PHENOMENOLOGY OF CONTROL BLASTING IN CLOSE PROXIMITY TO DENSELY POPULATED COMMUNITIES-A CASE STUDY OF MOPANI'S AREA J OPEN PIT

By

Mwale Timothy, B.Min.Sc (UNZA)

A Dissertation submitted to the University of Zambia in fulfilment of the requirements for the award of the Degree of Masters in Mineral Sciences (M.Min.Sc)

The University of Zambia

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Declaration

I, **Timothy Mwale** do hereby declare that this work was produced by me, with valuable assistance from various people from Mopani Copper Mines Plc., African Explosives Limited (AEL) and the School of Mine. I therefore declare that, to the best of my knowledge, this document ably supervised by Professor Radhe Krishna has never been submitted anywhere else except to the University of Zambia for the award of Masters of Mineral Sciences (M.Min.Sc). All works of other researchers have been duly acknowledged.

Signature……………………………………………

Date…………………………………………………

Approval

This dissertation for Timothy Mwale is approved as the fulfilment of the requirement for the award of the degree of Masters of Mineral Science in Mining Engineering by the University of Zambia.

Supervisor: Professor R. Krishna

Signature: ………………………………………………………………………..

Date: ……………………………………………………………………………..

Internal Examiner: Dr. Victor Mutambo

Signature: ………………………………………………………………………..

Date: ……………………………………………………………………………..

Internal Examiner: Dr. Besa Bunda

Signature: ………………………………………………………………………..

Date: ……………………………………………………………………………..

External Examiner: Dr. F.A. Dacosta (University of Venda)

Signature: ………………………………………………………………………..

Date: ……………………………………………………………………………..

Dedication

This dissertation is dedicated to my family. Without their input and assistance, it would not have been possible with the existing challenges we have had to undergo. Your input has yielded fruit and will always be appreciated. Thank you so much.

I would further and most importantly acknowledge our father, God almighty for guiding me through the journey and his endless provision during this study. It is clear that a workman can work but without the Lord, it's in vain, **(Psalms 127:1) "Except the Lord Build the house, they labour in vain that build it: except the Lord keep the city, the watchman watches but in vain."** Thank you Lord.

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It is with heartfelt gratitude and pleasure to acknowledge all the people that contributed to the successful completion of this dissertation. Without their effort, this work could not have been completed, successfully.

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ABSTRACT

Blasting is the process of fracturing material through the detonation of industrial explosives placed into drilled holes suitably located to a free surface. Every blast hole containing a calculated amount of explosive charge undergoes impulsive loading when detonated and results in the dynamic transfer of a powerful shock wave to the surrounding rock. It is this shock wave that brings about rock breakage when the tensile stress is produced after the stress wave reflects on the free face or change in material density of rock material tensile strength. However, this shock wave comes along with unwanted destructive consequences such as ground vibration, noise and flyrock.

The consequences highlighted above are likely to affect the Wusakile Community in Kitwe as Mopani Copper Mines (MCM) Plc. wishes to expand the pit towards the community that lies within 150m from the pit rim. As part of the commitment given by MCM to the Zambia Environmental Management Agency (ZEMA), all blasting operations close to a community must maintain at least below the maximum vibration threshold of 10 mm/s with no flyrock. Furthermore, all noise levels must be kept below the maximum threshold of 134dB in order to reduce environmental concerns raised by the community. In order to meet the above requirement, the phenomenology of rock breakage and other factors such as the maximum projection distance, size of the fragment, shape of the fragment, angle of projection, the state of confinement of an explosive charge, design faults, deviation in implementation and unforeseen geological conditions were considered during the study.

Analysis of data showed that vibrations directed towards the community were kept below the maximum acceptable limit of 10mm/s. Furthermore, all higher vibration readings recorded at MCM Synclinorium offices were quickly dumped on the subsequent seismograph positions located at Kalela Basic School thereby confirming its dumping effect. In addition, no flyrock was observed to have gone beyond the safe blast limit of 100m and over 90% of crack monitors on cracked houses revealed no expansion during the project life.

This dissertation therefore illustrates the possibility of conducting blasting operations close to a community with improved fragmentation while mitigating all adverse effects to the environment by use of controlled blasting techniques. It further suggests the possibility of incorporating such blasting techniques and their threshold values onto the Zambian Regulations.

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DEFINITIONS AND ABBREVIATIONS

AEL- African Explsoives Limited

AMC- African Mining Consultants

ANFO- Ammonium Nitrate Fuel Oil

ANN- Artificial Nueral Network

BME- Bulk Mining Explosive limited

CMRS- Central mining Research Center

CRSD- Cube root of Charge weight

GSI- Goelogical Strength Index

KCM-Konkola Copper Mines

MCM-Mopani Copper Mines Plc

MSD-Mines Safety Department

NOMIS- Name and type of a seismograph

NONEL-Non Electronic Detonators

OSM-Office of Surface Mines in the United States of America

PPV- Peak Particle Velocity

RBS-Relative Bulk Strength

RWS-Relative Bulk Strength

RQD- Rock Quality Designation

SRSD- Squared root scaled distance

SABS- South African Bureau of Standard

UCS- Uniaxial Compressive Strength

VOD-Velocity of Detonation

VS4- Ventilation shaft number 4 at Mopani Copper Mines

ZEMA-Zambia Environmental management Agency

CHAPTER 1

INTRODUCTION

1.1 Preamble

Rock breakage in open pit mining operations is achieved mainly through blasting. This process involves the application of explosives to fragment the rock and is often referred to as a science as well as an art. In the early years, explosives development became a science as chemists worked very hard to perfect stable but powerful explosive mixtures that have proved to be the cheapest and quickest way of breaking the rock through conventional blasting methods. These methods involve drilling and blasting of large blast holes that require a minimum safe distance of 500m from the community in order to ensure safety of personnel, structures and equipment. In recent years, the need to minimize costs of blasting and control environmental adverse effects such as ground vibration, flyrock, noise, dust and fumes; have brought about carefully engineered blast designs. Zambia is no exception as the need to expand the mining industry has brought about more exploration that has resulted in further discovery of more mineral deposits. However, these minerals are located in close proximity to communities. This is the case with Mopani Copper Mines plc. This mine is owned by Carlisa Investments Corporation, a joint venture company registered in Zambia and comprising Glencore International AG (73.1% shareholding), First Quantum Minerals Ltd (16.9% shareholding) and ZCCM Investment Holdings (10% shareholding).

MCM is one of the largest copper producing mines in Zambia and contributes to the world's copper ore production. The mine runs two mining divisions namely Mufulira and Nkana sites. The former being in Mufulira town with four vertical shafts and two portals while the latter is situated in Kitwe where a new Synclinorium shaft is being sunk and is expected to start production in the year 2015. The other four active shafts in Nkana include Mindola North, Mindola Sub-Vertical, Central and South Ore Body. Due to the shallow depth of as low as 15m from surface and orientation of the ore body in most parts of Nkana, the mine also runs open pit mining operations.

Area J Open Pit is one of the satellite pits in Nkana mine site. This pit lies 100m away from Wusakile community. However, the mining plans are to expand the pit on the footwall which entails getting even closer to community. Hard rock mining of such pits in close proximity to the community requires control blasting techniques in order to break rock safely and successfully. Control blasting techniques utilize bench blasting parameters with electronic detonation that have proved to reduce the scattering effect due to reduced cooperating charge, well-planned firing sequence and direction of ground movement. In order to achieve the above, an understanding of the phenomenology of control blasting and associated factors such as the maximum projection distance, size of the fragmentation, angle of projection, and the state of confinement of an explosive charge are required. Furthermore, design faults, deviations in implementation and unforeseen geological conditions have to be considered. This dissertation was therefore aimed at ascertaining the likelihood of blasting operations in close proximity to Wusakile community of Kitwe with reduced and / or no environmental impacts. Furthermore, it was also the aim of this dissertation to verify the cost implication of using electronic detonators. This research will ascertain whether the existing cracks on the houses within Wusakile would expand due to blasting operations.

Results obtained from this research would be used to confirm the possibility of conducting blasting activities close to the community. This would therefore, justify the need to open up other pits such as Area K and Nose open pits which also lie close to the community.

1.2 Location and description of the research site

This research was conducted at Mopani Copper Mines Plc. As can be seen from Figure 1, this mine is located in Kitwe town of the Copperbelt Province which lies 354km north of Lusaka, the capital city of Zambia. Currently, Mopani Copper Mine Plc. owns a total of 11,217 hectares of Nkana Mining license area.

Figure 1: Location of the Research site in Kitwe, (Nation Online, 2014)

1.3 Problem Statement

Area J Open pit was the only satellite pit that was operational at Mopani Copper Mines Plc. at the start of this research. This pit has a mine life of five years therefore expected to run until 2016. However, it contains some minerals that lie on the footwall which happens to be on the western side of the pit as mining is conducted towards the north as shown in Figure 2.

Two options are presented in this research; Option 1 is to proceed with expansion of the pit while employing control blasting methods, while option 2 is to leave the ore hanging in the pit but this entails reducing the life of mine. Mining of this material just like any other open pit requires 80% drilling and blasting. However, as can be seen in Figure 2, blasting operations at Area J will reduce the safe distance from 500m as recommended by Mines Safety Department (MSD) to less than 100m away from Wusakile community, (Blasting Licence Training Manual, 1990).

The following are the structures surrounding the pit; Kalela Basic School which lies less than 100m away from, Konkola Copper Mines (KCM) / MCM Synclinorium offices at 60m away, vent shaft at 50m, compressor house at 60m and the residential houses in the Wusakile community at less than 200m away from the pit rim. The above structures are prone to adverse effects generated during blasting such as **flyrock, noise, dust and vibrations**. However, Area J open pit had about 1,400,000tons of ore sitting within the footwall that would restrict mining of the 60mB bench level hence reduce the life of mine by two years.

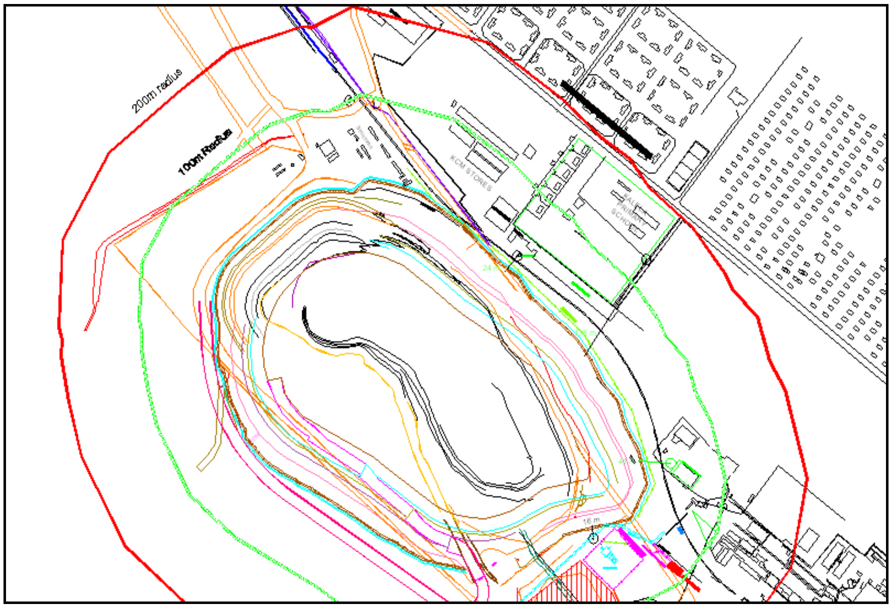


Figure 2: Area J Open Pit and it's radius of influence prone to environmental impact

Blasting operations in other countries such as South Africa have defined their threshold limits as indicated in Table 1, all private property close to an active blasting operation are vulnerable to damage should the limits exceed the threshold of 10mm/s:

Table 1: Maximum acceptable vibration (South African Bureau of Standards, 1990)

Status of Structure	**Maximum PPV(mm/s)**
Heavily reinforced concrete structure	120
Property owned by the concren performing blasting operations, where min or plaster cracks are acceptable	84
Strongly Masonary walls not affected by public concern	50
Commercial property in reasonable repair, where public concern is not an important consideration	25
Private property, where public concern is an important consideration and blasting conducted on a regular and frequent manner	10

Zambia currently does not have any laws regarding vibration control as most open pit operations have been located very far from densely populated communities. This consequently resulted into conventional blasting methods which do not raise any public environmental concerns. However, with the recent exploration that has resulted in more ore discoveries in close proximity to communities, the need to ascertain control blasting techniques has become vital especially where housing units are of low to medium standard as can be seen in Figure 3.

Figure 3: The condition of housing units in Wusakile close to the pit

According to the Environmental Impact Statement conducted for Area J open pit, all blasting operations close to the community must maintain a minimum of 10mm/s to the community (African Mining Consultants, 2009). In addition, there is need to ensure that no flyrock goes beyond the safe distance of 80m as there are structures with sensitive equipment such as can be seen in Figure 4 and 5. This is according to the 1990 South African Bureau of Standards provided for on Table 1.

Furthermore, all environmental nuisances such as flyrock, noise and dust must be kept to a minimum in order to sustain operations at Area J Open Pit. It was also observed that production was adversely affected as a result of poor fragmentation. Consequently, big boulders were generated as can be seen in Figure 6 thereby increasing the cost of mining because a rock breaker had to be hired, see Figure 7.

Figure 4: Infrastructure close to the pit perimeter

Figure 5: Synclinorium offices close to the pit

Figure 6: An excavator handling big boulders in the pit

Figure 7: Rock breaker downsizing big unbroken rock at Area J

With the understanding on the phenomenology of controlled blasting techniques, blast designs and methods are to be designed and implemented that will ensure the operation is addressing all environmental concerns while fragmenting rock to the correct size.

This research has, therefore, been initiated with the need to ascertain every possible way to control blast induced environmental nuisance and still be able to achieve better rock fragmentation.

Furthermore, the results of this research will form the basis for development of regulation on control blasting operations at close proximity to communities in Zambia.

1.4 Objectives of the Research:

The objectives of this study are:

a) To ascertain if controlled blasting in close proximity to a community is achievable

b) To establish the best-fit equation to compute safe charge weight per delay thereby determine the site characteristics of the peak particle velocity (PPV)

c) To determine ways of improving fragmentation without causing unsafe conditions to the nearby community.

1.5 Research Questions

In pursuing this research, the following questions were considered:

a) Why conduct blasting operations so close to the community?

b) What is the safe blast distance between the pit and the community?

c) How best has strain wave propagation (vibrations) been controlled in routine blasting at an open pit such as Area J Open Pit in Wusakile?

d) What are the basic considerations for success of the phenomenology of control blasting in close proximity to highly populated communities where blasting may cause public nuisance?

e) What are the prevailing site rock characteristics for optimum fragmentation?

f) What timing can be used to achieve better fragmentation?

g) Does the current fragmentation computation such as Kuzram consider control blasting and how may the expected size of volumes be accomplished?

h) What energy is expected in order to break the rock to its required size?

1.6 Significance of the Study

This research is vital for the successful implementation of control blasting methods and operations at Mopani Copper Mine Plc.

Controlled blasts with optimum fragmentation once achieved brings about reduced operating costs for the company due to reduced and or eliminated blast re-handling while successfully meeting the company's safety and environmental management policies.

The successful implementation of controlled fragmentation in blasting will greatly assist with mining of ore at Area J Open pit. Furthermore, the use of control blasting techniques will provide evidence on behalf of Mopani to the Chamboli community as the same techniques will be applied in the new pits, Area K and Nose open pits that also lie at less than 150m from the community. The technique will be used to conduct blasting operations in other pits close to the community owned by other mines.

This research further provides insights for Zambia Environmental Management Agency (ZEMA)'s revision of safe distance in mine blasting provided specified requirements are met.

1.7 Research layout

The following was the approach in undertaking this research:

a) Site investigations and review of geotechnical data

b) Understanding of blasting techniques currently used in the mining industry

c) Selecting of seismograph positions

d) Training of personnel to monitor vibrations

e) Designing of blasts and report writing format

f) Data collection, interpretation and analysis

g) Discussions, conclusion and recommendations

CHAPTER 2

LITERATURE REVIEW

This Chapter presents discussions on blasting as a process, phenomenology of rock blasting, different types of blasting techniques and blasting geometry.

2.1 Overview of Blasting

Blasting is the process in which industrial explosives are emplaced in blast holes suitably located to a free surface and detonated, (Dowding, 1992). Every blast hole containing a certain amount of explosive charge undergoes impulsive loading when detonated and results in the dynamic transfer of a powerful shock wave to the surrounding rock. The shock wave, upon reaching a free face or any change in material density reflects back to produce a tensile stress, (Dowding, 1992). Due to the low tensile strength of rock material, fragmentation takes place as can be seen in Figure 8.

Figure 8: Bench blasting (MAXAM, 2010)

During blasting operations, explosive energy is transferred into five primary components, Figure 9 shows explosives energy distribution and these include fragmentation, heave, air blast, and ground vibration and heat, (Sanchidrrian, Segarra, and Lopez, 2007).

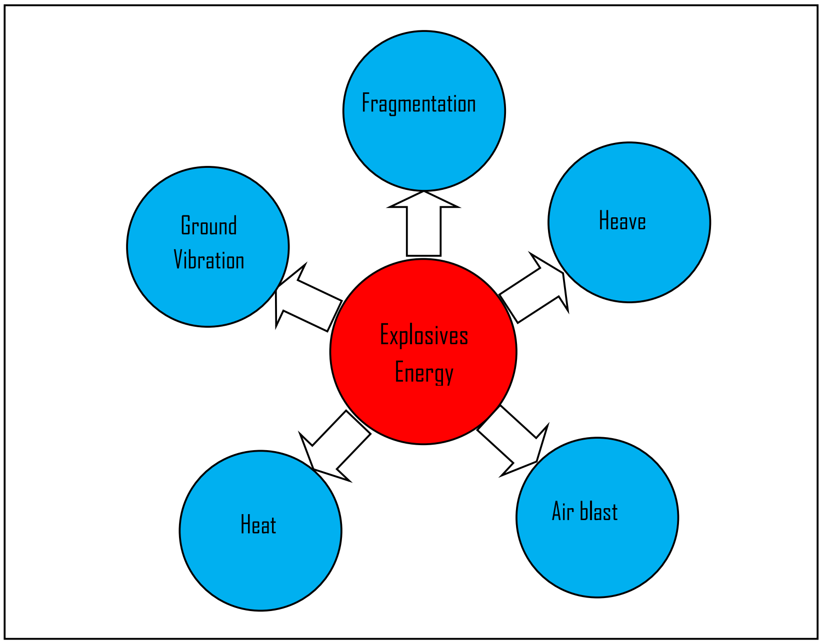


Figure 9: Explosive energy components

The aim of a good blasting engineer is to maximize explosive energy utilization in fragmentation and rock movement (heave) while minimizing energy loss to air blast, flyrock and ground vibration. Optimizing explosive energy use results into better blast fragmentation, reduced environmental impact and consequently reduced cost of operation.

2.1.1 Phenomenology of Rock Breakage

Blasting is the interaction of explosive induced waves and seismic rock waves. This interaction is based on the wave phenomena, transmission and reflection characteristics of interfaces between different media. It is believed that fragmentation of hard rock is the end result of compressive waves reflecting back into rock media as tensile waves. Some of the theories surrounding shock wave interaction include the following:

ZND theory: This theory is a fluid dynamic model that was arrived at by independently by Zeldovich in 1940, (Zeldovich, 1940). This theory embraces a thought of explosives detonating in the inert material, pressures get so high that material strength may become neglected. Furthermore, energy transport by heat conduction, viscosity, and radiation is negligibly small compared with the transport by motion. Figure 10 shows a plane, steady, unsupported detonation wave predicted by the ZND theory. The detonation, initiated by a pressure pulse from the piston at the left, is called unsupported because the piston velocity is less than the fluid or particle velocity of the explosive products. The detonation front is a shock wave, supersonic relative to the material ahead of it, so no signal precedes it. Compression heats the explosive, and rapid chemical reaction follows, then finally, reaction is complete, and the product gases expand as an inert flow, (William, 1979).

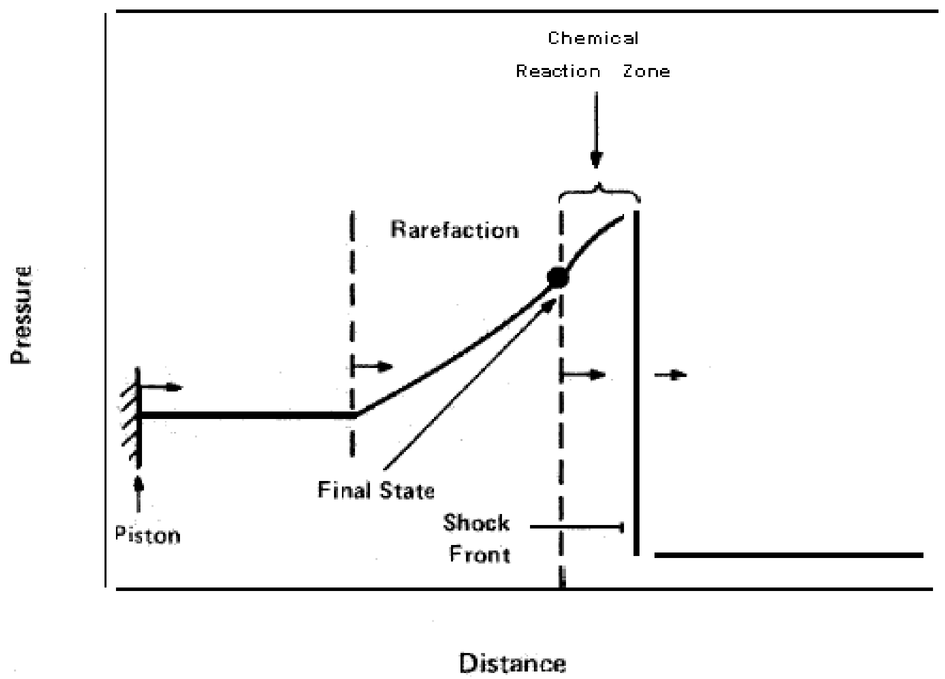


Figure 10: Plot of Pressure versus Distance for Detonation Wave (William, 1979)

The speed of the chemical reaction is seemingly an independent propagation of the detonation front that leads naturally to a division of the problem into two parts: the first study of the chemical reaction zone where the detonation process takes place, the second study looks at the acceleration of inert components, such as the metal of a hand grenade or the rock around a borehole, by the expansion of the explosive gases after the reaction is finished. The above two parts are interrelated even though they have been treated as separate problems until in the recent years. In most practical cases, the chemical reaction zone is so thin compared to the size of the explosive charge that its length is neglected completely in explosive performance calculations. The assumption is that the reaction takes place instantaneously at the detonation front and hence the expansion calculation of final explosive products as they push whatever material may enclose them. If this idealized calculation is compared with measurements, the effect of finite reaction zone length (or finite time of chemical reaction) appears as a small rise in pressure or velocity at the detonation front, (William, 1979).

The other theory known as the C-J theory calculates Detonation pressure (P_d) in N/m^2 at the C-J plane by using Equation 1:

$$P_d = \frac{\rho C_d^2}{4} \tag{1}$$

Where;

P_d is the detonation pressure given in N/m^2

ρ is the density of explosives in g/cm^3 and C_d is the velocity of detonation in m/s.

The detonation of explosives in cylindrical columns and in unconfined conditions leads to lateral expansion between the shock and C-J planes resulting in a shorter reaction zone and loss of energy, (Zhang et al., 2012).

This is why lower VOD is encountered in unconfined situations than in confined ones.

Detonation and interaction with rock upon detonation of an explosive charge causes the immediate rock surrounding the blast hole to get crushed due to the explosion pressure.

The outgoing shock wave, after passing through the crushed zone, travels at a velocity of between 2500m/s to 7000m/s and this results in tangential stresses that produce radial cracks, (Heiniö, 1999).

The pressure produced by the expanding shock wave from the blast source is compressive. When the shock wave reaches a free face, it will then reflect back towards the blast hole at a lower pressure but in the form of a tension wave through the rock. This is how the rock is broken in rock blasting.

Zhang in 2012 verified the C-J theory of detonation and based on the tests he conducted taking into consideration certain factors such as temperature and heat, it was concluded that the detonation pressure depended on the loading density of the explosive and hence derived Equation 2 to calculate detonation pressure.

$$P = P_{\max}\left(\frac{\rho_{O}}{\rho_{\max}}\right)^{2} \qquad (2)$$

Where

p is the detonation pressure (GPa) under the loading density ρ_o (g/cm^3) and p_{max} is the maximum detonation pressure (GPa) under the theoretical density ρ_{max} (g/cm^3), (Zhang et al., 2012).

The following phases describe detailed rock breakage through explosives detonation, (Brady and Brown, 2004).

- **Explosive initiation**; in this phase, detonation takes place by initiating the explosive charge in the blast hole. Upon initiation, the detonation wave rapidly transits to the orifice along the axis and the whole blast hole is filled with detonation gases under very high pressure and temperature. This causes the blast hole volume to expand dynamically concentrically thereby influencing the immediate periphery of the blast hole where high stress intensity results in the generation of the shock wave in the surrounding rock. This tends to cause the rock to behave mechanically as a viscous solid. The attenuation process also results in the reduction of the wave propagation velocity. Due to the above mentioned changes, the blast hole radius r_h expands to new radius r_s within the shock wave region and is believed to be twice the original radius (2r $_h$) as shown in Equation 3.

  $$r_s = 2rh \qquad (3)$$

  The rock in the near zone outside the shock wave region behaves as a non-linear elastic material hence new fractures are developed and propagated in the radially compressive stress field. This shock wave travels outwards as a compressive wave in all directions from the blast hole. This causes the immediate surrounding rock to get cracked and crushed.

- **Reflection of the shock Wave;** Crack formation continues in the radial direction resulting in a severely cracked annulus, called the rose of cracks. Generation of the radial extracts energy from the radial P- wave, resulting in reduction in the stress intensity. The radius r_t of the transition zone is about 4-6r_h. At this stage, the rock behaves linearly elastically and crack propagation continues at the speed of 0.2 to 0.25C_p. The P- waves rapidly outrun the crack tips, and propagation ceases. However, during transmission towards the free face, fractures are initiated in the cracks.

Figure 11 illustrates the movement of compressive waves that are reflected back as tensile waves from the free face, the outward bending compressive forces releases gasses at high pressure and a tensile wave is formed.

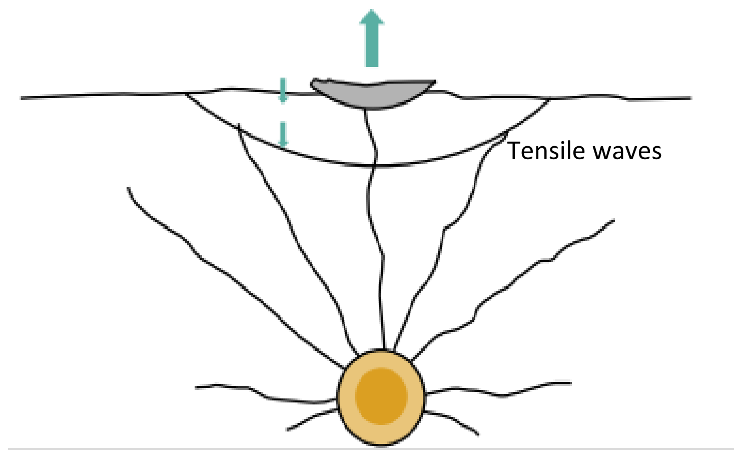


Figure 11: Shock wave reflects back as tensile waves (William and Huge, 2007)

These tensile waves in Figure 12 cause cracks which result into fragmentation. However, this has got to be within a certain distance between the free face and blast holes as the larger the distance, the more the wave energy is lost in the process hence reducing its influence on rock breakage.

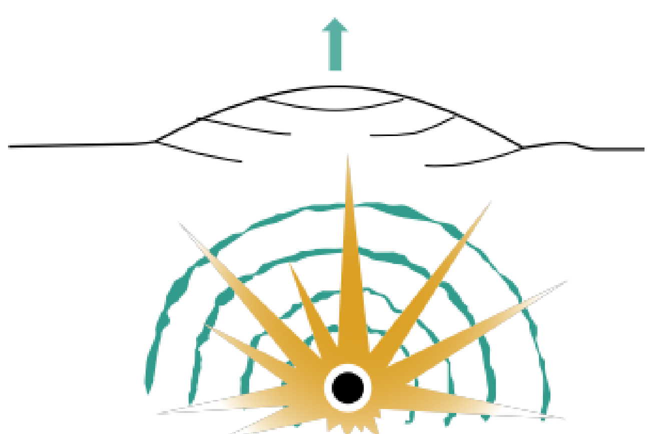


Figure 12: Rock Breakage, (William and Huge, 2007)

- **Gas expansion;** the cracks that are formed around the blast hole as illustrated by Figure 13 are filled in with superheated gas that is generated from the detonation of explosives. This gas is so hot that it occupies a space of around 10,000 to 20,000 times its original solid volume thereby exerting a pressure that is greater than 10,342,135 kPa.

  The fractured rock mass has a certain inertial that the gas pressure must overcome in order to initiate rock movement. This also depends on the timing and burdens between the blast holes.

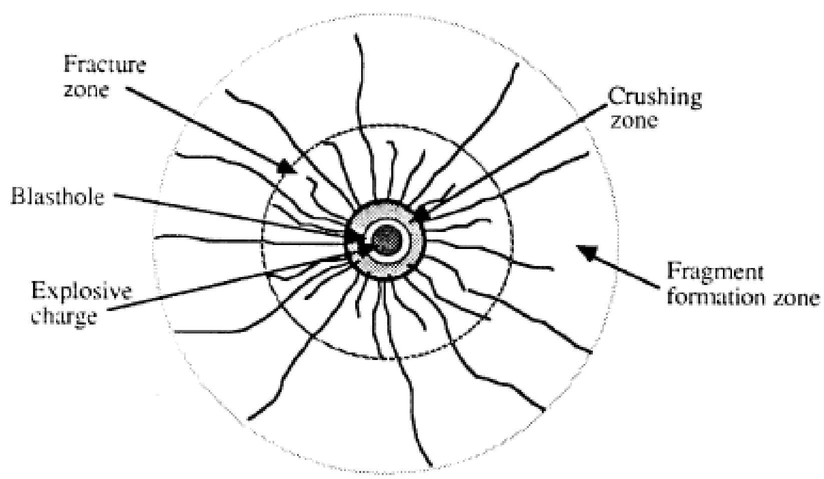


Figure 13: Seismic wave propagation after detonation (Whittaker et. al., 1992)

2.1.2 Rock distribution by Blasting

Once rock has been broken successfully, it disintegrates into different pieces. Such a process is referred to as fracturing. Fracturing is a complex process that depends on the character of loading and the material structure and their properties.

There are several theories on strength of materials and each theory has its own criterion. One well known theory of normal stresses is commonly used to describe rock breakage. However, it is important to note that all of them are based on the static loading, and they consider the material as homogenous and isotropic.

During blasting, the rate of strain is so high, and the times under stress are so short, that the stress property of material is considerably altered.

Fracturing and breaking of rock for multiple holes is fully dependant on the firing patterns. Firing patterns are designed so that every blast hole has enough free face to break into. This is determined by the delay given to each hole. Bernt Larsson of Nitro Nobel in 1974 has studied the impact of delay time on multiple holes and he concluded that the rock must be allowed to move 1/3 times the burden distance before the next detonation, (Larsson, 1974). He therefore recommended that rows may vary from 10ms/m for hard rock to 30ms/m for soft rock,

The energy generated during blasting is meant to break rock. However, not all the energy generated goes into its intended purpose, (Olofsson, 1990).

This section looks at the negative consequences blasting comes along with and further provides mitigations in reducing their effects to the surrounding.

2.2 Environmental impacts of blasting

Blasting is the science and art of breaking rock through the interaction of waves produced from detonation of explosives and the seismic waves of rock. Every time a blast is detonated, high pressure shock waves are released into the surrounding rock and it is from this interaction that breakage takes place. However, if these shock waves are not well managed, unwanted consequences resulting from seismic effects during the process are created. (Akande et al, 2014). The most likely consequences produced during blasting operations include the following:

Fly rock; Other than the forward movement of the entire rock in bench blasting, unwanted rock ejection from the blast occurs. This is known as fly rock. Fly rocks tend to travel long distances and are the main causes of on-site fatalities and damages to equipment.

Noise; this is sound generated from the blast that causes public nuisance when above 134 dB. This is usually due to gas release pulse from under burdened holes

Dust and fumes; this is the material that is generated during blasting operations and is likely to flow in the direction of wind. Explosives that produce fumes in most cases are such that have imbalance on the combination of ammonium nitrate and fuel oil with other additives. These may be in the form of ammonium nitrate fuel oil (ANFO) prills or emulsions and water gel containing ammonium nitrate or combinations of these.

Vibrations; this is the seismic movement in the ground caused by blasting. Ground vibrations are a form of energy transportation through the rock. If not well managed and kept below the threshold, they cause damage to adjacent structures. In Zambia, no threshold values have yet been set hence the use of the South African bureau of standards which has set the rate at 10 mm/s.

2.2.1 Flyrock

Upon firing an explosive charge in a blast hole, a compression wave travels to the free face. Rocks such as sandstone, shale, conglomerate and dolomite are generally strong in compression and little damage is inflicted by the compression wave. However, upon arrival at the free face, the compression wave is reflected back as a tensile wave. The tensile wave initiates cracks and fractures because most of the rocks are weak in tension. The gases then enter into the tensile-fractured spaces and resume their expansion work causing propagation of cracks. Gas pressurization then causes the fragmented rock mass to burst out from the bench. If there is excessive local gas pressurization due to the mismatch of the explosive energy with the geotechnical strength of the rock mass surrounding the explosive charge (Baipayee et al., 2004), flyrock could be generated. Factors responsible for this mismatch include:

1. High explosive concentration leading to localized high energy density
2. Inadequate delay between the holes in the same row, or between the rows
3. Improper loading and firing practice, including secondary blasting of boulders and toe holes.

Research conducted by Olofsson in 1990 has showed that maximum ejection of flyrock can be calculated using Equation 4 that takes into considering the blast hole diameter that is dependent on a given specific charge.

$$L_{\max} = 260\left(\frac{d}{25}\right)^{\frac{2}{3}} \qquad (4)$$

Where;

L $_{max}$ is projection distance in meters

d is hole diameter in millimeter

The following are causes and mitigation of flyrock;

a) Insufficient Burden: this is a primary cause of flyrock from a high wall face. This usually occurs whenever drilling is not correctly done on the bench resulting in irregular high wall faces that do not provide uniform burden from each point of the loaded borehole. High-pressure gases generated during blasting will vent out and therefore pose the greatest hazards at the weakest point in the high wall. Blasters need to visually examine the blast bench for zones of weakness, back break, concavity, unusual jointing and overhang in order to determine better ways of conducting safe blasting. This can only be accomplished by constant communication between the driller and themselves.

b) Blast hole layout and loading: the duty of the blaster is to ensure that the layout on the plan is strictly followed and all the holes are drilled correctly on the blast bench. This is very important because if the layout is not correctly drilled, there is an increase or decrease of the burden and spacing thereby affecting fragmentation. Furthermore, whenever hole deviation occurs, the likelihood of damaging the high wall increases because holes are likely to undercut the slopes. It is also the duty of the blaster to verify that the holes are drilled to the correct depth before loading of explosives into them. While loading a hole, blasters must frequently check the rise of the explosive column in the density cups before stemming the holes in order to prevent overloading or under loading. This is because some benches have voids, cracks, or other unknown reservoirs which tend to take in a lot of explosives that would lead to overloading thereby generate excessive release of energy that results into flyrock.

c) Geology and rock structure: Sudden change in geology or rock structure can cause a mismatch between the explosive energy and the resistance of the rock. Abrupt decrease in rock resistance is due to joint systems, bedding layers, fracture planes, geological faults, mud seams, voids and localized weakness of rock mass. It is prudent to try to detect such changes in advance and adjust the drilling patterns accordingly. Sedimentary rocks often change their geomechanical properties due to abrupt changes in the direction of laminations or bedding planes, inclusion of zones of weakness, and voids. Geological intrusions can compromise the strength of the parent rock. Presence of mud seams, voids, caverns raise a concern for the blaster. Inadvertently loading these areas will produce a high energy concentration. Change in rock formation is best mitigated by effective communication between drillers and blaster. Driller's log sheet must be completed during the drilling process and examined by the blaster prior to loading in order to locate any abnormality or irregularity. The log sheet should indicate items such as the depth and angle of each hole, location of voids, competency of rock, loss of air, and/or lack of drill cuttings (Mishra et al., 2011). In many operations, the drilling crew also participate in loading the holes for the blast and hence appreciate the importance of effective communication through logging.

d) Stemming: Stemming provides confinement and prevents the escape of high-pressure gases from the borehole. The purpose of stemming is to provide resistance to high-pressure gases whenever they want to escape from the blast hole during detonation. Konya and Walter in 1990 recommended a stemming length of about 0.7 to the burden been used. Improper or inadequate stemming can result in stemming ejections, undesired fragmentation of the collar zone and flyrock and air blasts, (Konya and Walter, 1990).

e) Detonation firing delay: Critical elements of any blast design are firing delays between adjacent holes in a row and also those between successive rows. The firing delay is a function of the burden, spacing, hole depth, rock type, and the quantity of charge fired per delay. Proper firing delay helps to achieve good fragmentation of the blasted material. It also reduces ground vibration, air blast, and flyrock. The rock fragmented by the previous hole must be given a chance to move out prior to firing subsequent holes

f) Lack of blast area security: Lack of blast area security has accounted for over 41% of blasting related accidents. (Verakis and Lobb, 2003). The blast area must be identified, cleared and entrances guarded prior to firing the blast. A blaster is often unable to see the entire blast zone area from the firing point due to the nature of the terrain or obstructions such as benches and dumping points. A last minute check of the blast area and all leading routes to the blast area must be checked and communication with guards as this could prevent such occurrences. An analysis of blasting injuries indicates that several factors have led to injuries due to lack of blast area security, (Siskind, 1995). Some of these factors include:

1. Failure to evacuate all employees and visitors from the blast area before blasting
2. Failure to understand the instructions and signals given by the blaster or supervisors
3. Inadequate guarding of the access roads leading to the blast area, or the secured area
4. Taking shelter at an unsafe location, or inside a weak structure. Blast area security issues could be addressed by providing adequate training and refresher courses (class room and hands-on type) to the blaster and other involved employees.

2.2.1.1 Flyrock prevention methods in construction

Flyrock can be prevented in control blasting techniques by covering the holes to be fired with heavy rubber mats (blasting mats). The correct specific charge will prevent the escape of fly-rock during the blasting process. The number of mats as well as their positioning is very important.

An important factor in flyrock prevention is the correct use of blasting mats. Different sizes of blasting mats are available such as 1.0 x 1.2 meters, 1.2 x 2.4 meters and 3.0 x 3.0 metres. Large blasting mats give maximum protection, but are difficult to handle and require many personnel.

In trench blasting, blasting mats are placed in such a way that the first mat is placed over the last holes to be fired, while the rest of the mats are placed towards the front row in such a way that proper overlapping occurs. Placing mats in this procedure will prevent the mats at the back of the trench to move from their position before all the holes have detonated. Blasting mats can be protected again sharp rock fragments, by placing a few pieces of thick welded mesh wire over the rock before covering it with the blasting mats.

If blasting mats are unavailable, old tyres can also be used provided they are tied together by means of rope or chain. The use of conveyer belts is not always advisable, due to the throw of these materials by the expending gases. This is especially true in cases where a hard material such as iron is blasted and the risk of blow out holes increases, (Verakis and Lobb, 2003).

The best cover available for blasting operations is ground; however, it should be free of rocks, not to be completely dry or over damp. Enough ground should be placed at the free face to prevent flyrock.

Whatever covering method is used, it is always a good habit to leave blasted material in front of the free face. This should help to keep the covering material in a horizontal position. Good covering of the free face is a necessity in order to prevent a disaster.

2.2.2 Noise

Noise is the sound generated from the blast that causes public nuisance when above 134 dB. This is usually due to gas release pulse from under burdened holes.

2.2.2.1. Air blast

Air vibrations are generated by the blast and propagated outward through the air under the influence of the existing topographic and atmospheric conditions. Four mechanisms are usually responsible for the generation of air blast vibrations: the venting of gases to the atmosphere from blown-out unconfined explosive charges, release of gases to the atmosphere from exposed detonating fuse (initiation system), ground motions resulting from the blast, and the movement of rock at the bench face. Audible air blast is called noise while air blasts at frequencies below 20Hz and inaudible to the human ear are called concussions. This is measured and reported as an “overpressure” meaning that air pressure is over and above atmospheric pressure. The noise can either be 1 second continuous or can be of impulsive nature such as a shock from explosions. Overpressure is usually expressed in pounds per square inch (psi), Pascal or Kilopascal (Pa, kPa), or in decibels (dB).
Peak pressures are reported in terms of decibels and can be calculated as shown in Equation 5.

$$dB = 20\log 10\left(\frac{P}{P_o}\right) \qquad (5)$$

Where P is the measured peak sound pressure and Po is a reference pressure of 2.9 x 10^{-9} psi (20 x 10^{-6} Pa).

Energy transmitted in acoustic waves behaves in the same manner as seismic energy. Air blast overpressures are greatly affected by atmospheric conditions, direction and strength of wind, temperature, humidity, and cloud cover. Like ground vibrations, the peak overpressure level is controlled by the charge weight of explosive per delay and the distance from the blast hole. Unlike ground motions, air pressure can be described completely with only one transducer, since at any one point; air pressure is equal in all three orthogonal directions. The pressure developed by noise and shock waves is the primary cause of window rattling. Nichols et al in 1971, through the Bureau of Mines conducted extensive research in blasting and concluded that overpressure less than 0.75 psi would not result in any window damage and overpressure of 1.5 psi or more would definitely produce window damage (Nichols, Johnson and Duvall, 1971). Table 2 illustrates overpressure equivalence for both types of units (dB and psi) based on Occupation Safety and Health Administration. In order to understand the overpressure levels, 0.01 psi is comparable to the maximum found in a boiler shop or to the pressure level present 4 ft. from a large pneumatic riveter.

Table 2: Convention for dB to psi, (Mohamed, 2010)

dB	psi	Comment
180	3	Structural damage
170	0.95	Most windows break
160	0.3	
150	0.095	Some windows break
140	0.03	OSHA maximum for impulsive sound
130	0.0095	USBM TPR 78 maximum
		USBM TPR 78 safe level
120	0.003	Threshold of pain for continous sound
110	0.00095	Complaints likely
100	0.0003	OSHA maximum for 15 minutes
90	0.000095	
80	0.00003	OSHA maximum for 8 hours

Air blasts causes noise as well as over pressures that arise due to detonation of explosives in the blast holes. Incidents in the mining industry regarding windows shattering and breaking off have been attributed to air over pressure from blasting operations. Not only does this damage equipment but has also been the cause for hearing disability if exposed for a long time. Table 3 shows 128dB as the acceptable threshold to a human ear. These values are based on the South African Bureau of Standard.

Table 3: Maximum acceptable Noise overpressure levels, (South African Bureau of Standards, 2000).

Item	Noise Overpressure level (Decibels-dB)	South African Authority Appreciation
1	100	Barely Noticeable
2	110	Readily Acceptable
3	128	Currently accepted by South African authorities as being an acceptable level for public concern
4	134	Currently accepted by South African authorities that concern will not occur below this level

In order to reduce air over pressure produced during operations, the following has to be done:

1. Ensure maximum confinement of explosives by:-
   - employing adequate stemming,
   - employing correct size of stemming materials,
   - not overburdening any of the blast holes, and
   - Employ correct tie up to avoid chocking some blast holes.
2. Reduce cooperating charge per delay by:-
   - employing optimum bench heights,
   - reducing number of blast holes detonated per delay,
   - Employing small diameter blast holes, and decking.
3. Time blasts, to tie in with highest ambient noise in the area,
4. Avoid blasting when wind is blowing in the direction of acoustic over pressures sensitive areas,
5. Employ toe priming rather than collar priming, and
6. Progression of firing detonation along the free face should be less than the speed of sound in the air.

2.2.3 Dust and Fumes

Dust is the fine, dry powder consisting of tiny particles of earth material or waste matter that is generated during blasting operations. This material tends to rise in the air and is likely to flow in the direction of wind.

Fumes are noxious gases that are produced from the detonation of explosives. The production of these gases is very critical in underground and other confined workings. Many factors affect the volume of poisonous gas produced including oxygen balance and adverse loading of explosives.

The *fume class* is a measure of the toxic gases in cubic feet per 0.44 lb. (200 g) of unreacted explosive that produce fumes are a combination of ammonium nitrate and fuel oil with other additives. These may be in the form of ammonium nitrate fuel oil (ANFO) prills or emulsions and water gel containing ammonium nitrate or combinations of these.

Fumes are easily controlled by ensuring there is a balance in the chemicals mixture of explosives. In order to reduce the effects of dust during blasting operation, the shot firer must observe the direction of wind and be in a position to determine the appropriate time of blasting so that the dust raised and fumes quickly get carried away into the air, (Barnhart, 2004).

2.2.4 Ground Vibration

Ground vibrations are seismic movements in the ground that are generated by rock blasting. These seismic movements are a form of energy transport through the ground that cause damage to adjacent structures when they go beyond a maximum threshold in relation to the type of structure. Energy during blasting operations is meant to break rock, but some of the energy propagates in all directions from the hole as seismic waves with different frequencies. It is believed that these seismic waves are damped with increasing distance and those with the highest frequency get damped much faster, (Lucca, 2003).

Blast induced ground vibrations are as a result of the detonation pressure pushing the blasted rock away from the bench in form of body waves to the wave front as shown in Figure 13.

This large force against the bedrock or unbroken portion causes the bedrock to vibrate. When the vibration is transmitted through the ground, this is called propagation. The propagation velocity is the speed at which the vibration wave travels. As the vibrations waves travel away from the energy sources, the vibration is reduced or diminished with increasing distance. This is called seismic attenuation.

Properly designed blasts utilize the majority of explosive energy to break rock. Poorly designed blasts will have higher vibration levels due to wasted energy, (Lucca, 2003).

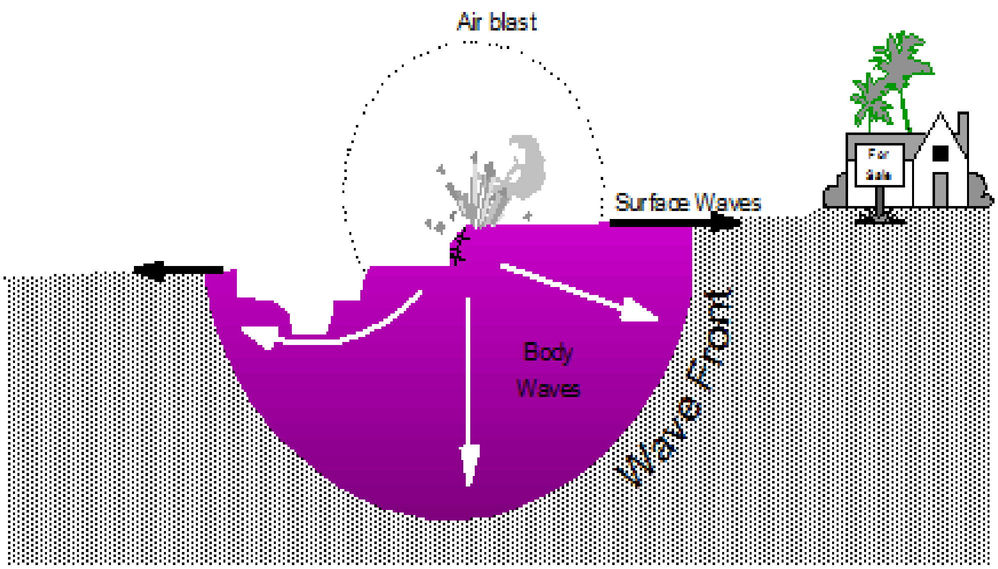


Figure 14: Blast induced vibrations (Bulk Mining Explosives handbook, 2009)

Explosive energy always seeks the path with the least resistance and hence the stress wave propagation at high pressures and temperature extends radial cracks in any discontinuity, joints and cracks surrounding the rock being blasted. Ground vibrations are as a result of any such explosive energy that is not utilized in breaking rock and is commonly known as wasted energy.

Mohamed (2010) illustrated through Figure 14 that whenever an explosive is detonated in a blast hole, rock in the immediate vicinity is crushed and shattered and an oscillatory wave is propagated through the rock mass causing particles along its path to move backwards and forward in longitudinal direction along the lines of advance of this primary wave.

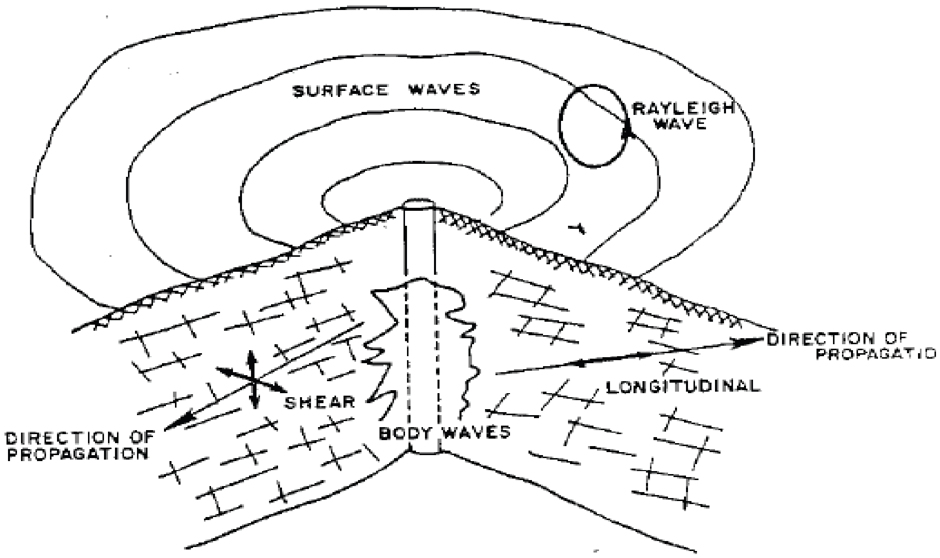


Figure 15: Common types of elastic waves induced by blasting, (Mohamed, 2010).

Blast induced vibrations waves are transmitted through the ground and this is called propagation. This process occurs in wave forms that can be divided into three main components namely compressive, shear and surface waves. In order to measure the ground motion, the three perpendicular components of vibrations namely, transverse, vertical and longitudinal waves must be measured.

The three main vibration wave types could further be defined in terms of body waves and surface waves. It is very important to be able to measure all three of them because it's from them that we determine the resultant vibration known as Peak Particle Velocity PPV. Each component can be represented as a sine wave.

a) **Body waves** are waves that travel in the body of rock. There are mainly two types of body waves and these include the P wave and S waves. These are the type of waves that are a push and pull arrangement. They move fastest in materials such as solids, liquids and gases hence are the first-arriving energy on a seismogram. They are generally smaller and higher frequency than the S and Surface-waves. P waves in Figure 16 are a push and pull waves that are compression / dilatation in nature and these are directed towards the wave propagation. The compression and tension effect of the longitudinal component is responsible for the motion of these particles. Whenever these waves travel in the solid, liquids and gas media, the compression wave creates change in volume of the medium. An example of such waves is that of a string held between two hands that move up and down when stretched. The P waves in a liquid or gas are pressure waves, including sound waves.

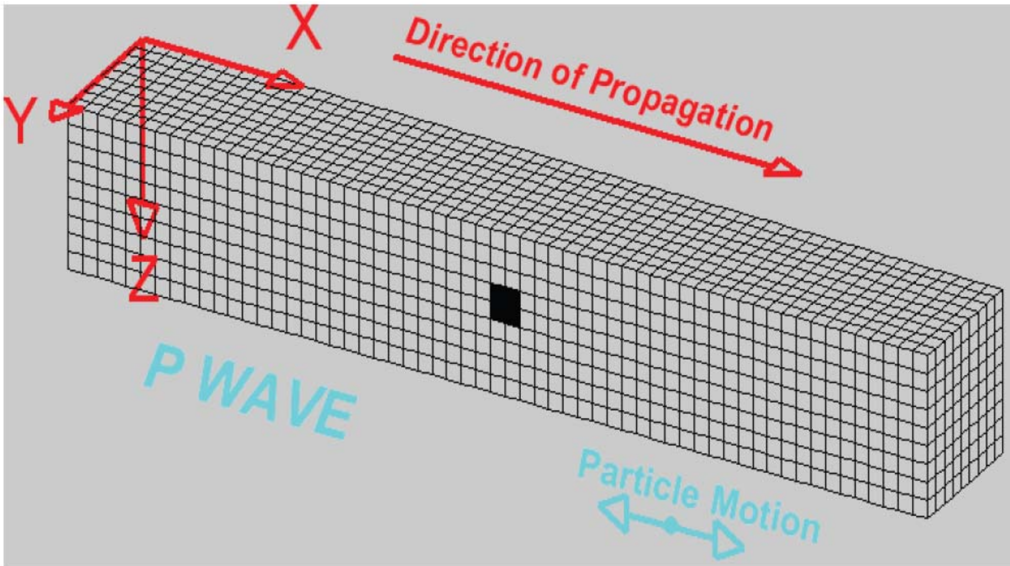


Figure 16: Compressive wave direction (Mohamed, 2010)

The other type of body wave is the S wave. As can be seen in Figure 17, the S wave, also known as the transverse wave moves at right angles to the direction of wave travel. These are only able to travel in a solid medium and they are responsible for change in shape of the medium.

Generally, body waves are produced predominantly at small distances during blasting operations. These waves propagate outward somewhat spherically until they contact any boundary. These boundaries can include another layer of rock, free face, fracture, joint surface and / or soil. Whenever body waves arrive at such intersections, surface and shear waves are produced.

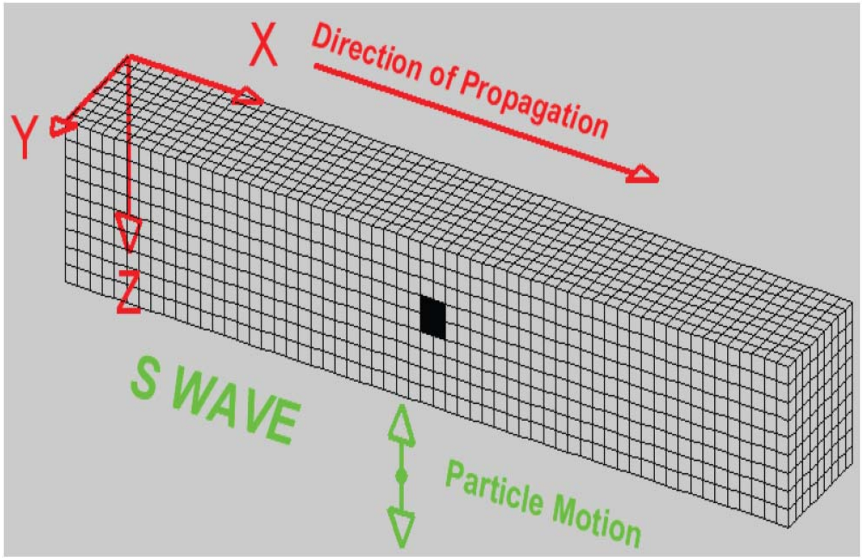


Figure 17: Shear wave direction (Mohamed, 2010)

b) **Surface waves** are also known as vertical waves which. They are larger than body waves hence they travel much slower. These waves are the major cause of complaint and vibration problems to the neighborhood because they carry a lot of energy with them and hence produce a lot of motion. There are two basic types of surface waves; these are Rayleigh and Love waves, (Mohamed, 2010). Figures 18 shows the Rayleigh wave moving in complicated motions in line with the motion particles that move both in the vertical and parallel directions to the direction of propagation (ellipse motion).

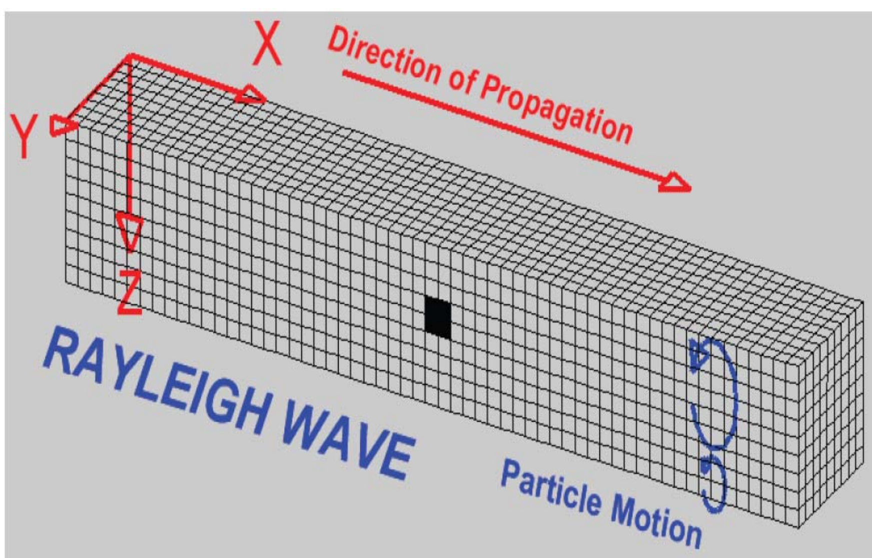


Figure 18: Rayleigh waves (Mohamed, 2010)

Love waves as shown in Figure 19 are transverse waves that travel in a surface layer on top of another medium overlying rock, (Lucca, 2003). Whenever a blast is conducted in extreme near field, surface waves are almost nonexistent without a soil layer.

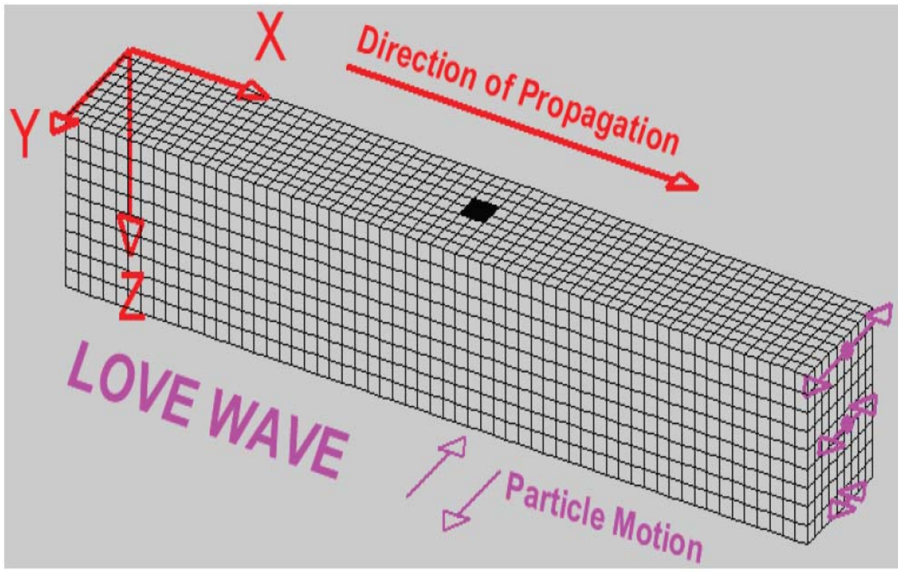


Figure 19: Surface wave motion (Mohamed, 2010)

The material involved in the transmission of surface waves is a zone about one wave length in thickness. All surface waves are generated at approximately the same time and, in the immediate vicinity of the blast, the total surface, displacement is controlled by the total energy contained within the waves. However, as the waves travel outwards at differing velocities, they quickly separate and maximum ground motion is then controlled by the energy contained within each individual wave. Hence maximum displacement decreases very rapidly at first but then diminishes more slowly as individual waves die out from loss of energy and dispersion, (Sang, 2003).

The Rayleigh or R-wave is the longitudinal wave that can cause vertical retrograde motion. The most common observation of the surface waves is that they carry a large amount of surface and ground energy, therefore, likely to cause damage to nearby structures.

The love or Q-wave (from the German querwellen) causes transverse vibration in the horizontal plane only and never in the vertical direction. As illustrated in Figure 20, the displacement of particles by C-waves is elliptical and inclined. They have both vertical and horizontal in directions hence are a combination of the P wave and S wave. The H-wave moves particles in an elliptical orbit similar to the R-wave but in the reverse direction. It has only been detected in nuclear blasting. P-waves have the highest velocity, usually in the order of 3000m/s - 6000m/s in hard rock formations, (Mohamed, 2010).

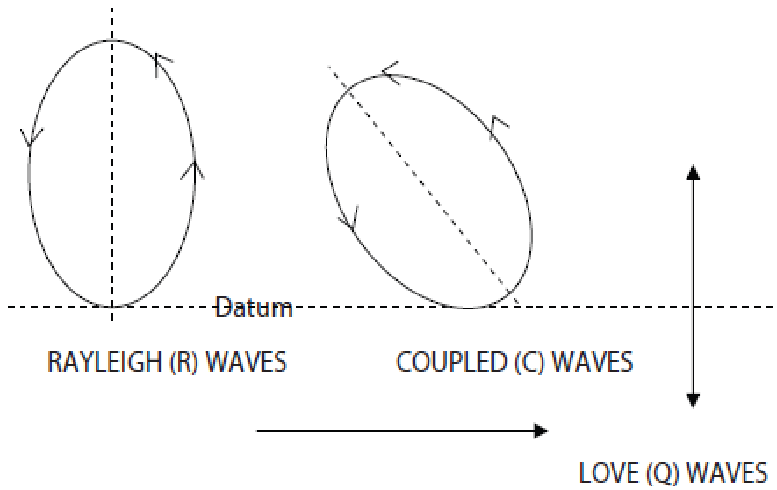


Figure 20: Particle motion associated with Rayleigh, Coupled, Love surface waves (Mohamed , 2010)

As waves travel outwards from the source of the explosion, higher frequencies are damped out and the frequency range thereafter is of the order of 3Hz-70Hz. This is because they are dependent on site geology and distance from the sources of detonation. Explosions in the ground are generated under a wide variety of conditions and this may lead to the formation of one type only, all types or combinations of surface waves, (Mohamed, 2010).

In dealing with environmental problems, it is the net resultant motion to which a structure or person is subjected to and hence it is usual to measure the net effect of all surface and body waves and not to attempt differentiation between the motion attributable to each type.

2.2.4.1 Measuring blast induced vibration

Blast induced vibrations can be measured using four main parameters that are interrelated and are used in order to define the magnitude at any location. These include:

Particle Displacement - the distance that a particle moves before returning to its original position, measured in millimeters (mm).

Particle Velocity - the rate at which particle displacement changes, it is measured in millimeters per second (mms^{-1}).

Particle Acceleration - the rate at which the particle velocity changes, measured in millimeters per second squared (mms^{-2}) or in terms of the acceleration due to the earth's gravity (g).

Frequency - the number of oscillations per second that a particle moves. This occurs in the back and forth direction and is measured in Hertz (Hz).

In blasting, ground particles oscillating in response to a vibration wave are measured in particle velocity. The maximum rate is the peak particle velocity (PPV) and is measured in millimetres per second or inches per second.

a) Peak Particle Velocity

This is the maximum rate of particle movement; displacement is the distance the particle moves back and forth or the distance a particle or object moves from its position of rest. The change in displacement over a unit of distance is called strain, (Lucca, 2003).

Other than peak particle velocity, frequency is another most important factor of controlling the response of structures.

Frequency is the number of times the particles move back and forth in one second. This back and forth movement can also be referred to as oscillations. The number of oscillations per second or cycle per second that a particle makes under influence from the vibration wave is measured in Hertz (Hz)

Frequency is dependent on site geology, distance from the blast and delay sequence. The nature of effect of frequency allows control blasting without damage and also allows higher peak particle velocity. Blast frequencies can be many thousands of hertz. Although they diminish quickly, frequency can still be high within a short distance.

Structural dynamic research has showed the importance of vibration wave frequency in control blasting methods. Frequency, although looked at in terms of standard blasting limits, is hardly ever taken into consideration when designing control blasts, (Lucca, 2003).

During blasting operations, the shock wave propagates through the rock material at a frequency spectrum from as low as 0.5Hz up to 200Hz. The **dominant frequency** is the frequency with the largest amplitude over the whole frequency spectrum. The frequency spectrum is influenced by the distance and type of rock material. High frequencies are filtered out faster in porous materials over shorter distances, while allowed to propagate in hard dense materials. This means that the lower frequencies are restricted to porous materials, while the higher frequencies dominate in high-density homogenous rock materials, (Stagg and Eagler, 1980).

If the frequency of the incoming waves falls in phase with **natural frequency** of the structure, this results in a resonance effect in the building. The possibility exists that the amplitude might exceed the maximum particle velocity. This can result in permanent damage to structures. Structures respond differently when a wave with a low frequency of 10Hz at a specific particle velocity moves through the ground compared to a wave with a higher frequency of 80Hz with the same particle velocity. This is because higher frequencies tend to produce less damage than lower frequencies. Buildings are vulnerable to frequencies of less than 10Hz, (the natural frequency is the frequency at which the building will start to vibrate resulting in permanent damage).

b) Amplitude

The Amplitude or peak particle displacement (mm) refers to the intensity or energy content of the wave. Parameters such as distance and explosive charge will affect the amplitude. The unit for amplitude is millimetres (mm). Amplitude limits may fall between fractions of a millimetre up to a few millimetres.

c) Wave Velocity (Sonic)

Mohamed (2010) describes sonic velocity of rock as one of the most important parameters to characterise rock material. This material could better be described in terms of the number of cracks present and faults. These factors are well known to affect sonic velocity to a large extent. Competent high density rock formations have high sonic velocities. However, it is believed that they decrease when faults and fissures increase.

Propagation velocities for the longitudinal, transversal and vertical components differ. Shear waves (transversal component) travel at the speed of about half to two thirds of the velocity of the compressive waves (longitudinal component) while the surface wave also known as vertical component travel 0.9 times the velocity of the shear wave. In most cases, surface wave would propagate even slower if there were a layer of sand on top of the rock material. The wavelength is a function of both the frequency and sonic velocity of rock materials. Both frequency and propagation velocity diminishes over time and distance resulting in longer wavelengths. It is important to remember that low frequencies with high amplitudes are responsible for damage to most structures, (Mohamed, 2010).

In order to measure ground vibrations, one or several points are selected on the ground away from the sources of vibration. It is recommended that a total analysis of the three waves is done. Normally the vertical component is dominant at shorter distances and it is therefore sufficient to measure vibrations in the vertical direction. For vibration analysis of the measured values, the vibration phenomenon is recorded as the function of time. Then the displacement, particle velocity and acceleration can be recorded.

Table 4 shows the seismic velocity of different rock formations, of interest is the shale up to quartzite which is present at Area J open pit. During blasting operations, the stress waves produced by explosives have to supersede that of the rock in order to break rock. The excess waves that are not absorbed in fragmenting rock turns into elastic waves and that is what causes ground vibration.

Table 4: different rock characteristics and their respective velocities (Mohamed, 2010).

Rock Characteristics	Velocity (m/s)
Soil	1500
Coal	3000
Shale	3500
Sandstone	4000
Quartzite	5700
Granite	5700
Dolerite	6000

Note that ground vibrations are measured on structures using instruments known as geophone and the acceleration on installations with an accelerometer. By measuring the ground vibration, the acceleration can be calculated and with these parameters, the damage criterion for the structures can be derived thereby protecting the structures. In this case, it becomes important then, to measure peak particle velocity.

2.2.4.2 Allowable Particle Velocities for various materials

Table 5 shows some of the allowable particle velocities with respect to foundations for residential properties.

Table 5: Allowable PPV for various materials (Mohamed, 2010).

Material type	Sand, gravel and clay	Sandstone (soft) and slate	Sandstone (hard) and granite	Visible effects
Sonic Velocity (mm/s)	1000-1500	2000-3000	4500-6000	
PPV (mm/s)	18	35	70	None
	30	55	80	Fine cracks
	40	80	150	Cracking
	80	115	225	Serious cracks

Table 5 only serves as a guideline. Different countries have different regulations pertaining to ground vibrations, it is better to always keep the levels of vibration below the threshold of acceptable limits.

One of the most troublesome and controversial issues facing mining and other industries related to blasting is that of ground vibration and air blast produced form blasts. (Mohamed, 2010). This has recently being the case for any green field project that shall require blasting as one of the major processes in the operations. All Environmental Impact Statement disclosure meetings have recorded questions and concerns regarding blasting and vibrations. It is imperative that such laws are passed in Zambia in order to govern the thresholds. The mining industry cannot do without blasting and even with the recent new explorations projects that have showed the availability of mineral ores close to the community. For that, accurate control must to be serious restricted to minimize blasting effect on people and environment. When a blast is detonated, some of the explosive energy not utilized in breaking rock travels through the ground and air media in all directions causing air and ground vibrations. Air and ground vibration from blasting is an undesirable side effect of the use of explosives for excavation. The effects of air and ground vibrations associated with blasting have been studied extensively by a number of blasting experts such as Dowding (1992) who have paid particular attention on criteria to control the vibration and prevent damage to structures and people.

There are a number of factors surrounding blasting activates which influence the outcome from blasting and the complex vibration waveform. Many parameters controlled and uncontrolled, influence the amplitude of ground vibrations such as: distance away from the blast; rock properties; local geology; surface topography; explosive quantity and properties; geometrical blast design; operational parameters (initiation point and sequence, delay intervals patterns, and firing method).

The propagation of ground vibration waves through the earth's crust is a complex phenomenon. It is very evident that rock formations change rapidly even in short distances hence making this blasting study a complicated one. However, with the understanding of blast frequencies generated and recorded from the machines have indirectly made the study on blasting effect bearable. Rock is generally unconsolidated material and these fall into anisotropy and non-homogeneous, this makes the study of geology important while evaluating blast induced vibrations. Rock interface on surface and at the body make a complex boundary effect. These difficulties restrict theoretical analysis and derivation of a propagation law, and consequently research workers have concentrated upon empirical relationships based on field measurements.

In order to control and protect the structures from devastating effects of ground vibrations, regulations have been formulated in different countries. These regulations vary from one country to another depending on the type of construction material used. Many damage criteria and propagation equations have been established and fulfilled with varying degrees of success.

2.2.4.3 Damage criteria and regulations

There are many damage criteria that have been established and fulfilled with different degrees of success (Mohamed, 2010). Its development stretches from Rockwell's vibration energy formula in 1934 to the present day Office of Surface Mines (OSM) regulations. A short account on the review of blast induced vibrations and their regulations from different countries in the world are presented below:

In 1934, Rockwell stated that vibration energy caused by blasting was proportional to frequency (f) and amplitude (A) (is proportional to f^2A^2), (Mohamed, 2010).

The United States Bureau of Mines (USBM) conducted studies from 1935 to 1940 on frequency and results revealed that a frequency range from 4Hz to 40Hz and amplitude from 0.0025mm to 12 mm are closely linked to damage to acceleration and have been fulfilled. These studies showed that , no damage with acceleration of lower than 0.1g, minor damage (fine plaster cracks) with acceleration ranges from 0.1 to 1.0g, but major damage (fall of plaster) when acceleration is above 1.0g. In 1942, USBM combined the effect of charge quantity, ground character and distance. This formula was found to be inadequate in view of the more complex blasting designs.

In 1949 Crandell developed the concept in which the energy ratio was defined as the ratio of the square of the acceleration to the square of the frequency ($ER = a^2/f^2$). Crandell's damage criteria was based on pre- and post-blast investigations of over 1000 residential structures, He recommended that the threshold level at which minor damage occurs is about 3 while that above 6 is more dangerous.

In 1958, Langefors described the relationship between blast induced ground vibrations and structural damage during a reconstruction project in Stockholm, (Laangefors et. al., 1958). Frequencies were measured between ranges from 50 to 500 Hz and amplitudes of 0.02mm to 0.5 mm. From their research, it was concluded that peak particle velocity gave the best guide to damage potential and derived the results as shown in Table 6.

Table 6: Damage criteria (Langefors et. Al., 1958)

Particle Vibration	Damage
2.8 in/s	No noticeable damage
4.3in/s	Fine cracks and fall of plaster
6.3in/s	Cracking of plaster and masonry walls
9.1in/s	Serious cracking

In 1959, Edwards and Northwood conducted an investigation in which they came up with a frequency range of 3Hz to 30Hz and amplitude range 0.25mm to 9mm. They concluded that damage was more closely related to velocity than displacement or acceleration. Minor damage was likely to occur with a peak particle velocity of 100mm/s (4in/s) to 125mm/s (5in/s). Table 7 presents damage risk greater than 4in/s to be the damage level applied in St. Lawrence project in Canada.

Table 7: Damage criteria for St Lawrence project in Canada (Edwards, 1959)

Particle Vibration	Damage
<2in/s	Safe no damage
2.4in/s	Caution
>4in/s	Damage

In 1971, USBM set damage criteria of peak particle velocity of less than 2in/s or 50mm/s and concluded that this would result in a low probability of structural damage to residential dwellings, see Table 8.

Table 8: Damage criteria (Nichols et al., 1971)

Particle Vibration	Damage
<2in/s	Safe no damage
2in/s to 4in/s	Plaster cracking
4in/s to 7in/s	Minor damage
> 7in/s	Major damage to structure

Medearis, in 1976 reported that a peak ground particle velocity alone could not cause damage to existing structures alone without taking into account two other very significant parameters, namely the predominant frequencies of the ground motion and the existing structure. He concluded that Pseudo Spectral Response Velocity (PSRV) was deemed to be the best predictor of damage due to blast vibrations. For a predicted PSRV of 1.5in/s, the probability of damage ranged from 0 to 1%, (Medearis, 1976).

Bauer, 1977 established damage criteria for equipment and structures that depended on peak particle velocity as shown in Table 9.

Table 9: Structural damage criteria (Bauer et al., 1977)

Type of Structure	**Type of damage**	**Particle velocity at which damage starts**
Rigidity mounted mercury switches	Trip out	0.5 in/s
Houses	Plaster cracking	2 in/s
Concrete blocks in a new home	Cracks in the block	8 in/s
Cased drill holes	Horizontal offset	15 in/s
Mechanical pumps compressors	Shafts misaligned	40 in/s
Prefabricated metal building on concrete pads	Cracked pads building twisted and disorted	60 in/s

In 1980, Siskind et al. conducted a comprehensive blast study close to a total of 76 houses and he later published the results of ground vibration produced by blasting from 219 production blasts in RI 8507, (Siskind et al., 1980).

It was then concluded that peak particle velocity is still the best single ground descriptor. Furthermore, it was deemed as the most practical safe criteria for blasts that generate low frequency ground vibrations. For frequencies above 40 Hz, a safe peak particle velocity of 2 in/sec is recommended for all houses.

In 1983, the United States Office of Surface Mine (OSM) published its final use of explosives regulations in order to control ground vibrations. These regulations were specifically designed for application in surface coal mining operations only. However, many ore mining firms resolved to use these guidelines in their operations. Consequently, the office of OSM regulations was tasked to offer more flexibility in meeting performance standards and to prevent property damage. The following options were presented by the OSM regulations.

• *Method 1*- Limiting particle velocity criterion: this requires each blast to be monitored by a seismograph capable of monitoring peak particle velocity. This is permitted provided all the maximum peak particle velocity stays below the acceptable levels.
• *Method 2*- Scaled distance equation criterion: this requires the operator to design shots in accordance with Table 10, which specifies a scaled distance design factor for use at various distances between a dwelling and blast site. No seismic recording is necessary. Provided that scaled distance in Table 10 is observed.

Table 10: Maximum permitted particle velocities, (Mohamed, 2010).

Distance from blast site (ft)	Method 1 maximum allowed peak partcile velocity (in/s)	Method 2 Scaled distance factor used without seismic monitoring
0 - 300	1.25	50
301 - 5000	1	55
5001 and above	0.75	65

• *Method 3*- Blast level chart criterion: This method allows an operator to use particle velocity limits that vary with frequency. This method requires frequency analysis of the blast-generated ground vibration wave as well as particle velocity measurements for each blast. This method may represent the best means of evaluating potential damage to residential structures as well as human annoyance from blasting.

Any seismic recordings for any component (longitudinal, transverse, or vertical) for the particle velocity at a particular predominant frequency that fall below any part of the solid line are considered safe. And any values that fall above any part of the solid line will increase the likelihood of residential damage and human annoyance.

The Australian "SSA explosive code AS 2187" has been presented in Table 11. These were recommended to be the maximum limits for the level of ground vibration in soil near the foundation buildings.

Table 11: Peak Particle Velocity Criteria from Australia AS2187, (Mohamed, 2010).

Particle Velocity (mm/s)	Type of building or structure
25	Commercial and industrial building or structure of reinforced concrete or steel construction
10	Houses and low rise residential buildings, commercial building not included in the third category
2	Historic building or monuments and buildings of significance

In Germany, the following vibration limits were set as a guideline to avoid damage to structures, see Table 11. Note that frequencies play an important role and hence have been included in Table 12;

Table 12: Peak Particle Velocity in Germany Din 4150 Standard, (Mohamed, 2010).

Building Class	Maximum resultant particle velocity Vr (mm/s)	Estimated maximum vertical particle volocity Vz (mm/s)
other similary built in the conventional and normal condition	8	4.8-8
Stall building in normal condition	30	18-30
Other building and historical monu	4	2.4-44

Table 13: PPV for Germany Standard, (Mohamed, 2010).

Structural type	Peak particle velocity (mm/s)		
	<10Hz	10 - 50Hz	40 - 50Hz
Commercial	20	20 to 40	40 to 50
Residential	5	5 to 15	15 to 20
Sensitive	3	3 to 8	8 to 10

Table 14: PPV for French Standard (Attewell et. al., 1984)

Structural type	Peak particle velocity (mm/s)		
	4 - 8Hz	8 - 30Hz	30 - 100Hz
Resistant	8	12	15
Sensitive	6	9	12
Very sensitive	4	6	9

India also came up with their own guiding regulations and threshold to avoid damage to nearby structures. See Table 15.

Table 15: Indian Standard (Basu et. al., 2005)

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

With much advancement in technology and control blasting techniques, Sweden as shown in Table 16 established the following guidelines to avoid structural damage that sit on different geological formations.

Table 16: PPV for Swedish Standard, (Mohamed, 2010).

Vibration (mm/s)	Subsoil
18	Unconsolidated strata of moraine sand, gravel, clay
35	Unconsolidated strata of moraine slate, soft limestone
70	Granite, gneiss, hard limestone, quartzitic sandstone, diabase

The Romans have also got a guide on blast generated vibrations and these include the following limits as shown in Table 17:

Table 17: PPV for the Roman Standard, (Mohamed, 2010).

Seismic intensity categories	Effects induced on the structures	Particle velocity mm/s	
		Allowed	Limited
IV	Possible damages for village type buildings, under pressure pipes, gas and petrol wells, mine shaft, and very fragile structures	5	10
V	The painting is falling down, small and thin cracks appear in mortar plaster in rural and urban buildings. Possible minor damage for industril construction	11	20
VI	Cracks in mortar (plaster) on the walls and pieces of mortar start to fall down in rural and urban buildings. Also minor damage for industrial construction	21	40
VII	Significant fractures are occurring in the basic elements of the rural buildings, great pieces of mortar are falling down in urban buildings and cracks are appearing in industrial constructions. Possible damage for pipes jointing system and fixed mounted equipment	41	80
VIII	Major fractures occur in the resistance elements of rural and urban buildings. Cracks are produced in the resistance elements of inductrial constructions	81	160
XI	Crumbing (collapse, falling down) of some joint elements of rural and urban buildings can occur. Fractures can take place in industrial structures. Dams and underground pipes can be damaged	161	320
XI	Rural buildings are destroyed, urban constructions are seriously damaged and industrial structures are affected seriously by fracturing and dislocation of resistance elements	321	642

As rock formation vary from one country to another, threshold limits must also vary as this is dependent on the nature of the structures, rock formations and geology, as well as the characteristics of vibration.

South African Bureau of Standard in 1990 also released a PPV damage criterion and it is shown in Table 18.

Table 18: PPV based on South African Bureau of Standards (South African Bureau Standard, 1990)

Status of Structure	Maximum PPV(mm/s)
Heavily reinforced concrete structure	120
Property owned by the concren performing blasting operations, where min or plaster cracks are acceptable	84
Strongly Masonary walls not affected by public concern	50
Commercial property in reasonable repair, where public concern is not an important consideration	25
Private property, where public concern is an important consideration and blasting conducted on a regular and frequent manner	10

The above standard in Table 18 was approved by the National Committee SABS, Construction standards, in accordance with procedures of the SABS Standards Division.

Both regulatory bodies in Zambia, ZEMA and Mines Safety Department (MSD) through the Environmental Management Act of 2011and Mining and Mineral Act of 2008 do not have any clause regarding blast induced vibration currently. Therefore, consultants concerned with blast vibrations in Zambia have borrowed the South African standard as the current threshold values. For the sake of this dissertation, SABS is also referred to as a guide where vibrations are concerned.

2.2.4.4 Effects of vibrations to structures

Vibration induced damage is quiet difficult to distinguish from that which is caused by environmental forces. Some of the forces that can also cause damage to structures include Temperature change, Humidity changes and Settling of rock material beneath the foundations due to subsidence.

The following as described by Mohamed in 2010 are some of the categories in which vibration damage criteria can be classified;

1. Cosmetic or threshold damage - the formation of hairline cracks or the growth of existing cracks in plaster, drywall surfaces or mortar joints. These may not be seen by the naked eye.

2. Minor damage – these are crack formations that are visible to the naked eye but do not affect the structural integrity of the structure.
3. Major or structural damage - damage to structural elements of a building that are given as guide values with respect to all 3 of these damage classifications for residential structures in terms of peak particle velocity and frequency. These values are based on the lowest vibration levels above which damage has been credibly demonstrated. In terms of cosmetic damage, at a frequency of 4 Hz the guide value is 15mm/s peak particle velocity, increasing to 20mm/s at 15 Hz and 50mm/s at 40 Hz and above.

Minor damage occurs when vibration magnitudes are greater than twice those given for the onset cosmetic damage with major damage to a building structure possible at values greater than four times the cosmetic damage values. These values apply even when a structure undergoes repeatedly vibration events. Even though this damage is a major concern for neighbors of surface mineral workings, the reality is that vibration levels at adjacent residential properties rarely, if ever, even approach the levels necessary for even the most cosmetic of plaster cracking, (Lucca, 2003). Engineered structures such as industrial and heavy commercial buildings and underground constructions are able to sustain higher levels of vibration than those applicable to residential type properties by virtue of their more robust design, (Mohamed, 2010).

2.2.4.5 Human response to air / ground vibration

Human beings have different responses to blast induced ground vibration and this makes the phenomenon complex. It is well known that the human body is very sensitive to the onset of vibration. Even though sensitivity to vibration varies significantly between individuals, a person would generally become aware of blast induced vibration at levels of 1.5mms^{-1} PPV, (Mohamed, 2009). As soon as the vibration released and received is greater than an individual's perception, the threshold becomes a possible concern for conducting blasting activities close to them. Such a concern normally relates to the vibration's potential for causing damage to the complainant's property.

The degree of concern may lead to complaints which arise from factors such as vibration magnitude, duration and frequency of the blast.

However, the vibration magnitude at which complaints arise varies greatly from one site to another and hence there is no common complaint threshold that exists.

Different individuals perceive ground vibrations differently and their perception is dependent on factors such as health, age and environmental exposure. It is usually the case that adverse comments are less likely once a neighbor has become accustomed to the perceived effects of blasting. An explanation of the need to blast and the significance of the vibration levels being received by a site's neighbors are very important as this is an understanding and sympathetic attitude from the blaster.

Human are quite sensitive to motion and noise that accompany blast-induced ground and airborne disturbances. Complaints resulting from blast vibration and air overpressure, to a large extent, are mainly due to the annoyance effect, fear of damage, and the after effect rather than damage. The human body is very sensitive to low vibration and air blast level, but unfortunately it is not a reliable damage indicator.

With air overpressure, blast levels of over 134 dB are likely to produce some annoyance to human beings. In most cases, there is need to assure the neighbors and alerting them on the likely time of the blast so that they are not caught unawares should the noise levels be louder than they are expected. A good public relations program with the residential owners surrounding the operations should be put in place as this is one way to alleviate the problem, assuming no structural damage. In this regard, psychological perception of the blast is generally more important than the numerical values of the ground vibration and air overpressure.

2.2.4.6 Analysis by scaled distance

The analysis of ground vibrations in terms of the peak particle velocity component and air blast intensities is evaluated based on scaled distance (SD). Scaled distance is a scaling factor that relates similar blast effects from various charge weights of the same explosive at various distances and hence has been incorporated by many regulations and studies in predicting vibrations.

Scaled distance is calculated by dividing the distance to the structure of concern by a fractional power of the weight of explosives material. These two include the square root scaling and cube root scaling.

Square root scaling is the general formula used in most regulations and general blasting, situations where the charge is considered linear. Cube root scaling is used for blasting in the extreme near field where the charge can be considered as the point charge or in explosions involving large quantities such as those created by nuclear explosions, (Lucca, 2003).

a) Square Root Scaling

This scaled distance calculation is required for designing construction blast. This is very useful as an initial estimate for vibration control and provides a conservative and safe charge weight for the test program. Since explosives confinement is not taken into consideration when calculating square root scaling, there can be a large variation in results especially in control blasting situations. It should be noted that small charges generate vibrations with higher frequencies and smaller displacement. Square Root Scaled Distance (SRSD) formula is calculated using Equation 6:

$$SRSD = \frac{R}{\sqrt{W}} \tag{6}$$

Where

R is the shot-to-seismograph distance (m)

W is the maximum charge weight (kg)

b) Cube Root Scaling

Cube root scaling must be used for vibrations predicting in the extreme near field that are less than 6m in control blasting. Cube root can also be used as the basis for predicting of frequency. The formula for cube root scaled distance (CRSD) is calculated using Equation 7:

$$CRSD = \frac{R}{\sqrt[3]{W}} \qquad (7)$$

Where

R is the shot-to-seismograph distance (m)

W is the maximum charge weight (kg)

Scaled distance is a means of incorporating the two most important factors contributing to the intensity of ground motion and air blast as intensity decreases proportionally with distance and inversely with the explosive weight detonated on one time delay. In the case of ground motion, the SRSD is used (commonly referred to as simply SD) as ground motion has been shown to correlate with the square root of the charge weight. In the air blast case. In airblast cases, air pressures correlate best with the cube-root of the charge weight (CRSD).

2.2.4.7 Blasting Sequencing

Blasting regulations usually state that only a certain amount of explosives may be detonated per delay. Through a lot of blast trials and experience in the field, it has come to be known that 8millisecond delay is suitable for conducting a good blast. It therefore means that, at every given charge per delay, a certain amount of explosive will be detonated not overlap another charge within 8 milliseconds, (Mohamed, 2010).

It was therefore recommended that 8ms could be used as the maximum time required for separation of explosives charges. Separation of explosives charges is extremely dependent on geological factors and with the coming in of electronic detonators, there is need to analyses results case by case.

2.2.4.8 Peak Particle Velocity (PPV) predictions

It is quiet challenging to predict vibrations as this has proved to be a complex phenomenon. These challenges have restricted theoretical analysis and derivation of a propagation law that predict the ground vibration, and consequently research workers have concentrated upon empirical relationships based on field measurements. Many researchers world over have taken time to study ground vibrations resulting from blasting. These research work dates way back as early as 1949, section 2.2.4.7.1 avails such work that was developed to explain the empirical data on blast vibration, (Mohamed, 2010).

a) Traditional empirical equations to predict vibrations

Predicting vibrations began long time ago with traditional empirical methods. This section looks into the usage of such equations that dates way back from 1949 by Crandell, (Crandell, 1949).

Crandell developed the concept of energy ratio in 1949. The concept is illustrated using Equation 8 and where acceleration is directly proportional to Energy ration while inversely proportional to frequency.

$$ER = \frac{a^2}{f^2} \tag{8}$$

He further suggested the following propagation as shown in Equation 9:

$$ER = kQ^2\left(\frac{50}{D}\right)^2 \tag{9}$$

Where:

ER = energy ratio

a = acceleration, ft/s^2

f = frequency, Hz;

k = site constant;

Q = quantity of explosives, Ib;

D = distance from measuring point to blast point, ft.

Studies conducted by Morris in 1950 showed that wave propagation phenomena as can be seen in Equation 10, Amplitude A of particle displacement is directly proportional to the square root of the weight of the charge (Q) and inversely proportional to the distance from the blast, (Morris, 1950).

$$A = k\left(\frac{Q^{\frac{1}{2}}}{D}\right) \qquad (10)$$

Where

k = the site constant (Varying from 0.05 for hard competent rock to 0.30 for clay up to 0.44, 0.5 for completely unconsolidated material)

A = maximum amplitude, in;

Q = quantity of explosives, Ib,

Habberjam and Whetton in 1952 suggested a higher power for the charge weight in Equation 11.

$$A \ \alpha \ Q^{0.085} \qquad (11)$$

Langefors and Kihlstrom in 1963 suggested the following relationship shown in Equation 12, (Langefors and Kihlstrom, 1963):

$$V = k\left(\frac{\sqrt{Q}}{D^{\frac{1}{2}}}\right) * B \qquad (12)$$

Where

k = site constants;

B = site constant;

V = mm/s or in/s

Q = Weight of explosives, kg or Ib;

D = distance from point of blast to measuring point, m or ft.

Duvall, et al. (1963) conducted a research that brought about the concept of linear dimension and they concluded that linear charge must be directly proportional to velocity assuming cylindrical explosive geometry for long cylindrical charge. The corresponding relationship assumes the form:

$$V = k\left(\frac{Q}{D^{\frac{1}{2}}}\right)^{-B} \qquad (13)$$

However, the U.S. Bureau of Mines investigated a similar scenario with Equation 13 and they suggested that for a spherical symmetry, any linear dimension should be scaled to the cube root of the charge size and this concept supported the theory of Ambraseys and Hendron in India, (Ambraseys and Hendron, 1968). An inverse power law was suggested to relate amplitude of seismic waves and scaled distance to obtain the following relationship:

$$V = k\left(\frac{Q}{D^{\frac{1}{3}}}\right)^{-B} \qquad (14)$$

In 1984, Attewell et al. proposed the following shape of propagation equation:

$$V = k\left(\frac{Q}{D^{2}}\right)^{n} \qquad (15)$$

Where

n = is the constant depending on site conditions (0.64 to 0.96)

K = site constant, (ranging 0.013 to 0.148, the softer rock increases in the constant)

Enesco in 1968 (the Roman method) evaluated the seismic effects of blasting based on the determination of apparent magnitude "Ma" with the following empirical relationship:

$$Ma = 0.67\left(\log V_{\max}^{2} * T + \log r + 4\log 4\rho V - 11.8\right) \qquad (16)$$

Where:

V_{max} = the maximum oscillation particle velocity, cm/sec;

T = oscillation period;

V = propagation velocity of elastic waves, m/sec;

ρ = the density of rock in which the seismic wave propagate, gm/cm^3;

r = distance from the shot point to measuring point, m.

The method above presumed the assessment of "acceptable intensity" for construction. Through field tests and usage of the above relationship showed the safe distance corresponding to "Ma" and consequently to a certain quantity of explosive could be determined, (Mohamed, 2010).

In 1968, the Russian method suggested by Medvedev to assess the safe distance "r" was deduced as follows:

$$r = K_e * K_t * K_c * R_{red} * Q^{\frac{1}{3}} \qquad (17)$$

Where:

K_e = firing sequence coefficient (instantaneous or delayed time) and the mining conditions (underground, open-pit or combined);

K_t = material characteristic coefficient in which wave propagation occurs;

K_c = construction coefficient (more or less damaged);

R_{red} = reduced distance depending on the function of the admissible intensity of vibration;

Q = explosive quantity

Table 19: Material Coefficient for different types of material, (Mohamed, 2010).

Class	Material	Material Coef. Fm
1	Armored concrete, steel, wood	1.2
2	Unarmed concrete, brick, brickwork, hollow concrete, stones, light weight concrete.	1
3	Porous concrete (gassed concrete)	0.75
4	Mixed bricks	0.65

Later on, Sadovski in Russia computed the non-dangerous explosive quantity by the following relationship:

$$V = \left(\frac{k}{D}\right)\left(\frac{Q}{D}\right)^{\frac{1}{3}} f(n) \qquad (18)$$

Where:

V = admissible particle velocity for construction, cm/sec;

k = global coefficient depending on the blasting and propagation conditions;

R = distance from the blasting point to construction, m;

Q = explosive quantity

f (n) = function for diminishment of seismic effect depending on firing system. It means on the number of blasting rows "n" and delay time between them. "Δt", the conditions are as follow:

i. For n Δt < 0.15 sec, f(n) = 1-12.9 (n Δt)2.

ii. For n Δt > 0.15 sec, f(n) = 0.275 / n Δt.

iii. For instantaneous blasting, f(n) = 1.

The Indian Standard in 1973 derived an empirical relationship that suggested the concept in which blast is scaled to the equivalent distance and the relation was expressed as:

$$V = k\left(\frac{Q^{\frac{2}{3}}}{D}\right)^{-B} \qquad (19)$$

The Swedish Detonic Research Foundation worked out an empirical formula to predict the vibration velocity as follows:

$$V = 700\left(\frac{Q^{0.7}}{D^{1.5}}\right) \qquad (20)$$

Davis et al., others investigators considered no particular charge symmetry. (Attewell P.B. and Farmer I.W, 1964) they proposed the most widely general formula was of the type:

$$V = kQ^{A} D^{-B} \quad (21)$$

Ghosh and Daemen in 1983 reformulated the propagation equation of the U.S. Bureau of Mines. This was later confirmed by Ambraseys and Hendtron and they incorporated the inelastic attenuation factor e-pD, (Ghosh at. el., 1983). The modified equations are:

$$V = k\left(\frac{Q}{D^{\frac{1}{2}}}\right)^{-B} e^{-pD} \quad (22)$$

$$V = k\left(\frac{Q}{D^{\frac{1}{3}}}\right)^{-B} e^{-pD} \quad (23)$$

Where

k, B, and p are empirical constants;

p is called the inelastic attenuation factor.

In 1991, Central Mining Research Station (CMRS) of India also established an efficient blast vibration predictor. The equation considers only geometrical spreading as the cause of the decrease in amplitude of ground vibrations:

$$V = n + k\left(\frac{Q}{D^{\frac{1}{2}}}\right)^{-1} \quad (24)$$

But in practical situation, the value of “n” was always negative, and this later resulted in the equation being:

$$V = -n + k\left(\frac{Q}{D^{\frac{1}{2}}}\right)^{-1} \quad (25)$$

Where:

n is the damping factor influenced by rock properties and geometrical discontinuities.

k is the empirical constant

Propagation law of blast-induced air overpressures has been studied by numerous investigators and is generally reported with cube-root rather than square-root scaled distances.

Equation 26 is commonly used for overpressure prediction:

$$P_{over} = 3300\left(\frac{D}{W^{\frac{1}{3}}}\right)^{-1.2} \qquad (26)$$

Where:

dB is the overpressure decibel level,

D is the distance from the blast hole (m),

W is the weight of explosive detonated per delay (kg)

P_{over} is the overpressure level (pa).

According to Nito-Consult AB the air overpressure propagation equation is estimated as follows:

$$P = 70 * \left(\frac{0.6Q^{\frac{1}{3}}}{R}\right) kPa \qquad (27)$$

Where:

Q is charge weight in kg;

R distance in m.

b) PPV Threshold predicting model

Particle velocity depends on both rock characteristics and the energy content of the incoming wave. During blasting operations, it is important to measure peak particle values for all three wave components namely Transvers, radial and vertical waves in order to pick the true reading of vibrations. Blast measuring instruments measure all three components and then work out the vector sum of the components. The peak particle velocity (PPV) is the point on which the velocity of particles gets to the maximum displacement, (largest amplitude) and at a specific frequency. It is very important to note that peak particle velocities of the various components may occur at different frequency values and therefore produce different values.

Another common way to measure damage on structures is through the calculation of the acceleration, (Mohamed, 2010). This can be achieved by using Equation 28 as shown.

$$PPV = 2\,\pi f A \qquad (28)$$

Where;

f'= frequency (Hz)

A = amplitude (mm)

π = 3.14 (amount of radians/rotation)

$$a = 4\,\pi 2 f\, 2A \qquad (29)$$

Where

a = acceleration (mm/sec/sec)

f = frequency (Hz) = 1/t

A = amplitude in mm

If acceleration is normalised through the gravitational pull of the earth which is 9814 mm/s^2, then the well-known "g" can be calculated.

$$(\text{Acceleration mm/s}^2) / (9814 \text{ mm/s}^2) = \text{amount of g} \quad (30)$$

According to the United States Bureau of Mines, it was recommended that the following index of damage, based upon acceleration is used. In cases where acceleration is less than 0.1g, no damage is expected. This is because acceleration between 0.1g to 1.0g minor damage is expected to be greater than 1^{-g} major damage.

At a given location, peak particle velocity (PPV) depends mainly on the distance from the blast and the maximum charge per delay. Scaled distance concept vs. particle velocity and air overpressure is generally used for blast vibration prediction. The most widely accepted propagation equation for ground and air vibration considering the damage to structures is of the format in Equation 31, (Mohamed, 2010).

$$V = K\left(\frac{R}{W^{\beta}}\right)^{-\alpha} \quad (31)$$

Where

K is a site constant;

R is the distance (m) from the blast source to a structure

W is the weight charge of explosives in (kg),

The model for determination of the allowed peak particle velocity has its origin in the Norwegian practice for prudent blasting in the last 30 to 40 years. The peak particle value is calculated by:

$$V = V_o * F_k * F_d * Ft \quad (32)$$

Where:

V_o = Uncorrected maximum value of vertical particle velocity measured in mm/s.

V_o is dependent on the kind of geological material of the ground,

F_k = construction coefficient = F_b * F_m

Fb = building factor,

F_m = material factor,

F_d = distance coefficient, which takes into consideration the distance between the blasting site and the critical object (0.5 and 1 for distance 200 and 5).

F_t = time coefficient, which takes into consideration how long the construction is exposed for blast vibration, see Table 20.

Table 20: Time coefficient, (Mohamed, 2010).

Duration of blasting work	Time Coef. F_t
Less than 12 months	1
More than 12 months	0.75

Scaled distance is the most commonly used method of predicting blast induced vibration by plotting the peak particle velocities versus scaled distance; this is achieved on the log graph for a particular area. Vibration data has to be collected through experimental field work, where various amounts of explosives are detonated at different distances from the blast and the peak particle velocities are recorded for each blast. The results are plotted on a graph. The y-axis represents peak particle velocity, while the x-axis represents scaled distance which is calculated using Equation 33:

$$ScalarDis\tan ce(c) = \left(\frac{D}{\sqrt{W}}\right) \tag{33}$$

Where

D = Distance (m)

E = Explosive mass per delay (kg)

A straight line is drawn by means of the method of least squares, after which Equation 34 is derived.

$$ppv = a\left(\frac{D}{\sqrt{E}}\right)^b \tag{34}$$

The straight line is given as $y = mx + c$, or in this case $V = -b(x) + a$,

Where

a represents the y-axis intercept,

b represents the slope of the graph,

V the peak particle velocity and x the scaled distance.

These equations are only valid for that particular area with the same geology and therefore, cannot be used to predict vibrations of another site with different geological formations, (Ozer, 2008).

2.2.4.9 Artificial intelligence prediction

Artificial predictions become very handy especially when there is an unusual noise or uncertainties that exist in the measured data of vibrations, statistical models. This type of intelligence predicts air vibration and peak particle velocity efficiently. Artificial neural network and fuzzy logic are the two most important concepts of artificial intelligence. They are very useful in modeling or prediction of one or more variables.

a) Artificial Neural Network (ANN)

Inspired by the function of the human brain, works on the AAN started when scientists realized that the human brain worked on a totally different basis when compared to common digital computers. The brain is a highly complicated processing system made from structural units called neurons hence ANN has been widely used in such different engineering branches as geo-technique, structure and tunneling, as well as medical sciences and management, (Meulenkamp and Grima, 1999).

Mohamed (2010) wrote that neural networks have become more popular compared to traditional statistical methods, neural network analysis has been found to be very useful in diverse, real-world applications. An artificial neural network can be defined as a data processing system consisting of a large number of simple, highly interconnected processing elements (artificial neurons) in an architecture inspired by the structure of the cerebral cortex of the brain (Tsoukalas et. al, 1996). These processing elements are usually organized into a sequence of layers or slabs with full or random connections between the layers. The input layer is a buffer that presents data to the network.

The following layer(s) is called the hidden layer(s) because it usually has no connection to the outside world. The output layer is the following layer in the network, which presents the output response to a given input. Typically the input, hidden, and output layers are designated the *i*th, *j*th, and *k*th layers, respectively. A typical neural network is "fully connected," which means that there is a connection between each of the neurons in any given layer with each of the neurons in the next layer. Artificial neural networks (ANNs) are a form of artificial intelligence that has proved to provide a high level of competency in solving many complex engineering problems that are beyond the computational capability of classical mathematics and traditional procedures. Back-propagation artificial neural network a Feed-forward network is considered the most popular, effective and easy-to-learn model for complex, multi-layered networks of the supervised learning techniques. The typical back-propagation network has an input layer, an output layer, and at least one hidden layer. Each layer is fully connected to the succeeding layers, as shown in Figure 21. In the back propagation training, the connection weights are adjusted to reduce the output error. In the initial state, the network begins with a small random set of connection weights. In order for the network to learn, a set of inputs is presented to the system and a set of outputs is calculated. The difference between the actual outputs and desired outputs is calculated and the connection weights are modified to reduce this difference, (Mohamed, 2010).

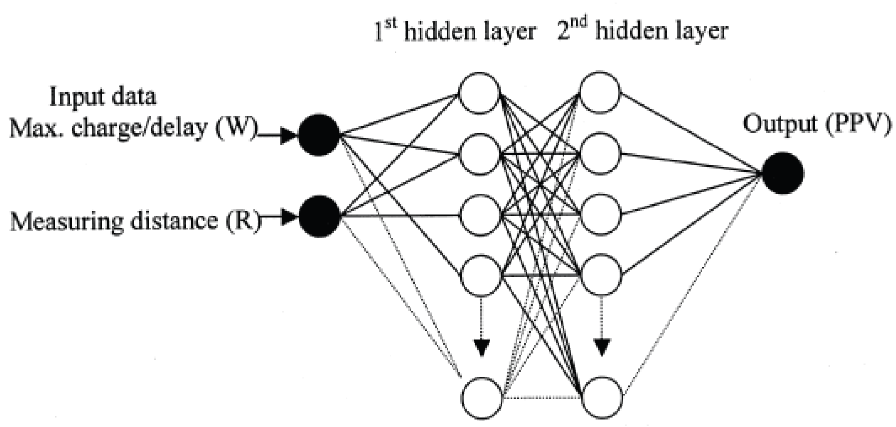


Figure 21: Back propagation training, (Tsoukalas, et al., 1996)

b) Fuzzy logic system

Whenever a mathematical problem becomes a challenge to solve and decisions have to be made with limited information, Fuzzy logic is preferable. Fuzzy models can be seen as logical models which use "if–then" rules to establish qualitative relationships among the variables in the model. Fuzzy set theory enables the processing of imprecise information by means of membership functions, in contrast to the classical set theory. The classical set (called crisp set) takes only two values: one, when an element belongs to the set; and zero, when it does not. In fuzzy set theory, an element can belong to a fuzzy set with its membership degree ranging from zero to one. The fuzzy logic operates under considerations of the state in which they take the form of subsets or fuzzy sets. The subsets are labeled in the sub sets: "low," "medium," "big," etc. A general fuzzy inference system basically consists of; fuzzification, knowledge base, a decision-making unit, and finally a defuzzification, the fuzzy system, (Sever, 2004).

2.2.4.10 Ground vibration control

The following parameters were suggested by Olofsson (1990) that affect ground vibrations generated during blast design:-

a. The maximum cooperating charge;
b. The number and frequency of delays. The introduction of a delay sequence can reduce the size of the maximum wave produced;
c. The bench height that consequently detects the blast hole depth;
d. The number of "decks" or layers of explosives and detonators in each hole;
e. Spacing, burden and number of holes, in the spacing to burden ratio;
f. The diameter of the shot hole, which will affect the amount of explosives used.

All the mentioned parameters can be addressed when we mitigate the effect of vibration through the following techniques:

1. Blasting techniques

It is very important to use a blast design that produces the maximum relief as practical as possible in the given situation. This is because all the holes which have sufficient relief such as those holes close to the free faces produce less ground vibration. The use of delay blasting techniques establishes internal free faces from which compressional waves produced later in the blast can delay patterns, maximum relief can be retained, (Mohamed, 2010).

In general, when blasting multiple holes, greater relief can be obtained by using a longer delay between rows than between the holes within a single row. A delay of at least 2–3ms/m of burden between the holes within a row is recommended for the necessary relief and best fragmentation.

It is also recommended that a spacing/burden ratio greater than one is used. The presence of weak seams or irregular back break may dictate the local use of a spacing/burden ratio close to one. It is further important to control blast limits for vibration control. This can be achieved by using reference points whenever drilling is about to commence. Make sure that the patterns are well defined and labeled, (Tose, 2010).

2. Sub-drilling

It is very important to restrict the amount of sub drilling to the level required to maintain good floor conditions. Typical sub drilling for holes inclined 3:1 is 30% of the burden at floor level. Tape each drill hole and match it to the face height. If hole depth is greater than intended, backfill with drill cuttings or crushed stone. Excessive sub drilling can increase vibration because of the lack of a nearby free face to create reflection waves.

3. Charge per delay

Charge per delay is the amount of explosives that detonates within a given period. By reducing the charge per delay, it is possible to keep vibrations under control. The following are some of the methods that will assist reduces the charge per delay:

- Reduce hole depths with lower bench heights and increase specific drilling,
- Use smaller diameter holes,
- Subdivide explosive charges in holes by using inert decks and fire each explosive deck with initiators using different delays,
- Use electronic or mechanical timers to increase the available number of periods of delays. Non electric delays coupled with surface delay connectors can provide similar flexibility.

4. Explosives

Eliminate or reduce hole-to-hole propagation between charges intended to detonate at different delay periods. Use explosives that are less abrasive such as P100 (an explosive mixture that contains straight emulsion produced by AEL). Hole to hole propagation occurs when the explosive charges or blast holes are only a few meters apart such as trenching, decked holes, or underwater excavations, or at greater distances when blasting inter-bedded soft and hard layer rock, such as coral or mud-seamed rock, that is saturated with water.

5. Using Electronic blasting system

Using NONEL blasting system increase vibration because they have a high scattering effect compared to ELECTRONIC detonators.

2.2.4.11 Air overpressure (Noise)

There are five principal sources of air overpressure from blasting at surface mineral workings (Mohamed, 2010):

1. The use of detonating cord which can produce high frequency and hence audible energy within the air overpressure spectrum.
2. Stemming release, seen as a spout of material from the boreholes, gives rise to high frequency air overpressure.
3. Gas venting through an excess of explosives leading to the escape of high-velocity gases, give rise to high frequency air overpressure.
4. Reflection of stress waves at a free face without breakage or movement of the rock mass. In this case the vertical component of the ground-vibration wave gives rise to a high-frequency source.
5. Physical movement of the rock mass, both around the boreholes and at any other free faces, which gives rise to both low and high-frequency air overpressure.

a) The steps to reduce air vibrations:

Detonating cord should be used as sparingly as possible, and any exposed lengths covered with as much material as possible. Just a few meters of exposed cord can lead to significant amounts of audible energy and, hence, high air overpressure levels. Stemming release can be controlled by detonation technique, together with an adequate amount of good stemming material. Drill fines, while readily available, do not make good stemming material. The use of angular chippings is better. It should be noted however that detonation cord and stemming release have been virtually eliminated with the use of in hole initiation techniques. Gas venting results from overcharging with respect to burdens and spacing or, perhaps, a local weakness within the rock, and is also typified by the occurrence of fly rock. Its control is essential for economic and safe blasting, and is considerably aided by accurate drilling and placement of charges, together with regular face surveys.

The controllable parameters surrounding site geology, topography, and meteorological conditions can be controlled to some extent by adjustment of blast pattern and blaster in charge judgment for blasting operation.

Rock mass damage in open pit mining usually occurs due to a change in the induced stresses and also due to blasting. As excavations are created, the in-situ stresses redistribute around the boundary of the openings. As a result, high stresses may be experienced on the backs and corners of the excavations, while low stresses may be experienced within some of the exposed open walls. The de-stressing of the walls will tend to open existing cracks, as a result of movement of the rock mass into the excavation. Extensometers can be used to measure the amount of wall relaxation and borehole camera surveys determine the exact location and nature of the damage. Blast damage is defined as the creation, extension and/or opening of pre-existing geological discontinuities in the rock mass. Blast induced damage weakens a rock mass, potentially leading to stability problems when the excavation size is increased. Empirical evidence from open pit mining suggests that near field peak particle velocity levels from blasting can be linked to rock mass pre-conditioning and damage. A relationship between the critical peak vibration velocity and rock mass damage in the near field (within a charge length) can be determined by correlating measured vibrations and the damage determined with geotechnical instrumentation (Persson, et al. 1994). The magnitude of the vibrations depends upon the nature of the rock mass, the blasting agent, the hole diameter used, the drilled pattern (burden, spacing, hole angle and distance of the holes to the exposed walls) and the hole deviation. Back analysis at Mount Isa Mines suggests that blasting may control up to 15% of the overall stope hanging wall behaviour, (Villaescus et al., 1997).

Blasting has positive results in hard rock fragmentation process. However, it also comes with negative impacts such as damage to in-situ remaining rock, cracking of neighbouring houses, physical damage on structures around the site etc. Most blast damage is attributed to peak particle velocities resulting from the dynamic stresses induced by the explosion. While it is recognized generally that the gas pressure assists in the rock breaking process, it has also shown the potential to cause destruction in the surrounding.

Rock jointed formations tend to have different strength as the degrees of interlocking between their masses vary from one rock to the other. Furthermore, the amount of damage may vary depending on the distance from the explosive charge. One of the major forms of damage is that of over-break in the design. This increases the cost of the operation in terms of drilling, extra explosives used and material movement. Furthermore, weakening of the remaining rock especially on the slope designs of the final pit limits thereby inducing slope failures which pose a great risk in the operation as this may result in catastrophic disasters if not well managed. Another form of blast damage although not easily quantified is that of vibrations produced from the shock wave in form of P and or, S waves. As earlier indicated, rock breakage takes place by wave interaction and most of the time; some of these waves tend to have unwanted effects on the surrounding rock. These waves are capable of damaging infrastructure in the neighbourhood. In order to reduce the damages resulting from blasting operations, planning the face for drilling, charging and blasting sequence has got to be done properly, coupled with discipline in executing the design. Furthermore, monitoring of vibrations has got to be done every time a blast is conducted. Below is a detailed blast design theory that is used to ensure that bench blasting is up to the required standard, (Tose, 2010).

2.3 Blast Designs

After having identified all environmental impacts that are generated during blasting operations, an effective and optimized blast design considers a number of variables that need to be synchronised into one unit. The following variables play a huge role in blast designs: safety, blast hole diameter, blast hole depth, stemming, stemming type, burden, spacing, sub drill, explosives type, explosives energy, explosives quantity, initiations system, inter hole delays and geology of the area being blasted.

This section therefore looks at parameters that bring about good blast designs through bench blasting parameters.

2.3.1 Bench Blasting

Bench blasting is the most commonly practiced methods in open pit mining. In this method, an optimum blast pattern is designed in order to fragment rock to the desired size at a minimum cost. To develop such patterns, basic understanding on the principles that affect a given blast design and the expected fragmentation must be known. This has brought about an empirical approach that is used in blast design; note that blasting is a never-ending process of fine-tuning and modifications, (Dowding and Aimone, 2002).

The parameters considered when coming up with blast include blast geometry, geological formation, geotechnical properties, and explosive's properties delay timing and initiation sequence of blasting.

2.3.1.1 Blast geometry

a) Drilling Diameter (D)

Factors governing the choice of blast hole diameter include bench height, type of explosives, rock characteristics, vibration control and specific charge of the blast hole. Diameter of a blast hole affects the burden and spacing to be used when designing the pattern, Figure 22 demonstrates the drilling and blast geometry and the parameters which are critical in determining the burden and spacing.

In strong, massive rock, smaller diameter blast holes have the advantage of better explosives distribution compared to large diameter holes, (Tose, 2010).

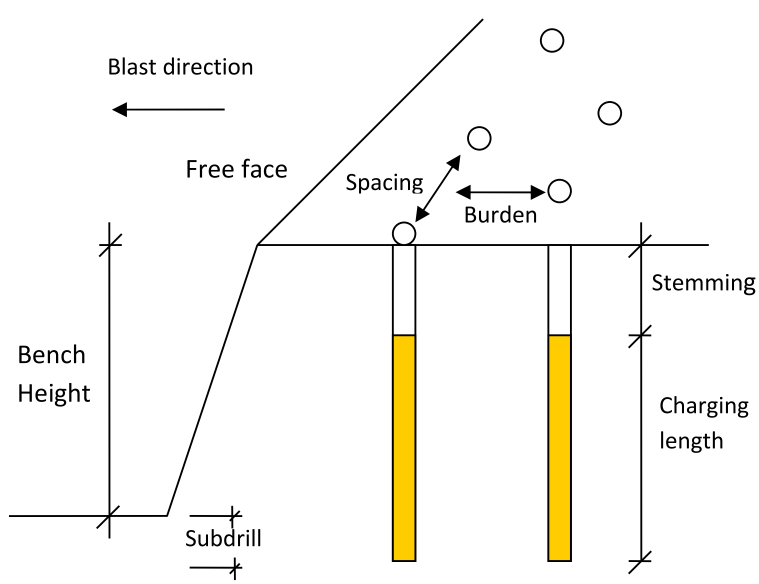


Figure 22: Blast geometry (Tose, 2010)

When the diameter is increased and the explosive powder factor remains constant, the larger blast hole patterns generally give coarser fragmentation and higher powder factors then have to be used to compensate for this. Where joints or pronounced bedding plane partings divide the burden into large blocks, acceptable fragmentation is often achieved only where each block is intersected by a blast hole. This structure usually necessitates the use of smaller diameter blast holes and correspondingly smaller blast hole patterns. In strata that are dense with a network of natural fractures, fragmentation tends to be structurally controlled. For this reason, increases in blast hole diameter can be accommodated with only a marginal penalty in fragmentation, (Tose, 2010).

The diameter of the hole is selected in such a way that it meets the combination with appropriate positioning of the holes and that which is able to give proper fragmentation suitable for loading, transportation equipment and crusher used. The other most important factor is that of the environmental concern to the public as hole diameter has a direct impact to the surrounding in that the bigger the hole, the more explosives are pumped into it and hence increase on the cooperating charge.

Drilling and blasting becomes economical with increase in diameter. When the blast hole diameter is increased and the powder factor remains constant the large blast hole pattern gives coarser fragmentation. By keeping burden unchanged and elongating spacing alone the problem can be overcome.

When joints or bedding plane divide the burned into larger blocks or hard boulder lie in a matrix of softer strata acceptable fragmentation is achieved only when each boulder has a blast hole, which necessitates the use of small diameter blast holes. Smaller diameter holes also tend to reduce on the amount explosive charge used hence reduces the vibrations produced during blasting operations. Blast hole diameter ranges from 35mm to 400mm. In Zambia, blast holes ranging from 45mm to 127mm are used in limestone operations and the same could be used in copper mines where vibration control is inevitable. Langefors and Kihlstrom in 1963 suggested that the diameter must be kept between 0.5 to 1.25 percent of the bench height.

Effect of excessive hole diameter

Whenever the hole diameter is very big, this results in a high explosive powder factor thereby increasing the levels of vibrations during blasting.

Effect of insufficient hole diameter

This lowers powder factor but may result in poor fragmentation if not well planned. This further increases time for charging, stemming and tie up (Bulk Mining Explosives, 2009).

Selection of hole diameter

There are advantages and disadvantages of drilling holes with a small or big diameter holes. As a rule within a particular range of applicable sizes, larger holes have more finance benefits in most big mining operations compared to small diameter holes (Tose, 2010). Tose indicated a number of advantages for drilling large diameter holes, some of which include the following: less susceptible to blocking, fewer to drill thereby making it easy to supervise, drilling is done more accurately, it is less likely for any hole to be omitted and timing is made easier with reduced working time both when drilling and timing the blast. Further advantages include:

- Suitability for bulk explosives;
- Explosives detonate more effectively;
- Fewer initiation system units needed;
- Better productivity on higher benches;
- More accurate.

On the other hand, there are advantages with drilling small diameters holes such as the following:

- Better fragmentation for same powder factor;
- Better ability to fragment hard capping;
- Better control of back break;
- Better control of flyrock, vibration and noise;
- Better ability to drill inclined holes; and
- Suitable for low benches.

It is always recommended that a best choice must fall where safety and environmental concerns are addressed in order to safe guard the operations from unwanted closures, therefore, small diameter holes are more appropriate for control blasting techniques in close proximity to the community.

b) Burden (B)

Burden is normally considered to be the distance between rows of holes in a direction that is perpendicular to the main free face, (Bulk Mining Explosives handbook, 2009). Burden is the closest distance to free face when the hole detonates. The burden has to match the blastability of the rock, energy of the explosive and the delay between the rows or vice versa, (Orica, 2008).

Burden can be calculated using Equation 34:

$$B = \sqrt{\frac{LM_c}{aHK_{tech}}} meters \qquad (34)$$

Where:

L is the charge length (m);

M_c is the specific charge (kg/m)

H is the bench height (m)

K_{tech} is the powder factor,

“a” is a value for the burden to spacing ratio, (range from 1 to 1.2, where 1 is used when the burden is equal to the spacing and a square pattern results. When the value is 1.15, then the staggered pattern was used), (Tose, 2010).

Effects of excessive burden

Whenever Burden is in excess, the amount of energy and burden velocity will be too small that the blast hole is not able to break the expected volume as interaction between holes is compromised to break the rock effectively. Figure 23 (b) shows the effect of excessive burden that results into poor breakage. This consequently results into high vibration, fly- rock and air blast. The muckpile will have a high profile of material not well fragmented and compact, (Orica, 2008).

Effects of insufficient burden

Whenever the burden is too small, in some cases as low as 2m, the energy will be too much with a high probability for face bursting and fly-rock. Figure 23 (a) shows the effect of small burden that results in flyrock to the surrounding. The burden velocity may also be too high. The front row may move too far and not shield from fly- rock from the next row resulting in fly- rock and air blast. The muckpile profile will be very low and often poorly fragmented (Orica, 2008).

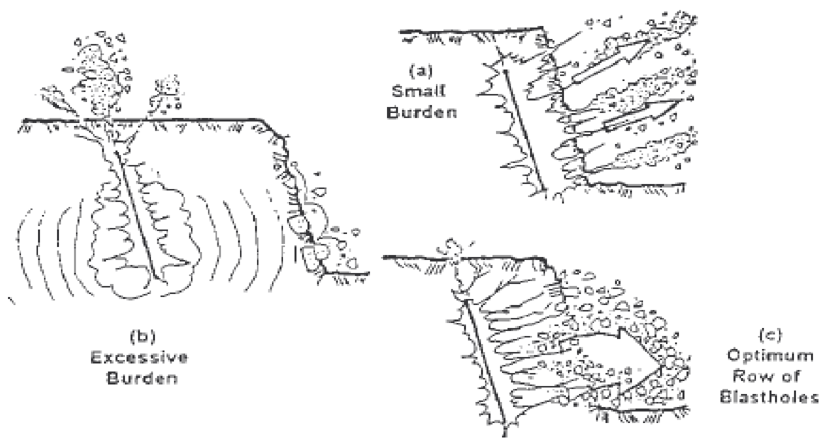


Figure 23: Effect of Burden on a blast design (Tose, 2010)

c) **Spacing (S)**

Spacing is the distance between holes in a row. The spacing is a function of burden (Bulk Mining Explosives handbook, 2009). This is calculated as shown in equation 35.

$$S = M * B \qquad (35)$$

Where

M is spacing to burden ratio

B is the burden (m)

Effects of excessive spacing

Whenever spacing between holes is too large, the rock between the holes is poorly fragmented. This problem will increase the burden of the next row, and multiply in size by the number of rows with a probability for air blast, fly rock and back break. This further result into toe problems and an irregular face, (Jimeno, 1995).

Effects of insufficient spacing

Whenever spacing is too small, the energy overlap between holes will be too large with a probability of face bursting, fly rock and air (Bulk Mining Explosives handbook, 2009).

Spacing to burden ratio (M)

The hole spacing should range between 1 and 1.2 times the burden in order to have better fragmentation results, (Tose, 2010). As can be seen in Figure 24, the optimal ratio between hole spacing and burden is 1.15 as this produces an equilateral triangle between holes (a triangle with sides of equal length) in a staggered pattern and therefore provides the most even distribution of explosive, (Tose, 2010).

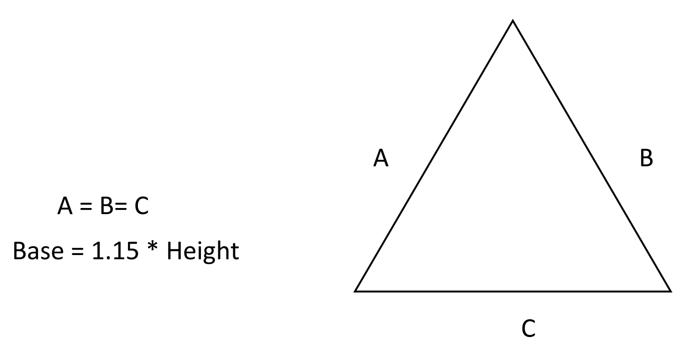


Figure 24: An even distribution of explosives

Bench height (H)

This is the vertical distance between the bench's toe to the bench's crest. It is determined by the working specifications of the loading equipment. It limits the charge of the hole diameter. When the bench height to burden ratio is large, it is easy to deform and displace rock especially at its centre (Ash, 1968). For Quarry and coal stripping operations, Height/ Burden ratio (H/B) must be greater than three due to the equipment being used. In other words, the bench height should be at least twice the burden, (Orica, 2008).

d) Sub drill (J)

In order to ensure an even and level bench surface, blast-holes are usually drilled slightly longer than absolutely required. This is because it is easier to fill the resultant slight over-break by filling the voids than to rip and bulldoze the mounds resulting from under-drilling. It is usually calculated from blast hole diameter when vertical blast holes are drilled. The sub drilling of the first row reaches value of 10D to 12D where D is the diameter of the drill hole. About 10% of sub drilling gives better fragmentation in the rock mass and lesser ground vibration.

In general, sub drilling should be 0.3 times the burden. Under different toe conditions sub drilling may be up to 50 percent of the burden.Sub drill is necessary because the stress waves at the bottom of a blast hole are not at maximum amplitude. At a finite distance from the hole bottom, the stress waves are at their maximum. One major reason for fragmentation problems such as boulders is presence toes at a blast area. Sub drills are therefore designed in order to reduce the stress wave amplitude at the bottom level of the bench. Sub drill for vertical blast holes should be 0.3 times the burden as shown in Equation 36.

$$J = 0.3 * B \qquad (36)$$

Where B is the burden

Effect of excessive sub drill

Excess sub drills increase vibrations because once the hole detonates below broken material, the waves propagation is supported to transmit in continuous formations. Furthermore, cost of drilling and blasting increases because the material on the top part of the underlying bench is fragmented excessively thereby posing challenges when drilling on the preceding bench and this ultimately results into loss of production time and reduced bit life.

Effect of insufficient sub drill

Whenever the sub drill is insufficient, uneven floors are an end result with high bottoms. This brings about difficult digging conditions at the bottom of the muckpile, (Bulk Mining Explosives handbook, 2009).

e) **Blast Hole Inclination (β)**

In recent years, attention has been given by open pit operators to the drilling of blast holes up to 20° vertical. The benefits from inclined charges are reduction of collar and toe region less sub drilling requirement. Uniformity of burden throughout the length of blast hole and drilling of next bench is easier. Air blast and fly rock may occur more easily due to smaller volume of material surrounding the collar inclined hole are successively used in Europe where high benches and smaller diameter holes in medium to higher strength rock exist. In case the face is high the use of vertical blast holes produce a considerable variation in burden between the top and bottom face which is the basic cause in the formation of toe. Angle greater than 25° are less used because of difficulty in maintaining blast hole alignment excessive bit wear and difficulty in charging blast holes. The blast hole length L increases with inclination as can be calculated using Equation 37.

$$L = \frac{H}{\cos\beta} + \left(1 + \frac{\beta}{100}\right) * J \qquad (37)$$

Where,

β in degrees represents the angle with respect to the vertical

H is the bench height (m)

J is the sub drill (m)

Effect of excessive hole inclination

When the hole inclination is too high, this results into flyrock ejection and poor break on the toe of the bench as there is an increase on burden and that consequently results into poor breakage.

Effect of insufficient hole inclination

When the hole inclination is too shallow, this brings about difficulties when drilling the holes thereby increasing the cost of drilling due to excessive bit wear.

f) Stemming (T)

This is the top part of the blast hole that remains uncharged and filled with material such as sand, drill chippings and aggregate. Stemming material prolongs the action of gas masses generated after detonation of explosive material and improves breakage. In order to extract the maximum energy from the expanding gases, the stemming plug must not blow out and allow gases to escape prematurely. Optimum stemming increases as the quantity and competence of the rock decreases, (Jimeno, 1995).

A suitable stemming column of suitable length and consistency enhances fracture and displacement by gas energy. The amount of unloaded collar required for stemming is generally from one half to two third of the burden, this length of stemming usually maintains sufficient control over the generation of the objectionable air blast and flyrock from the collar zone as can be seen in Figure 25. When the burden has a high frequency of natural crack and planes of weakness relatively long stemming column can be used. When the rock is hard and massive the stemming should be shortest which will prevent excessive noise, air blast and back break.

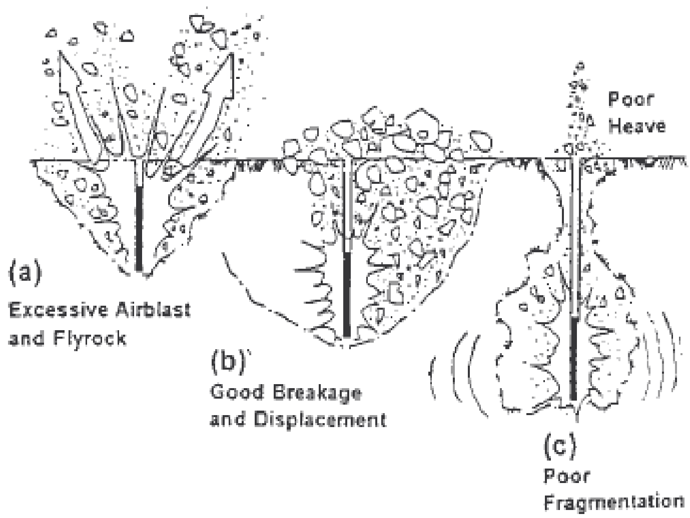


Figure 25: Crater tests to optimize stemming (Tose, 2010)

Theoretically, for an isotropic homogeneous material, burden and stemming should be equal for stress balance in solid rock, (Konya, 1991).

The amount of stemming or collar should be used as a direct function of the burden as is shown in Equation 38.

$$T = 0.7 * B \qquad (38)$$

Stemming must be effective for a period of about 5ms per meter of burden.

As a general rule, stemming length should be between 15 to 30 multiplied by the hole diameter. A practical and effective stemming length for good quality stemming material in a competent rock is 20 multiplied by hole diameters (Bulk Mining Explosives, 2009).

Effect of excessive stemming

Whenever the stemming height is excessive, rock breakage at the hole collar is poor and this results in big boulders, (SME, 2012).

Effect of insufficient stemming

Whenever the stemming height is too short, air pressure can evolve through the hole thereby disturbing nearby residents with flyrock and noise, (Roxborough, 1992).

g) Stemming Material

An optimum stemming particle diameter is equal to 0.05 of the hole diameter (Konya, 1991). This is calculated using Equation 39.

$$St = 0.05 * D \qquad (39)$$

Where

St is the stemming particle diameter in mm,

Dis the diameter of the blast hole (mm)

The type of stemming material that is used is very important for effective results. Crushed rock is suitable for use as stemming especially in hard rock since it is more resistant to ejection of the blast hole. The most effective stemming has high shear strength and a high density, (Bulk Mining Explosives handbook, 2009).

2.3.2 Geological formation

In order to design an optimum blast, there is need to understand the geological properties of rock. This is very important in blasting because it is from this understanding that the selection of explosive with the correct amount of energy is calculated in order to break the rock effectively. Below is the geology of Nkana and its geological properties.

2.3.2.1 Local Geology

Nkana ore bodies were discovered in 1910 and this form part of Zambian Copperbelt which happens to be one of the largest metallogenic provinces in the world. Nkana ore bodies cover the following; Area J Open pit, Mindola North, Mindola Sub vertical, Central and SOB.

Nkana Mine is on the western side of the Kafue Anticline in the Chambeshi-Nkana Basin elongated in a northwesterly direction. The predominantly mega structure of the Nkana Mine Site area is the northwest plunging Nkana Syncline. The structure is asymmetric. The axial zone of the Nkana Syncline is characterized by complex folding that dies out on the steeply dipping flanks, (Goelogical Guide to Nkana, 2005).

Current open pit and the underground workings include Area J open pit, South Ore body and part of Central Shaft. These are located on the northeastern edge of the folded axial zone, whereas the other part of Central, Mindola and Mindola North Shafts are mainly in the relatively under deformed, steep to shallow-deeping northeastern flank of the Nkana Syncline. See figures 26 and 27 for Area J open pits cross sections.

Figure 26: Nkana Syncline on Northern end of the Area J Open Pit

Figure 27: Folding's at Area J Open Pit

The Nkana Orebody consists of mineralised sedimentary rocks in a sequence of quartzite, sandstone, black shale, dolomite, and argillites. The orebodies cover a strike length of about 14 kilometres, with a 1.2 kilometres barren gap between Central and Mindola Shafts. The main copper ore minerals are chalcopyrite and bornite, with subordinate chalcocite oxidation occurring at North Shaft especially in the porous sandstone member. Minor malachite, pseudomalachite, chryscolla, native copper, cuprite and libethenite are associated with localized leach zones. Carrolite and cobaltiferous pyrite are the major cobalt bearing minerals. At Mindola and North Shafts, the minerals typically occur as fine dissemination, commonly aggregated into clots and blebs in richer units. In structurally complex SOB and Central shafts, the minerals occur predominantly in the dolomite bands, large streaks lenses and crosscutting veins. Enrichment on the crests and in the trough of the folds is common, (Geological Guide to Nkana, 2005).

2.3.2.2. Regional Structure Complexity

The deformation of the rocks of the Nkana Mining Area is related almost completely to the Lufilian Orogeny, which affected both the pre-Katanga and Katanga rocks. Little is known about the pre-Lufilian tectonic history of the Basement Complex within the Mining area, and so the following description is confined to a summary of the Lufilian structures.

The regional tectonic framework is described more fully by Garlick in 1961. The predominant mega-structure of the Mining Area is the northwest-plunging Nkana syncline. This structure is asymmetric, with a curved axial plane inclined steeply to the northwest in the trough of the syncline but upright to westward dipping at higher structural levels, (Garlick, 1961). A complex occupies the axial zone of the syncline but this folding dies out on its flanks. As a result of the northwesterly plunge of the Nkana syncline, the present land surface represents a horizontal section through progressively higher structural levels of the syncline from southeast to northwest. Similarly, current underground mining operations at South Ore body Shaft are located along the northeastern edge of the folded axial zone whereas at Central, Mindola and North Shafts these operations are mainly in the relatively undeformed steep to shallow-dipping northeastern flank of the Nkana syncline.

Outside the working areas, surface mapping and diamond drilling from both underground and surface provide the basis for structural interpretation.

Within the axial zone of the Nkana syncline, shear folding is characteristic of the deformation style of the Lufilian Orogeny, with transposed bedding attenuated to sheared-out folds limbs, axial swelling and chevron structures being developed. The structures indicate thrusting directed towards the northeast, with decollement present at the Basement/Lower Roan and Footwall Formation/Ore Formation boundaries. In addition, the Basement Complex adjacent to the Lower Roan cover has been sheared with refoliation developed parallel to the axial planes of the folds in the Katanga rocks, (Geological Guide to Nkana, 2005). During the early stages of deformation, open symmetrical folds such as the C anticline at South Ore body Shaft and the Zero Anticline at Central Shaft, were formed. As shearing stress built up, these earlier structures were modified; in some cases substantially. Post-shearing effects such as reversals of fold plunges, trans-current movements, faulting and kink zones are probably related to the late-tectonic up-doming of the Kafue anticline as the Katanga cover rocks adjusted themselves to essentially vertical movements between blocks of different competency within the underlying basement. In places, for example at 4800S (SOB), there is evidence to suggest that the late-tectonic vertical movements have accented the pre-Katanga palaeo-topography, which itself has influenced the deformation by the buttressing effect produced by the granite gneiss palaeo-hills, (Geological Guide to Nkana, 2005).

The following brief descriptions of the structure at Mindola and North Shafts and Central and South Ore body Shafts serve to demonstrate the contrasting deformational styles on the flanks and in the axial zone respectively of the Nkana Syncline.

i. Mindola and North Shaft

In the North Shaft area, that is between 1550N and 3700N the Ore body dips at an average of 30° westwards above 1380L with only minor rolls to disturb its planar habit. Below 1380L the dip increases to about 45° and increases both southwards to the dyke at 1300N and with increasing depth, locally it reaches about 70°. South of the dyke, the dip becomes steep to and assumes a maximum reverse value of 75° at 5660S. Dewatering and exploration drilling in the area between 2330S and 5000S have indicated folding of the Hanging wall Beds. This folding and the reversal of dip in the far south of Mindola is probably related to the buttressing effect of the Kitwe Barren Gap combined with the axial zone deformation where it plunges northwards through the barren gap from Central Shaft area.

ii. Central shaft

At Central Shaft, the folded lower part of the northeastern flank of the syncline is exposed by past and current mining operations. There are three major folds present on the northeastern flank. From east to west, these are the North, Zero and J folds. The plunges are variable between 20° and 45° to the northwest. The Zero Fold consists of an open upright symmetrical anticline with maximum amplitude of 350m and an asymmetric complementary syncline to the east. Axial planar shear has occurred on the crest of the Zero Anticline, affecting the Ore Formation and hanging-wall beds in particular, (Geological Guide of Nkana, 2005). Traced up-plunge, the Zero Anticline becomes a tight asymmetric structure similar to the North and J Anticlines, which flank it. These folds are shear folds, with axial planes steeply inclined to the southwest at the top but overturned to the northwest at the base of the fold. Typically, the anticline has short northeast facing limbs and long southwest-facing limbs, indicating that shearing stress was directed towards the northeast.

iii. South Ore body Shaft

The complex folded axial zone of the Nkana Syncline is best known at South Ore body Shaft, where past and current mining development, including hanging wall dewatering drilling and exploration drilling from both underground and surface has been and is being carried out in this part of the main syncline.

There are at least six major complex folds from east to west. The folds are arranged en echelon and plunges undulate erratically. There is a marked flattening of plunge in barren gap areas, which are associated with basement highs, with a corresponding steepening of plunge off the flanks of the barren gaps. The folds are tightly compressed, with attenuation of limbs, bedding plane crumpling, axial plane cleavage and axial plane shearing well developed. Minor shear folds are commonly developed on the limbs of the major structures, the axial zones of which are usually feathered out by axial plane shear. These shear features are particularly well displayed where units of different competency are juxtaposed for example the Footwall Formation and the Near Water Sediments and Dolomite Argillite Sequence on either sides of it. The C-anticline, with maximum amplitude of 250m, is a good example of an earlier more open, upright symmetrical fold which has been modified by subsequent axial plane shearing.

iv. The West Limb

The West Limb of the Nkana Syncline is only partially understood, the geological evidence being obtained from diamond drilling and surface mapping. However, it is clear that the West Limb is folded in a similar fashion to the axial zone of the main syncline. North of 7800N (Central Shaft), folding of the West Limb dies out gradually as the Nkana syncline opens out into the Chambishi-Nkana Basin. The major structure on the west Limb between the nose area and 7800N (Central Shaft) is the Luanshimba Anticline. This has a shallow undulating plunge and its limbs have been deformed by a complex of en enchelon shear folds with curved axial planes fanning outwards from the Luanshimba Anticline.

v. Metamorphism

Broadly speaking, the lithological terms applied to the Katanga rocks are misleading in that they imply a lower grade of metamorphism than is, in fact, the case. All the sediments have undergone a degree of metamorphic recrystallisation, although in the case of the quartzitic rocks this has not been sufficient to destroy the sedimentary micro – and micro-textures. The most common metamorphic minerals are biotite, chlorite, tremolite, talc, sericite and albite. This assemblage indicates a metamorphic grade in the Greenschist Facies. There is some variation in grade from the quatz-albite-epidote-biotite sub facies in the more highly deformed areas to the quartz-albite-muscovite-chlorite sub facies in the less deformed rocks, (Geological Guide to Nkana, 2005).

2.3.2.3. Stratigraphy

The local stratigraphic column for Mopani Copper Mines Plc is given in the Figure 27. The plane of unconformity between the basement Complex and the Katanga System is highly irregular in places although the paleotopography represented by this unconformity has probably been accented as a result of competency differences during deformation between the basement and the elements of the basement Complex and between the Katanga cover rocks. In places, a streakly, kaolinised, micaceous feldspathic arenite containing argillite fragments is present at the basal Katanga unconformity and has been interpreted as a regolith of the pre-Katanga land surface. Elsewhere, the basement-Katanga contact is represented by a zone of shearing.

The basic igneous rocks have intruded into the metasediments of the upper Roan, Mwashia and lower Kundalungu Groups, while a lamprophyre dyke cuts the Lower Roan rocks at Mindola Mine.

a) Basement Complex

The Basement is divisible into an assemblage of metasediments, assigned to the Lufubu System, and a suite of igneous rocks including granitoid gneisses and intrusive and segregation pegmatites.

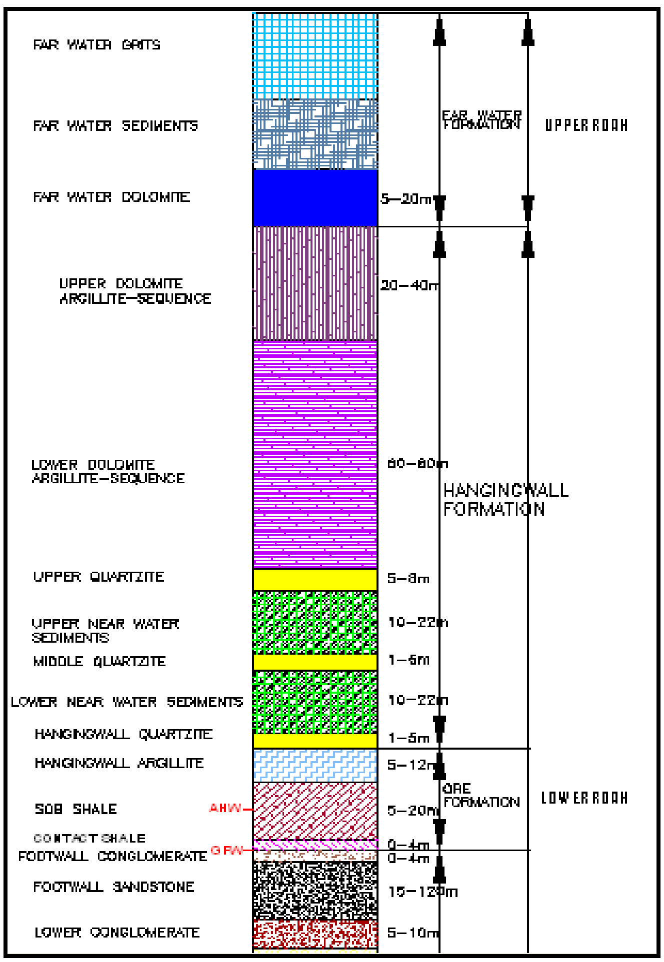


Figure 28: Stratigraphic column of Mopani (Geological Guide of Nkana, 2005)

At Mindola Shaft and North Shaft orebody, the Lufubu rocks consist predominately of impure quartzite, phyllite and semipelitic schist, with meta-arkose also present. At Central and South Shafts orebody, the Lufubu system includes migmatitic and banded gneisses, compared to the Mindola exposures, it is more schistose with two and three mica schist's predominating over the more psammitic rocks such as quartzite and meta-arkose. There is also a schistose conglomerate at Central Shaft.

The igneous rocks are intrusive into the Lufubu system but precise relationships are not always easy to decipher due to subsequent tectonism and metamorphism. This is particularly true at South Orebody Shaft. When undeformed, the granitoid gnesses are grey to pink in colour, medium-to coarse-grained, generally holocvyrstalline but occasionally prophyritic to porphyroblastic. They are mainly granodioritic in composition but basic gand to a lesser extent siliceous phases are not uncommon. The granite gneiss Lufubu contact is either sharp or gradational. The presence of flaser gneisses and chlorite-biotite-sericite shists at the contact suggests that marginal shearing of the gneisses has occurred.

Lenses, on all scale, of granite gneiss within the Lufubu schists indicate that more widespread tectonic interslicing of the granite gneiss and Lufubu metasediments has taken place. It is possible that some of the schistose elements within the Lufubu system are in fact cataclasites derived from the granite gneiss and basic intrusives. The migmatites and banded gneisses within the Lufubu system are presumably granitisation products related to the intrusion of the granite gneiss. There are no recorded occurrences of the granite intruding the Katanga cover rocks. There are at least two phases of basic intrusives present as ramifying networks of dykes cutting both the granite-gneiss and Lufubu metasediments. They also appear to be pre-Katanga in age.

b) The Katangan system

A detailed Stratigraphy of the sequence either exposed during underground mining or encountered by diamond drilling is shown in the figure below. The allocation of the members to a particular group is provisional and does not necessarily agree with the finding of the Copperbelt Stratigraphic Correlation Committee. The nature of the basal Katanga unconformity is not always clearly defined due to the presence of much locally derived basement material in the lowermost part of the Katanga sequence.

c) The Lower Roan Group

The Footwall Formation consists of six members, although not all of those are present everywhere. Much of the underground workings are developed in the upper part of the Footwall Formation. The Basal Conglomerate, 0-4m thick, is a variable member partly filling the valleys and mantling the hill slopes of the irregular pre-Katanga erosion surface. In composition it varies from a well-rounded pebble conglomerate to a massive boulder bed, while in some exposures it is an angular scree deposit. The clasts are locally derived from the Basement Complex. The Basal Quartzite Member is more widespread than the basal Conglomerate, which it overlies. The Basal Quartzite is either absent or greatly reduced in thickness over the palaeohills but thickness rapidly towards, and attains a maximum thickness of 150m in the intervening valleys. Typically this member is a somewhat glassy slightly feldspathic quartzite, which displays large scale cross-bedding. At the 3030L/600N position at Central Shaft, a black flinty fine-grained quartzite with thin impersistent pink quartzite with thin impersistent pink quartzo-feldspathic layer is developed at the base of this member. The Basal quartzite grades up fairly rapidly into Basal Sandstone Member, which has a maximum thickness of 20m and is absent over the palaeohills. The Basal Sandstone varies from a fairly massive argillaceous quartzite to a varied sequence of rapidly alternating feldspathic quartzite, dark siliceous argillite, feldspathic grit and pebble conglomerate. At the very top of the basal Sandstone member immediately below the lower Conglomerate, fine-grained beds are overlain by increasingly coarse ones, and interstitial anhydrite, which may be leached out locally, is abundant in places. The reverse grading and the presence anhydrite may indicate an off lap condition, (Garlick, 1969).

The overlying Lower Conglomerate member is a transgressive lenticular unit usually not more 10m thick. It is thickest on the flanks of the palaeohills but pinches out against these basement highs and lenses out over the intervening basement lows. This pinching out is well displayed at SOB Shaft on the 3140 level between 790S and 1090S. At Mindola Shaft, the lower Conglomerate attains its greatest thickness of 12.8m at 5600S on the 2800 Level on the northern edge of the Kitwe barren Gap, which is associated with a basement high. Although thick, the conglomerate contains fewer than normal coarse clasts and these, float in an arenaceous and sometimes argillaceous matrix.

The conglomerate is generally, poorly sorted and polymictic, consisting of pebble to cobble-size, rounded to sub-angular clasts of granite gneiss, schist, quartzite, vein-quartz and feldspar set in a medium to coarse-grained arkosic to argillaceous or micaceous matrix. At Central Shaft on the 3030 Level at 600N, it is possible to divide the lower Conglomerate into lower and upper units. The lower unit is coarser grained (up to boulder size), less well sorted and more polymictic than the upper unit. The matrix of the lower unit is arkosic and coarse-grained whereas that of the upper unit is more argillaceous. A coarse arkose to grit with scattered pebbles or bounders occur at the base of the lower unit.

There is a rapid Transition from the lower Conglomerate into the overlying Footwall Sandstone Member, which is between 15m and 20m thick. Where the lower Conglomerate is absent, the contact between the Footwall sandstone and the underlying Basal Quartzite and Basal Sandstone member is sharp, but without any obvious angular unconformity. The Lower part of the footwall Sandstone consists of a rapidly changing sequence comprising dark, almost black, fine-grained quartzite and argillite with lenticular interbands of feldspathic quartzite and grit displaying feston cross-bedding and wash-out features. There is a transition into the upper part of the footwall Sandstone, which is a fairly massive feldspathic quartzite and grit containing numerous scattered sub angular fine pebbles of pink and white feldspars. Cross–bedded heavy mineral layers, mainly containing hematite are present at intervals; Anhydrite is common in the lower part of the Footwall Sandstone especially where it overlies the basement complex directly, (Garlick, 1969). At 1500S on the 2880 level at South Ore body, an anhydrite seam up to 3m thick at the base of the Footwall Sandstone contains randomly oriented tabular fragments of schist derived from the immediately underlying basement. On the fringes of the barren gap area, the footwall Sandstone grades laterally into massive to schistose, white, pure to tremolitic or talcose dolomite, with thin pale grey dolomitic argillite interbands. The transition from the typical Footwall Sandstone lithologies to dolomite is on a bed-by- bed basis, beginning at the top of the member.

The top most member of the Footwall formation is the Footwall conglomerate, which has a maximum thickness of four meters and is very rarely present at South orebody but absent at Mindola Open Pit. It varies from a pebble conglomerate to a cross-bedded fine pebbly grit, which is polymictic, but schist, vein-quartz and feldspar predominate. The conglomerate tends to be coarser and more sorted at Central Shaft than at Mindola Shaft. The matrix is predominately arenaceous at Central Shaft but kaolinitic and argillaceous at Mindola Shaft. The bottom contact is gradational with the underlying arenite of the Footwall Sandstone, and probably represents a break in sedimentation.

d) The Ore Formation

The ore formation extends from the top of the Footwall Conglomerate to the base of the Hanging wall Quartzite. Two distinct faces are recognized: a South Ore body type typified by a black carbonaceous shale and a Mindola type comprising an interbanded sequence of argillite and thinly banded to laminated dolomite and argillite. The lateral change between the two faces is fairly rapid and can be traced obliquely through the Central Shaft area at about the zero position. There is no angular unconformity at the base of the formation but this contact is frequently the site for displacement, particularly at South Ore body Shaft, due to competency differences between the Footwall and Ore Formations. The upper contact is gradational, (Geological Guide to Nkana, 2005).

e) The Hanging wall Formation

The hanging wall Formation consists of four members namely the Hanging Quartzite member, Near Water Sediments, Upper Quartzite and the Dolomite Argillite Sequence.

f) The Hanging wall Quartzite Member

The hanging wall Quartzite member at the base is not always developed and varies in composition from a massive to laminated, medium grained pure quartzite to an interbanded and cross-bedded sequence of arkose, greywacke and anhydrite-rich dolomite. The base and top of the member is frequently dolomitic and often very pyretic. The upper and lower contacts are gradational. It has a maximum thickness of about five meters.

The Zambian Copper belt lies at the south-eastern end of the 800 km long Lufilian fold belt. It comprises deformed rocks of the late Precambrian Katanga system draped around the flanks of a major northwest trending late-tectonic structural feature, the Kafue Anticline. Various elements of the basement complex are exposed in the core of this Anticline. As a result of the Lufilian orogeny, the Katanga rocks have been thrown into a series of long narrow enechelon folds and late tectonic dome and basin like structures. The Chambishi-Nkana basin lies about midway on the southern flank of the Kafue Anticline and is elongated in a northwesterly direction. The Nkana Mining Area covers the northwesterly plunging Nkana syncline, which forms a south-eastward prolongation to the Chambishi-Nkana Basin. The regional tectonic framework is described more fully by Garlick, (Garlick, 1969). The predominant mega-structure of the mining area is the northwest plunging Nkana syncline.

This structure is asymmetric, with a curved axial plane inclined steeply to the northwest in the trough of the syncline but upright to westward dipping at higher structural levels. A complex occupies the axial zone of the syncline but this folding dies out on its flanks. As a result of the northwesterly plunge of the Nkana syncline, the present land surface represents a horizontal section through progressively higher structural levels of the syncline from southeast to northwest. The Area J mineralisation is hosted within an overturned isoclinal synform. Malachite oxide mineralisation is found between toe bounding faults, 26-68 m from surface. The whole oxide zone is enriched.

2.3.2.4. Aquifer Zones

The Near Water Sediments member consists of colour-laminated, greenish to purplish argillites and dolomitic argillites with lenses, stringers and interbands of buff feldspathic grit, carbonate-cemented sandstone, dolom ite and anhydrite. The lenses, stringers and interbands range in thickness from less than one centimetre up to three meters but are all laterally discontinuous. In the Central and Mindola Shaft areas one of more persistent interbands of well- banded cherty quartzite, one to three meters thick, is recognized as a separate unit, the middle Quartzite. It is difficult to estimate an average thickness for the Near Water Sediments as a whole as this member appears to have been particularly incompetent during deformation, which has resulted in considerable transposition of the bedding by shearing stress. Leaching-out of the carbonates, anhydrite and kaolinised feldspar gives the sequence a strongly pitted and fissured property and the member is consequently an important aquifer. There is an abrupt contact with the overlying Upper Quartzite member, which usually has a one meter thick dolomite band at its base. This dolomite band varies from a massive pink rock to a micaceous dolomite with cherty nodules, (Garlick, 1969).

2.3.2.5. Hydrogeology

The presence of water at Area J open pit is believed to be found in the Basement Complex formation contact and in the Sandstones immediately overlying the lower Conglomerate. This water appears to be stratigraphically controlled, and is randomly distributed underground. It is readily drained by means of horizontal holes drilled from the main development ends for South Ore body shaft.

2.3.3 Geotechnical Data

Good blast designs are generated after putting into consideration all aspects of rock formations surrounding the operations. One of the most important ones is that of geotechnical data. This information plays an important role in making sure that explosives used to break rock are of the correct strength and specification in order to overcome seismic strength of rock thereby fragmenting it. This section looks into geotechnical information obtained from Area J open pit.

2.3.3.1 Core Logging Data

In order to collect geotechnical data, core logging is required. This is the process by which all information of rock is analyzed and hence rock classification is obtained. For Area j open pit, a geotechnical study was conducted by African Mining Consultants (AMC) in 2010 and from it, a total of 271m diamond drilled cores from four boreholes. This process was conducted in order to collect the following rock specifications as below:

• Composition of the rock material include rock name, grain size, texture and fabric color.
• Condition of the rock material such as its strength as indicated in Table 21.
• Rock mass properties that include structure of rock, defects / characteristics such as rock type, orientation spacing, RQD, shape, roughness, aperture and weathering.

Table 21: Field estimates of Uniaxial Compressive Strength (Hoek , 1980)

Grade	Term	Uniaxial Compressive Strength (MPa)	Point load Index (KN)	Point load index (MPa)	Field Estimate of Strength	Examples
R6	Extremely Strong	>250	>62.5	> 0	Can only be chipped with a geological hammer	Fresh Basalt, chest, diabase, gneise, granite, quartzite
R5	Very Strong	100-250	25-62.5	4-9	Requires many blows of the geological hammer in order to fracture it	Amphibolite, Sanstone, Basalt, Gabbro, gneiss, grandiolite, limestone, marble, rhyolite,
R4	Strong	50-100	12.5-25	2-4	Requires more than one blow of a geological hammer to fracture it	Limestone, marble, phylite, sandstone, schist, shale
R3	Medium	12.5-50	6.2-12.5	1-2	Can not be scratched or pealed with a pocket knife, can be fractured by a single blow from a geological hammer	Claystone, coal, concrete, schist, shale, siltstone
R2	Weak	5-12.5	1.3-6.2	0.2-1	Can be pealled with a pocket knife, with difficulty, shallow indentations made from firm blows from geological hammer	Chalk, rock salt, potash
R1	Very weak	1.25-5	Too weak to index		Crumble under firm blows with point of a geological hammer, can be pealed by a pocket knife	Highly weathered or altered rock
R0	Extremely weak	<1.25	Too weak to index test		Indented by thumbnail	Stiff fault gorge

2.3.3.2 Geotechnical Environment

Geotechnical information collected from the core logging data was used to define the general geotechnical environment within and around the Area J Open pit. The high-wall that faces east as mining progresses in a north to south direction had previously been affected by mined out areas and such is expected to occur due to subsidence. Geotechnical design and analyses were carried out to the planned pit depth of 75m and a hypothetical extended depth to 125m with a proposed 18m thick barrier/crown pillar below the pit limit.

The major geotechnical input parameters include: uniaxial compressive strength (UCS), Geological Strength Index (GSI) and mi; values were varied for each analysis according to values found in each geotechnical domain. Geotechnical data collected were fully evaluated before arriving at the domain values for the Pit, (African Mining Consultants, 2011).

Domains

Geotechnical domains have been defined across Area J Open pit deposit based on the data obtained from the core logs. Geotechnical domains are used to divide the rock mass into volumes that are of the same ground behavior in three dimensions. It is from these domains that strength of rock and its behavior when blasted is understood. A summary of the domains is shown in Table 22.

Table 22: Area J Geotechnical Domains (African Mining Consultants, 2011)

Domain	Rock type
1	Saprock
2	Shale
3	Sandstone
4	Conglomerate

From Table 22, the Domains were further explained in more details as shown in Table 23 and the subsequent explanation.

Table 23: Strength of rock with different Domains (African Mining Consultants, 2011)

Domain	Rock type	Est. Rock Strength (Mpa)	Weathering Grade	GSI Value	Rock Shear Strength Parameter	
					Cohesion (Mpa)	Aangle of internal friction (Deg)
1	Saprolite	<5	Highly	N/A	0.087	28
2	Shale	53	Highly	32	0.19	30
3	Sandstone	51	Mod to Highly	31	0.165	29
4	Conglomerate	51	Mod to Highly	31	0.178	30

Domain 1

Domain I is made up of Saprolite material and is mainly defined by highly weathered, fine to medium grained material and lateritic soils extending vertically from surface to a depth of about 11m. The shear parameter for saprolite is shown in Table 23.

Domain 2

This domain consists of highly weathered, weak and fine grained shale material and extends to a vertical depth of 11 m to 31 m and is immediately below the Shale formation. See Table 23.

Domain 3

Domain 3 comprises of Sandstone rock formation that is generally weak to fair, moderately to highly weathered, medium to coarse grained and are in close proximity to shale. See Table 23 for the shear strength. The sandstone rock formation extends from 31 m to 76m.

Domain 4

As can be seen in Table 23, conglomerates form domain 4. This domain is generally moderately to highly weathered, weak to slightly fair due to its proximity to surface.

2.3.3.3 Rock Mass Properties

Estimates of the fundamental strength characteristics for the rock mass are required for the determination of blasting parameters and stability of both open pit and underground excavations. A technique developed by Hoek has been used for the conversion of the basic rock mass parameters determined from geotechnical mapping and core· logging to develop

Mohr-Coulomb shear strength parameters (cohesion and frictional angle) and Hoek Brown failure criteria.

The basic rock mass parameters required for the development of these strength parameters are Uniaxial Compressive Strength (UCS), Hoek-Brown intact constant m" and the Geological Strength Index (GSI)

1. Uniaxial Compressive Strength

The UCS of the rock provides an indication of the strength of the rock. It is measured by compressing core samples of the rock between two platens until the point at which the core sample fails. The pressure required to cause failure of the core sample is the uniaxial compressive strength of the rock, measured in units of pressure (MPa). These tests are carried out without confining the sides of the core sample and the loading force is applied only along the axis of the core. The reference to the unconfined uniaxial compressive strength (UCS) therefore applies only to this type of strength test. Other strength tests (triaxial confined tests) are possible, but are more difficult and costly to do, (Brown, 2004).

It is important to note that the UCS is only measured along the core sample axis. The strength measured in one direction may be higher than the strength measured in another direction. Usually the maximum uniaxial compressive rock strength is measured in core that is drilled perpendicular to layers in the rock whilst the minimum value is usually measured parallel to layers.

A summary of Uniaxial Compressive Strength values for the formations of Area J Open pit are presented in Table 24

Table 24: Summary of different UCS at Area J (African Mining Consultants, 2011)

	UCS (Mpa)		
Rock Units	**Min**	**Max**	**Mean**
Saprolite			<5
Shale	5	100	53
Sandstone	1.25	100	51
Conglomerate	1.25	100	51

Tensile Strength

The rock tensile strength is the pressure required to cause rock failure in tension (pulling the rock apart). This value is measured by tests such as the Brazilian Test. During the crack extension phase of blasting the rock fails in tension. The tensile strength therefore has an important bearing on blasting results.

The tensile strength is usually about 1/10th the UCS, (African Mining Consultants, 2011). For more accurate results, it is best to have samples tested.

2. Hoek-Brown mi Rock Constant

The Hoek-Brown constant mi, can be described as a parameter defining the fabric of the rock mass. This parameter is an important descriptor of the mode of rock failure *and* is used in the Hoek-Brown failure criterion when carrying out stability analyses. Higher mi, values represent coarse-grained rocks, while lower values indicate finer grained or highly foliated rocks. The Area J open pit "Mi" values have been estimated from a standard table of constants for similar rock types and are presented in the Table 25.

Table 25: Mi values of Area J rock units (African Mining Consultants, 2011)

Rock Formation	Mi Values
Saprock	4
Shale	6
Sandstone	17
Conglomerate	21

3. Geological Strength Index

Geological Strength Index (GSI) was introduced by Hoek in 1995, according to his earlier version of rock mass classification. GSI parameters estimate the reduction in rock mass strength taking into consideration the presence and condition of discontinuities. GSI values range from 0 to 100, with the higher value representing good undisturbed ground with few discontinuities. The lower values indicate weak, poorly interlocked or highly fractured rock masses. Table 26 gives a summary of GSI values representing the rock mass at Area J Open pit.

Table 26: GSI values of Area J rock formations (African Mining Consultants, 2011)

	GSI		
Rock Units	**Min**	**Max**	**Mean**
Saprolite			<5
Shale	20	56	32
Sandstone	20	42	31
Conglomerate	17	44	31

4. Rock Density

Rock density is the specific weight of a rock expressed in grams per cubic centimeter (gr/cc), (Olofsson, 1990). Rock density is a very important parameter in blasting because it has a bearing on blast fragmentation and heave results. This is because as density increases, the more the demand of explosives into the blast holes in order to break that particular rock.

Joints

Joints are planes within the rock mass which separate solid rock masses from each other, (Olofsson, 1990). If a rock has closely spaced joints, the jointing will control the fragmentation regardless of the rock strength. This is a fundamental rule. Each fragment in the jointed rock is defined by flat surfaces. These surfaces are the original joint planes. For such a rock, the explosive did very little work in creating new fractures.

Layers

Layers are similar to joints with a major difference of the orientation. Layers are formed in horizontal separations. The same rule applies if the rock is closely layered (example shale's). The joint spacing or layering will determine the outcome during blasting operations. If the rock formation is too layered, a lot of explosives are lost into the cracks. The opposite is also true when there is completely no joints, it becomes very difficult to break rock.

Massive rock

This is solid rock that has no joints within the formation. Rock that has no joints in it will be very difficult to blast even if it has a low strength. For example, semi-weathered sandstone has a low strength (UCS), but if it is massive (no layers or joints) it will be very difficult to fragment effectively because it yields and absorbs explosive energy rather than breaking. Therefore weaker massive rock can be more difficult to fragment than stronger massive rock because stronger massive rock will tend to yield less and shatter more easily.

2.3.4 Explosives properties

Explosives are chemical mixtures that release gasses at high velocity causing very high pressures. Some of the explosives types are discussed in the following sections.

1. **Dynamite**

In 1867, Alfred Nobel of Sweden discovered how to create dynamite. Most dynamites are nitroglycerin based, (Olofsson, 1990). Being the most sensitive of all explosives used; dynamite is more susceptible to accidental initiation. There are two major subclasses of dynamite, Granular dynamite and gelatin dynamite. Granular dynamite is a compound which uses nitroglycerin as its explosive base. Gelatin dynamite uses a mixture of nitroglycerin and nitrocellulose. This produces a waterproof compound

Blasting agents

Blasting agents are defined as chemical or mixtures that meet the description criteria of non-initiating but are capable of exploding when ignited by another high explosive charge or primer. The following are some of the blasting agents, (Olofsson, 1990).

2. **ANFO**

This is a powder of explosives consisting of Ammonium Nitrate Fuel Oil (ANFO). The ammonium nitrate compositions can be applied in “free running” form, i.e., poured into the blast hole. This provides effective coupling between explosive and blast hole wall. It has relatively safe handling properties and low cost, (Robert, 1981).

The density of ANFO ranges between 0.8 g/cm^3 to 0.85 g/cm^3, (Olofsson, 1990). It therefore, has a disadvantage of being used only in dry conditions since its density is less than that of water thereby making it non water resistant. It has a VOD of about 4500 m/s and contains 94% of Ammonium nitrate and 6% of fuel, (Bulk Mining Explosive, 2009).

3. **Heavy ANFO**

Heavy ANFO is composed of up to 45% to 50% Ammonium nitrate emulsion mixed with small porous sphere (Prill) used to manufacture ANFO in order to increase the bulk density of ANFO. The main advantage of heavy ANFO is that they can be mixed at the blast hole and quickly loaded into a hole. The ratio of the amount of slurry mixed with the ANFO can be changed to offer either a higher energy load or a load which is water resistant. The cost of heavy ANFO rises with increasing amount of slurry. These have an advantage over cartridge blasting agents because they are 100% coupling which fills the entire blast hole hence utilizing the most of the energy to break rock unlike with cartridges where some energy is wasted due to the coupling effect, (Mishra, 2009).

4. **Slurries**

Slurry explosives are also known as water gels. They are made up of ammonium nitrate partly in an aqueous solution containing 10% to 30% of water, sensitized with a fuel, thickened and cross-linked to a gelatinous consistency. Therefore, they are water-resistant and are able to be poured into a waterlogged blast hole to displace the water and form a dense, effectively coupled charge, (Olofsson, 1990).

They can be classified as blasting agents or explosives depending on the composition. Slurry blasting agents contain non explosive sensitizers or fuels such as carbon, sulfur, or aluminum. These blasting agents are not cap sensitive. On the other hand slurry explosives contain cap-sensitive ingredients such as TNT and the mixture itself may be cap sensitive. The slurries are thickened with a gum, such as guar gum. This gives them very good water resistance. “Slurry boosting” is practiced when slurry and a dry blasting agent are used in the same borehole. Most of the charge will come from the dry blasting agent. Boosters placed at regular intervals may improve fragmentation. The disadvantages of slurries include higher cost, unreliable performance, and deterioration with prolonged storage.

Slurries are characterized by high density (1.1 to 1.3 g/cm^3) and bulk strength, good oxygen balance, confinement and coupling within the borehole. They have a VOD ranging from 4115 m/sec to 6096 m/s, (Roberts, 1981).

5. **Emulsion**

An emulsion is a water resistant explosive material containing substantial amounts of oxidizers, often ammonium nitrate, dissolved in water and forming droplets, surrounded by fuel oil. The droplets of the oxidizer solution are surrounded by a thin layer of oil and are stabilized by emulsifiers. In order to achieve more sensitivity within the emulsion, voids are added by mixing the emulsion with carbonaceous fuels or certain organic compounds. These voids may include small nitrogen bubbles or micro-spheres made out of glass. Sensitivity of an emulsion decreases as the density increases. To adjust the density and strength of an emulsion, dry products are used and some examples are powdered aluminum and gasifying agents. It is therefore necessary to work above the minimum diameter of an explosive propagation with stable detonation known as critical diameter and hence use powerful initiators, (Olofsson, 1990). If the emulsion is not cap sensitive it is considered as a blasting agent. Emulsions have higher VOD ranging from 4400m/s to 5650 m/s, reliable performance, excellent resistance to water because they have a higher density ranging from 1.14g/cm^3 to 1.5g/cm^3 which is greater than that of water, and relative insensitivity to temperature changes. The direct cost of an emulsion explosive is higher but this is offset by time saved in loading and a reduction in nitrate content of broken muck, (Bulk Mining Explosives, 2009). Some other advantages of using emulsions in rock blasting include: a lower cost, excellent water resistance, high detonation velocities, and it's very safe to handle and manufacture.

Emulsion is an explosive that contains 6% Fuel, a percentage of Ammonium nitrate depending on the grade of emulsion and the remainder consists of Water, Stabilising and Gassing Agents, (Bulk Mining Explosives, 2009). Emulsion explosives work by releasing energy when an oxidizer reacts with a fuel to form a gas:

$$Oxidizer + Fuel \Rightarrow Gas \qquad (40)$$

The gas takes up much more volume than the oxidizer or fuel, and this is how work is done on the rock.

$$\begin{aligned} &AmmoniumNitrate + Fuel \Rightarrow Gas + WaterVapour \\ &3N_2H_4O_3 + CH_2 \Rightarrow 3N_2 + CO_2 + 7H_2O \end{aligned} \tag{41}$$

When an explosive reacts, the Oxygen in the oxidizer combines with the carbon in the fuel to form carbon-dioxide gas (CO_2). The oxygen also reacts with the hydrogen in the fuel to form water vapour (H_2O). Nitrogen gas is also released.

a) Oxygen Balance

For maximum energy, there must be the correct quantity of fuel (C and H molecules) to combine completely with the Oxygen in the reaction.

- **Oxygen Negative**

If the mixture is oxygen positive (too little fuel) the excess oxygen combines with nitrogen to form orange Nitrous fumes. Nitrous fumes are very poisonous.

- **Oxygen Positive**

If the mixture is oxygen negative (too much fuel) carbon monoxide (CO) forms instead of carbon dioxide (CO_2).

b) Explosive Density

The density of an explosive is a measure of how much potential energy in the form of hot, high-pressure gas is in a given volume, (Olinger, 2005). Density is normally expressed as g/cm^3. The density of emulsion explosives and Heavy ANFO have high emulsion content that can be adjusted by increasing or reducing the number of small sensitizing bubbles. Explosive density is very important because;

•It gives a general idea when selecting the explosive type because of the energy content required to break rock within a given volume.

•It enables us to know which explosive sinks or floats in waterlogged blast holes.

•It enables us to calculate the linear charge density.

•It also enables adjustment of small sensitizing bubbles.

c) Sensitivity of explosive

Sensitivity of Explosives is the explosive susceptibility to detonation upon receiving an external impulse such as impact, flame or friction. Commercial explosive is sensitized by small air voids.

d) Cap sensitivity

This is the measure of the easy of initiation of an explosive when subjected to a shockwave from a detonator. Blasting agents have very low cap sensitivity and are relatively safe to handle, whereas most high explosives have a high sensitivity and can be detonated with a small initiator, (Hopler, 1998).

e) Gap sensitivity

This the ability of an explosive charge to detonate another with an air gap between the two. Gap sensitivity must be controlled in industrial explosives so that one borehole's charge cannot detonate the other charge prematurely.

f) Sympathetic Detonation

Sympathetic detonation occurs when the shockwave from a nearby detonation is sufficient to initiate an explosive without a booster. This normally occurs when explosives are too sensitive or when timing problems occur in a blast.

g) Dynamic Desensitization

Explosive may be desensitized by the shockwave from a nearby detonation. This happens when the air bubbles are compressed and the void volume is temporarily reduced below critical levels by the shock wave. At this time a booster will be unable to initiate the explosive. The events that can desensitize explosives dynamically include Shock wave from detonating cord down lines and Detonations of explosive charges that are too close.

h) Relative Weight Strength (RWS)

RWS is the energy in 1 kg of explosive compared to the energy of 1 kg of ANFO. It enables us to quickly check which explosive will give more energy.

i) Relative Bulk Strength (RBS)

Relative Bulk Strength is the energy in 1 litre of explosive compared to the energy of 1 litre of ANFO. It gives information on how much energy of a given explosive in a hole when compared with ANFO.

j) Water Resistance

This is the ability of an explosive to withstand exposure to water without losing sensitivity or ore efficiency. Water resistance is minimal in ANFO and high in slurries and emulsions.

Fume class represents a measure of the amount of toxic fumes produced by a blast. Fume class is important if the explosive is to be used in underground mines, (Atlas Powder Company, 1987).

k) Detonation Pressure

This is the pressure exerted as the detonation wave travels through an area. High detonation pressures are required in boosters used to initiate blasting agents, (Hopler, 1998).

l) Velocity of Detonation (VOD)

VOD is the velocity at which detonation wave travels through the explosive column. May be seared confined or unconfined. It does not tell us how much energy is delivered.

Research conducted by Changshan Sun (2013) confirmed that the old assumption that stated that detonation velocity must equal that of rock seismic shock wave in order to break rock.

Explosive with high VOD gives high initial borehole pressures that are useful for creating fresh cracks in the rock around a blast hole (Changshau, 2013). This is effective in hard rock (dolerites, granites) that has few natural planes of weakness.

Explosive with low VOD delivers its energy more slowly. Initial borehole pressures are lower, but the pressure is sustained for a longer period of time. This is useful for softer rocks

(Shales, layered Sandstone) that are layered or closely jointed and are fragmented mainly by heave.

General Rule:

High VODs = higher shock energy - hard rock.

Low VODs = higher gas energy - softer rocks.

i. Useful energy

This is the heave and shock energy used for breaking the rock and displacing it.

ii. Wasted energy

Wasted energy could results into heat, light, noise, fly rock and vibration energy. Wasted energy is usually undesirable for the environment, and blast designs should ensure that there is minimal wasted energy.

m) Linear charge density (Q_c)

This is the weight of explosives that can be loaded into a blast hole per unit length of blast hole. This is calculated using Equation 42.

$$Q_c = \frac{D^2 * \rho}{1273} \tag{42}$$

Where

ρ is density (g/cm^3),

D is explosive column diameter in mm.

Q_c is given in kg/m. Knowledge of loading density is required for blast-design calculations, (SME, 2002).

n) Charge Length (L_c)

This is the total length of an explosive charge in a blast hole, (SME, 2002). It is given by Equation 43.

$$L_c = H + J - S \quad (43)$$

Where;

H is the bench height (m)

J is sub-drill (m)

S is stemming (m)

o) Charge per Hole (Q)

This is the amount of explosive pumped into a blast hole. The units are given as kg/blast hole. It can be calculated using Equation 44;

$$Q = Q_c * L_c \quad (44)$$

Where;

Q is the total charge per blast hole (kg/hole)

Q_c is the specific charge per meter, (kg/m)

L_c is the charge length in the blast hole, (m)

p) Powder Factor (P. F)

The powder factor is generally defined as the mass of explosive used to break a cubic meter of rock. This is calculated as shown in Equation 45:

$$P.F = \frac{Q}{B * S * H} \quad (45)$$

Where;

P.F is the powder factor in kg/m^3

B is the burden (m)

S is the spacing (m)

H is the bench height (m)

Once the design parameters are established, the powder factor PF is computed as the quantity of explosives used divided by the volume yield of material blasted (SME, 2002).

i) Effects of higher powder factors

Higher powder factors result in fine fragmentation and are required for small capacity removal equipment such as front-end loader, (SME, 2002). However, in some operations where coarse material is required, there may be need to ensure that powder factor is not too high in order to avoid over blasting.

ii) Effects of low powder factors

Low powder factors result in coarser fragmentation and are typically used for rock removal using draglines and large shovels. Powder factor is often reported as the rock yield in Kg/m^3.Powder factor does not tell us how much energy there is, it only tells the mass of explosive used. Powder factor is only useful for stock control and cost calculations. It does not have any value in comparing explosives and drill patterns when designing blasts (Bulk Mining Explosives handbook, 2009).

q) Energy Factor

The energy factor is a measure of explosive energy and its influence on the rock. This values provides a more realistic value to use for blast designs especially when comparing different explosive types. The energy factor is sometimes referred to as the ANFO equivalent powder factor and is calculated as shown in Equation 46.

$$EnergyFactor = P.F * RWS \qquad (46)$$

Where;

P.F is the powder factor,

RWS is the explosive relative weight strength

r) Explosive Selection Criteria

Selection of the type of explosives plays an important role in blast design as well as the outcome from the blast. An explosive has many characteristics that need to be analyzed in making this decision. The following factors were suggested by Dick et al (1983).

- Critical diameter
- Minimum primer weight
- Explosive density
- Relative Weight Strength (RWS)
- Relative Bulk Strength (RBS)
- Gap sensitivity
- Water resistance and Loading Procedure
- Shelf life, and
- Overall drilling and blasting economics and others.

Other things the technician must consider are charge diameter, characteristics of the rock to be blasted, volume of the rock to be blasted, environmental impact and supply problems.

2.3.5 Initiation Patterns and Delay timing for surface blasting

Blast delay timing is a critical and important blast design parameter which has direct impact on fragmentation, muckpile characteristics, flyrock direction, back break and ground vibrations (Jimeno et al., 1995). Blast hole delay should permit the succession of the following events:

1. Propagation of the compression and tensile waves from the blast hole to the free face;
2. Readjustment of the initial field of tensions, due to the presence of radial cracks and the effect of the reflection of the shock wave on the free face; and

3. Acceleration of the fragmented rock by the action of gases up to a velocity that assures an adequate horizontal displacement.

Rock mass movement depends on the material response in conjunction with the stress and gas pressure stimulus generated by the explosive, (Chiappetta, 2004).

Delays are incorporated into the blast design using electric, electronic or nonelectric caps or delay connectors with detonating cord, (Winzer, 1978). The delay patterns used in design will determine the sequence of hole or deck initiations, thereby, dictate the overall direction of blasted rock movement and resulting fragmentation. Depending on the Spacing / Burden (S/B) ratio, the actual timing (in milliseconds) between detonating charges will determine muck pile displacement height and distance from the bench. Figure 29 shows variations of timing patterns used for surface blasting. The effective spacing (ES) is the distance between holes in a row defined by adjacent time delays (such as delays by rows)such as is shown in Figure 29 (a) between holes. Effective burden (EB) is the distance in the direction of resultant rock mass movement and is represented by perpendicular to the free face as shown in figure 29 (b).

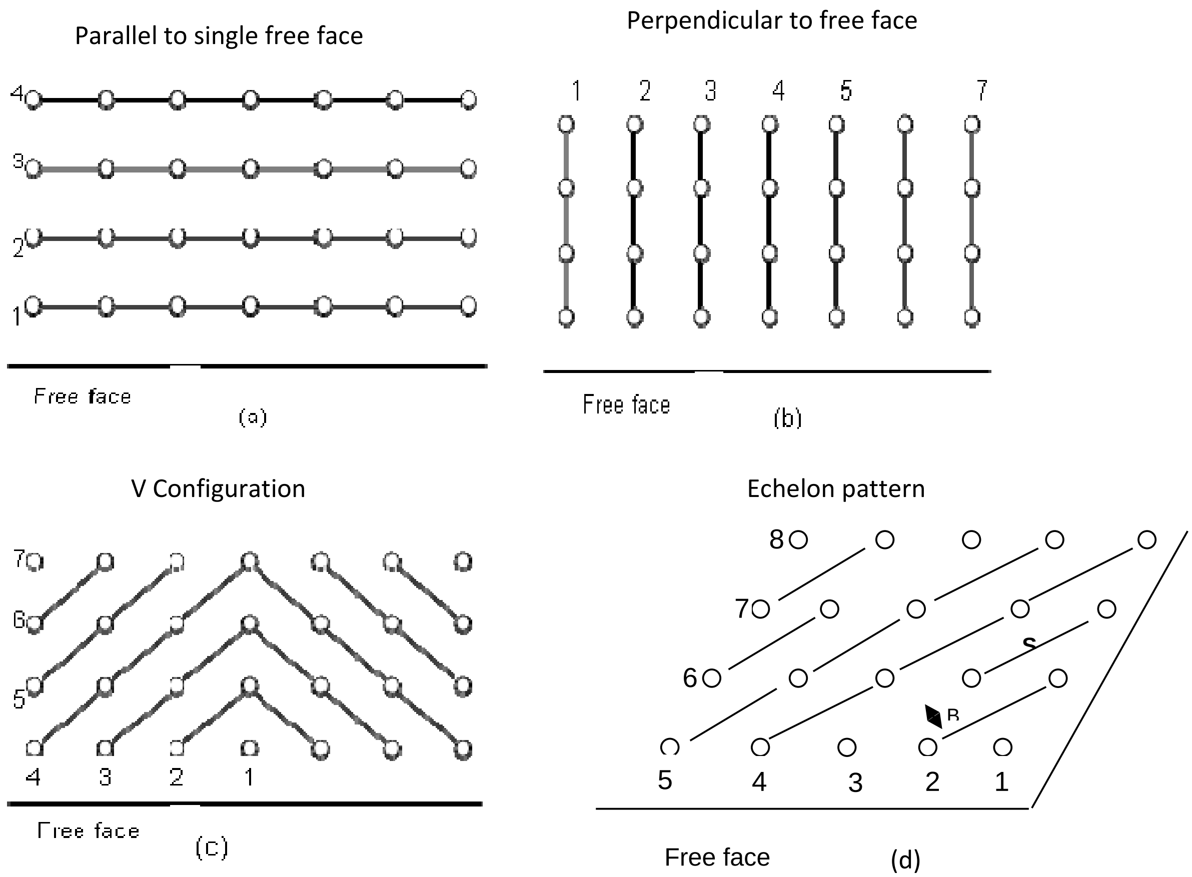


Figure 29: a, b, c, d shows the initiation patterns (SME, 1992)

The V and echelon (diagonal) patterns as shown in Figure 29 (c and d) are used when rock placement is restricted. Designs using two free faces usually provide improved fragmentation and throw control over those using a single face.

2.3.5.1 Delay Timings

Production-scale, multiple-row blasting has resulted in recommended timing to improve fragmentation:

Andrews in 1981 suggested that:

$$\text{Delay between rows} = (6.56 \text{ to } 49.22\text{ms/m}) * B_e \tag{47}$$

$$\text{Delay between holes} = (3.28 \text{ to } 16.4 \text{ ms/m}) * B_e \quad (48)$$

Roberts, (1981) suggested the following:

$$\text{Minimum delay time} \geq (1.5 - 3.0 \text{ ms/m}) * \text{Burden (m)} \quad (49)$$

$$\text{Maximum delay time} \leq (4.5 - 7.5 \text{ ms/m}) * \text{Burden (m)} \quad (50)$$

Onederra and Esen in 2008 wrote: The selection of appropriate delay times (inter-hole and inter-row) involves the implementation of preliminary design guidelines or more quantitative methods based on knowledge of the minimum response time (*Tmin*). Preliminary design guidelines are empirical in nature and include the application of simple rules such as those documented by Jimeno et al. (1995).

Recommendations by Bauer suggests delay times of 5 to 7 ms /m of a blast with holes of 38 to 311 mm diameter. On the other hand, Bergman suggested 3 to 6 ms/m and 4 to 8 ms/m, while Konya & Walter (1990) proposed delay times of 6 to 7 ms/m, 4 to 5 ms/m and 3 to 4 ms/m of burden for low, medium and high strength rocks respectively, (Konya and Walter, 1990).

Chiappetta in 1998 discussed a more objective methodology that can be adopted from knowledge of the minimum response time to determine optimum time interval. For example, the delay time between holes in a row could be less or equal to the minimum response time. This was done in an effort to encourage positive interaction for improved breakage and fragmentation, (Chipappetta, 1998). However the delay time between rows should be in the order from 1.5 to 3.0 times the minimum response time in order to maximize material displacement and hence create a loose muckpile.

Historical information has revealed that there are limited methods that are available to show the minimum response time for rock breakage. These include direct measurements with the application of high-speed video analysis techniques and the use of mechanistic and numerical modelling tools. High speed video analysis may be site specific, and that requires a lot of time as well as possess some difficult on implementation due to cost.

This has led to the development of an empirical approach that allows engineers to make preliminary estimates of the time at which rock mass movement is expected to occur, given a particular geometry and explosive/rock mass combination, the following is a model that is used to estimate the minimum response time. This model was arrived at from a detail compilation and analysis of several case studies in which comprehensive blast monitoring was done. A detailed examination of the research conducted by Onederra showed that minimum response time (T_{min}) is strongly dependent on the blast geometry (i.e. burden and hole diameter), stiffness of the rock mass and explosive/rock interaction, (Onederra and Esen, 2008), see Equation 51 and all the parameters as discussed earlier. This confirmed the suggestion made previously the in the blasting literature of Chiappetta in 1998.

$$T_{\min} = \left(K_{mass} ERI\right) \left[a\left(\frac{B}{d} \left(\frac{1}{K_{mass} ERI} \right) \right)^{b} \right] \qquad (51)$$

Where:

T_{min} is the minimum response time assumed to occur at the center of the explosive charge (ms)
B is the burden (m)
d the hole diameter (m)
a and b are site constants
ERI is an explosive rock interaction term. This term is based on the explosive performance term introduced by Bergmann, (Bergmann, 1983).

K_{mass} that can be calculated using Equation 52;

$$K_{mass} = \frac{Ed}{(1 + vd)} \qquad (52)$$

Where;

K_{mass} is the rock mass stiffness calculated in GPa
Ed is the rock mass dynamic Young's modulus
Poisson's ratio (vd).

$$ERI = (0.36 + \rho_e)\left[\frac{D^2}{\left(1 + \frac{D^2}{v_p^2} - \frac{D}{v_p}\right)}\right]\left(\frac{D}{D_{cj}}\right)\rho_e \quad (53)$$

Where:

ERI is the explosive-rock interaction term;
ρ e is the density of the explosive (g/cm^3);
D is the actual (non-ideal) detonation velocity (km/s);
v_p is the P–wave velocity of the intact rock (km/s) and
D_{CJ} is the Chapman-Jouguet detonation velocity (km/s).

In 2008, Onederra further proposed a model that could also be used to test for its ability to realistically predict changes of minimum response time caused by changes in geometry (i.e. namely burden distance). From the burden distance, the ratio of delay between holes to the burden distance is very important. The following was observed;
Timing ratios are found to vary over a wide range depending on the geology, scale and explosives used. Weak fractured rock formations require high delay ratios compared to competent formation, (Onederra, 2008).

If delay time is too long, this results into cut-off of surface delays whenever non electronic detonators are used. If delay time is too short, then cratering type of blast are produced with the movement of rows of burden being restricted hence affecting fragmentation. Furthermore, there is an increase in vibration and flyrock as well as excessive back break, (Cunningham, 2000). These are not desired during blasting operations.

2.4. Types of blasting

There are mainly three types of blasting methods, these include Cast blasting, concrete and control blasting, (Olofsson, 1990).

2.4.1. Cast Blasting

The purpose of cast blasting is to reduce the percentage of blasted material that has to be handled mechanically. The method involves using a controlled throw or cast of the material being blasted. This method is often used in coal mining to remove the soil overburden covering the coal, (Olofsson, 1990).

It is important to note that drilling and blasting costs increase with the introduction of cast blasting, but full economic benefit in terms of overall mining costs may only be visible in the long term.

Design Parameters and Explosive energy: An increase in powder factor will result in material displacement to the following extent:

Table 27 Powder Factor and Resultant Cast (%), (Olofsson, 1990).

Powder factor Range (kg/m^3)	**Material Displacement Cast (%)**
0.25 – 0.4	0 to 12
0.45 – 0.6	20 to 35
0.65 – 0.8	42 - 46
0.85 – 1	47 – 50

These values are only a guide and will depend on a number of factors such as the material being blasted, hole size and explosive type.

High wall height: in operations where the high wall is shorter than 18 metres, there is need for more energy to obtain the same cast percentage as large as the high walls. The cast distance can be calculated using Equation 54.

$$R = V_o Cos\theta \left[\frac{V_o Sin\theta + \sqrt{(V_o Sin\theta)^2 + 2gH}}{g} \right] \quad (54)$$

Where

R = cast distance (m)

V_o = initial burden velocity (m/sec)

θ = ejection angle (°)

H = height of burden above floor level (m)

g = acceleration due to gravity (9.8 m/s^2)

In terms of effective cast, the closer the high wall height to bench width ratio, (1:1), the better the results.

Blast hole diameter and inclination: Chiappetta (2004) recommended that a borehole diameter which corresponds to 10.5mm for every meter of overburden may be used successfully. He also stated that angled holes add a vertical component to the blast which results in better throw with a theoretical maximum at 45°.

Burden to Spacing Ratio: There are various opinions on this matter, but a burden to spacing ratio of 1:1.2 up to 1:1.5 seems to be generally accepted. A staggered pattern should be used in order to maintain an even distribution of energy, (Chiappetta et al, 1983).

Timing: For optimum cast results, all holes in a row must be fired instantaneously. To assure a systematic release of energy from one row to another, the effective delay between rows should be incrementally increased from the front row towards the back row.

Detonator Accuracy: To prohibit differential burden displacements and to eliminate detonator scatter, it is important that detonators must fire at the exact intended time. Detonator scatter may result in out of sequence firing. This in turn may give rise to vertical displacement of material, due to increase in burden and spacing between holes.

2.4.2. Concrete Blasting

It is quiet challenging to determine the rules that are applicable to conduct blasting of concrete due to the considerable difference in strength of different materials. This is because of the varying geometrical form of the concrete its degree of reinforcement hence the need to consider such factors while planning this blasts, (Olofsson, 1990).

Table 28 is a guide used to assist a blaster when attempting concrete blasting:

Table 28-The specific load to blast concrete, (Olofsson, 1990).

Material To Be Blasted	Specific Load (kg/m^3)	Spacing B*S (m)
Non-reinforced concrete poor quality	0.3	0. 7 – 0.8
Non-reinforced concrete	0. 35 – 0.4	0. 6 – 0.7
Surface-reinforced concrete	0.65	0.5 – 0.6
Heavily reinforced concrete	0.9	0. 5 – 0.6
Very heavily reinforced	1.8	0. 4 – 0.5

Where deep holes have to be loaded, the deck loading method should be used. The non-loaded section of the hole should be thoroughly filled and compacted with sand or similar material. Noise may be reduced by inserting the detonating caps into the hole before the last section is filled with sand. In the case of short holes, the detonator should be inserted into the bottom cartridge before filling the holes. In this way, bottom initiation is obtained which is a very effective method since the explosive is efficiently confined. The use of shock tube systems are very common these days, (Olofsson, 1990).

2.4.3. Close Proximity Blasting

The development of blasting techniques has made it possible to carry out advanced blasting operations close to existing structures. This practice has increased in the last decades due to newly discovered mineral reserves close to urban areas. Such a blasting technique is well known to be controlled.

In this blasting technique, ground vibrations and to certain extent air shock wave and flyrock constitute a threat to property and life, it is therefore necessary to control these hazards to avoid damage. This is easily achievable by ensuring that all the blasting parameters are well understood and followed. Furthermore, there is need to make sure that charges of explosives are well calculated and direction of throw is determined such that the nuisance is reduced.

It is usually ground vibrations that affect neighbouring structures but special attention has to be given to possible occurrence of flyrock and this happens to the main on-site fatalities and damages to equipment and houses.

2.4.3.1 Blasting methods

Blasting as earlier introduced is the breakage of material by detonation of industrial explosives emplaced into blast holes suitably located to the free surface. There are so many types of blasts conducted but in this research, two types were considered. These are conventional and controlled blasting techniques.

a) Conversional blasting

This is a type of blast commonly used in the open pit mining operations. This method of blasting basically utilizes bench blasting methods. However, non-electronic (NONEL) detonators are used hence areas blasted must have very little blasting concerns to the nearest neighbors. Different tie up patterns are used alongside the free face for effective fragmentation. Furthermore, detonators are initiated almost simultaneously and the cooperating charge per delay is not of great concern as the mine is situated far from the community.

b) Controlled blasting

This is the type of blasting method that requires extra caution when it comes to the type of explosives used and method. These methods require high precision and accurate detonation thereby eliminating the scattering effect that consequently increases on ground vibrations (Cunningham, 2003). Direction of throw, and wave propagation are very critical aspects of blasing when control is required. The timing and hole inclination also contribute to this technique in order to reduce on the adverse effects to the environment.The amount of explosives detonated per round also plays a very important role in achieveing this type of blast. It is therefore, only possible to achieve the above with the use of electronic detonators.

In order to control blasts at Area J open pit, Mopani utilized electronic detonators. These explosives as can be seen in Figure 30 comprise of Digi shot detonators, a tagger that is used to determine the hole ID of each detonator and giving them their positions in the blast, blasting box that is used to time and initiate the blast. However, it is not possible to conduct a blast unless a blasting safety key is insert into the blast box and the unique password is entered to fire the block.

Figure 30: Digi Shot explosives

CHAPTER 3

RESEARCH APPROACH

This chapter discusses research methods that were used in order to achieve the research objectives. The approaches used included;

3.1 Site Investigation.

This research was conducted at Area J Open Pit that lies 100m away from Wusakile community. In order to gather correct information and establish the feasibility of conducting this research, site investigations within Mopani and the nearby community was conducted. Some of the key parameters that were taken into consideration were distances from the blast to nearby housing units, condition of existing housing units, and the major geological discontinuities such as faults and joints.

3.1.1 Distance to the nearby housing units

The average distance from the pit limit to the housing units within Kalela Basic School is 100m. Using the conventional blasting methods, a safe blasting radius of 500m is recommended by Mines Safety Department to ensure safety of personnel and structures. A total of 450 houses were noted to be within a 500m radius, (Blasting Licence Training Manual, 1990). As can be clearly seen in Figure 31, within a safe blast radius of 500m is present a total of 450 housing units from the pit perimeter. The question then was why go ahead and proceed with such an operation so close to the community?

Figure 31: A total of 450 Housing units within 500m safe blast radius (Satellite image)

MCM discovered mineral wealth in close proximity to Wusakile community and hence engaged a consultant, African Explosives Limited (AEL) to conduct a feasibility study of conducting blasting operations safely within the community, (African Mining Consultants, 2009). Results showed that blasting operations by use of conventional methods would raise some public concerns from the community hence a new proposed control blasting method was suggested. This proposed method utilizes electronic detonators with delay interval of 50ms between holes and is known as controlled blasting. The following cost analysis was conducted to confirm the proposal:

- Controlled blasting methods would use Digi shot detonators and these would only increase blasting costs from US$0.16/ton to US$0.19/ton. That was because the drilling cost remained the same except for the detonators. In conventional blasting, NONEL detonators are used and these explosives cost $2/ unit whereas those of electronics' cost $20/unit. This consequently resulted in a net increase by US$0.03/tons of the total blasting cost.

- Conventional blasting produces better breakage as this method permits multiple holes going off simultaneous while utilizing the free face. However, non-electronic detonators have a high scattering effect and thus increase the vibration levels.

- A net increase of only US$ 405,000 resulting from the selection on explosives type was calculated for the total material of 13.5million tons to be mined at Area J open pit.

- Resettling the 450 housing units would require new land allocation and rebuilding of their houses in addition to compensation for loss of vegetation and business opportunities created or established over a long time. However only US$ 405,000 would then be shared among the residents of Wusakile community and this would mean each housing unit getting only US$900. This amount of money was not going to be sufficient to replace their current structures considering the price of land and building materials in Zambia today.

Therefore, the technique of Control blasting was a preferred option because it would benefit both the mining company and the residents of Wusakile community.

3.1.2 Conditions of housing units

The houses in Wusakile community are built from block work with cement and some with clay bricks. Most of these houses are quite old and require rehabilitation. During site investigation, about ten houses within Wusakile community as can be seen in Figure 32 were observed to have developed some cracks prior to Area J commencement. It was therefore necessary to monitor crack expansion while operating close to the community.

Figure 32: Cracks forming in some houses of Wusakile

A crack record card for capturing crack monitoring data is shown in Table 32. As can be seen, some houses had developed cracks as wide as 24mm before blasting operations commenced at Area J open pit, this clearly shows that there could be other factors that could result in crack formation other than blasting operations at the open pit.

Table 29: Crack monitoring report in Wusakile (JKL Associates, 2010)

DATE	PLOT NO.	DEVICE NO.	CRACK WIDTH BEFORE DEVICE INSTALLATION (mm)	CURRENT READING (mm)	Variance between Initial & Current Reading (mm)
30/11/2010	B1 - 1	1	5	0	Nil
30/11/2010	B1 - 4	2	9	0	Nil
30/11/2010	B1 - 6	3	5	0	Nil
30/11/2010	B1 - 17	4	4	0	Nil
30/11/2010	B1 - 21	5	6	0	Nil
30/11/2010	B1 - 25	6	24	0	Nil
30/11/2010	B1 - 31	7	7	0	Nil
30/11/2010	B1 - 70	8	6	0	Nil
30/11/2010	B1 - 71	9	4	0	Nil
30/11/2010	B1 - 85/B1 - 86	10	3	0	Nil

3.1.3 Major Geological Discontinuities

The geology of Nkana as already reviewed in literature review of Chapter 2, section 2.3.1.2 of this report. Information collected showed that most of the geology of Nkana is in the form of synclines and anticline formations. Such formation is forgiving because of change in material properties that consequently reduces vibration propagation as the distance increases.

3.2 Review of Geotechnical Data

This involved a detailed analysis of the geotechnical information for Area J open pit so as to determine strength of rock. African Mining Consultants (AMC) were subcontracted to conduct a study on geotechnical parameter at Area J open pit. AMC conducted the study of Area J open pit where a total of 271m of diamond drilled cores were geotechnical logged from four (04) bore holes drilled across the deposit. Their study revealed an average Uniaxil Compressive Stress (UCS) of rock to be in the following categories;

- Saprolite has less than 5 Mpa,
- Shale had 53Mpa,
- Sandstone had 51Mpa and
- Conglomerate has 51Mpa

With such formation, the pit would fairly be categorized to contain rock formations that ranged from soft to medium rock formations. Based on the Kuzram model of blasting, it was consequently designed that a low powder factor averaging 0.4 kg/m^3 could be used for rock breakage at the Area J open pit.

3.3 Controlled Drilling and Blasting Techniques

Controlled drilling and blasting could be summarized in the following step:

1. Designing the drilling plan for a particular blast, this included coming up with a planned depth of holes, burden and spacing on a particular rock formation, the actual position of the blast in relation to the positions of vibration and noise monitoring.
2. Implementation of the drilling plan by the mine captain. The plan contained catual hole depth to be drilled and this had to be verified a day before the blast. Using the principles of control blasting techniques, the charge per hole was calculated thereafter, the total tonnage of explosives were purchased and communicated to the explosives supplier.

3. On the same day, the blasting schedule was prepared and circulated to the community notifying them about MCM's plan to conduct a blast the following day. The schedule also indicated the affected areas and the actual time of the blast.

4. On the actual day of the blast, the blasting schedule was recirculated to the community while the bench was prepared for charging, see blasting procedure in the Appendix A.

5. Digishots detonators were prepared for Charging of holes. Note that for this type of blasting, electronic detonators were used. Connecting of the detonators and boosters was concluded and lowered in the ground thereafter, pouring of emulsion into the blast holes. This process was strictly monitored by explosives and blasting engineers. The holes were then allowed 20 minutes for the explosive gassing process. Thereafter, verification of stemming was conducted alongside tagging. Tagging is the process by which every detonator on the blast bench is given a hole identification number (ID) and hence included in that days blast and hence influencing the time of detonation (Rosmanith, 2003). This process is very important as it also confirms the status of each detonator and it is at this time that any failed detonator will be identified and replaced.

6. Stemming would begin upon approval from the personnel in charge of that day's blast. This process would be superseded with harnessing of the bench. Thereafter, verification of the holes would be done and then timing of the blast. Viewshot software was used to time and simulate the blast. It is at this time that personnel in the vibration monitoring crew were placed on all monitoring points with the seismographs. These machines were then mounted on the ground in readiness for monitoring and word sent to confirm all is in place for the blast. All access routes leading to the day's blast would then be secured with blast guards each possessing a red flag and portable radio for communication with the blaster at the blasting point. See Area J blasting procedure in the Appendix A.

7. Upon verification of safety in all leading routes to the blasting bench, the blast would be conducted. This process is only complete after rechecking all affected areas for safety even after the blast. Physical examination was conducted including verifications on such property. A contractor was appointed to be checking crack propagation on the structures that were noted to have developed cracks from inception. The report was produced on a monthly basis and all vibrations readings collected and analyzed to ensure safety and health for everyone within the blast radius.

3.3.1 Control Blasting Mitigation Key Parameters

The following parameters where set by South African Bureau of Standards in order to mitigate structural concern while conducting blasting operations, (Afriacn Explosives Limited, 2009):

a) Concern potential for low frequency generating blasts (<10Hz) are significantly higher than frequencies in excess of 40Hz (Low frequency coupled with high vibration levels present higher structural concern potential).
b) Chances of experiencing concern to buildings from blast generated peak particle velocities below 10mm/s are greatly reduced, and that factor greatly depended upon, to predict possibility of causing concern to structures.
c) Structural concern potential is increased with inferior building material employed, as well as age of the building which could have been affected by environmental stresses such as humidity and temperature changes, soil consolidation settlement due to variation in ground moisture and wind, and water absorption from tree roots.

3.3.2 Reducing levels of blast vibrations

African Explosives Limited (AEL) in 2009 proposed the following methods to be used in order to minimise blast vibration levels;

1. Improvement of explosives confinement could result in reduced blast vibrations by:-
    a) Reduction of blast pattern size (Burden and Spacing),
    b) Application of equilateral distances between blast holes as opposed to squared patterns,
    c) Provision of adequate free face,
    d) Increasing available explosive energy per unit volume,
2. Reduction of explosive cooperating charge,
3. Use of longer delays,

Equation 55 is used to determine the cooperating charge per delay considering the distance from the blast to vulnerable structures in close proximity to the pit.

$$PPV = a\left(\frac{D}{\sqrt{E}}\right)^{b} \qquad (55)$$

Where:-

PPV – Peak Particle Velocity in mm/s

a - Intersection point on the Peak Particle Velocity axis (site characteristic constant)

b -Gradient resulting from line generated by intersection points on the Scaled Distance and PPV axis (site characteristic constant)

D - Distance from detonation point to point of interest (Scaled Distance)

E - Highest quantity of detonating cooperating charge in the blast

3.3.3 Managing flyrock

AEL (2009) proposed the following methods to be employed in order to manage flyrock and acoustic over pressure:

a) Ensure maximum confinement of explosives by
- employing adequate stemming,
- employing correct size of stemming materials,
- not overburdening any of the blast holes, and
- employ correct tie up to avoid chocking some blast holes.

b) Reduce cooperating charge per delay by:-
- reducing the number of blast holes detonated per delay,
- employing small diameter blast holes, and decking.

c) Timing the blasts, to tie in with highest ambient noise in the area,

d) Employing toe priming rather than collar priming,

3.4 Factors that affect vibrations and Air Overpressure

When a blast is conducted in motionless atmosphere in which temperature is constant, the air overpressure intensity will solely depend on the distance of the source and reduce by 6dB as the distance doubles from the sources. Such conditions are very rare and is more usual for temperature both to decrease and increase with altitude in a fairly complex and changing manner. Winds also travel at different speeds and direction at differing altitudes. The overall result is that the nominal 6dB reduction may be greater in some directions from the sources and less in others. The atmospheric conditions may affect the intensity of air blast at a particular point. There are two regions of potential damage from air blasts, 'near field' regions surrounding the blast site and 'far field' regions far from the blast site at unknown distance and direction. When the ground vibrations is limited to 50mm/s, there is less likelihood of any damage due to air blast in the near region due to metrological conditions (Walter 1983).

When metrological conditions are such that there is isothermal air and zero wind velocity conditions which would produce isovelocity conditions resulting in straight ray path spherical wave fronts. This is an idea condition that does not exist in reality. Normally, as the altitude increases, the atmospheric temperature decreases, which is known as diabetic lapse rate, resulting in reducing the velocity of sound waves therefore refracting them upwards. Under such conditions, there would not be much effects of air blast in far field regions as the sound is absorbed in the atmosphere. Whenever a temperature inversion exists (where the temperature increases with altitude), the velocity of acoustic waves increases as the atmosphere becomes more dense near ground levels thereby refracting the waves downward back to the ground. It is for this reason that blasts conducted in the morning or evening or night usually result in high noise and air overpressure levels. In the early morning, following a clear night, with low wind velocities, the temperature at high altitude increases as compared to ground and the air immediately above it. However, as sunlight increases this temperature inversion is slowly eliminated. The presence of clouds also creates temperature inversion as they form a screen and prevent sunlight from providing ground heat to reduce the inversion.

Wind is another significant factor which influences the propagation of sound waves. Wind gradients are highly directional. The sound intensity and duration were found to be enhanced in the downward direction. On the downwind side, the winds adds to the velocity effects produced by the inversion and wind is as much as fifty times greater than that expected from a direct wave (Walter 1983). According to Downs and Stock (1977) in severe atmospheric conditions, the intensity of air overpressure levels may increase up to a hundred folds.

Therefore, if possible, blasting should be avoided when there is atmospheric inversion and string wind blowing towards highly inhabited areas.

As the distance from the shot increases, the particle velocity and frequency of ground vibration decreases due to absorption, dispersion and dissipation of elastic waves. The natural frequency of the structures vary with the foundation, condition and age of structure and construction of the structure.

3.5 Instrumentation for measuring Blast Vibration

In order to collect information required to make this dissertation a success, instruments such as seismographs, GIS, digital camera and crack monitors were used.

3.5.1 NOMIS Seismograph: In order to measure and control blast vibrations, instruments such as NOMIS seismographs as shown in Figure 33 were used. This modern vibration instrument is compact, robust and in some cases complex. It has built-in microcomputers driven by sophisticated software and used to perform complex mathematical calculations as well as wave analyses. Hundreds of explosions can be memorized and stored away for later analysis. Vibration monitors consists of a geophone, which is connected by means of a cable to an amplifier and microprocessor

Figure 33: NOMIS seismograph mounted on the ground

The operator can choose whether he prefers to make a print of the information on paper or whether he wants to save it as a file on disk.

Parts of the instrument

a) Geophone

The purpose of the geophone is to convert mechanical motion to an electrical signal. The geophone consists of a coil that moves inside a permanent magnet. (Olofsson, 1990) The output voltage is a function of the velocity of the magnet inside the coil. The geophone unit must be firmly installed into the material to be monitored. If the geophone moves more than the material, it decouples.

One of the most critical aspects for measuring accurately is by placing the sensor firmly in the ground and if possible burying it. When sensors are placed on rock, concrete or equivalent materials, it should be secured in place by means of epoxy resin or cement. Furthermore, calibration of these instruments is essential and should be done regularly to restrict incorrect information that might have disastrous effects.

b) Amplifier

The electrical signal generated by the geophone is most often at a very low voltage and therefore needs to be amplified before a readable signal can be achieved.

c) Software

Mathematical calculations to obtain particle velocity, frequency, acceleration as well as Fourier analysis, needs to be fast and reliable. Further information such as the day, time, the place and name of the operator can be added.

d) The Microphone

The microphone measures the air pressure wave from a blast. The microphones used in blasting seismographs are very sensitive units that measure air pressures at frequencies that are well below the human threshold of hearing. Air blast waves usually consist of low frequencies that cannot be heard. This means that a blast that generates high amplitude air blast waves may not necessarily produce a loud bang.

3.5.2 GPS equipment: this machine was used to measure the distances from the blasts to the seismograph positions. This instrument was also used to stake out drilling patterns in the pit and define blast limits.

3.5.3 Digital camera: A camera was used to take pictures while pursuing the research.

3.5.4 Crack monitors: these are small devices such as can be seen in Figure 35. These monitors contain a transparent measuring rule that is placed on the walls/slabs and floor where cracks are identified. The purpose of this instrument is to check for any expansion, if any, during mining / operating phases of the project. A total of 16 houses were identified to have developed cracks and hence were placed with these devices during this research.

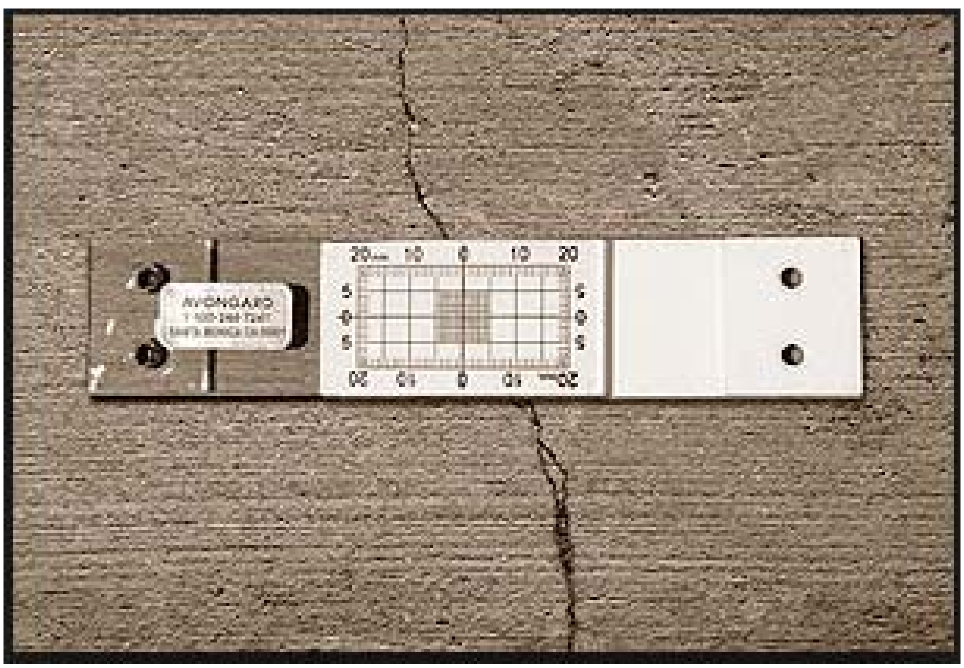


Figure 34: Crack monitors (Google wikimedia.org, 2014)

The readings are checked periodically and any variation is recorded in the table.

3.6 Positioning of the seismographs

In order to record blast induced vibrations generated during blasting activities, the selection of positions where seismographs would be positioned was done in such a way that the nearest vulnerable structures were given priority for installation of seismographs. These positions were basically chosen in order to measure all readings on structures close to the pit and these represented the remaining areas of Wusakile. Areas selected for positioning of the seismographs were: Kalela Basic School (within Wusakile community), KCM / Mopani synclinorium offices (Mine Property), VS4 fan within Mopani (Mine Property) and Compressor house (Mine Property). These areas are shown in Figures 35 and 36.

Note that on the eastern side of the pit, no position was selected as there are no structures. Furthermore, it is believed that subsidence must have been taking place due to previous underground workings.

Training of personnel to install equipment for monitoring blast induced vibrations was conducted by introducing the seismographs to team and explaining to them how to switch them on and ensure that the machines are correctly installed on the ground taking into consideration the precautions as provided for in the 3.4.1. The procedure was then given to them on how to install, and ensure the equipment was correctly placed so that readings were collected accurately.

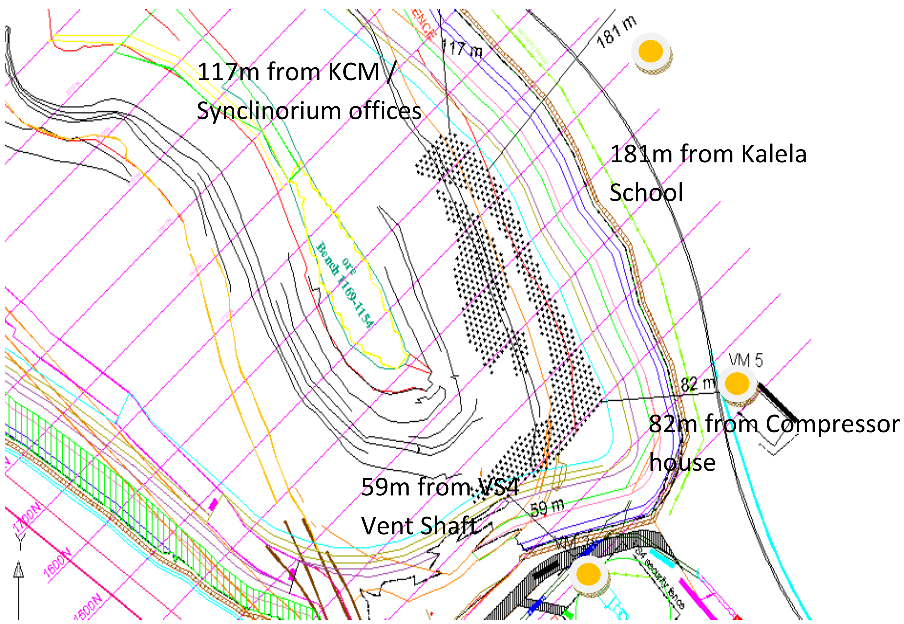


Figure 35: Positions where seismographs were placed during the study

Figure 36: Area J Open pit and the surrounding infrastructure

3.7 Drill and blast design

For the purpose of this dissertation, Kuzram analysis (Cunningham, 1983) was also conducted and the percentage passing was generated to be above 85% based on the crusher feed size of 600mm as shown in Figure 37 and 38:

KUZ-RAM FRAGMENTATION ANALYSIS

Project:	**Area J Open Pit**
Rock Type	**Conglomerate**

Intact Rock Properties		
Rock Factor		
Rock Type		
Rock Specific Gravity	2.65	SG
Elastic Modulus	10	GPa
UCS	51	MPa

Jointing		
Spacing	2	m
Dip	55	deg
Dip Direction	210	deg
In-situ block	0.6	m

Explosives		
Density	1.13	SG
RWS	95%	(% ANFO)
Nominal VOD	4800	m/s
Effective VOD	4500	m/s
Explosive Strength	0.83496094	

Pattern Design		
Staggered or square	1.15	
Hole Diameter	102	mm
Charge Length	3.5	m
Burden	3	m
Spacing	3.45	m
Drill Accuracy SD	0.1	m
Bench Height	6	m
Face Dip Direction	0	deg
Powder Factor	0.20	kg/tonne
Charge Density	0.52	kg/m³
Charge Weight per hole	32.32	kg/hole

Fragmentation Target Parameters		
Oversize	0.75	m
Optimum	0.65	m
Undersize	0.04	m

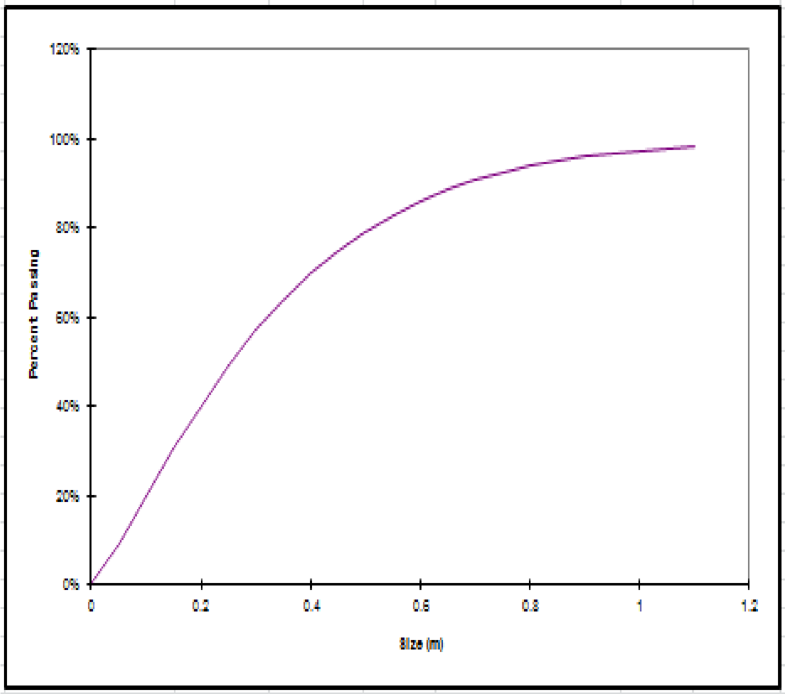
Blastability Index	6.935	
Average Size of Material	26	cm
Uniformity Exponent	1.21	
Characteristic Size	0.35	m

Notes

Square pattern = 1, staggered pattern = 1.1

Predicted Fragmentation		
Percent Oversize	7.7%	m
Percent In Range	85.2%	m
Percent Undersize	7.0%	m

Figure 37: Drilling parameters based on Kuzram model

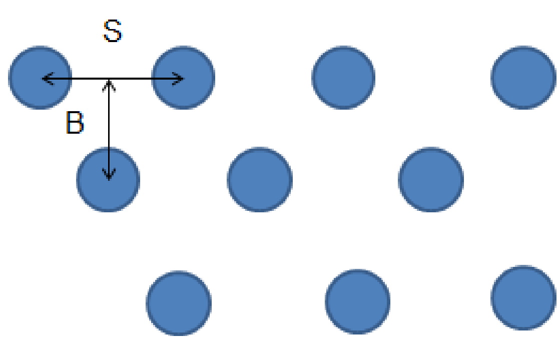


Percent Passing	Size (m)
0.0%	0
9.1%	0.05
19.9%	0.10
30.4%	0.15
40.2%	0.20
49.1%	0.25
56.9%	0.30
63.8%	0.35
69.7%	0.40
74.8%	0.45
79.1%	0.50
82.7%	0.55
85.8%	0.60
88.4%	0.65
90.5%	0.70
92.3%	0.75
93.7%	0.80
94.9%	0.85
95.9%	0.90
96.7%	0.95
97.3%	1.00
97.9%	1.05
98.3%	1.10

Figure 38: Percentage passing of the drill design based on Kuzram

From the above simulations, Figures 39 shows the drill and blast design that was derived and used in all blast trials:

Drilling pattern;



Hole diameter = 102mm (4in)

Spacing = 3.45m

Burden = 3.0m

Depth = 6.0m

Figure 39: Drilling design

Figure 40 shows the sequence of detonation and the resultant vibration direction after considering the following parameters for charging and timing the blast.

Charge length: 3.5m

Stemming: 2.5m, stemming was conducted with 10% of hole diameter aggregate

Explosives: P100, P400 and P700

Delay timing: 50ms between holes

Detonator type: Digi shot

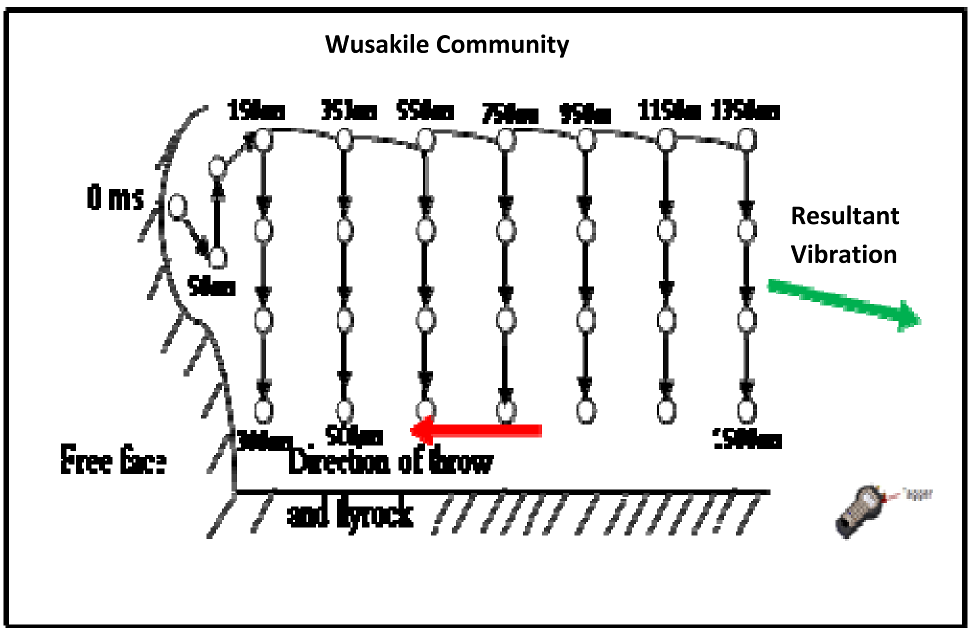


Figure 40: Firing sequence with 50ms inter hole delay

After every blast trial, the seismographs were connected to the computer and the readings were downloaded from the log-log graphs. These graphs show all the vibration readings of the three components of the blast. This data was then recorded in a table such as Table 30 then later placed in the master Table 31.

Table 30: Vibration recording table

Seismographs #	location	Distance from blast	R-waves	T-waves	V - waves	Vector velocity (mm/sec)	Acoustic levels
4104	Kalela basic school	181m	0.127	3.175	2.159	3.302	114dBL
12099	Compressor house	82m	-	-	-	-	- dBL
11677	VS4 fan	59m	16.51	22.23	18.669	29.58	123dBL
11679	KCM store/ Syn. offices	117m	5.207	5.588	4.064	6.61	117dBL

The whole information collected in the blasts was then transferred into the data sheet and analyzed.

CHAPTER 4

DATA PRESENTATION AND ANALYSIS

This chapter presents vibration results as shown in Table 31 obtained during the study. It further presents verification of current drill and blast designs and recommends better designs that address fragmentation concerns.

4.1 Data Presentation and Observations

During blast trials and results obtained in Table 31, three blasts highlighted in red recorded close to the threshold of 10mm/s, the following are the details:

31st May 2012;

Readings obtained at Area J open pit offices 50m away from the shot with Nomis seismograph number 11679 recorded 14.4mm/s. Blast was controlled and directed towards this area in order to reduce the impact heading towards the community. Adjacent to the blast was located the Mopani synclinorium offices 45m away. Seismograph readings on instrument number 11676 recorded 13.2mm/s. The seismograph number 4104 representing vibrations heading to the community recorded 2.667mm/s. This seismograph was located 101m away from the shot and was positioned ahead of the Mopani synclinorium offices. Note that vibrations were diminishing from 13.2mm/s to as low as 2.667mm/s with increasing distance from the blast.

11th June 2012;

Readings obtained at KCM/Mopani Synclinorium offices, 84m away from the shot with Nomis seismograph number 11679 recorded 17.3mm/s. Blast was directed towards VS4 fan. Kalela Basic School seismograph number 4103 which was representing the community 105m away had readings of 10mm/s.

16 August 2012;

Readings obtained at KCM/Mopani Synclinorium offices, 85m away from the shot with Nomis seismograph number 11676 recorded 10.4mm/s. Blast was directed towards Area J open pit offices. Kalela Basic School seismograph number 11679 which was representing the community 99m away had readings as low as 2.51mm/s.

Table 31: Vibration readings from different locations

Area J Open Pit Blast Vibration Report

Date	Time	Seis. ID	Seismograph Location	Blast Dist from Seismograph position	PPV Verticle Sum	Radial		Verticle		Transverse		Acoustic
						PPV (mm/s)	Hz	PPV (mm/s)	Hz	PPV (mm/s)	Hz	dB
09-May-12	16:46	11677	SOB VS4 Fan	74	8.320	8.255	12.4	8.001	18.2	7.620	21.3	113
	16:48	11679	KCM Sync Offices	86	5.990	5.969	30.1	5.080	26.9	4.953	20.4	114
	16:48	4104	Kalela Basic School	110	3.175	0.000	0.0	2.032	11.1	3.175	13.8	106
11-May-12	17:10	11677	SOB VS4 Fan	54	2.070	0.889	18.2	0.762	14.6	2.032	6.9	113
	17:10	11679	KCM Sync Offices	72	1.810	1.524	16.0	1.016	24.3	1.778	13.8	119
	17:04	4104	Kalela Basic School	98	1.016	0.000	0.0	0.508	15.5	1.016	10.6	106
12-May-12	16:52	11679	SOB VS4 Fan	60	3.450	3.429	11.1	1.524	15.5	1.9.5	13.4	113
	16:52	11677	KCM Sync Offices	90	8.440	7.747	16.0	5.207	23.8	7.366	16.5	127
	17:04	4104	Kalela Basic School	124	2.921	0.000	0.0	2.032	17.6	2.921	18.2	110
17-May-12	16:52	11679	SOB VS4 Fan	77	3.450	3.429	11.1	1.524	15.5	1.905	13.4	113
	16:52	11677	KCM Sync Offices	80	8.440	7.747	16.0	5.207	23.2	7.366	16.5	127
	16:47	4104	Kalela Basic School	100	2.921	0.000	0.0	2.032	17.6	2.921	18.2	110
21-May-12	16:56	11677	SOB VS4 Fan	90	5.370	5.207	21.3	2.921	17.6	3.302	19.6	115
	16:56	11679	KCM Sync Offices	87	8.580	7.874	23.2	5.588	22.2	4.572	19.6	120
	16:55	11676	Kalela Basic School	105	2.540	2.159	71.1	2.032	19.6	2.540	14.2	106
31-May-12	16:46	11677	SOB VS4 Fan	100	4.000	3.429	11.9	2.794	18.2	3.429	14.2	117
	16:46	11676	KCM Sync Offices	45	13.290	11.938	19.6	11.811	28.4	9.144	36.5	127
	16:45	4104	Kalela Basic School	101	2.667	0.127	0.0	2.667	25.6	2.413	22.2	106
	16:44	11679	Area J Offices	50	14.400	9.652	8.0	6.477	9.1	12.700	7.0	122
12-Jun-12	16:55	11679	SOB VS4 Fan	44	29.580	16.510	34.1	18.669	24.3	22.225	36.5	123
	16:55	11676	KCM Sync Offices	70	6.610	5.207	18.2	4.064	15.0	5.588	9.4	117
	16:55	4104	Kalela Basic School	93	3.302	0.127	0.0	2.159	24.3	3.175	20.4	114
25-Jun-12	16:55	11679	SOB VS4 Fan	65	21.660	16.510	28.4	141.00	25.6	13.843	30.1	141
	16:55	11676	KCM Sync Offices	124	1.260	0.254	0.0	1.143	20.4	1.016	13.8	113
	16:55	11677	Kalela Basic School	185	1.470	1.143	17.0	0.762	16.0	1.270	21.3	107
11-Jul-12	17:12	11677	SOB VS4 Fan	100	3.120	2.667	13.8	1.397	16.5	2.413	12.4	125
	17:12	11679	KCM Sync Offices	84	17.310	15.494	28.4	9.144	23.2	12.573	42.6	135
	17:12	4103	Kalela Basic School	105	10.141	8.001	20.4	5.334	24.3	6.731	26.9	127
13-Jul-12	17:23	11677	SOB VS4 Fan	62	2.900	2.413	12.1	1.270	15.5	2.159	13.8	110
	17:22	11679	KCM Sync Offices	104	1.960	1.651	13.1	1.524	15.0	1.778	13.4	127
	17:25	4103	Kalela Basic School	132	1.650	1.016	170.6	1.016	170.0	0.889	170.6	100
18-Jul-12	16:43	11676	SOB VS4 Fan	66	7.000	4.191	16.0	3.937	18.9	6.731	15.0	116
	16:43	11677	KCM Sync Offices	87	2.210	1.524	13.1	0.762	25.6	2.159	13.4	114
	16:42	11679	Kalela Basic School	104	1.530	1.524	22.2	1.016	16.5	0.508	34.1	110
23-Jul-12	17:41	11676	SOB VS4 Fan	70	4.560	3.683	17.0	2.540	18.9	2.921	18.2	114
	17:41	11677	KCM Sync Offices	88	3.670	3.048	26.9	2.540	18.9	3.556	18.9	107
	17:41	11679	Kalela Basic School	96	3.210	2.413	17.0	1.778	14.6	3.048	17.0	105
27-Jul-12	16:46	11676	SOB VS4 Fan	54	15.509	4.191	26.9	8.509	32.0	14.986	46.5	119
	16:46	11677	KCM Sync Offices	100	0.000	0.000	0.0	0.000	0.0	0.000	0.0	0
	16:46	11679	Kalela Basic School	120	0.000	0.000	0.0	0.000	0.0	0.000	0.0	0
02-Aug-12	17:21	11679	SOB VS4 Fan	55	25.280	23.495	28.4	19.685	26.9	12.954	36.5	117
	17:21	11677	KCM Sync Offices	86	8.820	8.636	24.3	4.826	32.0	5.588	32.0	112
	17:21	11676	Kalela Basic School	115	3.070	2.159	14.6	1.524	24.3	3.048	17.0	104
16-Aug-12	16:50	11677	SOB VS4 Fan	60	8.360	7.493	20.4	5.461	30.1	6.350	24.3	117
	16:50	11676	KCM Sync Offices	85	10.450	10.414	32.0	4.572	51.2	5.334	23.2	110
	16:51	11679	Kalela Basic School	99	2.510	2.320	17.6	1.778	15.0	2.286	15.5	108
21-Aug-12	16:45	11676	SOB VS4 Fan	65	3.830	2.794	17.6	3.683	20.4	3.048	16.5	118
	16:44	11679	KCM Sync Offices	80	1.310	0.762	15.5	0.889	20.4	1.270	19.6	107
	16:45	11677	Kalela Basic School	103	0.000	0.000	0.0	0.000	0.0	0.000	0.0	0
27-Aug-12	16:45	11676	SOB VS4 Fan	53	1.430	1.016	10.0	0.762	11.1	1.143	8.8	111
	16:45	11677	KCM Sync Offices	95	1.400	1.143	7.6	0.635	7.3	1.016	8.2	116
	16:41	11679	Kalela Basic School	128	1.200	1.016	5.6	0.508	8.5	1.143	6.8	110
04-Sep-12	16:45	11676	SOB VS4 Fan	53	14.560	13.335	20.4	10.414	22.2	5.842	5.7	129
	16:45	11677	KCM Sync Offices	70	3.330	2.921	9.6	1.651	25.6	1.651	6.8	111
	16:41	11679	Kalela Basic School	86	2.370	1.905	16.0	1.524	8.5	2.524	17.0	110
06-Sep-12	16:45	11676	SOB VS4 Fan	60	16.710	14.605	36.5	14.097	34.1	5.461	16.0	120
	16:45	11677	KCM Sync Offices	75	0.000	0.000	0.0	0.000	0.0	0.000	0.0	0
	16:41	11679	Kalela Basic School	98	1.590	1.143	20.4	0.889	23.2	1.524	11.1	108
10-Sep-12	16:59	11679	SOB VS4 Fan	102	3.36	2.413	9.800	2.2	10.400	2.5	9.300	115
	17:00	11676	KCM Sync Offices	76	3.160	2.286	26.9	1.778	30.1	2.921	18.2	116
	17:00	11677	Kalela Basic School	88	1.550	1.524	9.3	1.143	14.2	1.016	14.2	110
13-Sep-12	16:44	11676	SOB VS4 Fan	56	8.64	7.239	21.300	6.4	36.500	6.0	30.100	116
	16:45	11677	KCM Sync Offices	78	4.070	3.937	23.2	2.921	13.8	2.667	23.2	113
	16:45	11679	Kalela Basic School	91	1.670	1.397	18.9	1.524	22.2	1.270	26.9	105
21-Sep-12	16:44	11676	SOB VS4 Fan	66	6.13	3.429	16.500	5.2	21.300	4.4	18.900	116
	16:45	11669	KCM Sync Offices	86	1.440	1.143	8.8	0.762	13.4	1.016	9.3	117
	16:45	11677	Kalela Basic School	90	1.420	1.270	10.0	1.016	18.9	1.270	11.6	108
25-Sep-12	16:46	11676	SOB VS4 Fan	44	18.32	15.240	22.200	17.7	32.000	8.6	28.400	129
	16:47	11669	KCM Sync Offices	87	2.050	1.905	11.3	1.397	19.6	1.524	14.2	119
	16:47	11677	Kalela Basic School	99	2.220	1.651	20.4	1.397	15.5	1.778	16.5	116

4.2 Using Software and Data Analysis

During this study, a total of 23 blasts were monitored and tested and only three of them recorded close to or higher readings above the acceptable threshold of 10mm/sec PPV at KCM and synclinorium offices.

Blast conducted on 31 May the foot wall close to the community; direction of throw was planned towards the center of the pit hence directing most of the waves towards the community. This was done in an effort to prevent flyrock going towards the community as the rock was observed to be very hard. The resulting PPV of 14.4 mm/s and 13.29mm/s were recorded. However, as can be observed with the measurements recorded from Kalela Basic School on Table 31, there was a diminishing effect as the distance increased towards the community. It was also observed that the frequencies of those readings were higher than 10Hz; therefore, the likelihood of these structures with sound construction to get damaged is minimal because frequency of the waves must be lower than 10Hz and further be found to be in resonance to the natural frequency of the structures.

The same outcome was observed on blasts conducted on the 11th July 2012 and 16th August 2012. Furthermore, predominate blast wave propagation was directed towards VS4 fan, a vent shaft owned by MCM. This structure was constructed with heavy reinforced concrete hence its allowable peak particle velocity is 50mm/s as opposed to 10mm/s. Therefore, no damage is expected to be caused resulting from the blasts as all readings were below the expected threshold of 50mm/s for that type of structure.

It is very important to also note that vibrations were diminishing in all three blast trials as distance increased from the shot to positions where readings were being recorded.

Following the finding obtained in Table 31, Table 32 was then generated in order to determine the scaled distance in relation to the PPV obtained during the study. This information is very important because it is from Table 32 that we are able to plot the results of a PPV graph and come up with the best fit line which is used to determine the PPV prediction equation.

Table 32: Scaled distance from the various positions

Date	Time	Seismograph ID	Seismograph position	PPV Vector Sum mm/s	Scaled Distance
09-May-12	16:46	11677	SOB VS4 Fan	8.32	10.47
	16:48	11679	KCM Sync Offices	5.99	12.16
	16:48	4104	Kalela Basic School	3.18	15.56
11-May-12	17:10	11677	SOB VS4 Fan	2.07	7.64
	17:10	11679	KCM Sync Offices	1.81	10.75
	17:04	4104	Kalela Basic School	1.02	13.86
12-May-12	16:52	11679	SOB VS4 Fan	3.45	8.49
	16:52	11677	KCM Sync Offices	8.44	16.97
	17:04	4104	Kalela Basic School	2.92	14.43
17-May-12	16:52	11679	SOB VS4 Fan	3.45	10.89
	16:52	11677	KCM Sync Offices	8.44	11.32
	16:47	4104	Kalela Basic School	2.92	14.14
21-May-12	16:56	11677	SOB VS4 Fan	5.37	12.73
	16:56	11679	KCM Sync Offices	8.58	12.31
	16:55	11676	Kalela Basic School	2.54	13.44
31-May-12	16:46	11677	SOB VS4 Fan	4.00	14.14
	16:46	11676	KCM Sync Offices	13.29	6.36
	16:45	4104	Kalela Basic School	2.67	14.29
	16:44	11679	Area J Offices	14.40	7.07
12-Jun-12	16:55	11679	SOB VS4 Fan	29.58	6.22
	16:55	11676	KCM Sync Offices	6.61	9.90
	16:55	4104	Kalela Basic School	3.30	13.15
25-Jun-12	16:55	11679	SOB VS4 Fan	21.66	9.19
	16:55	11676	KCM Sync Offices	1.26	17.54
	16:55	11677	Kalela Basic School	1.47	26.17
11-Jul-12	17:12	11677	SOB VS4 Fan	3.12	14.14
	17:12	11679	KCM Sync Offices	17.31	11.88
	17:12	4103	Kalela Basic School	10.14	14.85
13-Jul-12	17:23	11677	SOB VS4 Fan	2.90	8.77
	17:22	11679	KCM Sync Offices	1.96	14.71
	17:25	4103	Kalela Basic School	1.65	18.67
18-Jul-12	16:43	11676	SOB VS4 Fan	7.00	9.34
	16:43	11677	KCM Sync Offices	2.21	12.31
	16:42	11679	Kalela Basic School	1.53	14.71
23-Jul-12	17:41	11676	SOB VS4 Fan	4.56	9.90
	17:41	11677	KCM Sync Offices	3.67	12.45
	17:41	11679	Kalela Basic School	3.21	13.58
27-Jul-12	16:46	11676	SOB VS4 Fan	15.51	7.64
	16:46	11677	KCM Sync Offices	0.00	14.14
	16:46	11679	Kalela Basic School	0.00	16.97
02-Aug-12	17:21	11679	SOB VS4 Fan	25.28	7.78
	17:21	11677	KCM Sync Offices	8.82	12.16
	17:21	11676	Kalela Basic School	3.07	16.27
16-Aug-12	16:50	11677	SOB VS4 Fan	8.36	8.49
	16:50	11676	KCM Sync Offices	10.45	12.02
	16:51	11679	Kalela Basic School	2.51	14.00
21-Aug-12	16:45	11676	SOB VS4 Fan	3.83	9.19
	16:44	11679	KCM Sync Offices	1.31	11.32
	16:45	11677	Kalela Basic School	0.00	14.57
27-Aug-12	16:45	11676	SOB VS4 Fan	1.43	7.50
	16:45	11677	KCM Sync Offices	1.40	13.44
	16:41	11679	Kalela Basic School	1.20	18.10
04-Sep-12	16:45	11676	SOB VS4 Fan	14.56	7.50
	16:45	11677	KCM Sync Offices	3.33	9.90
	16:41	11679	Kalela Basic School	2.37	12.16
06-Sep-12	16:45	11676	SOB VS4 Fan	16.71	8.49
	16:45	11677	KCM Sync Offices	0.00	10.61
	16:41	11679	Kalela Basic School	1.59	13.86
10-Sep-12	16:59	11679	SOB VS4 Fan	3.36	14.43
	17:00	11676	KCM Sync Offices	3.16	10.75
	17:00	11677	Kalela Basic School	1.55	12.45
13-Sep-12	16:44	11676	SOB VS4 Fan	8.64	7.92
	16:45	11677	KCM Sync Offices	4.07	11.03
	16:45	11679	Kalela Basic School	1.67	12.87
21-Sep-12	16:44	11676	SOB VS4 Fan	6.13	9.34
	16:45	11669	KCM Sync Offices	1.44	12.16
	16:45	11677	Kalela Basic School	1.42	12.73
25-Sep-12	16:46	11676	SOB VS4 Fan	18.32	12.73
	16:47	11669	KCM Sync Offices	2.05	12.31
	16:47	11677	Kalela Basic School	2.22	14.00

Based on the data as indicated in Tables 31and 32, the graph for PPV against scaled distance was plotted and is depicted in Figure 41.

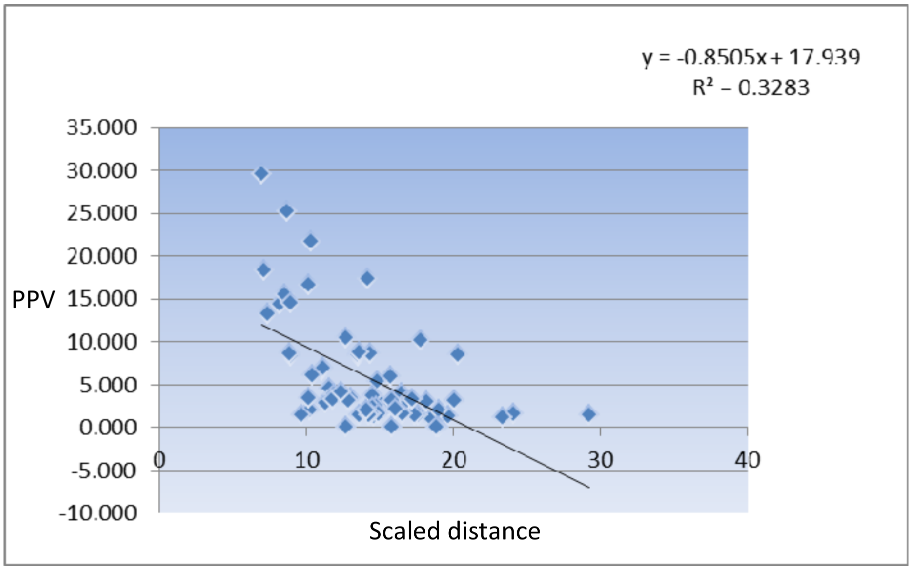


Figure 41: PPV plot scaled distance obtained from research

Using the best fit line from Figure 41, the new Equation 56 is derived from Equation 55 as was given earlier;

$$PPV = a\left(\frac{D}{\sqrt{E}}\right)^{b} \qquad (55)$$

Replacing the values of a and b generated from Figure 42 into Equation 55,

a=17.9 and b=-0.8505

Therefore, the derived equation for predicting PPV is Becomes:

$$PPV = 17.9\left(\frac{D}{\sqrt{E}}\right)^{-0.85} \qquad (56)$$

With Equation 56, it is now possible to predict PPV provided the distance and cooperating charge is known, (Mwale, et al., 2015).

Further to the analysis conducted above, Table 33 generated through the study conducted by JKL Consultants showed that only less than 6% of the 16 houses in Wusakile community had a 1mm crack expansion from the original cracks developed prior to commencement of blasting activities at Area J Open Pit.

Table 33: Crack monitoring results for December 2015 (JKL Associates, 2014)

DATE	PLOT No.	DEVICE No.	CRACK WIDTH BEFORE DEVICE INSTALLATION (mm)	CURRENT READING(CRACK WIDTH) (mm)	Variance between Initial and Current readings (mm)
23-11-2014	B1 - 6	2	5.0	6.0	1.0
23-11-2014	B1 - 17	3	4.0	4.0	0.0
23-11-2014	B1 - 21	4	6.0	6.0	0.0
23-11-2014	B1 - 25	5	24.0	24.0	0.0
23-11-2014	B1 - 31	6	7.0	7.0	0.0
23-11-2014	B1 - 70	7	6.0	6.0	0.0
23-11-2014	B1 - 71	8	4.0	4.0	0.0
23-11-2014	B1 - 19	10	2.0	2.0	0.0
23-11-2014	B1 - 28	11	5.0	5.0	0.0
23-11-2014	B1 - 29	12	5.0	5.0	0.0
23-11-2014	B1 - 30	13	2.0	2.0	0.0
23-11-2014	B1 - 85	14	4.0	4.0	0.0
23-11-2014	B1 - 87	15	4.0	4.0	0.0
23-11-2014	B1 - 18	17	13.0	13.0	0.0

It was further observed that some of the cracks in Wusakile were as a result of ground movement caused by natural settling of the material and not blasting operations. This is clear from the report generated by JKL consultants that showed that these cracks could have resulted from ground natural settlement during initial footing settlement. This is because the cracks were wider at the top than the bottom of the crack. This suggested that the footing of the building to the left or right of the crack had moved downwards.

It was however observed that fragmentation was compromised in most of the blasts trials. Investigations conducted revealed that the stemming column in most cases resulted in poor fragmentation while the charged length had better breakage. In order to improve on the results, the following trial was conducted to evaluate breakage:

Fragmentation trial: 13th September 2012

Initial blast design during the tests is shown in Figure 42 and this design had the following parameters:

Original design:

Burden: 3m

Spacing: 3.45m

Hole depth: 6.5m

Charge height: 3.5m

Total charge: 32kg

Stemming height: 3m with aggregate

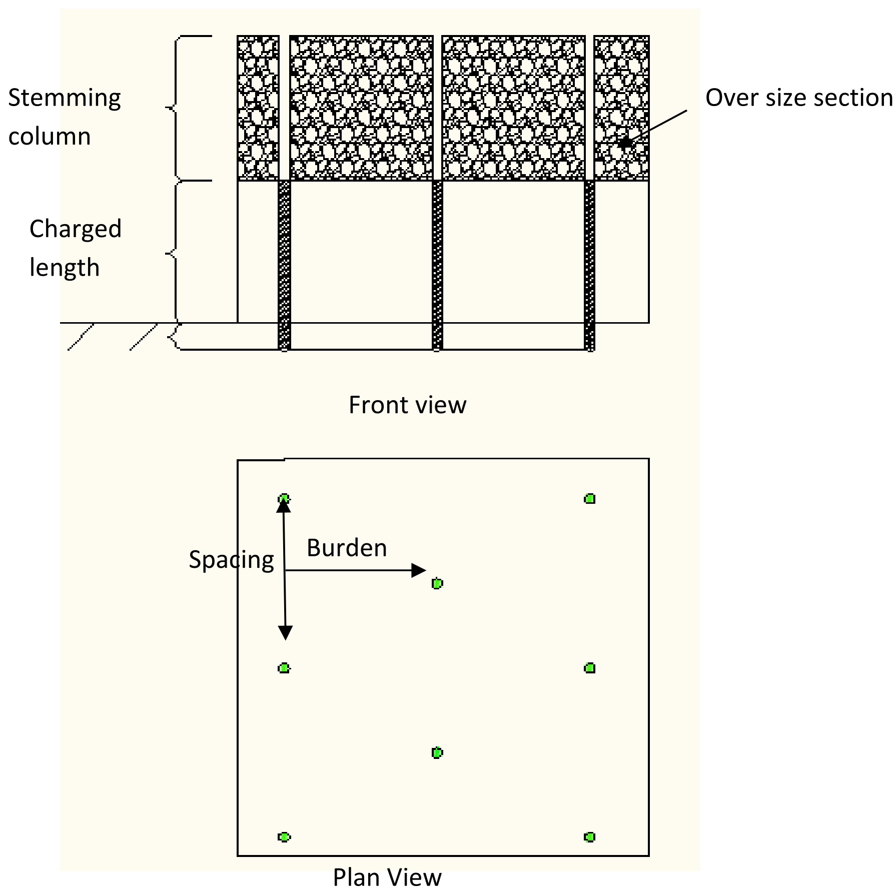


Figure 42: Initial drill and charging layout

Improved design:

Figure 43 illustrates the improved design that includes infill holes with 3m depth and 89mm diameter positioned between the initial patterns.

These holes were placed in between the original pattern

Charge height: 0.5m

*Total charge: 3.5*kg

Due to sensitivity of blasting the bench that is located required on the high wall close to the community, P100 emulsion was used with a 30ms delay between holes.

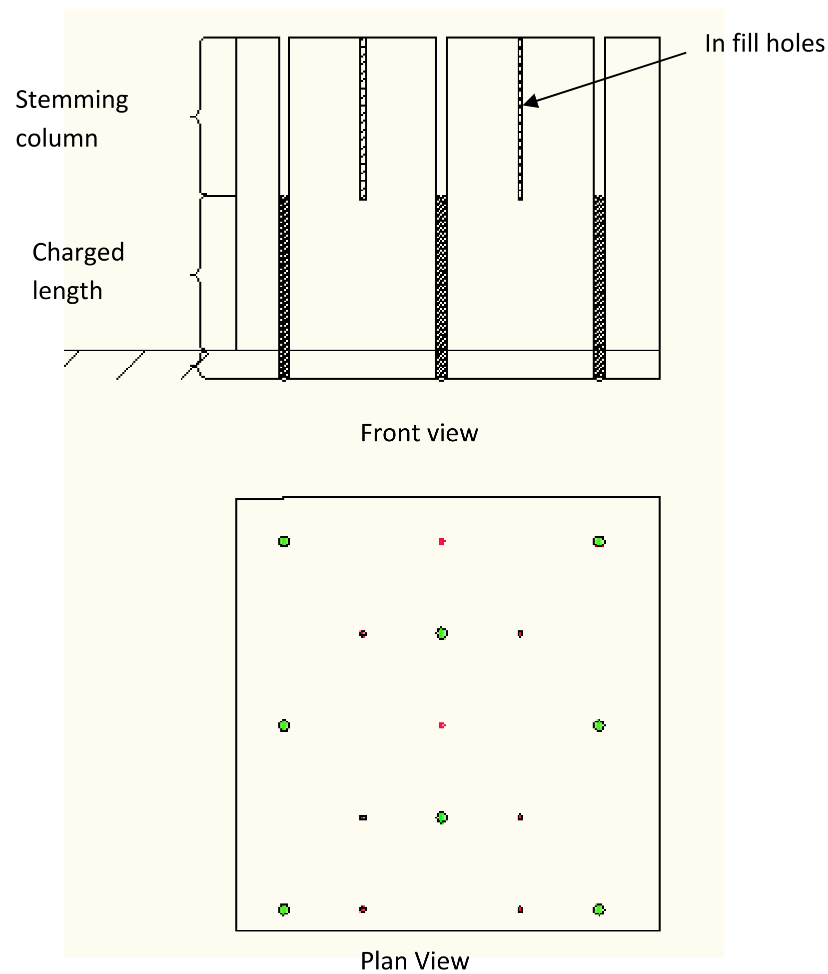


Figure 43: Revised drill and blast layout

Results obtained with regards to fragmentation looked good. Due to the number of holes, the delay between holes was reduced to 30ms between holes. The results during the blast trial were generated as shown in Table 34 and the highest readings were measured at KCM office from a distance of 78m. This is also within the maximum acceptable readings of 10mm/s.

Table 34: Blast trial on fragmentation control

Seismographs #	Location	Distance from blast	R-waves	T-waves	V - waves	Vector velocity (mm/sec)	Acoustic levels
11679	Kalela basic school	91m	1.39	1.52	1.27	1.67	105dBL
0	Area J Open pit	0m	0	0	0	0	0dBL
11677	KCM/ Syn. office	78m	3.94	2.92	2.67	4.0	117dBL
11676	VS4 shaft	56m	7.2	6.4	6	8.6	116dBL

Note that resultant vibration was directed towards VS4 fan and fragmentation on the top part of the bench was easily mineable.

The blast designs undertaken to conduct control blasting close to the community were done in a correct manner. Table 31 confirms that all vibration readings recorded in the community had vibrations lower than the threshold of 10mm/s. This was mainly due to the precautions taken into considerations while preparing for drilling, charging and blasting.

Furthermore, it was also observed that all blasts had no flyrock going towards the community and the noise levels were all kept below the limits.

CHAPTER 5

CONCLUSION AND RECOMMENDATIONS

5.1 Conclusions

5.1.1 All blast induced vibrations recorded at Kalela Basic School and the Wusakile community were kept below the threshold of 10mm/s. Furthermore, no flyrock was observed to have gone beyond the blast limit of 100m surrounding the blast patterns and all noise levels were kept below the acceptable limits of 134dB. It was also observed that vibrations were quickly diminished with increasing distances away from the blast as some blasts recorded vibrations higher than 10mm/sec at KCM / Mopani synclinorium offices and yet these values greatly reduced with increase in distance.

5.1.2 A total of 16 housing units within Wusakile were observed to have developed cracks prior to start of mining at Area J, these houses had some crack monitoring devices installed on them. Results have showed only 1mm expansion from one house out of the remaining 15 houses. This gives an average of less than 6%. It is, therefore, safe to conclude that no crack propagation is attributed to blasting activates from Area J open pit.

5.1.2 The prediction threshold Equation 55 was deduced with the data collected in the study. Based on this research, the prediction Equation for Mopani blasting site has the following site characteristics;

$$a=17.9 \text{ and } b=-0.8505.$$

With the derived PPV Equation, it is now possible to predict the levels of vibration that are likely to be generated given the total amount of cooperating charge detonated.

5.1.3 By drilling small diameter infill holes on a normal production blast pattern in hard formations, it is possible to improve fragmentation in non-homogeneous formations where rock is stratified in layers. This was observed to have been the reason for poor fragmentation during the blast as most of the material where the explosives was positioned could get well detonated whereas the parts on the column charge would remain un charged. Furthermore, vibration readings were kept below the limits even after the trial. It should be noted that infill holes must be drilled with a smaller diameter so that they do not produce fly rock hence posing a hazard to the environment. A light charge is to be used leaving a minimum of 2.5m for well controlled stemming.

Therefore, blasting in close proximity to densely populated communities is achievable provided that the phenomenology of rock blasting is clearly understood and the techniques of controlled drilling and blasting are implemented.

5.2 Recommendations

A good blast is one in which the blaster is in control of the majority of blast parameters. A better blast has all controlled parameters achieved with good fragmentation. It is always recommended that all drilling and blasting parameters are followed whenever a blast is conducted close to structures and the community. This minimizes concerns raised by the community.

In hard sedimentary rock blasting where formations happen to be layered, there may be need to drill infill holes on the stemming section so as to break the column section. This will require reducing the diameter of the bit thereby reduce the specific charge and consequently control the throw of rock and vibrations. This way, both lower vibration and good fragmentation will be achieved with minimum environmental concerns from the community.

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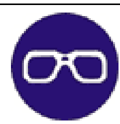
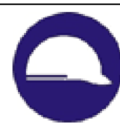
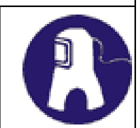
APPENDIX A: Drilling and Blasting Procedure

	MOPANI COPPER MINES PLC **HEALTH, SAFETY AND ENVIRONMENT** **FM-SA-013** **BEST OPERATING PRACTICE** **Drilling, Charging and Blasting**

Document No.		**Document Owner**	MANAGER OPEN PITS		
Revision No.	Initial	**Approver**	MINE MANAGER		
Date Approved		Department	MINING	Section:	OPEN PITS
Scope	The practices and standards to be adopted during planning of drilling, charging and blasting operations.				
Objectives	To comply with drilling, charging and blasting standards.				

PPE Requirements (Place a ■ below the appropriate PPE required)											
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STEP	ACTIVITY	HSEQ MESSAGE

STEP	ACTIVITY	HSEQ MESSAGE
	Preparation	
1.	The Mine Captain shall ensure the drilling site is exposed to solid.	To maintain the bench profile
2.	The Mine Captain shall ensure the drilling crew has the drilling lay out approved by the relevant authorities	To conform to mine planning
3.	The Surveyor shall stake the area and the Shift Boss will barricade off the drilling area.	To avoid the operator drilling on the wrong position and un authorised people accessing it.
4.	The Shift Boss shall ensure positions to be drilled are properly marked	To avoid the operator drilling on the wrong position
5.	The Mine Captain shall ensure tools and equipment are in good condition before commencing drilling	To do the job safely.
	Drilling	
1.	All personnel involved in drilling shall be competent and certified by MCM Mine Training.	To ensure accuracy in job performance
2.	All holes are to be drilled as per drilling lay out. The Blasting Officer shall ensure that all holes are drilled as per lay out and to check for accuracy in depth and position as soon as they are drilled and the information shall be kept on record.	To ensure accuracy and avoid drilling short holes which may result into flying rocks.
3.	Any hole found to have been drilled off the marked position and under drilled by more than 1.0 metre will be re-staked and labelled for re-drilling in the right position and the drill operator will be informed.	To ensure accuracy and avoid drilling short holes which may result into flying rocks.
4.	All drilled holes will be plugged by using bags filled with chippings	To avoid blockages.
5.	The Blasting Engineer will ensure that all the holes not drilled as per lay out are re-drilled.	To ensure correct blasting of drilled holes.

STEP	ACTIVITY	HSEQ MESSAGE
	Charging	
1.	The drilling and blasting Engineer shall prepare a blast plan before any blasting undertaking.	To show the areas to be affected by the blast.
2.	The Mine Captain shall ensure the correct material and equipment for blasting such as; anfex, emulsion, pentolite boosters, dig shots, blasting cable, shovels, wheel bow , approved charging rods ,sacks and aggregate are available. And ensure the charging area is free from any sparking material, radio communication and proper signage is put in place.	To do the job safely.
3.	On the day of charging red flags should be put in visible positions as a way of warning and sensitizing those close to the area.	To make people aware of the charging taking place.
4.	The blasting officer shall ensure that all the holes are toe primed.	To avoid over charging of holes.
5.	Charging the hole with emulsion, the Blasting Engineer shall give the charging crew the weight of explosives to be used in each hole.	To do a correct charge and avoid fly rocks.
6.	The Blasting Engineer shall specify the collar to be used at that particular blast. Currently 3.5m collar is being used (the holes should be marked 4m for a 3.5 m collar)	To prevent over charging of the hole.
7.	All holes shall be filled with ¾ aggregate as a stemming medium	To ensure maximum confinement of the explosives charge
8.	The blasting Officer shall ensure tagging, harnessing , testing of detonators are done	To do the job safely
	Blasting Clearance	
1.	The Mine Captain in Charge of Blasting shall discuss with the Duty Supervisors the safe parking of all equipment, cables and any other items that may be affected by the Blast. This discussion shall be concluded and the relevant equipment moved to safe areas by 10:00	To avoid personal injuries and property damage.

STEP	ACTIVITY	HSEQ MESSAGE
	hours on the day of blast. A list of equipment allocation shall be made available to the Mine Captain in Charge of Blasting at least one hour before blasting time	
2.	All roads leading to Area "J" Open Pit will be closed to the public only motor vehicles involved in blasting operation will access (Road to be closed; SOB Road and perimeter roads leading to the plants at designated points determined by the Mine Captain in Charge of Blasting).	To avoid personal injuries and property damage.
3.	The Mine Captain in Charge of Blasting shall appoint Officials in the blasting log book to clear respective areas where Open Pit personnel and contractors are working.	To avoid personal injuries and property damage.
4.	These appointed officials shall sign in the **Blast Register** to acknowledge the appointments. Preferably, supervisors will be appointed to clear employees in their sections.	To avoid personal injuries and property damage.
5.	In case of equipment on maintenance, the responsible foremen shall be appointed to clear all maintenance employees at the machine(s).	To avoid personal injuries and property damage.
6.	Where Townships, Workshops etc, are affected by the blast, the Mine Captain in Charge of Blasting shall post a red flag(s) at designated points in the area(s) likely to be affected not later than 10.00 hours on the day of blast	To avoid personal injuries and property damage.
7.	The Mine Captain in Charge of Blasting with the assistance of Mopani Security and Safety will ensure timely and adequate communication to the residents affected.	To avoid personal injuries and property damage.
8.	The Mine Captain in Charge of Blasting shall **acknowledge** the safety of all machines before blast.	To avoid personal injuries and property damage.
9.	Clearance of personnel from the blast area will only commence	To avoid personal injuries and property damage.

STEP	ACTIVITY	HSEQ MESSAGE
	when the Mine Captain in Charge of Blasting is satisfied that: • Charging operations are completed • All affected equipment is parked away from blast area • All cables are in safe position if electric equipment are been used • All guards are in their respective positions • Roads barricaded off	
10.	All Supervisors appointed to clear men from blast shall inform the Mine Captain in Charge of Blasting that the following tasks have been completed: • Declaration of clearance of the assigned area. • Headcount of the men that have been cleared and their Equipment location. • Exit route being used.	To avoid personal injuries and property damage.
11.	The Mine Captain in Charge of Blasting must be satisfied with the record of the individual headcount of men cleared from each appointed Supervisor.	To avoid personal injuries and property damage.
12.	The Mine Captain in Charge of Blasting and his Shift Boss will do the final examination of cleared areas prior to blast initiation.	To avoid personal injuries and property damage.
13.	Once satisfied with the above the Mine Captain in Charge of Blasting shall request for the warning first Siren to be sounded for five minutes, fifteen minutes before the scheduled blast time	To avoid personal injuries and property damage.
14.	Whenever blasting is taking place in an area within 200 m from the township the following will apply: a) Notice of blast stating the date and time of blast will be given to residents.	To avoid personal injuries and property damage.

STEP	ACTIVITY	HSEQ MESSAGE
	b) Red flags shall be posted near the township around 09:00 – 10:00 hours for the 15:30hours blasting. c) A radius of 200 m from the Open Pit limit of the blast shall be marked on the plan. d) Blasting shall be conducted only after the Mine Captain in Charge of Blasting is satisfied that the affected areas have been cleared/vacated.	
15.	Every time Primary Blasting is conducted Mine Captain in Charge of Blasting with the assistance of Mopani Security shall clear occupants of the building in proximity(houses affected to be tabulated), which fall within the **200m** radius from the pit according to the plan:	To avoid personal injuries and property damage.
16.	Flagmen shall be positioned at road junctions and all strategic positions to stop traffic and pedestrians entering danger areas.	To avoid personal injuries and property damage.
17.	The second warning Siren shall be switched on at the scheduled time of blast and will be sounded continuously until the blast operation is declared over.	To avoid personal injuries and property damage.
18.	The Mine Captain in Charge of Blasting will conduct the blast as follows: A) There shall be Radio communication between the Mine Captain in Charge of Blasting and the Person in charge of blasting. • Mine Captain: Five calling PIC • PIC: Receiving • Mine Captain: All affected areas have been cleared; you may blast now. • PIC: Rodger,	To avoid personal injuries and property damage.

STEP	ACTIVITY	HSEQ MESSAGE
	• Mine Captain: Rodger • PIC: Blast b) A thirty-second window shall be allowed before firing.	
19.	After the blast only the Mine Captain in Charge of Blasting shall authorize re-entry after satisfying himself that the blast(s) have gone off and the pit is free from blast fumes and dust.	To avoid personal injuries and property damage.
20.	Where electronic detonators have been used in the blast, the siren can be switched off immediately	To avoid personal injuries and property damage.
21.	In the event of a failed blast, the Mine Captain in Charge of Blasting shall inform the Open Pit Mine Superintendent accordingly. The blasting Siren shall continue sounding. Guards shall remain in their assigned positions and no open pit personnel except the blasters shall be allowed to enter the blast zone.	To avoid personal injuries and property damage.
22.	The following precautions shall be observed: • Re – entry set by Ventilation Engineer will be observed • Wait for fumes to clear The above, however, shall not apply to non-electric and non-electronic blasts e.g. safety fuses.	To avoid personal injuries and property damage.
23.	Should any unforeseen circumstances cause the blast time to be extended or brought forward, the Mine Captain in Charge of Blasting shall communicate to the Open Pit Mine Superintendent as well as all the affected parties in time.	To avoid personal injuries and property damage.
24.	Where the Blast schedule overlaps into the on- coming shift, the entire process must be repeated	To avoid personal injuries and property damage.
25.	On the day when conditions are not favourable to due to rains, the Mine captains and blasting Engineer shall suspend the blasting and	To avoid personal injuries and property damage

STEP	ACTIVITY	HSEQ MESSAGE
	inform the Mine Manager.	

OCCUPATIONAL ILLNESSES RELATING TO WORK

ILLNESS	SYMPTOM

DEFINITIONS AND ABBREVIATIONS

TERM	DESCRIPTION
Hole	Any drilled in rock for the purpose of containing explosives.
Charging	Insertion of explosives into a hole.
Blasting	Detonation of hole filed with explosives.
Harnessing	Clipping of detonators to harness wire

REFERENCES

REFERENCE	AUTHOR	TITLE
Guide to the Explosives Act and the Explosives Regulations 1974	Republic Government of Zambia	Ministry of Mines and Mineral Resources.

DOCUMENT REVISION CONTROL

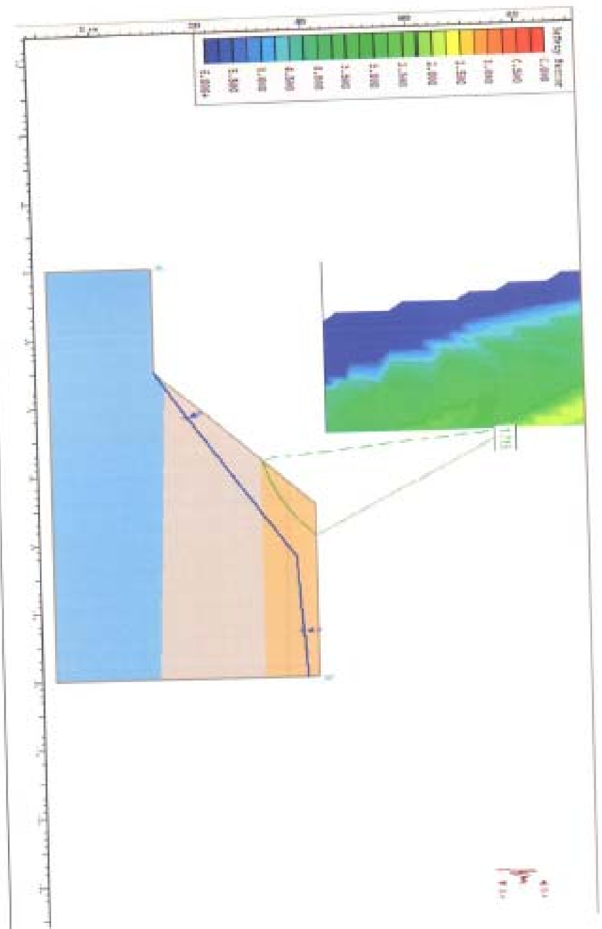
REVISION	PAGE	CHANGE EFFECTED	DATE OF CHANGE

NUMBER	NUMBER/S		
N/A	N/A	N/A	N/A

REVIEW AND APPROVAL SIGNATURES RECORD

REVIEWER ROLE	TITLE	SIGNATURE	DATE
Originator	Drilling, Charging & Blasting		
Reviewer 1			
Reviewer 2			
Reviewer 3			
Reviewer 4	Document Administrator		
APPROVED BY: Mine Manager			

Area J open Pit – Slope Angle 37°, Slope Height 30m



Appendix B: Geotechnical data for area J Open Pit

Appendix C: Crack monitoring data.

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CRACK MONITORING RECORD CARD

DATE	PLOT No.	DEVICE No.	CRACK WIDTH BEFORE DEVICE INSTALLATION (mm)	CURRENT READING(CRACK WIDTH) (mm)	Variance between Initial and Current readings (mm)
24-11-2014	B1 - 6	2	5.0	6.0	1.0
24-11-2014	B1 - 17	3	4.0	4.0	0.0
24-11-2014	B1 - 21	4	6.0	6.0	0.0
24-11-2014	B1 - 25	5	24.0	24.0	0.0
24-11-2014	B1 - 31	6	7.0	7.0	0.0
24-11-2014	B1 - 70	7	6.0	6.0	0.0
24-11-2014	B1 - 71	8	4.0	4.0	0.0
24-11-2014	B1 - 19	10	2.0	2.0	0.0
24-11-2014	B1 - 28	11	5.0	5.0	0.0
24-11-2014	B1 - 29	12	5.0	5.0	0.0
24-11-2014	B1 - 30	13	2.0	2.0	0.0
24-11-2014	B1 - 85	14	4.0	4.0	0.0
24-11-2014	B1 - 87	15	4.0	4.0	0.0
24-11-2014	B1 - 18	17	13.0	13.0	0.0

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CRACK MONITORING RECORD CARD

DATE	PLOT No.	DEVICE No.	CRACK WIDTH BEFORE DEVICE INSTALLATION (mm)	CURRENT READING(CRACK WIDTH) (mm)	Variance between Initial and Current readings (mm)
12-12-2014	B1 - 6	2	5.0	6.0	1.0
12-12-2014	B1 - 17	3	4.0	4.0	0.0
12-12-2014	B1 - 21	4	6.0	6.0	0.0
12-12-2014	B1 - 25	5	24.0	24.0	0.0
12-12-2014	B1 - 31	6	7.0	7.0	0.0
12-12-2014	B1 - 70	7	6.0	6.0	0.0
12-12-2014	B1 - 71	8	4.0	4.0	0.0
12-12-2014	B1 - 19	10	2.0	2.0	0.0
12-12-2014	B1 - 28	11	5.0	5.0	0.0
12-12-2014	B1 - 29	12	5.0	5.0	0.0
12-12-2014	B1 - 30	13	2.0	2.0	0.0
12-12-2014	B1 - 85	14	4.0	4.0	0.0
12-12-2014	B1 - 87	15	4.0	4.0	0.0
12-12-2014	B1 - 18	17	13.0	13.0	0.0

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CRACK RECORD CARD

DATE	PLOT NO.	DEVICE NO.	CRACK WIDTH BEFORE DEVICE INSTALLATION (mm)	CURRENT READING (mm)	Variance between Initial & Current Reading (mm)
23/11/2010	B1 - 1	1	5	0	Nil
23/11/2010	B1 - 4	2	9	0	Nil
23/11/2010	B1 - 6	3	5	0	Nil
23/11/2010	B1 - 17	4	4	0	Nil
23/11/2010	B1 - 21	5	6	0	Nil
23/11/2010	B1 - 25	6	24	0	Nil
23/11/2010	B1 - 31	7	7	0	Nil
23/11/2010	B1 - 70	8	6	0	Nil
23/11/2010	B1 - 71	9	4	0	Nil
23/11/2010	B1 - 85/B1 - 86	10	3	0	Nil

Date	Time	Seismograph ID	Seismograph position	PPV Vector Sum mm/s	Scaled Distance
09-May-12	16:46	11677	SOB VS4 Fan	8.32	10.47
	16:48	11679	KCM Sync Offices	5.99	12.16
	16:48	4104	Kalela Basic School	3.18	15.56
11-May-12	17:10	11677	SOB VS4 Fan	2.07	7.64
	17:10	11679	KCM Sync Offices	1.81	10.75
	17:04	4104	Kalela Basic School	1.02	13.86
12-May-12	16:52	11679	SOB VS4 Fan	3.45	8.49
	16:52	11677	KCM Sync Offices	8.44	16.97
	17:04	4104	Kalela Basic School	2.92	14.43
17-May-12	16:52	11679	SOB VS4 Fan	3.45	10.89
	16:52	11677	KCM Sync Offices	8.44	11.32
	16:47	4104	Kalela Basic School	2.92	14.14
21-May-12	16:56	11677	SOB VS4 Fan	5.37	12.73
	16:56	11679	KCM Sync Offices	8.58	12.31
	16:55	11676	Kalela Basic School	2.54	13.44
31-May-12	16:46	11677	SOB VS4 Fan	4.00	14.14
	16:46	11676	KCM Sync Offices	13.29	6.36
	16:45	4104	Kalela Basic School	2.67	14.29
	16:44	11679	Area J Offices	14.40	7.07
12-Jun-12	16:55	11679	SOB VS4 Fan	29.58	6.22
	16:55	11676	KCM Sync Offices	6.61	9.90
	16:55	4104	Kalela Basic School	3.30	13.15
25-Jun-12	16:55	11679	SOB VS4 Fan	21.66	9.19
	16:55	11676	KCM Sync Offices	1.26	17.54
	16:55	11677	Kalela Basic School	1.47	26.17
11-Jul-12	17:12	11677	SOB VS4 Fan	3.12	14.14
	17:12	11679	KCM Sync Offices	17.31	11.88
	17:12	4103	Kalela Basic School	10.14	14.85
13-Jul-12	17:23	11677	SOB VS4 Fan	2.90	8.77
	17:22	11679	KCM Sync Offices	1.96	14.71
	17:25	4103	Kalela Basic School	1.65	18.67
18-Jul-12	16:43	11676	SOB VS4 Fan	7.00	9.34
	16:43	11677	KCM Sync Offices	2.21	12.31
	16:42	11679	Kalela Basic School	1.53	14.71
23-Jul-12	17:41	11676	SOB VS4 Fan	4.56	9.90
	17:41	11677	KCM Sync Offices	3.67	12.45
	17:41	11679	Kalela Basic School	3.21	13.58
27-Jul-12	16:46	11676	SOB VS4 Fan	15.51	7.64
	16:46	11677	KCM Sync Offices	0.00	14.14
	16:46	11679	Kalela Basic School	0.00	16.97
02-Aug-12	17:21	11679	SOB VS4 Fan	25.28	7.78
	17:21	11677	KCM Sync Offices	8.82	12.16
	17:21	11676	Kalela Basic School	3.07	16.27
16-Aug-12	16:50	11677	SOB VS4 Fan	8.36	8.49
	16:50	11676	KCM Sync Offices	10.45	12.02
	16:51	11679	Kalela Basic School	2.51	14.00
21-Aug-12	16:45	11676	SOB VS4 Fan	3.83	9.19
	16:44	11679	KCM Sync Offices	1.31	11.32
	16:45	11677	Kalela Basic School	0.00	14.57
27-Aug-12	16:45	11676	SOB VS4 Fan	1.43	7.50
	16:45	11677	KCM Sync Offices	1.40	13.44
	16:41	11679	Kalela Basic School	1.20	18.10
04-Sep-12	16:45	11676	SOB VS4 Fan	14.56	7.50
	16:45	11677	KCM Sync Offices	3.33	9.90
	16:41	11679	Kalela Basic School	2.37	12.16
06-Sep-12	16:45	11676	SOB VS4 Fan	16.71	8.49
	16:45	11677	KCM Sync Offices	0.00	10.61
	16:41	11679	Kalela Basic School	1.59	13.86
10-Sep-12	16:59	11679	SOB VS4 Fan	3.36	14.43
	17:00	11676	KCM Sync Offices	3.16	10.75
	17:00	11677	Kalela Basic School	1.55	12.45
13-Sep-12	16:44	11676	SOB VS4 Fan	8.64	7.92
	16:45	11677	KCM Sync Offices	4.07	11.03
	16:45	11679	Kalela Basic School	1.67	12.87
21-Sep-12	16:44	11676	SOB VS4 Fan	6.13	9.34
	16:45	11669	KCM Sync Offices	1.44	12.16
	16:45	11677	Kalela Basic School	1.42	12.73
25-Sep-12	16:46	11676	SOB VS4 Fan	18.32	12.73
	16:47	11669	KCM Sync Offices	2.05	12.31
	16:47	11677	Kalela Basic School	2.22	14.00