ Plane (Clark, 1987) travels with the speed of the shock wave front as shown in Figure (2.2a). The actual pressure acting on the boundary of the bore hole, known as the bore hole pressure (P_b), depends not only on amount of gas produced, its temperature and velocity of detonation, but also on the rock mass stiffness properties and its strain-rate dependant strength properties as well as the bore hole volume. As can be seen from Figure (2.2b), the pressure P_B responsible for fracturing the rock, is less than P_d but is considerably higher than the original pressure P_0 .

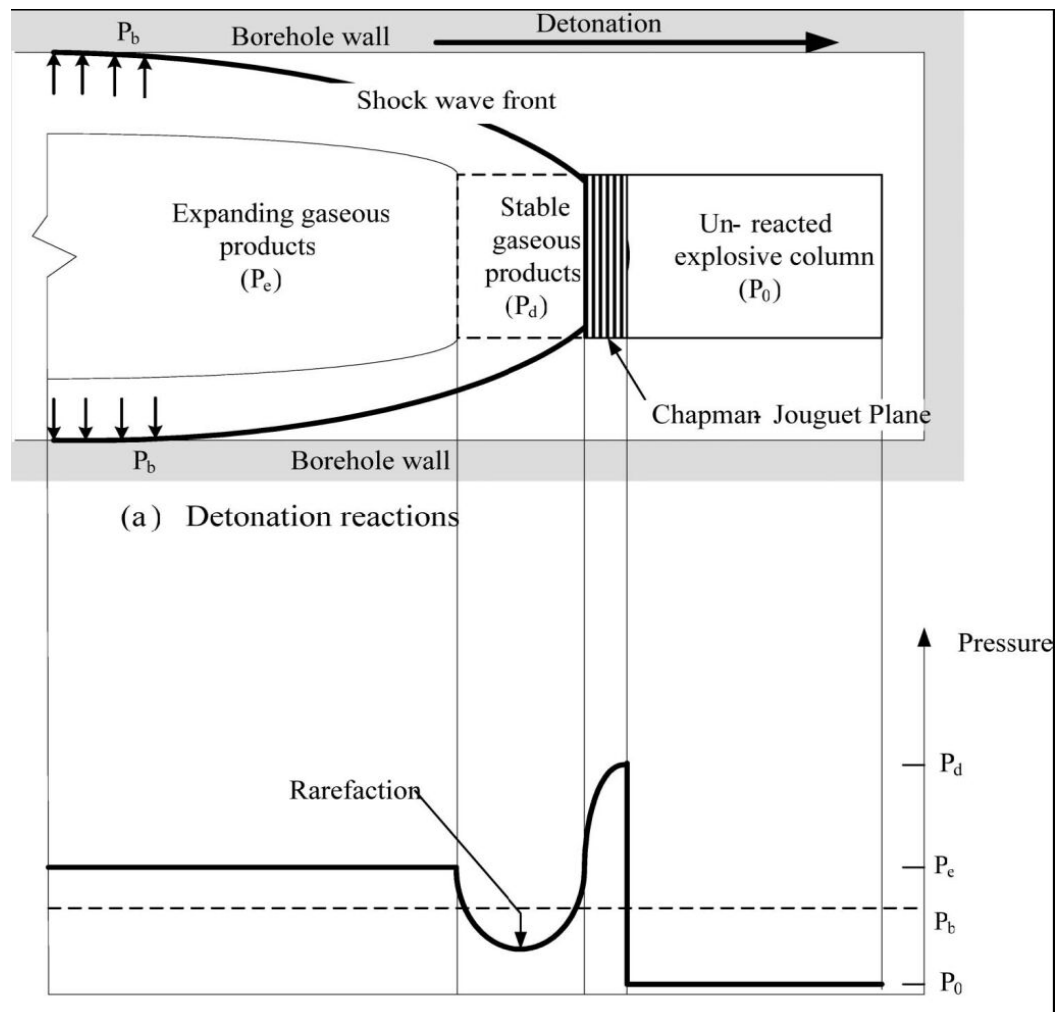


Figure 2.2 Detonation process of an explosive in a bore hole (Saharan, Mitri & Jethwa, 2006)

The following simplified equations are in use for the estimation of P_d , P_e and P_b , which are derived using the equation of state of ideal gases.

$$\text{Detonation pressure, } P_d = \rho V_d^2 / 4 \quad (2.2)$$

$$\text{Explosion pressure, } P_e = P_d / 2 = \rho V_d^2 / 8 \quad (2.1)$$

$$\text{Bore hole pressure, } P_b = P_e (r_c^{2\gamma}) \quad (2.6)$$

Where; ρ =explosive density in kg/m^3 ; V_d =velocity of detonation in m/s ; r_c =coupling ratio=explosive diameter/bore hole diameter; γ =adiabatic exponent=1.5

$$\gamma = \sqrt{(1 + V_d^2 / Q)} \quad (2.7)$$

and Q is the heat of explosion of the explosive in kJ/kg .

One can find contradicting blast hole pressure measurements with regard to time needed to attain the peak-pressure values but the complete duration time for the blast hole pressure-time profile is similar in these contradicting measurements. The peak-pressure time profile falls to a stand-off pressure within a few milliseconds (the stand-off pressure is much less than the static tensile strength of most of the rocks). Many publications cite explosive peak-pressure attainment time in order of several milliseconds (Holloway, Barnholt & Wilson 1986; Schatz, Ziegler & Chritianson 1987) but there are also in-blast hole measurements suggesting that the peak pressure attainment time is in between 20 and 150 microseconds depending on explosive type and confinement (Olsson, Nie, Bergqvist & Ouchterlony, 2001; Daniel, 2003; McHugh & Keough, 1982). The above described action of detonation in rock mass is illustrated in Figure 2.3. As can be seen, the impingement of the transient blast hole pressure on the blast hole boundary generates a primary compressive shock wave into the rock mass. Part of the wave energy is reflected where it meets a free surface or a discontinuity (shown as interface between rock 1 and rock 2 in Figure 2.3). The reflected wave, termed ‘secondary’ wave, may be compressional or tensile in nature depending on the characteristic acoustic impedance difference at the interface of the original and the new face or discontinuity. Those stress waves, through their interaction with existing micro flaws in the rock, are the dominant force in

fragmenting it (Winzer, Anderson & Ritter, 1983). Fracture generation, propagation and coalescence at a point of consideration will continue until superimposed values of the incident and reflected waves attenuate to a value lower than the tensile strength of the rock at that point under stress. These theoretical explanations for dynamic rock fracturing, however, ignore the very characteristic of a rock, namely the strain rate response of the material. It is now proven beyond doubt that rocks exhibit strain rate-dependant response under severe dynamic loading conditions (Prasad, Mohanty & Nemes, 2000). Therefore, it would be more precise to state that the fracture generation, propagation and coalescence at a point of consideration continue until superimposed values of the primary and secondary shock waves attenuate to a value lower than the characteristic strength of the rock, which is the result of the strain rate-dependant response at that point under existing state of stress.

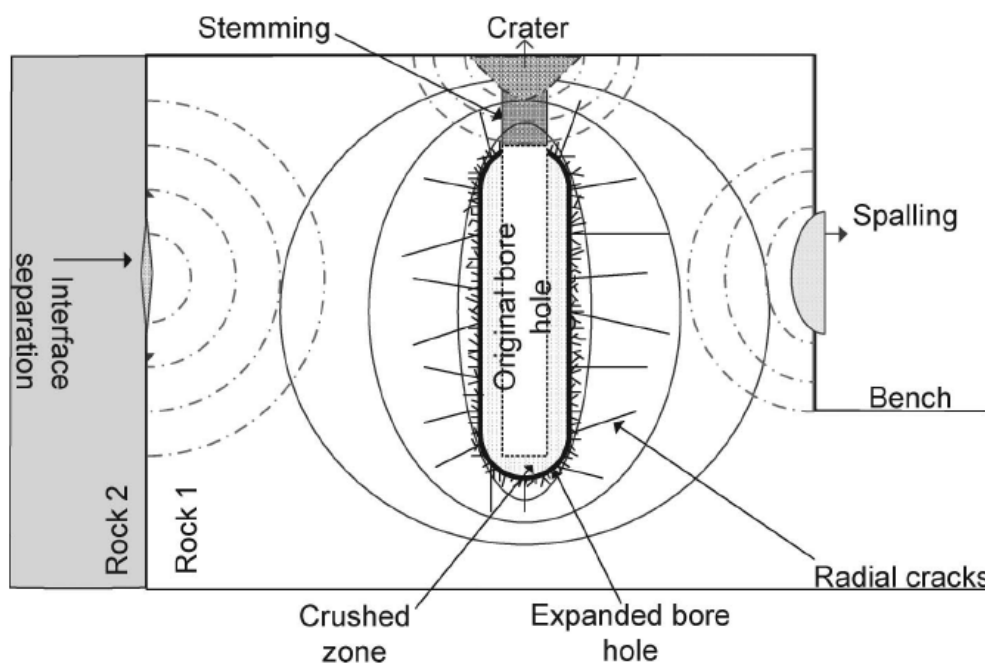


Figure 2.3 Vertical section in a bench blast hole showing the effects of explosive energy on the rock mass (Saharan, Mitri and Jethwa, 2006)

Considerable efforts were made in the last four decades to understand the rock fracturing mechanism. The work done so far has led only to general agreement on development of stress waves and gas pressure by explosive loading, though in the past other theories were also propounded such as flexural rupture and nuclei stress

flaw. Table 2.1 compiles a list of references and the blasting theories that they support for the interpretation of breakage mechanism. As can be seen from Table 2.1, rock breakage mechanism is attributed to one or more of the following factors: tensile reflected waves, compressional stress waves, gas pressure, flexural rupture and nuclei stress flow. Controversies still exist to precisely define the roles and mechanism of fracturing either by stress waves or by gas pressure. (Daehnke, Rossmannith & Napier, 1997), who are propounders of gas pressure breakage theory distinguished between the shock wave velocity and the crack propagation speed. The shock wave velocity is three times the crack propagation speed as demonstrated in (Edgeston & Barstow 1941). They, however, ignored the interaction between primary and secondary waves emanated from pre-existing flaws in the rock mass. Supporters of stress wave theories believe that the large number of cracks developed around the blast hole is due to stress waves since a quasi-static pressure could only cause a couple of cracks to develop (McHugh, 1983).

Table 2.1 blasting theories and their breakage mechanisms (Saharan, Mitri & Jethwa, 2006)

Reference	Breakage Mechanism				
	Tensile reflected waves	Compressional stress waves	Gas pressure	Flexural rupture	Nuclei stress flow
Obert & Duval 1949	X				
Hino 1956	X				
Duval & Atchison 1957	X				
Rinehart 1958	X				
Langefors & Kihlstrom 1963		X	X		
Starfield 1966	X				
Porter & Fairhurst 1970		X	X		
Persson et. al. 1970		X			
Kutter & Fairhurst 1971		X	X		
Field & Ladegaard-Pederson 1971		X	X		
Jhansson and Persson 1972	X		X		
Lang and Favreau 1972	X	X	X		X
Ash 1973			X	X	
Bhandari & Vutukuri 1974	X	X	X		
Hagan & Just 1974		X			
Barker et. al. 1978					
Winzer et. al 1983					X
Margolin & Adams et. al. 1983					X
McHugh 1983					X
Brinkman 1987		X			X
Daehnke, Napier & Rossmannith 1977			X		
Nie and Olsson 2000		X			

One can make the following observations barring the controversies of respective roles of stress wave and gas pressure in rock fracturing:

- (a) Reduction in unwanted damage to host rock by explosive action can be achieved by a 'hydrofrac' action rather than a dynamic action. The acronym hydrofrac stands for hydraulic fracturing in which the peak borehole pressure of lower magnitude pressurizes the rock for a much longer duration. The energy applied is mostly used for rock fracturing only in this process. Such hydrofrac action can be achieved by non-ideal detonation of a low energy explosive and a prolonged action of the reduced pressure level by de-coupling of a high energy explosive (Barkley, Lee & Rodgers, 2001).
- (b) Reduction in the crushing zone and the intense fracturing zone can be achieved by air-decking techniques (Bussey & Borg, 1989).
- (c) Insertion of a viscous medium, such as water or sand, or a material with different impedance from the rock also helps in reducing the crushing zone as well as reducing the number of radial fractures. Bhandari & Rathore (2002) suggest the use of cardboard liners to reduce the damage to the host rock.
- (d) The rock structure plays an important role in fracturing of rock. Granular minerals produce more fines than the minerals with elongated fibers.
- (e) The fractures propagation path needs to be specified in the borehole should directional fracture growth be needed. The notched borehole technique and shaped charges can be employed for such effects.

2.3.3.1 Ideal Detonation

Ideal detonation is steady state and one-dimensional with complete reaction between the shock front and sonic locus, and no dependence on time, dimension or external physical parameters. CJ detonation treats only the boundary states of the

unshocked explosive and the sonic locus, while ZND detonation treats also the detonation driving zone between. In an ideal detonation, the entire explosive reacts in the detonation driving zone (DDZ) between the shock front and the sonic (CJ) plane. Figure 2.4 aligns a physical representation of this in an ideal container, with the classical CJ pressure-volume curve and tangential Rayleigh line defining the CJ locus. The DDZ is shown as long in order to clarify what happens with non-ideal detonation. (Cunningham, 2006)

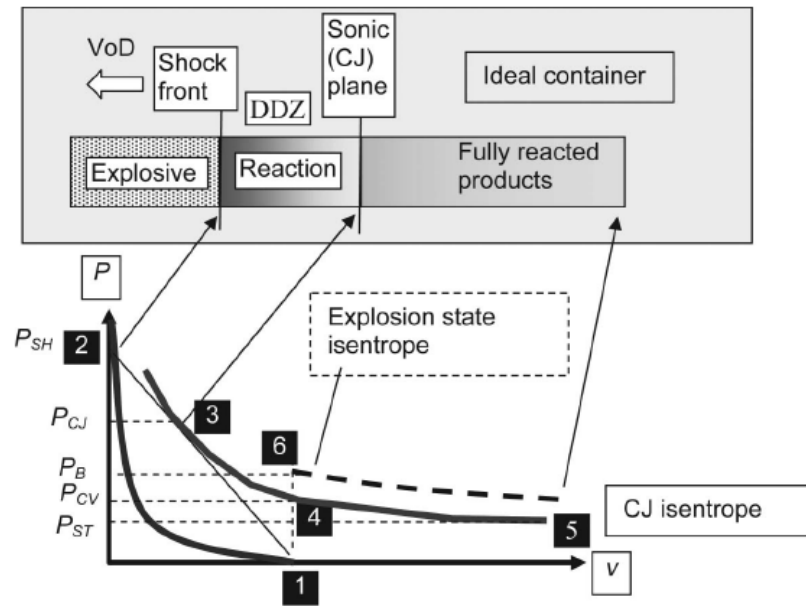


Figure 2.4 Ideal detonation. Top: Physical process. Bottom: pressure-volume curves (Cunningham, 2006)

In practice all detonation is non-ideal, but some situations are so close to ideal that it is a good approximation. The unreacted Hugoniot is the locus for points 1 and 2, representing the pressure and specific volume of the explosive as it goes from the unshocked to the shocked state. Points 3 to 5 lie on the CJ isentrope defining the energy states of the product gasses as in expansion from the CJ state. The well known conservation of energy equation applies (Cooper, 1996);

$$E + Pv + 0.5u^2 = C - W \quad (2.10)$$

Where; E is internal energy, MJ/kg; P is pressure, GPa; v is specific volume, $\text{m}^3/\text{kg} \times 1000$; u is particle velocity, km/s; C is the constant within the isentrope; W is work on environment. Because of the ideal container, no external work is done and the pressure gradient is a result of the velocity gradient: temperature, particle velocity and pressure adjust until a stable pressure is achieved, at point 6.

The key things to notice are that:

- If the explosive components react slowly but ideally in ideal confinement, then the pressure in the container would be the explosion pressure, also known as the borehole pressure, P_B .
- With Ideal detonation in ideal confinement, the container will also settle at pressure P_B , once dynamic interaction of the reacted fluids is complete.
- The pressure gradient after the sonic locus is caused by the dynamic balance between pressure, velocity and heat content, regulated by conservation of energy, mass and momentum. The kinetic energy of the detonation fluids plays a major role in creating a transient pressure wave, even in ideal confinement. The time of transition of the DDZ is of the order of a few microseconds, and the decline in pressure behind the sonic locus is initially rapid but plateaus as it approaches the stagnation condition.
- The ideal sonic or CJ pressure P_{CJ} is about half the shock pressure, P_S , and the borehole pressure P_B is about half P_{CJ} , as reflected in equations (2.1) and (2.2). The pressure difference is created by the particle velocity, resulting from the densification of the DDZ by the shock wave. In Figure 1, particle velocity is zero at the starting end of the container (point 5), and pressure is about $P_{CJ}/3$. The instantaneous pressure at original density is P_{CV} , also below P_B . Equalization of the pressure pulses within the vessel would then rearrange the pressure to P_B , about $P_{CJ}/2$.
- By definition, the sonic (CJ) locus isolates the DDZ from any later physical processes. In non-ideal detonation, chemical energy released after the DDZ detracts from the VoD, but not from the energy of the explosive.

2.3.3.2 Non-Ideal Detonation

In a truly ideal container, non-ideal detonation would not be possible, as non-ideality is caused by the DDZ losing radial energy, which is prohibited by ideal confinement. Thus, in order to illustrate the point that needs to be made, we define Ideal confinement as “quasi ideal”, in that it allows partial reaction within the DDZ without the confinement yielding. The conventional pressure calculation, equation (2.1), is usually undertaken with measured VoD's that are less than the ideal value, signifying that reaction is incomplete at the sonic locus, proving non-ideal detonation. Unless quenched, reaction will continue after the sonic plane and the gas build up must cause a slower pressure drop than is the case for ideal detonation, in which reaction is complete at the sonic locus. In addition, the incomplete DDZ reaction produces a lower pressure at the sonic locus, so the reduced pressure drop feeds into a lower initial pressure. Thus in non-ideal detonation, PCJ will be lower, but as the explosive reacts after the DDZ, pressure drops less, as illustrated in Figure 2.5, for a point in the middle of an ideal container. For clarity, the stabilization of pressure as particle velocity comes to rest after detonation is shown as a smooth recovery of pressure. It indicates that, because of energy conservation, both ideal and non-ideal detonation should result in the same ultimate borehole pressure.

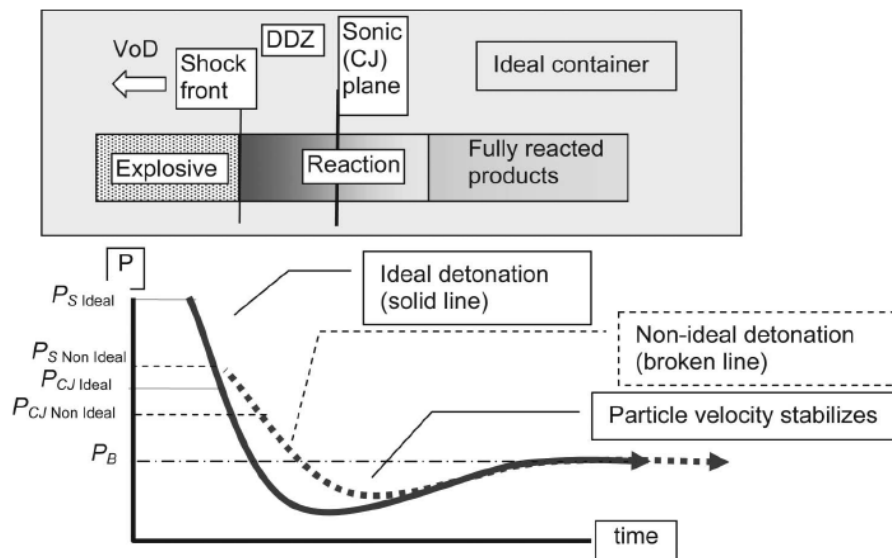


Figure 2.5 Non-ideal detonation has lower VoD but same and of blast hole pressure concept (Cunningham, 2006)

Higher ideality of reaction will result in a higher sonic pressure but deeper rarefaction. This process cannot be represented on a conventional $P - v$ chart, since rates of reaction are involved. The usual equations used to derive PCJ and PB (equations (2.1) and (2.2)), depend on reaction being complete at the sonic plane, so it is clearly wrong to expect them to be valid for non-ideal VoDs. After all, even if there were no detonation and the formulation reacted slowly in Ideal confinement, the pressure would reach the ideal value of PB. Clearly equation (2.2) expects lower pressure for a lower VoD, and in equation (2.3) this translates into holes being spaced closer together, which must be questioned in the light of neither energy nor borehole pressure being related to VoD. If the above reasoning is correct, then there should be little or no difference in work, provided the energy is similar. However, the conclusion drawn from conventional use of the equations leads to an expectation that higher VoD explosives will do more work than lower VoD explosives. This represents a vital change in understanding which needs to be supported.

2.3.4 Significance of Stress Wave and Gas Pressurization Stages of Blasting Process

The long-term controversy is whether the action of stress wave or that of the high-pressure gases is mainly responsible for the rock breakage. The earliest attempts to describe the fragmentation process are those of Duval & Atchison (1957), Hino (1956), and Starfield (1966). These authors emphasized the role of stress wave in fragmentation. They postulated that when the explosion-induced stress waves reach the free face or open discontinuity surfaces, there is a strong reflection wave, as a tensile pulse, back into the rock mass. Then, considering that the tensile strength of rock is very low, the reflected wave carries regions of the rock into tension leading to spalling parallel to the free faces. Field observations of real blasts support that reflection of stress waves from free faces and as a result spalling types of rock breakage alone cannot be the governing rock mechanism. Notwithstanding, this theory can be used to describe the damage occurring in the vicinity of the free faces. Winzer & Ritter (1980), Fourny et al. (1983), and Wilson (1987) are other researchers who further acknowledged the significance of stress waves in the

fragmentation process. According to these authors, fracturing of the rock is mainly due to the passage of stress waves through the rock mass and their multiple reflections from free faces and discontinuity surfaces. Fourney et al. (1983) used high-speed photography in conjunction with the photoelastic technique to observe the propagation of stress wave and cracks. They concluded that the stress waves are of extreme importance in conditioning the rock mass by way of initiating cracks at a multitude of locations. After this preconditioning by the stress wave, rock fracture and fragmentation continues as the gases fill up the newly created fractures. Wilson (1987), in his experimental and numerical studies, indicated that it is unlikely that any new cracks would be initiated during the later period of gas pressurization, assuming that the media is homogeneous, isotropic, and elastic. In addition, even though the gas pressurization phase acts over a much longer period of time than stress wave, the tensile magnitude in the stress wave was found to be higher than that created by the later gas pressure.

The work of this second group of researchers demonstrates that the initiation of compressive, shear, and tensile stress waves with flaws and discontinuities is the governing fragmentation mechanism as opposed to spalling theory. Moreover, it is believed that the role of gas pressure is to only move the fragmented rock mass. There is a third group of researchers who believe in the roles of both the stress wave action and the expansion of detonation products in fragmentation. Kutter and Fairhurst (1971) hinted at the important role of gas pressurization in blasting, but it was also shown that the gas action is effective only if the cavity and free faces are pre-conditioned by the stress wave. They acknowledged that the amount of useful energy contained in the wave is a small fraction of the total explosive energy. They referred to the experimental findings of Fogelson et al. (1959) and Nicholls et al. (1962) that revealed that stress wave carries up to 10-18% of blast energy in granite gneiss (brittle rocks) and 2-4% of total energy in salt rock (soft rocks). The remaining part of explosive energy is contained to a large extent in the expanding high-temperature and high-pressure gases. Hagan (1974) suggested that the gas pressurization is the governing mechanism in fragmentation and in order to minimize the damage zone around the blasthole and avoid energy losses, pressure should be

kept below a certain limit. This idea was also supported by Persson et al. (1969) and Warppinski et al. (1979). In a series of experiments, McHugh (1983) used steel liners to assess the relative importance of internal gas pressurization and stress-wave-induced circumferential tensions for extending radial cracks originating at the blasthole. He used cylindrical Plexiglas samples to simulate the field conditions. His experimental findings indicated that gas pressurization could extend the crack length by a factor of 5 to 10 (when 6.9 MPa confining pressure was applied to the cylindrical samples) compared with the stress-wave-induced tensile stresses acting alone. A summary of his experimental findings can be seen from Figure 2.6.

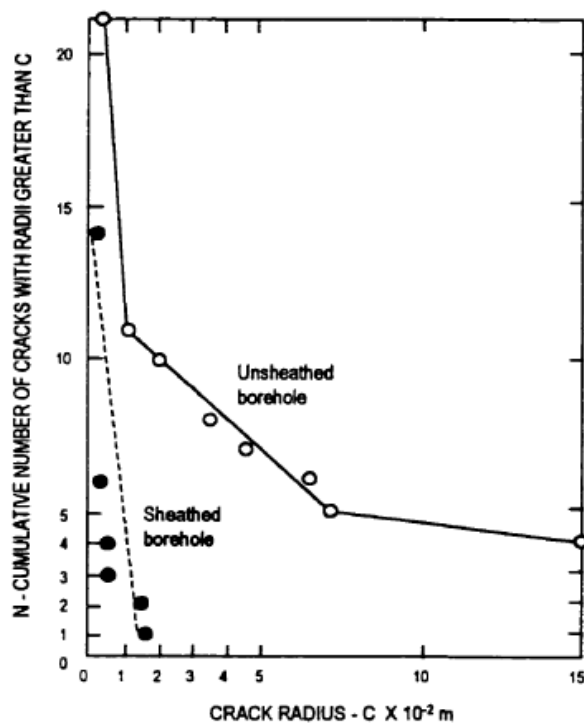


Figure 2.6 Cumulative crack number versus crack radius from laboratory experiments (McHugh, 1983)

Similar experiments were conducted by Brinkman (1987) using aluminum pipes to separate the shock gas components of explosive energy. He concluded that the shock wave is only responsible for fracturing in the close vicinity of the blast hole. This in turn provides the initial fracture pattern for gas penetration and expansion as well as final fracturing, fragmentation, and heaving of the burden.

In spite of the existence of various theories, the dynamic rock fragmentation process cannot be mathematically described yet. Ambiguities such as the precise role of stress wave or gas pressure, partitioning of explosive energy in discontinuous rock masses, dynamic properties of rock and its influence on explosive energy absorption hinder the complete mathematical or numerical description of the fragmentation process. Nevertheless, with regard to the theories discussed above as well as existing knowledge and experience on the rock breakage process, the following consecutive stages could be speculated for the process of rock fragmentation by blasting. This is also shown schematically in Figure 2.7.

Stage a:

Upon detonation shock wave impacts the blast hole walls violently and a thin annulus of rock, surrounding the blast hole, yields completely. The size of this yielded or crushed zone depends on the rate at which pressure builds up (rise time). The rise time is affected by the type of explosive as well as the properties of the material surrounding the blast hole.

Stage b:

Initiated plastic and elastic waves from the crushed zone induce significant fracturing and form a non-linear zone surrounding the crushed zone. The size of this zone defines the effective radius of the blast hole for subsequent gas pressurization.

Stage c:

The plastic wave is almost fully attenuated in the non-linear zone. The elastic stress wave is further propagated out into the rock losing its energy due to attenuation. When it reaches the free faces, the elastic wave is reflected as a tensile pulse, thus causing tension cracks and slabbing at the free faces. Moreover several major cracks originating from the blast hole are produced due to the wave action.

Staged d:

High-pressure detonation products expand and push the modified (slightly expanded) blast chamber outward. Gas expansion and penetration into the cracks promote further fragmentation and the burden starts to detach at this stage. Additionally, crushed material may penetrate into some of the generated cracks and block the gas penetration path.

Stage e:

Full penetration of gas into the rock mass has already occurred and gas has forced a path to the atmosphere. At this stage fragmentation is almost completed, burden is fully detached, and fragments are flying. Additional fragmentation may occur due to the release of strain energy stored in fragments, due to the impact and interaction of fragments, and due to inertial resistance of fragments against preceding accelerating forces generated by high-pressure gases (Mortazavi, 1999).

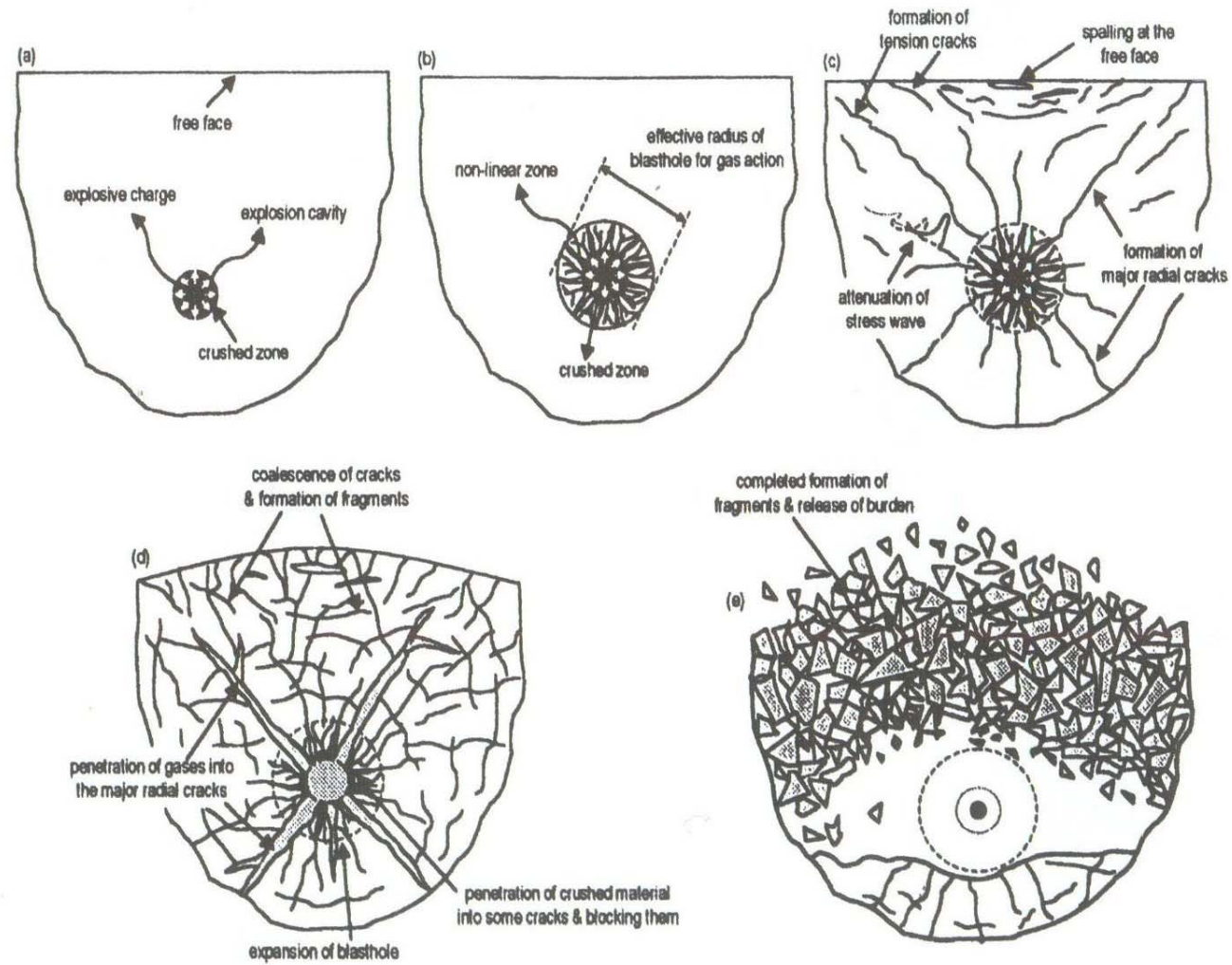


Figure 2.7 Schematic illustration of process of rock breakage by blasting (Mortazavi, 1999)

2.4. Blast Design

To be able to conduct a good blasting operation, the blasting engineer should be aware of the fundamental concepts of blast design. These concepts are useful as a first approximation for blast design. Field testing is the best way to determine the individual blast designs. During the blast design process, two crucial principles must be kept in mind;

- 1) Explosives function best when there is a free face approximately parallel to the explosive column at the time of detonation.
- 2) There must be adequate space for the broken rock to move and expand. Excessive confinement leads to the poor blasting results such as back break, ground vibrations, air blast, unbroken toe, fly rock, and poor fragmentation.

2.4.1. Properties and Geology of the Rock Mass

The rock mass properties are one of the most critical variables affecting the design and results of a blast. Rock properties are very qualitative and cannot be sufficiently quantified numerically when applied to blast design. Rock properties even vary greatly from one blasting operation to another. The specific rock mass properties should be taken into consideration to be able to decide the explosive selection, blast design and delay pattern to be used.

2.4.2. Characterizing the Rock Mass

The geologist and a blasting driller play an important role in characterizing the rock mass. The geologists must concentrate on detailed mapping of the rock surface for blast design. Jointing probably has the most significant effect on blast design. The geologists should document the direction, density, and spacing between the joint sets. At least three joint sets, one dominant and two less pronounced and two less pronounced are in most sedimentary rocks. The strike and dip of bedding planes, foliation, and schistosity are also important to blast design and should be documented

by the geologist. The presence of major zones of weakness such as faults, open joints, open beds, solution cavities, or zones of less competent rock or unconsolidated material are also important to blast design and must be considered. Samples of freshly broken rock can be used to determine the hardness and density of the rock.

An observant blasting driller can be of great help in assessing rock variations that are not apparent from the surface. Slow penetration, excessive drill noise, and vibration indicate a hard rock that will be difficult to break. Fast penetration and a quite drill indicate a softer, more easily broken zone of rock. Total lack of resistance to penetration, accompanied by a lack of cuttings or return water or air, means that the drill has hit a void. Lack of cuttings or return water may also indicate the presence of an open bedding plane or other crack. A detailed drill log indicating the depth of these various conditions can be very helpful in designing and loading the blast. The log should be kept by the driller. The driller should also document changes in the color or the nature of the drill cuttings which will tell the geologist the location of various beds in the formation.

The characterization of rock/ore is also of fundamental importance for optimization of any mine to mill process. Characterization of ore for blasting will be different from that for crushing and grinding operations. For example, the rock mass structure in addition to its strength is very important for blasting, whereas the micro fracture network, grain size and grain characteristics may be important in comminution process such as crushing and grinding. Therefore it is essential to understand these differences and characterize the rock mass in terms of its blastability and comminution properties. (anonymous, 2007)

2.4.3. Blastability

Blasting will loosen the existing rock mass structure to liberate rock blocks as well as creating new fractures within the intact material. Therefore any meaningful description of rock mass blastability should be a function of mechanical properties of

intact rock (such as stiffness and strength) and the rock mass structure (size of “in situ blocks”). Zones within a mine that are of similar strength and structure should blast in a similar way and hence form a blasting domain. (Kanchibotla et al, 2007).

2.4.3.1 Structural Properties

Structural parameters can be determined through scan-line mapping of the exposed bench face or estimated from visual inspection. The volumetric joint count (Jv) is a simple measure of the degree of jointing or the in situ block size of a rock mass. The number of joints intersecting one unit of rock mass is defined as the volumetric count and can be estimated from a joint count made within a window defined on the face. In the case of a new deposit, structural parameters can be estimated using rock quality designation obtained from exploratory diamond drilling.

2.4.3.2 Mechanical Properties of Rock

The mechanical properties of rock such as density, Young’s modulus and uniaxial compressive strength (UCS) influence the rock mass blastability and can be determined from laboratory tests. The samples for conducting these tests are usually obtained from the blast muckpiles. In order to properly characterize the rock strength it may be necessary to conduct tests on a large number of samples, which may be expensive. In such cases the UCS can be obtained indirectly from point load tests performed on irregular samples. Point load testing (Broch & Franklin 1972) is inexpensive and therefore can be performed cheaply on a number of rock samples.

2.4.4. Rock Density and Hardness

Some displacement is required to prepare a muckpile for efficient excavation. The density of the rock is a major factor in determining how much explosive is needed to displace a given volume of rock (powder factor). The burden-to-charge diameter ratio varies with rock density, changing the powder factor. The average burden-to-charge diameter ratio of 25 to 30 is for average density rocks similar to the typical

rocks listed in Table 2.2. Denser rocks such as basalt require smaller ratios (higher powder factors). Lighter materials such as some sandstone or bituminous coal can be blasted with higher ratios (lower powder factor).

Table 2.2 Typical rocks, densities, and unit weights

Rock Type	Density range (gr/cm³)	Range of unit weights (kg/m³)
Limestone	2.5 to 2.8	2.5 to 2.8
Schist	2.6 to 2.8	2.6 to 2.8
Rhyolite	2.2 to 2.7	2.2 to 2.7
Basalt	2.7 to 2.9	2.7 to 2.9
Sandstone	2.0 to 2.6	2.0 to 2.6
Bituminuous coal	1.2 to 1.5	1.2 to 1.5

The hardness or brittleness of rock can have a significant effect on blasting results. If soft rock is slightly underblasted, the rock will still be excavated. If soft rock is slightly overblasted, excessive violence will not usually occur. On the other hand, slight underblasting of hard rock will often result in a tight muckpile that is difficult to excavate. Overblasting of hard rock is likely to cause excessive flyrock and airblast. Blast designs for hard rock require closer control tighter tolerances than those for soft rock.

2.4.5. Voids and Zones of Weakness

Undetected voids and zones of weakness such as solution cavities, “mud” seams, and shears are serious problems in blasting. Explosive energy always seeks the path of least resistance. Where the rock burden is composed of alternate zones of hard material, weak zones and voids, the explosive energy will be vented through the weak zones and voids resulting in poor fragmentation. Depending on the orientation of zones of weakness with respect to free faces, excessive violence in the form of airblast and flyrock may occur. When the drill hole intersects a void, particular care must be taken in loading the charge, the void will be loaded with a heavy

concentration of explosive resulting in excessive airblast and flyrock. If these voids and zones of weakness can be identified and logged, steps can be taken during drill hole loading to improve fragmentation and avoid violence. The best tool for this is a good drill log. The depths of voids and zones of weakness encountered by the drill should be documented. The geologist can help by plotting the trends of “mud” seams and shears. When charging the drill hole, inert stemming material rather than explosives should be loaded through these weak zones. Voids should be filled with stemming. Where this is impractical because of the size of the void, it may be necessary to block the hole just above the void before continuing the explosive column.

If the condition of the drill hole is in doubt, the top of the powder column should be checked frequently as loading proceeds. A void probably exists if the column fails to rise as expected. At this point, a deck of inert stemming material should be loaded before powder loading continues. If the column rises more rapidly than expected, frequent checking will ensure that adequate space is left for stemming. Alternate zones of hard and soft rock usually result in unacceptably blocky fragmentation. A higher powder factor seldom will correct this problem; it will merely cause the blocks to be displaced farther. Usually, the best way to alleviate this situation is to use smaller drill holes with smaller blast pattern dimensions to get a better powder distribution. The explosive charges should be concentrated in the hard rock.

2.4.6 Jointing

Jointing can have a pronounced effect on both fragmentation and the stability of the perimeter of the excavation. Close jointing usually results in good fragmentation. Widely spaced jointing, especially where the jointing is pronounced, often results in a very blocky muckpile because the joint planes tend to isolate large blocks in place. Where the fragmentation is unacceptable, the best solution is to use smaller blast holes with smaller blast pattern dimensions. This extra drilling and blasting expense will be more than justified by the savings in secondary blasting. Where possible, the perimeter holes of a blast should be aligned with the principal joint sets. This

produces a more stable excavation, whereas rows of holes perpendicular to a primary joint set produces a more ragged, unstable perimeter. (Figure 2.8). The jointing will generally determine how the corners at the back of the blast will break out. To minimize backbreak and flyrock, tight corners, as shown in Figure 2.9, should be avoided.

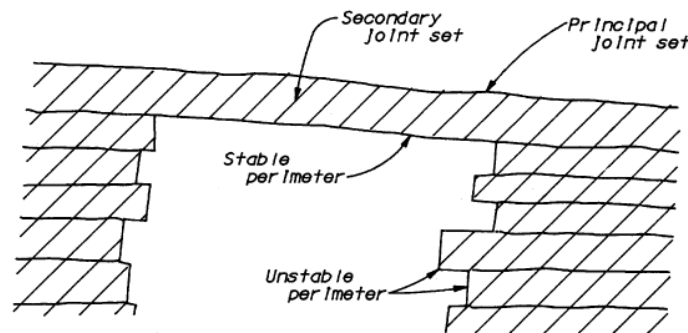


Figure 2.8 Effect of jointing on the stability of an excavation (plan view) (anonymous, 2007)

The open corner at the left of Figure 2.9 is preferable. Given the dominant jointing in Figure 2.9, more stable conditions will result if the first blast is opened at the far right and designed so that the hole in the rear inside corner contains the highest numbered delay.

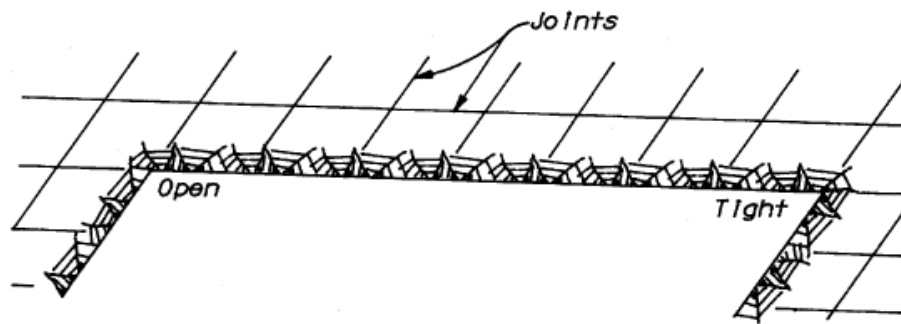


Figure 2.9 Tight and open corners caused by jointing (plan view) (anonymous, 2007)

2.4.7 Bedding/Foliation

Bedding has a pronounced effect on both the fragmentation and the stability of the excavation perimeter. Open bedding planes, open joints, or beds of weaker materials should be treated as zones of weakness. Stemming, rather than explosive, should be

loaded into the borehole at the location of these zones as shown in Figure 2.10. In a bed of hard material (greater than 1m thick), it is often beneficial to load an explosive of higher density than is used in the remainder of the borehole. To break an isolated bed or zone of hard rock (1m thick or greater) near the collar of the blast hole, a deck charge is recommended, as shown in Figure 2.11, with the deck being fired on the same delay as the main charge or one delay later. Occasionally, satellite holes are used to help break a hard zone in the upper part of the burden. Satellite holes Figure 2.4 are short holes, usually smaller in diameter than the main blast hole drilled between the main blast holes.

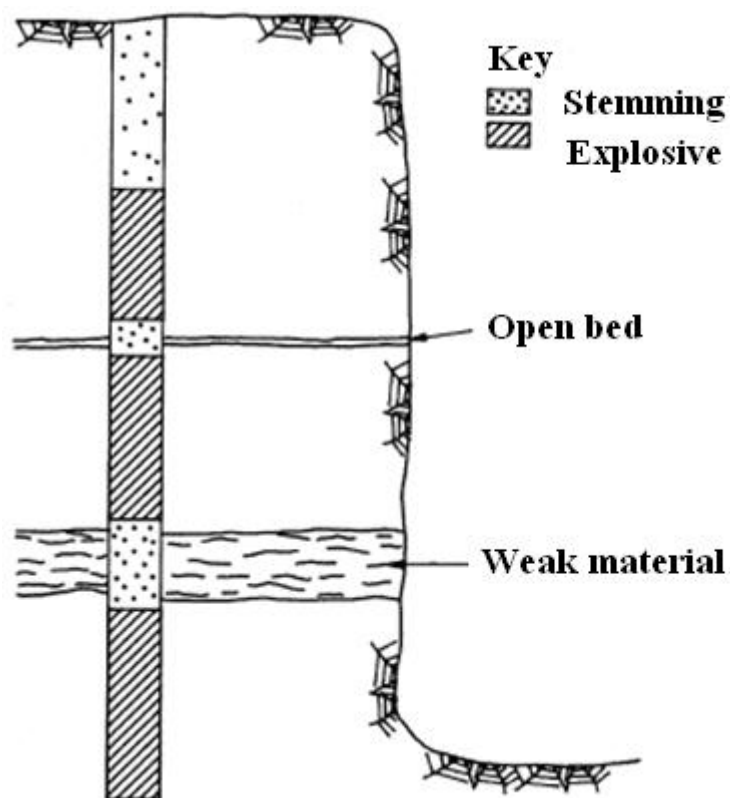


Figure 2.10 Stemming through weak material and open beds
(anonymous, 2007)

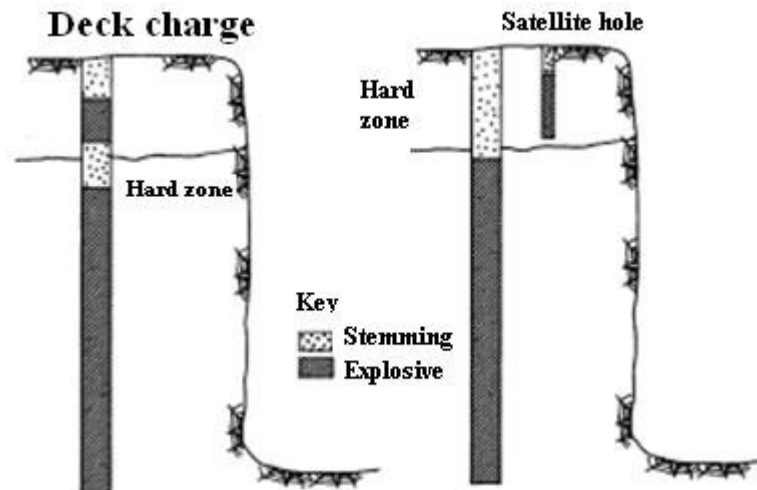


Figure 2.11 Two methods of breaking a hard collar zone
(anonymous, 2007)

A pronounced foliation, bedding plane, or joint is frequently a convenient location for the bench floor. It not only gives a smoother floor but also may reduce subdrilling requirements. Dipping beds frequently cause stability problems and difficulty in breaking the toe of the excavation. When bedding or foliation dip into the excavation wall, the stability of the slope is enhanced. When the dip is out of the wall, slip planes exist that increase the likelihood of slope deterioration or failure. Blasthole cutoffs (part of a column of explosives not fired) caused by differential bed movement are also more likely. Beds dipping out of the final slope should be avoided wherever possible.

Although beds dipping into the face slope stability, the beds can create toe problems because the toe rock tends to break along the bedding or foliation planes. Dipping beds such as these require a tradeoff, depending upon which is the more serious problem—a somewhat unstable slope or an uneven toe. In some cases, advancing the opening perpendicular to dipping beds may be a compromise. Many blasting jobs encounter site-specific geologic conditions not covered in this general discussion. Good blasting technique requires constant study of the geology to make every effort to advantageously use the geology, or at least minimize its unfavorable effects in blast design.

2.5 Surface Blasting

2.5.1 Blast Hole Diameter

The blast hole size is the first consideration of any blast design. The blast hole diameter, along with the type of explosive being used and the type of rock being blasted, determines the burden (distance from the blast hole to the nearest free face). All other blast dimensions are a function of the burden. This discussion assumes that the blaster has the freedom to select the blast hole size based on available drilling equipment.

Practical blast hole diameters for surface construction excavations range from 75mm to approximately 38cm. Large blast hole diameters generally yield low drilling and blasting costs because large holes are cheaper to drill per unit volume and less sensitive, cheaper blasting agents can be used in larger diameter holes. Larger diameter blast holes also allow large burdens and spacings and can give coarser fragmentation. Figure 2.12 illustrates this comparison using (50mm and 500mm)-diameter blast holes as an example.

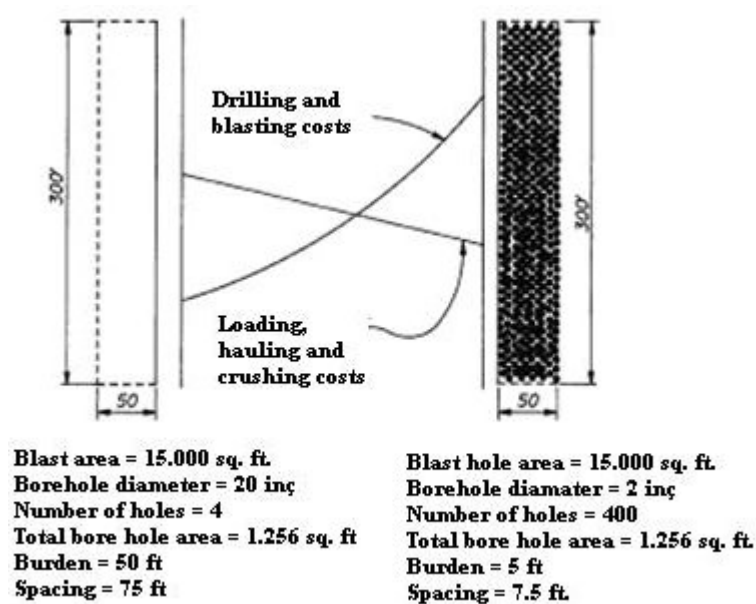


Figure 2.12 The effect of large and small blastholes on unit costs

Pattern A contains four 500mm blast holes, and pattern B contains 400 5mm blast holes. In all bench blasting operations, some compromise between these two extremes is chosen. Each pattern represents the same area of excavation (1400m^2). Each pattern involves approximately the same volume of blast hole, and each can be loaded with about the same weight of explosive. As a general rule, large diameter blast holes (15 to 38cm) have limited applications on most construction projects because of the requirements for fine fragmentation and the use of relatively shallow cuts. However, bore hole size depends primarily on local conditions. Large holes are most efficient in deep cuts where a free face has already been developed.

In most construction projects, small diameter drilling with high-speed equipment provides relatively low unit costs and permits fairly close spacing of holes. This close spacing provides better distribution of explosives throughout the rock mass, which in turn produces better breakage. An additional advantage of small diameter blast holes is that the reduction in the amount of each explosive used in each hole reduces ground vibrations. Construction blast hole diameters usually vary from (90 to 114mm), and the normal drilling depth is less than 12m. reclamation generally limits blast hole diameters for structural excavation drilling to 90mm. blasting patterns usually range from (1,8 by 2,4m) to (2,4 by 4,6m) and are usually rectangular with the burden being less than the spacing.

In a given rock type a four-hole pattern will give relatively low drilling and blasting costs. Drilling costs for large blast holes will be low, a low-cost blasting agent will be used, and the cost of detonators will be minimal. In a difficult blasting situation, the broken material will be blocky and non-uniform in size, resulting in higher loading, hauling, and crushing costs as well as requiring more secondary breakage. In sufficient breakage at the toe may also result. The 400-hole pattern will yield high drilling and blasting costs. Small holes cost more to drill per unit volume, powder for small diameter blast holes is usually more expensive, and the cost of detonators will be higher. The fragmentation will be finer and more uniform, resulting in lower loading, hauling and crushing costs. Secondary blasting and toe problems will be minimized. Size of equipment, subsequent processing required for

the blasted material, and economics will dictate the type of fragmentation needed and the size of blast hole to be used.

Geologic structure is a major factor in determining the blast hole diameter. Planes of weakness (i.e., joints, shears, or zones of soft rock) tend to isolate large blocks of rock in the burden. The larger the blast pattern, the more likely these blocks to be thrown unbroken into the muckpile. Note that in the top pattern in Figure 2.13, some of the blocks are not penetrated by a blast hole. In the similar bottom pattern, all the blocks contain at least one blast hole. Because of the better explosives distribution, the bottom pattern will give better fragmentation.

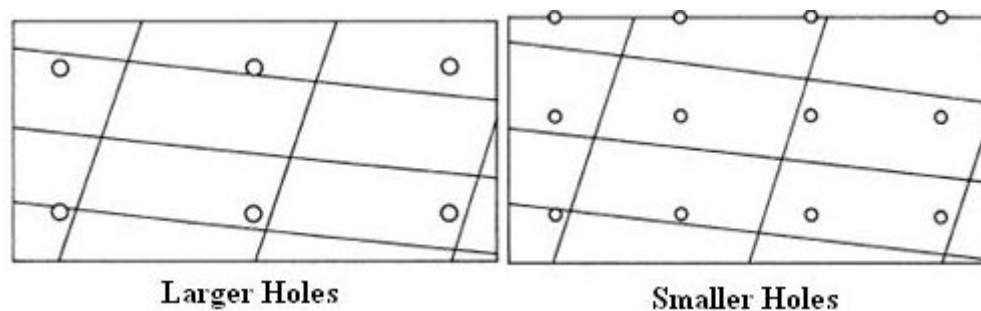


Figure 2.13 The effects of jointing on selection of blast hole size

Airblast and flyrock often occur because of an insufficient collar distance (stemming column) above the explosive charge. As the blast hole diameter increases, the collar distance required to prevent violence increases. The ratio of collar distance to blast hole diameter required to prevent violence varies from 14:1 to 28:1, depending on the relative densities and velocities of the explosive and rock, the physical condition of the rock, the type of stemming used, and the point of initiation. A larger collar distance is required where the sonic velocity of the rock exceeds the detonation velocity of the explosive or where the rock is heavily fractured or low density. A top initiated charge requires a larger collar distance than a bottom initiated charge. As the collar distance increases, the powder distribution becomes poorer, resulting in poorer fragmentation of the rock in the upper part of the bench.

Ground vibrations are controlled by reducing the weight of explosive fired per delay interval. This is done more easily with small blast holes than with large blast

holes. In many situations where large diameter blast holes are used near populated areas, several delays, along with decking, must be used within each hole to control vibrations. Large holes with large blast patterns are best suited to an operation with: (1) a large volume of material to be moved, (2) large loading, hauling, and crushing equipment, (3) no requirement for fine, uniform fragmentation, (4) an easily broken toe, (5) few ground vibration or airblast problems (few nearby buildings), and (6) a relatively homogenous, easily fragmented rock without many planes of weakness or voids. Many blasting jobs have constraints that require smaller blast holes. The final selection of blast hole size is based on economics. Savings realized through inappropriate cost cutting in the drilling and blasting program may well be lost through increased loading, hauling, or crushing costs.

2.5.2 Blast Patterns

The three drill patterns commonly used are square, rectangular, and staggered. The square drill pattern shown in Figure 2.14 has equal burdens and spacing, and the rectangular pattern has a larger spacing than burden. In both the square and rectangular patterns, the holes of each row are lined up directly behind the holes in the preceding row. In the staggered pattern shown in Figure 2.15, the holes in each row are positioned in the middle of the spacings of the holes in the preceding row. In the staggered pattern, the spacing should be larger than the burden.

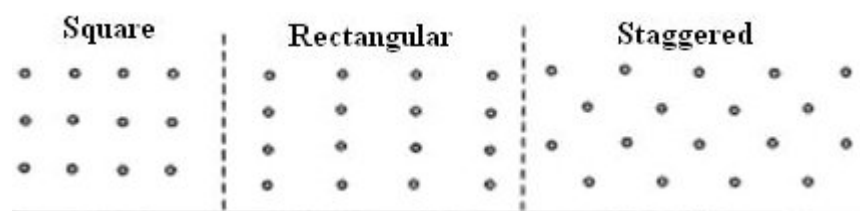


Figure 2.14 Three basic types of drill patterns

Square or rectangular drilling patterns are used for firing V-cut shown in Figure 2.16 or echelon rounds. The burdens and subsequent rock displacement are at an angle to the original free face either side of the blast round in V-cut or echelon patterns. The staggered drilling pattern is used for row-on-row firing where the holes

of one row are fired before the holes in the row immediately behind them as shown in (Fig. 2.16). Looking at Figure 2.16, with the burdens developed at a 45-degree angle to the original free face, the original square drilling pattern has been transformed to a staggered blasting pattern with a spacing twice the burden. The three simple patterns discussed here account for nearly all the surface blasting.

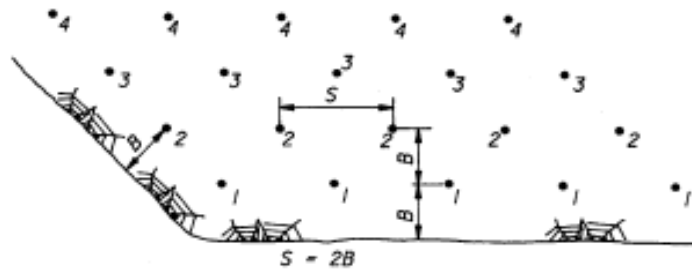


Figure 2.15 Corner cut staggered pattern with simultaneous initiation within rows (blast hole spacing, S , is twice the burden, B).

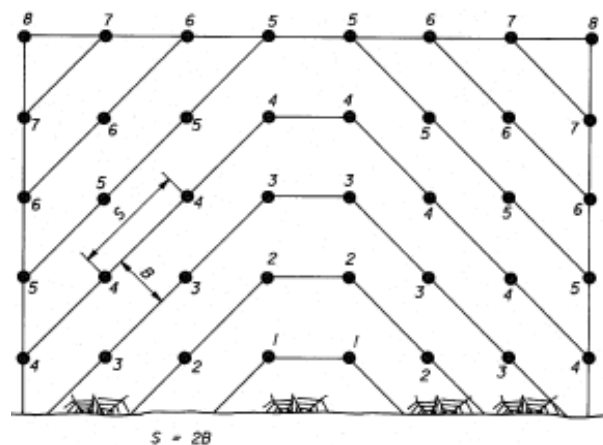


Figure 2.16 V-Echelon blast round (true spacing, S , is the true burden, B).

2.5.3 Burden

Figure 2.17 is an isometric view showing the relationship of the various dimensions of a bench blast. The burden is defined as the distance from a blast hole to the nearest free face at the instant of detonation. In multiple row blasts, the burden for a blast hole is not necessarily measured in the direction of the original free face.

The free faces developed by blast holes fired on lower delay periods must be taken into account.

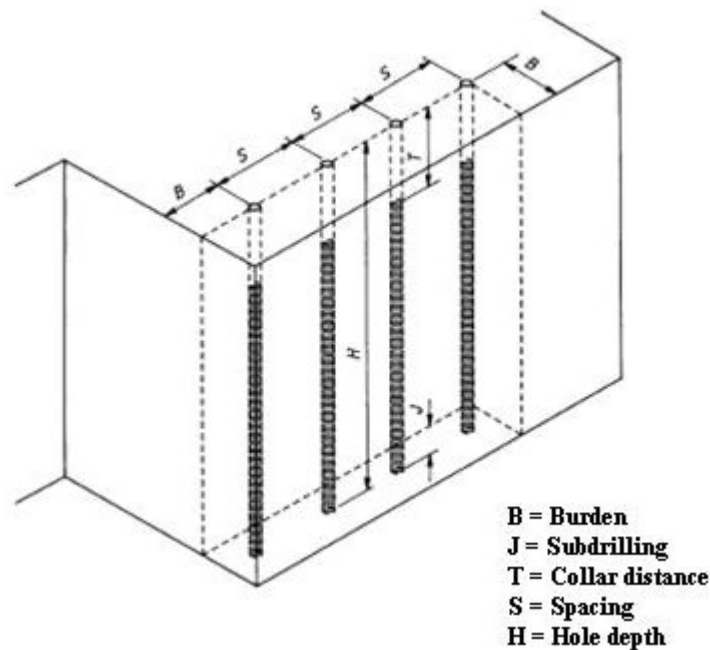


Figure 2.17 Isometric view of a bench blast (anonymous, 2007)

As an example, in Figure 2.15, where one entire row is blasted before the next row begins, the burden is measured perpendicular between rows. In Figure 2.16, the blast progresses in a V-shape. The true burden on most of the holes is measured at an angle of 45 degrees from the original free face. It is very important that the proper burden be calculated, accounting for the blast hole diameter, relative density of the rock and explosive, and, to some degree, the depth of the blast hole. An insufficient burden will cause excessive airblast and flyrock. Too large a burden will produce inadequate fragmentation, toe problems, and excessive ground vibrations. If it is necessary to drill a round before the previous round has been excavated, it is important to stake out the first row of the second round before the first round is fired. This will ensure a proper burden on the first row of blast holes in the second blast round.

For bulk-loaded charges (the charge is poured down the hole), the charge diameter is equal to the blast hole diameter. For tamped cartridges, the charge diameter will be between the cartridges diameter and the blast hole diameter, depending on the degree

of tamping. For untamped cartridges, the charge diameter is equal to the cartridges diameter. When blasting with ANFO (ammonium nitrate/fuel oil mixture) or other low density blasting agents with densities near ($0,85\text{gr/cm}^3$), in typical rock with a density near ($2,7\text{gr/cm}^3$), the normal burden is approximately 25 times the charge diameter. When using denser products such as slurries or dynamites with densities near ($1,2\text{gr/cm}^3$), the normal burden is approximately 30 times the charge diameter. These are first approximations, and field testing usually results in adjustments to these values. The burden-to-charge-diameter ratio is seldom less than 20 or seldom more than 40, even in extreme cases. For instance, when blasting with a low density blasting agent such as ANFO in a dense formation such as basalt, the desired burden maybe about 20 times the charge diameter. When blasting with denser slurries or dynamites in low density formations such as sandstones, the burden may approach 40 times the charge diameter. Table 2.3 summarizes these approximations.

Table 2.3 Approximate burden charge diameter for bench blasting

Material	Ratio
ANFO (density, 0.85 gr/cm^3)	
Light rock (density, 2.2 gr/cm^3)	28
Average rock (density, 2.7 gr/cm^3)	25
Dense rock (density, 3.2 gr/cm^3)	23
Slurry, dynamite (density, 2.2 gr/cm^3)	
Light rock (density, 2.2 gr/cm^3)	33
Average rock (density, 2.7 gr/cm^3)	30
Dense rock (density, 3.2 gr/cm^3)	27

High-speed photographs of blasts show that flexing of the burden plays an important role in rock fragmentation. A relatively deep, narrow burden flexes and breaks more easily than a shallow, wider burden. Figure 2.18 shows the difference between using a 150mm blast hole and a 310mm blast hole in a 12m bench with a burden-to-charge-diameter ratio of 30 and appropriate subdrilling and stemming dimensions. Note the inherent stiffness of the burden with a 310mm blast hole as compared to a 150mm blast hole. Based on this, lower burden-to-charge-diameter

ratios should be used as a first approximation when the blast hole diameter is large in comparison to the bench height. Care must be taken that the burden ratio is not so small as to create violence. Once the ratio has been determined, it becomes the basis for calculating subdrilling, collar distance (stemming), and spacing.

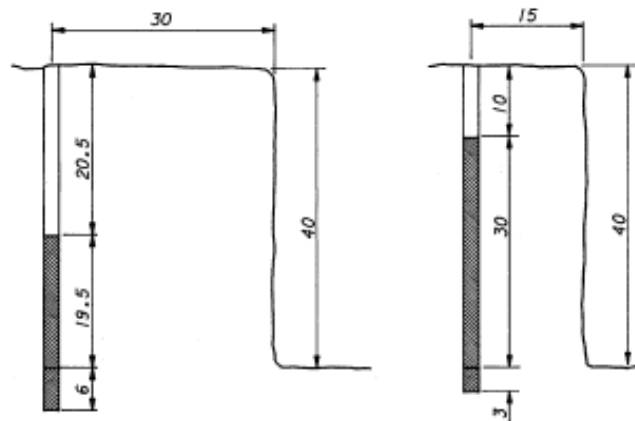


Figure 2.18 Comparison of a (300mm) diameter blast hole (stiff burden) on the left with (150mm) diameter blast hole (flexible burden) on the right in a (12m) bench

2.5.4 Subdrilling

Subdrilling is the distance drilled below the floor level to ensure that the full face of rock is removed. Where there is a pronounced parting at floor level for the explosive charge to conveniently break, subdrilling may not be required. In most surface blasting jobs, some subdrilling is necessary to make sure the shot pulls the grade. In most construction blasting, subdrilling is generally limited to 10 percent or less of the bench height. In blasting for civil engineering structures where a final grade is specified, subdrilling of the final lift is severely restricted. The final lift in structural excavations is usually limited to 1.5 to 3m. Subdrilling is not allowed in a 1.5m lift and is limited to 0.6m for the 3m lift. To prevent damage to the foundation, the diameter of the blast hole is limited to 90mm.

Priming the explosive column at the toe level (bottom of the drill hole) gives maximum confinement and normally gives the best breakage. Toe priming usually requires less subdrilling than collar priming. Toe priming should be restricted in

structural foundations because of the potential damage to the unshot rock. Excessive subdrilling is unnecessary, expensive, and may cause excessive ground vibrations because of the high degree of confinement of the explosive in the bottom of the blast hole, particularly when the primer is placed in the bottom of the hole. In multiple-bench operations, excessive subdrilling may cause undue fracturing in the upper portion of the bench below, creating difficulties in collaring holes in the lower bench. Insufficient subdrilling causes a high bottom, resulting in increased wear and tear on equipment and expensive secondary blasting or hand excavation in structural foundations.

2.5.5 Collar Distance (Stemming)

Collar distance is the distance from the top of the explosive charge to the collar of the blast hole. This zone is usually filled with an inert material called stemming to give some confinement to the explosive gases and to reduce airblast. A well-graded, crushed gravel works best as stemming, but it is common practice to use drill cuttings because of availability and economics. Collar distances that are too short result in excessive violence in the form of airblast and flyrock and may cause backbreak (breaking beyond the back wall). Collar distances that are too long create large blocks in the upper part of the muck pile. The selection of a collar is often a tradeoff between fragmentation and the amount of airblast and flyrock that can be tolerated. This is true especially where the upper part of the bench contains rock that is difficult to break or is of a different type. The difference between a violent blast and one that fails to fragment the upper zone properly may be a matter of only a few meters of stemming. Collar or direct priming (placing the primer at or near the collar of the blast hole with the blasting cap pointing toward the bottom of the hole) of blast holes normally causes more violence than center or toe priming and requires the use of a longer collar distance.

Field experience has shown that a collar distance equal to 70 percent of the burden is a good first approximation. Careful observation of airblast, flyrock and fragmentation will enable further refinement of this dimension. Where adequate

fragmentation in the collar zone cannot be attained while still controlling airblast and flyrock, decks charges or satellite (mid-spaced) holes may be required.

A deck charge is an explosive charge near the top of the blast hole, separated from the main charge by inert stemming. If large blocky materials are being created in the collar zone and less stemming would cause excessive airblast or flyrock, the main charge should be reduced slightly and a deck charge added. The deck charge is usually shot on the same delay as the main charge or one delay later. Do not place the deck charge too near the top of the blast hole, or excessive fly rock may result. Alternatively, short satellite holes between the main blast holes can be used. The diameter of the satellite holes is usually smaller than the main blast holes, and the satellite holes are loaded with a light charge of explosives.

Collar distance is a very important blast design variable. One violent blast can permanently alienate neighbors. In a delicate situation, it may be best to start with a collar distance equal to the burden and gradually reduce this if, conditions warrant. Collar distances greater than the burden are seldom necessary.

2.5.6 Spacing

Spacing is defined as the distance between adjacent blast holes, measured perpendicular to the burden. Where the rows are blasted one after the other as in (Figure), the spacing is measured between holes in a row. Where the blast progresses at an angle to the original free face, as in (Figure 2.16), the spacing is measured at an angle from the original free face. Spacing is calculated as a function of the burden and also depends on the timing between holes. Spacing that is too close causes crushing and cratering between holes, large blocks in the burden, and toe problems. Spacing that is too wide causes inadequate fracturing between holes, toe problems, and is accompanied by humps on the face (Figure 2.19).

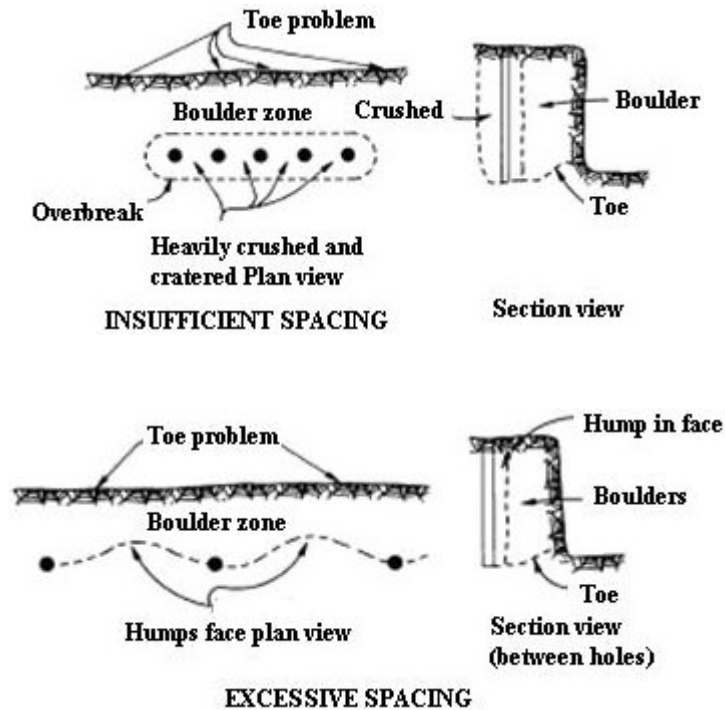


Figure 2.19 Effects of too small and too large spacing

When the holes in a row are initiated on the same delay period, a spacing equal to twice the burden usually will pull the round satisfactorily. The V-cut round in Figure 2.16 illustrates simultaneous initiation within a row, with the rows being the angled lines of holes fired on the same delay. The true spacing is twice the true burden, even though the holes were originally drilled on a square pattern. Field experience has shown that the use of millisecond (ms) delays between holes in a row results in better fragmentation and also reduces ground vibrations produced by the blast. When ms delays are used between holes in a row, the spacing-to-burden ratio must be reduced somewhere between 1.2 and 1.8, with 1.5 being a good first approximation. Various delay patterns may be used within the rows, including alternate delays in Figure 2.20 and progressive delays in Figure 2.21. Generally, large diameter blast holes require lower spacing-to-burden ratios (usually 1.2 to 1.5 with ms delays) than small diameter blast holes (usually 1.5 to 1.8). Because of the geology complexities, the interaction of delays, differences in explosive and rock strengths, and other

variables, the proper spacing-to-burden ratio must be determined through on site experimentation, using the preceding values as first approximations. Except when using controlled blasting techniques such as smooth blasting and cushion blasting, the spacing should never be less than the burden.

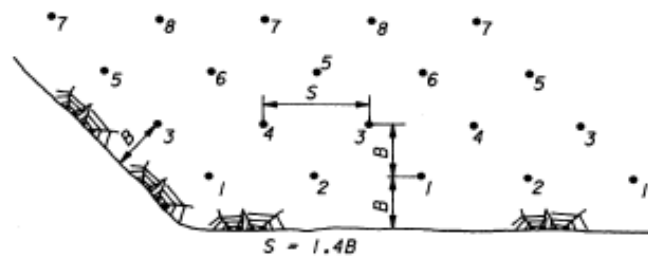


Figure 2.20 Staggered blast pattern with alternate delays
(blast hole spacing, S , is 1.4 times the burden, B)

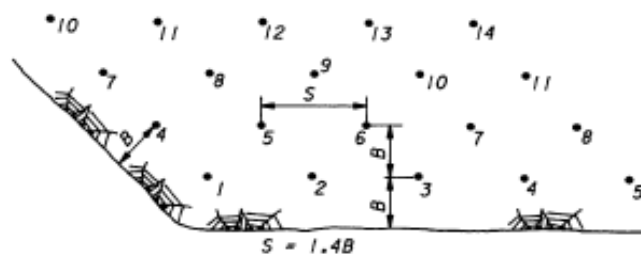


Figure 2.21 Staggered blast pattern with progressive delays
(blast hole spacing, S , is 1.4 times the burden, B)

2.5.7 Hole Depth

In any blast design, the burden and the blast hole depth (or bench height) must be reasonably compatible. The rule of thumb for bench blasting is that the hole depth-to-burden ratio should be between 1.5 and 4.0. Hole depths less than 1.5 times the burden cause excessive airblast and flyrock and, because of the short, thick shape of the burden, give coarse and uneven fragmentation. Where operational conditions require a ratio of less than 1.5, a primer should be placed at the toe of the bench to assure maximum confinement. Keep in mind that placing the primer in the subdrill can cause increased ground vibrations and unacceptably irregular final grades for engineering structures. If the use of a hole depth-to-burden ratio of less than 1.5 is

necessary or specified, consideration should be given to increasing the bench height or using smaller drill hole diameters.

Hole depths greater than the four times the burden are also undesirable. The longer a hole is with respect to its diameter, the more error there will be in the hole location at toe level (hole wandering), the most critical portion of the blast. A poorly controlled blast will result. Extremely long, slender holes have been known to intersect.

High benches with short burdens can also create safety hazards, such as small equipment having to drill the front row of holes near the edge of a high ledge or a small shovel having to dig at the toe of a precariously high face. The obvious solution to this problem is to use a lower bench height. There is no real advantage to a high bench. Lower benches yield more efficient blasting results, lower drilling costs, less change of cutoffs, and are safer. If it is impractical to reduce the bench height, larger rock handling and drilling equipment should be used, effectively reducing the blast hole depth-to-burden ratio. A major problem with long slender charges is the greater potential for cutoffs in the explosive column. If it is necessary to use blast designs with large hole depth-to-burden ratios, multiple priming should be used as insurance against cutoffs.

2.5.8 Delays

Millisecond delays are used between charges in a blast round to:

- To ensure that a proper free face is developed to enable the explosive charge to efficiently fragment and displace the rock.
- To enhance fragmentation between adjacent holes.
- To reduce the ground vibrations created by the blast.
- The delay time between holes in a row should be between 1 and 5 ms for 0,3m of burden. Delay times less than 1ms for 0,3m of burden cause premature shearing between holes, resulting in coarse fragmentation. If an excessive delay time is used between holes, rock movement from the first

hole prevents the adjacent hole from creating additional fractures between the two holes. A delay of 3ms for 0,3m of burden gives good results in many types of rocks.

- The delay time between rows should be two to three times the delay time between holes in a row. To obtain good fragmentation and to control flyrock, a sufficient delay is needed so that the burden from previously fired holes has enough time to move forward to accommodate moving rock from subsequent rows. If the delay between rows is too short, movement in the back rows will be upward rather than outward as shown in Figure 2.22.

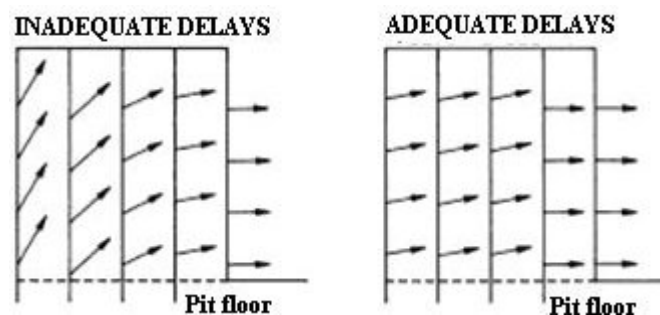


Figure 2.22 The effect of inadequate delays between rows
(anonymous, 2007)

- Where airblast is a problem, the delay between holes in a row should be at least 2ms for every 0,3m of spacing. This prevents airblast from one charge from adding to that of subsequent charges as the blast proceeds down the row.
- For controlling ground vibrations, most regulatory authorities consider two charges to be separate events if they are separated by a delay of 9ms or more.

These rules-of-thumb generally yield good blasting results. When using surface delay systems such as detonating cord connectors and sequential timers, the chances for cutoffs will be increased. To solve this problem, in-hole delays should be used in addition to the surface delays. When using surface detonating cord connectors, the use of 100ms delays in each hole is suggested. This will cause ignition of the in-hole delays well in advance of rock movement, thus minimizing cutoffs. The same effect can be accomplished with a sequential timer by avoiding the use of electric caps with delays shorter than 75 to 100ms.

It is best if the entire explosive in a blast hole is fired as a single column charge. When firing charge blast holes in populated areas, two or more delays within a blast hole can be used to reduce ground vibrations. Blast rounds of this type can become quite complex. All currently delay detonators employ pyrotechnic delay elements that depend on a burning powder train for their delay. Although these delays are reasonably accurate, overlaps have been known to occur. When it is essential that one charge fires before an adjacent charge such as in a tight corner of a blast, it is a good idea to skip a delay period.

2.5.9 Powder Factor

In bench blasting, the powder factor (P.F.) is termed as a measure of the explosive weight required to break a unit volume of rock. However, powder factor is not the best tool for designing blasts. Blast designs should be based on the dimensions discussed earlier in this chapter. Powder factor is a necessary calculation for cost accounting purposes. In construction blasting where the excavated material has little or no inherent value, powder factor is usually expressed in pounds of explosives per cubic yard of material broken. Powder factors for surface blasting can vary from (0.1 to 1.1 kg/m³), with (0.2 to 0.45 kg/m³) being most typical. Powder factor for a single blast hole is calculated by the following:

$$P.F. = \frac{L(0.34d)(D^2)}{27BSH} \quad (2.11)$$

$P.F.$ = Powder factor in pounds of explosive per cubic yard of rock

L = Length of explosive charge in feet

d = Density of explosive charge in grams per cubic centimeter

D = Charge diameter in inches

B = Burden in feet

S = Spacing in feet

H = Bench height

A comparable formula can be developed for metric equivalents. Many companies that provide explosives also publish Tables that give loading densities in pounds per foot of blast hole for different combinations of d and D . A common nomograph found in most blasting handbooks can be used to calculate the density in pounds per foot of borehole. The powder factor is a function of the explosive type, rock density, and geologic structure. Table 2.4 includes typical powder factors for surface blasting.

Table 2.4 Typical powder factors for surface blasting

Rock breakage difficulty	Powder Factor (kg/m³)
Low	0.1-0.18
Medium	0.18-0.34
High	0.34-0.57
Very high	0.57-1.14

CHAPTER THREE

AIR-DECK MECHANISM AND ITS USE IN PRODUCTION BLASTING

3.1. Introduction

In conventional blasting much of the explosive energy is wasted in the generation of undersize fragments or super fragmentation. The production of oversize fragments, on the other hand, lowers loading and transportation efficiencies of the equipment and requires secondary blasting. Melnikov and Marchenko (1971) and Melnikov et al. (1979) reported that, by introducing one or more air gaps in the explosive column, a secondary shock wave can be readily and inexpensively generated thereby increasing the duration of the shock wave action on the surrounding rock mass by a factor of 2–5. The explosive consumption was reported to be reduced by 10–30% with apparent benefits in both fragmentation and movement. Air-Deck was originally applied in the late 19th century as a means to distribute the explosives pressure more evenly throughout the rock mass being blasted. This was critical to reduce shattering in dimensional stone, yet was also recognized as a means to “stretch the powder” in conventional shooting. The technique was restricted primarily to small diameter holes using black blasting powder in dimensional stone work. Today, black powder shooting in dimensional stone quarries is the exception rather than the rule. Today Air-Deck is being used successfully around the world to allow reductions of from 10% to 30% in total explosives required for production blasting. With the increase pressure on operators in large volume operations to reduce blasting costs, potential savings of this magnitude cannot be ignored. Other common applications of Air-Deck are for wall control, rip-rap production, to reduce vibration/overpressure and to create a barrier to groundwater using pre-splitting techniques. In all these cases, Air-Deck can result in substantial savings and improved efficiencies over conventional methods.

3.2 Overview of Air-Deck Related Researches

The Air-Deck blasting technique has been used over a number of years. The earliest reference regarding the use of Air-Deck in production blasting date as far back as 1893 to the work of Knox (Knox, 1893). In the recent past, most of the original research on this topic the area has been conducted in the former Soviet Union (Melnikov & Marchenko 1971, Melnikov et al. 1976, 1979 and Marchenko 1982). According to their research, air-decking results in an increase in the degree and uniformity of fragmentation, a decrease of average fragment size and the amount of lump, while the specific consumption of explosives can be reduced by 10-30%. Laboratory experiment carried out by Fournery et al. (1981) in U.S.A involving the use of thick plexiglass blocks has supported these findings and indicated that the introduced air-decks increase the duration of the shock wave action on the surrounding material by a factor of two to five. Chiappetta et al (1987) used high-speed photography to evaluate air decks, stemming retention and gas confinement in pre-splitting, reclamation and gross motion application. Moxon et al. (1993) carried out a laboratory and theoretical study of air-decks to assess the effect of the length and location of air-decks on fragmentation. Liu and Katsabanis (1996) carried out a series of numerical simulations to investigate the physical processes and effects of blasting with air decks. Many references describe the successful application of air decking techniques for rock fragmentation in mines and quarries (Wlodarczyk, 1983; Stachura, et al, 1991; Woolf, et al, 1994; Pal Roy, et al 1995; Yang, et al 1999; Jhanwar, et al 1999, 2000a, 2000b, etc.).

Moxon et al.(1993) showed that as the length of the air-deck is increased the degree of fragmentation is reduced relative to that of a full-column charge. The reduction however is relatively small until a critical length is exceeded. The critical length depends on the strength and structure of the surrounding material. A critical air-deck length of 30-35% of the original explosive column was determined for the model materials used. They inferred that a mid-column air-deck has a better fragmentation effect than that of the top or bottom air-deck. Liu and Katsabanis (1996) found that there exists a minimum beneficial air deck length and that variation

to top air deck such as bottom and mid-column air-decks do not benefit production blasting.

3.3 Analysis of the Gasdynamic Process of the Detonation Products in a Blast Hole with an Air Deck

By introducing an air deck in a traditional explosive column, the expansion processes of the detonation products can be completely different from that of the fully charged blast hole. In the case of air-decking, the processes involve the complicated interactions between the detonation products and the end of stemming or the bottom of blast hole. Several researchers have studied the processes (Borovikov, 1975, Kucheryavyi, et al, 1976, Borovikov, et al 1984). Effort devoted in this part is to find some kind of theoretically correct and easy to use method, so as to decide easily the changing pressure distribution of the detonation products within blast hole. In order to simplify the analysis processes, the following assumptions has been made: (1) During the expansion of detonation products in blast hole, only the axial movement is taken into account; (2) the detonation products are treated as ideal gases; (3) the expansion of the detonation products in the blast hole is an adiabatic process

3.3.1 Some Basic Equations for Pressure Wave in a Shock Tube

The equations of motion of a one-dimensional compressible gas (Wright, 1961) can be expressed as follows.

Equation of conservation of mass:

$$A \frac{\partial P}{\partial t} + \frac{\partial(\rho v A)}{\partial x} = 0 \quad (3.1)$$

Equation of conservation of momentum:

$$A \frac{\partial v}{\partial t} + v \frac{\partial v}{\partial x} = - \frac{1}{\rho} \frac{\partial P}{\partial x} \quad (3.2)$$

Equation of state of gas:

$$P = \rho RT \quad (3.3)$$

For a adiabatic process there still is:

$$P\rho^{-\gamma} = \text{const} \quad (3.4)$$

In Equations (3.1)-(3.4), A=cross-section area of blast hole, P=pressure of the detonation product, v=flow velocity, ρ =density, T=temperature, R=the universal gas constant, γ =adiabatic index of detonation product.

As for weak pressure waves in a blast hole with a constant cross-section, it follows that, at any point in the wave, the flow velocity of gas is:

$$v = \frac{2c_0}{\gamma - 1} \left\{ \left(\frac{P}{P_0} \right)^{(\gamma-1)/2\gamma} - 1 \right\} \quad (3.5)$$

Where, c_0 and P_0 are original sound speed and pressure of gas respectively, c_0 is expressed as:

$$c_0 = \sqrt{\frac{\partial P}{\partial \rho}} = \sqrt{\frac{\gamma P_0}{\rho_0}} \quad (3.6)$$

Shock wave propagation in constant cross-section tube is shown in Figure 3.1. A shock wave is moving into still air with a velocity V, the region ABCD contains the shock transition. The pressure P, temperature T, density ρ , internal energy (e) per unit mass of the gas and the sound speed in front and behind the shock are discriminated with subscript 0 and 1 respectively.

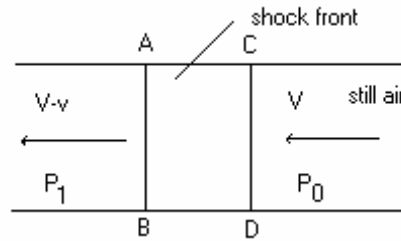


Figure 3.1 Propagation of shock wave
in a constant cross-section tube
(Lu&Hustrulid, 2003)

The equations of motion can be easily as follows. Equation of conservation of mass:

$$\rho_0 V = \rho_1 (V - v) = m \quad (3.7)$$

Equation of conservation of momentum:

$$mv = P_1 - P_0 \quad (3.8)$$

Equation of conservation of energy:

$$P_0 V + me_0 + \frac{1}{2} m V^2 = P_1 (V - v) + me_1 + \frac{1}{2} m (V - v)^2 \quad (3.9)$$

For ideal gas, the internal energy of unit mass of gas can be expressed as:

$$e = \frac{P}{(\gamma - 1)\rho} \quad (3.10)$$

Equations (3.6)-(3.10) yield:

$$\frac{V}{c_0} = \left(\frac{y + \mu}{1 + \mu} \right)^{\frac{1}{2}} \quad (3.11)$$

$$\frac{v}{c_0} = \frac{(1 - \mu)(y - 1)}{[(1 + \mu)(y + \mu)]^{\frac{1}{2}}} \quad (3.12)$$

$$\frac{\rho_1}{\rho_0} = \frac{\mu + y}{1 + \mu y} \quad (3.13)$$

$$\frac{T_1}{T_0} = Y \frac{1 + \mu y}{\mu + y} \quad (3.14)$$

Where, $\mu = (\gamma - 1)/(\gamma + 1)$ and $y = P_1 / P_0$ (Lu&Hustrulid, 2003).

3.3.2 Pressure Waves in Blast Hole with Air Deck

The real shock wave propagation of detonation products in a blast hole with an air deck is shown in Figure 3.2. The initial average detonation pressure P_e is:

$$P_e = \frac{\rho_e D^2}{2(\gamma + 1)} \quad (3.15)$$

Where ρ_e =density of explosive, D =detonation velocity of explosive.

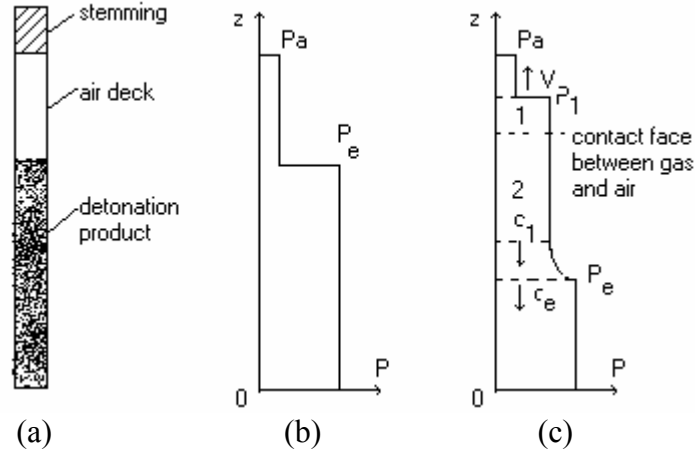


Figure 3.2 Pressure waves in a blast hole with top air deck
(Lu&Hustrulid, 2003)

Figure 3.2(b) corresponds to the instant when the detonation wave arrives at the top of explosive charge. Thereafter a shock wave propagates in THE air while rarefactions propagate back into detonation product, shown in Figure 3.2(c).

The flow velocity v_1 of air behind the front of shock wave is given by Equation (3.12), that is:

$$\frac{v_1}{c_0} = \frac{(1 - \mu_a) \left(\frac{P_1}{P_a} - 1 \right)}{\left[(1 + \mu_a) \left(\frac{P_1}{P_a} + \mu_a \right) \right]^{\frac{1}{2}}} \quad (3.16)$$

Where the subscript a stands for air. The flow velocity v_2 of detonation product behind the rarefaction wave is given by Equation (3.5), that is:

$$v_2 = -\frac{2c_e}{\gamma - 1} \left\{ \left(\frac{P_2}{P_e} \right)^{(\gamma-1)/2\gamma} - 1 \right\} \quad (3.17)$$

Where the sign “-” means the direction of flow velocity of detonation product is opposite to the direction of the propagation of the rarefaction wave. At the contact interface between the air and the detonation products, there must be pressure and flow velocity continuity:

$$P_1 = P_2 \quad (3.18)$$

$$v_1 = v_2 \quad (3.19)$$

From Equation (3.16)-(3.19), v_1 and P_1 can be decided. Other parameters V_1 , ρ_1 and T_1 then can be calculated by Equations (3.11), (3.13) and (3.14).

At time t_1 , soon after the shock wave in air deck reaches the bottom of the stemming, the incident shock wave is reflected as another shock wave that propagates toward the detonation products. This is shown in Figure 3.3. For the incident shock wave in the air, the shock wave parameters have been described earlier. For the reflected shock wave, Equation (3.12) yields:

$$\frac{v_r}{c_1} = \frac{(1 - \mu_a)(\frac{P_r}{P_l} - 1)}{[(1 + \mu_a)(\frac{P_r}{P_l} + \mu_a)]^{\frac{1}{2}}} \quad (3.20)$$

Given that the movement velocity of the stemming material is v_s , then there is:

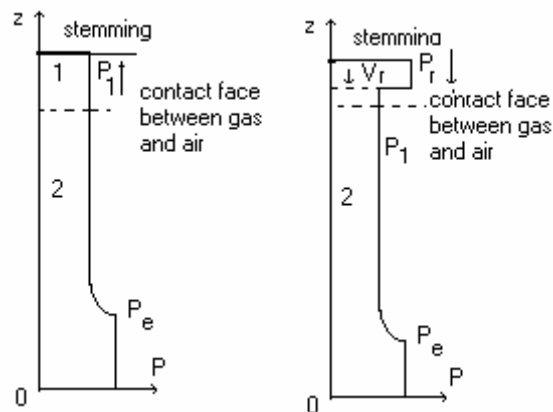
$$v_1 - v_r = v_s \quad (3.21)$$

Combining Equations (3.16), (3.20), and (3.21) (Lu&Hustrulid, 2003), the pressure P_r of the reflected shock wave can be decided. When the reflected shock wave reaches the contact interface between the air and the detonation product, a refraction of shock wave occurs. This is shown in Figure 3.4.

The reflected wave in compressed air can be a reflected shock wave or a reflected rarefaction wave. The procedure is to write down expressions for the flow velocity on either side of the interface in terms of the unknown pressure behind the transmitted shock and reflected rarefaction or shock wave (Equation (3.5) or (3.12)

respectively). Then in accord with the continuous conditions of velocity and pressure on the contact interface, it can be decided whether there is a reflected shock wave or a rarefaction.

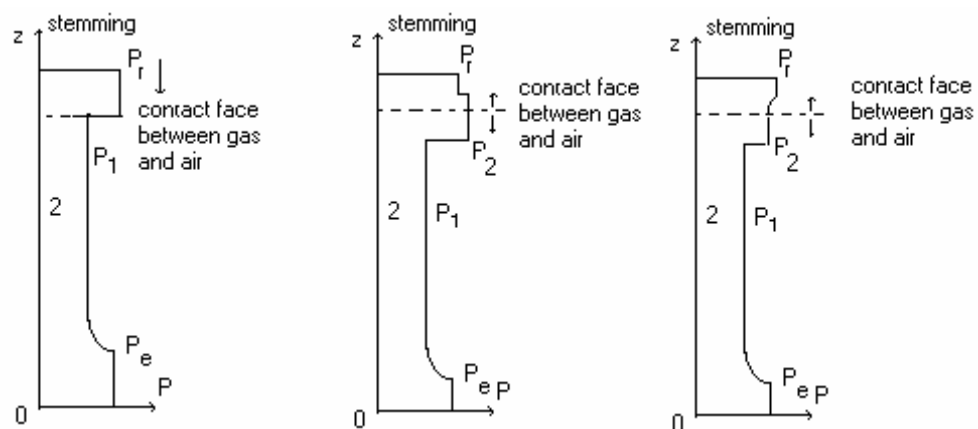
After calculation, we can find that the reflected wave is also a shock wave. When the reflected shock wave propagating in the compressed air reaches the end of stemming, another interaction between air and stemming begins, again the reflected wave from the stemming is a shock wave.



(a) Before reflection (b) After reflection

Figure 3.3 The reflection of the incident shock wave at the bottom of the stemming

(Lu&Hustrulid, 2003)



(a) Before refraction (b) Refraction with reflected shock (c) Refraction with reflected rarefaction

Figure 3.4 The refraction of the reflected shock wave (Lu&Hustrulid, 2003)

However because the flow velocity of the compressed air is much lower than that of the first time, the pressure rise for the second reflected shock wave is far lower than first one. The interactions between air and detonation products and between air and stemming repeat several times, until the reflected rarefaction from the bottom of the blast hole arrives. After detonation products expand into the air, a rarefaction wave propagates inward detonation product at sound speed, this is shown in Figure 3.2. This wave is reflected as another rarefaction wave after it reaches to the bottom of the blast hole. Because both sound speed and flow velocity are varying in the region of the rarefaction fan, it would be very complex to calculate the detailed parameters of the reflected rarefaction wave in this region. This up-propagating reflected rarefaction wave will finally meet with the down-propagating reflected shock wave, another refraction of shock waves occurs. After several times of interactions between shock and rarefaction waves and between waves and the end of stemming or the bottom of the blast hole, the gas in the chamber of the blast hole finally reaches an average pressure P_0 , shown in Figure 3.5.

From the above analysis, we find that it can be very difficult and complicated to calculate theoretically or analytically all the pressure waves parameters in a blast hole with an air deck. From the viewpoint of the engineering application, it is necessary to find an approximation method that is simple to use but is still theoretically correct.

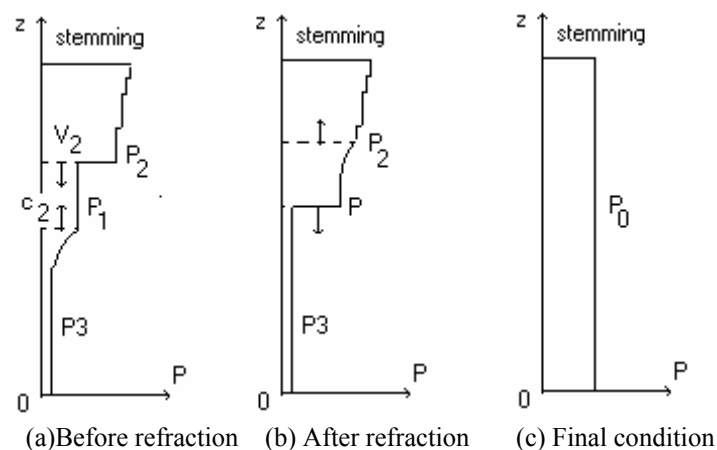


Figure 3.5 The interaction between shock wave and rarefaction wave (Lu&Hustrulid, 2003)

3.3.3 A Simplified Method to Decide the Pressure Distribution in Blast Hole with Air Deck

Compared with the detonation products of typical solid explosives such ANFO, the density of the air (1.29kg/m^3 , at 0°C and 760 mm Hg) is only one seven-hundredth of that of ANFO, which is negligible. During a blasting, the air is initially static and at room temperature and pressure. The air material under such physical conditions offers virtually no resistance to the expansion of the detonation products, which has a temperature and pressure about 3-4 orders higher in magnitude. So, during analysis, the air deck can be treated as a “vacuum deck” (Liu and Katsabanis, 1996). This will greatly simplify the calculating process.

At $t=0$, the detonation of explosive in blast hole is just completed, as shown in Figures 6(a) and 6(b). At $t=t_1$, the detonation products are just reaching the end of stemming (shown in Figure 3.6(c)). From previous analysis, it can be concluded that only after the air in air deck is compressed to same order of the pressure and heated to same order of the temperature of the detonation product, can the magnitude of the pressure of the reflected shock wave from the end of the stemming reach a maximum. That means the contact interface between the air and the detonation products is now moved to some point very near to the end of stemming. If we can decide the average movement velocity of the interface, then we can decide the time t_1 when the detonation product is just reaching the end of stemming.

As an approximation, assume the wave parameters behind the front of the rarefaction wave in the detonation products is constant.

The equation of the conservation of the mass is:

$$\rho_e c_e t_1 = (c_e t_1 + L_a) \rho_1 \quad (3.22)$$

Where L_a = length of air deck, as shown in Figure 3.6(a). The time t_1 is defined as:

$$t_1 = \frac{L_a}{v} \quad (3.23)$$

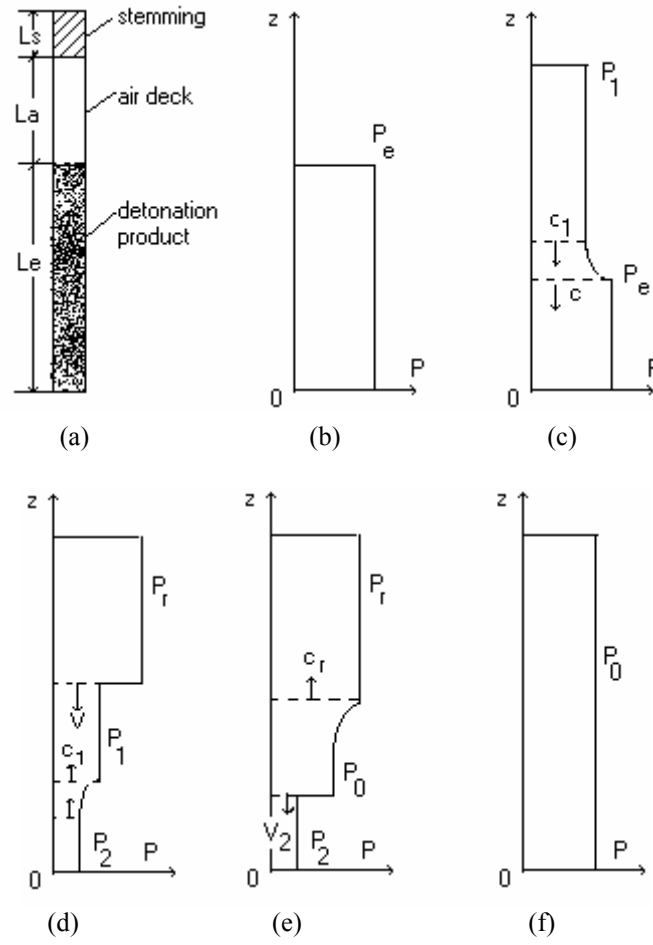


Figure 3.6 The pressure history of blast chamber with an air deck (Lu&Hustrulid, 2003)

Then the equation of conservation of momentum is:

$$P_e = \rho_e c_e v \quad (3.24)$$

Where, c_e = sound speed of the original detonation product. The values of c_e and P_e can be calculated with Equations (3.6) and (3.15), respectively. Combining Equations (3.22)-(3.24) and Equation (3.4), it yields:

$$P_1 = P_e \left(\frac{c_e}{v + c_e} \right)^\gamma \quad (3.25)$$

$$v = \frac{P_e}{\rho_e c_e} \quad (3.26)$$

After the detonation products expand to the bottom of stemming, the incident pressure is reflected as a shock wave as shown in Figure 3.6(c). Again using Equation (12) to decide the flow velocity of the reflected shock wave, it turns out that

$$\frac{v_r}{c_1} = \frac{(1 - \mu)(\frac{P_r}{P_l} - 1)}{[(1 + \mu)(\frac{P_r}{P_l} + \mu)]^{\frac{1}{2}}} \quad (3.27)$$

Then combining with the continuous condition of flow velocity at the end of the stemming, e.g. Equation (3.21), we can finally decide P_r , the pressure of reflected shock wave, shown in Figure 3.6(d).

As mentioned in the previous paragraphs, the rarefaction wave traveling in the detonation products is reflected as another rarefaction wave from the bottom of blast hole. It will be very complicated to calculate the detailed parameters of this wave, especially after the interaction with the reflected shock wave. Fortunately, simulations show that the detailed pressure magnitude of the reflected rarefaction wave has little influence on the final stress and velocity fields in surrounding rock mass. So as an approximation, the pressure (P_2) of the gas behind the front of the reflected rarefaction wave is just taken as zero. It will simply be assumed that after the rarefaction wave meeting with the reflected shock wave, the pressure of detonation product behind the front of the rarefaction is reduced to the average pressure P_0 , where P_0 is defined as:

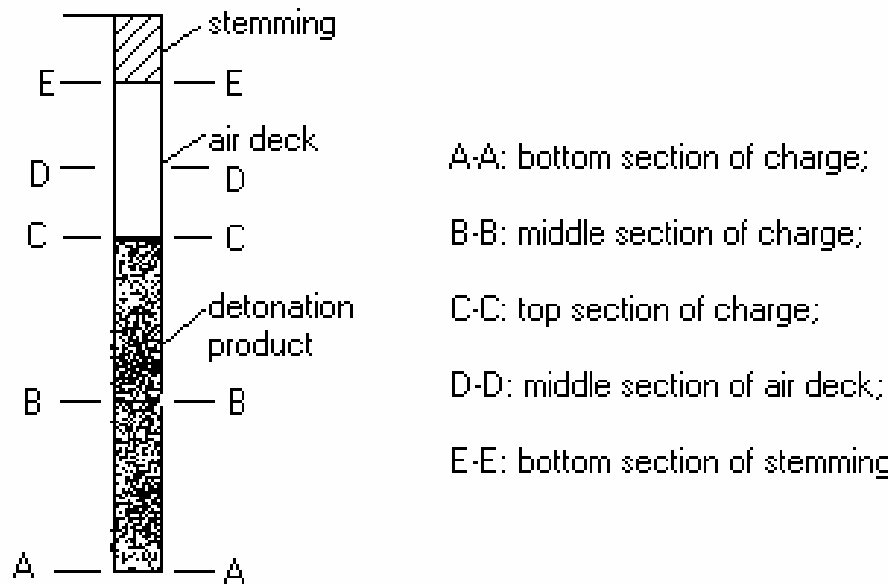
$$P_0 = P_e \left(\frac{L_a}{L_a + L_e} \right)^\gamma \quad (3.29)$$

The simulation parameters adopted are as follows:

Blast hole: diameter=250mm, depth=20.0m, length of stemming (L_s)=2.0m, length of air deck (L_a)=4.5 m, length of the charged column (L_e)=13.5m.

Explosive: type=ANFO, density of explosive (ρ_e)= 850kg/m³, detonation velocity (D)=4000m/s, adiabatic index (γ)=2.4. Then it is easy to decide that P_e =2000MPa (Equation (3.15)), c_e =2376m/s and c_1 =1862m/s (Equation (3.9)), v =990m/s (Equation (26)), P_1 =867MPa (Equation (3.25)), P_0 =1003MPa (Equation (3.28)).

If the movement velocity of the stemming material is considered as negligible, then from Equations (3.27) and (3.21), P_r can be calculated, that is P_r =3765Mpa. Then the velocity of shock front can be decided by Equation (3.11) that is 3368m /s. Now combined with the arrival times of expanding gas, reflected shock wave, rarefaction wave and reflected rarefaction wave, the pressure history of every section of the blast chamber can be decided. The pressure-time histories at some typical sections are shown in Figure 3.7.



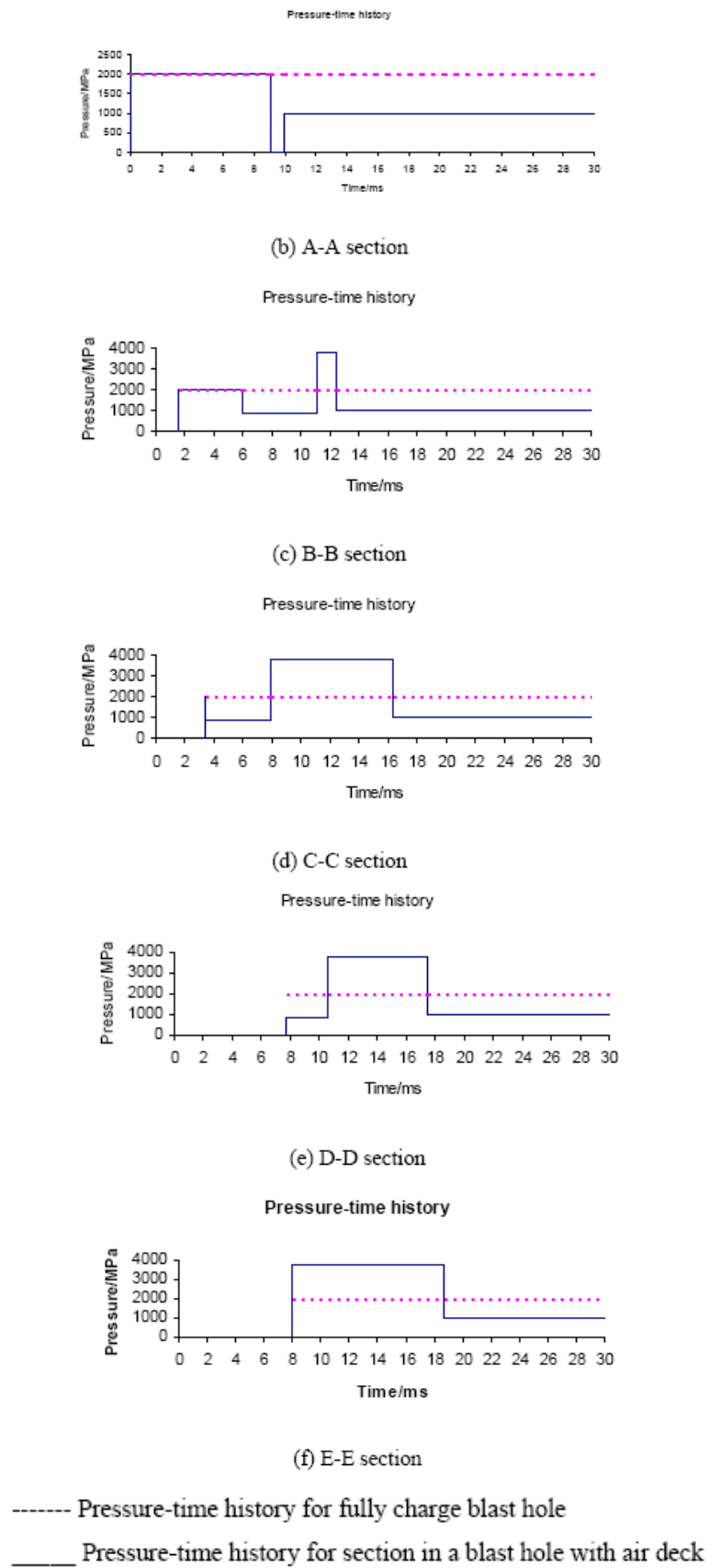


Figure 3.7 Pressure time histories of some typical sections

3.3.4 Further Understanding of the Mechanism of Air Decking

In agreement with the understanding of the mechanism for air decking proposed by previous researchers (Melnikov and Marchenko 1971, Melnikov et al. 1979 and Marchenko 1981, Fourny et al. 1981, Moxon et al. 1993, Liu and Katsabanis, 1996), combined with the analysis and discussion of the simulated result presented in this paper, a further understanding of the mechanism of air decking is showed as follows:

The pressure-unloading process caused by the propagation of the rarefaction wave and reflected rarefaction waves in the detonation products plays an important role in air-decking's beneficial breakage effect. The pressure unloading process can induce high magnitudes of tensile stress and shear strain in the rock mass around blast hole. This helps to form an initial fracture network before the action of quasi-static pressure of detonation products.

Owing to the shock wave reflected from the end of the stemming caused by the incidence of fast expanding detonation product, the upper part of the blast hole is acted upon by a dynamic gas pressure with a magnitude higher than that of the average detonation pressure caused by the detonation of the explosive in a fully charged blast hole. The reflected shock wave acts actually as a main energy source to break rock in the air deck and stemming part.

The second and following strain waves induced by the unloading or reloading of pressure within the blast hole also contribute to form the initial fracture network in rock around the blast hole. Introducing an air deck lowers the final average pressure of the detonation products in the blast hole. Hence the overall of the degree of the crush of rock in a near zone around blast hole is reduced. This helps to utilize the explosive energy more efficiently.

3.3.5 Reasonable Air-Decking Ratio

During the application of air –decking blasting technique, the most important design parameter should be the “air decking ratio (R_a)”. This is defined as the ratio

between the length of air deck and the total length of explosive column including the air deck.

$$R_a = \frac{L_e}{L_a + L_e} \quad (3.30)$$

Where; R_a = air deck ratio, L_e = length of the charged column, L_a = length of air deck.

Two guidelines for choosing this design parameter can be obtained from the aforementioned understanding of the mechanism for blast with air decks: (1) The pressure-unloading process plays an important role in inducing high magnitudes of tensile stress and shear strain in rock mass, so it is reasonable to let every section of the charged part of the blast hole undergo a process of pressure-unloading. It means that the rarefaction wave from the contact interface between detonation product and air should reach to the bottom earlier than that of the reflected shock wave from the end of the stemming. (2) While the pressure magnitude of reflected shock wave can be higher than the average detonation pressure in a fully charged blast hole, it should be kept in mind that the magnitude of the final average gas pressure in blast hole with an air deck is absolutely lower than the average detonation pressure, it is necessary to assure that the wall of the part of blast hole with air deck should undergo the action of high-magnitude pressure of the reflected shock wave. This implies that the reflected shock wave should be reach to the contact interface between detonation products and the air than that of the reflected rarefaction wave from the bottom of the blast hole. These turn out that

$$\frac{L_a}{v} + \frac{L_a}{V} \leq \frac{L_e}{c_e} + \frac{L_e}{c_1} \quad (3.31)$$

$$\frac{L_a}{v} + \frac{L_a + L_e}{V} \geq \frac{L_e}{c_e} \quad (3.32)$$

Combining Equation (3.31) and (3.32), (Lu&Hustrulid, 2003) yields

$$\frac{v(V - c_e)}{c_e(v + V)} \leq \frac{L_a}{L_e} \leq \frac{vV(c_e + c_1)}{c_e c_1(v + V)} \quad (3.33)$$

Substitute the detailed values of parameters in Equation (3.33), and combine with Equation (3.30), it turns out

$$0.087 \leq R_e \leq 0.423 \quad (3.34)$$

In practice, the procedure of the failure of rock mass under dynamic loading takes some time, so we should modify Equation (3.31) and (3.32) as

$$\frac{L_a}{v} + \frac{L_a}{V} + t_f \leq \frac{L_e}{c_e} + \frac{L_e}{c_1} \quad (3.35)$$

$$\frac{L_a}{v} + \frac{L_a + L_e}{V} \geq \frac{L_e}{c_e} + t_f \quad (3.36)$$

Where, t_f is the time taken for the failure of rock mass under dynamic loading. If t_f is set as 1ms and 2ms, then the values for R are showed in Equations (3.37) and (3.38) respectively.

$$0.135 \leq R_a \leq 0.398 \quad (3.37)$$

$$0.164 \leq R_a \leq 0.374 \quad (3.38)$$

3.4 General Applications Related to Air-Deck

To day air decking is being used successfully around the world to allow reductions of from 10% to 30% in total explosives required for production blasting. With the increased pressure on operators in large volume operations to reduce blasting costs, potential savings of this magnitude cannot be ignored. Other common applications of air decking are for wall control, rip-rap production, to reduce vibration/overpressure and to create a barrier to ground water using pre-splitting techniques. In all these cases, air decking can result in substantial savings and improved efficiencies over conventional methods.

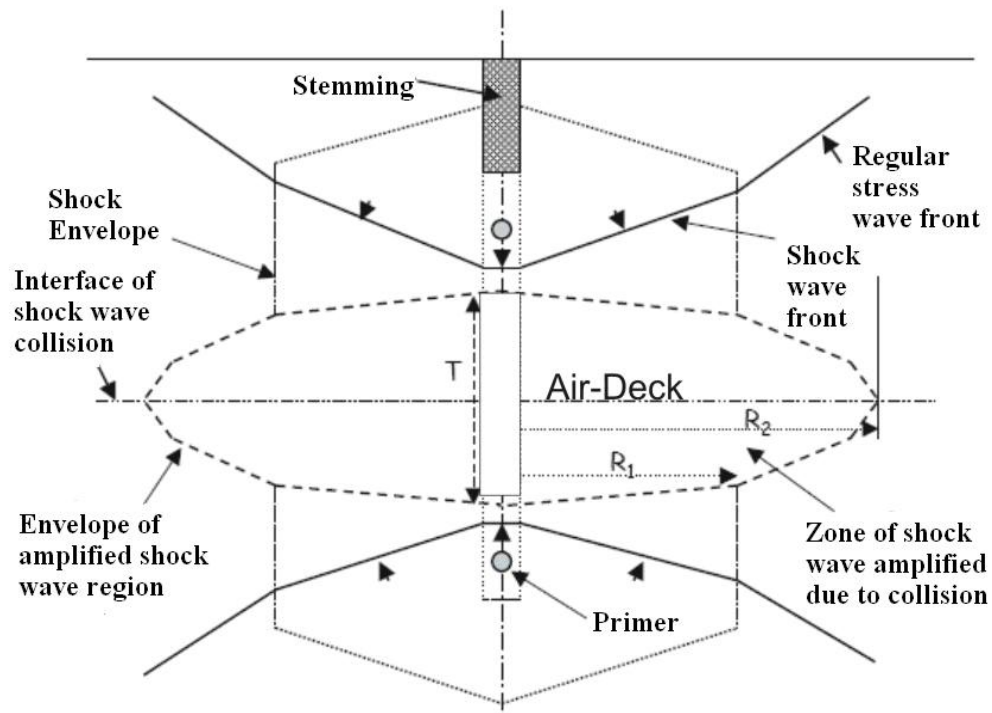


Figure 3.8 Cross section of explosive charge with airdeck

When the blast hole round charged conventionally is detonated, undesirable effects could be encountered both in muck pile and in environmental parameters such as boulders need to be blasted again, inadequate fragmentation size, too much back break, toe occurrence, unproductive hauling operations due to the inadequate fragmentation as well as ground vibration and noise amount. Air deck introduced in predetermined location within a blasthole according to the kind of blasting operation can almost eliminate all the above mentioned parameters. However, creating an air volume within the blast hole is an important and special matter. Since this technique has become common in America and Australia, various kinds of devices produced by different companies are used for mentioned operations. The main aim of these devices is to create an air gap at the proper location within the blast hole. For instance, the device named gasbag is one of the most commonly used devices. The usage of the mentioned devices is quite simple. The pim on the device is pulled in order for two chemical substance to get reaction and the gas produced after the reaction make the device wider and wider. Additionally, mechanical working based air decks are also available for the operations. Air deck applications are used for coal production blasting, quarry blasting, metal mines production blasting from surface

and underground mines, road constructions and controlled blasting such as pre splitting, rip-rap blasting, trim blasting. It enables most blasting operations to be conducted safely because of its flexible usage.

3.4.1 Production holes

In designing the burden for production blast holes, the engineer is restricted to how much ground the hole can pull at the toe. This is the most heavily constricted area of the hole and proper fragmentation and movement in this zone is critical. At the top (or “collar”) of the hole, there is much less vertical constriction, so it follows that considerably less explosive energy is required to fragment and move the material at the top of the hole than at the bottom. Common practice to compensate for this is to use a lower energy/density explosive in the upper region of the hole. Generally speaking, there is no lower density, lower cost explosive for the upper region than ANFO, and this in fact is the one most commonly used. When blast engineers design the pattern to pull the toe with ANFO as the total charge, things get a bit more complex. Operators can use bulked out ANFO mixtures to achieve similar energy reduction in the collar zone, but the added time and effort make this impractical. Air decking offers an economical and energy efficient alternative to this practice.

The physical steps in creating an air deck are sample:

1. lower the design the top of the explosive column
2. set a barricade to the stemming material at some point above the original top of column location
3. stem from this barricade to the top of the hole, as usual

The explosive gases will be able to expand into the air deck created above the column and exert a reduced but prolonged stress in the collar zone of the hole. This can allow significant reductions in the total explosives loads in production holes without significant loss in either fragmentation or movement of the collar zone.

3.4.2 Design Criteria

The most common design criteria used to estimate an air deck volume is a percentage of the total explosive column length. When working in a fairly tight range of diameters and hole lengths, this approach works quite well. This method has the added advantage of automatically giving the operator an accurate estimate of how much will be saved in explosive costs.

The new stemming length is usually started at 75% of the original, but this dependant on the degree of risk for stemming ejection or fly rock. Gradual reductions in stemming length will determine the minimum allowable, and this can often be as little as half of the original stemming length provided good stemming material is used. Carefully controlled field tests are required for exceptional conditions (rock or explosive types, holes diameters or depths). It is recommended that these tests begin with the minimum amount of stemming and explosives removed with gradual increases in each until the maximums that can be removed are determined.

3.4.3 Pre-shearing

Pre-shearing (or “Pre-splitting”) is generally an expensive process, both from a drilling and from a blasting perspective. Pre splitting is the controlled blasting in which air deck applications are most commonly used. Pre splitting operations are one of the most commonly applied operations in mining and construction sectors. In pre splitting operations, blast holes are drilled with narrow spacing are charged with less explosive. The fact that the diameter of the blast hole drilled are narrow and the explosives used are of specialty make these operations very costly. The aim of conducting these operations is to split the rock mass to be blasted from the major rock mass in order to minimize the damage induced by blasting. In pre splitting applications, generally the detonating cord, booster and less amount of explosives are used. In this technique all blast holes are detonated at the same time. This results in high amount of noise occurrence. Moreover, stemming is sometimes not done in pre splitting operations. This increases the risk of over pressure and fly rock. Air deck

applications in pre splitting operations are carried out by charging very little explosive and then creating an air gap within a blast hole. Especially, this method is really effective in the pre splitting operations carried out in coal production blasting. The cross section of charging the pre split blast hole with air deck can be seen in Figure 3.9

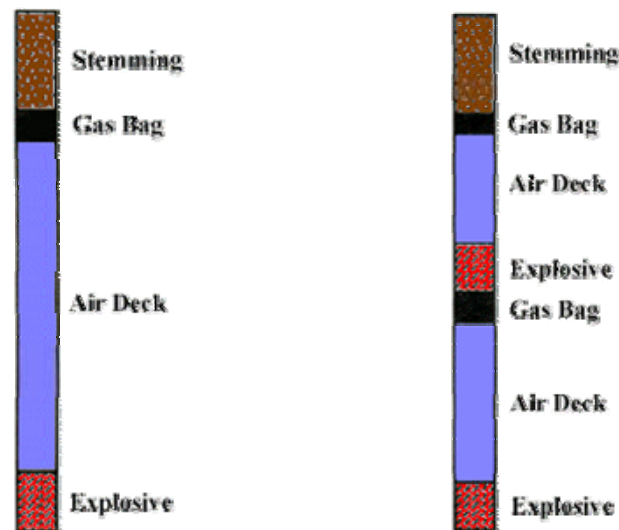


Figure 3.9 Cross section of application of air deck in pre splitting operation

According to the observations from the applications;

- The spacing is 15-24 times the diameter of the blast hole in which the air deck is applied.
- If the hole length is more than 15m, two or more charge columns should be created in order to get the better explosive distribution
- The blast holes should be stemmed in order to get good explosive productivity
- In pre splitting operations, the total charge amount should be 3%-10% times the conventional one
- 60% of the total charge is generally done in the bottom of the blast hole.

Drill factors can be as low as 15% of normal production which in turn means that the drilling cost per ton is increased over six-fold. In addition, there is a low yield of rock per man-hour loading when compared to normal production holes. In the loading process, high cost, specialty explosives are often used, or labour intensive preparations of string charges are involved.

Air decking with bulk explosives offers two methods of reducing the impact of lower drill factors and high explosive costs.

1. air deck pre-shearing is conducive to large diameter holes. Normal production diameters can be used, usually at much wider spacings than small diameter holes. This, in effect, reduces the impact of lower drill factors on drilling and loading costs.
2. bulk explosives are considerably less expensive to purchase and load than specialty or hand-prepared string charges.

3.4.4 Air Deck Applications in Production Blasting

Using the air deck technique in the production blasting operations is of great importance in terms of increasing the blast productivity and making reduction in the ground vibration intensity. However, it should not be forgotten that the air deck volume is of difference in terms of the length of air deck and the command geological structure. In the present time, air deck has been applied in different geological structures and the proper air deck length has been determined for the given mine site and then the desirable fragmentation could be achieved. the design criteria used for estimating the length of air deck is the percentage ratio of the total explosive column. This technique gives the practical information about the saving of the explosive. According to the formation, the first air deck volume is determined by decreasing the original stemming column at about 25%. The reduction in the stemming column is done by considering the overpressure and fly rock conditions. Additionally, the first air deck application should be started with the least reduction in the stemming and blast charge length. After that the gradual increase in the air

deck length should be done until the undesirable fragmentation distribution is observed.

According to the observations from the use of air deck in production blasting

- Where as 15%-35% saving in total charge is achieved, fragmentation distribution still remains at permissible level.
- Reduction in ground vibration intensity and noise amount.
- If the air deck is introduced in the middle of the blast hole, the best and most productive results can be obtained.
- 15% of the explosive amount should be removed first for as an air deck volume; where as the stemming length remains the same. According to the data obtained from blasting, the reductions should be made at 0.5%-2% and these reductions are continued until the optimal air deck volume is achieved. The loading of the production blasting with air deck can be seen in Figure 3.10.

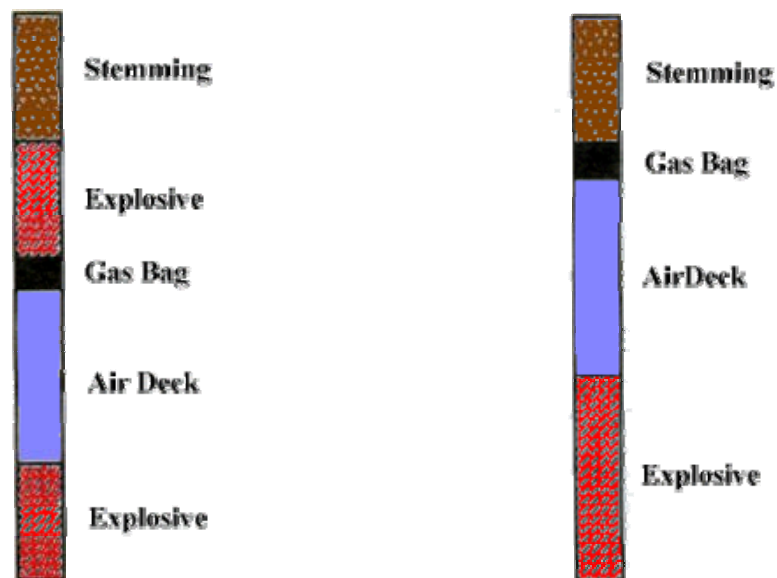


Figure 3.10 The cross section of application of water deck in production blasting

3.4.4.1 Vibration and Over-Pressure Reduction

When vibration problems are the result of charge per delay, the simple expedient of introducing air decks in production holes can help. The resultant reductions in vibration levels are primarily due to the simple reduction of charge in each hole. There is some evidence to support the claim that the air deck also acts as an “accumulator”, trapping previously wasted energy in the collar zone and converting it into useful work. This may further reduce vibration levels in the near field.

Over pressure due to stemming ejection can also be reduced through the use of air decks. In addition to effectively lowering the top of the explosive column, air decks can act as stemming enhancers. Coal operators have been able to solve stemming ejection problems by simply inflating an air bag directly on top of the explosive column before stemming the hole. A second major cause of overpressure is the transmission of the explosive shock wave from the rock surface into the atmosphere. Efficient pattern design and implementation, coupled with proper delay sequencing, can go a long way to reducing the magnitude of the shock wave introduced into the atmosphere, but cannot eliminate it. Air decking, by the simple expedient of reducing the total explosive amount, can reduce it even further.

Two surfaces must be considered. The vertical surface at the front of the shot (normally called the free face) and the horizontal surface on the top. Shock waves from the vertical face are least affected by air decking, since the air deck itself comprises a relatively small portion of the vertical section of the blast. the shock wave of the horizontal face is most affected by air decking, where the deck (1) reduces the total explosive amount, and (2) increases the distance the shock wave must travel through the rock before it reaches the surface. A third mechanism in over-pressure reduction could be the “accumulator” effect noted previously.

3.4.5 Other “Decking” Applications

As mentioned above, since the air deck applications are quite flexible, it can be used for various reasons within a blast hole. However, the air deck application cannot always be done in the blast hole. Two major problems encountered within a blast hole is the cavities due to the weak zones and the water amount within a blast hole. These mentioned situations can be eliminated by using air deck within a blast hole and then the blast holes with problems can also be detonated without facing any further problems.

3.5 Some Comprehensive Researches Regarding the Use of Air-Deck

Jhanwar and Jethwa (2000) carried out a comprehensive survey in order to interpret the use of Air-Deck in production blasting. They conducted their survey at the mine situated in central India near the city of Nagpur in the state of Maharashtra.

3.5.1 The Geology of the Mine at Issue

The mine area is a part of the eastern limb of the Wardha valley coal field and the entire area is covered by black cotton soil. The Wardha valley coal field is confined between the latitudes $E19^{\circ}30'$ and $20^{\circ}27'$ and longitudes $N78^{\circ}50'$ and $79^{\circ}49'$ in the south-eastern part of the Maharashtra state of India. The regional structure of this coal field is a broad anticline plunging towards NNW. Both the western and eastern limbs of this anticline have been proved to be coal bearing. The Kamthi formation unconformably overlies the coal bearing Barkar formation, preventing coal seams from outcropping on the surface. The Kamthi beds usually have low dips of 5° to 10° whereas the underlying Barakar beds generally have relatively higher dips of 8° . NW-SE trending normal faults are major structural features of the Wardha valley coal field. These strike faults have caused repetition of strata in many parts of the coal field. The geological succession of this area is given in Table 3.1. The upper part of the Kamthi formation is subjected to differential weathering which has given a brownish to yellowish tint to the sediments. The thickness of this zone varies from

1 to 12 m from the surface. The soil in this block is mostly black cotton and sandy soil. It varies in thickness from 0.5 to 6.0 m.

The overburden at this mine consists of coarse grained reddish ferruginous sandy soil of 2.3 m thickness, followed by fine to medium grained yellowish, white sandstone which measures approximately 35-45 m thickness. The coal seam is 17-20 m thick with a dip of 1 in 8 at S 54°30' E. The coal seam floor consists of 3.25 m thick grey sandstone. The geological section from a typical bore hole in this area is shown in Figure 3.11 The general strike of the coal seam is NE-SW and it becomes almost NS towards North. The coal seam dips due east and the amount of dip varies from 5° to 8°. In the south part of the block, the dip is steeper than in the northern part.

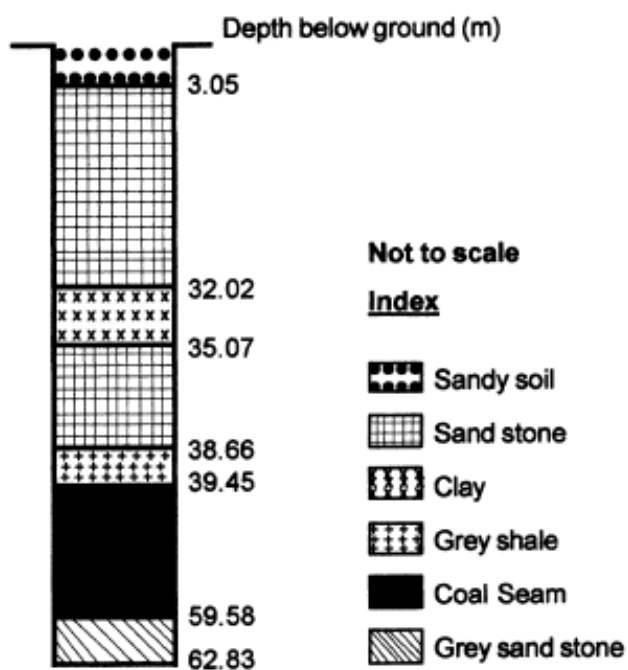


Figure 3.11 The cross section of geological structure
(Jhanwar & Jethwa, 2000)

Table 3.1 The geological succession of this area

Age	Formation	Lithology	Thickness range (m)
Recent to sub recent	Soil	Black cotton soil, sandy soil	0.50-6.00
Upper Permian to lower Triassic	Kamthi	Red, brown, yellow coloured fine to coarse grained sandstone, variegated clay.	4.77-100.77
Lower Permian	Barakar	Grey to white fine to medium grained sandstone shale, intercalation of shale & sandstone	84.00-127.19
Upper Carboniferous to Lower Permian	Talchir	Greenish sandstone, green shale.	0.31-4.00

3.5.2 Conventional Blasting Carried out at the so called Mine

In conventional blasting at this mine, solid decks of drill cuttings were placed in the blast hole to separate the explosive charges. The deck length varied from 2 to 3 m for 8-10 m and 4 to 5 m for 16 m deep holes. The charge pattern in a typical conventional blast is shown in Figure 3.12. The blast patterns used were spacing and burden at 5-6 m and 4-5.5 m respectively. The explosive used was slurry type in cartridge form. Hole depth varied from 8 to 16 m. The explosive cartridge diameter was 125 mm. The length and weight of each such explosive cartridge were 0.45 m and 6.25 kg respectively. The blast hole diameters were 150 mm and 250 mm. Explosive quantity per hole depended on the hole depth and the rock formation and varied over a wide range. It was in the range of 62.5-87.5 kg, 87.5-100 kg and 125-150 kg for the hole depths of 8 m, 10 m and 16 m respectively. Charging was done by putting 12.5-18.75 kg of primer explosive cartridges tied with detonating cord in the hole bottom followed by 18.75-68.75 kg of slurry explosive in cartridge form. This bottom charge was overlain by a 2-5 m long solid deck of drill cuttings. On the top of this deck, primer charge (6.25-18.75 kg) followed by column charge (12.5-43.75 kg) was put maintaining a stemming column of 4-6 m length in the top. Stemming was also done by using the drill cuttings (Figure 3.12). The detonating cord tied with primer explosive in the hole bottom passed through the hole length and was kept protruding at the surface for connections. Initiation was done by using electric delay detonators on the surface. No inter deck delays were used. All holes in a row were connected together by a detonating cord and were given the delay using short delay electric detonators of 25 ms. Row to row delay was thus given and the

delay interval between two successive rows was 25 for burden less than 4.5 m and 50 ms for burden more than 4.5 m. In soft and medium strength sandstones, conventional blasting produced excessive fines. The blast induced fragmentation were estimated by observing the shovel loading operation. The total blasted muck volume was estimated by the total number of dumper trips. The estimation of fines volume was made by the number of dumpers loaded by fines. It was revealed that about 60-80% of the blasted muck volume consisted of undersize/fines.

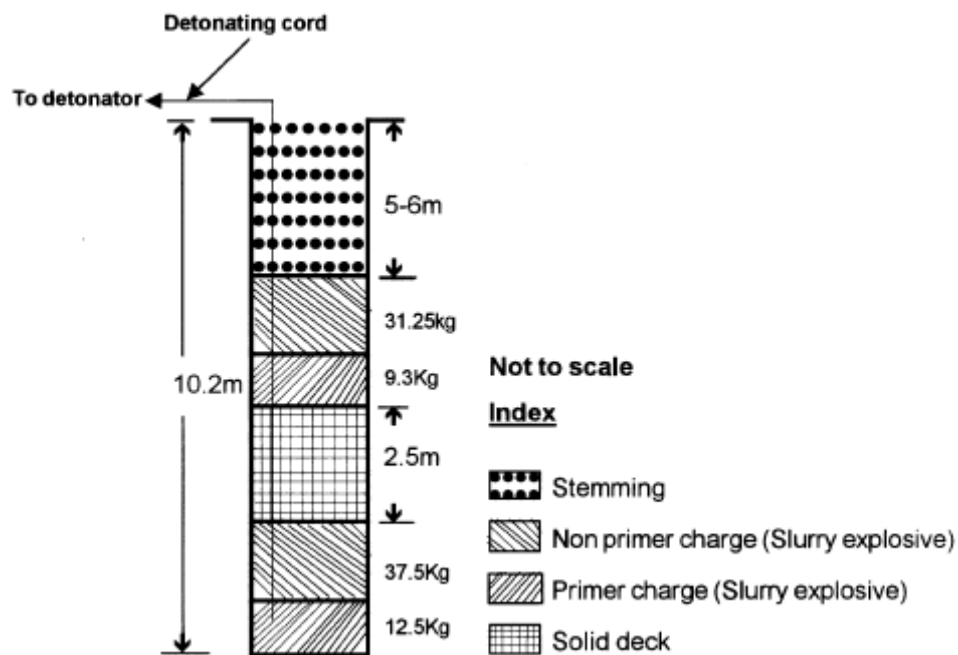


Figure 3.12 The cross section of charging of the blast hole with solid deck (Jhanwar & Jethwa, 2000)

The back break as an additional breakage behind the last row of holes was measured by steel tape. This was found to be in the range of 0.5-5.0 m which indicated wasteful utilization of explosive energy. In medium strength blocky sandstones, it produced poor fragmentation with a large number of oversize in situ boulders which required secondary blasting. Due to poor fragmentation, shovel loading efficiency was very low in these formations.

3.5.2.1 Ground Vibration in Conventional Blasting

Blast induced ground vibrations were measured at different distances using a digital seismograph for a total of nine such conventional blasts taken at different

locations in the overburden benches (Table 3.2). Analysis of vibration data was performed at 95% confidence level. The following attenuation relations were derived (Equations (3.39) and (3.40) and Figure 3.13).

$$V = 240.60(D/Q^{0.5})^{-0.62} (r = 0.87) \quad (3.39)$$

Table 3.2 Ground vibration details of conventional blasts

Blast Number	Location	Distance from blast (m)	Maximum charge per delay (kg)	Vibration (mm s ⁻¹)
1	Sector 4; (a)*	100	96.25	41.66
2	Sector 4; (c)*	35	1500	109.73
3	Sector 4; (b)*	75	900	31.26
4	Sector 4; (b)*	250	195	14.65
5	Sector 4; (d)*	250	790	26.06
6	Sector 4; (a)*	225	175	12.47
7	Sector 4; (d)*	200	450	21.36
8	Sector 4; (e)*	50	1200	106.68
9	Sector 4; (e)*	200	600	45.72

$$V = 44.009(D/Q^{0.5})^{0.62} (r = 0.87) \quad (3.40)$$

Where V is the peak particle velocity (mm/s), D is the distance from the blast site (m), Q is the maximum charge per delay (kg) and r is the correlation coefficient.

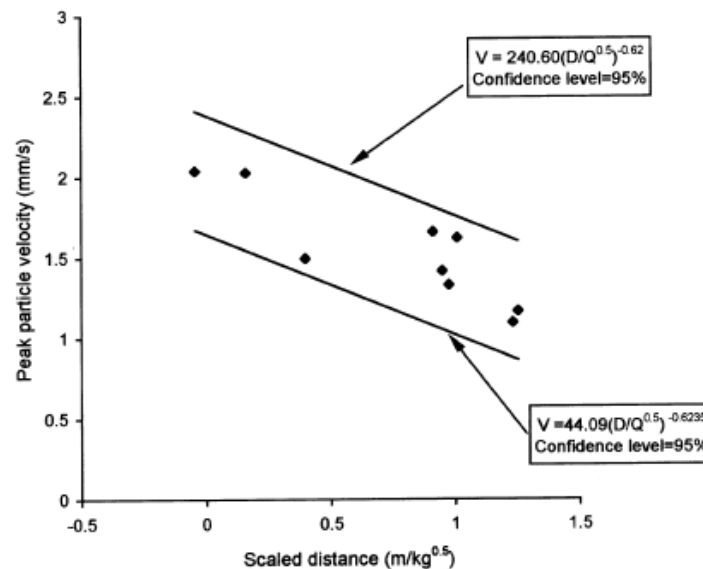


Figure 3.13 Variation in ground vibration with scaled distance in conventional blasting (Jhanwar & Jethwa, 2000)

3.5.3 Air Deck blasting

A total of ten blast trials were conducted using air decked explosive charges (Jhanwar, 1998). These blasts were conducted at different locations in the overburden benches with various combinations of spacing, burden, bench height etc. (Table 3.3). The design parameters and the performance parameters of all the ten blasts are listed in Tables 3.3 and 3.4 respectively. The number of blast hole rows in these blasts depended on the total number of holes and these varied between 2 and 7. These blasts were monitored for fragmentation, throw, backbreak, ground vibration, powder factor and explosive consumption. Air decks were introduced in the blast hole in the middle of the explosive column by lowering a dumbbell shaped wooden spacer (Figure 3.14). Explosive quantity per hole depended on the hole depth and the rock formation and varied over a wide range, as shown in Table 3.3. Charging was done by first downloading the 12.5-18.75 kg of primer explosive cartridges tied with detonating cord in the bottom followed by 12.5 to 37.5 kg of column charge. This was overlain by an air-deck of 0.5-3.0 m length depending on the hole depth and the rock formation (Table 3.3). Above this air-deck, 6.25-12.5 kg of primer explosive cartridges were placed followed by a 18.75-37.5 kg of column explosive charge cartridges. A stemming column of 5-6 m length was maintained in the top using the drill cuttings. The detonating cord tied with primer explosive in the hole bottom ran through the hole length and was kept protruding the surface. The charge section and air deck placement in a typical air deck blast are shown in Figure 3.15. Initiation was done by using electric delay detonators on the surface. No inter deck delays were used. All holes in a row were connected together by a detonating cord and were given the delay using short delay electric detonators of 25 ms. Row to row delay was thus given and the delay interval between two successive rows was 25 ms for burden less than 4.5 m and 50 ms for burden more than 4.5 m. Ground vibrations induced by these air-deck blasts were measured at different distances using digital seismographs (Table 3.3). Analysis was done to derive attenuation relations which include 95% confidence interval (Equations (3.41) and (3.42) and Figure 3.16).

Table 3.3 Design parameters of air deck blast (Jhanwar & Jethwa, 2000)

Blast Number/ Location	Bench height (m)	Hole depth, dia, (m, mm)	Number of total holes number of rows	Spacing × Burden (m×m)	Charge/hole (kg)	Maximum charge/delay (kg)	Air-deck length (m)	Air-deck length/ original charge length (ADF)
1/Sector 4 (a)*	10.0	10.2, 250	8/3	6×5.5	68.75	206.25	2.0	0.31
2/Sector 4 (a)*	10.0	10.2, 250	8/2	5.5×5.5	56.25	225	3.0	0.47
3/Sector 4 (b)*	10.5	10.5, 250	12/3	7×6	62.50	312.5	3.0	0.46
4/Sector 4 (d)*	15.0	150, 150	7/2	6×4.5	96.50	385.5	3.0	0.30
5/Sector 4 (c)*	15.0	150, 150	12/4	7×4.5	83.3	250	3.0	0.35
6/Sector 4 (c)*	10.5	10.0, 250	22/3	6×4.5	750	1200	1.5	0.20
7/Sector 4 (d)*	8.0	7.5, 150	51/7	5.0×4.5	500	1350	2.0	0.41
8/Sector 4 (b)*	16.0	15.5, 250	17/3	5.5×6.0	137.5	750	2.0	0.17
9/Sector 4 (e)*	8.0	6.5–8.0, 150	28/2	5–6×5–9	46	750	0.6	0.42
10/Sector 4 (e)*	8.0	6.0–8.0, 250	14/3	6×6.0	87.5	525	2.0	0.28

Table 3.4 Performance parameters of air deck blast(Jhanwar & Jethwa, 2000)

Blast Number/ Location	Powder factor (kg m ⁻³)	Fragmentation	Back break (m)	Throw (m)	Toe	Vibration, Distance (mm s ⁻¹ , m)	Explosive savings(%)
1/Sector 4 (a)*	0.20	Good	0.2–0.5	8	Nil	—	22
2/Sector 4 (a)*	0.18	Fair	Almost nil	6	Nil	17.27, 80	36
3/Sector 4 (b)*	0.14	Poor	Almost nil	4	Nil	30.48, 75	30
4/Sector 4 (d)*	0.24	Very good	0–3	8	Nil	7.25, 100 and 1.97, 150	30
5/Sector 4 (c)*	0.27	Satisfactory	Almost nil, 0.4 m on one side	10	Nil	2.67, 140 and 12.621, 90	30
6/Sector 4 (c)*	0.27	Good	Clean face	10.5	Nil	7.62, 120	25
7/Sector 4 (d)*	0.29	Good	0.3–0.5	12	Nil	21.08, 90	20–25
8/Sector 4 (b)*	0.27	Excellent; Ideal muck pile	0–0.2	9	Nil	30.48, 55	10–15
9/Sector 4 (e)*	0.16	Excellent; Ideal muck pile	0.1–0.5	5–7	Nil	6.86, 110	25–30
10/Sector 4 (e)*	0.30	Excellent; Ideal muck pile	Almost nil	15	Nil	48.77, 40	10

$$V = 183.27(D/Q^{0.5})^{-1.57} (r = 0.80) \quad (3.41)$$

$$V = 59.66(D/Q^{0.5})^{-1.57} (r = 0.80) \quad (3.42)$$

Where V is the peak particle velocity (mm/s), D is the distance from the blast site (m), Q is the maximum charge per delay (kg) and r is the correlation coefficient.

3.5.3.1 Influence of Air Deck Blasting on Blast Performance and Economics

The impact of air deck blasting on blast performance and economics was evaluated in terms of various parameters like fragmentation, backbreak, throw and ground vibration as detailed below. This was done by comparing these parameters from air deck blasting with those from conventional blasting (Jhanwar, 1998).

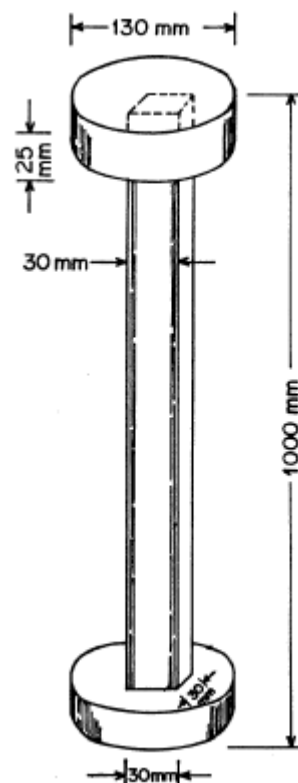


Figure 3.14 Wooden spacer for air decking (Jhanwar & Jethwa, 2000)

3.5.3.1.1 Fragmentation Assessment of blast induced fragmentation was made by studying the shovel loading operation and visual analysis of muck piles at different stages of loading operation. The total blasted muck volume was estimated by the total number of dumper trips. The estimation of fines volume was made by the number of trucks loaded by fines. In soft and weathered sandstones, fines produced in conventional blasting were estimated to be of the order of 60-80% of the total muck volume which indicated wasteful utilisation of explosive energy. Air deck blasting in such formations produced fine at only 20-30%. The fines were thus reduced by about 70%. In medium strength blocky sandstones, conventional blasting produced in situ boulders of the order of 60-80% of the muck volume, and approximately 50% of these required secondary blasting. The boulders produced were physically measured for their approximate volumes.

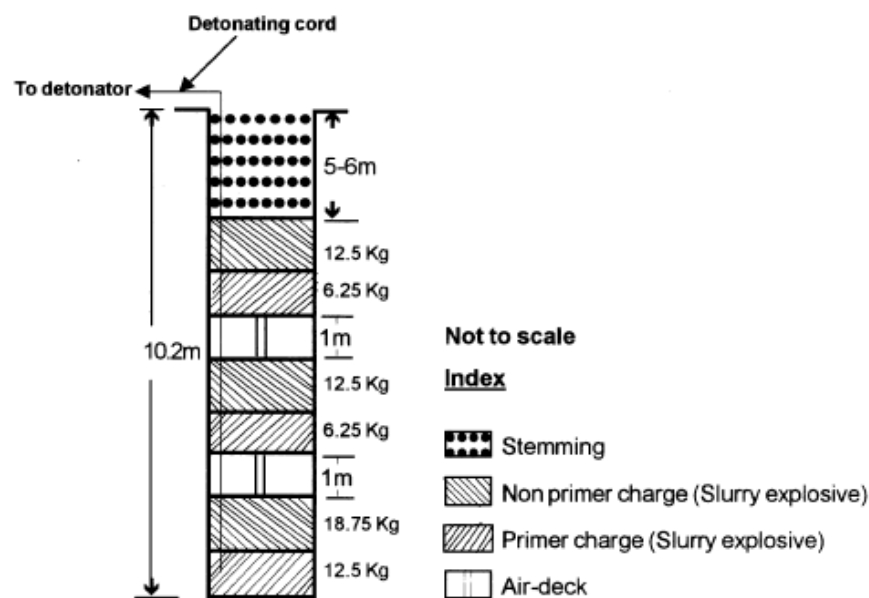


Figure 3.15 Charge pattern and air deck in atypical air deck blast (Jhanwar & Jethwa, 2000)

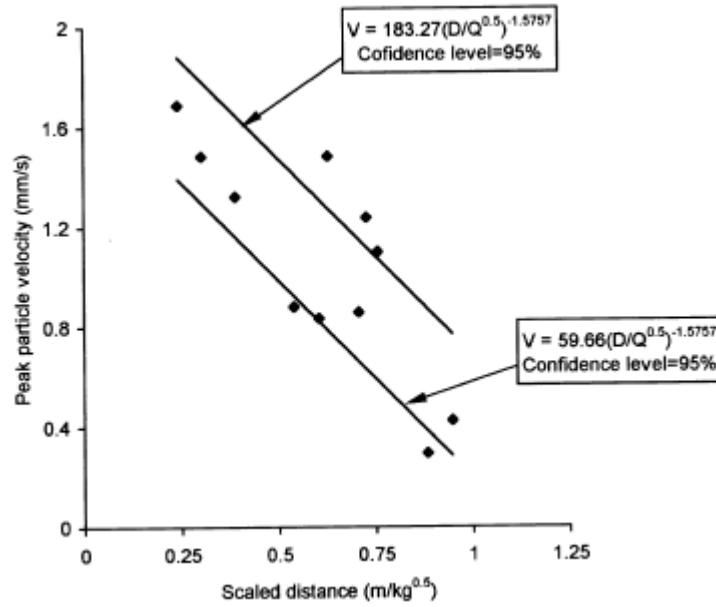


Figure 3.16 Variation in ground vibration with scaled distance in air deck blasting (Jhanwar & Jethwa, 2000)

Air-deck blasting in such formations reduced these in situ boulders by 50-70%. Due to improved fragmentation in these formations, shovel loading efficiency in terms of volume handled per hour was improved by 20-40% as revealed by the time study of shovel operation. The improvement in fragmentation due to air deck blasting can be explained by two facts. First, due to the reduction in explosive charge, the shock energy responsible for crushing is significantly reduced. Second, the air-gap separating explosive charges in the blast hole allowed expansion of explosion products into the air gap and produced repeated oscillations of shock waves in the medium. This in turn caused a reduction in initial borehole pressure and an increase in duration of the shock wave action on the surrounding rock mass. In a particular rock mass and blast design environment, the air-deck length (ADL) which replaced the original charge length (OCL) influenced the fragmentation significantly. This parameter has been designated here as air deck factor (ADF) and has been calculated as below (Equation (3.43)).

$$ADF = \frac{AirDeckLength(ADL)}{OriginalChargeLength(OCL)} \quad (3.43)$$

Where OCL is the sum of ADL and the charge length. As shown in Table 5, ADF in air deck blast trials varied from 0.17 to 0.47. In blast numbers 2, 3, 7 and 9, ADF was high at 0.47, 0.46, 0.41 and 0.42 respectively. In all other blasts, it varied from 0.17 to 0.35. It was significant to note that blasts with higher ADF produced poor or fair fragmentation and inadequate burden movement. In contrast, blasts with ADF below 0.35 produced good to excellent fragmentation. In blast number 9, despite a higher ADF at 0.42, it produced very good fragmentation probably because of very soft and weathered sandstones. Hence, for optimum fragmentation, ADF should preferably be within a range of 0.10 and 0.35. The lower value is appropriate for blocky rocks and the higher value is appropriate for softer rocks.

3.5.3.1.2 Backbreak Backbreaks (identified as an additional breakage beyond the last row of holes) were measured by a measuring tape for both conventional and air deck blasts. Air deck blasting significantly reduced or in some cases even eliminated the backbreak. In conventional blasting, the backbreak varied from 0.5 to 5.0 m, whereas in air deck blasting, it varied from 0.15 m to 0.5 m. In blasts, 2, 3, 5, 6 and 10, clean the stable faces were produced (Table 3.4). Backbreak in air deck blasting was thus reduced by 50-80% as compared to backbreak in conventional blasting. Air deck blasting, could therefore, be effectively used as a controlled blasting technique to obtain stable faces.

3.5.3.1.3 Throw Forward movement of the blast induced muck pile from the bench face was considered as the throw and it was measured by a tape for each blast. The throw of fragmented material was perfectly contained within the bench width in all air deck blast trials. In air deck blasts, it varied from 4 to 15 m, whereas in conventional blasts, it varied from 4.5 to 22 m. The throw was thus reduced by 10-35% as compared to conventional blasting. Throw was found to be related to the powder factor in the case of air-deck blasting (Equation (3.44) and Figure 3.45) (Jhanwar & Jethwa, 2000).

$$T = 49.27 * PF - 2.47(r = 0.9) \quad (3.6)$$

Where, T is the throw (m), PF is the powder factor (kg/m^3), and r is the correlation Coefficient. The throw of fragmented rock during blasting is caused by the gas energy component of explosive energy. In an air deck blast, due to the use of reduced charge and due to the expansion of gaseous products in an air gap, the available gas energy is also reduced and hence the effective throw of material.

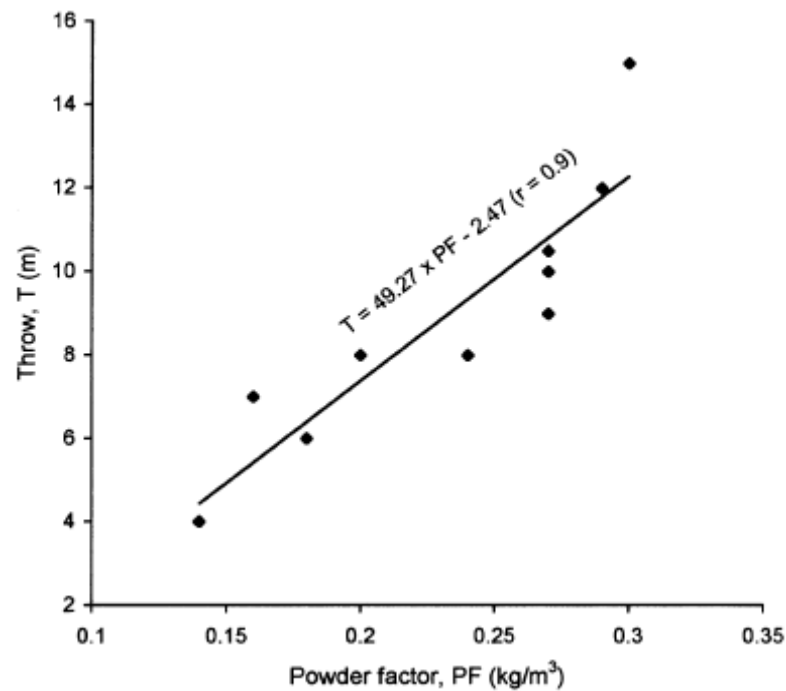


Figure 3.17 Variation of throw with powder factor in air deck blasting (Jhanwar & Jethwa, 2000)

3.5.3.1.4 Ground vibration To evaluate the impact of air-deck blasting on ground vibration, Equations (3.1) and (3.3) corresponding to upper bounds of 95% confidence interval in conventional blasting (Figure 3.13) and air-deck blasting (Figure 3.16) were used to predict ground vibrations for various combinations of distance (D) and maximum charge per delay (Q) in respect of conventional and air-deck blasting respectively as shown in Table 3.5. The results so obtained were used to derive per cent reduction in ground vibrations due to air deck blasting for various combinations of D and Q (Figure 3.18). It can be seen that ground vibration reduces in the range of 30–94%. It is interesting to note that at a particular Q , the reduction in ground vibration in air deck blasting increases with an increase in D and almost

stabilizes at a D of 300 m. At a particular D, the vibration reduction potential of air deck blasting increases with a decrease in Q.

Table 3.5 Predicted ground vibrations for conventional and air deck blasting (Jhanwar & Jethwa, 2000)

Distance (m)	Maximum charge/delay (kg)	Scaled distance (m/kg ^{0.5})	Ground vibration (air-deck) (mm/s)	Ground vibration (conventional) (mm/s)
50	250	3.2	29.9	117.4
100	250	6.3	10.0	76.2
150	250	9.5	5.3	59.2
200	250	12.6	3.4	49.4
250	250	15.8	2.4	43.0
300	250	19.0	1.8	38.4
50	500	2.2	51.6	145.7
100	500	4.5	17.3	94.6
150	500	6.7	9.1	73.4
200	500	8.9	5.8	61.4
250	500	11.2	4.0	53.4
300	500	13.4	3.1	47.7
50	750	1.8	71.0	165.3
100	750	3.7	23.8	107.3
150	750	5.5	12.6	83.3
200	750	7.3	8.0	69.6
250	750	9.1	5.6	60.6
300	750	11.0	4.2	54.1
50	1000	1.6	89.0	180.8
100	1000	3.2	29.9	117.4
150	1000	4.7	15.8	91.2
200	1000	6.3	10.0	76.2
250	1000	7.9	7.1	66.3
300	1000	9.5	5.3	59.2
50	1500	1.3	122.6	205.2
100	1500	2.6	41.1	133.2
150	1500	3.9	21.7	103.4
200	1500	5.2	13.8	86.5
250	1500	6.5	9.7	75.2
300	1500	7.7	7.3	67.1

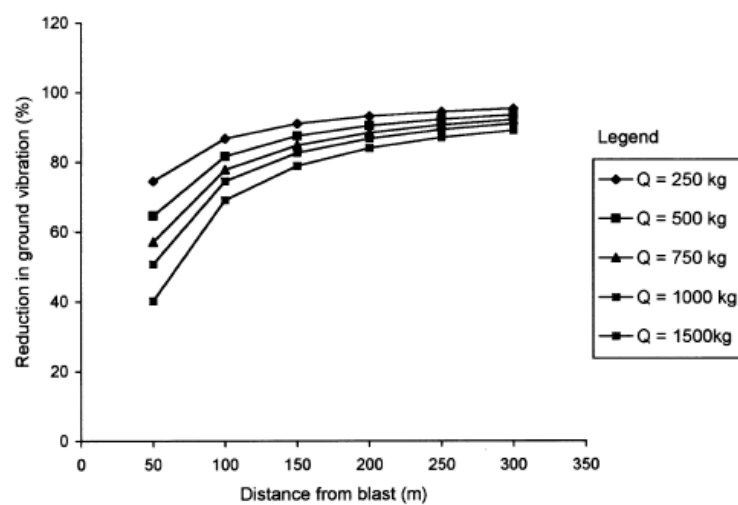


Figure 3.18 Reduction in ground vibration in air deck blasting (Jhanwar & Jethwa, 2000)

In this particular study, the amount of explosive per delay was less in air deck blasting as compared to conventional blasting. Vibrations are, therefore reduced in air deck blasting as a result of the reduced mass of explosive per delay. These conclusions are based on a small number of observations. However, the trends in Figures 4 and 7 on which the results are based do show clear trends.

3.5.3.1.5 Blast economics The use of air deck blasting at this mine induced explosive savings of the order of 10-36% with an average saving of 20% as compared to conventional blasting. Explosive loading in each holes was reduced from 15 kg to 40 kg depending on the hole depth and on the rock formation. Softer formations allowed higher explosive savings in the range of 20-36% as compared to harder formations where it varied between 10% and 20%. Annual savings in the explosive cost at this mine were estimated to be at Indian Rupees 4.46 million (US\$ 0.1 million). As a result of reduced explosive consumption, overall powder factor at this mine was reduced from approximately 0.35 kg/m³ to approximately 0.25 kg/m³. In addition to the above benefits, improved fragmentation on account of air deck blasting in blocky sandstones, resulted in improved shovel efficiency by 20-40%. (Table 3.6)

Table 3.6 Impacts of air deck blasting on blast performance and economics

Serial number	Parameter	Improvement
1	Fragmentation	● Fragmentation becomes uniform ● Fines reduced by 70%
2	Backbreak	Reduced by 50-80%
3	Throw	Reduced by 10-35%
4	Ground vibration	Reduced by 30-94%
5	Explosive cost	Reduced by 10-35%
6	Powder factor	Reduced from 0.35 kg m ⁻³ to 0.25 kg m ⁻³
7	Shovel loading efficiency	Improved by 20-40% in blocky formations

CHAPTER FOUR

ENVIRONMENTAL EFFECTS OF BLASTING CARRIED OUT AT OPEN-CAST MINES

4.1 Introduction

Ever since black powder was first used in mining applications for blasting in the early 17th century there have been undesirable side effects of rock breaking. The problem of so-called undesirable side effects increased in scale considerably in 1960 when mass blasting was made more accessible through the creation and use of Ammonium Nitrate Fuel Oil (ANFO) type explosives (Atlas Powder Company, 1987). Side effects are caused by the sudden release of large amounts of energy stored within the explosives. The majority of the energy created by the explosives is designed to crack, break, fragment and displace the rockmass and allow for the following stages of mining and processing. However, a significant amount of the energy is converted into ground vibrations as well as noise, fly rock, gas and fumes. Mining operations have a continuing trend to increase in scale and thus blasts at mines must also increase in magnitude to accommodate for the overall increase of the operation. In conjunction with this trend of continually increasing blast sizes and the associated vibrations the constraints placed on mines to protect the environment and surrounding communities are becoming more stringent (DYNO Nobel et al, 2001). Due to these fact vibrations will continue to be an issue if not become a more significant issue. Thus preventative measurements must be taken to reduce vibration levels.

The following literature review in Table 4.1 gives an understanding of the basic concepts of environmental effects of blasting such as ground vibration, fly rock, noise, air blast, dust and fumes.

Table 4.1 Unwanted effects of blasting in mines (Raina et. al. 2004)

Effect	Quantity	Unit	Symbol	Damage Potential	Prediction descriptors
Ground vibration	Peak particle velocity	mm/s	V_{max}	Structural and human response annoyance	Scaled distance maximum charge per delay
Air-overpressure	Over-pressure	Pa	P	Structural and human response annoyance	Same as above
Noise	Loudness	$dB(A)$	-	Annoyance	Level of noise
Flyrock	Throw and size of fragment	m, kg	-	Serious, fatal incidents and damage to equipment thus resulting in human annoyance and reduced confidence of people living in and around the mines	Initial velocity of the fragment
Dust and toxic fumes	Parts present per unit volume of air	ppm	-	Hazardous to health	Measured in terms of suspended particulate matter and noxious fumes

4.2 Noise and Air Blast

Noise from mining is a common source of community concern because operational noise emissions frequently occur on a continuous basis. This can interfere unreasonably with day to day activities, particularly concentration, recreation and sleep, and result in an adverse impact on residential amenity. Vibration and airblast from blasting can lead to community concern primarily due to the fear of structural damage. This fear occurs because people are able to detect vibration at levels which are well below those which result in even superficial damage to buildings and items of heritage value (Needham & Brooks, 2005)

The term air over pressure is often used to describe air waves that are generated by blasting activities. Oriard (1996) simply defines this as the pressure over and above that of the atmospheric pressure that is always present. This air pressure can

also be described as air wave that passes through the atmosphere. Air waves are compressed waves that travel through the air. When air blast is within the range of hearing it is called “sound” (20-20000 Hz). When its frequency is below the range of hearing, it is generally referred to as “concussion” or “air blast” (less than 20Hz). Air blast radiates outwards from the blast site in a similar manner to ground vibration, but at a slower rate. The time between the arrival of the ground vibration and air blast depends on the distance from the blast. At one kilometer, the air blast arrives approximately 2.5 seconds after the ground vibration. People may experience the blast as two separate events, such as air blast and ground vibration. The amount of “noise” perceived by the human ear depends on a) the mass of explosives detonated at any one moment of time, and b) the distance between the listener and the point of detonation.

4.2.1. Measuring the Air Blast

Sound is measured in decibels (dB). When measuring environmental noise, a weighting network is used, which filters the frequency of the sound so that it corresponds to the response of the human ear. Noise measurements made using this filtering network are expressed as dBA. The decibel scale is logarithmic to manage the enormous range of sound pressures able to be detected by the human ear. Airblast is measured in decibels but no weighting scale is used so the results are expressed as dB Linear (Richards & Moore, 2006). The air pressures are measured to determine the effect of air blast energy on structures (L scale) or as perceived by the human ear (A scale) (Chen & Huang, 2001). Based on work carried out by Siskind (1980), monitored air blast amplitudes up to 135 dB are safe, provided the monitoring instruments is sensitive to low frequencies (down to 1 Hz). Persson (1994) has published the following estimates of damage thresholds based on empirical data as shown in Table 4.2.

Table 4.2 Blast induced air overpressure limits proposed by Perrson

120 dB	Threshold of human pain for continuous sound
> 130 dB	Resonant response of large surfaces such as roofs and ceilings. Complaints start
150 dB	Some window may be broken
170 dB	Most window may be broken
180 dB	Structural damage to buildings

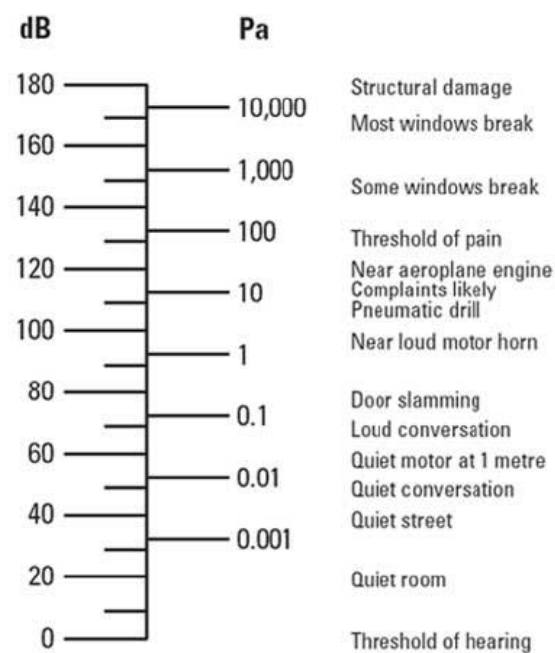


Figure 4.1 Examples illustrating the decibel (A) scale

Table 4.3 Decibel A scale

20 dB	Quite countryside
40 dB	Living room
60 dB	Conversational speech
80 dB	Busy road, loudly conversation
90 dB	Haul truck at 7m
100 dB	Pneumatic hammer, rock breaker at 7m
110 dB	Rock band, production blast at 100m
120 dB	Threshold of discomfort
140 dB	Threshold of pain, jet plane at 100m

4.2.1.1. Estimation of Air Blast Via an Equation

The air pressure intensity can be estimated by the equation similar to one used for estimating the ground vibration intensity. The formula given below, which is used extensively in most of the investigations, has been used as a predictor for the estimation of the air pressure intensity;

$$P = K \left(\frac{R}{(Q)^{0.33}} \right)^B \quad (4.1)$$

P = pressure (kPa)

K = state of confinement

Q = Maximum instantaneous charge per delay

R = Distance from charge (m)

$R/(Q)^{0.33}$ = is defined as a scaled distance

B = constant related to the rock and site (usually -1.2)

Typical K factors: Unconfined = 185, fully confined = 3.3

The US Office of Surface Mining has presented the following regulatory limits for air blast from blasting (depending on the low frequency limit of the measuring system);

<132 dB (Linear) at 2Hz or lower

<130 dB (Linear) at 6 Hz or lower

<133 dB (Linear) at 2.0 Hz high pass system

Also the standards proposed by USBM are shown in Table 4.4.

Table 4.4 U.S. Bureau of Mines Standards for Air Concussion

Instrument Frequency Range	Maximum dB Level At Closest Structure
0.1 to 200 Hz. Flat response	134 dBL
2.0 to 200 Hz. Flat response	133dBL
6.0 to 200 Hz. Flat response	129dBL
C-Weighted Slow Response	105 dBC

Major causes of air blast overpressure are;

- Blowouts in the face
- Insufficient stemming in the blast hole or overcharging
- Incorrect stemming material or not enough stemming retention time.
- Detonating cord used as downlines in the blast hole.
- Detonating cord used as surfaced trunklines (often used when cast blasting)
- Detonating cord used as a lead-in to the shot.
- Incorrect inter-row timing which chokes the shot, causing blowouts on the surface of the shot. (Burrow, 2006)

How to control noise and air blast

- Minimize charge per delay of explosive
- Use a hole spacing and burden which will ensure that the explosive force is just sufficient to break the ore to the required size.
- Eliminate exposed detonating cord and secondary blasting. In the event that an explosive detonating cord is used to detonate the blast holes, it should be covered with a suitable aggregate material
- However, the potential for initiation related air blast emissions can be minimized with the use of nonel (non electric) initiation systems.
- Orient quarry faces away from potentially sensitive receivers.

- Ensure stemming depth and type is adequate
- Restricts blast to favorable weather conditions. Wherever possible blasting should be confined to between 09:00 hours to 17:00 hours to minimize the noise enhancing effects of temperature inversions.
- Avoid shallow blasting or secondary blasting.
- Use barrier between blast area and reception points.
- Minimize frequency of blast.

4.3 Ground vibration

One of the most controversial issues facing the mining, quarrying and construction industry is ground vibration. There are quite a number of legal cases with these industries and citizens in close proximity to their operations, some issues at times are a bit fictitious while other stem from ignorance. Some of these complaints and alleged blast damage structures are exaggerated as a result of limited knowledge about the subject (Dowding 1996). However, these complaints and issues are likely to continue as mining companies are trending towards large blasts to maximize the use of large scale equipment. The problems of blast vibrations are therefore likely to escalate in and around residential areas. The onus is therefore on the blaster to design blasts in such a way to minimize ground vibration and maximize production.

4.3.1. Generation of ground vibration

When an explosive charge is detonated in a blast hole, instant dynamic stresses are set around it due to sudden acceleration of rock mass by detonating pressure on borehole walls. The stress wave transmitted to surrounding rock sets up a wave motion in the ground. The strain energy carried by these waves breaks the rock by different breaking mechanisms such as crushing, radial cracking and reflection breakage in the presence of a free face. The stress wave propagates through the medium in the form of elastic waves, oscillating the particle through which it travels. These waves in an elastic zone are known as PPV, which closely confirms to the

visco-elastic behavior. The wave motion spreads concentrically from the blast site in all the directions and gets attenuated due to spreading of fixed energy over an origin. The intensity of ground vibrations depends on various parameters given before but all can be broadly divided into two categories namely controllable parameters and uncontrollable parameters. Controllable parameters are mainly related to explosive characteristics and blast design parameters, which can be changed by mine administration. But on the other hand, some parameters, which are natural and cannot be controlled, are geological conditions, rock characteristics and structural setup of the ground.

The most important controllable parameters related to ground vibrations are classified into two categories (Siskind et al., 1980). The blast geometry parameters have a direct relation with the ground vibration and generate higher vibration with increasing values. But other three have an interrelationship and that is why these are studied collectively. It was found that if burden is higher than the gases produced to detonation of explosive charges to properly fragment and move the rock forward. These gas confined in the bottom of the hole more than the optimum time. The energy of the explosive not utilized in fragmentation and throw will cause an increase in ground vibration. It has been proved that spacing and burden ratio more than one increases fragmentation and if more energy is consumed in fragmentation then less energy is released to generate ground vibration. The main objective of stemming is to confine the explosive gasses, so that they have enough time to fracture the ground. A stemming length, which is more than critical length, may enhance the ground vibrations due to long confinement and high pressure, developed in boreholes (Hagan and Kennedy, 1977).

So far, as explosive parameters are concerned, explosive blasted per delay is the most important parameter. The ground vibration characteristics depend on maximum charge per delay for any delay interval, instead of total explosive blasted (Nichols et al., 1971). Delay interval is the lag in firing timing between two consecutive detonations in blasting round. Dick et al. (1983) reported 8 or 9 ms minimum delay interval that can be used between charges if they are to be considered as separate

charges for vibration purpose. The explosives which generate higher gaseous energy than shock energy such as ANFO are more likely to induce higher amplitude air blast overpressures and lower ground vibrations. Briefly, high velocity of detonation explosive generates high intensity ground vibration, and low velocity of detonation explosive generates low intensity ground vibration.

The main uncontrollable parameters are rock characteristics and site geology. The amplitude, frequencies and duration of the vibration change as it propagates, because of interaction with various geological medium and structural inferences, spreading out the wave train through dispersion and absorption which is greater for higher frequencies. Wiss and Linehan (1978) investigated the influence of local geology, lithology and rock characteristics and found moderate influence on ground vibration characteristics. The mechanical properties of rock like compressive strength, tensile strength, modulus of elasticity, Poisson's ratio, dynamic strength and propagation response of wave show distinct influence on ground vibration. The frequency of joints and other joint geometry parameters in geological media also affect the ground vibration. The effect of discontinuities is well known on the ground vibration (Fourney et al., 1993). It was shown that the direction of strata gradient also influences the ground vibration.

4.3.2. Peak Particle Velocity

As the seismic waves travel through the rock, there are movements of the particles. This is commonly referred to as vibration. The motion of the ground particles (vibration) occurs in three dimensions which are vertical, longitudinal and transverse. Transverse ground vibrations agitate soil/rock particles in a side to side motion. Vertical ground vibrations agitate particles in an up and down motion. Longitudinal ground vibrations agitate particles in a forward and backward motion progressing outward from the blast site (Chen & Huang, 2001). When there is vibration each particle has a velocity and the maximum velocity is referred to as the peak particle velocity. This motion is usually recorded by the use of a seismograph and the maximum velocities of all three directions are given. The particle by most is

to use the reading of the peak particle velocity as the standard for measuring the intensity of ground vibration. In reporting, the maximum measurement of any of the three components combined. In most cases the PPV is closely linked to the potential to damage structures rather than the acceleration or displacement of the rock. A standard unit is used to measure this peak vibration in millimeter per second (mm/s). (Nicholson, 2005)

4.3.3. Ground Vibration Waves

Ground vibration radiates outwards from the blast site and gradually reduces in magnitude; in the same manner as ripples behave when a stone is thrown into a pool of water, schematically shown in Figure 4.2. The motion of the wave can be defined by taking measurements of a float on the surface of the water.

Ground vibration wave motion consists of different kinds of waves;

- Compressional (or P or longitudinal) waves
- Shear (or S or secondary or transverse) waves
- Rayleigh (or R) waves and Love (or L) waves

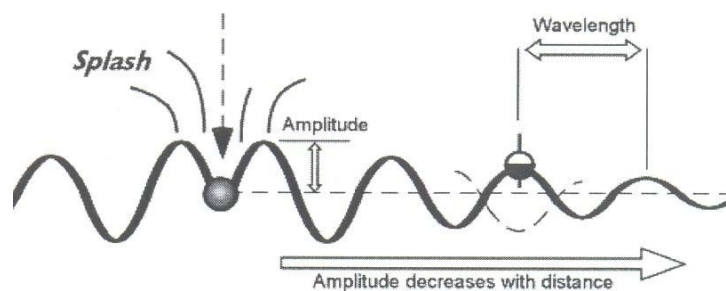


Figure 4.2 Schematic diagram of vibration terminology (Richards & Moore, 2006)

4.3.3.1. Wave Propagation

When a rockmass is suddenly impacted, vibration waves will propagate away from the location in expanding spheres (Figure 4.3). The waves will continue to

expand in this method via the path of least time, not necessarily the path of least distance (Fermat's Principle). The waves will continue to propagate away from the blast site until they have dissipated all of their energy into the surrounding rockmass or reach a material they cannot propagate through. The major factors associated with the loss of energy of vibrations are geometric spreading, discontinuities and inelasticity of the rock mass (Atlas Power Company, 1987).

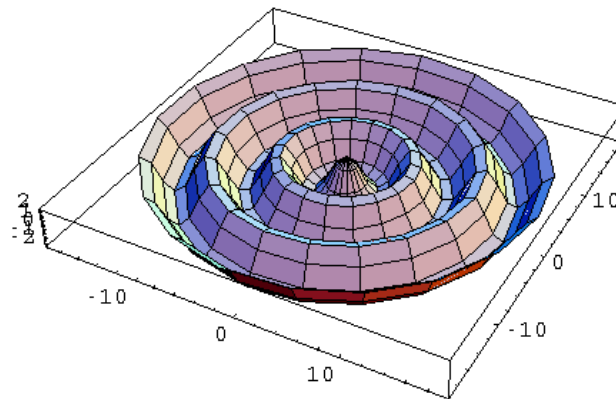


Figure 4.3 Schematic diagram of propagation of ground vibration

4.3.3.2. Longitudinal Waves

Longitudinal waves are one of the two types of body waves produced from blasting. They are the fastest traveling of all the waves and can travel through all mediums. They are also known as primary, compression or P waves. The simplest illustration of the motion of the particles within the 'P' wave is to consider a long steel rod struck on the end. The particles of the rod move to and fro as the compressive pulse travels along the rod. The particles in the wave move in the same direction as the propagation of the wave. The 'P' wave moves radially from the blast hole in all directions at velocities characteristic of the material being traveled through. The wave motion of 'P' wave is illustrated in Figure 4.4.

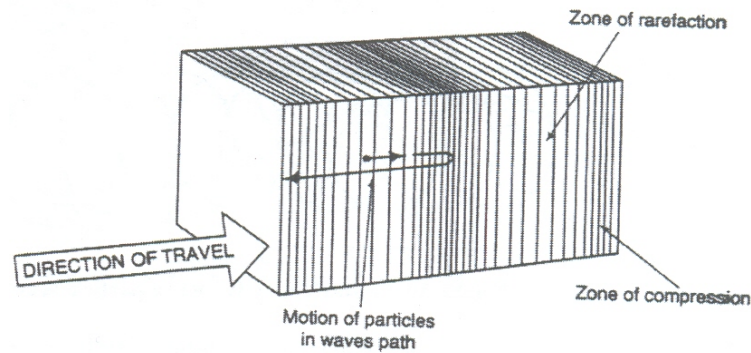


Figure 4.4 Compressional “P” wave particle motion (Richards & Moore, 2006)

4.3.3.3. Transverse Waves

Transverse waves are also commonly known as shear, secondary or S-waves and are the other type of body wave that can be produced from blasting. These waves can only travel through solid materials and are commonly created when P-waves come into contact with a free surface, irregularities or a change of material (Atlas Power Company, 1987). S-waves can cause shear deformation as they travel through a material due to the fact that individual particles move perpendicular to the direction of wave propagation. The shear or ‘S’ wave travels at approximately 1200m/s (50% to %60 of the velocity of the ‘P’ wave). The motion of the particles within the wave can be illustrated by shaking a rope at one end. The wave travels along the rope, but the particles within the wave move at right angles to the direction of motion of the wave. The wave motion of ‘S’ waves is illustrated in Figure 4.5. The ‘P’ waves and ‘S’ waves are sometimes referred to as “body waves” because they travel through the body of the rock in three dimensions.

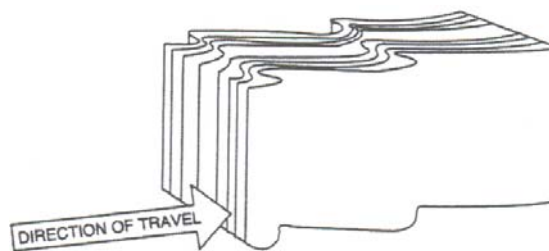


Figure 4.5 Shear “S” wave particle motion (Richards & Moore, 2006)

4.3.3.4. Rayleigh and Love Waves (Surface Waves)

Surface waves are not created directly from blasts however are the result of interaction between the body waves and the surface. As their name indicates these waves propagate on the surface. Their magnitude decreases approximately exponentially with depth. Surface can be split into two more specific groupings; these are Rayleigh waves and Love waves. Rayleigh waves have a vertical and horizontal component of particle movement and are similar to the circular motion of waves in the ocean (Figure 4.6). It propagates more slowly (750 m/s) than the body waves.

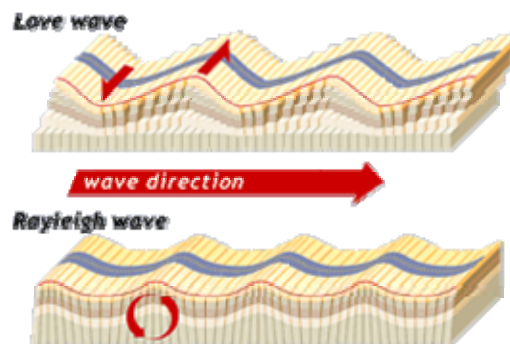


Figure 4.6 Rayleigh and love wave (Faultline, 1999)

Love waves consist only of horizontal motion with the movement of the individual particles perpendicular to the direction of propagation of the wave these waves are confined to the shallow surface zone. In general S-wave travels at a velocity of $\frac{1}{2}$ to $\frac{1}{3}$ of that of the P wave and a rayleigh wave at a velocity of 0.9 to 0.95 of that of the S wave. Table 4.5 shows velocities of P and S waves in different rock mediums.

Table 4.5 Typical P and S wave velocities of rock (Nicholson, 2005)

Material	P-velocity, m/s	S-velocity, m/s
Granite	4000-6000	2000-3000
Basalt	5500	3000
Sandstone	2000-3500	1000-2500
Limestone	3000-6000	2000-3000
Schist	4000-5000	2500-3000
Soil	150-1000	100-700

4.3.4. Vibration Prediction

There are a range of different vibration prediction methods that are commonly used the most common methods of vibration prediction are Scaled Distance Equations. Scaling is the designation of relationships correlating ground vibrations to various distances (Atlas Powder Company, 1987). These relationships are based on predicting the peak particle velocity using, instantaneous charge weight, distance from the blast and rock mass properties, the three main variables affecting vibration levels. Using these variables the generic empirical formula can be created and can be seen below in equation (4.2).

$$PPV = K \left(\frac{R}{(Q)^{0.5}} \right)^{-\beta} \quad (4.2)$$

Where;

PPV = Peak Particle Velocity

d = Distance from blast

Q = Charge weight per delay

K and β = Site constants

$(R / Q^{0.5})$ = Scaled distance

This equation illustrates the energy attenuation and the charge mass relationships with respect to the vibration levels. The site constants can be determined by plotting data points on a logarithmic graph of PPV versus Scaled Distance. The y-intercept of the line of best fit will give the 'K' site constant and the slope of the line will relate to the ' β ' site constant.

This equation is used by various researchers at 50% confidence interval. However, if the correlation coefficient (R^2) of equation determined at 50% confidence interval is close to 1.0, it may be used reliably. If it is very low, these equations should not be used. Several authorities (Erkoç and Esen, 1998; Bilgin et al., 1999b) have obtained satisfactory increase in correlation coefficient by detailed analyses that involve FFT analysis and filtering. However, these analyses are rather

complex and software of the seismograph must be capable of carrying out them (Bilgin et al., 2000). The most practical and reliable method developed till now is to determine this equation at 95% confidence interval. 95% confidence value (upper) is that particle velocity that will not be exceeded 95 out of 100 blasts at that scaled distance. This criterion may be provided by the software or one may use the methodology outlined by Dowding (1985). Figure 4.7 shows the example of Regression analysis of ground vibration data.

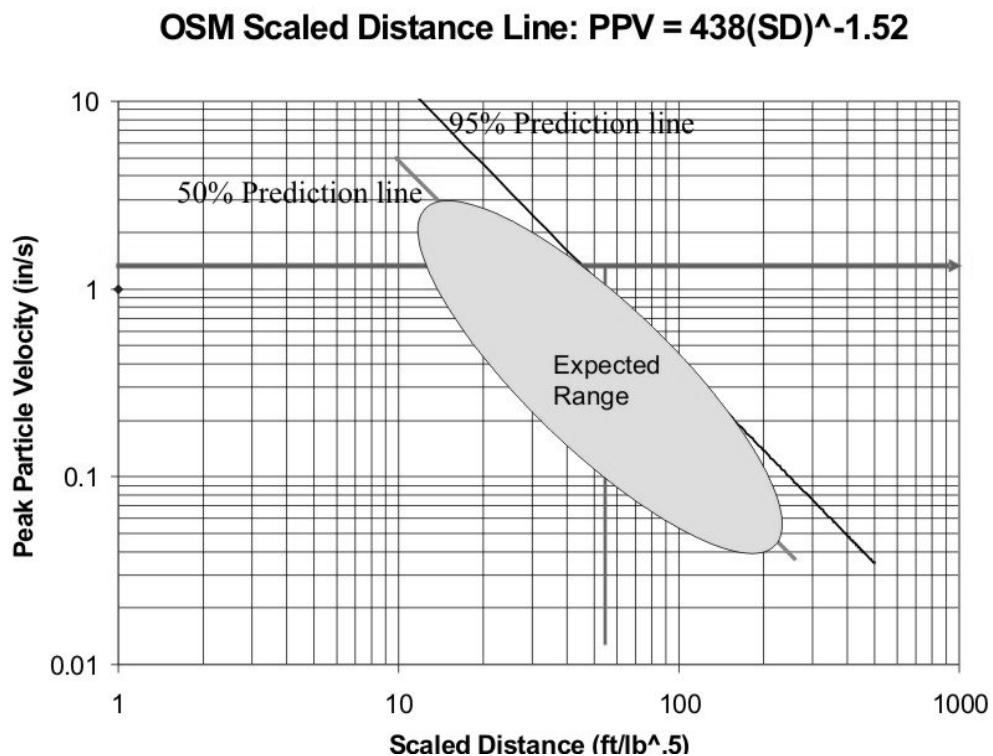


Figure 4.7 Example of Regression analysis of ground vibration data

Slight variations to the scaled distance equation are available with the charge weight being subjected to a square-root, the cube-root (0.33) or the two-thirds (0.66) relationship. However the square-root relationship is thought to be the most accurate for cylindrical blast holes and has been recommended by several investigations.

When using the scaled distance equations to predict the vibrations created from a blast it must be noted that there is a considerable vibration within these values due to the large range of variables that will have an influence on the attenuation of vibrations. As a result of this variations are generally due to the unpredictable

interaction between vibrations and discontinuities within the rock mass (Atlas Powder Company, 1987). The same principle of scale distance equations can be applied to the frequency of blast vibrations. This is because similar to the vibration magnitude the vibration frequency will change if there is an alteration to the distance the vibration has traveled of the initial charge weight that has created the vibration. The relationships can be trialled with square-root, cube-root or the two-thirds relationship similar to the vibration magnitude scale distance relationship. Site-specific constant can be calculated in the same way as for the magnitude empirical relationship. These scaling relationships can then be used to scale the seed wave magnitude and frequency if required. Table 4.6 shows the different vibration equations proposed by the researchers in question.

Table 4.6 Different vibration predictor equations (Khandelwal and Singh 2006)

Name of predictor equation	Equations
USBM (Duvall and Fogelson, 1962)	$v = k \left(\frac{R}{\sqrt{Q_{Max}}} \right)^{-\beta}$
Ambraseys-Hendron (1968)	$v = k \left(\frac{R}{\sqrt{Q_{Max}^{1/3}}} \right)^{-\beta}$
Longefors-Kihlstorm (1978)	$v = k \left(\sqrt{\frac{Q_{Max}}{R^{2/3}}} \right)^{\beta}$
Indian Standard Predictor (1973)	$v = k \left(\frac{Q_{Max}}{R^{2/3}} \right)^{\beta}$

4.3.5. Measurement of Vibration

Early researchers into ground vibration discovered a closer relationship between velocity of the ground surface and the response of structures than either displacement or acceleration. Measurement of velocity of the motion of the surface of the ground near where it enters a building has become the standard by which ground vibration is

measured and regulated. Ground vibration is measured with a blasting seismograph and is commonly expressed in terms of peak particle velocity and measured in terms of millimeters per second (mm/s). to define the motion in three dimensions, it is necessary to use three transducers to measure the vibration in three mutually perpendicular directions and then determine a Peak Particle Velocity or Peak Vector Sum, which is the instantaneous maximum vector of the three individual measurements.

$$\text{PPV (PVS)} = \sqrt{V_t^2 + V_l^2 + V_v^2} \quad (4.3)$$

Rather than being the simple wave type in the pond illustration, the ground vibrations are more complicated seismic events. The blast vibration consists of the different waves from each hole in the blast with propagation controlled by the physical and structural properties of the ground through which it travels.

It is obvious that ground and structure vibrations should be measured properly. Three issues are important to receive correct records. First, a contemporary transducer for velocity measurement is the portable assembly of a triaxial pack of geophones with the frequency response from 2 Hz to 300 Hz. These transducers are convenient tools with many options for measurement of vertical, radial and transverse vibration components. It is necessary to point out that a geophone's sensor has the natural frequency between 6.5 Hz and 9.5 Hz and therefore a flat velocity-frequency response starts at the frequency about 15 Hz. Manufacturers use software to compensate a resonance amplification of the sensor and receive a flat frequency response between 2 Hz and 15 Hz. Therefore the set of measured calibration curves of the whole transducer has to be submitted by manufacturers because results obtained without calibration curves could be misleading. Dowding (1996) demonstrated examples of the measured velocity-frequency responses. Also, it is important to verify linearity of the flat velocity-frequency response in the seismic range of transducers. The transducer should have a calibrated linear response in the region of the measured frequency, Skipp (1978).

Second, ground motion may excite natural vibrations of the seismograph-ground system and distort records of ground vibrations. Natural vibrations of the seismograph-ground system can be excited at a field test by slight impacts on the seismograph and the natural frequencies can be determined from the records. It is important to find the frequency zones where records can be changed, Svinkin (1973). Third, there are simple mathematical relationships between peaks of displacement, velocity and acceleration. Displacement calculations from velocity records are correct, but analogous acceleration calculations do not always yield proper results because velocity measurement cannot detect high frequency components of ground vibrations. Accelerometers and geophones have the opposite principles for vibration measurements. If acceleration limits are available for sensitive devices or foundation settlements, acceleration must be measured in parallel to velocity measurements.

4.3.6. Human Response to Ground Vibration and Regulations

Blast-produced vibrations at levels those are lower than the damage thresholds. A person will generally become aware of blast induced vibration at levels of around 1.5 mm/s, although under some circumstances this can be as low as 0.5 mm/s. Human reaction to vibrations depends on the amplitude as well as frequency and duration of vibration (Bilgin et al., 2000). Particle velocity values perceived by people at different levels depending on the duration of the vibration are illustrated in Figure 4.8 (Siskind et al., 1980). Relevant to whole-body vibration reaction is the degree that the vibration interferes with activity (sleep, speech, TV viewing, reading), presents a health hazard, and affects task proficiency. For people at home, the most serious blast vibration problems are house rattling, fright (fear of damage or injury), being startled, and for a few, activity interference. Complaints from these causes should be definitely considered in blast vibration control (Bilgin et al., 2000). Table 4.7 shows the human perception levels for vibrations.

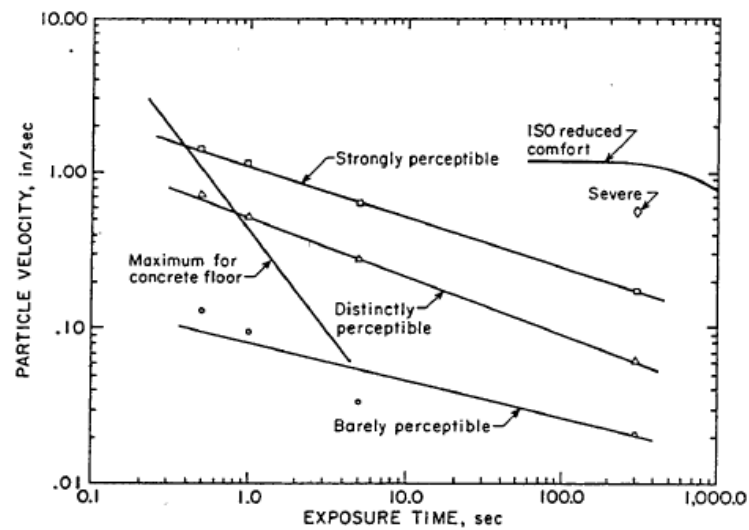


Figure 4.8 Human response to vibration of various duration
(Siskind et. al., 1980)

Table 4.7 Human perception levels for vibrations (Raina et. al. 2004)

$V_{\max}$ (mm/s)	Steady state	Vibration (impulsive, no noise)	Vibration (impulsive, with noise)
< 0.5	Not detectable	Not detectable	Noticable
0.8	Noticable	Not detectable	Noticable complaints possible
1.5	Noticable	Noticable	Noticable complaints possible
5.0	Disturbing	Noticable	Severe complaints likely
10.0	Very disturbing	Disturbing	Severe complaints
> 15.0	Severe	Very disturbing	Severe complaints

4.3.6.1 Vibration Regulations

A number of attempts have been made to connect vibration parameters (displacement, velocity, acceleration, and frequency) with observed human annoying, disturbances of sensitive devices, and structural damage, *e.g.* Crandell (1949), Medearis (1977), Siskind et al. (1980), Dowding (1996) and others. Duvall and Fogelson (1962), Wiss (1968), and Nichols et al. (1971) found that structural

damage could be well correlated with the peak particle velocity (PPV) of ground vibrations. The PPV is a measure of vibration intensity.

4.3.6.1.1 U.S. Bureau of Mines (USBM) The most extensive study of ground vibrations from blasting was made during the 10-year research program by the U.S. Bureau of Mines (USBM), Nichols et al. (1971). Vibrations were measured and evaluated damage criteria for residential structures and empirical safe blasting limits were established. The USBM's 51 mm/s (2 in/s) vibration criterion for ground vibrations with no regard to frequency was used to regulate blasting but could not properly reflect the effect of a variety of dynamic soil-structure interaction. Therefore this criterion was not able to prevent numerous complaints from habitants. It became necessary to take into account the dominant vibration frequency to assess the vibration effect on structures. Intensive studies of residential structural damage in connection with measured displacement and velocity at the frequency range of 1-100 Hz was made by the USBM. As a result, the frequency-based safe limits for cosmetic cracking threshold shown in Figure 1 were developed, RI 8507 (Siskind et al. 1980). (Fig. 4.9). As safe-level guidelines, these limits reduced the existing 51 mm/s (2 in/s) safe-level criterion by a factor of 3 to 4 in the frequency range of 4-12 Hz. It is necessary to point out that the RI 8507 has been recognized as the great achievement that provided the safety of low-rise residential structures from vibrations generated by mining blasting. Then the USBM's criteria were modified for regulation of blasting by U.S. Office of Surface Mining (OSM). The OSM criteria have the following displacement and velocity values for the four ranges of the dominant frequency: 0.76 mm (0.03 in) for 1-3.5 Hz, 19 mm/s (0.75 in/s) for 3.5-12 Hz, 0.25 mm (0.01 in) for 12-30 Hz, and 51 mm/s (2.0 in/s) for 30-100 Hz. Siskind (2000) mentioned also a simple distance-dependent set of the PPV criteria used by OSM and other regulatory bodies: 32 mm/s (1.25 in/s) for 0 to 91 m (0-300 ft), 25 mm/s (1.0 in/s) for 91 to 1,524 m (300- 5,000 ft), and 19 mm/s (0.75 in/s) for distances greater than 1,524 m (5,000 in/s) as seen in Table 4.8.

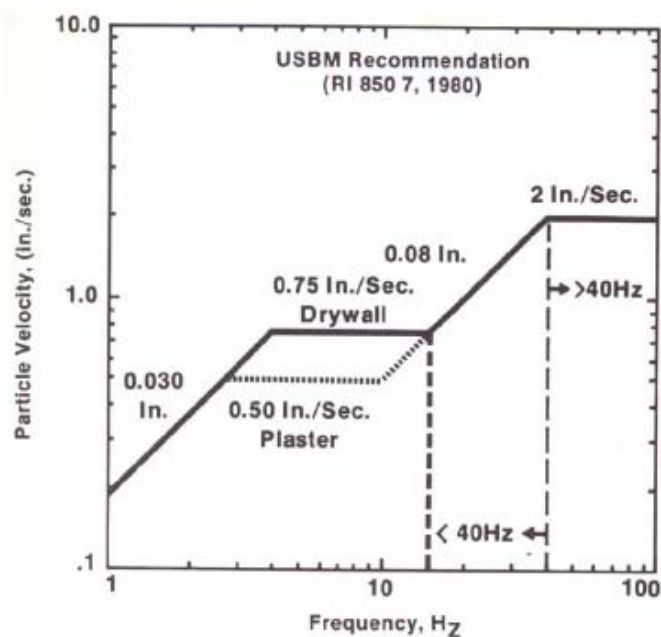


Figure 4.9 USBM Standard for vibration as a function of frequency and peak particle velocity

Table 4.8 OSMRE (Office of the Surface Mining, Reclamation and Enforcement, United States, Department of the Interior MARCH, 1987) regulation 816.67

Distance for Blasting Site in Meters	Maximum allowable Peak Particle Velocity in (mm/s)
0-90	31.25
90-1500	25.00
1500 and above	18.75

4.3.6.1.2. European Standard The European Standards are peak particle velocity level as a function of two factors;

1. The frequency of vibration
2. The type of structure

In the following tables, German, French, and Swedish British standards can be seen in Table 4.9, 4.10, 4.11, 4.12 and Figure 4.9 respectively.

Table 4.9 German standards from DIN 4150 (Basu and Sen, 2005)

Structure Type	Peak Particle Velocity in (mm/s)		
	< 10 Hz	10-50 Hz	50-100 Hz
Commercial	20	20-40	40-50
Residential	5	5-15	15-20
Sensitive	3	3-8	8-10

4.10 French standards (87/70558) (Basu and Sen, 2005)

Structure Type	Peak Particle Velocity in (mm/s)		
	4-8 Hz	8-30 Hz	30-100 Hz
Resistant	8	12	15
Sensitive	6	9	12
Very sensitive	4	6	9

4.11 Swedish standards (SS 460 48 66) (Basu & Sen, 2005)

Sub Soil	Vibration in (mm/s)
Unconsolidated strata of moraine sand, gravel, clay	18
Consolidated strata of moraine, soft limestone	35
Granite, gneiss, hard limestone, quartzite sandstone, diabase	70

Table 4.12 British Standards (7385:1993) (Richards & Moore, 2006)

Line	Type of Building	Peak component particle velocity in frequency range of predominant pulse	
		14 Hz to 15 Hz	15 Hz to above
1	Reinforced or framed structures. Industrial and heavy commercial buildings.	50 mm/s at 4 Hz and above	
2	Unreinforced or light framed structure. Residential or light commercial type buildings.	15 mm/s at 4 Hz increasing to 20 mm/s at 15 Hz.	20 mm/s at 15 Hz increasing to 50 mm/s at 40 Hz and above
NOTE 1: Values referred to are at the base of the building.			
NOTE 2: For line 2, at frequencies below 4 Hz, a maximum displacement of 0.6mm (zero to peak) should not be exceeded.			

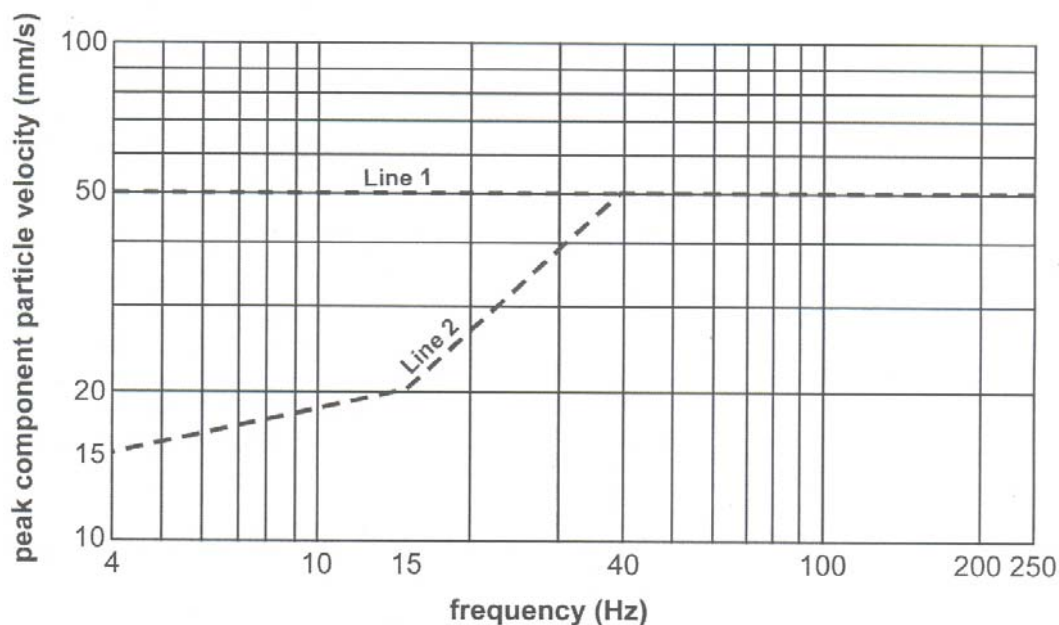


Figure 4.9 Transient vibration guide values for cosmetic damage (British Standards 7386:1993)

4.3.6.1.3 Indian Standards As per present Indian standards, as mentioned in DGMS (Tech) (S&T) Circular No. 7 dated August 29, 1997, depending on the type of structures and dominant excitation, the peak particle velocity (PPV) on the ground adjacent to the structure shall not exceed the values given below in Table 4.13

Table 4.13 Indian Standards (Basu & Sen, 2005)

Type of Structure	Dominant excitation frequency, Hz		
	< 8 Hz	8-25 Hz	> 25 Hz
(A) Buildings/structures not belong to the owner			
i) Domestic houses/structures (Kuchha brick and cement)	5	10	15
ii) Industrial buildings (RCC and framed structures)	10	20	25
iii) Objects of historical importance and sensitive structures	2	5	10
(B) Building belonging to owner with limited span of life			
i) Domestic houses/structures (Kuchha brick and cement)	10	15	25
ii) Industrial buildings (RRC and framed structures)	15	25	50

4.3.6.1.4 Turkish Standard The ministry of Environment and Forest has proposed a standard dealing with the blast induced ground vibration. This regulation has been prepared for evaluating the ground vibration intensity induced by blasting. This standard is shown in Table 4.14

Table 4.14 Frequency based Turkish Standard (Çevre ve Orman Bakanlığı, 2007)

Ground vibration frequency (Hz)	Permissible peak vibration velocity (Peak value-mm/s)
1	5
4-10	19
30-100	50

4.3 Fly Rock

Fly rock is considered to be the most undesirable movement of rocks during the blasting activities. Danger and damage from fly rock in rock blasting has been a serious problem ever since blasting was introduced. Not only have man been killed and injured but also buildings, equipment and materials have been damaged (Lundborg et. al., 1975). Damage by a fly rock cannot be refuted; the evidence is usually present and visible. An example is a case where a fly rock might go through a window; the rock will be there as well as the splinters, if not disturbed by individuals. This type of undesirable activity stems from a number of factors and easier to control than ground vibration and tends to be of a greater concern to the blaster than the neighbors. The major causes of fly rock are inadequate burden, inadequate stemming length, drilling inaccuracy, excessive powder factor, unfavorable geological conditions (open joints, weak seems, cavities), inappropriate delay timing and sequence, in accuracy of delays, back break and loose rock on top of the bench (Fletcher and D'Andrea, 1986; Workman and Calder, 1994; Kopp, 1994; Bhandari, 1997). The energy spent in creating fly rock is less than 1% of the total energy transferred to the rock (Berta, 1990), hence the wastage of explosive

energy in this form may be insignificant. However the risk of damage due to the fly rock is so high that it merits serious consideration in blast design.

4.3.1 Safe Distance and Regulations Related Fly Rock

Safe distance is the minimum distance beyond which the throw of fragments does not appreciably affect the surroundings. The Metalliferous Mines Regulations, 1961 prescribes a danger zone of 300 m for surface blasting. As the accidents related to flyrock did occur outside this zone, the Director General of Mines Safety, Dhanbad recommended to increase the radius of the danger zone to 500 m for surface blasting (DGMS, 1982).

- The U.S. Code of Regulations (CFR), Title 30, Part 57.6000, defines ‘Blast Area’ as the area in which concussion (shock wave), flying material, or gases from an explosion may cause injury to persons.
- The US Federal Office of Surface Mining (USOSM) regulations [30 CFR §816.67 and 817.67] help to characterize the bounds of the blast area by specifying that flyrock shall not be cast from the blasting site.
- Institute of Makers of Explosives (IME) has defined flyrock as the rock(s) propelled from the blast area by the force of an explosion [IME1997]. A flyrock related injury is sustained when a blast propels rock beyond the blast area and it injures someone.

4.3.2 Primary Factors for Fly Rock

4.3.2.1 Influence of Burden

It should not be interpreted that burden has no influence on fly rock. From a single hole tests (Bilgin, 1991), it is known that the fly rock decreases as the burden increases. In the present study, the variation of burden distance is not large enough to

observe the behavior of fly rock as in the case of single hole tests. Moreover, the burden of 20-40 times the blast hole diameter is within the accepted range (Bhandari, 1997).

4.3.2.2 Influence of Stemming Length

Generally the maximum fly rock distance decreases with an increase in stemming length. The decreasing trend of fly rock distance with increasing stemming length is obvious from the role of confinement in controlling premature venting of high pressure gases. Persson et. al. (1994) has provided an explanation for this phenomenon. When the stemming length is short, shock wave component of the explosive energy is sufficient to fragment the rock at the collar. If the rock is finely fragmented, the small fragments will not be thrown very far because they have a very low mass and are stopped effectively by the air drag. If the stemming length is long, large piece of rock will form. They have large mass and, for the same impulse, their velocity of thrown distance will be consequently short. Between these two extremes, a maximum throw distance may be expected. The water in the blast hole exerts significant influence when it rises up to the stemming column. The drill cuttings used as stemming material form slurries which cannot provide adequate confinement to the bore hole pressure. The gases energy is vented into the atmosphere causing fly rock. The effect of water on fly rock may be reduced by dewatering blast holes or by using suitable stemming material.

4.3.2.3 Influence of Powder Factor

In bench blasting, the powder factor (P.F.) is termed as a measure of the explosive weight required to break a unit volume of rock. Previous study (Lundborg et. al. 1975) also shows that fly rock can be avoided if the powder factor is smaller than a critical value. However it is not recommended to reduce the powder factor significantly because it will adversely affect the fragmentation and displacement. An optimum powder factor can satisfy both economic and environmental considerations

in blast design. Excess powder factor is not used at limestone quarries and unless charge concentration becomes high due to undetected geological structures.

4.3.2.4 Influence of Blast Hole Diameter

When a blasthole diameter is increased, the loading density of fully coupled charges increases by the square of the ratio of the diameters. Using a semi-empirical method, Lundborg et. al. (1975) has derived the following equation to estimate maximum fly rock distance:

$$L_{max} = 260d^{2/3} \quad (4.4)$$

Where (L_{max}) is maximum fly rock distance (m); (d) is the hole diameter (inch).

4.3.2.5 Influence of Initiation Systems

Significant progress has been made during the last two decades in the areas of blast hole initiation and development of initiating devices. The influences of delay timing and sequence on generation of flyrock are known (Konya and Walter, 1990; Bhandari, 1997), but the degree to which flyrock can be controlled is also influenced by the selected initiation system. For the various initiation methods commonly used at limestone quarries,

If the individual holes are connected with short delay electric detonators, there is a probability of overlap due to scattering of delay periods. If any of the succeeding detonators in the second or third row fires earlier than the preceding detonator, it would cause cratering and generation of flyrock from top of the bench. The problem still exists if the group of holes are connected with detonating cord and initiated by short delay electric detonators.

If the holes are initiated with detonating cord relay system, the initiation proceeds sequentially and the problem of flyrock due to overlap does not arise. However, there

is limited scope for optimizing the delay intervals as the detonating relays have fixed intervals. A group of holes can be connected with detonating cord, which in turn is connected to instantaneous electric detonators. The delays are provided with circuit delays of the Sequential Blasting Machine. In this case, the delays are accurate and the delay intervals can be optimized to control flyrock. The detonation of a 10 gm/m detonating cord downline, being used in limestone quarries, can laterally compress the stemming material (drill cuttings) so as to create a gas chimney through which gases can escape into the atmosphere. The detonating cord downline system should be replaced by the shock tube initiation system or low energy detonating cord. Besides flyrock control, there are several advantages of these initiation systems (Konya and Walter, 1990; Bhandari, 1997).

4.3.3 How to Control Fly Rock

- Orient quarry faces away from potentially sensitive receivers. (Select the direction of blasting face so as to place any person or buildings at the rear of the face)
- Ensure stemming depth and type is adequate
- Choose a burden of sufficient magnitude to reduce the possibility of blast through Take special care when carrying out secondary blasting to avoid overcharging
- Avoid shallow blasting that is generally less than 1meter
- Cover the rock to be blasted with suitable material, to control fly rock
- Make sure that the order of firing is correct
- Cover the rock to be blasted with suitable material, to control fly-rock, if necessary.
- Make sure that the blast area security is adequate

4.4 Dust and Fumes

There is always some dust associated with blasting. However, a well-controlled blast will always produce less dust than a poorly controlled blast. If the blast is

properly planned and executed then the dust is largely taken care of. The prevailing wind strength and direction plays a large part in dust control. Ideally, the wind should be strong enough to disperse the dust as quickly as possible without it being carried into the nearby buildings. Noxious fumes are sometimes associated with blasting. These fumes are normally nitrous oxides (NO_x) or carbon monoxides (CO) or a combination of both. Fumes, if present, are easily seen in the post-blast dust cloud. The fumes appear as yellow/orange clouds and in some cases can be red in color. Generally, the darker the colours of the cloud, the more toxic are the fumes contained inside it. Fumes are normally caused by poor or inefficient detonation of the explosives. This generally happens when the ground is wet, or when the rock formation is strongly stratified and cannot provide sufficient confinement for the detonation to take place.

From the many toxic gases that are formed only carbon monoxide and nitrogen oxide reach the level where it can be dangerous after a blast. However, there are threshold values stated for these fumes. The blasters should ensure that these are not exceeded. The inhalation of carbon monoxide above the threshold level of 50ppm will lead to Asphyxiation. Nitrogen oxides are very irritating to the skin and the mucus membrane. When further oxidation takes places, nitrogen oxide becomes nitrogen dioxide. It becomes reddish brown in color can be fatal if inhaled at a high concentration level. High levels of nitrogen dioxide contamination can be seen as anything above 30ppm. Exposure to nitro-glycerine based explosives can lead to low blood pressure and headaches. The nitro-glycerine will dilate blood vessels which can lead to fatality if over exposed. This exposure can come in the form of contact with the skin during blast hole charging or inhalation of fumes after blast.

CHAPTER FIVE

FRAGMENTATION PREDICTION BY IMAGE ANALYSIS METHOD

5.1. Introduction

From the whole mining activities, drilling and blasting operations are of great importance in terms of over all cost of mining. Mine administrators always struggle to excavate as much rock mass as possible so that they can minimize the cost of excavating. However, optimization of fragmentation size is the most essential aspect in rock excavation process. Therefore, many researchers have made some studies regarding determination of optimum size of fragmentation due to blasting. Moreover, fragmentation is a key factor in order to control and minimize the loading, hauling, crushing, classification and processing costs in aggregate and industrial minerals operations.

Some models have been introduced so far in order to predict the optimal size of fragments such as Kuz-Ram Model, Larsson's model, The SVEDEFO (Swedish Detonic Research Foundation) model. These mentioned models are of some limitations in terms of prediction of muck pile size. For this reason digital image softwares were developed through the 1990s and at present they are worldwide accepted tool in the mining and mineral processing industries. In chapter 5, the Kuz-Ram Model and digital image processing software named WipFrag will be introduced.

5.2 Blasting Fragmentation Prediction

Assessment of the blast performance is critical for optimal blasting, and the size distribution of the blasted material is essential to determine the degree of fragmentation. However, fragmentation is influenced by both controllable and uncontrollable parameters: rock properties, the geometry of rock, and blasting patterns, and the optimal size distribution of the blasted material depends on the mining objective. Optimal blasting is a very complex and difficult issue.

Furthermore, there is no method or equation which can predict the blast fragmentation exactly because of varying desired blasting fragmentation and numerous controlling parameters involved in the process. Many researchers have recently developed models and computerized simulations. Following is one of the most widely accepted models called Kuz-Ram. (Jimeno, C. L., Jimeno, E. L., & Carcedo, F. J. A. , 1995).

5.2.1 Kuz-Ram Model

Kuz-Ram is the combination of Kuznetsov and Rosin-Rammler equation, and an empirical fragmentation model. Since its introduction by Cunningham, the Kuz-Ram model has been used by many mining engineers to predict rock fragmentation arising from blasting, and many researchers have attempted to improve the Kuz-Ram fragmentation prediction model (Cunningham, 1983 and 1987).

The model has two main factors;

The characteristic size (X_C): It was derived by the Kuznetsov model (Kuznetsov, 1973). The uniformity index (N): It is based on geometric parameters of the drilling and blast design. The size distribution of the muckpile rock after blasting is determined by these two main factors. However, this original Kuz-Ram fragmentation model has the limitation of application and a high margin of error (Spathis, 2004).

5.2.2 The Mean Size of the Fragments Formed by Blasted Rock

The distribution function, an analytical representation of the fragment size composition of blasted rock, has been suggested by Rosin-Rammler model (Lilly, 1986), (Chung & Katsabanis, 2000), and (Kuznetsov, 1973).

$$\Phi_{(x)} = 1 - R_{(x)} = 1 - \exp \left[- \left(\frac{x}{x_0} \right)^N \right] \quad (5.1)$$

Where:

$\Phi_{(x)}$ is the distribution function (the total relative volume of fraction not longer than x).

X_0 is the Characteristic Size.

N is the Uniformity index

$R_{(x)}$ is the fraction of material retained on screen.

Using the Rosin-Rammler equation, the formula for the mean fragment size was suggested with given rock volume and needed explosives by Kuznetsov (Kuznetsov, 1973).

$$\langle X \rangle = A \left(\frac{V_0}{Q} \right) Q^{1/6} \quad (5.2)$$

Where:

$\langle X \rangle$ is the mean fragment diameter (cm)

V_0 is the volume of blasted rock per hole (m^3).

Q is the weight of explosives of TNT equivalent explosives per hole (kg)

A is rock factor:

$A=7$ the medium hard rocks, $f=8-10$

$A=10$ the hard but highly fissured rocks, $f=10-14$

$A=13$ very hard and weakly fissured rocks, $f=12-16$

f is the Protodyakonov factor.

An equivalent quantity of any explosives, Q_e related to TNT is calculated by Equation 2.5 because TNT is not currently used in blasting as a standard explosive (Clark, 1987).

$$Q = Q_e \left(\frac{E_e}{1090} \right) \quad (5.3)$$

Where:

E_e is the absolute weight strength of the explosives (cal/g).

The factor 1090 is the absolute weight strength of TNT.

5.3 WipFrag Software

WipFrag is a software system specially designed to quantify the size distribution of unconsolidated materials while in the field, lab or office. WipFrag is the pioneer, innovator and leader in fragmentation and particle photoanalysis. WipFrag offers a fast, convenient and accurate way of quantifying any material which can be photographed or videotaped. WipFrag uses tested and proven algorithms which measure particle size distribution for any sample that can be photographed. It allows size distribution comparisons of material in order to determine the optimum size for higher production and lower cost. WipFrag also provides data on uniformity, oversize, fines and fragment shape. WipFrag image analysis is extremely fast and easy, with most users becoming proficient in their very first day.

Using digital image analysis of rock photographs and videotape images with granulometry system, grain size distribution may be obtained by WipFrag. Photographic images are digitized by using WipFrag from slides, prints or negatives, using a desktop copy stand. In order to overcome size limitations inherent with a single image, WipFrag has the function for zoom-merge analysis. Therefore, combined analysis of images taken at different scales of observation may be analyzed. In addition, Using Edge Detection Variables (EDV), fragment boundaries are analyzed efficiently, and manual editing can improve edge detection (Maerz, Palangio, and Franklin, 1996)

WipFrag is specially designed software which identifies and measures particles and displays the results in easy to use form. WipFrag can use images from digital cameras, scanned images or video-tape playback, (NTSC or PAL). Once the photo is captured, analysis is easy. After opening the sample photo in the WipFrag software,

you can generate results in seconds. More importantly, the material is untouched during measurement so it is never degraded, contaminated, damaged or altered due to sieving, handling or segregation.

WipFrag outputs all of the data into three easy to read screens: Chart, Cumulative Curve and a Histogram. As well, WipFrag can log all of the information so that it can be easy merged together at a later date, or imported into another program such as excel. Results can be displayed as an easy to read histogram, cumulative weight percent passing curve and sieve size. Monitoring particle size has never been easier (Fig. 5.1).

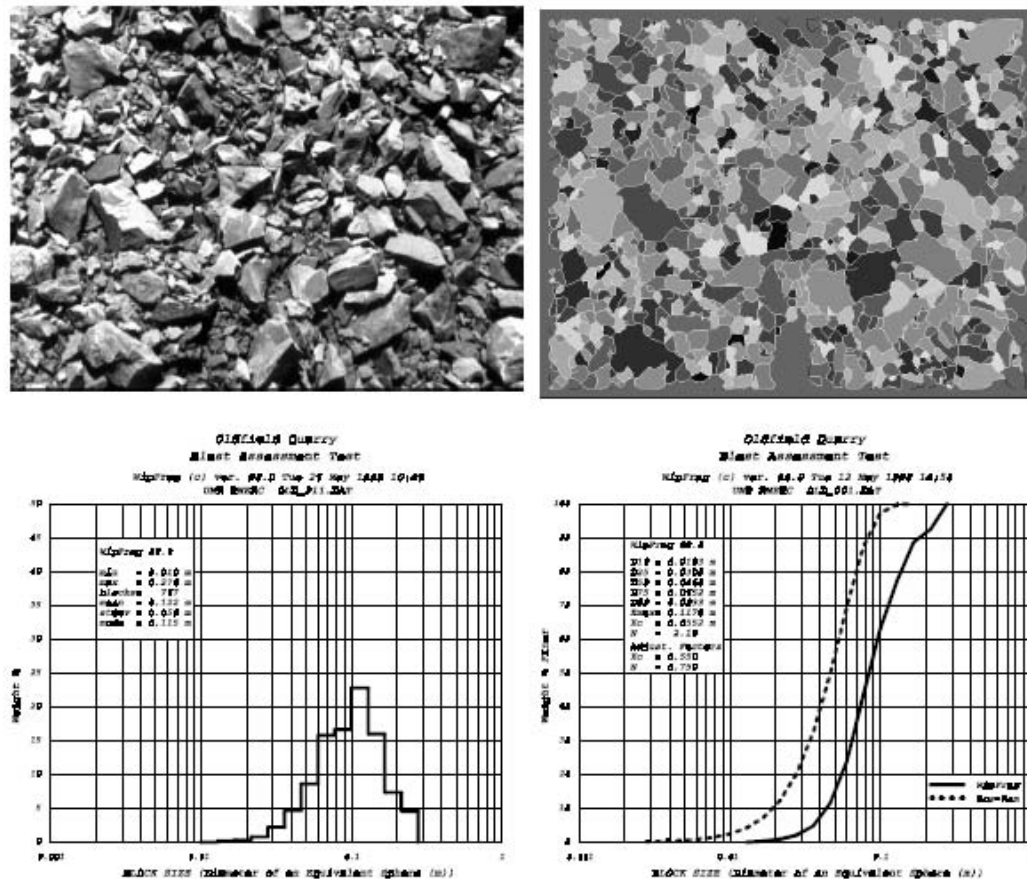


Figure 5.1 Top left: Typical image of rock fragmentation. Top right: Computer generated block outlines Bottom right: Measured Histogram. Bottom left: Cumulative weight percent distribution, with calibrated curve (dashed line) (Palangio et. al, 2005).

CHAPTER SIX

CASE STUDY ON CIMENTAS LIMESTONE QUARRY

6.1 Introduction

As it is well known, there are some bad outputs generating from blasting such as ground vibration, dust, fly rock, air/over pressure. One of the most disturbing outputs is certainly ground vibration. Blast induced ground vibrations create some effects similar to earthquakes and these effects make residents annoyed and disturbed. In order for the production blasting operations conducted at limestone quarries to continue, necessary precautions should be taken. Especially both the proximity between blast point and residential areas and the charge weight per delay should be taken into consideration. As a result of all these calculations and analyses, the proper site constants can be derived from the equation.

In order to observe the decrease in the ground vibration intensity when the water deck is introduced within the borehole, Cimentas limestone quarry was chosen. Necessary data were collected to determine the site constants and optimal vibration intensity. Then the water deck was introduced within the borehole. Some production blasting operations were carried out to observe the decrease in so-called bad outputs arising from blasting. Monitoring points were selected in some places where people live and complain about blasting. According to measurements taken from the monitoring points, there was a considerable decrease in the ground vibration intensity from blasting conducted by using water deck. From other outputs such as dust amount, air/over pressure, fly rock, there was a decrease in the dust amount as well. Some noise measurements were also taken in order to see the difference and it is because of less charging, the decrease was also observed in air/over pressure intensity. Chapter 6 will introduce the application of water deck and monitoring process in Cimentas limestone quarry.

6.2 Description of Quarry Site

Cimentas limestone quarry is located in central Izmir in Aegean region in the west of Turkey. The location of the quarry at issue can be seen in Figure 6.1

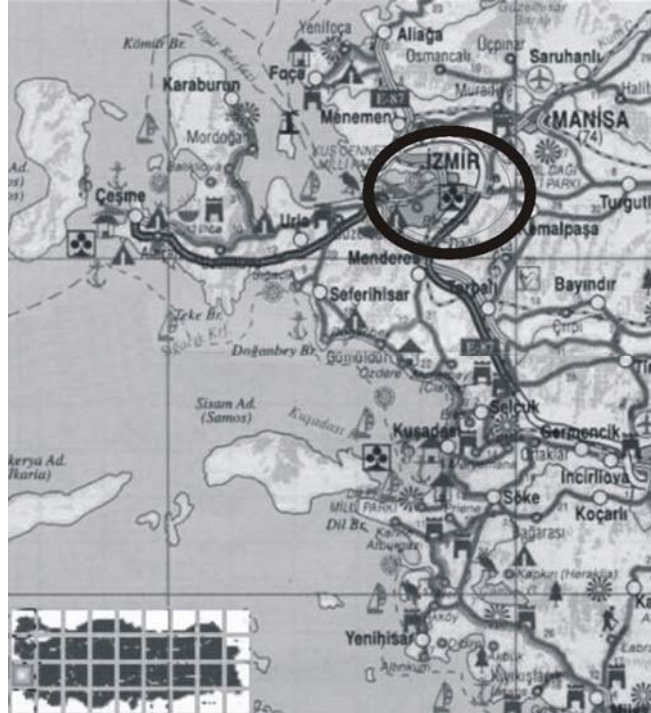


Figure 6.1 Location of the quarry site

Cimentas limestone quarry excavates approximately 2.500.000 tons of limestone yearly and Cimentas Cement Factory produces nearly 1.800.000 tons of cement yearly. A huge amount of cement is exported to countries all over the world. Since uniaxial compressive strength value of the limestone excavated from the quarry is 300 kg/cm^2 , so the drilling and blasting operations have to be used to break the rock.

6.3 General Mine Geology

The geology in the vicinity of the mining area consists of mesozoic and cenozoic depositional units. These so-called units are as follows from bottom to top; upper cretase limestones, paleozoic sandstones-shale, and neogene aged marl and clay.

6.3.1 Limestone Unit

The Limestone Unit, upper cretase aged, houses the basement. It is recrystallized and has brownish-gray-dirty whitish hue. The thick layered unit is of massif character and is quite durable. It contains many cracks and joints are relatively thick and filled with calcite. The unit is of thin and moderate grain sizes. Rudis and Foraminifer contain fossils. Much of the mining area comprises this abovementioned unit.

6.3.2 Sandstone-Shale Unit

It can be observed in the area which falls the northern part of the mine site. Since this paleogen aged unit is not durable enough, it cannot form the hillsides. The so-called unit is of complex structure. the unit is composed of gravelstone, sandstone-shale, and shale with calcite member as well as limestone blocks.

6.3.3 Sandstone Unit

Sandstones that the so-called formation includes are of dirty yellowish-dark gray surfaces and the cleat planes present the yellowish-reddish hue. The thickness of layers varies in the range of 2-30 cm. this unit is of numerous cracked structures, and its cracks are filled with calcite. The unit in question presents finely grained structure in some areas; however, it presents coarse grained structure in other areas.

6.3.4 Shale Unit

Decomposition color of the unit is of greenish, yellowish, grayish, and dirty whitish. The unit is very thin layered and quite undurable. In most areas, shale is of muddy shale. The boundary between the sand stone and shale presents almost the same vision in every part of the unit. Bottom boundary of sandstone is of immediate and upper boundary of it is of graded. sandstone-shale sequence contents olistolith belonging to upper cretase aged limestone.

6.3.5 Marl and Clay Stone Unit

This unit located in the southern part of the area is Neogene aged. The unit is composed of medium thick layered limestone with gastropod fossils and limestone with clay (marl) and clay stone sequences. It contains undurable mudstone with 1-3cm thick. (Konak et. al., 2005)

6.4 Interpretation of Blast Design and Optimization Applied in Cimentas Limestone Quarry

Maximum productivity can be obtained from blasting as long as the proper optimization is achieved. Many researchers have proposed several approaches for determining the best blast pattern in terms of productivity and further mining processes. However, the site constants should be kept in mind and the blast pattern which can provide desirable fragmentation and the least ground vibration intensity should be designed. Additionally, determination of the optimum burden and spacing is of crucial importance in terms of the fragmentation size of the muckpile obtained after blasting.

In Cimentas limestone quarry, the blast pattern determined by the contractor is based on the observation of muckpile and on the least boulder occurrence. In fact the best optimization is that some changes should be made in the determined blast design by observing the muckpile. This situation is called “rules of thumb” in the literature. The blast pattern data determined by contractor can be seen in Table 6.1

Table 6.1. Details of Cimentas Quarry blasting practice

Blasthole diameter (mm)	89
Spacing (m)	3.5
Burden (m)	2.5
Drill pattern	Staggered
Hole length (m)	15
Sub drill (m)	1
Stemming length (m)	3
Stemming material	Drill cuttings
Initiation system	Nonel
Drilling inclination	Vertical

6.4.1 Blasthole Diameter

In an open cast mine, many parameters can be effective in determining the optimum blast hole diameter. Especially, hourly production capacity, cycle time of the working machines, vibration problems, unit cost and geological structures are the major parameters that should be noted. In general, blast holes are drilled from 65mm up to 165mm in quarries. The more the diameter of the blast hole is decreased the more the drill cost is increased. Therefore, determination of the most proper of blast hole diameter is of great importance in terms of economy of the mine. Additionally, the geological structure (weak zones, joints and cavities) plays an important role in determining the optimum blast hole diameter. The larger the blast patterns, the more likely these blocks are to be thrown unbroken into the muckpile. In Cimentas limestone quarry, the blast holes are drilled with the diameter of 89mm. as far as the above mentioned parameters are concerned, it can be said that the blastholes with diameter of 89mm is proper for the site and production capacity.

6.4.2 Bench height

Two of the most important parameters in determining the bench height is blast hole diameter and the boom length of the working machine operating in the mining site. In many applications bench height of 8m-10m is used at the blast hole diameter of 65mm-90mm. certainly if the bench height is decreased then the charge weight per delay is automatically decreased. There are several bench heights in Cimentas limestone quarry because of the changing altitude. However, the highest bench is 14m-15m in the so-called quarry. The image of the bench can be seen in Figure 6.2.

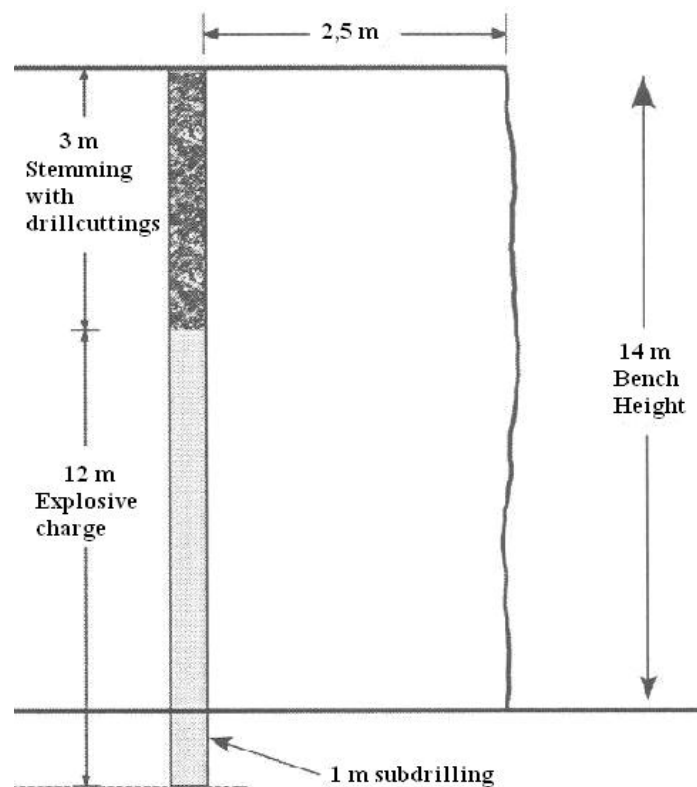


Figure 6.2. Cross section for the bench height used in Cimentas Quarry

6.4.3 Burden

Burden is described as the distance between the center of the blast hole in the first row and the free face. Some parameters that are effective in determining the burden; blast hole diameter, rock type, explosive type, bench height and fragmentation size.

In many applications, burden is determined to be 25-40 times the blast hole diameter. Burden less than adequate causes uncontrolled blasting operation and air/over pressure, and the burden more than adequate causes the blasting energy to convert into seismic energy. Moreover, the burden should always be less than spacing in most quarry blasting. The burden value which is applied in blasting in Cimentas limestone quarry is 2.5m, and this value is quite proper as compared to the blast hole diameter.

6.4.4 Spacing

Burden and delay timing between the blast holes are effective in determining the spacing. In most applications, when the blast holes are fired by using delay time, spacing may be 1.2-1.8 times the burden. The spacing applied in Cimentas limestone quarry is 3.5m, and it can be said that this spacing value is quite proper for optimization.

6.4.5 Stemming

Stemming is necessary for confining the shock energy due to blasting. Stemming more than adequate causes the boulder to occur in the upper part of the bench, and the stemming less than adequate causes flyrock and air pressure. Stemming material is also of importance in terms of determining the length of stemming. In general, crushed stone is the best material to use for stemming, but the drill cuttings is commonly used because of its availability and ease. In many applications, stemming length is 20-60 times the blast hole. In Cimentas limestone quarry, the stemming length is 35 times the blast hole diameter.

6.4.6 Delay Timing

There are two reasons for using delay between blast holes; better fragmentation and decreasing the vibration intensity. Moreover, since frequency is defined as the number of waves occurred in a unit time, delay timing can be effective in order to

increase the frequency to some extent. In many applications, 1ms-5ms delay is used for each 30cm of burden. The most appropriate delay is determined as 3ms. The delay between rows should be 2-3 times the delay between holes. If the delay between the rows is less than adequate then the burden motion will be upward rather than forward; in other words, the back break can be occurred because of the improper delay timing. In Cimentas limestone quarry, the delays between holes were determined as 25ms and the delays between rows were determined as 65ms (Fig. 6.3.).

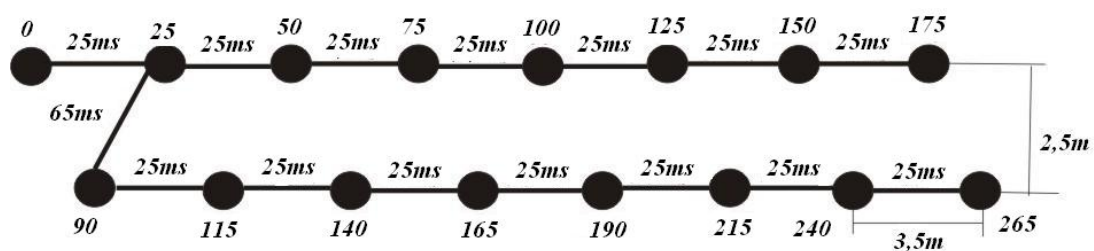


Figure 6.3. Application of delay timing in a blast pattern with diameter of 89mm blast hole

6.4.7 Powder Factor

Powder factor is defined as the necessary explosive amount (kg) in order to blast one (m^3) of rock mass. In other words, powder factor is the function of explosive type, rock density and the geological structure. Powder factor is not directly an effective parameter for the blast design; however, it is of great importance for obtaining the data regarding energy consumption and economy of blast design. Powder factor can generally vary from 0.1 to 1.1kg/m^3 . however, most typical value of powder factor used in blasting operations conducted in open cast mines varies from 0.2 to 0.45kg/m^3 . Table 6.2 includes typical powder factor for surface blasting.

Table 6.2 Typical powder factors for surface blasting

Rock breakage difficulty	Powder Factor (kg/m ³)
Low	0.1-0.18
Medium	0.18-0.34
High	0.34-0.57
Very high	0.57-1.14

$$q = \frac{Q_{Top}}{E * V * K} \quad (6.1)$$

Where, (q) is the value of powder factor (kg/m³); (Q_{top}) maximum charge per delay (kg); (E) is the spacing (m); (V) is the burden (m); (K) is the bench height.

When it is noted that the uniaxial compressive strength of the limestone excavated from Cimentas Quarry is about 300kg/m² and that this value takes place as medium strength-strength in the rock strength classifications systems prepared by Bieniawski, it can be understood that the powder factor value higher than 0.34 should be used for the so-called quarry. This can be explained by two blasting operations conducted in two different locations;

Location 1: Near transmitting line

Bench height: 11m

Spacing: 3.5m

Burden: 2.5m

Excavated material amount: (for 15 blast holes) 4095 tones

$$q = \frac{45}{3,5 * 2,5 * 11}$$

$$q = 0,47 \text{ kg/m}^3$$

Location 2: Short bench

Bench height: 6m

Spacing: 3.5m

Burden: 2.5m

Excavated material amount: (for 15 blast holes) 2388.75 tones

$$q = \frac{20}{3,5 * 2,5 * 6}$$

$$q = 0,38 \text{ kg/m}^3$$

As it is seen from the examples, powder factor values obtained from two bench blasting are quite economical.

6.5 Site Survey and Application of Water-Deck in Production Blasting in Cimentas Quarry

6.5.1 Site Survey

Two major surveys were conducted in the scope of this thesis. First survey was carried out to determine the safe charge amount for bench blasting in order to decrease the vibration intensity, and the second survey was related to the application of water deck in production blasting to determine the effect of water deck on ground vibration, noise, back break, dust amount and fragmentation size.

Because of the location of Cimentas Quarry, it is obvious that the ground vibration generated by blasting carried out in the quarry at issue attenuates in the vicinity of the residential areas as shown in Figure 6.4.



Figure 6.4 The proximity between Cimentas limestone quarry and residential areas

For one thing, the settlers of the residential areas around the quarry have claimed that there is visible damage in their houses and also they have got the physiological problems based on worries. On the one hand these claims could be true, on the other hand these claims could be untrue because some settlers want to get compensation from the quarry management for the damage in their house. Upon a request of

management of the quarry, a set of ground motion measurements were conducted in the vicinity of the site said to be suffering from blasting.

In the scope of the survey, first of all the blast pattern used were investigated in terms of the ground vibration intensity. Then all measurements were carried out through the use of the seismographs belonging to the University of Dokuz Eylul and to Nitromak Explosive Company. In order to obtain the data required for the analyses, first a certain station points were selected for blasting operations, and the distance between blast point and the station point in question was calculated by means of GPS. Second ground motions induced by blasting were carefully recorded in longitudinal, transverse and vertical directions. Lastly all data obtained were evaluated in order to determine the charge weight per delay to be used not to exceed the vibration limit in the vicinity of the so-called sites. The model used for the linear Regressional analysis is the one proposed by USBM given as follows;

$$PPV = k \left(\frac{R}{\sqrt{Q_{Max}}} \right)^{-\beta}$$

Where; (k) and (β) is the site constants, (R) is the distance between shot point and monitoring station, (Q_{max}) is the charge per delay.

In the scope of this field study, 60 data were recorded from 50 shots. To determine the site constants “k” and “ β ”, the logarithmic graph of PPV versus Scaled Distance (SD) was plotted, and as a result of this, the site constants “k” and “ β ” were determined as 159564 and 2,5792 respectively with correlation $r = 0,71$. All data obtained; the amount of charge weight per delay, the distance from the blast point and scaled distance values were presented in Table 6.3 Also, PPV versus scaled distance graph can be seen in Fig. 6.4

Table 6.3 Ground vibration data taken from different stations around Cimentas limestone quarry

Shot No	Particle velocity (mm/s)				Frequency (Hz)			Distance (m)	Scaled Distance (m/kg ^{0.5})	Charge Per delay (kg)
	Vert. (PVL)	Long. (PVT)	Trans. (PVV)	Peak Vec. (PVS)	Vert.	Long.	Trans.			
1	0,54	0,35	0,00	0,54	7,75	8,62	0,00	710	129	30,30
2	0,44	0,28	0,03	0,51	11,9	15,4	333,0	559	125	20,00
3	0,86	1,03	0,00	1,34	13,1	13,5	0,00	500	92	29,60
4	1,71	1,39	0,00	2,07	13,1	14,1	0,00	649	100	41,79
5	1,31	1,18	0,00	1,58	12,2	10,2	0,00	502	83	36,62
6	0,84	0,72	0,00	1,03	11,1	15,6	0,00	472	80	34,38
7	0,45	0,61	0,42	0,76	14,1	14,9	10,4	725	115	40,00
8	0,49	0,34	0,44	0,59	12,7	9,43	8,40	712	120	35,00
9	0,80	0,63	0,51	1,05	10,0	13,9	12,7	513	115	20,00
10	1,89	1,79	1,51	2,84	11,7	11,0	11,8	527	91	33,53
11	1,60	1,52	1,06	1,90	10,5	14,3	15,6	722	113	40,89
12	1,92	1,64	1,50	2,67	11,9	11,3	10,3	678	105	41,58
13	1,64	1,94	2,18	2,83	14,3	11,9	10,5	508	80	40,00
14	0,80	0,62	0,70	0,96	9,71	10,0	10,8	550	116	22,50
15	0,44	0,50	0,50	0,58	10,8	10,3	6,06	697	118	35,00
16	1,27	1,27	1,19	1,78	12,8	10,9	12,7	730	91	65,00
17	0,89	0,75	0,49	1,11	13,3	8,62	11,1	509	107	22,50
18	1,15	0,96	1,00	1,51	14,1	13,0	15,9	470	79	35,00
19	1,56	1,29	0,92	1,78	9,59	8,85	7,69	698	102	46,50
20	1,60	2,26	1,15	2,78	11,4	16,2	5,52	532	75	50,00
21	2,08	2,61	1,31	3,47	14,5	15,5	17,9	508	72	50,00
22	0,74	0,94	1,12	1,18	11,8	11,7	8,55	489	69	50,00
23	0,55	0,62	0,80	1,00	10,3	10,2	8,33	692	117	35,00
24	1,74	2,41	1,76	2,86	10,2	10,5	11,1	703	91	60,00
25	0,70	0,91	1,19	1,39	10,4	8,40	9,68	465	81	32,50
26	0,80	0,57	0,91	1,09	10,3	10,1	9,90	482	79	37,50
27	0,62	0,72	0,47	0,78	11,1	9,17	10,0	735	104	50,00
28	1,78	0,88	1,09	2,01	11,3	9,29	10,5	719	93	60,00
29	2,50	2,13	2,40	3,24	10,8	9,52	10,7	617	78	63,00
30	0,19	0,23	0,34	0,38	16,7	12,7	7,46	507	101	25,00
31	0,45	0,77	1,13	1,14	9,52	8,55	8,06	559	88	40,00
32	1,76	2,31	2,04	3,41	12,5	10,3	16,4	546	71	60,00
33	2,15	2,58	2,14	3,65	12,3	14,0	14,7	528	65	65,00
34	2,27	2,31	2,04	3,58	13,8	13,6	15,8	638	76	70,00
35	1,23	1,00	0,88	1,70	14,4	15,4	13,3	463	78	35,00
36	0,57	0,56	0,53	0,80	10,3	11,1	5,41	735	101	53,00
37	0,25	0,44	0,26	0,49	11,9	16,9	16,7	493	99	25,00
38	0,27	0,34	0,30	0,42	10,8	9,43	6,76	556	121	21,00
39	0,30	0,30	0,24	0,46	13,7	15,4	8,26	550	120	21,70
40	1,65	1,39	1,51	1,98	9,63	10,5	11,6	730	88	68,70
41	0,20	0,20	0,20	7,26	0,50	1,60	2,90	200	24	70,00
42	0,20	0,20	0,20	10,42	1,00	1,50	1,00	300	35	75,00
43	0,20	0,20	0,20	1,16	0,00	0,00	0,00	570	85	45,00
44	0,68	0,87	0,00	0,91	8,13	10,9	0,00	695	90	60,00
45	0,50	0,42	0,36	0,59	12,3	11,4	12,7	590	152	15,00
46	0,73	0,47	0,49	0,89	11,2	7,94	14,1	520	116	20,00
47	0,89	0,76	0,54	1,16	13,2	13,3	11,1	500	112	20,00
48	2,76	2,21	1,57	2,96	10,4	10,6	11,7	645	83	60,00
49	2,54	4,17	1,66	4,44	15,7	12,4	15,7	620	72	75,00

*Continuing Table 6.3

50	4,40	5,50	3,01	5,65	13,8	10,4	12,9	555	64	75,00
51	0,20	0,20	0,20	7,53	0,40	0,40	0,20	503	58	75,00
52	1,77	2,26	0,07	2,55	19,8	15,6	66,7	550	64	75,00
53	2,77	3,77	0,11	3,91	16,4	18,8	90,9	615	71	75,00
54	0,20	0,20	0,20	2,39	0,10	0,30	0,00	805	104	60,00
55	0,21	0,27	0,53	0,53	8,77	8,20	8,40	725	162	20,00
56	0,22	0,36	0,78	0,78	8,26	9,62	6,45	665	149	20,00
57	0,56	1,68	2,35	2,62	12,1	10,5	10,4	800	92	75,00
58	1,71	2,30	3,32	3,42	11,4	7,89	11,6	710	82	75,00
59	0,64	0,94	0,92	1,22	10,8	11,0	9,76	665	86	60,00
60	0,50	0,64	0,49	0,79	11,7	9,40	9,80	800	92	75,00

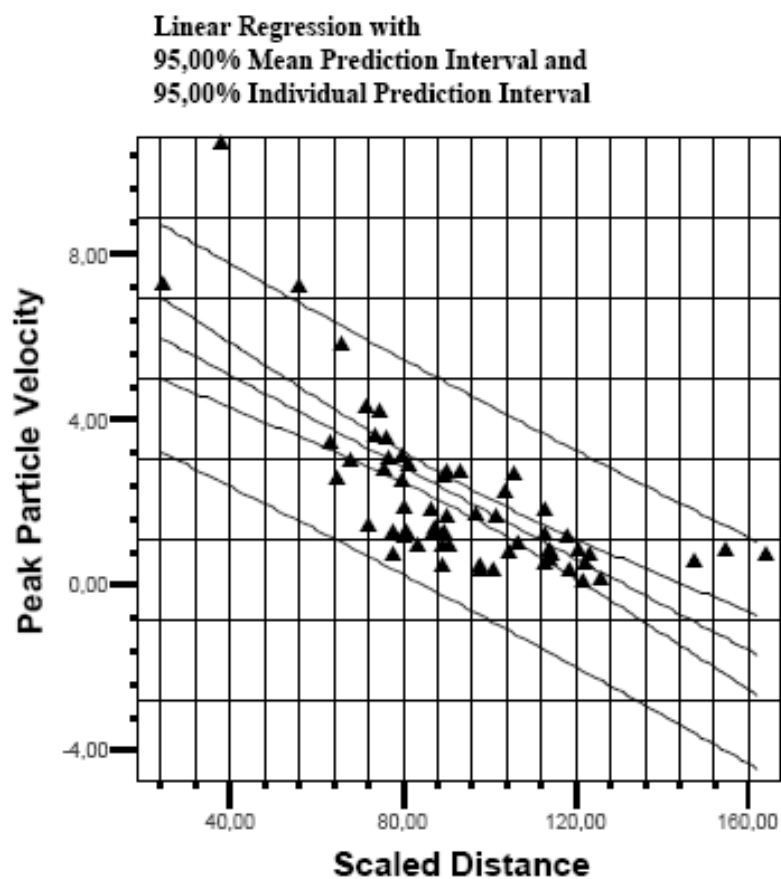


Figure 6.4 Peak particle velocity versus scaled distance

The result of the regression and correlations are presented in Table 6.4, which shows the statistical calculations using the full data set. It presents standard summary output from SPSS (version 15.0) regression analysis performed within the data analysis tool. The R^2 quantity is a basic measure of the quality of the fit. In this case a value of 0.594 indicates that 59.40% of the PPV variability is explained by the linear regression. It must be taken into consideration that its use in the blast design could give erratic results. In order to support this formula, more events should be monitored in various directions and the regression analysis should be updated, considering the results of further measurements. At the end of evaluation of data pairs, an empirical relation, which gives 50% prediction line with a reasonable correlation coefficient was established between PPV and Scaled distance. The prediction of particle velocity requires that average and upper bound values should be well known.

Table 6.4 SPSS regression analysis software outputs

Variables Entered/Removed ^b			
Model	Variables Entered	Variables Removed	Method
1	Inx ^a	.	Enter

a. All requested variables entered.
b. Dependent Variable: Iny

Model Summary ^b									
Model	R	R Square	Adjusted R Square	Std. Error of the Estimate	Change Statistics				
					R Square Change	F Change	df1	df2	Sig. F Change
1	,771 ^a	,594	,587	,50441	,594	84,857	1	58	,000

a. Predictors: (Constant), Inx
b. Dependent Variable: Iny

ANOVA ^b					
Model		Sum of Squares	df	Mean Square	Sig.
1	Regression	21,590	1	21,590	,000 ^a
	Residual	14,757	58	,254	
	Total	36,347	59		

a. Predictors: (Constant), Inx
b. Dependent Variable: Iny

Coefficients^a

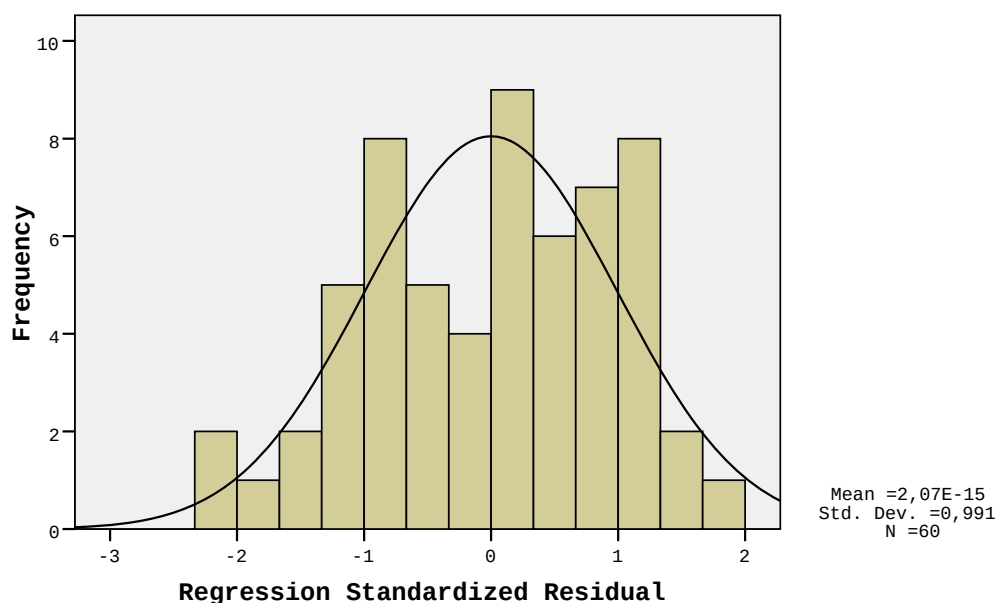
Model		Unstandardized Coefficients		Standardized Coefficients	t	Sig.
		B	Std. Error	Beta		
1	(Constant)	9,209	,955		9,645	,000
	lnx	-1,951	,212	-,771	-9,212	,000

a. Dependent Variable: lny

Residuals Statistics^a

	Minimum	Maximum	Mean	Std. Deviation	N
Predicted Value	-,7156	3,0096	,4338	,60493	60
Std. Predicted Value	-1,900	4,258	,000	1,000	60
Standard Error of Predicted Value	,065	,287	,085	,035	60
Adjusted Predicted Value	-,7224	3,5018	,4410	,63989	60
Residual	-1,17373	,85170	,00000	,50012	60
Std. Residual	-2,327	1,688	,000	,991	60
Stud. Residual	-2,477	1,706	-,006	1,019	60
Deleted Residual	-1,51943	,86990	-,00717	,53183	60
Stud. Deleted Residual	-2,596	1,736	-,010	1,033	60
Mahal. Distance	,000	18,130	,983	2,617	60
Cook's Distance	,000	1,470	,036	,189	60
Centered Leverage Value	,000	,307	,017	,044	60

a. Dependent Variable: lny

Histogram**Dependent Variable: lny**

After regression analysis had been performed, maximum charge that could not cause any damage in the residential areas in the vicinity of the mine site. DIN 4150 German Standard was used for the calculation of the permissible amount of charge. However, the value of 3mm/s given for the sensitive houses in the DIN 4150 was changed because of the fact that the structure of buildings around the mine site are not of the same durability as the ones in Germany. Therefore, the values of 3mm/s were considered as 2mm/s in the scope of this thesis. All the calculations related to standard of DIN 4150 and USBM are shown in Table 6.5 and 6.6 respectively.

Table 6.5 Charge amount per delay calculated for the different values based on the USBM and DIN 4150 damage criteria

Distance (m)	Damage Beginning Limit (mm/s)		Maximum Charge (kg)	
	DIN 4150 (mm/s)	USBM (mm/s)	DIN 4150 (mm/s)	USBM (mm/s)
50	2	20	0,40	2,36
100	2	20	1,58	9,42
150	2	20	3,55	21,20
200	2	20	6,32	37,70
300	2	20	14,23	84,80
400	2	20	25,29	150,82
500	2	20	39,52	235,65
600	2	20	56,91	339,34
700	2	20	77,46	461,87
800	2	20	101,18	603,26
900	2	20	128,00	763,51

Table 6.6 Particle velocity values calculated according to the different charge amount per delay

Distance (m)	Charge Amount Per Delay (kg)			
	50	55	60	70
200	28,78	32,54	36,41	44,42
300	10,11	11,44	12,79	15,61
400	4,81	5,45	6,09	7,43
500	2,70	3,06	3,43	4,18
600	1,69	1,91	2,14	2,61
700	1,14	1,29	1,44	1,76
800	0,81	0,91	1,02	1,24
900	0,59	0,67	0,75	0,91

When frequency values belonging to the vibration intensity induced by blasting is considered, 60% of the frequency values are in the range of 4-12Hz and 40% of the frequency values are in the range of 13-20Hz. As a result, it can be said that buildings and other structures were vibrated much more than the value read in the seismograph because of the resonance. The frequency distribution of the blast shots can be seen in Figure 6.5.

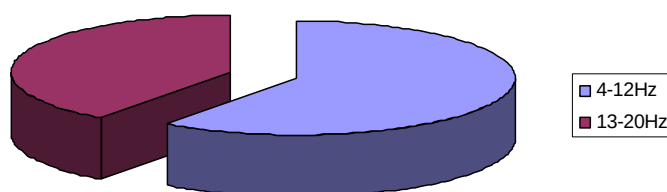


Figure 6.5 Frequency distribution of the blast shots

It follows from the calculations made with the data obtained that if the amount of 50 kg ANFO should be used as charge weight per delay in order not to exceed the vibration limit around the distance of 600m, which contains most of the residential areas from where the complaints arises then the PPV value will be recorded as 1.69 mm/s, and this value is below the value of acceptable limits proposed by USBM RI 8507. Although the ground vibration intensity measured is in acceptable limit, 5 more shots were conducted in order to decrease the precision of ground motions by humans (Table 6.7).

Table 6.7 Predicted PPV values

Shot point	Peak Particle Velocity (mm/s)				Predicted PPV	Scaled distance (m/kg ^{0.5})
	Vertical	Longitudinal	Transverse	Vector sum		
1	0,70	0,78	0,58	0,92	1,10	100
2	1,99	2,14	2,08	3,07	2,10	78
3	0,36	0,40	0,48	0,56	1,16	98
4	0,93	0,66	1,03	1,40	1,19	97
5	0,65	0,77	0,99	1,20	1,54	88

6.5.2 Application of Water Deck in Production Blasting

After the safe charge amount had been determined, the water deck was introduced within a bore hole in order to see the difference between conventional blast and blasting with decking. The normal production blast pattern was not interrupted during the site survey. As mentioned before, the decking application has some advantages over conventional blasting such as decreasing in the ground vibration intensity, dust amount, back break and noise. However, in the beginning of the site survey, the proper material needed to be chosen in order not to get the hole wet. Before the site survey was started, the proper material for decking was investigated and the cylindrical shaped plastic material was considered as the best decking material for the site survey at issue. After that 15 applications were trialed in the so-called quarry. The evaluation was done by considering the ground vibration intensity, dust amount and back break observations. Fragmentation size analysis was also experienced. All of the applications were done within the blast hole diameter of 89mm. blast hole length varied from 6m to 14m. The loading of the blast holes required attention because of the danger of piercing the water deck.

The loading was started with the booster and a detonator. After that the adequate ANFO was charged for the given charge length and then the water deck was lowered within a borehole by using string with a hook. Firstly the hook was hooked the string around the top of the material then it was lowered within a blast hole. The hook and string were taken back out side the blast hole. After the water deck had been introduced within a blast hole, second booster and the detonator with same delay time as the detonator introduced first were lowered within a blast hole then the adequate ANFO was again charged within a blast hole. Lastly the drill cutting was charged in the blast hole as stemming material. The schematic diagram of application of water deck can be seen in Figure 6.6

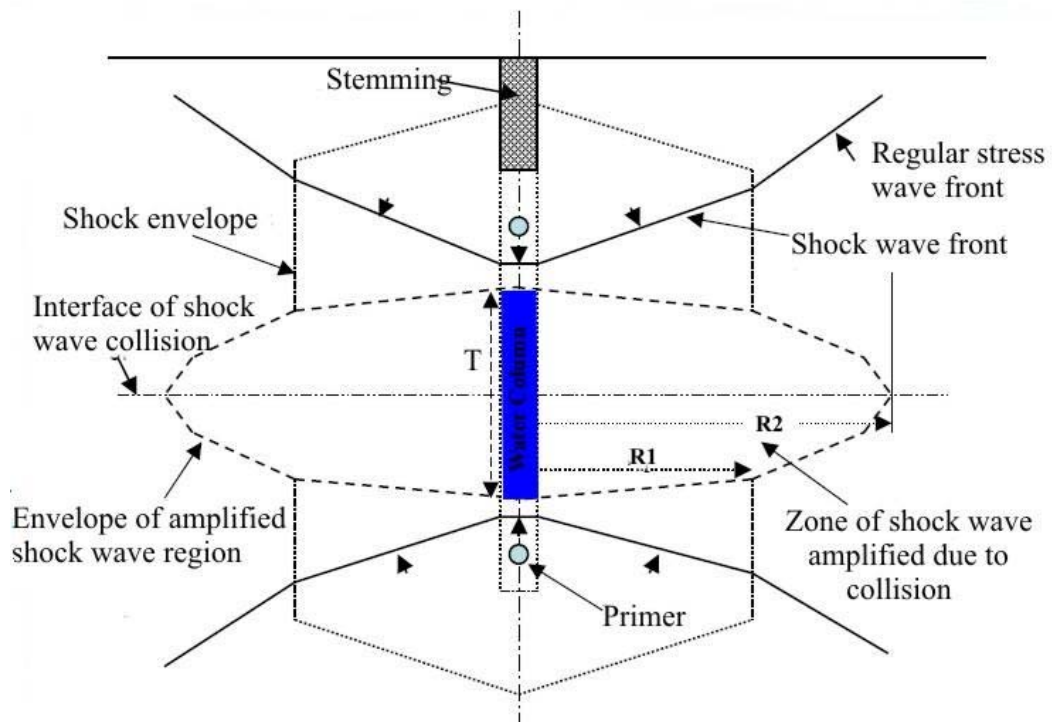


Figure 6.6 Cross section of the application of water deck in bench blasting

6.5.2.1 Decking Material

After the long investigations for determining the best decking material, the plastic based cylindrical shaped material was considered as the most proper one for the applications conducted in Cimentas lime stone quarry. The so called decking material has three major parts which are of importance in terms of confining the water volume within the material. First part is the short hose with narrow diameter. This hose's diameter was chosen quite narrow since the connection of the hose and the plastic was getting difficult to connect because of the wider hose diameter. Second part of the material is the plastic which contains the calculated amount of water within it. During the site survey two different volumes of Water deck were used in order to see the difference in the so called outputs of blasting. Third part of material is the valve attached to the hose in order for the water not to leak. The whole decking material filled with water can be seen in Figure 6.7 and 6.8 respectively.



Figure 6.7 Production blast hole with 89mm diameter, ANFO, booster with detonator and a water deck (length of 50cm)



Figure 6.8 Application of water deck by using string with hook

6.5.3 Site Survey with Water Deck

During the studies carried out in Cimentas limestone quarry, 20 applications were trialed with Water Deck. These applications were evaluated in terms of ground vibration intensity, back break, noise and fragmentation. Most of the surveys were conducted in two major slopes situated in Cimentas limestone quarry as seen in Figure 6.9.



Figure 6.9 Slopes where water deck application were trialed in Cimentas limestone quarry

6.5.3.1 Economical Analysis of Application of Water Deck

ANFO: 1.02 YTL

Booster named Emulite: 3.88 YTL

Electrical detonator (30ms): 2.22 YTL

Nonel detonator: 3.11 YTL

Price of water deck: 1.0 YTL

The diameter of the blast holes at Cimentas limestone quarry is 89mm. Thus, it can be easily calculated that 5kg of ANFO is needed in order to charge per meter of the hole. In conventional blast, adequate ANFO amount, booster and a nonel detonator are normally used. From this;

In order to charge one meter of hole with ANFO and other necessary equipment such as booster and a nonel detonator. The cost would be;

$$3.11 + 3.88 + (1.02 \times 45) \\ = 52.89 \text{ YTL (for 12 m bench height)}$$

When blasting with water deck is considered, the charging of the blast hole is slightly different from conventional one. The cost of the application of water deck is as follows;

$$3.88 \times 2 + 2.22 \times 2 + 43.35 + 1 \\ = 56.55 \text{ YTL (for 12m bench height and a water length of 50cm). Table 6.8 indicates the ANFO amount to be removed and the water amount to be used.}$$

Table 6.8 Total ANFO amount to be removed and water amount to be used

	5%	10%	15%	20%	25%	30%	35%	40%
ANFO amount need to be removed (m ³)	0.03 27kg	0.06 54kg	0.09 81kg	0.12 108kg	0.15 135kg	0.18 162kg	0.21 189kg	0.24 216kg
Water deck length	42.1	84.2	126.2	168.3	210.4	252.4	294.6	336.6
Water amount need to be used (dm ³)	2.5	5	7.5	10	12.5	15	17.5	20

Location 1: Next to transmitting line

Number of holes: 16

Unit charge per hole: 5kg/m

Initiation system: electrical detonator

Bench height: 12m

Spacing x burden: 3.5mx2.5m

Water deck length: 50cm

Stemming length: 3m

Total ANFO saving: 40kg

Comments: not economical

Table 6.9 outputs of the first trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.50	0.63	0.44	0.74	76.4	Almost nil	dusty

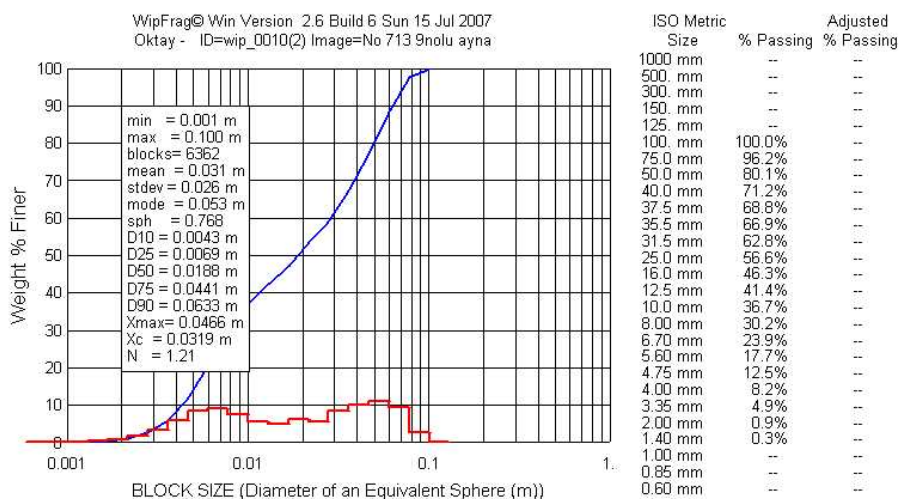


Figure 6.10 Fragmentation distribution of first trial calculated by WipFrag

Table 6.10 outputs of the second trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.47	0.24	0.04	0.51	84.4	Clean face, back break of 0.5m on one side	Negligible

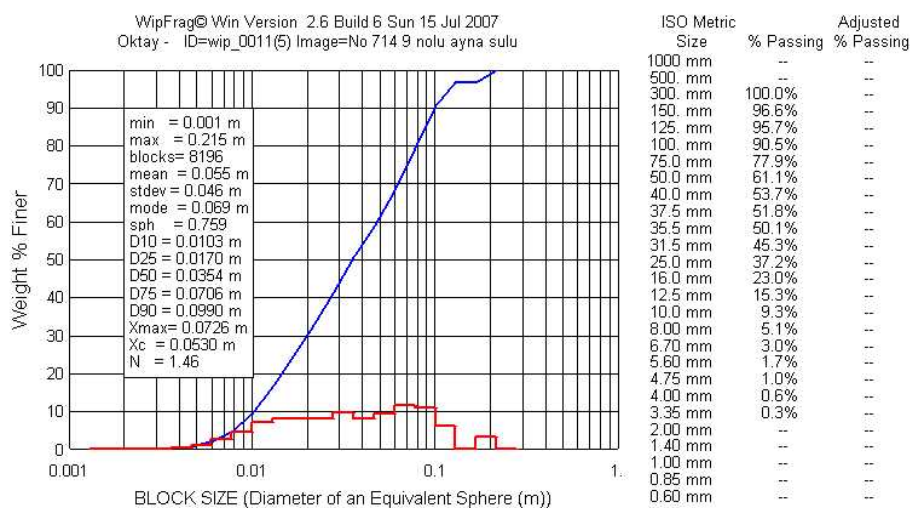


Figure 6.11 Fragmentation distribution of second trial calculated by WipFrag

Table 6.11 Outputs of the third trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.46	0.47	0.48	0.67	79.0	nil	Medium dusty

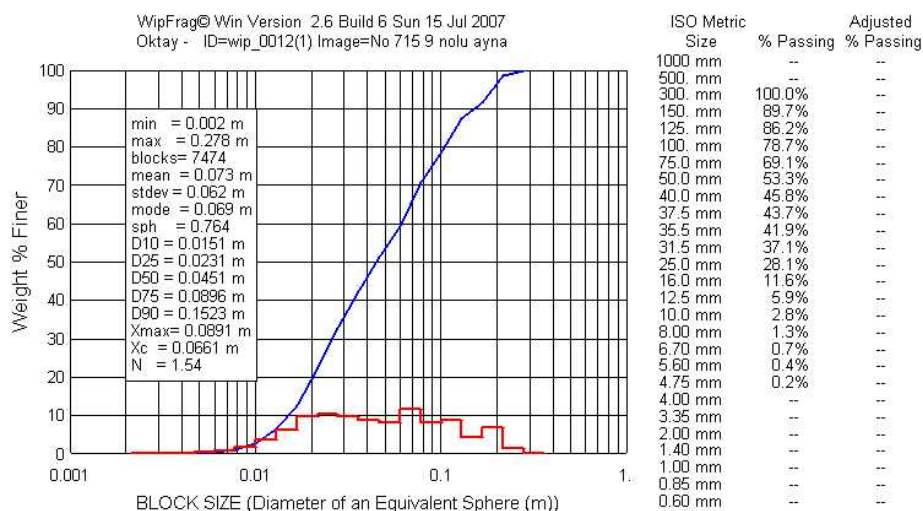


Figure 6.12 Fragmentation distribution of third trial calculated by WipFrag

Table 6.12 Outputs of the forth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.49	0.37	0.46	0.59	77.8	nil	dusty

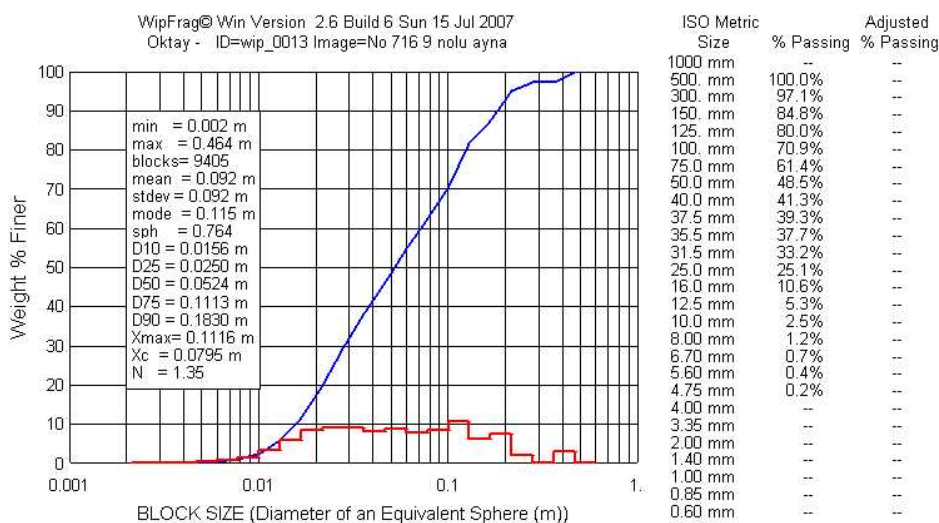


Figure 6.13 Fragmentation distribution of forth trial calculated by WipFrag

Table 6.13 Outputs of the fifth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.44	0.53	0.51	0.58	73.8	nil	Dusty

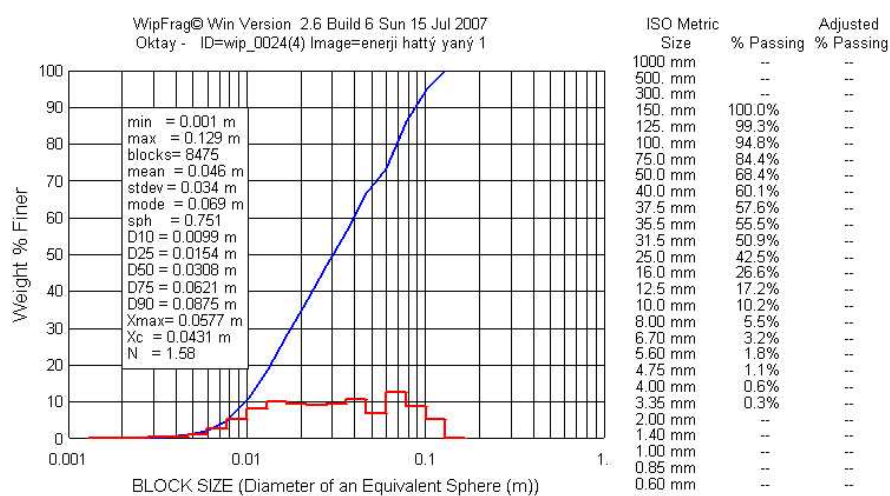


Figure 6.14 Fragmentation distribution of fifth trial calculated by WipFrag

Location 1: Long slopes

Number of holes: 16

Unit charge per hole: 5kg/m

Initiation system: electrical detonator

Bench height: 15m

Spacing x burden: 3.5mx2.5m

Water deck length: 50cm x 2

Stemming length: 3m

Total ANFO saving: 80kg

Comments: barely economical

Table 6.14 Outputs of the sixth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.80	0.64	0.71	0.91	81.8	nil	dusty

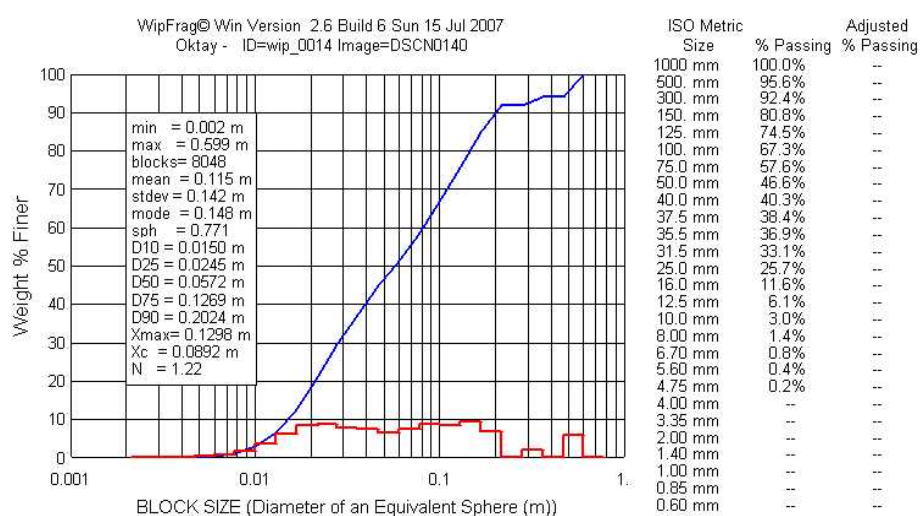


Figure 6.15 Fragmentation distribution of sixth trial calculated by WipFrag

Table 6.15 Outputs of the seventh trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.83	0.74	0.49	0.98	81.8	Almost nil	dusty

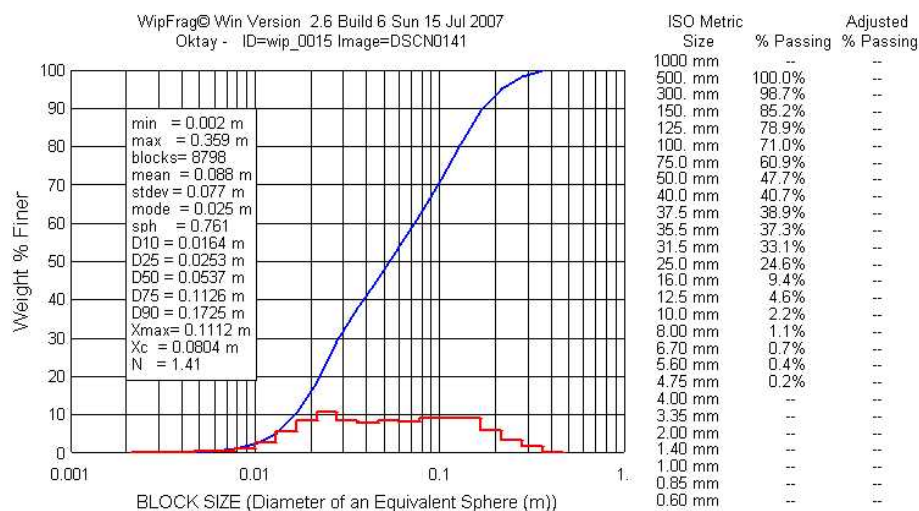


Figure 6.16 Fragmentation distribution of seventh trial calculated by WipFrag

Table 6.16 Outputs of the eighth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.55	0.64	0.81	1.00	87.8	nil	dusty

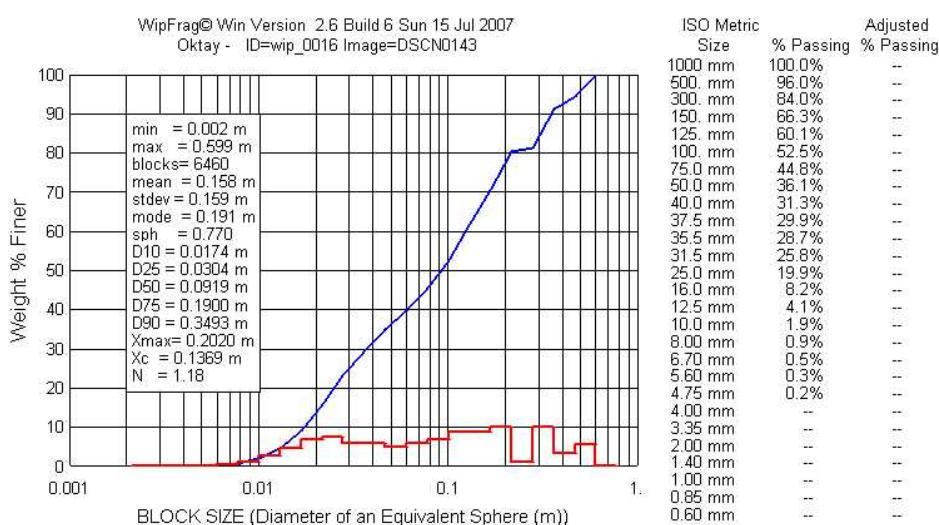


Figure 6.17 Fragmentation distribution of eighth trial calculated by WipFrag

Table 6.17 Outputs of the ninth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.80	0.57	0.91	1.05	79.1	nil	Medium dusty

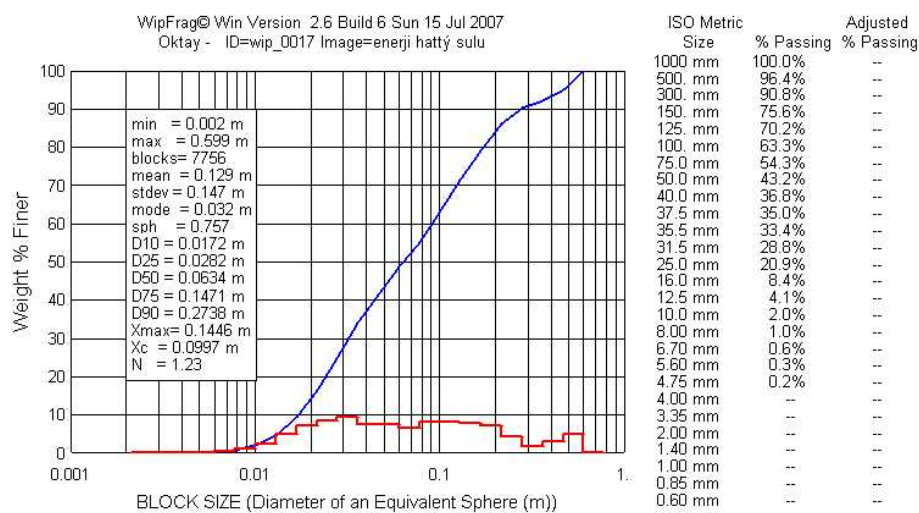


Figure 6.18 Fragmentation distribution of ninth trial calculated by WipFrag

Table 6.18 Outputs of the tenth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.63	0.71	0.45	0.77	80.0	nil	Negligible

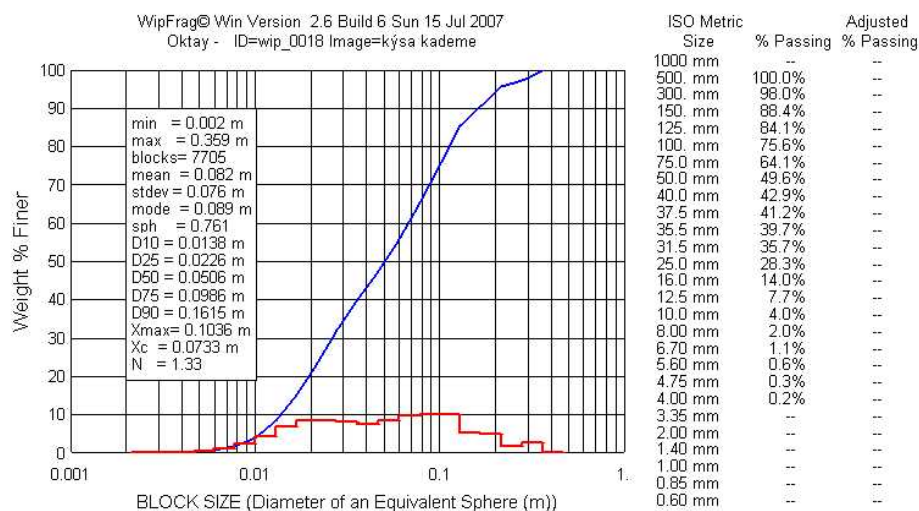


Figure 6.19 Fragmentation distribution of tenth trial calculated by WipFrag

Location 1: Short slopes

Number of holes: 16

Unit charge per hole: 5kg/m

Initiation system: electrical detonator

Bench height: 6m

Spacing x burden: 3.5mx2.5m

Water deck length: 50cm

Stemming length: 3m

Total ANFO saving: 40kg

Comments: barely economical

Table 6.19 Outputs of the eleventh trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.19	0.23	0.34	0.38	68.9	nil	Negligible

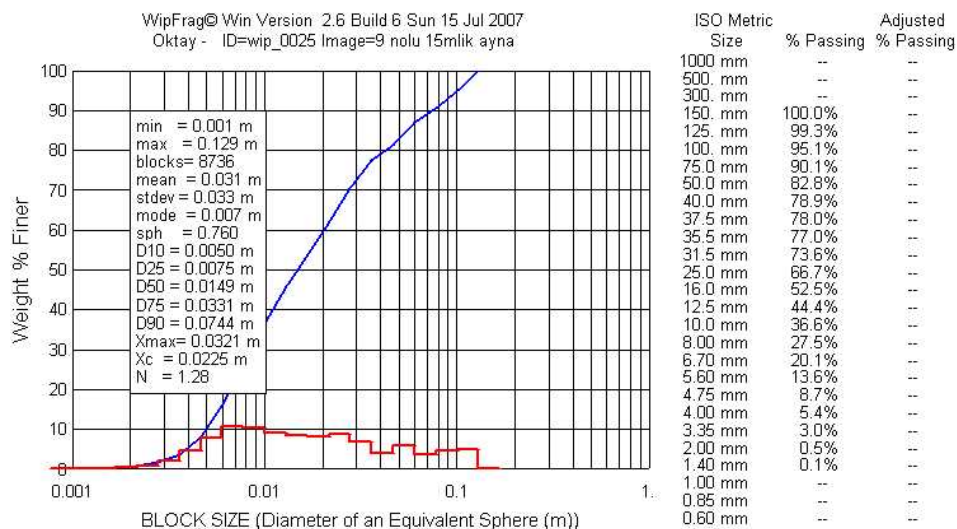


Figure 6.20 Fragmentation distribution of eleventh trial calculated by WipFrag

Table 6.20 Outputs of the twelfth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.25	0.44	0.26	0.49	65.9	nil	Negligible

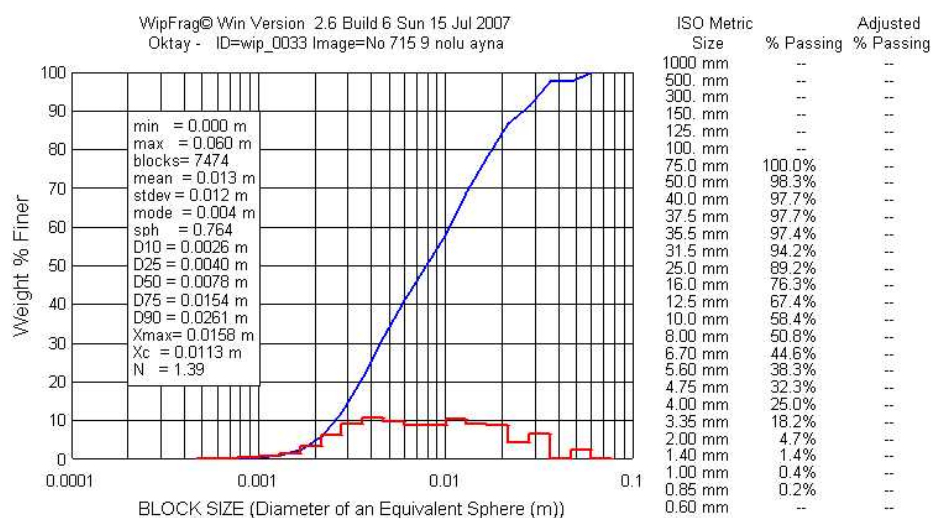


Figure 6.21 Fragmentation distribution of twelfth trial calculated by WipFrag

6.21 Table outputs of the thirteenth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.27	0.34	0.30	0.44	72.9	nil	Dusty

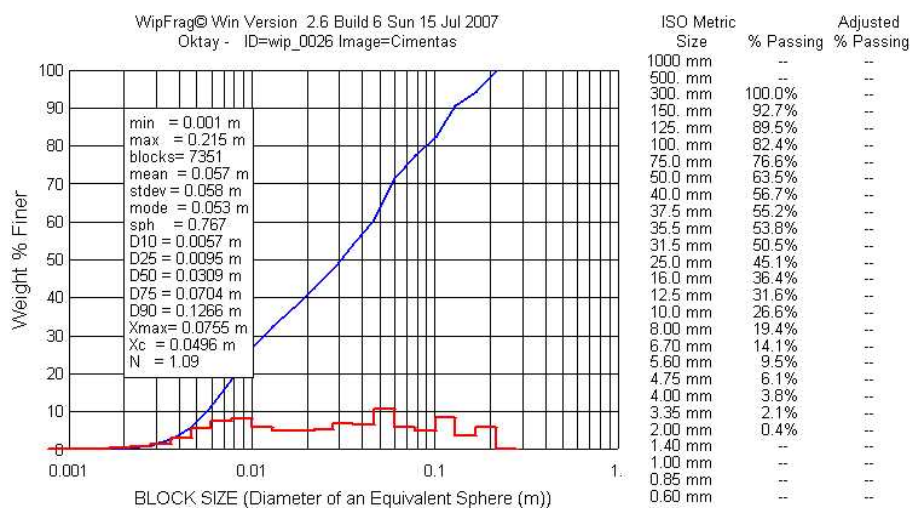


Figure 6.22 Fragmentation distribution of thirteenth trial calculated by WipFrag

Table 6.22 Outputs of the fourteenth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.30	0.30	0.24	0.43	73.0	nil	Negligible

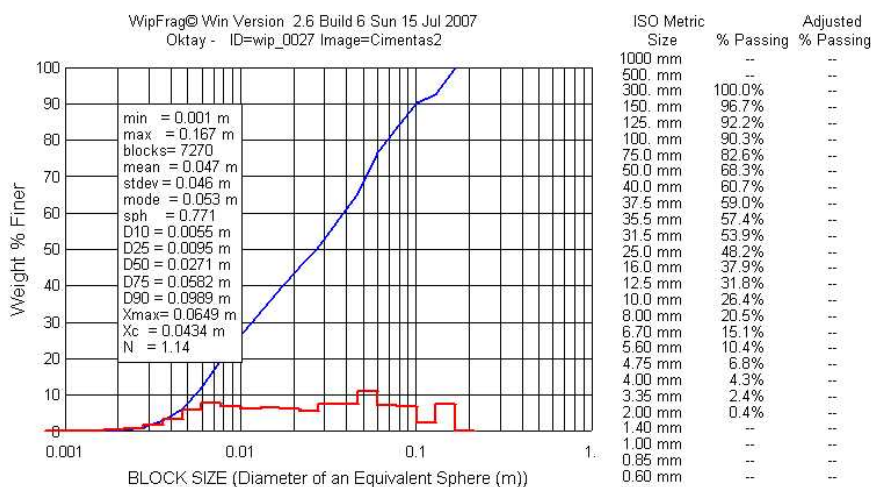


Figure 6.23 Fragmentation distribution of fourteenth trial calculated by WipFrag

Table 6.23 Outputs of the fifteenth trial

Location of the shot point	Peak Particle velocity (mm/s)				Noise (30m) dBA	Back break based on observations	Dust amount based on observations
	Trans.	Vert.	Long.	Vector sum			
Next to transmitting line	0.20	0.22	0.26	0.32	69.1	nil	Almost nil

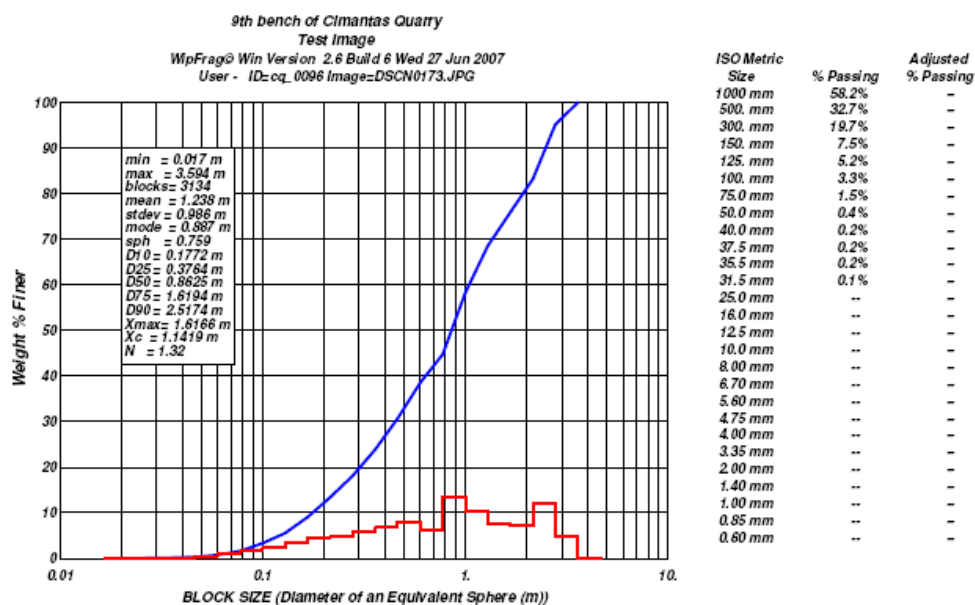


Figure 6.24 Fragmentation distribution of fifteenth trial calculated by WipFrag

6.5.4 Definitions of WipFrag Statistics

Granulometry is the measurement of particle sizes and shapes. The science of granulometry, when applied to blast fragmentation, has been developed to an advanced stage in earlier studies of comminution (crushing and grinding), petrology (study of rocks and minerals under the microscope), soil mechanics (classification of soils), metallography, powder technology, and other fields. A confusing range of alternative definitions exists even for such fundamental items as fragment size, which can be measured in terms of projected lengths, diameters of circles with the same area or in several other ways. The choice becomes even more complex when selecting statistics to represent fragment size distributions and fragment shapes.

C_u = Coefficient of Uniformity = D_{60}/D_{10} , a measure of the slope of the cumulative weight % curve between the 60- and 10-percentiles.

D_n = Nominal diameter, or equivalent spherical diameter, i.e. the diameter of a sphere with the same volume as that computed for the fragment.

D_{10}, D_{25} etc. = Percentile sizes. For example D_{10} is the ten-percentile, the value of D_e for which 10% by weight of the sample is finer and 90% coarser. In terms of sieving, D_{10} is the size of sieve opening through which 10% by weight of the sample would pass.

D_{50} = The Median or 50-percentile, the value of D_n for which half the sample weight is finer and half coarser.

blocks = Number of net elements detected in the NET image

max = Maximum size of fragment in the image [D_n (m)]

mean = Arithmetic mean (average) fragment size, equal to the sum of all equivalent spherical diameters divided by the total number of particles [D_{av} (m)]

min = Minimum size of fragment in the image [D_n (m)]

mode = Most common sized particle, the geometric mean D_n size class interval for the class containing the greatest number of net elements (fragments) [D_n (m)]

n = Rosin-Rammler (and Gaudin-Schuhman) Uniformity Coefficient, equal to the slope of the Rosin-Rammler straight line fitted to the data in log-log co-ordinates.

Sphericity = D_n/D_s , the ratio of equivalent spherical diameter to the diameter of a circumscribing sphere (long axis of the fragment)

stdev = Standard deviation of fragment size D_{av}

X_c = Characteristic Size, the intercept of the Rosin-Rammler straight line fitted to the WipFrag D_n data in log-log co-ordinates. This is equivalent to the $D_{63.2}$.

X_{max} = Gaudin-Schuhman characteristic size, the intercept of the 100 % passing and the slope of the Gaudin-Schuman straight line.

CHAPTER SEVEN

CONCLUSIONS AND FUTURE WORK

This section draws general conclusions of the thesis. Original discussions have been presented earlier. In this thesis, one of the methods applied in blasting operations called Air Deck and its effects on environmental products induced by blasting were investigated.

Firstly, some fundamental properties of blasting were examined. Secondly the blast pattern used in Cimentas limestone quarry was examined and it was decided that the blast pattern currently used by the contractor named NITROMAK is quite optimized in terms of the ground vibration effects on the residential areas around the mine site. After the site constants had been determined for Cimentas limestone quarry. The optimal charge per delay was calculated for different benches situated in Cimentas lime stone quarry.

After all the surveys conducted regarding determination of the safe charge amount were completed, the water deck was introduced within a blast hole with the help of string with hook. 15 blasts were trialed by the application of water deck and it was concluded that;

- There was a reduction in the ground vibration intensity because of the less explosive amount used in the round
- There was also reduction in the noise amount again, because of the less explosive used
- There was a reduction in the dust amount but this reduction was not enough for this method to be applied in industrial scale.
- The fragmentation size remained in the permissible level when the 50cm length of water deck was used. Moreover, when the 120cm water deck was used, the toe problems were encountered.

- The application process were also of risk because of the fact that the water deck can get punctured in any time due to the coarse material around the blast hole
- This situation made the charging process slower and time consuming.
- When considered, water is not a desirable parameter within a blast hole because of the fact that ANFO can get badly affected by the water.
- The water deck longer than 90cm could not be introduced within a blast hole because of the high risk.
- The aim of this application was to make use of the energy transmitted on the water deck since the liquids cannot be pressed under the pressure. However, the diameter of the water deck should be almost same as the diameter of the blast hole in order to get this benefit. However, the diameter of the water deck could not be same as the diameter of the blast hole because of the fact that the walls of the blast hole is generally of quite coarse. This meant that puncturation of the water deck could be encountered in any time.
- In future work, site surveys regarding decking systems will be conducted by using air deck as it is carried out all over the world.

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