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CONTROL MEASURES AND SAFETY PRACTISES USED FOR FLY ROCK ISSUES IN MINING DUE TO BLASTING

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ABSTRACT
Blasting involves the breaking of rocks using explosive that rapidly change due to chemical reaction forming huge volumes of gases with high pressure and temperature causing kinetic energy. The blasting process is primarily a rock–explosive interaction that entails application of pressure generated by detonation of explosives, on rock mass, over a few milliseconds. This rock–explosive interaction results in rock breakage and heaving of the broken rock mass (muck). In comparison to the mechanical methods that rely predominantly on the compressive breakage, blasting exploits the tensile strength of the rock mass. This is probably the reason that blasting is still the most prevalent and economical method for rock breakage. Blasting, in general, results in 'desired' and 'undesired' outcomes that may be 'regular' or 'random' in nature. Any mismatch between the energy available and the work done will increase the adverse or undesired blast results like excessive throw and fly rock. Fly rock and excessive throw occur due to deviations in blast design execution, use of excessive explosive energy than the required levels to fragment and throw the rock mass, and/or presence of rock mass features, not accounted for during blasting
Key words: Drilling, Blasting, Fragmentation, Explosives, Fly rocks, Rifling, Catering, Injuries, Mucking.

1. Introduction

Blasting involves the breaking of rocks using the chemical energy in the explosive. The blasting process is primarily a rock explosive interaction that entails application of pressure generated by detonation of explosives, on rock mass, over a few milliseconds. This rock explosive interaction results in rock breakage and heaving of the broken rock mass and muck. In comparison to the mechanical methods that rely predominantly on the compressive breakage, blasting exploits the tensile strength of the rock mass. This is probably the reason that blasting is still the most prevalent and economical method for rock breakage used in both Mining and Construction industries. Blasting, in general, results in 'desired' and 'undesired' outcomes that may be 'regular' or 'random' in nature as shown in table1.1.1. These also form the objectives of the mine mill fragmentation system MMFS. Any mismatch between the energy available and the work to be done will increase the adverse or undesired blast results like excessive throw and fly rock. Fly rock and excessive throw of muck or heavy sand particles occur due to deviations in blast design execution, use of excessive explosive energy than the required levels to fragment and throw the rock mass, and/or presence of rock mass features, not accounted for during blasting or poor preparation of the blasting field. The said rock mass and blast design anomalies favour the hallow out of high-pressure gases emanating from the blast holes in the direction of the weakest zone and result in fragments travelling unwanted distances than desired. Such fragments are called 'fly rock'. These Fly Rock can be as deadly as a bullet or a missile in term of destruction and devastation.

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Fly rock is one of the crucial issues in bench blasting, as it is not only a safety concern but also affects the productivity. Percentage of accidents occurring due to fly rock Table1.1.2, justifies its importance irrespective of the fact that the problem is seldom reported.
For instance, Mishra, (2013) reported 17.35% in total accident due to explosive in both coal and non-coal mines of deep hole blasting, Adhikari (1999) reports that 20% of accidents that were related to fly rock occurred in mines in India, Mishra, (2003) reports more than 40% of fatal and 20% of serious accidents resulting from blasting occur due to fly rock in mines in India. It can be noted that almost 70% of all injuries is directly contributing to the fly rock and lack of blast area security. So the prediction of fly rock and its control is still elusive.
Fly rock, arising from open-pit blasting, still eludes rock excavation engineers, despite a reasonable understanding of throw. Fly rock distance predictions have witnessed a refocus in the past few years due to want of probable solution. Such attempts also have raised certain pertinent questions that need to be answered in order to develop a proper understanding of thefly rock phenomenon, which is expected to facilitate a better investigation regime for forthcoming R&D efforts on its prediction.

2. Literature Survey

Despite the fact that fly rock consumes only 1% of the explosive energy used in a blast, it is more serious in nature, in comparison to ground vibrations, as it can inflict damages, injuries and fatalities. Several authors have reported that 20–40% of the blasting related accidents are due to fly rock. The research on fly rock is, however, abysmal and considering the above-mentioned facts, the problem deserves more attention from the researchers.
Hence, it is essential to identify the reasons for lack of R&D on fly rock. Under or non-reporting of fly rock probably due to heavy penalties imposed by regulatory agencies, high cost of experimentation, and the random nature of fly rock are some of the reasons identified for inadequate R&D on fly rock. Such limitations are the cause for low confidence with regard to the existing predictive models of fly rock distance.
Ash (1963) investigated the effect of stemming material as well as the length of stemming material on fragmentation size. It is realized from their experiment that stemming length of 70 percent of the burden dimension is good and it has a sufficient control over production of objectionable air blast and fly rock from the Collar zone. If there are number of structural discontinuities the collar region scattering of energy may reduce the stress levels to the extent that inadequate breakage of the top rock results where discontinuities are pronounced. The field tests indicate that efforts to keep explosive gases from entering the stem and thereby reducing
Langefors (1965) demonstrated from laboratory model scale tests that ratio exceeding three for simultaneously fired charges in a single row gave their fragmentation. This was observed by reducing the conventionally used burden. For the same model tests with multiple rows of charge fired together, but rows of holes delayed relatively resulted in good fragmentation effective stem wall friction Improved stem performance.
Ash (1969) observed the variable characteristics of spacing by model test made from block of cement mortar, acrylic and dolomite rock. From the result of these tests, it was concluded that the larger spacing could be used because of enhancement of stress wave energy in simultaneously blasted holes. However, this conclusion is not acceptable because the conventional burden (i.e. 50 to 100 times the charge diameter) is used, therefore, large spacing are not suitable. It was concluded that the charge length were affecting the hole spacing.
Gregory (1973) stated that whenever operators try to increase the hole spacing more than twice that of the burden, the problem of incomplete breakage occur and results in a poor fragmentation.
Hagan (1973) had recommended that even larger hole spacing can be used, whereas the Closer hole spacing can be possible when joints on most dominating discontinuity across the free face

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Person and Ladegaard Pedersen (1973) verified successfully wide hole spacing technique on the production scale blasting. Better fragmentation results were achieved when the hole spacing as large as eight time of the burden was used in laminated limestone quarry. The method suggested became popular in early 1970's and is known as Swedish Wide Spacing Technique.

Bhandari (1975) demonstrated this hypothesis on model scale test using cement mortar blocks. He recommended small burden with larger hole spacing preferably 3 to 4. After this ratio separate hole breakage occurred. It was explained that reduced burden allowed better utilization of explosive energy. He had shown that jointed rock increase in burden given coarser fragmentation.

Ash and Smith (1976) showed that the spacing twice the burden gave better fragmentation with delay timings. He also observed that when the ratio of spacing increase 3 to 4 times the burden unbroken rock in between the holes Occur.

Knoya and Davis (1978) recommended that the crushed and sized angular rock fragments works best as stemming material.

Hagan (1983) suggested that smaller burden is required when the distance between discontinuities is larger. He also stated that the spacing equal to the burden gave adequate results.

Singh & Sarma (1983) and Sigh & Sastry (1987) observed that the orientation of joints have influence on blasting results because the optimum burden for variable orientations was different. But no consideration is given to other blasting parameters in relation to orientation of joints. They also observed that the hole spacing ratio between 3.0 to 4.0 provide optimum fragmentation results.

Verma (1993) advocated that performance rating of explosives has become a primary need because of the growing requirement and competition mining industries. In experiments, the usually accessed parameters are the strength though there is no such parameter still to compare the performance index of the explosives. At present, the only way out is to compare the lab results and the company or manufacturers claimed results about the explosive properties. The ratio must be 1 but due to factors it must be close to it, if not equal. By the ratio the explosives can be classified into different categories.

Biran (1994) observed that many empirical formulas have been used over 200 years for selection of proper charge size and other parameters for good fragmentation. But for blasting efficiency and uniform fragmentation, there should be uniform distribution of explosives in holes. The blasted material heap should have more throw for loaders and hydraulic shovels and more heave for rope shovels and loaders. For good economic blasting the holes should not be deviated from the plan. It requires meticulous planning on the use of site mixed slurry explosives, stemming of holes with mechanical means and blasting after pilot blasting of holes to access various details.

Adhikari and Venkatesh (1995) suggested that drilling and blasting cost in any project can be as high as 25% of the total production cost. So the design and implementation of a blast must be given some priority. By the blast design parameters optimization the profitability would increase. For this the study of the existing practice was done followed by pre-blast, in-blast, and post-blast survey. Then the data were analyzed and a model was interpreted. All the parameters were then compared and worked on for the best suiting result. They observed that to achieve a certain degree of refinement in blast design, scientific and systematic approach is needed. With instruments like VOD probes, laser profiling system, etc the monitoring becomes easier, efficient and cost effective.

Singh and Dhillon (1996) pointed out that to optimize the cost in an opencast mine, there is a need to optimize the drilling and blasting parameters. In case of blasting operations; for optimization of explosives, the first step is to optimize the booster cartridges and cast boosters along with column explosives. The booster for initiation of the whole column of the explosive must be reduced by experimentation. It saves a large share of expenditure. By the use of a total top initiation system instead of a down the hole for bottom initiation reduces the use of detonating fuse. By use of air decks, the explosive cost can be saved to some extent. By introduction of top-initiation system and

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non-electric initiation the desensitization effect has been completely eliminated, thus enabling optimum utilization of explosive energy.

Uttarwar and Mozumdar (1996) studied the blast casting technique that utilizes explosive energy to fragment the rock mass and cast a long portion of it directly into previously worked out pits. The technique depends on factors like bench height and helps in efficient trajectory of thrown rock and so in the height to width ratio. This technique is most effective with explosives that maximize ratio of heave energy to strain energy. Higher powder factor supports the technique. Optimal blast-hole diameter and inclination, stemming and decking method used the burden to spacing ratio, delay intervals and initiation practices help in effective blasting.

Thote and Singh (1997) observed that the blasting results of fragmentation are influenced by various factors. For example, rock strength decreases the fragmentation; it is also affected by the blast ability index, porosity and the geological disturbances. In case of discontinuities, the shock wave gets reflected causing higher attenuation at a smaller area. This leads to boulder formation. All these factors need a detailed study and in-field experiments to judge the blasting parameters and decide the quantity of explosives to be used to avoid boulder formation or enable good fragmentation.

Karyampudi and Reddy (1999) observed that the toe formation has always been a drawback in the opencast mines. There are certain factors that result in toe formation like the burden and spacing, size of drill block, condition of drill holes and condition of face before blasting; charging of blast holes and the type of initiation are the factors that can be avoided. But the strata variation, fractured strata and watery holes are unavoidable. So it should be tried to achieve a drill block where the unavoidable factors are non-existent. It is marked with crest, burden, spacing. They were of the view that blast holes must be charged as per proper charging pattern with appropriate percentage of booster, base and column and holes by charging from bottom initiation leads to toe-less blasting.

Pal and Ghosh (2002) studied the optimization of blasting pattern implemented at Sonepur Bazari opencast project for control of ground vibration, noise or air over pressure and fly rock with improved production and productivity. Their study revealed that by proper design of blast parameters the desired results in fragmentation, vibration were achieved where as fly rock needed good supervision. They recommended use of non-electric initiation system instead of detonating fuse; this increased the cost but gave back in productivity reducing chances of misfire, flies rock and achieved proper fragmentation with reduced sub-grade drilling. The direction of invitation was also important. They suggested a blast design for proper balance between environmental aspects and productivity criteria.

Pradhan (2002) studied the trend of blasting in Indian opencast mines and observed that it has been changing with requirements. There are new explosives, use of electronic delay detonators for accurate delays, blast design as per Physico-mechanical properties of rock, initiation of shock tubes, air-deck system, blast performance monitoring, cost-effective explosive formulations, etc. Now-a-days GPS is also used for blast planning. He pointed out that in spite of optimum blasting pattern and scientifically chosen explosives, still a lot has to be done for blast management and control.

Nanda (2003) advocated that operation research facilitates in describing the behavior of the systems, analyzing the behavior by constructing appropriate models and predicting future behavior by using these models. They studied the Queuing, Markov and Reliability models and concluded that with the help of operations research an appropriate mathematical model for situations, processes and systems can be developed. The model can then be tested and operated by changing the variable values to implement optimization of parameters. They were also of the view that in the present era optimal use of resources are essential and operation research can facilitate to take proactive decisions to make the system profitable and competitive.

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3. METHODOLOGY

3.1BLAST DESIGN PARAMETERS

The following are the some of the important parameter which generally govern for blast design

3.1.1.1Physico-mechanical properties of rock:

Here type of the rock, dynamic tensile strength, tensile strength, compressive strength,young's modulus, Poisson's ratio, density and hardness of the rock mass, presence of
Discontinuities, bedding plane and joints, etc. are very important.

Geology

Pit geometry

Under this heading thickness of coal seam or ore body and bench height, over burdenbench height, bench slope angle, strip width, height to width ratio, and length to width ratio are generally considered.

Explosive characteristics

Factors generally considered under this heading are type of explosive, type of booster,bulk strength, energy release per unit mass of explosive, detonation pressure, explosionpressure, ratio of decoupling, strength of explosive used, time taken for explosive wave totravel to the free face and back, volume of gaseous product per unit mass of theexplosive, velocity ofdetonation, velocity of explosion propagation, explosion wavelength, weight strength, number of spalls that an explosive wave may produce, length, diameter and weight of the cartridge, loading density, bottom charge and column chargedensity, etc. are very important. Characteristics of blasting accessories – type, thermalproperties are also important.

3.1.1.2 Powder factor

The size of the fragmented rock should match the bucket size of the excavator and alsothe grizzly size of the primary crusher.
Length of stemming column, the size and quality of stemming
Angle drilling
Amount and direction of throw requirement and problems of fly rock.
Requirement of muck profile
Vibration level
Presence of water
Some of the important parameter considered in blast design; given above are discussed in details as follows

3.2.2 Bench Geometry

3.2.2.1 Bench Height (H): The bench height is the vertical distance between each horizontal level of the pit. Unless geologic conditions dictate otherwise, all benches should have the same height.The height will depend on the physical characteristics of the deposit; the degreeof selectivity required in separating the ore and waste with the loading equipment; the rate ofproduction; the size and type of equipment to meet the production requirements; and the climatic conditions. The elements of a bench are illustrated in the above fig2.2**.**

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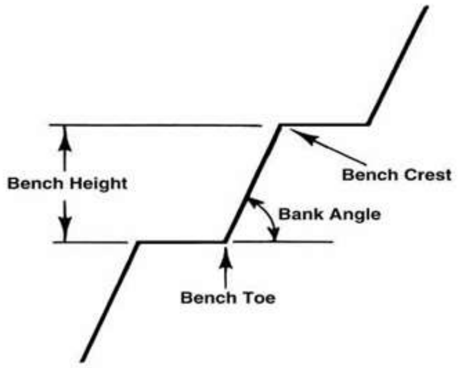

Fig 1Bench cross section

The bench height should be set as high as possible within the limits of the size and type ofequipment selected for the desired production. The bench should not be so high that it willpresent safety problems of towering banks of blasted or unblasted material or of frost slabs in winter. The bench height in open pit mines will normally range from 15 mts in large copper mines to as little as 1 m in uranium mines. But in special case such as rip-rap blasting height can be reached 20 mts. The bench height is directly related to degree of heaping and spreading of material broken by blasting, thus, directly affecting displacement requirement to accomplished by round blasting. The height also limits the maximum and minimum charge diameters and drill diameters. The most economical may be also determined by the drill penetration rate; whenever penetration rate decreases significantly, it is generally uneconomical to drill deeper. High faces pose the problem of considerable bit wander, especially with small diameter hole. The deviation of blast hole places a limit on the maximum allowable bench height. The bench height is also highly depend on capacity of loading equipment The following are some of the factor that should be considered in the selection of the bench height:

Optimum blast Hole diameter increases with the height: In general an increase in blastHole diameter decreases in drilling costsIn some cases the bench height is limited by the geology of the ore deposit due to imperatives of the ore dilution of the control and safety measures .

Bench Width (W): There is a minimum bench width, measured horizontally in a directionperpendicular to the pit wall. For each bench height and set of pit operation conations whosevalue is established by the working requirements of the loading and hauling equipment. Thewidth also must be such so that to ensure stability of excavation both before and after blasting, because each blast effectively reduced the restraint sustains the pit walls at higher elevation. Because of the limit set by requirements for equipment operating room and bank stability, there is a maximum width that should not be exceeded by any blast.

3.2.2.2 Blast Geometry:

Drilling Diameter (D): The hole diameter is selected such that in combination withappropriate positioning of the holes, will give proper fragmentation suitable for loading,transportation equipment and crusher used. Additional factor that should be considered in the determination of the hole diameter are:

Bench height

Type of explosive

Rock characteristics

Average production per hour

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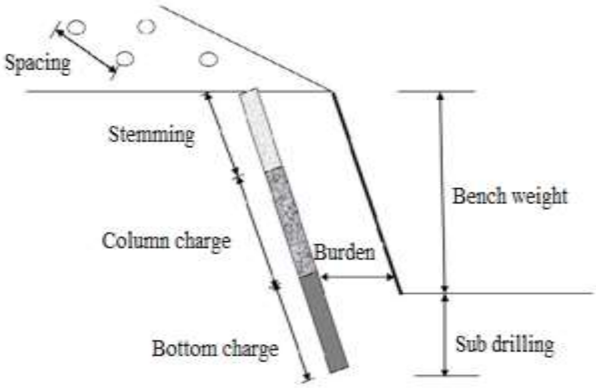

Fig.2. General layout its having different parameters of blast design

The drilling and blasting will become economical with increase in diameter. When the blast hole diameter is increased & the powder factor remains constant the large blast hole pattern gives coarser fragmentation. By keeping burden unchanged & elongating spacing alone the problem can be overcome. When joins 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 boulders has a blast hole, which necessitates the use of small diameter blast holes. Hole diameter varies from 35 in small benches up to 440 mm in large benches. In India 100-150 mm blast hole diameter are used inlimestone mines,150-270 mm in coal mines & 160 mm or above blast hole are used in iron ore mines is used. Langefors and Kihlstrom suggested that the diameter be kept between 0.5 to 1.25 percent of the bench height.

3.2.2.3 Sub Drilling (J):

To avoid formation of toe in bench blasting the blast hole are drilledbelow the floor or grade level. This is termed as sub grade drilling or sub drilling. Ifthe toe formation will not avoid it may increase the operating costs for loading, haulingequipment. The optimum effective sub drilling depends on

The structural formation

Density of the rock

Type of explosive

Blast hole diameter & inclination

Effective burden

Location of initiators in the charge.

It is usually calculated from blast hole diameter when vertical blast holes are drilled. The subdrilling of the first row reaches value of 10D to 12D .About 10% of sub drilling gives betterfragmentation in the rock mass and lesser ground vibration. In generally sub drilling should be0.3 times the burden. Under different toe conditions sub drilling may be up to 50 percent of theburden. A relation is also shown in the fig 2.4 below.

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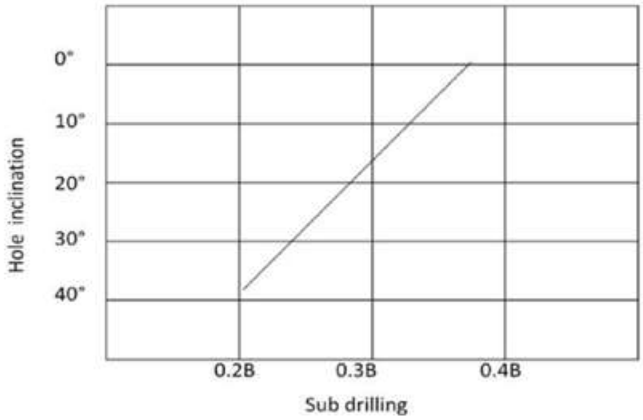

Fig 3 Sub drilling with inclination of blast hole

Excessive sub grade drilling causes more vibrations, under fracturing at the bottom anddepressed floor conditions. It should be avoided since it:

a. waists drilling and explosives expenditure
b. increased ground vibration level
c. may cause undesirable shattering of the pit floor
d. Increase the vertical movement of the blast.

3.2.2.4 Stemming (T)

The primary function of the stemming is to confine the gas produced by the explosive until they have adequate time to fracture and move the ground. A suitable stemming column of suitable length and consistency enhances fracture & 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, fly rock from the collar zone. When the burden has a high frequency ofnatural crack and planes of weakness relatively long stemming column can be used. When the rock is hare and massive the stemming should be shortest which will prevent excessive noise, air blast and back brake.For blast hole diameter in the 230-380 mm range, angular crushed rock in the approximate size of 23 to 30 makes a very effective stemming column larger fragments tends to damage the detonating curd and the detonator lead wire.Dry granular staining is much more efficient then material behave like plastically or tend to flow. In coal blast inert stemming material should be used rather than coal cutting. In multi row blast when the mean direction of rock movement tends to more and more towards the vertical with successive rows a longer stemming column is often used in the last row to avoid over break. When large stemming is kept in rocks with discontinuities large boulders may result. In such cases pocket charge or satellite charge are recommended.

3.2.2.5 Blast Hole Inclination (β)

In recent year attention has been given by open pit operators to the drilling of blast holes up to 20 degree 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 Drilling of next bench is easier Air blast and fire 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 degree 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.

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This is one of the most critical parameter in designing of blast. It is the distance from a charge axis to the nearest free face at the time of detonation. As the boreholeswith lower delay periods detonates, they create new free faces. As a result the effective burden will depend upon the selection of the delay pattern. When the distance between discontinuities is larger, smaller burden is required. A relationship between burden with blast hole diameter has been shown in the fig 2.5 below.

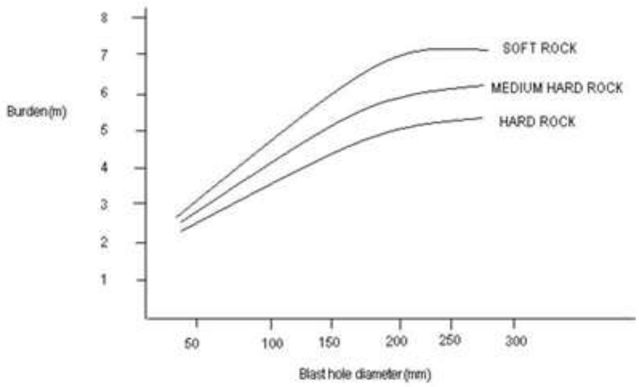

Fig 5 Size of burden in function with drilling diameter

3.2.2.6 Spacing (S): Spacing is an important parameter in blast design. It is defined as the distance between any two adjacent charges in the same row and it controls mutual stress effect betweencharges. Spacing is calculated as a function of burden, hole depth, relative primer locationbetween adjacent charges and depends upon initiation time interval. Over past several decades inmost mining operations the spacing distances have been decided in relation to burden. The valueof the spacing to burden ratio (S: B) which has been commonly used in different formulas liesbetween 1 and 2. From the production scale test with the spherical charges breaking to cratergeometry, many workers suggested that the spacing be kept about 1.3 times the burden. Whenthis ratio increases more than 2, unexpected results were found.

3.2.2.7 Powder Factor

The powder factor is defined as the explosive necessary to fragment 1 m^3 of rock. This equationcan also be defined as the amount of explosives over the cubic yards of material desired to beblasted.Kg of explosive used/volume of material blasted = kg/ m^3It is the opinion of many specialists this is not the best tool for designing a blast, unless it isreferring to pattern explosives or expressed as energetic consumption. The size of the fragmentedrock should match the bucket size of the excavator and also the grizzly size of the primarycrusher. It can be also expressed in ton/kg.

A relation of average fragmentation size in function with burden and powder factor is shown in the fig 2.6 below.

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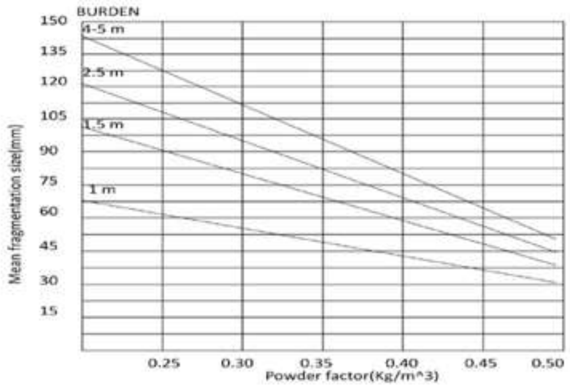

Fig 6Average fragmentation size in function with burden and powder factor

4. CONTROL MEASURES FOR FLY ROCK SUPRESSION Explosive Selection Criteria

This selection plays a major role in the blast design and the blast results that will occur. Anexplosive has many characteristics that need to be analyzed in making this decision. Theseinclude: minimum diameter in which detonation will occur, the ability to resist water and waterpressure, generation of toxic fumes, ability to function under different temperature conditions,input energy needed to start reaction, reaction velocity, detonation pressure, bulk density, andstrength. Other things the technician must consider are: explosive cost, charge diameter,characteristics of the rock to be blasted, volume of the rock to be blasted, presence of water,safety conditions, and supply problems.

4.1 Types of Explosives

The explosive used as the main borehole charge can be broken up into four categories. Thesecategories are dynamite, slurries, emulsions, and dry blasting agents because all the categoriesmentioned contain explosives that will detonate, they are considered high explosives.

4.1.1 Dynamite

In Sweden in 1867, Alfred Nobel discovered how to create dynamite. Mostdynamites are nitroglycerin based. Being the most sensitive of all explosives used; dynamite ismore susceptible to accidental initiation. There are two major subclasses of dynamite, Granulardynamite and gelatin dynamite. Granular dynamite is a compound which uses nitroglycerin as itsexplosive base. Gelatin dynamite uses a mixture of nitroglycerin and nitrocellulose. Thisproduces a waterproof compound.

4.1.2 Slurry Explosives

Slurry explosives, also called water gels, are made up of ammoniumnitrate partly in an aqueous solution. Depending on the rest of the ingredients slurries can beclassified as a blasting agent or an explosive. Slurry blasting agents contain non explosivesensitizers or fuels such as carbon, sulfur, or aluminum. These blasting agents are not capsensitive. On the other hand slurry explosives contain cap- sensitive ingredients such as TNT andthe 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 dryblasting agent are used in the same borehole. Most of the charge will come from the dry blastingagent. Boosters placed at regular intervals may improve fragmentation. The disadvantages ofslurries include higher cost, unreliable performance, and deterioration with prolonged storage.

4.1.3 Emulsions

An emulsion is a water resistant explosive material containing substantialamounts 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 oiland are stabilized by emulsifiers. To achieve more sensitivity within the emulsion voids areadded. These voids may include small nitrogen bubbles or micro-spheres made out of glass.Sensitivity of an emulsion decreases as the

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density increases. To adjust the density and strengthof an emulsion dry products are used. Some examples being, powdered aluminum, gasifyingagents to reduce density. It is therefore necessary to work above the critical diameter and usepowerful initiators. If the emulsion is not cap sensitive it is considered a blasting agent.Emulsions have high energy, reliable performance, excellent resistance to water, and relativeinsensitivity to temperature changes. The direct cost of an emulsion explosive is higher but this isoffset by time saved in loading and a reduction in nitratecontent of broken muck. Some otheradvantages to using emulsions in rock blasting include: a lower cost, excellent water resistance,high detonation velocities, and it's very safe to handle and manufacture.

4.1.4Dry Blasting Agents

Dry blasting agents are the most widely used explosive used in theworld. ANFO is the most common dry blasting agent. An oxygen balanced mixture of ANFO isthe lowest cost source of explosive energy today. To increase energy output, ground aluminumfoil is added to dry blasting agents. A downfall of this however, is that the cost is increased. Twocategories make up dry blasting agents: cartridge blasting agents and bulk ANFO. Bulk ANFO iseither blown or augured into a blast hole from bulk truck. These blasting agents will not functionproperly if placed in wet holes for extended periods of time. Cartridge blasting agents however,are made for use in wet blasting holes. Cartridge blasting agents are available with densities thatare greater than that of water if you would like them to sink, or less than that of water if youwould like them to float.Heavy ANFO is made up of mixtures of ammonium nitrate prills, fuel oil, and slurries.

4.2 Explosive Characteristics

4.2.1 Physical properties

There are many physical and chemical attributes and properties that must be considered in the selection of explosives. These factors affect six characteristics of the explosives: sensitiveness, water resistance, water pressure tolerance, fumes, and temperature resistance.

4.2.2 Sensitiveness

It is the characteristic of an explosive which defines its ability to propagate a stable detonation through the entire length of the charge and controls the minimum diameter for practical use. By determining the explosive's critical diameter you can measure the sensitivity of the explosive. The critical diameter is the minimum diameter of explosive column which will detonate reliably.

4.2.3 Water Resistance

Water resistance is the explosive's ability to withstand exposure to water without suffering detrimental effects in performance. Explosives have two types of water resistance: internal and external. Internal water resistance is water resistance provided by the composition of the explosive. External water resistance is the water resistance is provided by the packaging or cart ridging in which the explosive is placed. Water is harmful to the explosive because it can dissolve or leach out some of the explosive ingredients.

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Table 1 List of important physical properties of explosives

Type	Water Resistance	Quality of Fumes	Temperature Resistance between 0° F – 100° F
Granular Dynamite	Poor - Good	Poor – Good	Good
Gelatine Dynamite	Good – Excellent	Fair to Very Good	Good
Cartridge Slurry	Very Good	Good to Very Good	Poor Below 40° F
Bulk Slurry	Very Good	Fair to Very Good	Poor Below 40° F
Emulsion	Very Good To Excellent	Good to Very Good	Good
Poured ANFO	Poor	Good	Poor above 90° F
Packaged ANFO	Very Good	Good to Very Good	Poor above 90° F
Heavy ANFO	Poor To Very Good	Good	Poor above 40° F

4.2.4 Water Pressure Tolerance

Water pressure tolerance is the explosive's ability to remain unaffected by high static pressures. These high pressures will occur when you have deep boreholes that are filled with water. Explosives may be densified and desensitized in these conditions. Some examples of explosives that have big problems with water pressure tolerance are slurries and heavy ANFO.

4.2.5 Fumes

The fume class of an explosive is a measure of the amount of toxic gases produced in the detonation process. The most common gases considered in fume class ratings are carbon monoxide and oxides of nitrogen. Commercial explosives are made to get the most energy out as possible while minimizing these gases. This is done by balancing the oxygen in chemical reaction of the explosive. This alone doesn't solve the problem of toxic fumes. These can still occur due to environmental conditions. The Institute of Makers of Explosives (IME) has adopted a method of rating fumes and the test is conducted by the Bichel Gauge method. The cubic meter of poisonous gases released per 200 grams of explosive are measured. If less than 0.05 m^3 of toxic fumes are produced, the fume class rating would be 1. If 0.05 to 0.1 m^3 is produced, the fume class rating is a 2, and if 0.1 to 0.2 m^3 of toxic fumes is produced, the fume class rating is 3.

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4.2.6 Temperature Resistance

The performance of explosives can be affected a great deal if they are exposed to extremely hot or cold conditions. Under hot conditions, above 18 degrees C, many explosive compounds will slowly decompose or change properties. Shelf life will also be decreased. Cycling can occur when you store ammonium nitrate blasting agents in temperatures above 18 degrees C. This will affect not only the performance of the explosive, but also the safety.

4.2.7 Performance Properties

After considering all of the environmental factors, the performance characteristics of explosives must be considered in the explosive selection process. These characteristics include: Sensitivity, velocity, detonation pressure, density, and strength.

4.2.8 Sensitivity

The sensitivity of an explosive product is defined by the amount of input energy required for the product to detonate reliably. Other common names for this are the minimum booster rating, or minimum priming requirements. While some explosives require very little energy to detonate reliably with just a blasting cap, others require the use of a booster or primer along with a blasting cap to get a reliable detonation. Factors such as water in the blast hole, inadequate charge diameter or temperature can affect the sensitivity of an explosive. Sensitivity of an explosive defines its primer requirements, primer size, and energy output. When reliable detonation fails to happen, the amount of fumes increase, and ground vibration levels tend to rise. Sensitivity is also the measure of the explosive’s separation distance between a primed donor cartridge and an unprimed receptor cartridge, where reliable detonation transfer will occur. Hazard sensitivity is the explosive’s response to accidental addition of energy, an example being a fire.

4.2.9 Velocity

The speed at which a detonation occurs through an explosive is called the detonation velocity. Detonation velocity is important to consider only on explosive applications where a borehole is not used. Detonation velocity is used to determine the efficiency of an explosive reaction. If it is suspected that and explosive is performing subpar then you can test the detonation velocity. If this measured velocity is significantly lower than its rated velocity the explosive is not performing as should be expected. The greater the detonation velocity the more the breakage will occurs. Factors that affect the detonation velocity include: charge density, diameter, confinement, initiation, and aging of the explosive.

Conclusion

The damages and injuries that are cause by the fly-rock and their effect are devastating especially when there are softer and loosely packed earth strata. Therefore, it is important to identify those engineering design parameters and control measure that could be used for controlling fly-rocks no matter how loosely packed or soft/hard rock we are dealing and this could be achieved only by using systematic engineering control measures and the best part is there are only two activities and procedure that need studious calculation and effort namely Drilling Pattern and Diameter of the drill hole and Blasting pattern and the type of explosives and delay techniques used. The parameters that we have worked on in Phase II in order to come to find a less common platform on best methods that could be adopted to avoid fly-rocks are starting with theEmpirical Equations that Supports Blast Designs which could be used by engineering’s to reduce fly-rocks followed by the best suitable surface blast design that could be adopted which will optimise only the fly-rock fragmentation and the most suitable drilling parameters that would enable optimum use of explosive for a good throw in the bench having heavy or medium fragmentation of the rocks. The analysis of the fly rock distances for the blasts conducted during the study period reveal that the computed safe distance is 135m for maximum face velocity of 30m/s. The measured fly rock distance during the field investigation was

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within 50m, it is suggested that a distance of 135m can be considered safe from fly rock point of view. The other factor is the direction of the drilling hole to make sure proper toe is obtained while good fragmentation is obtained. It is very important that our hypothesis results could be predict which will be possible by establishing a derivation method where fly-rock could be predicted and appropriate safety actions could be taken and this could be validated by conducting and recording the field investigation and check if the designed parameters are optimum followed by Fly-rock analysis and chart out an observation based on pre-blast and post blast observation and this will help in prediction of the fly-rock distance so that adequate safety methods and avoidance procedure could be adopted.

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