DESIGN OF SURFACE BLASTS- A COMPUTATIONAL APPROACH

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE
REQUIREMENTS FOR THE DEGREE OF

Bachelor of Technology
In
Mining Engineering

By

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Under the Guidance of

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CERTIFICATE

This is to certify that the thesis entitled **"*SURFACE BLAST DESIGN-A COMPUTATIONAL APPROACH*"** submitted by Sri Ashutosh Mishra, Roll No. 10505029 in partial fulfillment of the requirements for the award of Bachelor of Technology degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any other University/Institute for the award of any Degree or Diploma.

Date: (**Kaushik Dey)**

ACKNOWLEDGEMENT

My heart pulsates with the thrill for tendering gratitude to those persons who helped me in completion of the project.

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ABSTRACT

A major part of mineral production comes from surface mining and there has been a rapid growth in this sector with the deployment of high capacity equipment .Increased production can be achieved from large capacity surface mines using heavy earth moving machineries. These machineries involve high capital cost, and thus, the mining engineers should plan to achieve the best performance from these machineries. Performance of them, especially the excavating and transporting equipments are largely depending on the blast results, particularly, fragment size, distribution and muck profile. Thus, proper blast design is a vital factor that affects the cost of the entire mining activities. Various approaches to blast design for surface mines have been reviewed to understand the present state of knowledge in this field. The blast design approaches such as trial and error and cratering are not suitable for large scale blasts in surface mines. Till date, the blast design for a particular mine is established through trial blasts. The blast designer may make use of available computer aided models for prediction of fragmentation, muck pile profile and vibration. The empirical method continues to be the most common way to calculate the design parameters. Nevertheless, an integration of empirical method, computer modeling and instrumented field trials effectively contributes to the state-of-the- art in blast design. The use of computational approach is meagre. In this paper, the controllable and uncontrollable parameters, which have significant effect on surface blast design, are identified. Based on the model proposed by Langefors and Kihlstrom (1978), a computer model is prepared. The computer model was initially developed in C++ & then built in java platform with the help of software NetBeans IDE 6.5.The developed model is tested for a coal and an iron ore mine and is found reasonable accurate. A data base is also available with the software to make it more useful and less time consuming and user friendly.

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Introduction

1.1 Overview:

Mining began with Paleolithic people, perhaps 30, 0000 years ago [11], during the Stone Age and has been the backbone for the sustainable development of any country. Some of the earliest known mines were those developed by the Greeks in the sixth century B.C. Basically mining operation is divided into two section; underground mining, surface mining. The vocabulary of underground mining has developed over several centuries. Underground mines are excavated using a variety of methods. Room-and-pillar or Board-and-pillar mining is the excavation of large open rooms supported by pillars, where as Long wall mining is a form of underground mining widely used in the coal industry for more production, where a coal seam is completely removed using specialized machines, leaving no support and allowing the overlying rock to slowly subside as the seam is mined. Different stoping method has been adopted for the extraction of metal from underground metal mines. Surface mining may be less expensive and safer than underground mining as well as it accounts a higher production.

With the advancement of civilization, the requirement of different minerals has increased significantly and to meet this demand, large surface mines with million ton production targets are established. The basic aim in mining operation is to achieve maximum extraction of minerals with profit, environmental protection and safety. A rapid growth in this sector with the deployment of high capacity equipment has been observed since last 30 years. Improvement in production has been achieved with the help of large capacity earth moving machineries, continuous mining equipments, improved explosives and accessories, process innovations and increased application of information and computational technologies. These machineries involve high capital cost, and thus, the mining engineers should plan to achieve the best performance from these machineries. Performance of them, especially the excavating and transporting equipments, are largely influenced by the blast results, particularly, fragment size, distribution and muck profile. Thus, proper blast design plays a vital role on the cost of the entire mining activities. In spite of introduction of continuous rock cutting equipments, drilling and blasting continue to dominate the production due to its applicability in wide geo-mining condition.

Therefore, to minimize the cost of production, optimal fragmentation from properly designed blasting pattern has to be achieved. Large fragments adversely affect the loading and hauling equipments and increase the frequency of sorting of oversize boulders and secondary blasting, thereby increasing the cost of mining. Similarly, fines are also undesirable as indicates excessive explosive consumption. It is, therefore, desirable to have a uniform fragment distribution, avoiding both fines and oversized fragments to optimize the overall cost of mining. In most of the surface mines, blast patterns are established through trial blasts. The blast pattern proposed from trial blast often fails to achieve the required blast results. Thus, it is felt necessary to develop a software for surface blast design based on the methodology proposed by Langefors and Kihlstrom (1978).

1.2 Objectives

The basic objective of the project is to develop a computer model which has the following facility

a) Designing of different parameters of a surface blast
b) Achieving the desired fragmentation size

1.3 Research Strategies

Extensive literature review has been carried out for identifying the controllable & uncontrollable parameters, which have significant effect on the blast design. Existing relationships for blast design are also reviewed of different researcher. Using those established relationship a computer model is developed in C & C++ language for designing a surface blast. For making it more user friendly, it was developed in java platform with the help of software “Net Beans 6.0 IDE”. Presently, the software OCBLASTS 1.0 is in the trial version and it will be further modified accordingly if needed. Developed computer model is tested in an iron ore & a coal mine and the result are analyzed.

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BLAST DESIGN PARAMETERS

Overview

Bench Geometry

Blast Geometry

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Blast Design Parameters

2.1 Overview

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

- *Physico-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 burden bench 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, explosion pressure, ratio of decoupling, strength of explosive used, time taken for explosive wave to travel to the free face and back, volume of gaseous product per unit mass of the explosive, velocity of detonation, velocity of explosion propagation, explosion wave length, weight strength, number of spalls that an explosive wave may produce, length, diameter and weight of the cartridge, loading density, bottom charge and column charge density, etc. are very important. Characteristics of blasting accessories - type, thermal properties are also important.
- Burden distance
- Spacing of the hole
- Ratio of spacing to burden
- Depth of hole
- Diameter of blast holes
- Consideration of toe and depth of sub-grade drilling

- Blasting technique:

  Here objective of blasting, drilling pattern, number of availability of free faces, manner of charging, charge per hole and per delay, sequencing of initiation i.e. delay between two holes in a row and delay between two rows, decking, length of explosive column, height of the bottom charge, volume of the explosive in the blast hole, etc., are to be considered

- Powder factor:

  The size of the fragmented rock should match the bucket size of the excavator and also the 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

2.2 Bench Geometry:

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 degree of selectivity required in separating the ore and waste with the loading equipment; the rate of production; the

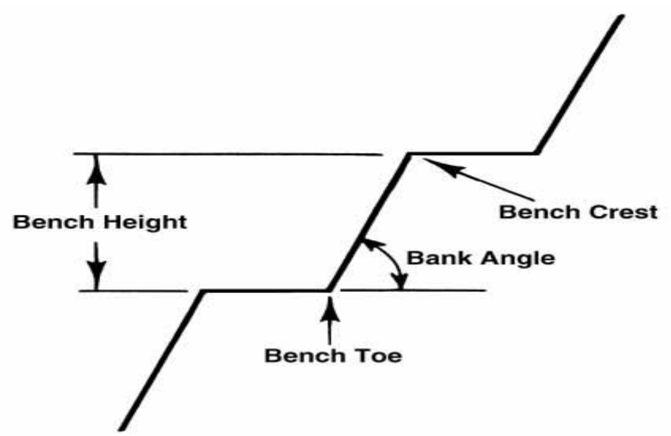


Fig. 1-Bench cross section

size and type of equipment to meet the production requirements; and the climatic conditions. The elements of a bench are illustrated in the above figure 1**.**

The bench height should be set as high as possible within the limits of the size and type of equipment selected for the desired production. The bench should not be so high that it will present 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 m 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 m.

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:

a) Optimum blast hole diameter increases with the height. In general an increase in blast hole diameter decreases in drilling costs

In 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 (figure 2).

2.2.2 Bench Width: There is a minimum bench width, measured horizontally in a direction perpendicular to the pit wall. For each bench height and set of pit operation conations whose value is established by the working requirements of the loading and hauling equipments. The width 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 equipments operating room and bank stability, there is a maximum width that should not be exceeded by any blast.

2.3 Blast Geometry:

2.3.1 Drilling Diameter (D): The hole diameter is selected such that in combination with appropriate 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

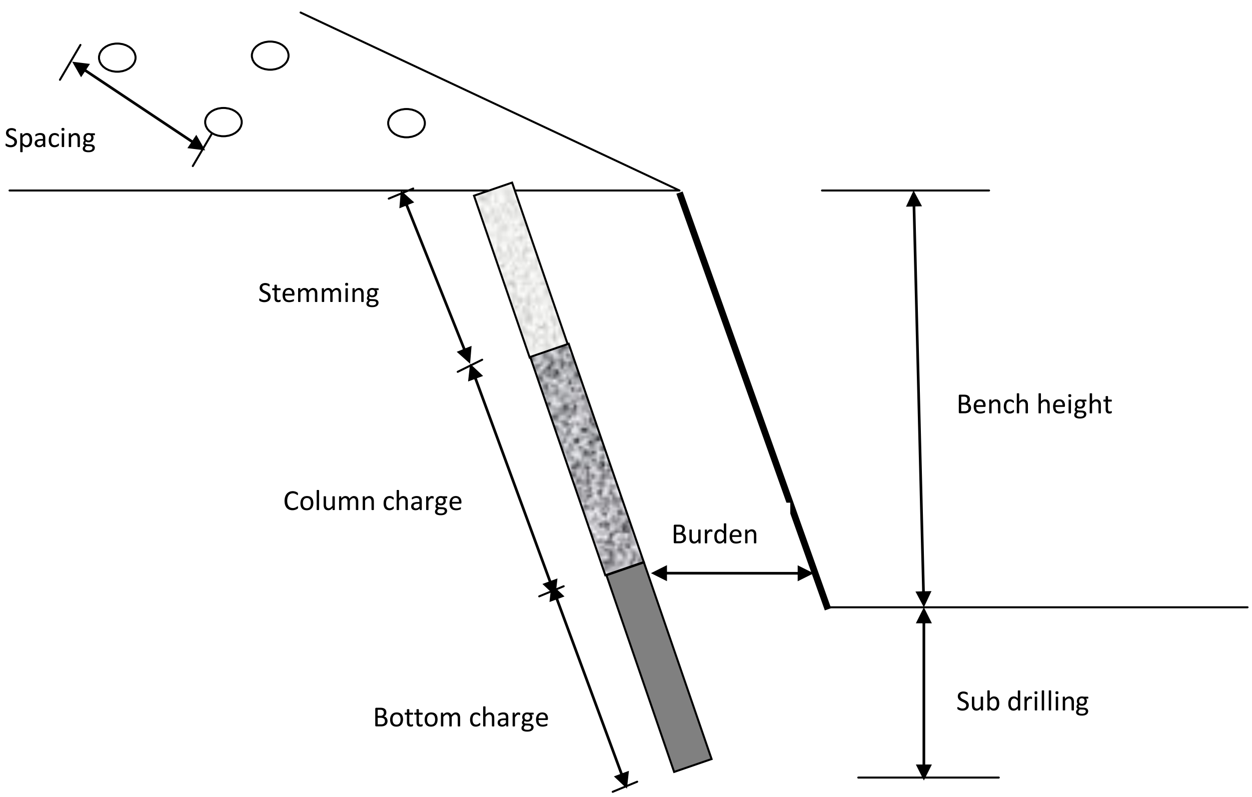


Fig.2-General Layout showing different parameters of blast design

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 in limestone 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.

2.3.2 Sub Drilling (J): To avoid formation of toe in bench blasting the blast hole are drilled below the floor or grade level (figure 2). This is termed as sub grade drilling or sub drilling. If the toe formation will not avoid it may increase the operating costs for loading, hauling equipment. The optimum effective sub drilling depends on

- The structural formation
- Density of the rock
- Type of explosive
- Blasthole 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 sub drilling of the first row reaches value of 10D to 12D .About 10% of sub drilling gives better fragmentation in the rock mass and lesser ground vibration. In generally sub drilling should be 0.3 times the burden. Under different toe conditions sub drilling may be up to 50 percent of the burden. A relation is also shown in the figure 3 below.

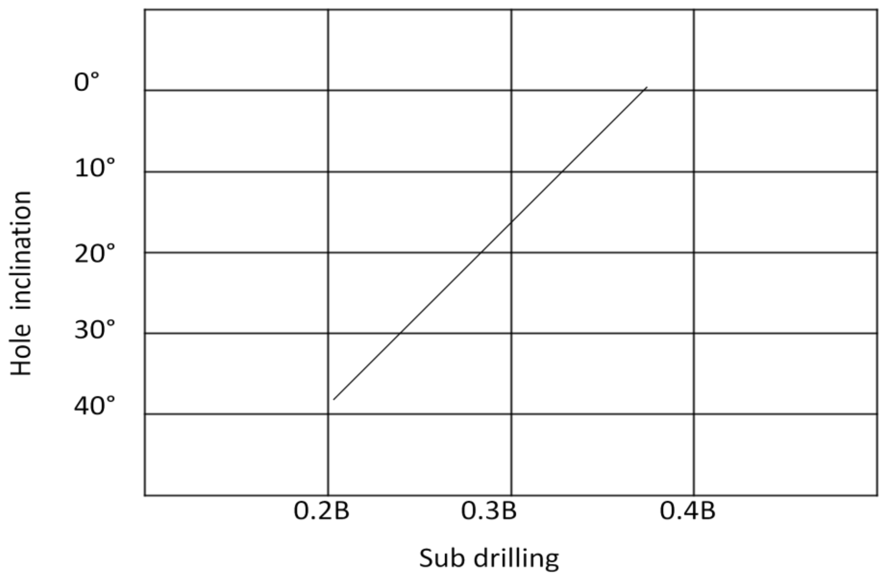


Fig.3-Sub drilling with inclination of blast hole

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

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

2.3.3 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 of natural 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.

From the field experience, it is realized that stemming length of 70 percent of the burden dimension a good approximation. This length has a sufficient control over production of objectionable air blast and fly rock from the Collar zone. It is recommended that the crushed and sized angular rock fragments works best as stemming. But it is common practice to use drill cuttings as a stemming material.

2.3.4 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.

To calculate **L,** the following equation is used:

$$\mathbf{L} = \frac{\mathbf{H}}{\cos\beta} + \left(1 - \frac{\beta}{100}\right) \times \mathbf{J}$$

Where, β in degrees represents the angle with respect to the vertical.

2.3.5 Burden (B): 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 boreholes with 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 figure 4 below.

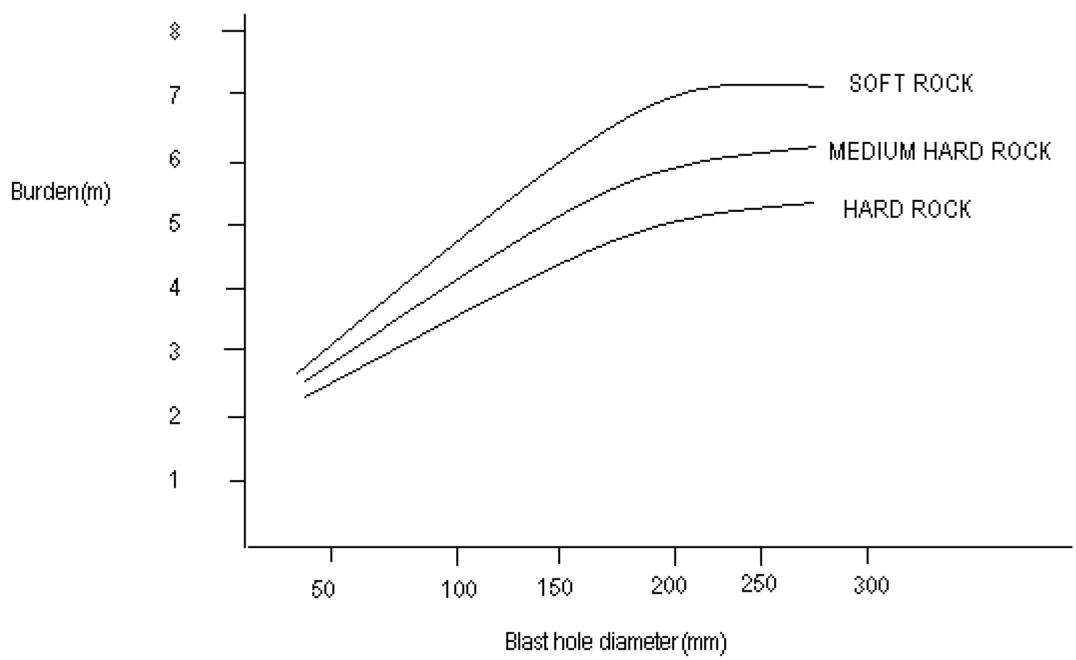


Fig. 4-Size of burden in function with drilling diameter

2.3.6 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 between charges. Spacing is calculated as a function of burden, hole depth, relative primer location between adjacent charges and depends upon initiation time interval. Over past several decades in most mining operations the spacing distances have been decided in relation to burden. The value of the spacing to burden ratio (S: B) which has been commonly used in different formulas lies between 1 and 2. From the production scale test with the spherical charges breaking to crater geometry, many workers suggested that the spacing be kept about 1.3 times the burden. When this ratio increases more than 2, unexpected results were found.

2.4 Powder Factor:

The powder factor is defined as the explosive necessary to fragment 1 m^3 of rock. This equation can also be defined as the amount of explosives over the cubic yards of material desired to be blasted.

$$\text{Kg of explosive used/volume of material blasted.} = \text{kg/ } m^3$$

It is the opinion of many specialists this is not the best tool for designing a blast, unless it is referring to pattern explosives or expressed as energetic consumption. The size of the fragmented rock should match the bucket size of the excavator and also the grizzly size of the primary crusher. it can be also expressed in ton/kg. The following figure 5 shows, how the total operating cost varies with the powder factor.

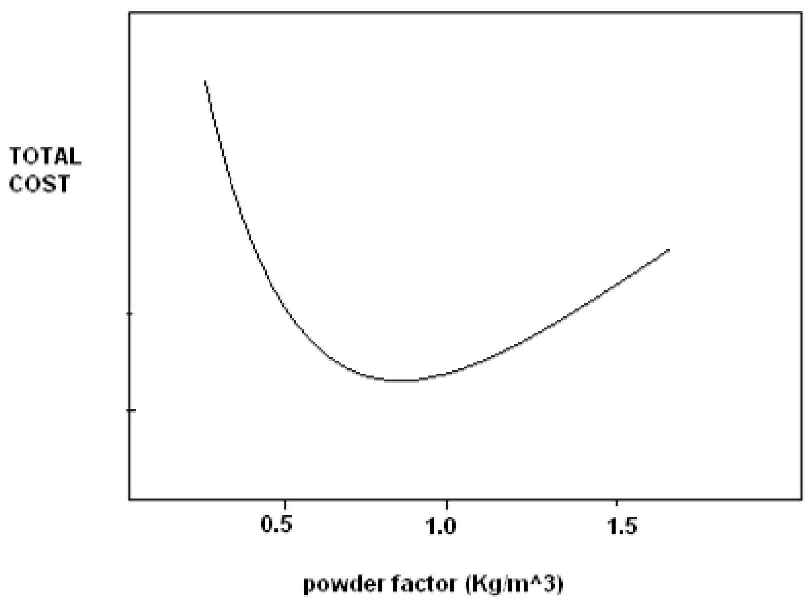


Fig. 5-Reduction of total cost with powder factor

A relation of average fragmentation size in function with burden and powder factor is shown in the figure 6 below.

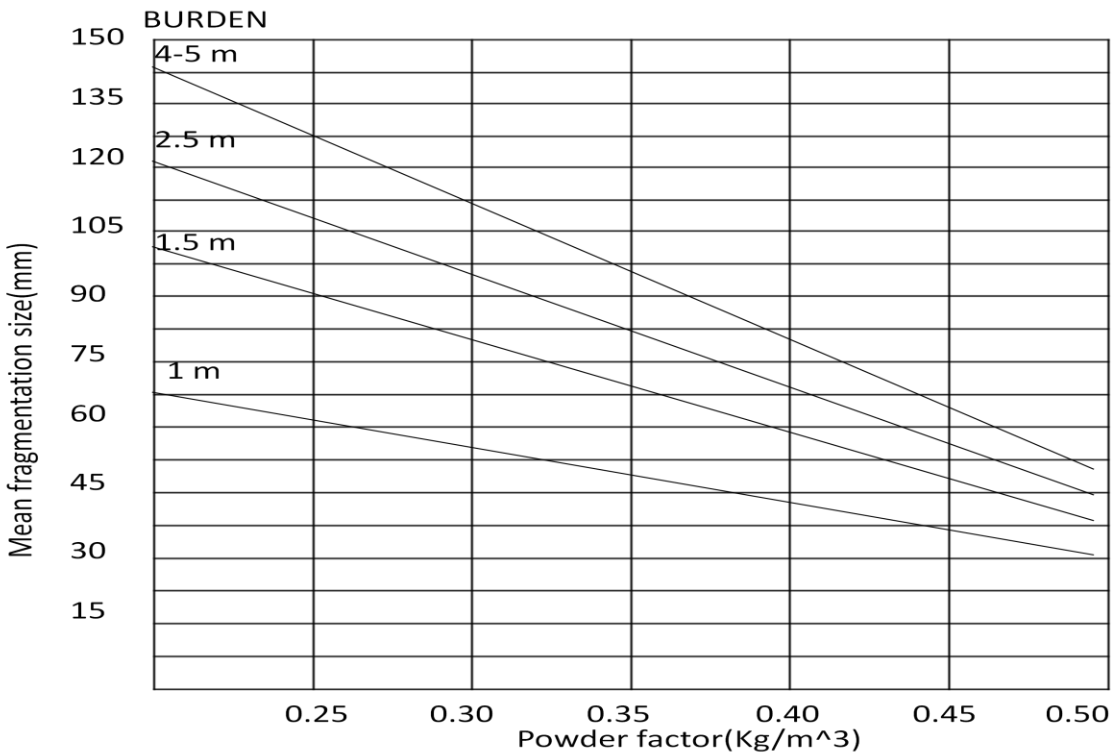


Fig. 6-Average fragmentation size in function with burden and powder factor

2.4 Explosive Selection Criteria

This selection plays a major role in the blast design and the blast results that will occur. An explosive has many characteristics that need to be analyzed in making this decision. These include: minimum diameter in which detonation will occur, the ability to resist water and water pressure, generation of toxic fumes, ability to function under different temperature conditions, input energy needed to start reaction, reaction velocity, detonation pressure, bulk density, and strength. 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.

2.4.1 Types of Explosives

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

2.4.1.1 Dynamite: In Sweden in 1867, Alfred Nobel discovered how to create dynamite. Most dynamites are nitroglycerin based. 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.

2.4.1.2 Slurry Explosives: Slurry explosives, also called water gels, are made up of ammonium nitrate partly in an aqueous solution. Depending on the rest of the ingredients slurries can be classified as a blasting agent or an explosive. 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.

2.4.1.3 Emulsions: 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. To achieve more sensitivity within the emulsion voids are added. 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. Some examples being, powdered aluminum, gasifying agents to reduce density. It is therefore necessary to work above the critical diameter and use powerful 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 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. Some other advantages 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.

2.4.1.4 Dry Blasting Agents: Dry blasting agents are the most widely used explosive used in the world. ANFO is the most common dry blasting agent. An oxygen balanced mixture of ANFO is the lowest cost source of explosive energy today. To increase energy output, ground aluminum foil is added to dry blasting agents. A downfall of this however, is that the cost is increased. Two categories make up dry blasting agents: cartridge blasting agents and bulk ANFO. Bulk ANFO is either blown or augured into a blast hole from bulk truck. These blasting agents will not function properly 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 that are greater than that of water if you would like them to sink, or less than that of water if you would like them to float.

Heavy ANFO is made up of mixtures of ammonium nitrate prills, fuel oil, and slurries. 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 fill the entire blast hole with energy and have to wasted volume that would occur with cartridges.

2.4.2 Explosive Characteristics

2.4.2.1 Physical properties

There are many physical attributes 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.

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. This diameter has quite a wide range between different explosives. Some may have a critical diameter of three inches, while others may have a critical diameter of a few thousandths of an inch. The explosive diameter, the diameter of the borehole, must be greater than the critical diameter of the explosive you choose to use for this blast to function. Thus, if your borehole size is already determined you may eliminate explosives that have a critical diameter which is greater than your predetermined explosive diameter.

Table 1- Critical diameter of explosives

Type	Critical Diameter		
	< 1 in.	1-2 in	> 2 in
Granular Dynamite	x		
Gelatin Dynamite	x		
Cartridge Slurry	x	x	x
Bulk Slurry		x	x
Emulsion		x	x
Poured ANFO		x	
Packaged ANFO		x	
Heavy ANFO		x	x

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. It can also cool the explosive to a point where it will not function properly. To describe the water resistance you can use the terms excellent, good, fair, or poor. If there is water in your blast sight you are going to want to use an explosive with at least a fair rating. The more water resistant an explosive is, the higher the cost.

Table 2 -List of important physical properties of explosives			
Type	**Water Resistance**	**Quality Of Fumes**	**Temp. Resistance Between O°F - 100°F**
Granular Dynamite	Poor to Good	Poor to Good	Good
Gelatin Dynamite	Good to 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 Below 40°F
	* Becomes poor if package is broken	*Can be poor under adverse conditions	

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.

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.

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.

2.4.2.2 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.

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.

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 sub par 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.2

Table 3 -List of important performance properties of explosives

Type	Hazard Sensitivity	Performance Sensitivity	Detonation Velocity (m/s)	Detonation Pressure (K bars)	Density (g/cc)
Granular Dynamite	Moderate to High	Excellent	2100-5700	20-70	0.8-1.4
Gelatin Dynamite	Moderate	Excellent	3600--7500	70-140	1.0-1.6
Cartridge Slurry	Low	Good to Very Good	3900-5700	20-100	1.1-1.3
Bulk Slurry	Low	Good to Very Good	3600-5700	20-100	1.1-1.6
Emulsion	Low	Very Good to Excellent	4200-5500	40-90	1.0-1.2
Poured ANFO	Low	Poor to Good*	1800-4500		0.8-0.85
Packaged ANFO	Low	Good to Very Good	3000-4500	20-60	1.1-1.2
Heavy ANFO	Low	Poor to Good*	3300-5500	20-90	1.1-1.4

*Heavily dependent on field condition

Detonation Pressure: The detonation pressure is the pressure associated with the reaction zone of a detonating explosive. It's is measured in the C-J plane, behind the detonation front, during propagation through an explosive column. This pressure can be estimated using the following formula:

$$p_d = \frac{1}{2}\rho_e c_d^2 10^{-6}$$

Where,

P_d = Detonation pressure (MPa)

ρ_e = Density of explosive (kg/m ^{3})

C_d = Velocity of detonation (m/s)

Detonation pressure is related to the density of the explosive and its reaction velocity. When selecting explosives for primers, detonation pressure is an important consideration. In hard and competent rocks the fragmentation is done more easily with high detonation pressure explosives, owing to the direct relationship that exists between detonation pressure and the breakage mechanisms of the rock.

Density: The density of an explosive is important because explosives are purchased, stored and used on a weight basis. Then density of an explosive determines the weight of explosive that can be loaded into a specific borehole diameter. In the bottom of the blast holes where more energy concentration is required, higher density explosives such as gelatin explosives or water gels are used. In column charges where lower density is required, ANFO based or powder explosives are used. Loading density is the weight of explosive per linear foot of charge in a specified diameter. Loading density is used to determine the total amount of explosive which will be used per borehole and per blast. Loading density can be calculated using the following equation:

$$q_1\left(\frac{kg}{m}\right) = 7.854\times10^{-4}\times\rho_e\times D^2$$

Where,

ρ_e = Explosive density (g/cm^3)

D = Charge Diameter (Mm)

Strength: The strength of an explosive refers to the energy content of an explosive which in turn is the measure of the force it can develop and its ability to do work. Strength is rated in two different ways. One is on an equal volume basis, called bulk strength. The other is rated on an equal weight basis, called weight strength. Strength is measured using various methods and tests. Some of these include: the Ballistic mortar test, seismic strength test, Traulz test, and cratering.

2.4.3 Important Considerations:

There are a few more parameters that must be considered when selecting an explosive other then the before mentioned. These include: the explosive cost, charge diameter, characteristics of the rocks being blasted, volume of the rocks being blasted, safety conditions, and supply problems.

2.4.3.1 Explosive Cost: When selecting an explosive, the cost of the explosive is a very important thing to consider. The goal is to find the lowest cost explosive that is able to complete the job at task. Being the cheapest explosive, ANFO is used the majority of the time. Explosive cost is more correctly expressed in terms of cost per unit of energy available rather than cost per weight because energy is what is used to break the rock. For a blast design using a fixed hole size requiring an explosive or explosives of particular bulk strength the lowest blast costs will be achieved by selecting the explosive having the required bulk strength at the lowest cost per unit length of loaded blast hole. The best explosive is not always the least expensive but rather the one that achieves the lowest blasting costs.

2.4.3.2 Charge Diameter: If explosives with detonation velocities that vary greatly with the diameter are used, you should take the following precautions:

- With blast hole under 50mm diameter, it is better to use slurries or cartridge dynamites
- With blast holes between 50mm and 100mm diameter, ANFO is adequate for bench blasting as a column charge and in inner charges increasing the density by 20 percent with pneumatic chargers and effective priming
- With blast holes above 100mm in diameter, there are no problems with ANFO, although in hard rocks it is better to design columns with selective charges and a good initiation system.
- In large diameters with different mixtures of bulk explosives it is very economical to charge by mechanical means.
- Gelatin and granular cartridge explosives are still used in small diameters, but in medium type calibers they are being substituted for cartridge slurries and emulsions.

2.4.3.3 Rock Characteristics: When blasting rocks, they are categorized into four types, resistant massive rocks, highly fissured rocks, rocks that form blocks, porous blocks. Different types of explosives are recommended for each one of these types.

Resistant massive rock formations have very few fissures and planes of weakness. As a result, an explosive is needed that creates a large number of new surfaces based on its strain energy. The strain energy is the potential energy stored in the linear part of a strained elastic solid. An explosive with a high density and detonation velocity will work well in this case. Thus slurries and emulsions would be good choices.

Highly fissured rock formations have many preexisting fissures. Explosives with high strain energy don't work in this case. ANFO is the recommended choice here because of its high gas energy.

When masses with large spacing between discontinuities that forms large blocks, and in ground where large boulders exist within plastic matrixes, the fragmentation of the rock is more based on the geometry of the blast than the properties of the explosive. Thus, you want an explosive with a balanced strain/gas energy relationship such as heavy ANFO.

In porous rock formations there are many things to consider when blasting along with selecting the proper explosive. The proper explosive would be one with low densities and detonation velocity, such as ANFO. To retain gases in the blast hole for as long as possible the blaster should:

- control the stemming material and height
- Properly sized burden
- priming the bottom
- reduce blast hole pressure by decoupling the charges

2.4.3.4 Volume of Rock Being Blasted: The volume of the rock being blasted will determine the amount of a certain explosive you will use for the blast. When this volume is very large you are going to want to consider the use of bulk explosives. This makes mechanized charging possible from the transports, thus lowering labor costs.

CHAPTER: 03

BLAST DESIGN CONCEPTS - A REVIEW

Blast Design Concepts

Empirical Equations

Fragmentation Analysis

CHAPTER: 03

Blast Design Concepts - A Review

3.1 Blast design Concepts

In order to examine the existing practices in blasting, it is desirable to collect as many blast records and blast designs as possible from different researchers. A critical review of the blasting practices in vogue helps in identifying the shortcomings and exploring the possibility of improving the blast results, by introducing modified techniques and updated products. some of the important concepts including empirical equation supporting blast design proposed by different researchers are discussed as follows

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

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. Incase 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 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 inspite 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.

Konari et al (2004) observed that blast casting is the most recent innovation on blasting for overburden removal in opencast mines. It is implemented in due regard of the growing demand in coal due to rise of power sector needs. It can be implemented by considering some aims like increase of production levels, reduce capital outlay, improving productivity, equipment replacement. The parameters to be considered for blast casting are the overburden rock characteristics, blast geometry, spacing to burden ratio, delay interval, stemming and decking, bench height to width ratio, explosive used etc. They were of the opinion that by improvement in all these parameters, blast casting has a good future in India keeping in view the increasing depth of opencast coal mines. It has high potential to equipment productivity, safety and overall operational economics.

He tried to evaluate the potential of bulk explosive due to increase in rock excavation targets. They studied performance of the explosive in Nigahi and Jayant mines, and observed that with increase in tensile strength of rock there is decrease in the powder factor. They observed that by increase in blastability index, there is increase in density and p-wave velocity, and the fragmentation decreases with powder factor. They were of the opinion that the explosive consumption should be taken care of to get proper fragmentation size. They pointed out that more efforts should be put on assessing the VOD of the explosive as it increases the shock energy and more studies are needed to justify the results from the work done.

Sethi and Dey (2004) studied the blast designs in Indian mines and found that most of the designs are based on trial and error to a large extent. They pointed out that utilizing computerized blast designing method; the disadvantages of the previous used ones can be eliminated. After studying all the parameters related to blasting, they observed their share of weightage and found that parameters like the fragmentation size and hole diameter are more significant on powder factor where as charge per hole has negligible impact on overall performance. The hole length and bench height has equal weightage. Similar are the spacing and burden. They pointed out that calculating and manipulating the extent of significance of all the factors, software can be designed to provide an appropriate solution to the blast design.

Bhandari (2004) developed a blast information management system (BIMS) where all the data in the mining operation are stored, analyzed, audited, documented and managed. These can be used to optimize the whole process. They observed that use of software for blasting operation i.e. BIMS makes the job simpler. It is easy to use, user friendly, data entry, reliable storage and analysis and can be customized easily. It saves time and cost to get the impact of a particular design. It helps to train and assess the effects of a certain drill and blast design for people and organizations that use blasting.

2.2 Empirical Equations Supporting Blast Designs

Fraenkel (1944)

$$Bmax = \left(\frac{K}{50}\right) \times \left(\frac{d^{\,8}}{3 \times \; hc \times H}\right)^{\frac{1}{3.3}}$$

Where,

B_{max} = Maximum burden for good fragmentation, m

d = Borehole diameter, m

h_c = Charge height, m

H = Depth of the blast hole, m

Andersen (1952) determined the burden value in feet and its value increases with the length of the blast hole but not indefinitely as usual happens in practice.

$$B = K \times \sqrt{(D') \times L}$$

Where,

B= Burden, ft

D'= Diameter of hole, ft

L= Length of the blast hole, ft

K= Empirical constant

This formula does not take into account the rock properties or those of the explosives.

Pearse (1955)

$$B = K \times d \times \sqrt{\left(\frac{Ps}{\sigma t}\right)}$$

Where,

B = maximum e burden (m)

K = Constant, value varies from 0.7-1.0

Ps = Detonation pressure of the explosives (Kg/cm^2)

σt = Tensile strength (Kg/cm$^{2)}$

d = Diameter of borehole

Hino (1959)

The equation proposed by Hino is:

$$B = \frac{D}{4} \times \left(\frac{PD}{RT'}\right)^{\frac{1}{n}}$$

Where,

B= Burden, m

D= Blasthole diameter, cm

PD= Detonating Pressure, Kg/cm^2

RT'=Dynamic Tensile Strength, Kg/cm^2

n= Characteristics constant depending upon the par explosive-rock and calculated through the catering test

$$n = \frac{\log \dfrac{PD}{RT}}{\log \dfrac{2\mathrm{D}'}{d}}$$

Where,

D'= Optimum depth of the center of gravity of the charge, cm and it determined graphically from the following equation values,

$$D'' = \Delta \sum V'^{\,0.33}$$

d= diameter of the explosive charge

D"= depth of the center of gravity of charge

Δ= Relationship of depths D''/D_c

D_c=Critical depth of the center of gravity of charge

∑= Volumetric constant of charge

V'= Volume of charge used

Allsman (1960)

The equation for maximum burden value proposed is;

$$B_{max} = \sqrt{\frac{Impuse \times g}{\pi \times u \times \rho}} = \sqrt{\frac{PD \times t \times D \times g}{u \times \rho}}$$

Where,

PD= Mean adverse detonating Pressure, N/m^2

t= Duration of average detonation, sec

ρ= Specific rock weight, N/m^3

u= minimum velocity which must be imparted to the rock, m/s

g= acceleration due to gravity=9.81 m/s^2

D= Diameter of blasthole, m

Ash (1963)

Burden, B (ft) = 0.084 × K_B × D (in)

Where, K_B = Depends upon the rock group and the type of explosive used, See **Table A**

Blast hole depth, L= K_L× B (K_L between 1.5 & 4)

Sub drilling, J= $K_J \times B$ (K_J between 0.2 & 0.4)

Stemming, T= $K_T \times B$ (K_T between 0.7 & 1)

Spacing, S=$K_s \times B$

K_s= 2.0 for simultaneous initiation, 1.0 for sequenced blasthole with long delay between 1.2 & 1.8 for sequenced blasthole with Short delay

Langefors and Kihlstrom (1968)

$$B_{max} = \frac{D}{33}\sqrt{\frac{\rho e \times PRP}{Co \times f \times \frac{S}{B}}}$$

Where,

B_{max} = Maximum burden for good fragmentation (m)

D = diameter of hole (m)

ρ_e =Density of the explosive in the borehole ($Kg/m^{3)}$

PRP = Relative Weight strength of the explosive

f = Degree of confinement of the blasthole.

S/B = Spacing to burden ratio

Co = Corrected blastability factor (Kg/m^3)

= C + 0.75 for B_{max} =l.4-1.5m

= C + 0.07/B for B_{max} < 1.4m

When C = rock constant

Lopez Jimeno, E (1980)

He modifies the ash's formula by incorporating the seismic velocity to the rock mass, resulting in

$$B = 0.76 \times D \times F$$

Where,

B= Burden, m

D= Diameter of blasthole, inches

F= correction factor based on rock group = Fr× Fe

$$Fr = \left[\frac{2.7 \times 3500}{\rho' \times VC}\right]^{0.33}$$

$$\mathrm{Fe} = \left[\frac{\rho'' \times VD}{1.3 \times 3660^2}\right]^{0.33}$$

Where,

ρ'= specific gravity of rock, gm/cm^3

VC= seismic propagation velocity of the rock mass

ρ''= specific gravity of explosive charge, gm/cm^3

VD= Detonation velocity of explosive, m/s

The indicated formula is valid for diameter between 165 & 250mm.For large blasthole the burden value will be affected by a reducing coefficient of 0.9.

Konya and Walter (1990)

$$\mathrm{B} = 3.15 \times \mathrm{D} \times \sqrt[3]{\frac{\rho e}{\rho r}}$$

$$B = [1.5 + \frac{2 \times \rho e}{\rho r}] \times D$$

Where,

B = Burden, (ft)

ρe = Specific gravity of explosive, (lb/in3)

ρr = Specific gravity of rock, (lb/in3)

D = Diameter of explosive, (in)

Correction factor, Bc = Kd. Ks. Kr. B

Where,

Bc = Corrected burden (ft)

Kd = Correction factor for rock deposition. Its value is as follows,

- for bedding steeply dipping into cut Kd = 1. 18
- for bedding steeply dipping into face Kd = 0.95
- for other cases Kd = 1.0

Ks = Correction factor for geologic structure. Its value is as follows,

- for heavily cracked, frequent weak joints, weakly cemented layers Ks = 1.30
- for thin well cemented layers with tight joints Ks=1.1
- for massive intact rock Ks = 0.95

Kr = correction factors for number of row. Its value is a follows,

- for one or two rows of blastholes Kr = 1.0
- for third or subsequent rows Kr = 0.95

Konya and walter also suggested the following empirical relationships-

For instantaneous initiations system,

$$S = \frac{H + 2\times B}{3}, \ H < 4B$$

$$S = 2 \times B, \ H \geq 4B$$

For delay initiation system,

$$S = \frac{H + 2 \times B}{8}, \ H < 4B$$

$$S = l.4 \times B, \ H >= 4B$$

Where,

H = depth of blast-hole, m

B=burden, m

S=Spacing, m

Konya and Walter also suggested the following empirical relationship-

$$B = 0.67 \times d \times (S_{ANFO} / \rho r)^{0.33}$$

Where,

S_{ANFO} = relative strength of explosive

ρr = density of rock, gm/c.c.

d = diameter of blast-hole, m

2.3 Fragmentation Analysis

Kuz-Ram model (1983)

According to the Kuz-Ram model, the mean fragment size can be calculated by the following equation

$$X = A \times \left(\frac{V}{Q}\right)^{0.8} \times (Q)^{0.167} \times \left(\frac{E}{115}\right)^{-0.633}$$

Where,

X = mean fragment size, cm

V = volume of blasted rock, m^3

Q = mass of explosive charge per hole, kg

E = relative weight strength of explosive (ANFO= 100)

A = a constant based on rock factor (depends upon rock density, strength and jointing).

Rosin-Rammiler equation

An estimate of the fragment size distribution is given by the Rosin-Rammiler equation which is as follows:

$$R(x) = 1 - e^{-\left(\frac{X}{Xc}\right)^{n}}$$

Where,

R(x) = proportion of the material passing through the screen size x.

X = screen size, cm

X_c = characteristics size, cm

n = index of uniformity, varies from 0.8 - 2.0

$$n = \left\{2.2 - 14 \times \frac{B}{d}\right\} \times \left(1 - \frac{W}{B}\right) \times \frac{L}{H} \times \left\{1 + \frac{(R-1)}{2}\right\}^{0.5}$$

Where,

d= charge diameter, mm

B = burden, m

W = standard deviation of drilling accuracy, m

R =Spacing/Burden

H = Bench height, m

L = Charge length, m

CHAPTER: 04

SURFACE BLAST DESIGN

Introduction

Small Diameter Bench Blast

Large Diameter Bench Blasting

Computational Approach

CHAPTER: 04

Surface Blast Design

4.1 Introduction

This chapter on bench blasting will help you understand and use geometric configuration of blasthole, explosive charges, initiation sequence, and the delay timing. With the continued evolution of drilling equipment, and the extension of surface mining, bench blasting is fast becoming the most popular method of rock fragmentation with explosives. Bench blasting for surface are classified according to their purpose. Mentioned below are some of the more common types blasting are Conventional bench blasting, Rip-rap blasting, Cast blasting, Road and railway blasting, Trench and ramp blasting, Ground leveling and foundation blasting.

The main focus of this chapter will be on bench blasting (both small and large diameter). Many formulas and methods for calculating geometric parameters such as burden, spacing, and sub drilling have been around since the early 1950's. The previously mentioned formulas use one or more of the following parameters: hole diameter, characteristics of explosives, compressive rock strength, and many more. Bench blasting can also be classified by the diameter of the blast hole. These falls into two categories, small diameter blasting (65 mm to 165 mm,) and large diameter bench blasting (180 mm to 450 mm).

In small diameter blasting the most common technique developed by **Langefors** and **Kihlstrom** is used; however, it is better to use the crater technique by **Livingston** or the American criteria for the larger diameter blasts. Due to the different nature of rocks the best method is continuous trial and error to arrive at the best conclusion.

Obviously, every situation in the field cannot be predicted, and is beyond the scope of this chapter. What this chapter will do is give an initial approach to the approximate geometric design of blasting, the calculation of charges, and characterization of rocks by their uniaxial compressive strengths. It will be necessary to adjust patterns, explosive charges to suit the need in the field according to the type and make up of the material encountered.

4.2 Small Diameter Bench Blast

As stated before, the dimensions of the small diameter bench blast range from 65 mm (2.56 in) to 165 mm (6.50 in). The small diameter bench blasts are mostly used in small surface mining operations, construction excavations, and quarries. Many variables must be considered when preparing for any blast. The variables that need to be considered are: drilling diameters, bench height, drilling/sub drilling and stemming patterns, inclination of blasthole and charge distribution.

Drill Diameters: While selecting the proper blasthole diameter, the average production per hour, or excavation, must be taken into account (Table 4). In addition, the type of material excavated must also be accounted. An important aspect when drilling is the drilling cost. The cost usually goes down as the diameter of the hole increases.

Table 4-Average production with variation of drill hole diameter

Blast hole diameters(mm)	Average production per hour(m^3b/h)	
	Medium-soft rock <120 MPa	Hard-very hard rock >120 MPa
65	190	60
89	250	110
150	550	270

Bench Height: When determining the bench height it is important to take into account the drilling diameter and the loading equipment used (Table 5).

Table 5- Relation between bench height, blasthole diameter and loading equipment

Bench Height H(m)	Blasthole diameter D(mm)	Recommended loading Equipment
8.0-10	65-90	Front end loader
10.0-15	100-150	Hydraulic or rope shovel

Burden (B) and Spacing(S): The burden is the minimum distance from the axis of a blasthole to the free face, and the spacing is the distance between blasthole in the same row. These parameters are dependent on the following variables: drilling diameter, properties of the rock and explosive, the height of the bench, and the degree of fragmentation and displacement.

There are many formulas that have been suggested for calculating the burden, taking into accounts one or more of the variables mentioned (Table 6).

Table 6-Variation of parameters with UCS of rock & Diameter of hole				
Design Parameter	Uniaxial compressive strength (MPa)			
	Low < 70	Medium 70-120	High 120-180	Very High > 180
Burden - B	39 x D	37 x D	35 x D	33 x D
Spacing - S	51 x D	47 x D	43 x D	38 x D
Stemming - T	35 x D	34 x D	32 x D	30 x D
Sub drilling - J	10 x D	11 x D	12 x D	12 x D

Values that are outside those that are established can lead to some of the following situations.

- -Marking and collaring errors.
- -Inclination and directional errors.
- -Deflection errors while drilling.
- -Irregularities in the face of the slope.

If the burden is too great, then the explosion gases encounter too much resistance to effectively fracture and displace the rock. Part of the energy used is turned into seismic energy and intensifies ground vibration. This is most evident in pre-splitting blasts where there is total confinement and vibration levels can be as much as 5 times larger then normal bench blasting.

If the burden is not large enough, the gases escape and expand at high speeds towards the free face. This pushes the fragmented rock, and projects it uncontrollably causing an increase in overpressure of the air and noise.

The spacing S value is calculated with burden and the delay timing between blasthole. The value for spacing is approximately 1.15 x B for hard rocks, and 1.30 x B for soft rocks (Table 3**)**. As with burden, if the dimensions for spacing are inadequate then irregularities occur in the rock face. If the spacing is too large then the fracturing between the charges is inadequate and leads to toe problems. If the spacing is too close together then excessive crushing between charges occurs, along with superficial crater breakage, large blocks in front of the blast hole, and toe problems.

Stemming (T): Stemming is the inert material packed within the blasthole meant to confine the gases produced with the explosion, improving the quality of the blast. Just as with any other calculations, this too must be accurate. If the stemming is too great (excessive) then this leads to a large quantity of boulders coming from the top of the bench, poor swelling of the muck pile, and an elevated vibration level. However, if the stemming is too small (insufficient) then this leads to a premature escape of the gases leading to an air blast and a danger of fly rock, the hurling of rock fragments in a blast.

To properly calculate stemming, the type and size of material used, and the length of the stemming column must be taken into account. Studies have shown that coarse angular material, such as crushed rock, is the most effective stemming product. Crushed rock effectively lowers the stemming length by up to 41%. The optimal stemming length varies between 20 and 60 times the diameter of the blast hole with at least 25 times the diameter maintained to avoid the problems listed above in Table 6.

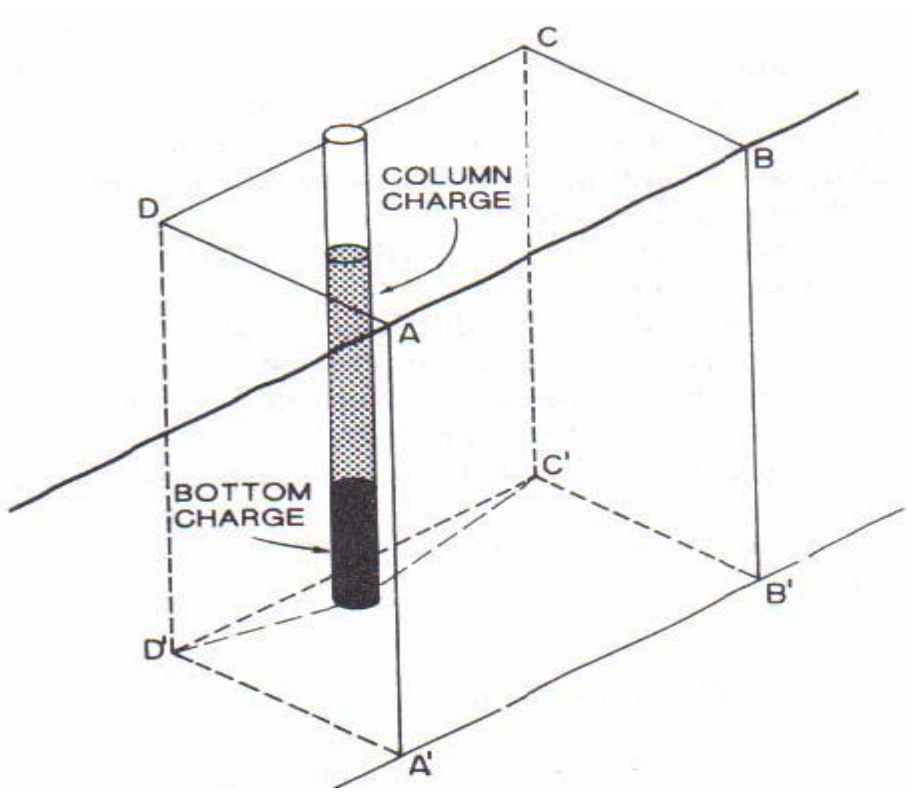
Sub Drilling (J): Sub drilling is the length of the blasthole underneath the floor level needed to break the rock at bench height and achieve adequate fragmentation and displacement; this allows the loading equipment to achieve optimum level of productivity. However, sub drilling is not used in calculating the volume of rock being blasted. If sub drilling is too small, the rock will not completely shear off resulting in a toe appearance (this leads to an increase in loading costs). However, if the sub drilling is too large the following can happen:

- Increase in drilling and blasting costs
- An increased vibration level.
- Excessive fragmentation of the bench, affecting slope stability in the end zones
- Increased risk of cutoffs and over break.

The value of sub drilling that produces the optimum level of breakage is roughly 0.3 times of Burden (Table 6).

Inclination of the Blasthole (β): In bench blasting it has been discovered that inclined drilling gives the most benefits with few disadvantages. Some of the benefits include: better fragmentation, less sub drilling, increased drilling productivity, and a lower powder factor. Some of the disadvantages are an increased drilling length, more wear on bits, and problems in charging the explosive. The blasthole length increases with inclination; however, the sub drilling decreases.

Charge Distribution: The required energy needed to produce rock breakage is not uniform in bench blasting. The energy generated by the explosive must overcome the tensile strength of the rock (section CDD'C') and the shear strength (section A'B'C'D'). To achieve this effect the explosive with the greater density and strength should be placed on the bottom of the blasthole, known as the bottom charge. It should be noted that placing this charge on the bottom of the blasthole increases the diameter of shaped charges by roughly 10%. The explosive with the lighter density should be placed in the column; this is known as the column charge (**figure 7**).



(**Fig. 7-**Charge distribution)

The energy per unit length for the bottom charge should be roughly 2 to 2.5 times more then the energy necessary for rock breakage. Recommended lengths of bottom charges are given in Table 7.

Table 7- Variation of bottom charge length with UCS & Diameter				
Design Parameter	Compressive strength (MPa)			
	Soft < 70	Medium 70-120	Hard 120-180	Very Hard > 180
Bottom charge length l_f	30 x D	35 x D	40 x D	46 x D

The height of the column charge is calculated by the difference between total lengths of blast hole and the sum of stemming and bottom charge lengths.

Powder Factor: Powder factor is nothing but the specific charge or we can say it is the m3 of material excavated per kg of explosive used. For the rock groups shown in Table 7**,** the powder factor varies between 0.25 and 0.55 kg/m^3.

4.3 Large Diameter Bench Blasting

Diameters from 165 mm to 450 mm are considered to be large diameter bench blasts. Large diameter bench blasts are used mostly in large surface mining operations and certain civil engineering excavations like power stations and quarries for the construction of dams. Many of the same variables are required for the proper calculations.

Drilling Diameters: Much of the same criteria for drilling parameters are the same for large diameter blasts as they are for small diameter blasts. The average production per hour and type of rock being fragmented is still the variables needed for consideration (Table 8).

Table 8-Variation of average production with diameter and rock type			
Blasthole Diameter **D** (mm)	Average production per hour (m^3b/h)		
	Soft Rock < 70 MPa	Medium Hard 70-180 MPa	Very Hard Rock > 180 MPa
200	600	150	50
250	1200	300	125
311	2050	625	270

Bench Height: There are a couple of ways to calculate the bench height of a large diameter blast hole, the first of which relates to the size and reach of the rope shovel. The height in meters can be estimated by the following equation:

$$H = 10 + 0.57\,(C_c - 6)$$

Where,

C_c = the bucket size of the shovel (m^3),

H= bench height (m)

Another way to calculate bench height which take into account the compressive rock strength and relate it to the diameter can be seen in Table 9.

Table 9-Relationship of bench height, stemming with diameter & UCS of rock			
Design Parameter	Compressive rock strength (MPa)		
	Low < 70	Medium-high 70-180	Very High >180
Bench Height H	52 x D	44 x D	37 x D
Stemming - T	40 x D	32 x D	25 x D

Stemming: To determine the proper length of the stemming refer to Table 9. The table uses the relationship between diameter and compressive rock strength.

Sub drilling: Sub drilling **is** usually calculated from blasthole diameter, as show in Table 10.

Table 10-Relationship of sub drilling with blasthole diameter		
Design Parameter	Blasthole Diameter (mm)	
	180-250	250-450
Sub drilling - J	7-8 x D	5-6 x D
	J=5+ (0.450-D)/0.09467	

When drilling vertical blasthole the first row should reach values of approximately 10 to 12 times **D**. Shorter lengths then those that are indicated if used in the following cases:

- -Horizontal bedding planes that coincide with the bench toe.
- -Application of select explosive charges.

Inclination: Most drills have a difficult time drilling holes of diameters of a large magnitude. Because of the difficulty in this, most blast holes are drilled vertically. There are a few exceptions though, when drilling in soft rocks with a bench height over 24 meters, it is recommended that inclined drilling be used. The best example of the use of inclined drilling in large diameter bench blasting is in coal mining operations.

Drill Patterns: The burden as indicated previously is a function with the charge diameter, compressive rock strength, and specific energy of the explosive used. The diameter of the column charge is usually the same as the drilling diameter. List of burden and spacing values for various compressive rock strengths and explosives are given in Table 11**.**

Table 11-Burden and spacing values for various compressive rock strengths and explosives				
Type of Explosive	Design Parameter	Compressive rock strength (MPa)		
		Soft < 70	Medium-Hard 70-180	Very Hard > 180
ANFO	Burden - B	28 x D	23 x D	21 x D
	Spacing - S	33 x D	27 x D	24 x D
Water gels/ emulsions	Burden - B	38 x D	32 x D	30 x D
	Spacing - S	45 x D	37 x D	34 x D

Charge Distribution: When doing large surface operations ANFO**,** ammonium nitrate fuel oil, is primarily used due to the following advantages.

- -Low cost & high Bubble Energy.
- -Safety & Easy mechanization.

In the cases where ANFO cannot be used, when the blasthole might be filled with water or when the charges on the bottom have been used as an initiator or primer for the rest of the charge column, water gels have been used as a substitute. Currently the system consists of creating a bottom charge of a high density explosive with a length approximately 8 to 16 times the diameter of the blast hole, in accordance with the rock type, and filling the rest of the blasthole with ANFO. It should be noted that the diameter of the bottom charge does not increase due to compression as there was in small diameter bench blasting. The technique listed above gives the minimum costs in drilling and blasting, while allowing for the optimum results in fragmentation, swelling, floor conditions and geometry of the muck pile.

Powder Factor: Powder factor is nothing but the specific charge or we can say it is the m3 of material excavated per kg of explosive used In large diameter blasting the powder factors range from 0.25 to 1.2 kg/m^3.

4.4 Computational Approach

Based on the methodology proposed by Langefors and Kihlstrom, software has been developed for surface blast design. The software follows the logic as depicted in fig.H1, fig.H2, fig.H3, fig.H4, fig.H5, fig.H6 given below. The coding of software is written with the C++ language initially and again it is developed in NetBeans. The required input data is given through screen entry which can be written in IN.txt file and out put will be given by the software on screen as well as text pad as OUT.txt. It is user friendly and suggests the user in case of wrong entry. However as the software uses the empirical relationship, it is more useful for trial blasts. In the near future the application of the software will be extended for modern explosive as well.

4.4.1 Flow Sheet : Surface blast design methodology, proposed by Langefors and Kihlstrom, is utilized to develop this software. The methodology is expressed in flow sheet to depict the step by step calculation (logical and mathematical) as given below -

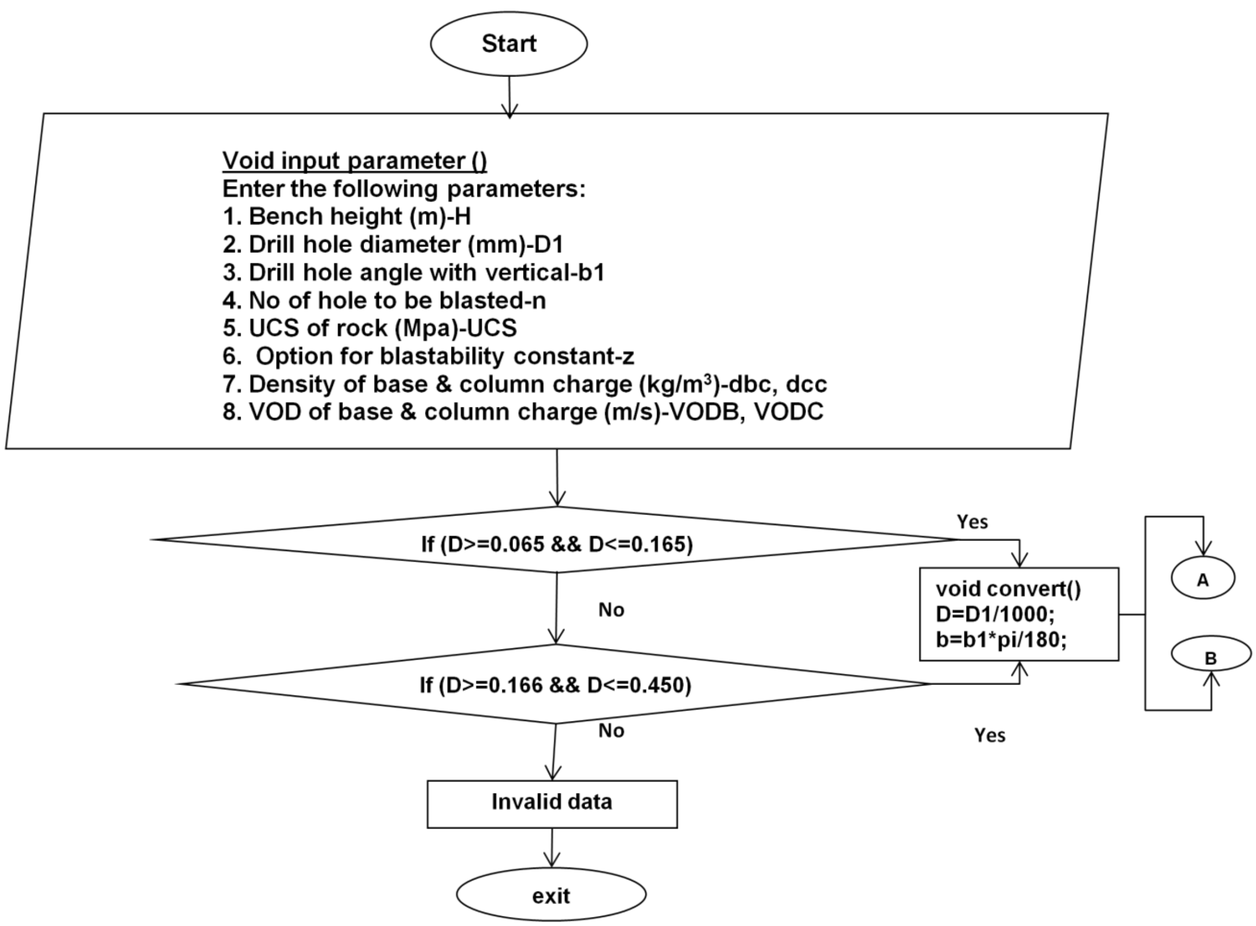


Fig H1- input parameters

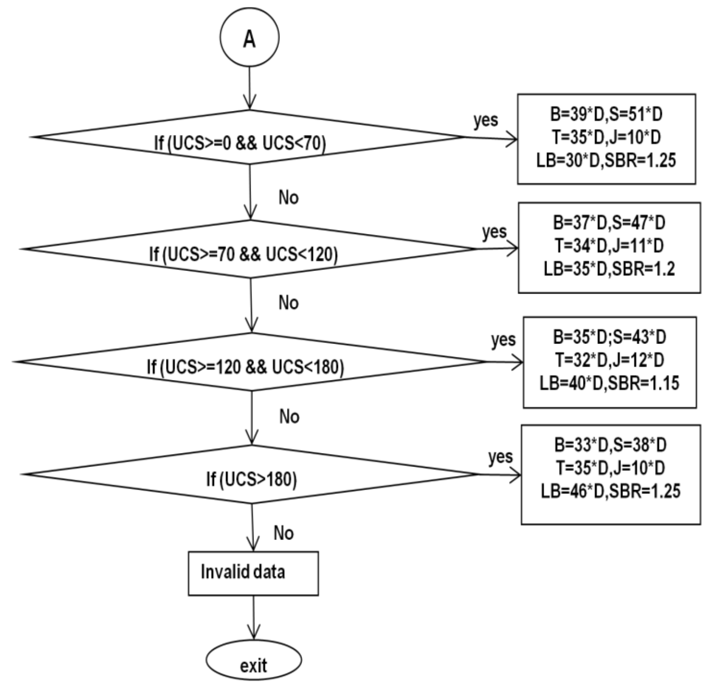


Fig. H2- Designed parameter for small diameter blasthole

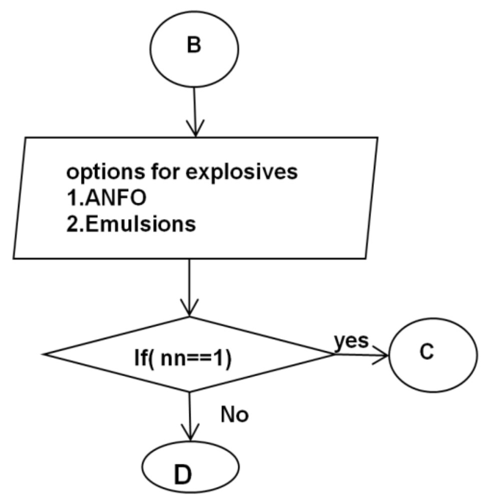


Fig. H3-Selection of explosive

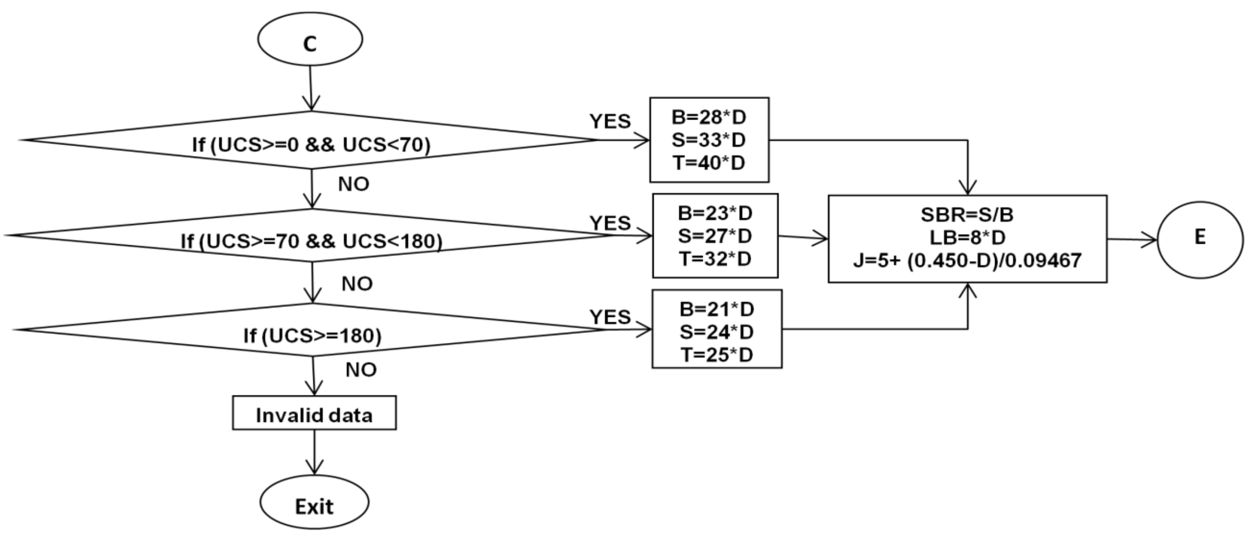


Fig. H4 -Designed parameter for large diameter blasthole for use of ANFO

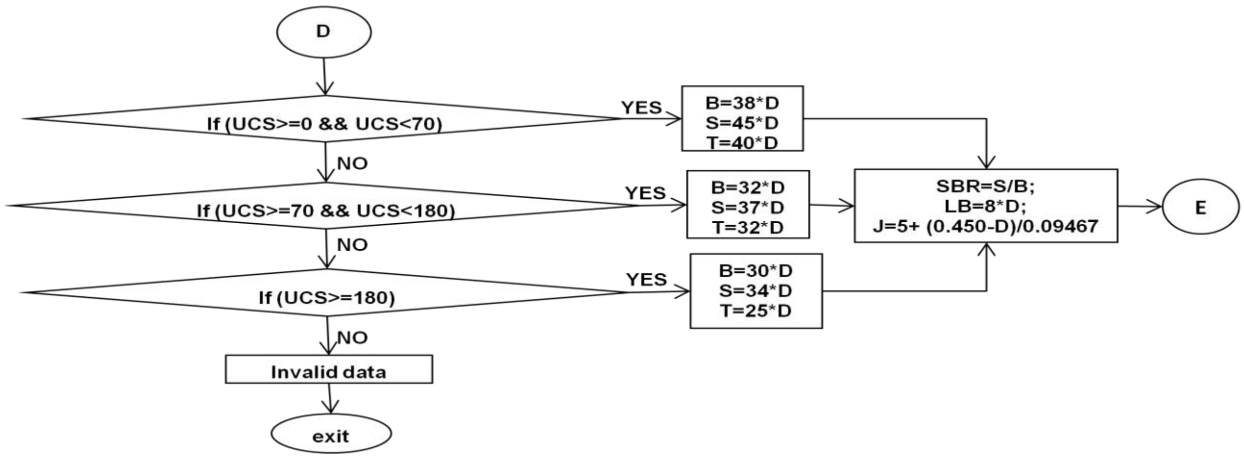


Fig. H5-Designed parameter for large diameter blasthole for use of Emulsion

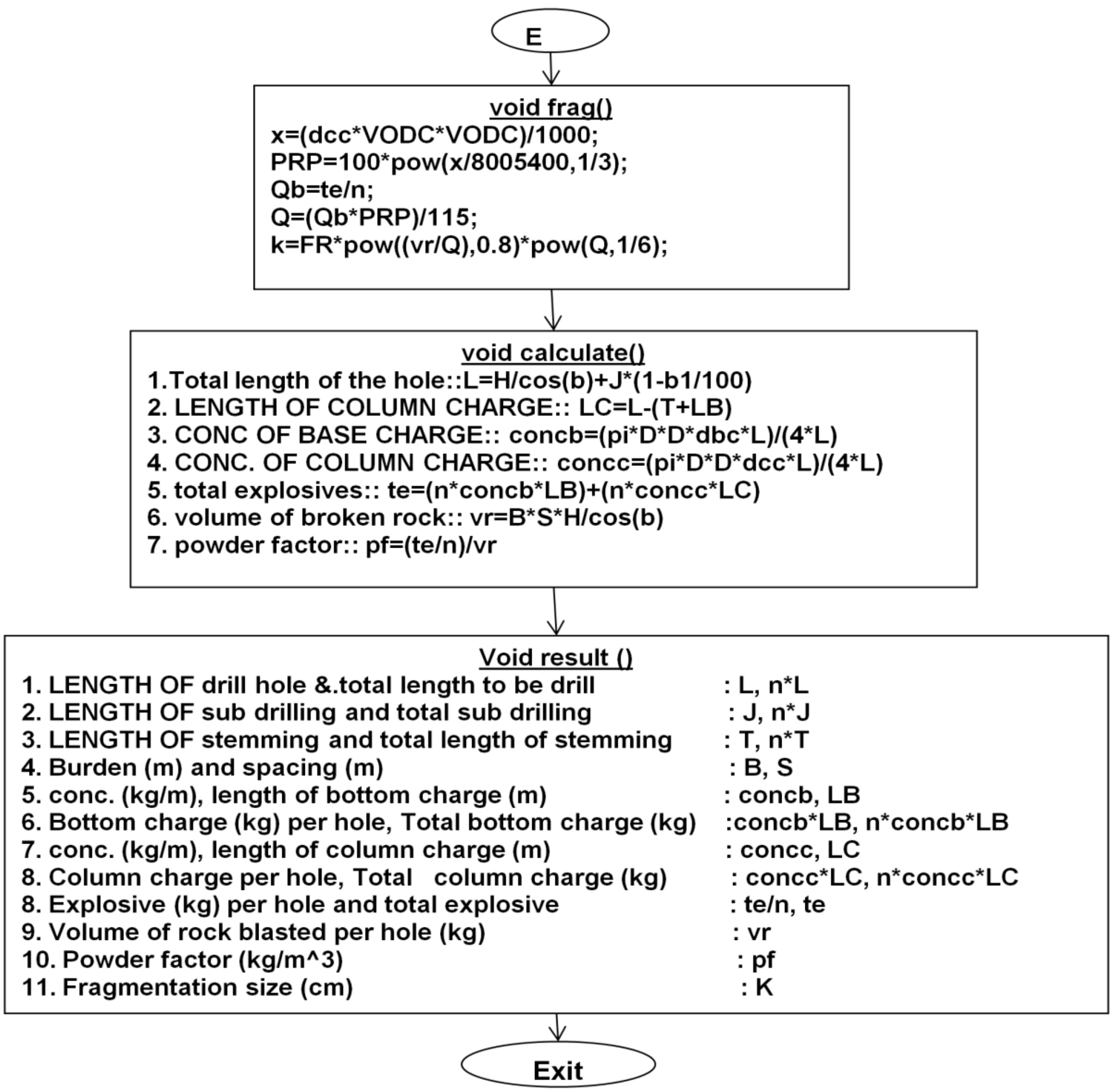


Fig. H6-Calculation and result

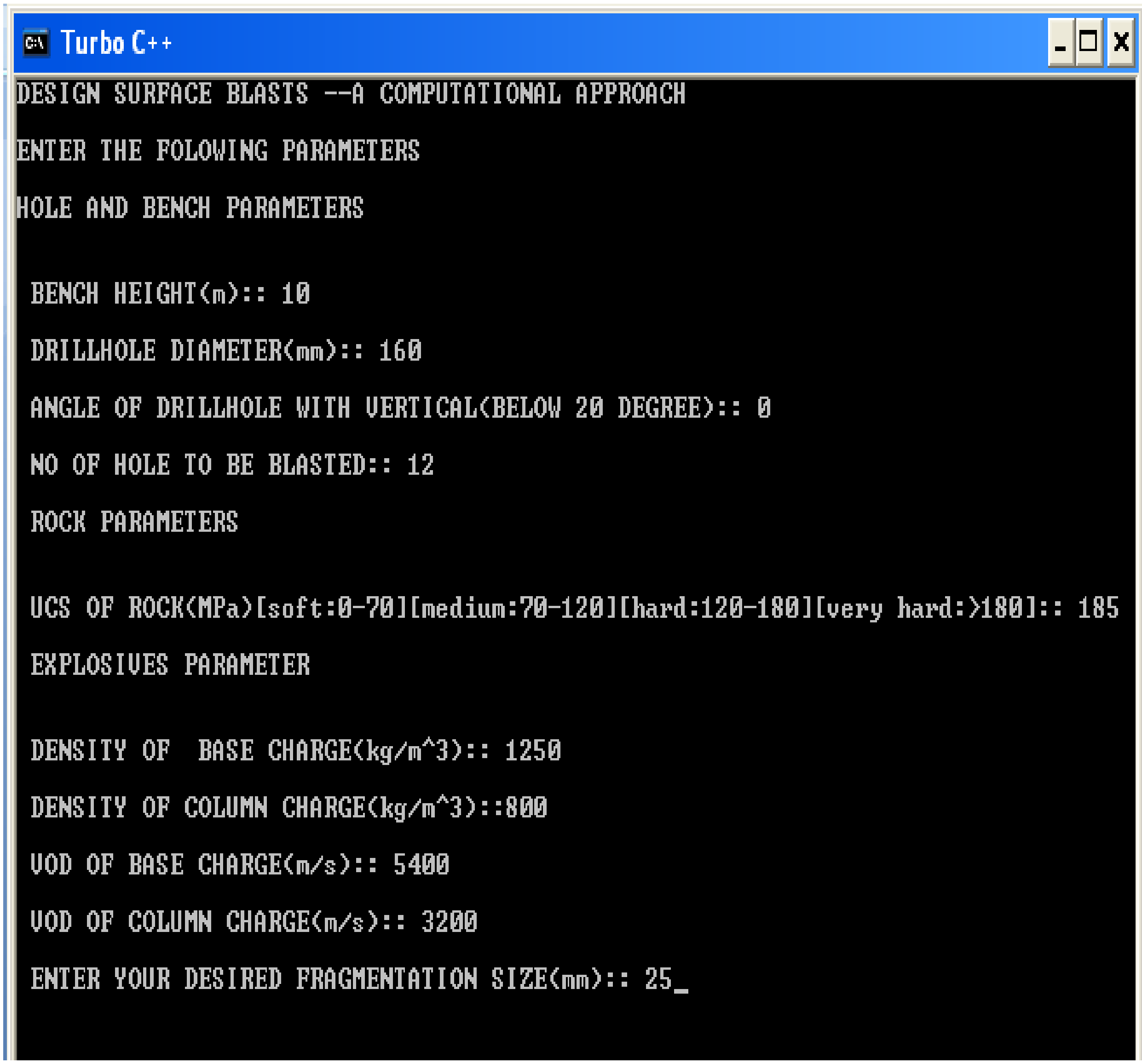
4.4.2 The software (*OCBLASTS 1.0)*

4.4.2.1 Designed model in C++

This module is developed in previous semester with the help of c++ language,which have the facilities of screen input and output.it aslo can write thre input & output to a text files depending upon the user command.

SCREEN INPUT:

it allows the user to provide the values of input parameter one by one,in case of mistake in between the user can reload the program by pressing the bottom “R/r” which will take the user to initial screen to give new inputs.



SCREEN OUTPUT:

it gives the out put of all the designed parameters like burden,spacing,stemming,total explosice used,powder factor,fragmentation size etc.

```
Turbo C++
1.LENGTH OF DRILL HOLE::11.92

2.total length to be drill::143.040009

3.LENGTH OF sub drilling and total sub drilling :1.92 23.039999

4.LENGTH OF stemming and total length of steming::4.68 56.160004

5.burden(m)::   spacing(m) ::5.28 6.08

6.conc(kg/m) and length of bottom charge(m),bottom charge(kg) per hole ::

25.132797
4.62
116.113533

7.total bottom charge(kg)::1393.362427

8.conc(kg/m) and length of column charge(m) ,column charge per hole::

16.084991
2.619999
42.14267

9.total column charge(kg)::505.712006

10.explosive(kg) per hole and total explosive ::158.25621   1899.074463

11.volume of rock blasted per hole(m^3)321.023987

12.powder factor(kg/m^3)  0.492973

enter rock factor 3,5,7,10,13
3.very soft rock
5.soft rock
7.medium rock
10.hard fissured rock
13.hard homogeneous rock
5

fragmentation  size in cm:
9.846223
Do u want to write input and output  to  textpad
y_
```

4.4.2.2 Design model In Net Beans

Software: To make it more user's friendly, presently it has been developed in java platform with the help of software "Net Beans 6.0 IDE".The Net Beans Platform is a reusable framework for simplifying the development of other desktop applications. When an application based on the Net Beans Platform is run, the platform's Main class is executed. Available modules are located, placed in an in-memory registry, and the modules' start-up tasks are executed. Generally, a module's code is loaded into memory only as it is needed. Applications can install modules dynamically. Any application can include the Update Centre module to allow users of the application to download digitally-signed upgrades and new features directly into the running application. Reinstalling an upgrade or a new release does not force users to download the entire application again.

Among the features of the platform are:

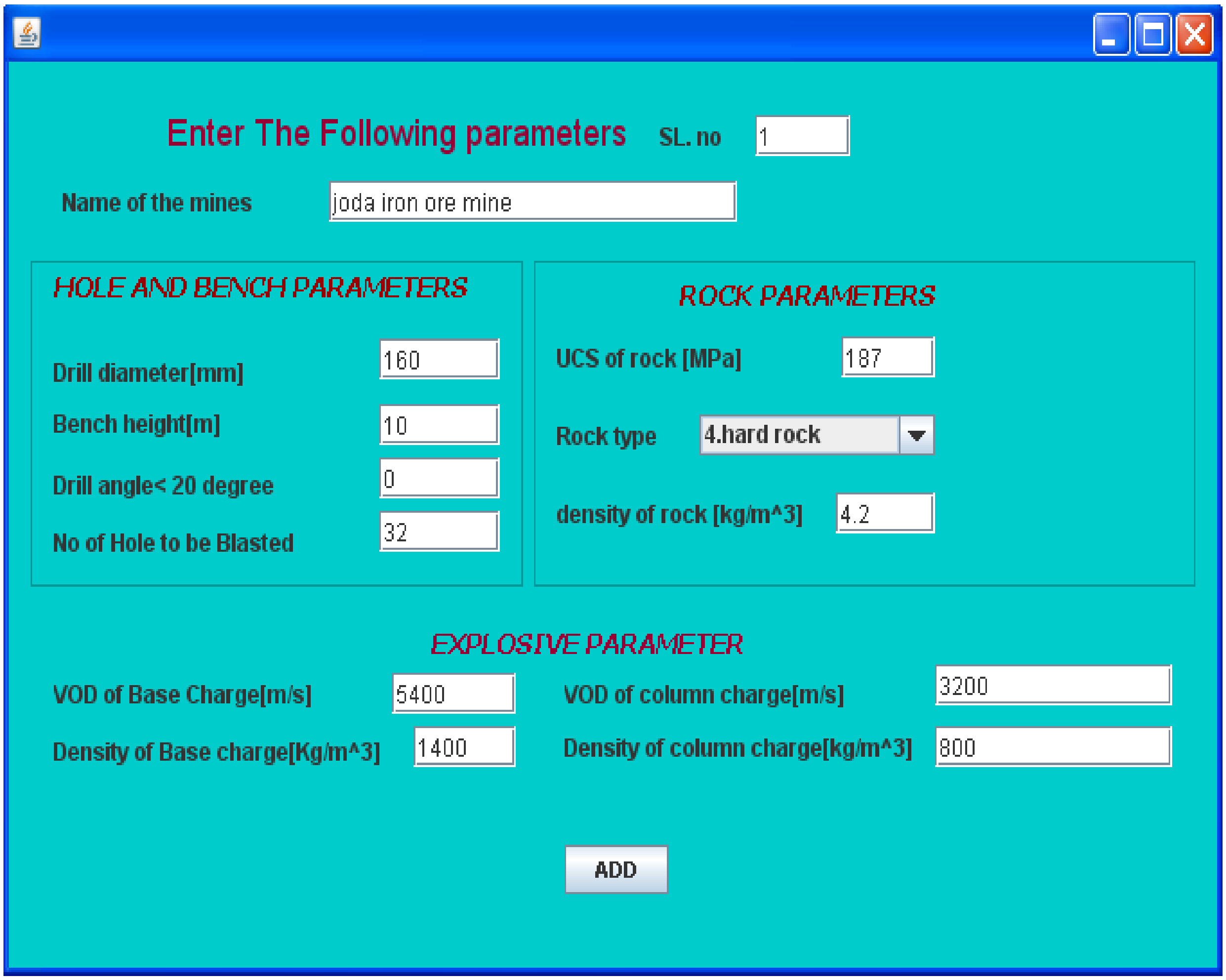
- User interface management (e.g. menus and toolbars)
- Storage management (saving and loading any kind of data)

STARTING SCREEN:

It is the starting page which welcomes the user and asks the user to click the "Ok" button to continue & will show the input page. The user can add new data to the database by clicking on the button "Add New DATA". The user can also see the existing data base.

ADD PAGE:

This page allows the user to enter the name of the mines ,hole parameters,rock parameters,bench parameters, explosives parameters for storage & further calculation. For each field it has got a check to catch the invalid data. when the user clicks the submit bottom and all the data provided are correct,it automatically adds the data to the pre existing database.it got the facilities to edit & delete an pre-existing data in the database but it is password protected.

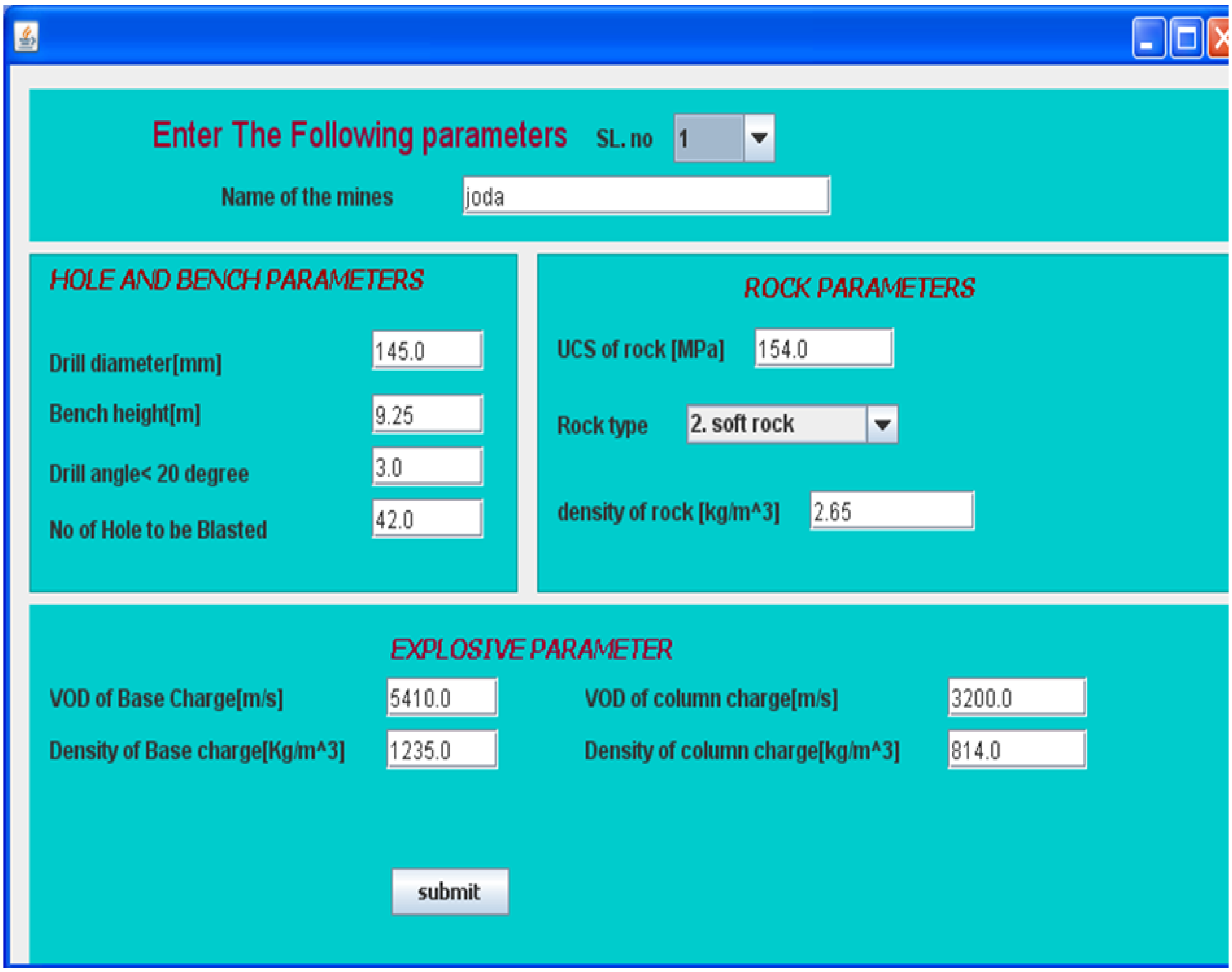


WRONG ENTRY:

If a user makes a mistake during adding a data to the databases or providing an erotic data or enter an invalid number it will show error message box. if in case the user provides an charcter, in a number valued place,the software will show an “invalid decimal” error.if the data provided are not in range then it will show an error “invalid data”. it will check the error at a particular field at one time showinig ”****” in red colour.After correcting this field it will check for another field.when all the data are correct and user press the “continue ” button, the data will be automatically added to the data base and will return the result page.

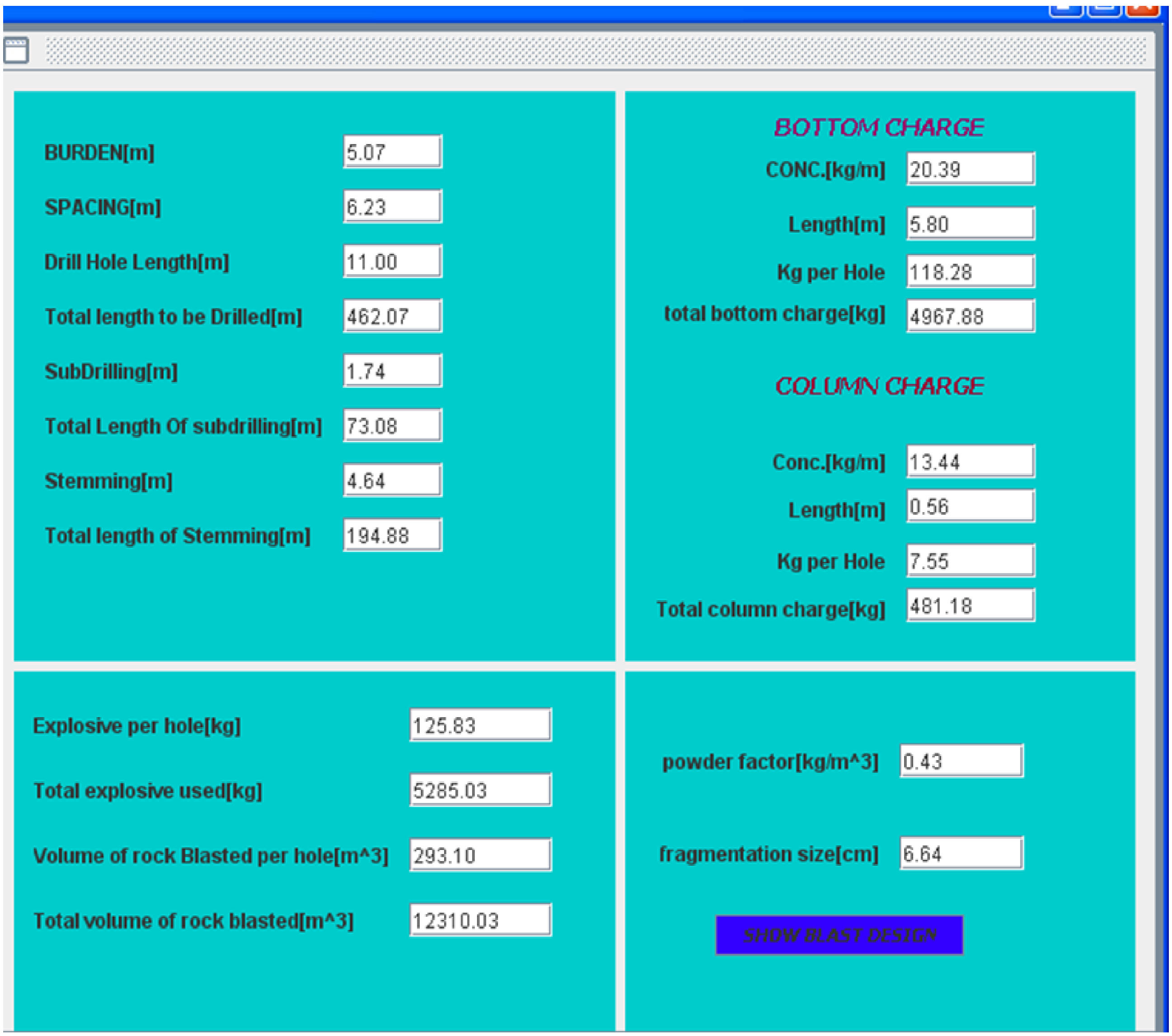
INPUT PAGE:

This allows the user to select a serial number from a combo box (in which pre designed data are already stored) which refers to a particular mine or blasting parameters. Actually it is retrieving the data from the database (ocblast.accdb) Here the user can not change, edit or delete the data. When the user clicks "SUBMIT" button it calculate and show the desired blast parameters in the result page.



OUTPUT PAGE:

This page shows the desired parameters of blast design after calculation. Mainly the parameters are burden, spacing, sub drilling, stemming, length, hole, bottom charge, column charge, explosives used, volume of rock blasted, and fragmentation size. It gives details about column charge and bottom charge used in blasthole. From this page the user can terminate the program or can move to the home page. In future version of the software the user can also see the blast design graphics and different blast patterns.



4.4.3 Field Trials

The software has been tasted in two mines; one coal mine in Orissa and an iron ore mine in eastern India. Some of the important parameter like volume of rock blasted, powder factor and average fragmentation size are quite matching

INPUT DATA

Name of mines	UCS of rock [MPa]	Bench height [m]	Drill hole Dia. [mm]	Angle of drill [<20 °]	VOD of base charge [m/s]	Density of base charge [kg/m^3]	VOD of column charge [m/s]	Density of column charge [kg/m^3]	No of hole
Iron ore mine	187	10	160	0	5400	1400	3200	800	32
Coal mines	35	12	160	0	5400	1400	3200	800	24

OUTPUT DATA

PARAMETER	Iron mine in eastern India	Coal mines in Orissa
Length Of Drill Hole[m]	11.920000	13.600000
Burden[m]	5.280000	6.240000
Spacing[m]	6.080000	8.160000
Length Of Sub Drilling[m]	1.920000	1.600000
Length Of Stemming [m]	4.680000	5.600000
Explosive Per Hole (Kg)	172.189819	186.585897
Volume Of Rock Blasted Per Hole(m^3)	321.023987	611.020752
Powder Factor(Kg/m^3)	0.577213	0.305368
Fragmentation size(cm)	12.88	14.6

CHAPTER: 05

CONCLUSIONS

Results and Discussions

Future Work

CHAPTER: 05

CONCLUSIONS

5.1 Result & Discussion:

Parameters influencing surface blast design have been reviewed extensively. The key parameters having significant influence are identified. Different researchers, namely, Langefors and Kihlstrom, Lopez &Jimeno, Ash, Bhandari, Singh & Sarma, Thote and Singh, Andersen etc have utilized some of these parameters to arrive at suitable blast design. Among these, the most popular one is blast design theory proposed by Langefors and Kihlstrom (1978). The calculation of this design concept needs longer time duration (for hand calculation) to arrive at the design solution. So, it was felt to establish a user-friendly computer program to assist the basting engineers to arrive at the blast design.

The developed software is user-friendly and easy to use. As the software is developed based on the empirical relationship, the software has limited utilization. However, with the invented new explosives, drilling and blasting pattern the software should be modified in the subsequent versions. In this software the input has to be provided through keyboard and the input & output can be written to a text file for further use. The software is using many important parameters like rock parameter, explosives parameter as well as bench parameter. The software has been tested in two mines; one coal mine in Orissa and an iron ore mine in eastern India. Most of the important parameter like volume of rock blasted, powder factor and average fragmentation size are closely matching with the field results.

To make the software more user-friendly, the software has also been developed in net beans. A data base is also available with the software to make it more useful and less time consuming. This software helps in reducing time and allows the user to come out with the best solution with number of iterations.

5.2 Future work

At present the software is in demo version which was initially developed in C++ language and subsequently modified in net beans. The graphics of the blast design will be included in subsequent versions. Further, it will be modified for newly developed explosives. A cost calculation package would also be included for optimization of the blast pattern.

CHAPTER: 06

REFERENCE

CHAPTER: 06

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DESIGN OF SURFACE BLASTS- A COMPUTATIONAL APPROACH

Author: Ashutosh Mishra

Supervisor: Kaushik Dey

Abstract: A major part of mineral production comes from surface mining and there has been a rapid growth in this sector with the deployment of high capacity equipment .Increased production can be achieved from large capacity surface mines using heavy earth moving machineries. These machineries involve high capital cost, and thus, the mining engineers should plan to achieve the best performance from these machineries. Performance of them, especially the excavating and transporting equipments are largely depending on the blast results, particularly, fragment size, distribution and muck profile. Thus, proper blast design with a computational approach is a vital factor that affects the cost of the entire mining activities.

Introduction: Various approaches to blast design for surface mines have been reviewed to understand the present state of knowledge in this field. The blast design approaches such as trial and error and cratering are not suitable for large scale blasts in surface mines. The empirical method continues to be the most common way to calculate the design parameters. Nevertheless, an integration of empirical method, computer modeling, and instrumented field trials effectively contributes to the state-of-the- art in blast design. In this paper, the controllable and uncontrollable parameters, which have significant effect on surface blast design, are identified. Based on the model proposed by Langefors and Kihlstrom (1978), a computer model is prepared.

Objective: The basic objective of the project is to develop a computer model which has the following facility

a) Designing of different parameters of a surface blast
b) Achieving the desired fragmentation size

Methodology: The primary concept behind this blast design is the model proposed by Langefors and Kihlstrom (1978) to design different parameters, which is given in following tables.

Blast design for small blast hole diameters(65mm-165)				
Design Parameter	Uniaxial compressive strength (MPa)			
	Low < 70	Medium 70-120	High 120-180	Very High > 180
Burden - B	39 x D	37 x D	35 x D	33 x D
Spacing - S	51 x D	47 x D	43 x D	38 x D
Stemming - T	35 x D	34 x D	32 x D	30 x D
Sub drilling - J	10 x D	11 x D	12 x D	12 x D
Bottom charge length l_f	30 x D	35 x D	40 x D	46 x D

Blast design for large blast hole diameters(166mm-450mm)			
Design Parameter	Compressive rock strength (MPa)		
	Low < 70	Medium-high 70-180	Very High >180
Bench Height H	52 x D	44 x D	37 x D
Stemming - T	40 x D	32 x D	25 x D
Burden – B(ANFO)	28 x D	23 x D	21 x D
Spacing – S(ANFO)	33 x D	27 x D	24 x D
Burden – B(EMULSION)	38 x D	32 x D	30 x D
Spacing - S (EMULSION)	45 x D	37 x D	34 x D
Bottom charge length l_f	8 x D		
Sub drilling - J	J=5+ (0.450-D)/0.09467		

The mean fragment size can be calculated (Kuz-Ram model) by the following equation

$$X = A \times \left(\frac{V}{Q}\right)^{0.8} \times (Q)^{0.167} \times \left(\frac{E}{115}\right)^{-0.633}$$

Where, X = mean fragment size, cm

V = volume of blasted rock, m^3

Q = mass of explosive charge per hole, kg

E = relative weight strength of explosive (ANFO= 100)

A = a constant based on rock factor (depends upon rock density, strength and jointing).

Discussion and Conclusion:

Different parameters to be considered for designing a surface blast have been reviewed extensively. The key parameters having significant influence are identified. The calculation of this design concept is complex and needs larger time for hand calculation, in turn, arrive at the design solution. So, it was felt to establish a user friendly computer program to assist the basting engineer to arrive at the blast design. The developed software is user-friendly and easy to use. The software is using many important parameters like rock parameter, explosives parameter as well as bench parameter. As the software is developed based on the empirical relationship, the software has limited utilization. However, with the invented new explosives, drilling and blasting pattern the software should be modified in the subsequent versions. The software has been tested in a number of mines in India. A data base is also available with the software to make it more useful and less time consuming.

References:

1. Jimeno L.C & Jimeno L.E. Drilling & Blasting of Rocks. New York, 2006, pp.191-196,328
2. Bhandari S. Engineering Rock Blasting Operations,3000 BR Rotterdam, Netherlands,1997,pp. 185-207
3. Software: NetBeans 6.5 IDE, TURBO C++, Windows XP.