OPTIMIZATION OF BLASTING PARAMETERS IN OPENCAST MINES

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

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

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CHINMAY KUMAR PARIDA

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National Institute of Technology
Rourkela

C E R T I F I C A T E

This is to certify that the thesis entitled "**Optimization of Blasting Parameters in Opencast Mines**" submitted by Sri Manmit Rout (Roll. No.: 10305019) and Sri Chinmay Kumar Parida (Roll. No.: 10305017), in 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.

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A C K N O W L E D G E M E N T

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Date: **Chinmay Kumar Parida**

Manmit Rout

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ABSTRACT

Drilling and blasting are the major unit operations in opencast mining. Inspite of the best efforts to introduce mechanization in the opencast mines, blasting continue to dominate the production. Therefore to cut down the cost of production optimal fragmentation from properly designed blasting pattern has to be achieved. Proper adoption of drilling and blasting can contribute significantly towards profitability and therefore optimization of these parameters is essential.

Introduction

Rock breaking by drilling and blasting is the first phase of the production cycle in most of the mining operations. Optimization of this operation is very important as the fragmentation obtained thereby affects the cost of the entire gamut of interrelated mining activities, such as drilling, blasting, loading, hauling, crushing and to some extent grinding. Optimization of rock breaking by drilling and blasting is sometimes understood to mean minimum cost in the implementation of these two individual operations. However, a minimum cost for breaking rock may not be in the best interest of the overall mining system. A little more money spent in the rock-breaking operation can be recovered later from the system and the aim of the coordinator of the mining work should be to achieve a minimum combined cost of drilling, blasting, loading, hauling, crushing and grinding. Only a “balance sheet” of total cost of the full gamut of mining operations vis-à-vis production achieved can establish whether the very first phase- rock breaking- was “optimum” financially; leaving aside factors of human safety.

An optimum blast is also associated with the most efficient utilization of blasting energy in the rock-breaking process, reducing blasting cost through less explosive consumption and less wastage of explosive energy in blasting, less throw of materials, and reduction of blast vibration resulting in greater degrees of safety and stability to the nearby structures.

Development of a Blast Optimization Model

Selection of proper explosive in any blasting round is an important aspect of optimum blast design. Basic parameters include VOD of explosive (m/s), Density (g/cc), Characteristic impedance, Energy output (cal/gm), and Explosive type (ANFO, Slurry, Emulsion etc.). However, all these parameters can not be taken for optimizing the blasting method successfully. Some of the parameters are taken for minimizing the blasting cost. These cost reduction and optimum blast design parameter will give an economical result. The parameters are

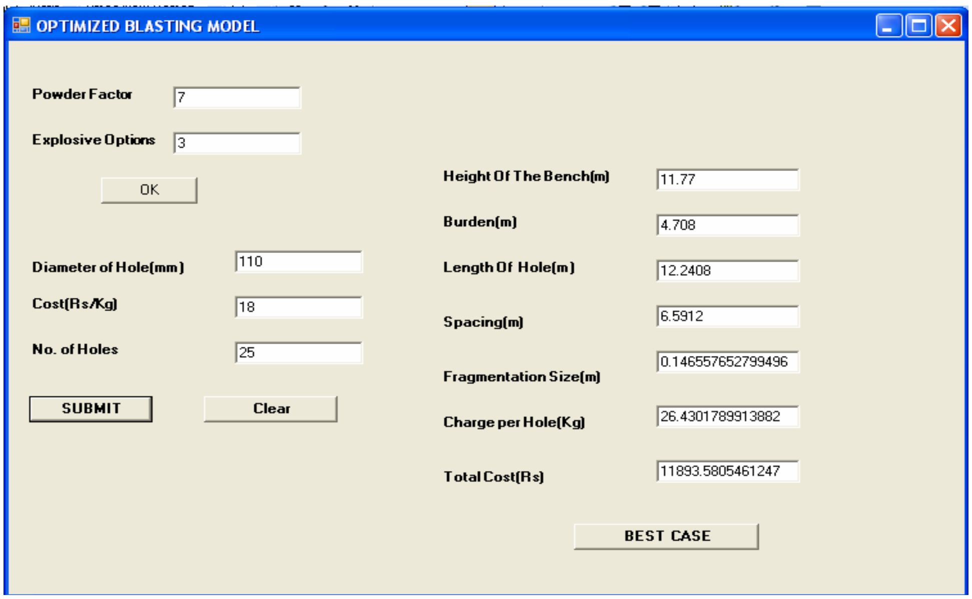
i. Drill hole diameter,
ii. Powder factor (desired),
iii. Cost of explosive,
iv. Numbers of holes required to blast.

Methodology

The study of the various parameters of blasting suggests that the powder factor should be constant as per the requirement. The number of holes desired as per the explosive, the drill

hole diameter as available and the cost of explosive are kept as input. The spacing, bench height, burden, charge per hole as depending on the previous parameters can be calculated. From the different input and calculated parameters the total cost of the method is calculated and the least expensive method is selected as the optimized model.

Blasting related information were collected from three different mines of Mahanadi Coalfields Ltd.(MCL) for implementation of the optimization model. A program was designed using visual basic on .net platform taking the above parameters into consideration to select the optimized model. It was observed that the program gives satisfactory results. A sample output of the program is as presented below:



Conclusion

The blast optimization model has been developed with simple methodologies which can be adopted by the mining industry to compare the explosive costs and achieve better blasting results and. The model developed is a user friendly one, since by keeping the powder factor and number of choices of explosives available as constant and by varying the parameters like drill hole diameter, number of holes and cost of explosives one can compare the explosive performance and accordingly take a decision to select the proper type of explosives for blasting.

It may be noted, that the model has been developed based on case studies of three different mines of MCL, and it can be modified with collection of information from a large number of mines.

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CHAPTER 1

INTRODUCTION

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INTRODUCTION

Mining industry is the backbone for the development of any nation. In mining the basic aim is to achieve maximum extraction of minerals keeping in view the environmental, economic and lease constraints. With the advancement of civilization, the requirement of different minerals has increased manifold to meet this demand. There is an upsurge in interest and action in opencast mining because of the improved productivity, recovery and safety of mining operation. Improvement in production has been achieved with the help of large capacity opencast machineries, continuous mining system with improved design, development of modern generation, explosives and accessories, process innovations and application of information technologies and increased adoption of computerized mine planning and control.

Drilling and blasting are the major unit operations in opencast mining. Inspite of the best efforts to introduce mechanization in the opencast mines, blasting continue to dominate the production. Explosives contribute currently about 5% of the direct cost of production and if the aggregate cost of drilling and blasting is taken together, this may go as high as 30% of direct cost of production. Therefore to cut down the cost of production optimal fragmentation from properly designed blasting pattern has to be achieved. Fragmentation of rock represents one of the key problems in maximizing economic efficiency for exploitation of mineral deposits. 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. 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 overall cost of mining to optimum level.

Drilling and blasting cost in any project can be as high as 25% of the total production cost. Inspite of this the design and implementation of a blast is not given that much priority in our country. Proper adoption of drilling and blasting can contribute significantly towards profitability and therefore optimizations of these parameters are essential.

Optimization means achieving the best i.e. to achieve maximum or minimum value of the operating parameters. Optimization of blast is dependent on a host of complex factors related to the rock, explosive, initiation, drill-hole parameters and their layout. The present work is a step in the direction of developing a suitable blast model, with simple methodologies which can be adapted by the mining industry to achieve better blasting results.

CHAPTER 2

LITERATURE REVIEW

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LITERATURE REVIEW

Verma (1993) advocated that performance rating of explosives has become a primary need because of the growing requirement and competition. 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 blastability 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, fly 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 choosen explosives, still a lot has to be done for blast management and control.

Nanda (2003) advocated that operation research facilitates in describing the behaviour of the systems, analyzing the behaviour by constructing appropriate models and predicting future behaviour 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.

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.

Kumar et al (2004) 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.

Chapter 3

DRILLING AND BLASTING IN LARGE OPENCAST MINES

Drilling

Blasting

Recent Advancements in Drilling and Blasting Techniques

CHAPTER - 2

DRILLING AND BLASTING IN LARGE OPENCAST MINES

3.1 DRILLING

There are two forms of rock breakage viz., rock penetration and rock fragmentation. The former includes drilling, cutting, boring etc., while the latter includes blasting etc. The term rock penetration is preferred for all methods of forming a directional hole in the rock. There are many types of rock penetration depending on the form of energy application, viz. mechanical, thermal, fluid, sonic, chemical etc. The mechanical energy, of course, encompasses the majority (about 98%) of rock penetration applications today. The application of mechanical energy to rock can be performed basically in only one of the two ways: by percussive or rotary action. Combining the two results in hybrid methods termed roller-bit rotary and rotary-percussion drilling.

In surface mining, roller bit rotaries and large percussion drills are the machines in widest current use, with rotary drills being heavily favoured. Drilling is performed in order to blast the overburden, ore deposit, coal seams etc., so that the power requirement for excavators to extract the materials becomes less. This also reduces the wear and tear of the excavators, increases their life, reduces clearing time of materials, and decreases operation cost. Drilling holes are usually made in a zig-zag pattern .The spacing between the rows and column is of equal length. Certain empirical rules are followed for this spacing and the depth of holes as indicated below (Dey, 1995).

For hard rock: 1 : 4

Where 1 = horizontal space, 4 = vertical drill depth

For loose material: 1 : 6 ratio

3.1.1 Classification of Drilling Systems

Drilling machines used in surface mining projects, construction work, etc., can be classified in the following ways (Dey, 1995):

1) Depending upon the principle of working
    i) Percussive Drilling

ii) Rotary Drilling
iii) Rotary-percussive Drilling

2) Depending upon types of prime mover
   i) Used diesel driven drilling machine
   ii) Electrically driven drilling machine

3) Depending upon the means of power transmission
   i) Pneumatically operated machine
   ii) Hydraulically operated machine
   iii) Electrically operated machine in combination with hydraulic and pneumatic system.

3.1.2 Percussion Drilling

Percussion drilling penetrates rock by the effects of successive impacts applied through the bit which is typically of chisel type. The bit/tool rebounds and impacts again after rotating slightly thus every time hammering a new surface and also to maintain a circular shape of hole. The stress effective in breaking the rock acts essentially in an axial direction and in a pulsating manner. The rotational torque applied is not responsible for breakage of rock by the tool. This torque is usually small in magnitude and operates during rebound only.

Under the action of these impacts the rock is first elastically deformed with crushing of surface irregularities. Then main sub-surface cracks are formed. These are radial cracks from the edge of the bit. At the edge of the bit a wedge of the crushed rock is also formed. This leads to the formation of rock chips which are removed by the cleaning action of any circulating fluid. The sequence is repeated with succeeding blows and turning of the bit. The two predominant mechanisms in percussion drilling are CRUSHING and CHIPPING.

Percussion drills generally play a minor role as compared with rotary machines in surface mining operations. Their application is limited to production drilling for small mines, secondary drilling, development work, and wall control blasting.

There are two main types of drill mounting. The smaller machines utilize drifter-type drills placed on self-propelled mountings designed to tow the required air compressor. Typical hole sizes are in the 63 to 150mm (2.5 to 6 in.) range. The larger machines are crawler-mounted and self contained. Drill towers permit single pass drilling from 7.6 to 15.2m with hole sizes in the range of 120 to 229mm. These larger machines are almost exclusively operated using down the hole hammers. For many years these machines were exclusively operated using pneumatic hammers. But in the last 20 years hydraulic machines have been introduced in the smaller size range.

3.1.3 Rotary Drilling

As the name suggests, the boring tools used in this method are rotated and they crush, cut or abrade the rock. The rate of drilling depends on

- Nature of the rock
- Pressure exerted by drilling bits and rods
- The rpm of the bit
- Type of drilling bit

The simplest form is the hand augur. These are attached to rods and rotated by means of a simple cross bar. In this method hollow drill rods of steel or aluminum are used. These are thread connected and transmit torque and feed pressure to the drilling bit or drilling tool, which is attached at the end of column of the drill rods. Rotation of the drill rods is through gearing driven by a prime mover at the surface. The drill bit attacks the rock with energy supplied to it by a rotating drill rod, while a thrust is applied to it by a pull down mechanism using upto 65% of the weight of the machine, forcing the bit into the rock. As the rods rotate, the drilling tool/bit breaks the rock (by either a ploughing-scrapping action in soft rock, or a crushing-chipping action in hard rock, or by combination of the two) and the cuttings are cleared by pumping water under pressure or compressed air down the hole through the hollow drill rods. The air both cools the bit and provides a medium for flushing the cuttings from the hole.The water or air, along with the cuttings, comes to the surface through the space between the drill rods and the sides of the drill hole.

The bit moves forward by the effect of torque and thrust simultaneously applied to the rock surface. The mechanism of penetration rate are related to shearing and friction processes. The shearing action of the leading edge of the cutting component produces chipping, whereas friction creates wear of the bit-rock interface.

Blast hole sizes produced by rotary machines vary in the size range of 100 to 445mm diameter with the most common sizes being 200, 250, 311, and 381mm in diameter. These drills usually operate in the vertical position although many types can drill up to 25 or 30^0 off the vertical. To achieve high drilling speeds, and to drill holes to greater depths, three power driven rotary methods are available viz., hydraulic rotary drilling, diamond drilling, and chillied –shot drilling

3.1.4 Rotary –Percussion Drilling

This is a hybrid form of drilling. In a rotary- percussive machine the advantages of both rotary and percussive principles are used for making the hole. Here blows are imparted and the tool is also rotated during drilling action. Generally, percussion bits (with buttons or asymmetric wings) or sometimes roller bits are used. The superimposing of percussion on a rotary system means that higher impact forces are realized than in straight rotary drilling, but thrust and torque-induced forces are still operative. In rotary-percussion drilling, rock failure occurs by crushing and chipping, the proportion being a function of the drilling action.

3.1.5 Drill Bits

A bit is the applier of energy in the system, attacking rock mechanically to achieve penetration. The common drilling bits being used in large opencast mines are Drag bit, Carset bit, Tricone rock roll, Button bit.

Drag bit: they have three or four cutting wings tipped with carbide inserts and usually an A.P.I. regular threaded pin connection. Blade bits have a similar cutting action, except that the blades can be replaced.

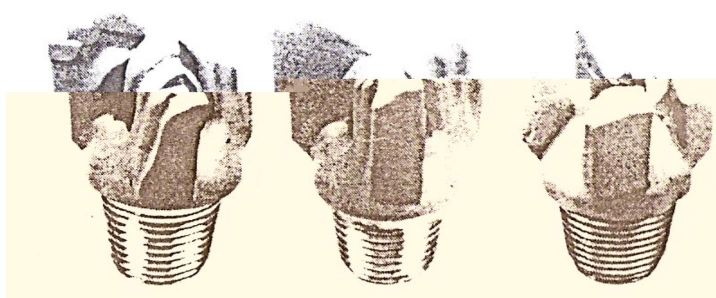


Figure 3.1: Drag Bits

Carset bit: The drill bit in this case is essentially a cross bit tipped with tungsten carbide and it is an integral part of the unit called a carset bit. It has five air holes (one at the center and four on the sides). The drill bit is usually fitted in a 1.5 m long pipe like device known as hammer. This hammer contains a piston and valve arrangement. During operation, compressed air passes down the hollow drill rods through filipper valves and experts pressure on the piston, which in turn strikes the bit. Air then enters the bottom of the piston through a passage way around the cylinder when it is at the ends of its down stroke, and lifts the piston up. In this piston the passage way is cleared and the entrapped air below is released through the carset bit and there by cleared again. The piston is then struck by an air stream at its top and this forces it down and thus the process of up and down movement of the piston gets going.

In pneumatically operated drilling machines, the piston strikes the carset bit about 1000 times/minute at full air pressure. The drilling action in such cases, that is, movement of the drill down hole, takes place on account of atomization of rock due to constant pounding on it by the carset bit. Theses bits have line contact with the rock and constant impact while breaking and atomizing wears out the contacts. The cutting edge as well as the periphery needs grinding for further use.

Tricone Rock Roller Bit: In rotary drilling machines, which are electrically driven in combination with pneumatic and hydraulic systems, the drilling tool is a tricone rotary bit. This consists of three truncated cones placed 120° to each other. The surface of these truncated comes have a number of cutting teeth and they are mounted on two bearings, one a roller bearing and the other a ball bearing and. Roller bearings are positioned to support the radial load and free of thrust loads acting longitudinally along the bearing pin. Bearing pins in rock bit heads are forged integrally with each section of the bit body. The teeth of the cones are hard faced to give resistance to abrasive wear. The bits are fitted with three air blast nozzles which direct the air blast coming through a drill pipe to the bottom of the wall intersection. This helps in quicker and more efficient removal of cuttings. The size of the nozzle required in the bit

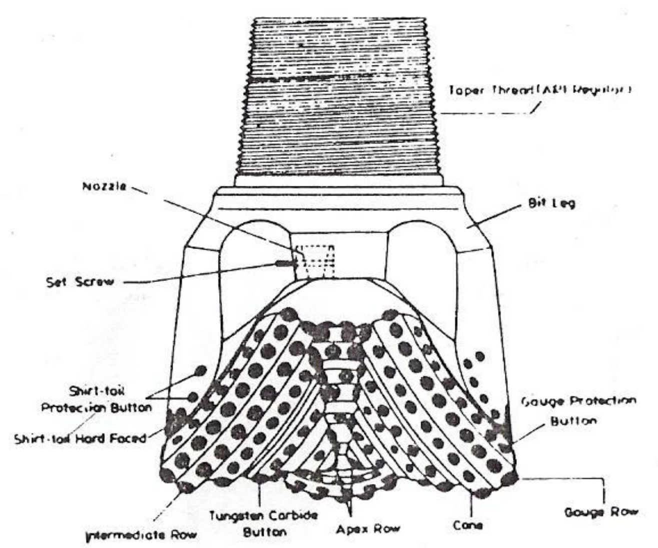


Figure 3.2: Tricone Rock Roller Bit

depends on the volumetric output of the compressor and its operation pressure capacity. The nozzle size should be such that it only clears but also cools the equipment. Rotary speed varies from 60 to 120 rpm for a steel- toothed bit and 50 to 80 rpm tungsten carbide bits. The normal life of such a bit is about 2500 m. this bit is not repairable and has to be disposed of after use for 2500 m.

Button Bit: Button bits have cylindrical bodies with a larger diameter head on the top and the stem is spline shaped. The head is chamfered on the sides. A number of hard metal balls in the shaped of a hemisphere are sintered on the head and on the side to flush cutting from the drill holes. There are certain vertical slots at the side to provide a passage for the cutting to come out of the holes. Rotational speed varies from 10 to 20 rpm. The bodies are made of alloy steel and heat treated. The hole diameter varies from 100 to 210 m.

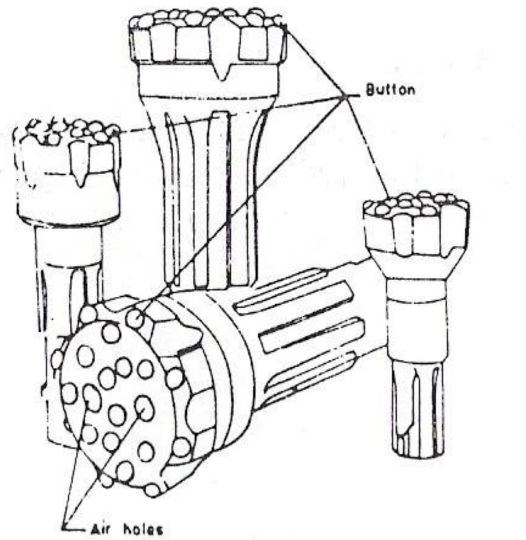


Figure 3.3: Button bit

3.1.6 Feed Mechanism

The pressure acting on the bit into the rock is controlled by an arrangement known as "*feed mechanism*". The feed mechanism is hydraulic for deep holes, but may be replaced by a screw feed for shallow holes. Beyond a depth of nearly 60m, the weight of the rods keeps the bit pressed against the rocks and the feed mechanism may not be necessary. At greater depths the feed mechanism is operated in such a way that the weight on the drill bit is not excessive.

Three types of such mechanisms are used in drilling machines. They are:

1. Pneumatically operated mechanism
2. Hydraulic operated mechanism
3. Rope pulley operated mechanism

Pneumatically operated mechanism: This consists of an air motor, transmission system and chain drive. The air motor is driven by compressed air, drives the sprocket chain arrangement through a gear box or a belt pulley system. The rotary head is placed on a chain which reciprocates during the raising and lowering of the chain.

Hydraulic operated mechanism: This can be classified as two different types: (a) by the use of hydraulic rams only, (b) by the use of hydraulic rams in combination with a rope pulley arrangement. The first is consisting of a hydraulic tank, a hydraulic pump, a two-way valve, a feed control valve, a hydraulic cylinder, a cross head, and pipelines.

The hydraulic pumps is a vane type variable delivery unit which discharges the hydraulic fluid either at the top of the cylinder or at the bottom of the cylinder through a two way valve, there by extending or retraction the piston cylinder assembly, which finally provides the feeding action of the machine. There are two return lines, one form the pumps and the other form the two-way valve so that excess oil may be allowed to come back to the tank. This two-way valve is equipped with built-in relief valve of a differential plunger design to ensure accurate and uniform maintenance of pressure. The oil pressure gauge on the oil pump line indicates the oil pressure in the system. There is a feed control valve in the piping, leading form the bottom of the cylinder. It is adjusted to regulate or stop the advance of the bit.

The second method is usually used in electrically driven drilling machines. The pistons of the hydraulic cylinder actuate the hydraulic motor through a rope pulley arrangement as shown in the figure.

Rope pulley mechanism: A rope pulley operated system uses purely mechanical components. Here the rotary head is allowed to move on a guided path. The top of the head is connected to a rope and this rope allows to pass over the auxiliary reel at the top and then around a bull reel in the middle. Finally, the rope is connected to the bottom of the rotary head structure after passing over the bottom reel. The middle reel, that is the bull reel, is powered by a prime mover. When the bull reel rotates in a clockwise direction the rotary head is raised and as the reel is rotated in a counter in a counter-clockwise direction the rotary head is lowered, providing feed for the drill rod.

3. 1.7 Power Transmission System

Transmission of power in drilling machines is of two types:

(a) In combination with pneumatic and mechanical means.

(b) Electric, pneumatic and mechanical means

Flow system for the first type is as follows:

This type of machine usually consists of an engine, which drives and air compressor. Air on being compressed is stored in a tank, and then taken into a separator and control chamber, from where it is feed into three different sub systems. On one side, air is used to drive the rotary head, which drives the drill rod and bit through the gear box. On the other side, the air drives a feed motor, which drives the driving sprocket of the endless chain through a reduction gear box to provided feeding of the machine. The last portion of the compressor air is forced through the drill rod and bit, which finally forces the cutting chips out of the drill hole, that is, at the top of the hole, which is sucked in by the vacuum pump and is discharged at a distance from the hole.

3.1.8 Drilling Machines Used in Large Opencast Mines

In the mining cycle, drilling performed for the placement of explosives is termed as production drilling. Some of the very common and widely used drilling machines for production drilling are discussed here.

Pneumatically Operated Wagon Drill

Compressed air operated drills mounted on a mobile frame are known as wagon drills. The frame is usually mounted on tyred wheels. Figure 3.4 shows such a machine. They can be pulled by the operator and his helper to the hole site on level ground. The size of the drill hole varies form 50 to 100 mm for a depth of 3 to 15m. These drills have a separate compressor unit. Most drills are usually 3 m long providing a 3 m vertical travel.

The mast is capable of swiveling from the vertical to a horizontal position and it can be kept at any angle between the horizontal and the vertical, thereby facilitating vertical, horizontal or inclined drilling up to $40^{0.}$ The framework rests on three wheels. There is only one wheel at the rear which helps in steering the machine, while the two front wheels are the main load bearing wheels. The mast of the drill is placed near the rear of the machine. The control units, such as the valve, are placed behind the mast. On the right hand side of the machine is a hand operated reciprocating

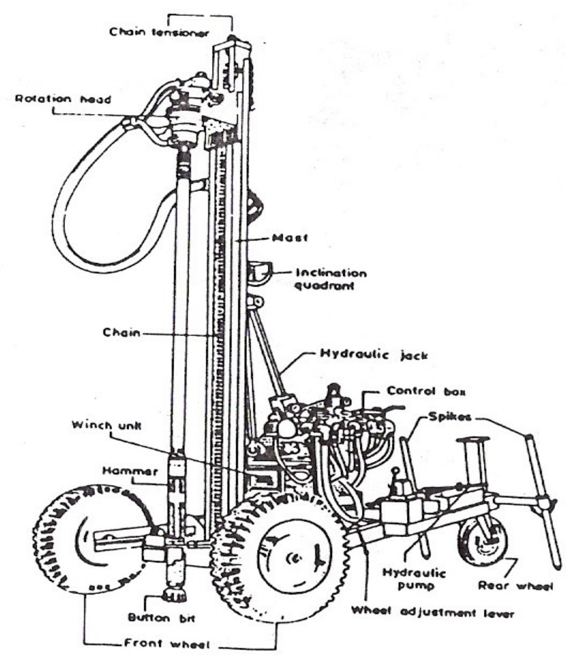


Figure 3.4: Pneumatically operated wagon drill

pump which raises the mast form the horizontal to the vertical position. At the rear there are two spikes which help to hold the machine during drilling action.

Blast hole drills

Bigger drilling machines, which produce holes to be blasted in order to facilitate higher capacity excavators, are termed as blast hole drilling machines. In fact, all drilling machines which make holes for blasting purposes should be termed as blast hole drills. However, it is a common practice to refer only to bigger capacity machines as such. The hole diameter varies form 100 to 300 mm up to depth of 60 m with a drilling speed varying form 1.8 to 24 m/hours.

This machine usually consists of a prime mover (either a diesel or an electric motor) which drives the air compressor, the hydraulics pumps, the rotary head and other auxiliary components. The main function of the air compressor is to supply compressed air, which is forced down the hole through drill rods. To remove cutting from the hole so formed during drilling action. For pneumatically driven drilling machine, a portion of the compressed air is utilized to run the rotary head through an air motor and also to operate the feed mechanism of the machine, besides cuttings. All other main functions such as drill rod rotation and feed mechanism movement are performed by a hydraulic motor run by hydraulic pumps which are in turn operated by electric motors. The third type of machine, which is electrically driven in combination with pneumatic and hydraulic system, uses an electric motor for drill rod rotation while a hydraulic motor run by the hydraulic system operates the feed mechanism in addition to the removal of cuttings by compressed airs previously.

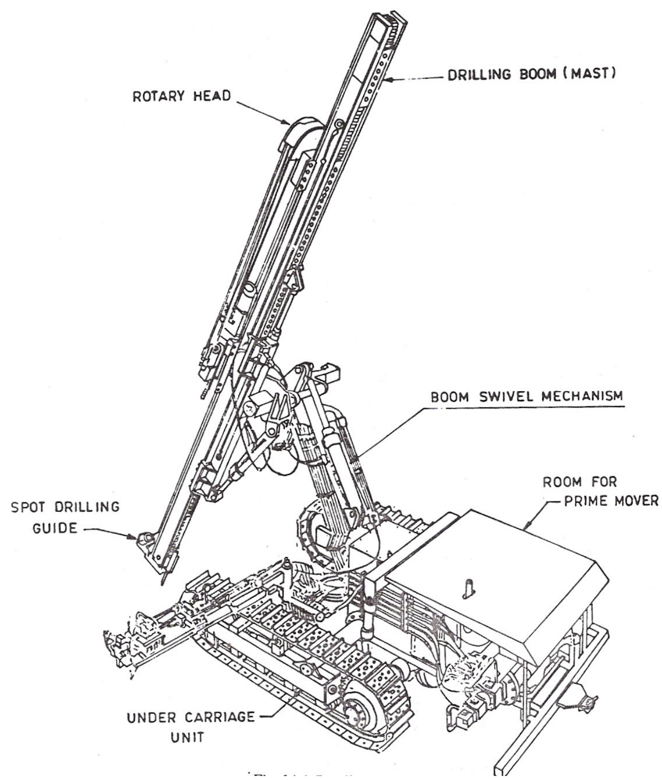


Figure 3.5: Blast hole drill

The components of these drilling machines are as follows:

i) Drill bits	ii) drill rods	iii) mast assembly
iv) Feed mechanism	v) power transmission system	vi) undercarriage unit

Drill bit: There are three types of drill bits used in the above type of drilling machine. They are

a) Carset but b) Tricone rock roll c) Button bit

Mast assembly

This is vertical structural frame work with feed mechanism that is either a chain or hydraulic type. The rotary head (air motor/hydraulic motor/electric motor) is placed at the top and is capable of traveling along the feed mechanism, downward or upward. Drill rods are attached to the rotary head through a coupling and gear box. The whole of this frame work is held in position by means of hydraulic piston cylinder arrangement and is capable of swiveling the mast in vertical plane.

Drill Rod

Blast hole drills use a heavier type of hollow steel called rod or pipe that is designed to convey torque rather than impact. It uses America Petroleum Institute (API) steeply tapered threads, male type at one end female type at the other. They are made of medium carbon.

It has got three parallel paths for performing three different functions. The AC induction motor is used to drive a screw type air compressor and the air is stored in a tank. This air is allowed to pass through a separator for removing moisture and fairly dry air is forced through the drill rods and bit to remove the cuttings form the hole.

There is a blower fan run by a motor, which is placed near the drill hole. As the cuttings are lifted form the hole, this blower fan laterally throws the cuttings some distance away form the hole by the air stream. The AC supply is used to drive the rotary DC motor after passing through a rectifier. This DC motor drives the drill rod and bit through the gear box and tyre coupling.

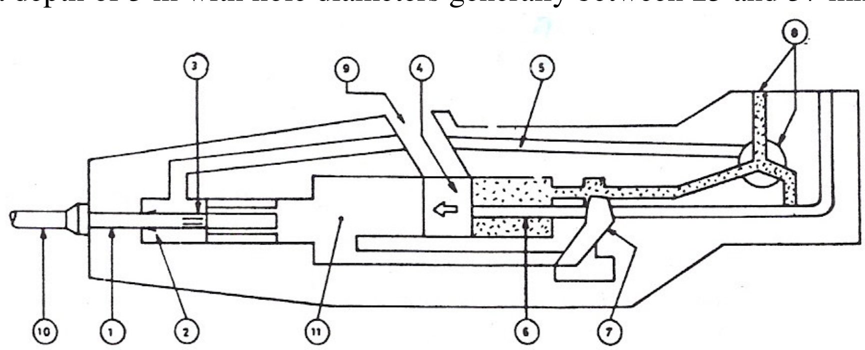
Undercarriage unit: This unit is mounted on both a tyred wheel system and a crawler mechanism. A structural framework mounted on three wheels is the common feature of the former type while the latter type consists of two crawler mechanisms on which the whole machine rests. Each mechanism is provided with a separate driving sprocket, driven wheel and an endless chain run by the driven sprocket. The tension adjustment arrangement for each mechanism is provided to adjust the tension as required for the purpose. There are certain carrier rollers, which guide and carry the endless chain. The undercarriage unit is also provided with hydraulic jacks over which the whole machines rest during action. Each crawler unit has a separate framework and they are rigidly connected to form a single unit.

A pneumatically operated machine uses compressed air to remove cuttings for the drilled holes, for operation of drill rods and for forcing/withdrawing the drill rods from the hole. A hydraulically operated machine, on the other hand, uses compressed air for removing cuttings only. The drill rod rotation and the up and down movement of the drill rod are done hydraulically.

An electrically operated machine in combination with hydraulic and pneumatic systems also uses compressed air for removing cuttings only. The drill rod operation is done electrically, while the raising and lowering of the drill rod is performed hydraulically.

JACK HAMMER DRILL

This is the best example of a percussive type drilling machine and is a familiar equipment to mine workers and civil construction labourers. This is a hand held and unmounted drill used to bore vertically down ward holes. The weight of the machine varies for 15 to 25 kg and is capable of making drill holes up to a depth of 3 m with hole diameters generally between 25 and 37 mm. It is a compressed air operated drilling machine to which air is supplied from external compressor(s) through hose pipes at a pressure of about 6kg/cm^2. The drill rod is hexagonal in cross-section. Suitably shaped at one end to form



1.flushing hole 2.water or air tube 3.Air tube 4.piston 5.air channel for extra blowing 6.cylinder chamber 7.rocker valve 8.air inlet valve 9.air outlet valve 10.drill steel 11. front cylinder chamber

Figure 3.6: Sectional View of Jack Hammer Drill

the shank and the other end is so shaped as to form a nondetachable single chisel bit with a tungsten carbide insert. It may also be used for drilling inclined holes.

This machine consists of the following mechanisms:

i) Percussive mechanisms ii) Rotating mechanisms

iii) Flushing mechanisms iv) Feed mechanisms

The percussive mechanism is aided by a piston-cylinder arrangement. The rotation mechanism consists of a rifle bar with ratchet, ratchet box, rifle nut and rotation chuck. Air is used as a flushing fluid through the central hole in the drill. The machine is fed by hand.

Operation

Figure 3.6 shows the cut-away view of a jack hammer. The compressed air enters the machine thought the air inlet valve at point 8 and flows via the channel past the disc shaped rockeer valve (7) into the rear cylinder chamber (6) Here compressed air actuates the piston (4) so that the piston moves in a forward direction. The air outlet (9) in the cylinder wall is uncovered due to the forward movement of the piston, allowing the air in the rear cylinder chamber to flow out freely. The piston continues to move forward and strikes the drill steel. Due to the uncovering of the air outlet the pressure of air causes the rocker valve to be reversed. Compressed air now flows to the front of the cylinder chamber (11) and forces the piston (4) in a backward direction during the backward movement of the piston; the air outlet is again uncovered, resulting in the expulsion of air from the front cylinder chamber. Owing to the uncovering of the air outlet, the pressure of air causes the rocker valve to be reversed. The whole working cycle is repeated.

Flushing the drill hole: This is done in two ways, (i) with water under pressure, and (ii) with compressed air. In the first method, the drill hole is flushed with water under pressure conducted throught the water tube (2) in the rock drill and the flushing hole (1) in the drill steel. In the second method, compressed air is fed to the drill via the tube of the tool and the flushing hole in the drill steel. This eliminates the need for tube (3)

Rotation of drill bit: The rifle bar is splined and slightly twisted. Four pawls are attached to its upper end, which allow the rifle bar to rotate in one direction that is in the direction of twist. The rifle nut placed on the top of the piston has matchine splines. As the piston moves up, the rifle nut splines exert twisting force on both the rifle bar and the piston. The rifle bar is held by the ratchet, so the piston turns. On the return stroke, the rifle bar turns having less rotational resistance than the piston, which drives the rod straight down.

Comparison of pneumatic, hydraulic and electrically operated drilling machines.

Pneumatic	Hydraulic	Electric
The machine is compact, rugged in nature, having high power-to-weight ratio, Pollution is less.	It is more compact, precise control is possible, low noise level, High rate of penetration	Pollution is less, power loss
Moisture in compressed air may freeze in low temperature operating zone. Mosisture may cor-rode the exposed metal Surfaces.	Possibility of pollution in case of leakage in oil line. They required greater maintenance.	Requires more space, Creates noise.

3.2 BLASTING

3.2.1 Explosives

The basic objective of drilling and blasting program is to achieve optimum fragmentation. Blasting in Overburden is mainly done either to fragment and shatter the rock or to displace the rock in the mine area by casting of Overburden.

According to explosive rules the explosives and their accessories are classified into eight classes. These are:

- Class—1 : Gunpowder
- Class—2 : Nitrate mixtures (like ANFO, Aquadyne, Energel, GN-1, Godyne, Permadyne, Powerflow, Permaflow, Powerite, Superdyne, Supergel, Toeblast.)
- Class—3 : Nitro compounds
  - Div:-1:Blasting gelatine, Special gelatine, O.C.G., Ajax-G, Viking-G, Soligex, etc.
  - Div:-2: Gun cotton, PETN, TNT, etc,
- Class—4 : Chlorate mixtures,
- Class—5 : Fluminate,
- Class—6 : Ammunition
  - Div:-1: Safety fuse, Igniter cord, Connectors, Electric lighters etc,
  - Div:-2: Cordtex, Detonating fuse, Plastic igniter cord, fuse, igniters, etc.
  - Div:-3: Detonators, Delay detonator, relays, etc.
- Class—7 : Fireworks,
- Class—8 : Liquid Oxygen Explosive (LOX)

The commonly used explosives in the opencast mines of our country are:

Ammonium Nitrate: It is very high explosive and having a good oxidizing and cooling agent and very safe to handle. AN is mixed with a sensitizers (fuel oil or NG or Trinitrotoluene) to form an explosive. It is hygroscopic in nature. It is having low temperature of detonation and less power as compared to NG. It is cheap, safe to handle and give better fragmentation. Prilled AN of fertilizer grade mixed with diesel oil is used for larger diameter hole in opencast mines.

Ammonium Nitrate and Fuel Oil: It is a mixture of prilled AN and fuel oil, at the nearly oxygen balanced ratio of 94/6 AN/FO. Both sensitivity and performance depend upon prill properties. It does not detonate ideally and its performance properties depend upon charge diameter and confinement. For dry hole condition it is excellent, and also it should be initiated as soon as it is loaded. It is initiated by small quantity of O.C.G. or booster.

Slurry Explosive: This type of explosive incorporate besides oxidizer (AN, sodium Nitrate etc.) water, sensitizer hydrophilic colloid which results in viscosity build up of the matrix. Water resistance is due to the cross-linking agent forms a network of bonds involving the polyvalent metal ion and hydrated gum molecules. Proper density control is crucial for maximizing the shock sensitivity of theses explosives. These are the safest of all explosives as they are not ignited easily and insensitive to the type of shock, bullet, impact and friction. It has good fume properties, water resistant. Slurry automation is accomplished by pump truck method. It led to the successful field implementation of SMS (site mixed slurry) concept. It gives a high loading rate (150-300 kg/min) and minimizes the blasting efficiency.

Site Mixed Slurry: These explosives are used for blasting on a large scale in an opencast mine. It involves specially designed pump trucks for transport to the blasting site ingredients required for SMS system. It basically comprises a mother support plant where intermediate non-explosive slurry is, initially, prepared for its application. This intermediate slurry subsequently, is transferred to a 10 tonne capacity stainless steel tank.

Emulsion Explosive: It is a mixture of oxidizer and fuel which are both in liquid form. With the help of emulsifying agent an intimate mix of oil and water is possible. Delivery rate of 200-300 kg/min can be achieved. Load cells/digital meters indicate amount explosive being loaded into boreholes. Straight emulsion explosive has high bulk strength. Emulsion matrix can be carried in support tankers of 10-12 tones capacity or more which is transferred at site thus saving time. It ensures uninterrupted charging. It is recommended that 500 gm Pentolite boosters are used for boosting. Manpower savings are obtained with less deployment of van drivers or helper, blasting crew and magazine staff. Full borehole coupling expanded burden/spacing parameters on blasting efficiency. It does not give explosive pilferage.

Heavy ANFO: It is latest development of 1980's had been use of emulsion slurry mixed with different proportion of ANFO. The ratio of emulsion to ANFO is 20:80 to 50:50 depending on the severity of water conditions and need of stronger blast energy. It is of low cost with higher density, higher energy and better water resistance than ANFO and AN. Its concentrate system allows expansion of drilling pattern, thereby reducing drilling cost.

The comparative VOD of ANFO, Slurry and emulsion are 2000 to 4800 m/s, 3300 m/s and 5000 to 6000 m/s.

3.2.2 Accessories

Detonator: High explosives are initiated by detonator or detonating fuses. It is a small cupper or aluminum tube containing a small auxiliary charge of special explosive. Due to chemical reaction initiated by flame or electric current in the special explosives, an explosion of sufficient intensity result throughout the high explosive enclosing the detonator. It is of plain ordinary electric detonators. It is having a ⅓rd with A.S.A. composition and P.E.T.N. No.6 detonator is suitable for normal requirement of mining work. No.8 is more power full than No.6. The current of 0.5amp is required for ignition of fuse-head so single detonator can blasted with minimum voltage of 3.5 volt. Delay detonator is used for more efficient blasting due to supply of immediate free face for multi-row blasting.

Booster: For effective detonation of slurry explosives and ANFO mixture such as GN-1, use of high detonation velocity booster is necessary. It is water resistant and VOD of 7000 m/s, wet strength 82 and can be detonated by detonating fuse or detonator. Cast booster is not substitute for explosive charge. It is a very power full detonator of large size and is preferred for deep large diameter, blast hole in opencast mines.

Safety fuse: It looks like a cord consists of core of fine grained gunpowder warped with layer of a tape or textile yarn and water proof coating. The burning speed is 100-120sec/m. it carries a flame of uniform rate of ignition to detonate an ordinary detonator.

Detonating fuse: For shallow depth (<3m), and for small number of holes, a detonator is inserted in the cartridge itself and detonated and detonated by ignition of safety fuse or incase of electric detonator, by an exploder. It contains core of PETN enclosed in a tap wrapped with cloth. It looks like a plastic cord. Its diameter is 5 mm external and weight about 20 g/m length.

It has a VOD of 6500 m/s. A large number of shots connected with detonating fuse can be blasted by a single detonator. Nonel is non-electric detonator.

Detonating Relays: In opencast working; it use detonating fuse for initiation provide a non-electric delay firing system. It avoids electrical connection which are required when using delay detonators. A detonating relay is an assembly of two open ended delay detonator coupled together with flexible neoprene tubing in an Aluminum sleeve. The delay interval for each detonating relay varies from 15-45 milliseconds. In use, the main or branch line of detonating fuse cut at required point of delay and detonating relay is crimped between two cut ends of the line. Cord relays manufactured by IDL chemicals.

Circuit Tester: The blasting circuit is tested by circuit tester because to avoid accidental explosion of detonator. Blastomer is manufactured by IDL chemical. It is an electronic solid circuit tester.

Crimper: A crimper is a paid of pliers to crimp or press the end of the detonator tube on safety fuse so that fuse cannot come out from the detonator.

Shot firing cables: During electric shot firing the leads for the detonator are connected to long shot firing cables to fire the shots from safe distance.

Exploder: The portable apparatus which provide the current necessary for firing electric detonator is called exploder. There are three types of exploder used in Indian mines i.e. Magneto (or Dynamo) Exploder, Battery condenser Exploder and Condenser dynamo Exploder.

3.2.3 Blasting Pattern Followed in Opencast Mines

In opencast mines both vertical and inclined holes parallel with bench face is practiced. Row of the holes may be in single or multiple. So there are mainly two types of blasting pattern followed in opencast mines. These are:

a) Single Row blasting pattern
b) Multi-row blasting pattern.

3.2.3.1 Single row firing pattern: In single row blasting the fragmentation is low and specific explosive consumption is more than multi-row blasting, so multi-row blasting pattern is preferred.

In this the following patterns are used:

a) The alternate delay pattern (used for softer rocks),

b) Consecutive shot delay pattern (rock with medium hardness),

c) Short delay firing with a cut (used for hard rocks).

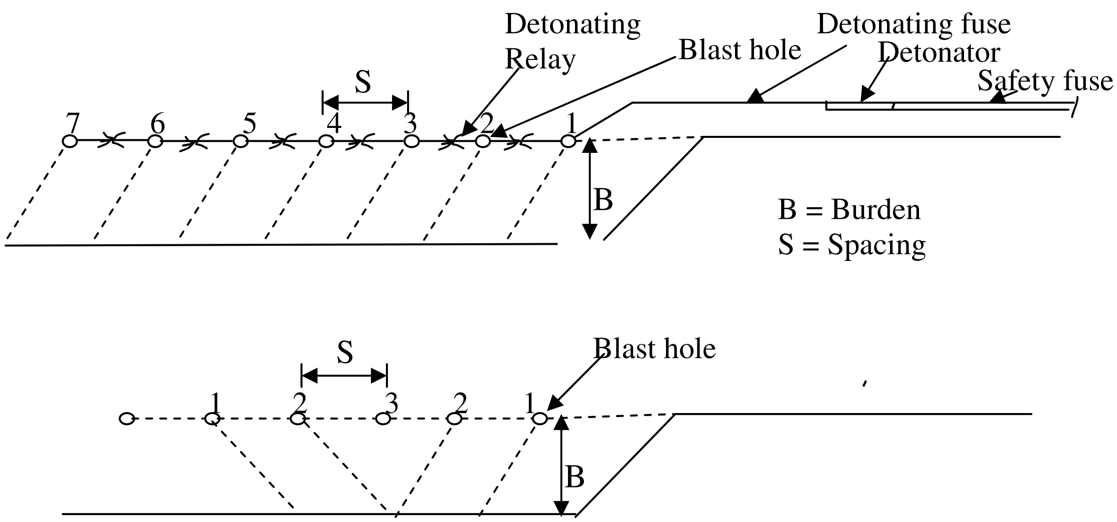


Figure 3.7 Sequence of initiation in single row blasting

3.2.3.2 Multi Row firing pattern: The Multi Row Firing pattern is of mainly five types:

a). Square grid in-line initiation (spacing(S) = effective burden (B)).

b). Square grid 'V' pattern (S = B; S_E =2.B_E).

c). Square grid 'V_1' pattern (S = B; S_E = 5.B_E).

d). Staggered grid 'V' pattern (S = B; S_E = 1.25B_E).

e). Staggered grid 'V_1' pattern (S = B; S_E = 3.25B_E).

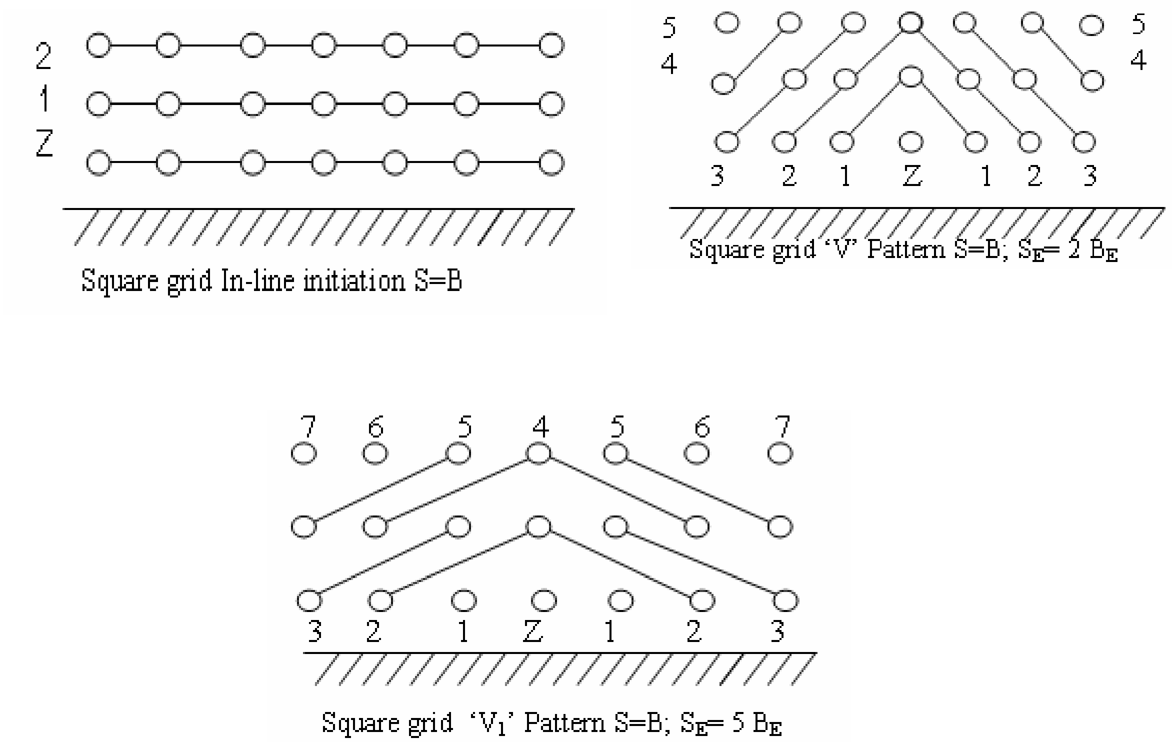


Figure 3.8(a): Multi row firing patterns

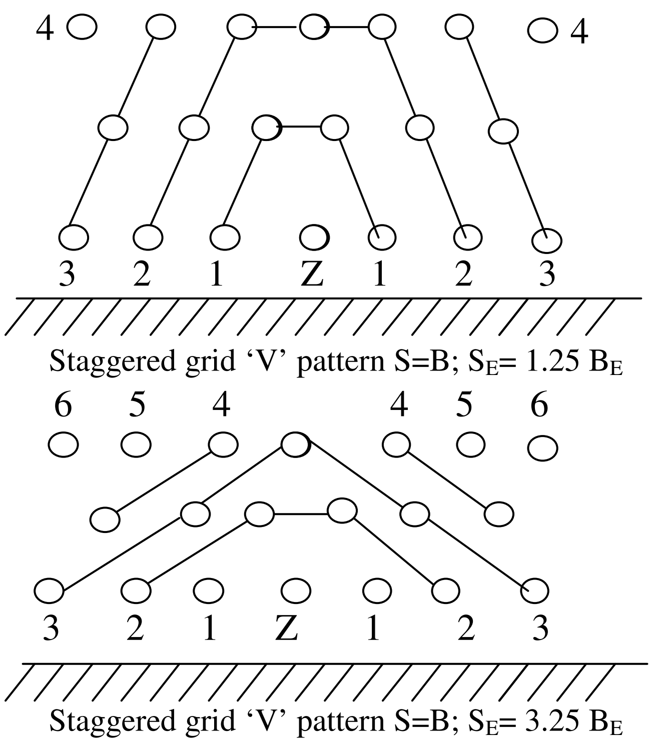


Figure 3.8(b): Multi Row firing patterns

Beside cut pattern other pattern of blasting in multi row of firing are as given below:

- **Transverse cut pattern**: They are used where smaller width of muck pile is desired.
- **Wedge or trapezoidal blasting pattern**: They are used when the rocks are medium hard and hard one. Due to the motion in opposite direction in this case the big boulders are broken by supplementary collision.

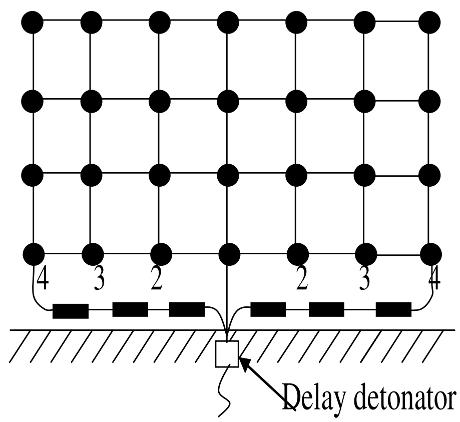


Figure 3.9: Transverse cut pattern

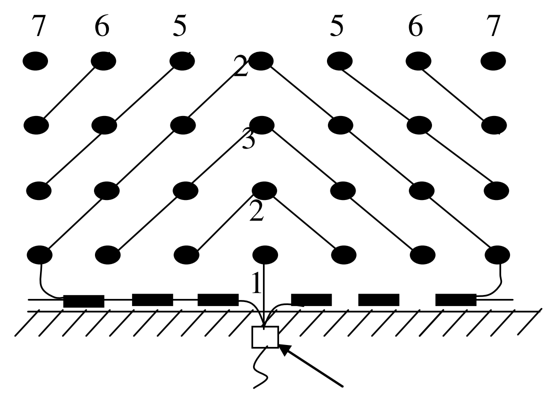


Figure 3.10: Wedge blasting Pattern

- **Diagonal blasting pattern**: With this it is possible to blast the rock towards the least resistance and improve the fragmentation of rock.

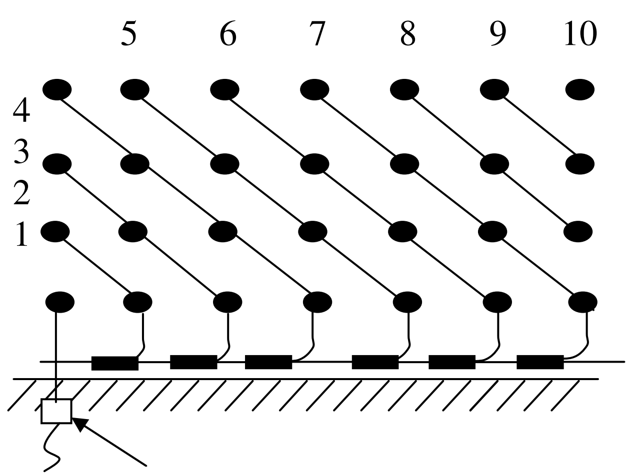


Figure 3.11: Digital blasting pattern

3.3 RECENT ADVANCEMENT IN DRILLING AND BLASTING TECHNIQUES

Segregation Blasting:

In production blasts, in must of the coal mines in India, a thin layer of coal parting (0.5 to 1.2m) sandwiched between overburden rocks is generally lost during overburden bench blasting. This may also lead to spontaneous heating besides creating endanger to unauthorized manual pickers. This problem may be sorted by segregation blasting technique, which encounters the difference in densities of coal and overburden and their impact sensitivity of different energy level so that when the composite medium is blasted, coal and overburden are separated and thrown away at different distances.

Baby Decking:

This method is an effective mean to control the coal damage due to sub-grade drilling in normal production blasting. In this technique, the total explosive column is divided into two decks, a small baby deck and a main deck. The primers and timing of down-the-hole delays are designed in such a way that the baby deck is initiated 25ms after the main deck. The amount of explosive charge in baby-deck will depend on the hole depth, hole diameter, explosive type and rock properties.

Air Decking:

This is an increasingly popular technique in which an explosive column is combined with an air chamber in blast holes. This technique helps in controlling the breakage process through effective distribution of explosive energy and there by enhancing the breaking power on the rock.

3.3.1 Cast Boosters

Economy and Reliability: Boosters optimize borehole initiation of all common booster sensitive explosives.

Application Flexibility: Composition boosters are available in standard, cord-sensitive, slider and stinger configurations to make them easy to use with all types of initiation systems.

High Visibility: Fluorescent color packaging reduces the possibility of losing or misplacing charges.

Unlimited Shelf Life: Shelf life is unlimited and freeze/thaw cycles are no problem.

No Headaches: Cast pentolite boosters do not contain nitroglycerin.

3.3.2 Nonel

NONEL Initiation System

The invention of NONEL by Dyno Nobel's Swedish organization in the 1970's revolutionized the blasting industry. Instead of electric wires, a hollow plastic shock tube delivers the firing impulse to the detonator, making it immune to most of the hazards associated with stray electrical current. NONEL shock tube is a small diameter, three-layer plastic tube coated on the innermost wall with a reactive explosive compound, which, when initiated, propagates a low energy signal, similar to a dust explosion, at approximately 6,500 ft/sec (2,000 m/sec) along the length of the tubing with minimal disturbance outside of the tube. The design of NONEL detonators incorporates patented technology, including the Cushion Disk (CD) and Delay Ignition Buffer (DIB) to provide reliability and accuracy in all blasting applications.

NONEL MS: It is non-electric delays detonators consist of a precise, millisecond delay detonator crimped to a length of shock tube, and are used in open pit mining, quarrying, construction and underground mining. NONEL MS units are available in firing times ranging from 0 to 1000 milliseconds.

NONEL LP: It is a non-electric delay detonators are precise in-hole delay detonators used extensively in underground mining, tunneling, shaft sinking and special construction applications. NONEL LP units consist of a high-strength detonator crimped to a length of shock tube, available in 19 delay periods from 0 to 8000 milliseconds.

NONEL SL: These are non electric delay detonators comprising of 30 inch lengths of shock tube with a precise in-hole delay detonator on one end and a loop of shock tube on the other, to be used in conjunction with low-energy detonating cord down lines. When used in a slider configuration, they provide independent deck initiation from a single down line. NONEL SL units are available in firing times ranging from 0 to 1000 milliseconds.

NONEL TD: These delay detonators provide precise, reliable surface delay times, lengths and hardware suited for initiating detonating cord or shock tube down lines in various surface applications such as open pit mining, quarrying and construction. They consist of a precise surface delay detonator housed in a plastic bunch block on a length of shock tube, and are available in delay times ranging from 9 to 109 milliseconds.

NONEL EZ DET: This non-electric blast initiation system was developed for a variety of blasting applications, including construction, surface and underground blasting. They eliminate the need to inventory various in-hole delays and provide fast, simple hook-ups while allowing an unlimited number of holes to be shot with independent hole initiation. An EZ DET unit consists of a surface delay detonator housed in a plastic connector and a precise in-hole delay detonator, linked by a length of shock tube. They are available in a variety of lengths and delay times.

NONEL EZTL: These non-electric trunk-line delay detonators are precise, reliable millisecond delays, with delay times and hardware suited for use as trunk-lines in open pit mining, quarrying, construction and underground mining. EZTL units consist of a precise in-hole delay detonator housed in a connector block and crimped to a length of shock tube. They are available in various lengths and delay times.

NONEL EZ DRIFTER: These non-electric blast initiation systems are used extensively in underground mining. An EZ DRIFTER unit consists of a surface delay detonator housed in a plastic connector and a precise high strength in-hole delay detonator, linked by a length of shock tube. They are available in various lengths, with a delay time of 200/5400 milliseconds.

NONEL STARTER: It is a non-electric delay detonators are used as the primary initiator for mining, quarrying and construction blasts. They consist of a spool of non-electric shock tube factory-assembled to a detonator that is housed in a plastic bunch block. NONEL STARTER is available in various spooled lengths for easy application and deployment.

NONEL LEAD LINE: This is NONEL shock tube spooled at the factory in 2,500 ft (763 m) lengths for easy application and deployment. NONEL LEAD LINE provides maximum flexibility to the blaster in choosing a position of safety from which to initiate non-electric blast rounds in either underground or surface applications.

NONEL TWINPLEX: These delay detonators are designed to provide two independent paths of initiation between boreholes and/or rows of boreholes (twin-path applications) in open pit mining, quarrying, construction and underground applications. TWINPLEX units consist of two precise millisecond delay detonators of the same delay, with individual yellow shock tube leads, joined by an over-extruded orange plastic sheath. Each detonator is housed in a separate connector block.

NONEL MS CONNECTOR: These delay detonators consist of an 18-inch (46 cm) length of shock tube with detonators of the same delay on each end. The detonators are housed in connector blocks designed to accept detonating cord quickly, easily and securely. MS CONNECTORS are used to provide millisecond delay timing between holes of detonating cord initiated blasts. They are available in delay times ranging from 9 to 109 milliseconds.

NONEL and detonating cord combination

Nonel in-hole detonators are, in some applications, combined with surface cord trunk-lines. However overpressure from the cord can be a constraint in this system. For this combination Dyno Nobel Special 18 or Special 25 detonating cord with 3.6 and 5.0g/m PETN core load respectively are recommended.

All NONEL MS and LP detonators are fitted with Cord Clips for easy connection to detonating cord. The position of the Cord Clip on the NONEL tube is easily adjusted by sliding it along the tube. It is recommended that the connection point is adjusted to be as close as possible to the collar of the hole (without putting excessive strain on the tube). This ensures the shortest possible active tube on the surface, thereby avoiding potential tube cut-off by the detonating cord.

Surface NONEL Blasting Patterns

The most commonly used NONEL products in surface blasting operations are a combination of the NONEL MS Series and the NONEL Snap line Series. This combination offers an unlimited number of delay times that can be used to design different types of initiation patterns. It also offers a reduction in noise levels compared to the use of detonating cord for surface hook up.

The principle for all patterns is that one delay time from the NONEL MS Series is chosen as the in-hole delay. One or several delay times from the NONEL Snap line Series are chosen to provide the surface delay pattern. To get the best blast performance and use the explosive in a blast hole in the most efficient way, bottom initiation is practiced in most blasting operations.

For a variety of reasons a blast hole may require a second detonator. This back-up detonator is generally placed in the upper part of the explosives column. In order to retain control over bottom initiation of the holes a suitable combination of bottom and top detonator delay times is chosen.

If the top and bottom detonators in a hole are located at a distance greater than 25 meters apart it is recommended to increase the difference in delay time between the detonators, due to the additional time taken for the signal to travel the length of the NONEL tube (1 millisecond for every 2 meters of tube).

3.3.3 Digital blasting using electronic detonators(E-Blast)

In the past, all in-the-hole delays have employed a slow-burning pyrotechnic charge to create delay time. The electric detonator with pyrotechnic delay is a superficially simple product. An electric detonator with pyrotechnic delay contains several elements in the chain that leads from ignition of the detonator to detonation of the charge in the drill hole. Each element involves a delay time, which is not exactly the same for all nominally equal detonators i.e. each element has a certain amount of scatter in time. The scatter of the detonators total delay time is influenced by all these elements. To keep within reproducible delay time, the raw material used in the manufacture of these components must be of even quality. The finished product must undergo little change in its properties during transport and storage so as to undergo the least change in its delay time. Many chemical ingredients in these detonators must be handled with care because they are suspected of being a health hazard. The practical difficulties of controlling the anomalies in the pyrotechnic delay detonator to achieve a radically increased timing precision appear to be insurmountable.

The electronic detonator offers such a radically increased timing precision. Using integrated circuits, timing precision will be measured in microseconds rather than in milliseconds. High accuracy timing can help to develop new blasting methods where better fragmentation control can be achieved. The blast can be so formulated that there will be minimal ground vibration and an increase in frequency of wave. The ongoing research being conducted into the use of electronic detonators reveal that they have great potential in the following areas.

- Improving contour blasting and decreasing the need for rock support
- Controlling ground vibrations
- Controlling rock fragmentation and heave

Thus the increasing disadvantages and the environmentally disturbing results with ordinary electric and shock tube detonators and the great potential in the use of electronic detonators highlights the need for the digital blasting system.

Components of digital blasting system

The components of a digital blasting system are the same as that of an ordinary blasting system. Only the type of blasting machines and the detonators used change. The components of a digital blasting system are listed below

- Electronic detonator
- Connecting wires
- Computerized blasting machine

The preparatory work for a blasting operation includes determining the delay time for each blast hole in the round and charging the holes with detonators with detonators with suitably chosen period numbers. The blasting machine's time memory is then programmed with the necessary time information adapted to the period number chosen.

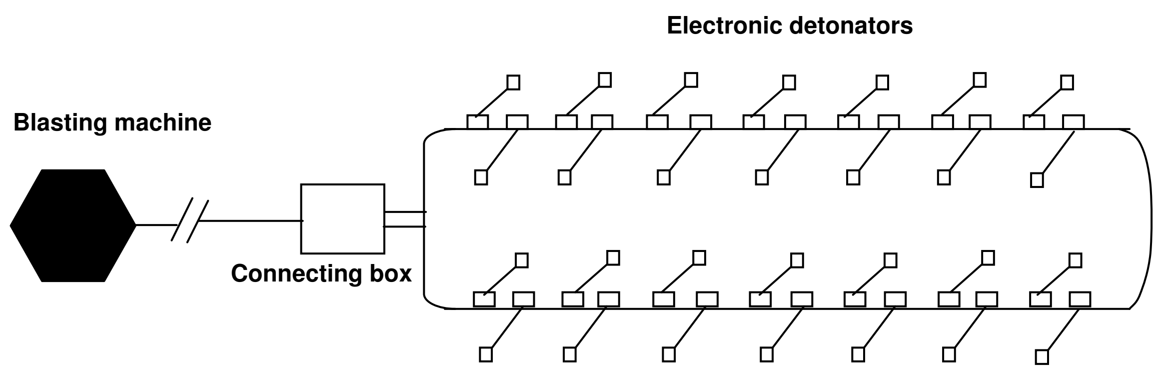


Figure 3.12: Model circuit of digital blasting system

Electronic detonator

From the exterior, the electronic detonator looks exactly like a conventional electric detonator. In principle, the detonator consists of an electronic delay unit in combination with an instantaneous detonator. Electronic detonators utilize stored electrical energy inside the detonator as a means of providing the timing delay and initiation energy. All other detonator technologies, including shock tube, electric or safety fuse initiated, utilize pyrotechnic energy as a means of delay and initiation.

An integrated circuit on a microchip constitutes the heart of the detonator. In addition the detonator has a capacitor for energy storage, and separate safety circuits on the input side in order to protect against various forms of electric overload. Thus fuse head for initiating the primary charge is specially developed to provide a short initiation time with minimal time scatter.

Programmable electronic detonators contain a tiny circuit board that enables the detonator to store a blast sequence number in it's on- board memory. It also can perform a self-check of its functionality when connected to a hand-held programming unit or digital blasting machine. Depending on the brand used, delay periods of 1 to several thousands milliseconds, in 1-ms increments, can be programmed into the detonators. This level of precision eclipses the minimum 25-ms delay capabilities found in most pyrotechnic based detonators.

The most striking characteristic of an electronic detonator is its flexibility. Period numbers, which are marked on the detonators, do not state the delay time but the order in which the detonators will go off. Each detonator has its own time reference but the final delay time is determined through interaction between the detonator and the basting machine immediately before initiation.

Typical characteristics of an electronic detonator are.

- ♦ The detonator initially has no initiation energy of its own
- ♦ The detonator cannot be made to detonate without a unique activation code
- ♦ The detonator receives its initiation energy and activation code from the blasting machine
- ♦ The detonator is equipped with over voltage protection. Low excess loads are dissipated via internal safety circuits and higher voltages limited by means of a spark plug
- ♦ The initiation systems operate with low voltages, which is a great advantage considering the risk of current leakage.

Blasting machine

The blasting machine constitutes the central unit of the digital blasting system. It supplies the detonators with energy and also determines the delay time to be adopted. The unit is microcomputer controlled and its mode of operation can thus be altered with various control programs.

The controls for initiation appear as simple as possible in spite of the advanced internal design of the machine. A panel with lamps indicates what is happening and gives the go-ahead signal when the shot is to be fired.

Delay time allocation to the detonators is carried out by uniquely coded signals to eliminate any possibility of error. The detonator responds to the code only from the blasting machine and thus eliminates any risk of initiation from external energy sources. The blasting machine also performs an operation status control, which is done automatically by the machine.

System characteristics

The typical characteristics of digital blasting system are

- The shortest delay time between two adjacent delay period can be as low as 1 ms.
- A detonator with a lower period number cannot be given a longer delay time than a detonator with a higher period number.
- Detonators with different period numbers cannot be closer to each other than the difference in their period numbers.
- About 500 detonators can be connected to each blasting machine.

From the system characteristics we can deduce that many different number combinations can be used. Thus for a successful blast, a user requires only a sufficient number of detonators with different period numbers in stock.

Mode of operation

The preparatory operations for a blast includes determining the delay time for each blast hole in the round and charging the holes with detonators with suitably chosen period numbers. An outline of a round using this system is given in the figure. The detonators in the round are connected in parallel with arbitrary polarity. This is done by connecting the detonators to a two-wire bus cable via a terminal block, using special pliers. Finally the bus cable is connected to the blasting machine via terminal box and a firing cable. After getting the go-ahead sign, energy is supplied to the detonators from the blasting machine, which in turn initiates the explosives.

Advantages of digital blasting system

The digital blasting system has a number of advantages over the one, which uses pyrotechnic delay. Vibrations resulting from blasting are one of the major problems of pyrotechnic delay. Of the three components of ground vibration—peak particle velocity (PPV), duration and frequency—frequency is the most important when it comes to human reaction from blast effects. The geology of the area surrounding the blasting site influences the transmission of vibrations. Once the influence of the geology is understood, blast can be configured to disrupt rather than reinforce the natural frequency of the site t5hrough the use of millisecond delays. High precision timing obtained using the electric detonator and the reduction in the time of delay intervals has the advantages of decreasing the charge per delay thus leading to reduce levels of vibration. Apart from reduction in the vibration, there is also an increase in the frequency thus avoiding structure resonance.

Use of the digital blasting system also to helps achieve a higher fragmentation of the rock being blasted and better throw of the blasted material. Increased fragmentation and good throw can result in better loading conditions. Boulder formation can be greatly avoided with a well designed blast thus eliminating the need for secondary blasting. Drilling costs are reducing since there is not a need for deck holes and in turn the explosives cost. Since a large number of holes can be blasted without any worry about vibrations, and overall frequency of the blasting operations can be greatly increased, thus bringing down the time taken for blasting per tonne of rock. This will lead to minimum suspension in operations such as loading and drilling. Hall these factors will influence a higher productivity. Increase in output of the mine means an increase in profit for the mine owner.

From the advantages summarized above, the potential applications of digital blasting can be listed as follows:

- Vibration control situations, close to sensitive structures and environments
- Complex blasts, difficult to design with standard pyrotechnic detonators and delay imings
- Minimizing of ore dilution, maximum separation of ore/waste
- Improving cast or throw
- Improving fragmentation and diggability
- Simplification of detonator inventory control and re-ordering

3.3.4 Digital Drilling

Computerized drills and MWD systems are two technologies that can provide feedback on important aspects of the drilling function:

- Correct location and depth of blastholes; and
- Accurate recording of as-drilled conditions.

According to Atlas Copco, computerized drilling systems provide the capability to automate the drilling process, which can increase machine utilization, and to improve drilling accuracy, which helps optimize blasting and improve safety. The company developed computerized control systems for its jumbo drills in the early 1990s and more recently introduced the technology to surface crawler drills.

The Atlas Copco ROC D7 C top-hammer drill uses a CAN-bus system-Controller Area Network — comprising a number of small on-board computers linked by a single cable. All hydraulics has been removed from the cab and a color monitor replaces traditional dials and gauges.

A Rig Control System (RCS) senses variations in rock conditions and adjusts drilling functions to optimize drill penetration and accuracy and to minimize consumption of drill tools, the company says. RCS works in conjunction with ROC Manager software for monitoring drilling operations. Drill plans can be designed in the office and uploaded to the drill rig. While drilling, the system logs deviations and other parameters for later analysis, which can reveal geologic or hole conditions that can impact explosives loading or other blasting plans. MWD parameters include penetration rate, percussion pressure, feeder pressure, damper pressure, rotation speed, rotation pressure, and flush air pressure.

The ROC D7 C automatically adds rods to the drill string to a preset hole depth, allowing drilling to continue during operator breaks and between shifts, Atlas Copco says. Rig utilization through using the automated functions can increase 10 to 15 percent, according to the manufacturer.

The modular design of the RCS allows future expansion or upgrading of the system by adding control modules. Options under development include the use of a Global Positioning System (GPS) for accurately positioning the rig to drill holes according to the blast design without surveying or staking.

GPS and MWD systems developed by several companies currently are available as retrofit components for drill rigs. Thunderbird Mining Systems' Drill NAV Plus is a blasthole drill monitoring and navigation system that allows operators to get within approximately 8 in. of the designed hole location without staking, according to the company. A color VGA display shows the operator where every hole should be drilled as well as the location of previously drilled holes. As the drill moves around the drill pattern, the map moves on the operator's display to allow quick navigation between holes and a visual lock ring function indicates when the drill is positioned within tolerance of the design location of a hole. Map data is transferred to and from the drill via spread spectrum-radio.

The system also indicates the elevation above sea level of the drill bit as the hole is drilled. Consequently, the target bottom of each hole can be specified as an elevation, rather than a depth, to reduce under- and over-drilling, eliminate hard-digging toes, and create level benches. Drill NAV Plus also provides drill monitoring by measuring, displaying and recording feet or meters drilled; penetration rate; drilling, non-drilling, and tramming time; and hole, pattern, and drill bit numbers.

Aquila Mining Systems, a Caterpillar company, offers drill monitoring systems for mining applications. Its DM products, based on the AMP computer hardware and QNX operating system software, are available factory installed on new drills or retrofitted to older drills. Six stand-alone products allow operators to upgrade drill-monitoring capabilities as needed The DM-1 Production Monitoring System provides feedback to the operator on drilling productivity and performance. DM-2 Material Recognition System uses a vibration sensor and pattern recognition software to analyze drill variables and determine hole geology while drilling.

The DM-5 Guidance System for vertical drilling uses RTK GPS receivers to allow the operator to spot the hole within centimeters of the target location without surveying or staking, according to the company. Once the drill is leveled and the hole started, the DM-5 automatically determines collar elevation and calculates the required drilling depth. Blasthole position information is stored and transmitted to the mine office for use in blast design. The DM-6 Guidance System is for inclined drilling.

CHAPTER 4

REVIEW OF OPTIMIZATION TECHNIQUES

General
Optimization of Mine Production System through Operation Research Techniques

REVIEW OF OPTIMIZATION TECHNIQUES

4.1 GENERAL

The optimization of mining process can be achieved in different ways and methods. These are;

1 Optimization of drilling productivity
2 Optimization in casting of overburden
3 Optimization of explosives and blast design
4 Optimization in transportation etc.
5 Safety aspects

4.1.1 Optimization of Drilling Productivity

Improvement in the area of productivity / meter of drilling was achieved though gradual enlargement of drilling parameters from those existing parameter for the different benches of the mines. By studying the rock properties scientifically and type of explosives for different benches for achieving the optimum drilling parameter. So by the help of computer aided method or module, 'SABREX' was helpful.

Aim of the computer aided Drilling

The CAD design programs consider all aspects of drilling process. They are:

- Design drilling patterns to maximize drilling efficiency while lowering overall, operation cost.
- Provide the capability for remote and unattended operate.
- Simplify the control, which decreases training time required for drilling function.
- Reduce maintenance requirement
- Reduce the drill bit life.
- Control the cycle time of the drilling operation
- Calculate and optimize drilling costs for any given blasting pattern.
- Selection of proper hole size, inclination and depth.

Inputs required for computer model

- **Geology:** Rock density, tensile and compressive strength of rock, young's modules of rocks, and Poisson's ratio. Strike and dip value, joint structure and frequency.
- **Equipment:** Feed thrust, impact frequency, piston strike, Impact pressure, rotation rate. Type of drill rig, type of bit.
- **Site factors:** Dimensions of the face, Diameter of hole ratio of spacing and burden, length of hole, inclination of hole, number of rows, wet or dry holes, drilling sequence.
- **Cost factor:** Cost of drilling equipment and depreciation cost, number of operators, wages and efficiency factor, the unit cost of drill rods, blast hole bit and consumables, the cost of power and lubrication oil.

Computer programs for optimum drilling

a) Dialog's program,
b) Fan drilling and explosives loading design program
c) Roof support hole drilling program for coal mines.
d) 'Care' program
e) Program for jumbo drill machine.

Therefore, it can be concluded that computer aided method of drilling can be very much useful in achieving the higher production targets by maximizing the drilling efficiency, rate of drilling, higher availability of equipment through saving in drilling cycle time, reduction in cost of drilling, and capabilities for re mute working in hazardous face.

4.1.2 Optimization in Casting of Overburden

Factors considered for application of blast casting

- Pit geometry which refers to bench height, pit width and the ratio of these two parameters.
- O/B characteristic like depth and hardness.
- Location of O/B dumps as to availability of in pit dumping conditions.
- Capacity of striping
- Environmental restriction etc.

Aim of blast casting

- Increasing the production lends.
- Reducing the capital out lay for excavating equipment.
- Improving productivity of existing operation.
- Replacement of equipment.

4.1.3 Optimization of Explosive and Blast Design

We know mainly drilling and blasting cost is more significant part of the overall operating cost, i.e. explosive cost may very from 4-12 % or the total operating cost. So this cost can be controlled by;

- Optimum use of booster cartridges and cast boosters.
- Optimum use of detonating fuse.
- Saving of explosives by using air decks.
- By eliminating the desensitization of explosive column on the hole.

Influence of rock parameters on blasting

- Rock strength,
- Density,
- Blast ability index,
- Porosity,
- Effect of geological disturbances, etc,

Problems associated with blasting

(i) Fragmentation : The influencing factor is ;

Design parameter

1) Drilling pattern
2) Hole diameter
3) Sub-grade drilling
4) Steaming column
5) Initiation system
6) Delay timing

Explosive parameters

1) Density
2) VOD,

3) Shock and gas energy released.

Rock parameter

1) Strength,

2) Stiffness,

3) Compressive, shear wave velocity.

(ii) Blast induced vibration

(iii) Noise /Air Over-Pressure

(iv) Fly Rocks: It can be controlled by giving proper attention to blast design layout, drilling and loading of explosive.

4.2 OPTIMIZATION OF MINE PRODUCTION SYSTEM THROUGH OPERATION RESEARCH TECHNIQUES

Optimization means achieving the best that is to achieve maximum or minimum value of the operating parameters. Operations Research uses suitable techniques or tools available to achieve the goal. The techniques like linear program, waiting line theory, game theory, inventory control models and non-liner programming, integer programming, dynamic programming sequencing theory, Markov process, network scheduling PERT and CPM, Symbolic logic, information theory and utility/ value theory are popular (Nanda, 2003).

The different optimization techniques being utilized for the purpose are**:**

1. Queuing Model: used for optimization dumper shovel combination.
2. Markov model: used for production potential prediction.
3. Reliability model: used for assessing reliability of mine production system.
4. Cargo-loading model: can be used for selection of explosives.
5. work force size model:

4.2.1 Queuing Model

It was introduced by A.K. Erlang. There are a specified number of dumpers allotted to a particular shovel for operation. If the number of dumper is not matching perfectly then either dumper or shoved will be idle, so optimum number of dumpers to be allotted for economically viable operation.

Time study is to be conducted to obtain the following parameters

a. Traveling time (haul, + dump + return),

b. Loading time,

c. Spotting time,

d. Service time,(loading time+ spotting time)

The expression for probability of a busy system is $\boldsymbol{\eta = 1- P_0}$ where:

$$P_0 = 1 / \sum_{n=0}^{n=m} \frac{M!}{(M - N)!} \cdot (\lambda/\mu)^n$$

Where:

λ = mean arrival rate, μ = mean service rate
M = number of dumpers already in the queue,
$\rho = \lambda/\mu$ = service factor = 0.259
η = utilization of shovel.

4.2.2 Markov Model

The behavior of equipment or a system can be appropriately analyzed like a renewal process, if the component or the system is maintained properly. This process consists of states of operation and failure. The system is such that the equipment or the subsystem can remain in following three states.

1 Operating state,

2 Failed state, where repair action is not yet initiated.

3 Repair state.

Application: By using Markov Model the transition probabilities of different states can be obtained.

Series System: The machines are kept in series and at least, drill machines are in operation. It means that if one machine goes under breakdown there want be any more failure. In this situation the transition matrix for three machines with different values of λ & μ, can be solved.

$$P_0 = 1 / \left(1 + \sum_{i=1}^{n} (\lambda_i / \mu_i) \right)$$

and $\mathbf{P_0 + P_1 + P_2 \ldots\ldots + P_n = 1}$

Where:

$\mathbf{P_1 = P_0 ((\lambda_i / \mu_i)}$

P_0 is transition probability of initial state and η is number of components in the system.

Parallel System: Instead of keeping all the three M/C in series for operation (two are operation and one as stand by). In such case assumptions are,

1) Two components in parallel are required to keep the system operating.
2) Upon failure of one on-line component the stand by unit is brought to operation mode.
3) Repair work starts when ever a component transits to failure mode and each component has its own failure rate and repair rate.
4) If the failed component is repaired and is ready for operation before the stand by unit completes its schedule, work, the repaired M/C becomes a stand by.
5) No failure is possible when the M/C is off the line.

The steady state reliability of each m/c is calculated by derivation of Markov process for steady state reliability i.e. $R(t) = \mu/\mu+\lambda$ where, μ = repair rate, λ= failure rate.

4.2.3 Reliability Model

Probability of success is a measure of reliability and the probability of failure is the measure of unreliability.

To provide a quantitative basis for evaluation of reliability, there is some well-known time to failure distributions in standard use. The important role in evaluating them is time between failures of system. The distributions are commonly used in failure and repair analyses are

1) Exponential distribution
2) Weibull distribution
3) Log-normal distribution
4) Gamma distribution
5) Poisson distribution.

Exponential distribution : The distribution has no memory i.e. if a unit has survived (t) hours then the probability of its surviving an additional 'h' hours is exactly the same as the probability of surviving 'h' hours of a new item.

- If items under study are replaced or renewed like "as good as new" after failure, the time periods between failures are independent and identically distributed. It is widely new for representing operating and repair times.

4.2.4 Cargo-Loading Model

It deals with the problem of loading items on a vessel with limited volume or weight capacity. Each item produces a level of revenue. The objective is to load the vessel with the most valuable cargo. The recursive equation is developed for n-item W-ton vessel. Let m_i be the number of units of item (i) in the cargo. The general problem is represented by:-

Maximize $z = r_1m_1 + r_2m_2 + \ldots\ldots + r_nm_n$

Subject to $w_1m_1 + w_1m_1 + \ldots\ldots + w_nm_n <= W$

$m_1, m_2, \ldots, m_n >=0$ and integer

4.2.5 Work Force Size Model

In it the hiring and firing is exercised to maintain the labor force required for the project. Hiring and firing both add on cost. Let us assume the project will be executed in a time span of n weeks and minimum labor force required in any week i is b_i. Let xi is the actual number of employees employed in week i, so cost in week i is.

1) $C_1(x_i\text{-}b_i)$ cost of labor.
2) $C_2(x_i\text{-}x_{i-1})$ cost of hiring additional labor

Giving the equation

$$f_i\ (x_{i-1}) = \min_{x_i>=b_i} \{C_1(x_i-b_i)+C_2(x_i-x_{i-1})+f_{i+1}(x_i)\},$$

$$I=1, 2, \ldots, n$$

CHAPTER 5

DEVELOPMENT OF A BLAST OPTIMISATION MODEL

Parameters affecting explosive performance

Selection of Parameters for Blast Optimization

Collection of Information for Implementation of the Optimization Methodology

Optimization Methodology

Flowchart of the Program

Algorithm of the Program

DEVELOPMENT OF A BLAST OPTIMISATION MODEL

Rock breaking by drilling and blasting is the first phase of the production cycle in most of the mining operations. Optimization of this operation is very important as the fragmentation obtained thereby affects the cost of the entire gamut of interrelated mining activities, such as drilling, blasting, loading, hauling, crushing and to some extent grinding. Optimization of rock breaking by drilling and blasting is sometimes understood to mean minimum cost in the implementation of these two individual operations. However, a minimum cost for breaking rock may not be in the best interest of the overall mining system. A little more money spent in the rock-breaking operation can be recovered later from the system and the aim of the coordinator of the mining work should be to achieve a minimum combined cost of drilling, blasting, loading, hauling, crushing and grinding. Only a “balance sheet” of total cost of the full gamut of mining operations vis-à-vis production achieved can establish whether the very first phase- rock breaking- was “optimum” financially; leaving aside factors of human safety.

An optimum blast is also associated with the most efficient utilization of blasting energy in the rock-breaking process, reducing blasting cost through less explosive consumption and less wastage of explosive energy in blasting, less throw of materials, and reduction of blast vibration resulting in greater degrees of safety and stability to the nearby structures.

Important and highly controllable blast parameters include: diameter and length of blast holes; type and configuration of charges; shape, condition and development of effective faces; available expansion volume of broken rock; type and dimension of the blasthole pattern; initiation sequence and delay timing.

Ground vibration from mine blasting may be expressed by three important characteristics. They are amplitude, frequency and duration of the blast. The variables which influence ground vibration parameters may be divided in to two groups (Siskind, 1973):

(i) Non-controllable, and

(ii) Controllable

Controllable variables are those, which can be manipulated or changed by trial and error depending on the characteristics of ground vibration. On the other, non-controllable variables are those, over which the blasting engineer has no control. Non-controllable variables are:

(i) General surface terrain

(ii) Type and depth of overburden

(iii) Wind

The important controllable variables associated with the characteristics of ground vibration are (Siskind, 1973; Wiss and Linehan, 1978):

- Type of explosive
- Charge per delay
- Delay interval
- Direction of blast progression
- Burden, spacing and specific charge

Most of these variables are interrelated. A change in one variable in the operating system can change the others. The net change in the magnitude, frequency and duration of ground movement is the combined influence of all variables rather than anyone of them independently. It is of course a very difficult task to quantify the measures or extent of the effect of each variable individually. Basic parameters involved in the process of optimum blasting may be classified as follows:

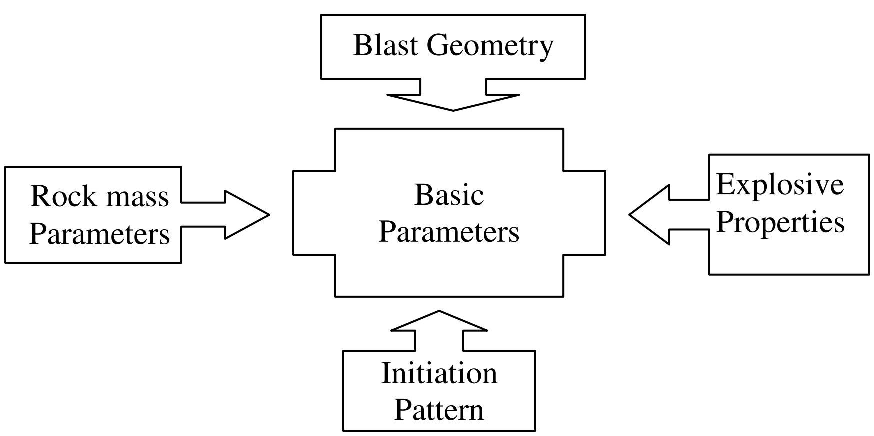


Fig 5.1: Classification of basic parameters

5.1 PARAMETERS AFFECTING EXPLOSIVE PERFORMANCE

Selection of proper explosive in any blasting round is an important aspect of optimum blast design. Basic parameters include

i. VOD of explosive (m/s)
ii. Density (g/cc)
iii. Characteristic impedance
iv. Energy output (cal/gm)
v. Explosive type (NG, ANFO, Slurry, Emulsion etc.)

5.1.1 Selection of Proper Explosive

A number of different types of explosives are presently available in the market today and hence it is rather difficult to select the right type for a specific purpose. Some of the explosives extensively used today are:

1) Special Gelatine
2) NG –based
3) Gun powder
4) Slurry explosives (both cartridge and site –mixed)
5) ANFO (ammonium nitrate prills mixed with fuel oil)
6) Emulsion (both cartridge and site-mixed)

Although all of them have relative merits and demerits in utilization, the major factor influencing the mine operator's ultimate choice of the correct explosive is blasting cost. Since this cost includes the cost of drilling plus other fixed expenditures, a more expensive explosive could well mean money saved in the total cost of a blasting operation.

The following simple example illustrates how certain basic considerations could influence the selection of explosives. (P. Pal Roy, 2005)

Let us consider,

N = Number of boreholes

W = Weight of explosives loaded in each hole (kg)

C = Cost of explosive per kg

D = Cost of drilling and loading of each hole

Then total cost becomes

$$T = N\,(D + W\,.\,C)$$

When comparing total cost for different explosives, A and B, the formula is:

$$T_A - T_B = N_A\,(D + W_A.C_A) - N_B\,(D + W_B.C_B)$$

The criterion for determining the quality of explosive for a given job is simply whether the left hand side of the equation is positive or negative. If it is positive, explosive B is better than explosive A. if it is negative, the reverse is true.

5.1.3 Parameters Related to Geo-Mechanical Properties of Rock Mass

The total geology and rock characteristics have greater influence upon proper fragmentation, control of fragmentation, control of vibration, fly rock and other safety aspects. They may be divided into following ten parameters:

i. P-wave velocity (m/s)
ii. S-wave velocity (m/s)
iii. Density of rock mass (g/cc)
iv. Compressive strength (MPa)
v. Tensile strength (MPa)
vi. Characteristic impedence of rock
vii. RQD
viii. Dip direction (deg) and dip amount (deg) of joints
ix. Rock quality factor

5.1.4 Parameters Related to Blast Geometry

Good fragmentation, displacement and less explosive consumption depend mostly on blast geometry. Basic blast geometry parameters include:

i. Spacing (m)
ii. Burden (m)
iii. Bench height (m)
iv. Depth of hole (m)
v. Stemming length(m) and type
vi. Diameter of hole (mm)
vii. Loading density (kg/m)
viii. Charge factor (kg/m^3)

5.1.5 Parameters Related to Initiation Pattern

Initiation pattern is important for proper fragmentation, proper throw of blasted materials and fewer blasting hazards. The two most commonly used parameters related to initiation pattern are:

(i) Delay interval (ms)

(ii) Delay pattern or connection

Although it is unlikely that all of the above-listed parameters can be accounted for in detail by means of simple closed-form expressions, it is probable that better predictability can be achieved by explicitly incorporating those parameters, which predominantly govern the blasting operation.

5.2 SELECTION OF PARAMETERS FOR BLAST OPTIMIZATION

As discussed earlier, there are many parameters available for optimum blasting. All these parameters can not be taken for optimizing the blasting method successfully. Some of the parameters are taken for minimizing the blasting cost. These cost reduction and optimum blast design parameter will give an economical result. The parameters are

v. Drill hole diameter,

vi. Powder factor (desired),

vii. Cost of explosive,

viii. Numbers of holes required to blast.

Drill hole diameter: The drill hole diameter are taken as fixed parameter because the bit size available in the market is limited. The hole diameter is also vary as the geology condition of the strata. From the drill hole diameter other information required to design a blast geometry.

Powder factor: The powder factor for a particular mine is fixed as to give a continuous product size and also avoiding the oversize and under sized product.

Cost of explosive: This is a parameter where the value depends on the explosive strength and type of the explosive used.

Numbers of hole required to blast: It depends on the production of the mine. More the hole required to be blasted when the demand is more. These parameters are generally decided from the other parameters also. These are density of explosive and energy output from explosives. More hole for blasting at a time means the strength of explosive should be more so, the density will be high and the energy released from explosive should be more.

5.3 COLLECTION OF INFORMATION FOR IMPLEMENTATION OF THE OPTIMISATION METHODOLOGY

Information regarding the blasting and related parameters for implementation of the optimization technique were collected from three different mines of Mahanadi Coalfields Ltd.(MCL), viz. Basundhara Opencast Project, Ananta Opencast Project and Bharatpur Opencast Project. The results are given in Tables5.1 to 5.3.

Table 5.1: Blasting and other related information for Basundhara OCP

Sl. No.	Parameters	Specifications
1.	Strike length	1.5km
2.	Number of seam	Rampur1–3m (E grade) & Rampur2-10m(F) IB (Top)– 2m (D) & IB (Bottom) – 2.5m(C)
3.	Average thickness	Rampur – 35.94m to 45.26m IB – 0.15m to 3.34m & 1.23m to 3.41m
4.	Dip	5° to 8° towards south west
5.	Type of coal	D to E and F grade
6.	Total reserve	36.76 MT
7.	Stripping ratio	0.78
8.	OMS	20 T/ manshift
9.	Production	3 Lakh Tonne / month
10.	Pattern of drilling implemented	Square and zig-zag
11.	Bench height (detail)	1m – 6m
12.	Drill hole diameter	100mm, 160mm
13.	Burden	1→ 2.5m, 2.5→ 2.5m, 5→ 3.5m, 6→ 4m
14.	Spacing	1→2.5m, 2.5→ 2.5m, 5→ 4m, 6→ 4.5m

15.	Width of spread of blast(no of rows blasted at a time and length of row)	Variable
16.	Explosives used	Slurry Explosive
17.	Powder factor(o/b & coal)	O/B→5.5m^3, Coal→5.6m^3
18.	Stemming height	1→.75m, 2.5→ 1.5m, 5→ 3.5m, 6→ 4m
19.	Cost of stemming material	Cuttings
20.	Cost of explosive per kg (prime) (column)	Prime→Rs.17.25/kg, Column→Rs.13.92/kg
21.	Density of stemming material	Same as Coal
22.	Cost of detonating cord per meter	Rs.2.21/m
23.	Cost of booster per kg	Rs.17.25/kg
24.	Quantity of explosives used in a hole	30kg/m
25.	Delay used	25ms, 17ms
26.	Density of coal	1.6
27.	Cost of coal (all grades) per tonne	D→710/T, E→695/T, F→570/T
28.	Cost of drilling per tonne of coal	Not available
29.	Cost of blasting per tonne of coal	Not available

Table 5.2: Blasting and other related information for Ananta OCP

Sl. No.	Parameters	Specifications
1.	Strike length	2.5km East and West
2.	Number of seam	Talcher Seam – II & III
3.	Average thickness	Seam II – 30to 35m & Seam III – 12 to 15m
4.	Dip	1 in 18 to 12
5.	Type of coal	E and F grade
6.	Total reserve	148.48 MT
7.	Stripping ratio	1:0.52
8.	OMS	27.91 T/ manshift

9.	Production	89.13Lakh Tonne / year
10.	Pattern of drilling implemented	Square grid
11.	Bench height (detail)	Coal – 4 to 5.5m & O/B – 8 to 14m
12.	Drill hole diameter	160mm (Coal), 259mm(O/B)
13.	Burden	6m(O/B), 5m(Coal)
14.	Spacing	6.5m(O/B), 5m(Coal)
15.	Width of spread of blast(no of rows blasted at a time and length of row)	Variable
16.	Explosives used	SMS (Power Gel B & Navashakti)
17.	Powder factor(o/b & coal)	O/B→2.6m^3, Coal→4.9m^3
18.	Stemming height	1→.75m, 2.5→ 1.5m, 5→ 3.5m, 6→ 4m
19.	Cost of stemming material	Cuttings
20.	Cost of explosive per kg	Rs. 14/kg
21.	Density of explosive	Initial – 1.35, Final – 1.14 to 1.16
22.	Density of stemming material	Same as Coal
23.	Cost of detonating cord per meter	Rs.8/m
24.	Cost of booster per kg	Rs.2 Lakh/T
25.	Quantity of explosives used in a hole	Coal – 50kg/hole & O/B – 160 to 180kg/hole
26.	Delay used	25ms
27.	Density of coal	Seam II – 1.64 & Seam III – 1.6
28.	Cost of coal (all grades) per tonne	E→675/T, F→441/T
29.	Cost of drilling per tonne of coal	Not available
30.	Cost of blasting per tonne of coal	Not available

Table 5.3: Blasting and other related information for Bharatpur OCP

Sl. No.	Parameters	Specifications
1.	Strike length	2.5km East and West
2.	Number of seam	Talcher Seam – II & III
3.	Average thickness	Seam II – 30to 35m & Seam III – 12 to 15m
4.	Dip	1 in 14 to 10
5.	Type of coal	E and F grade
6.	Total reserve	118.39 MT
7.	Stripping ratio	0.84
8.	OMS	24.91
9.	Production	5.341 MT / year
10.	Pattern of drilling implemented	Square grid
11.	Bench height (detail)	8 to 12m
12.	Drill hole diameter	160mm
13.	Burden	4.5m
14.	Spacing	4.5m
15.	Width of spread of blast(no of rows blasted at a time and length of row)	Variable
16.	Explosives used	SMS (Power Gel B)
17.	Powder factor(o/b & coal)	O/B→2.2m^3, Coal→4.6m^3
18.	Stemming height	Not available
19.	Cost of stemming material	Cuttings
20.	Cost of explosive per kg	Rs. 14/kg
21.	Density of explosive	Initial – 1.35, Final – 1.14 to 1.16
22.	Density of stemming material	Same as Coal
23.	Cost of detonating cord per meter	Rs.8/m
24.	Cost of booster per kg	Rs.2 Lakh/T
25.	Quantity of explosives used in a hole	60 – 65kg
26.	Delay used	25ms
27.	Density of coal	Seam II – 1.64 & Seam III – 1.6
28.	Cost of coal (all grades) per tonne	E→675/T, F→441/T
29.	Cost of drilling per tonne of coal	Not available
30.	Cost of blasting per tonne of coal	Not available

5.4 OPTIMIZATION METHODOLOGY

In the suggested method implemented in the present work, priority is given to the number of holes and the powder factor desired. These factors combinedly take into consideration the energy output of the explosive, the density and type of explosive used.

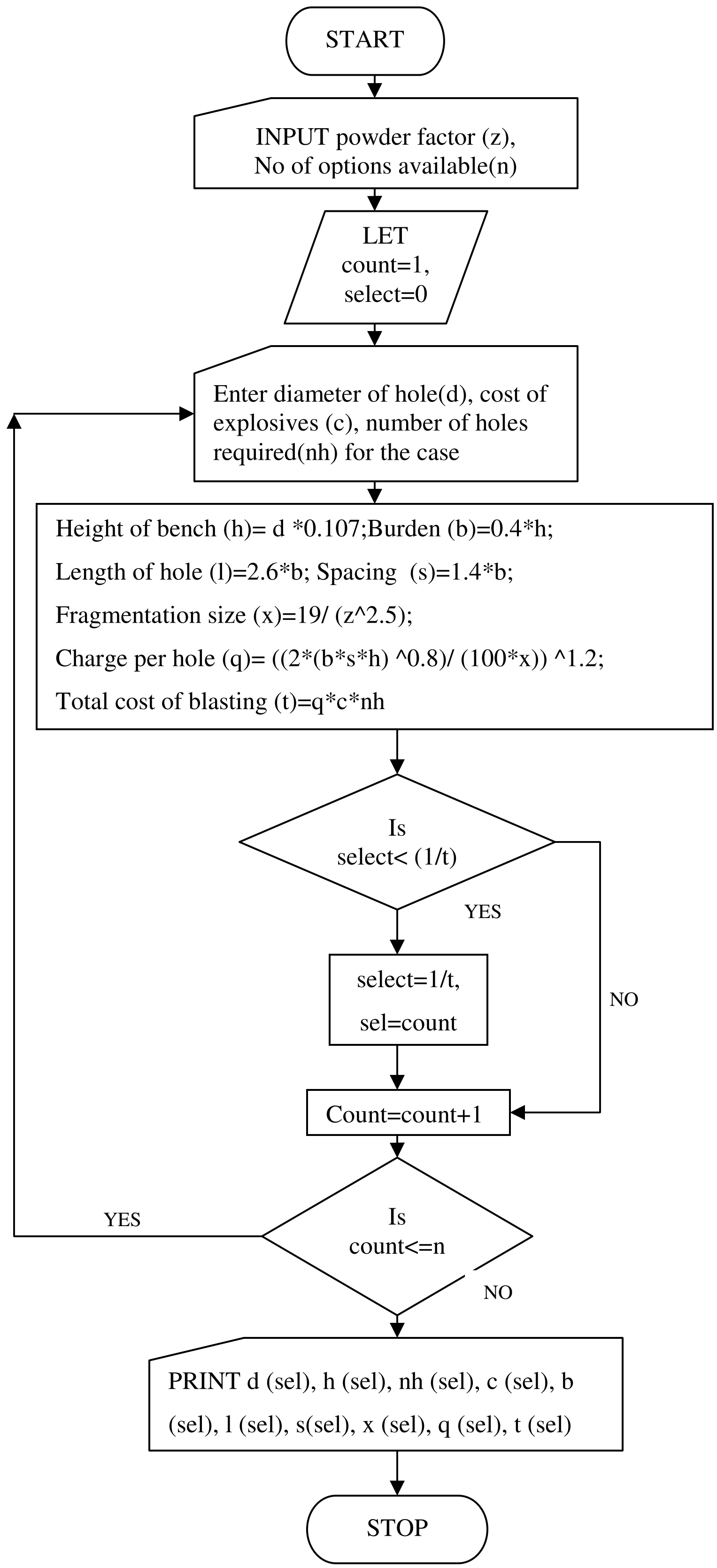
The main emphasis here is to optimize the cost with the desired production. The equations derived for the optimization are from the basic data available from the mines and the best combination and as per the optimum standards (Sethi and Dey, 2004).

In the present case, since the drill holes are fixed and cannot be varied as they depend on the available drill machines, this is taken as an input in the process. As per the customers requirement the fragmentation size needs to be constant which varies by the powder factor, so the powder factor is made constant or input data. The cost of the explosive which is the driving data, is also to be considered as an input.

As per the optimized models practiced till now; the burden, spacing, height of bench and the length of the hole can be easily correlated with the diameter of the drill hole by some derived, simple equations. From all the previous data the charge per hole and the total cost of the configuration can be calculated. The constants used in the equations are derived from the field data available from some Indian coal mines and some optimization equations already implemented in coal and metal mines (Sethi and Dey, 2004). The input parameters are those available explosive data from the coal mines in India.

The flow chart and the algorithm for implementation of the optimization methodology are as given below:

5.5 FLOWCHART OF THE PROGRAM



5.6 ALGORITHM OF THE PROGRAM

STEP 1: Start

STEP 2: Input the desired powder factor i.e. (z), let (select) =0

STEP 3: Input number of explosive options available as (n)

STEP 4: Let count = 1

STEP 5: Enter diameter of hole (d (count)), cost of explosive (c (count)) and number of holes required for the required output for case (nh (count))

STEP 6: Let height of bench (h (count)) is given as = d (count)*0.107

Burden (b (count)) =0.4*h (count)

Length of the hole (l (count)) =2.6*b (count)

Spacing (s (count)) =1.4*b (count)

Fragmentation size (x (count)) =19/ (z^2.5)

Charge per hole (q (count)) = ((2*(b (count)*s (count)*h (count)) ^0.8) / (100*x (count)))^1.2

Total cost (t (count)) =q (count)*c (count)*nh (count)

STEP 7: Is select< (1/t (count)), goto step 8 else goto step 9

STEP 8: Let select=1/t (count), sel=count

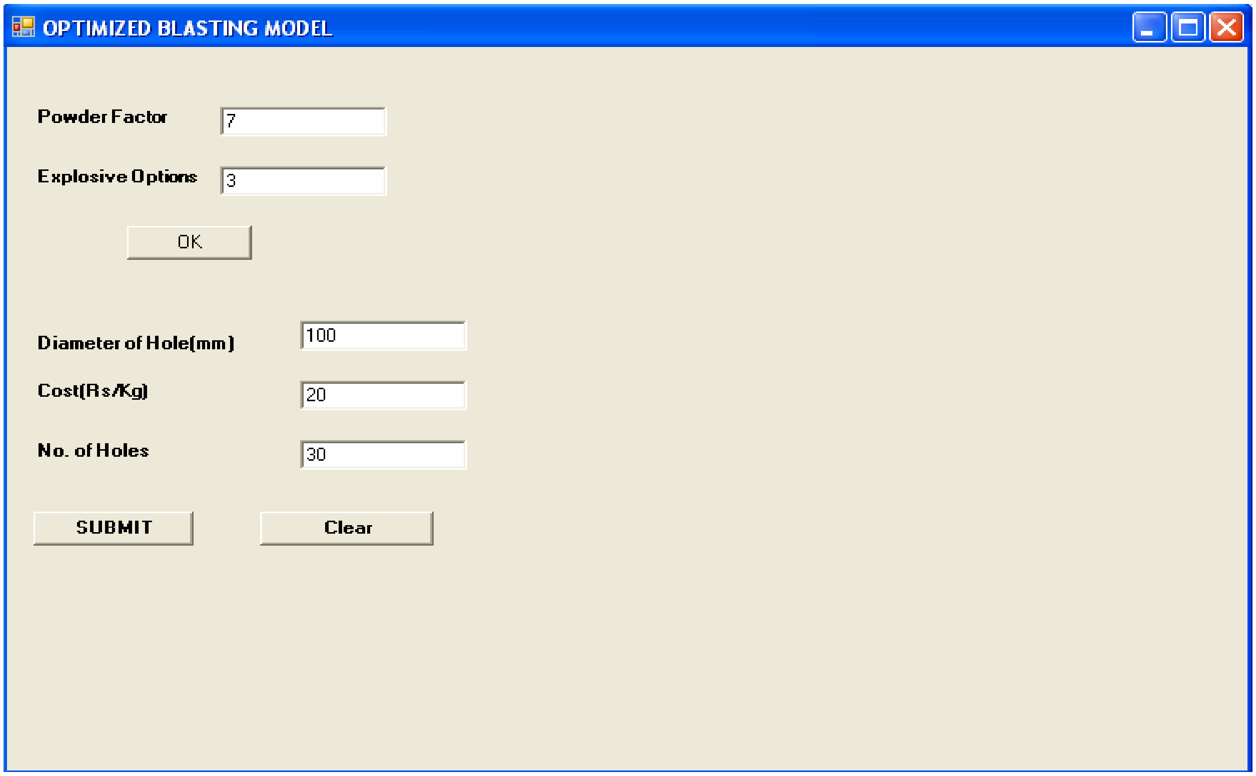
STEP 9: Let count=count+1

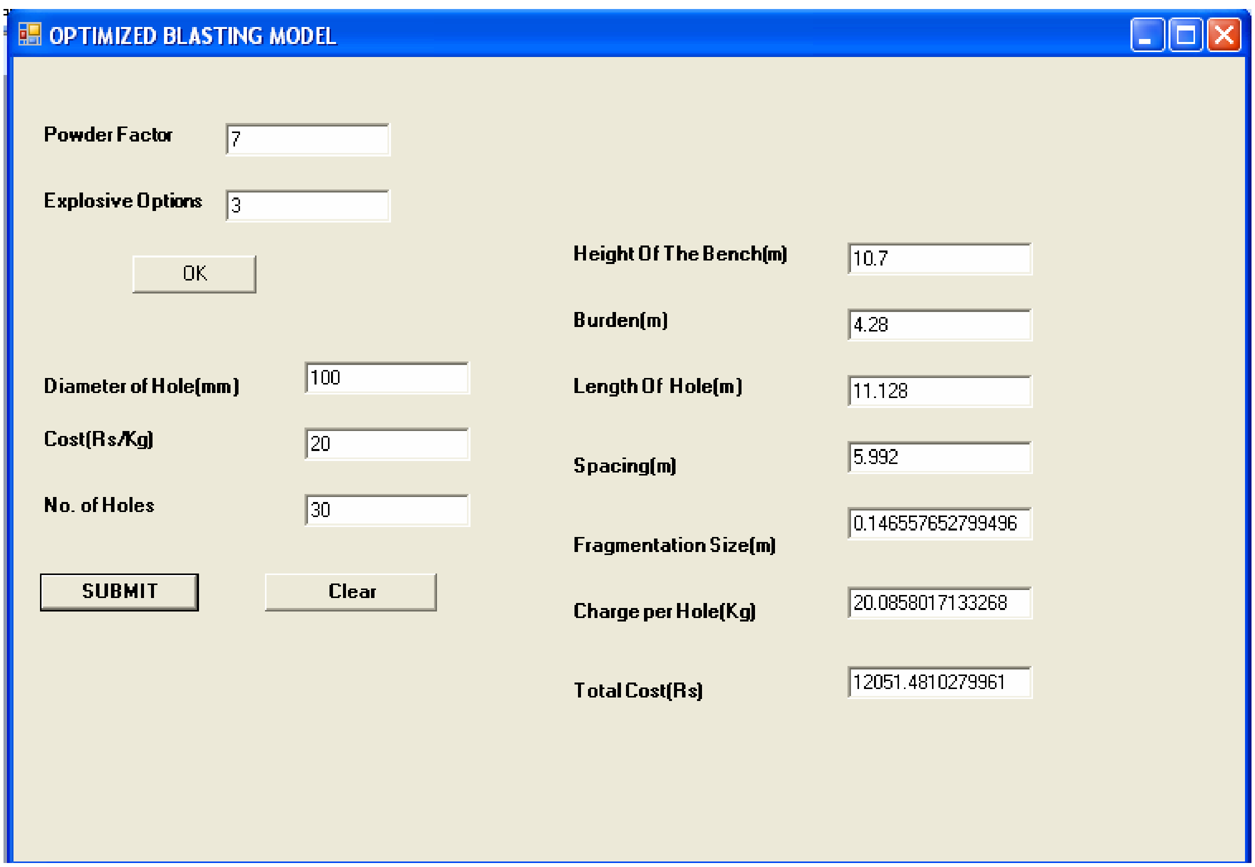
STEP 10: Is count<=n, goto step 5

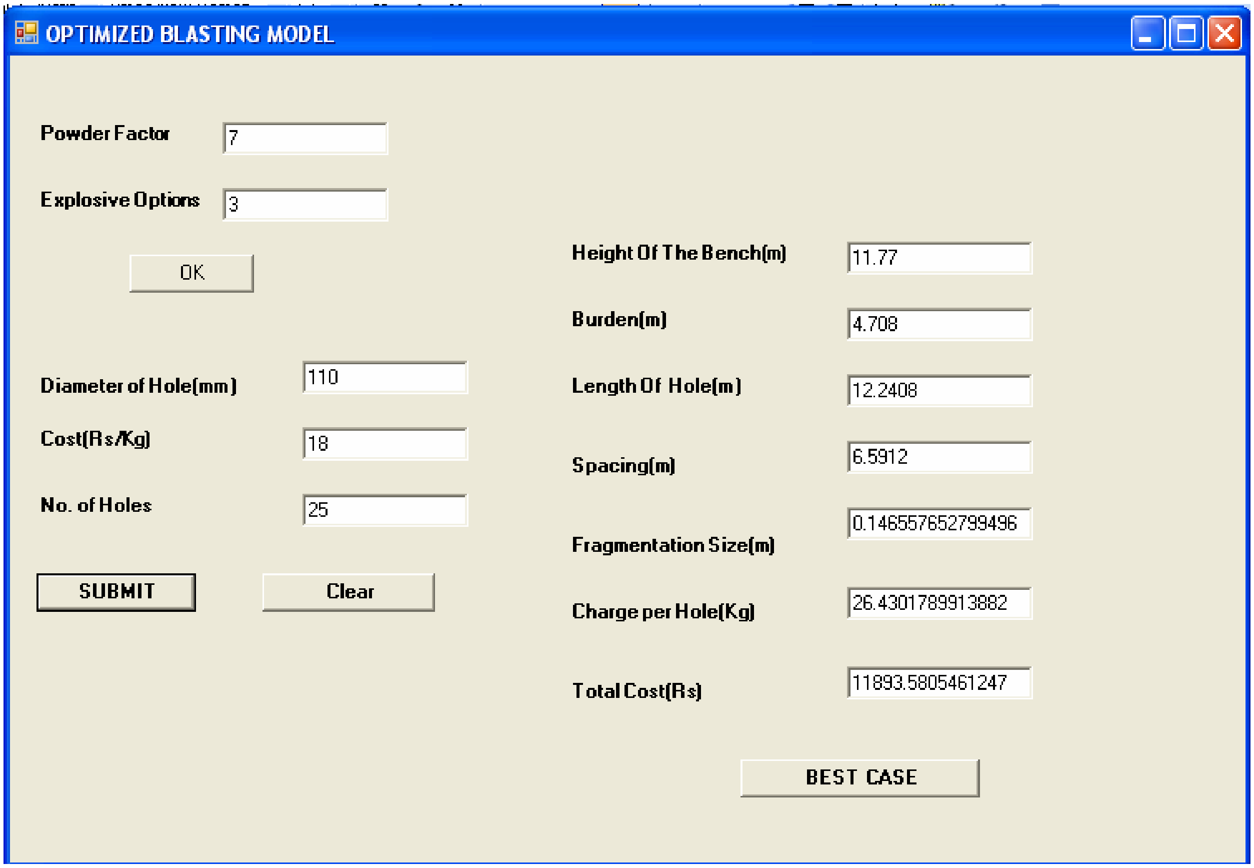
STEP 11: The selected option is (sel), and the other constraints are ((d (sel)), (h (sel)), (nh (sel)), (c (sel)), (b (sel)), (l (sel)), (s (sel)), (x (sel)), (q (sel)), (t (sel))

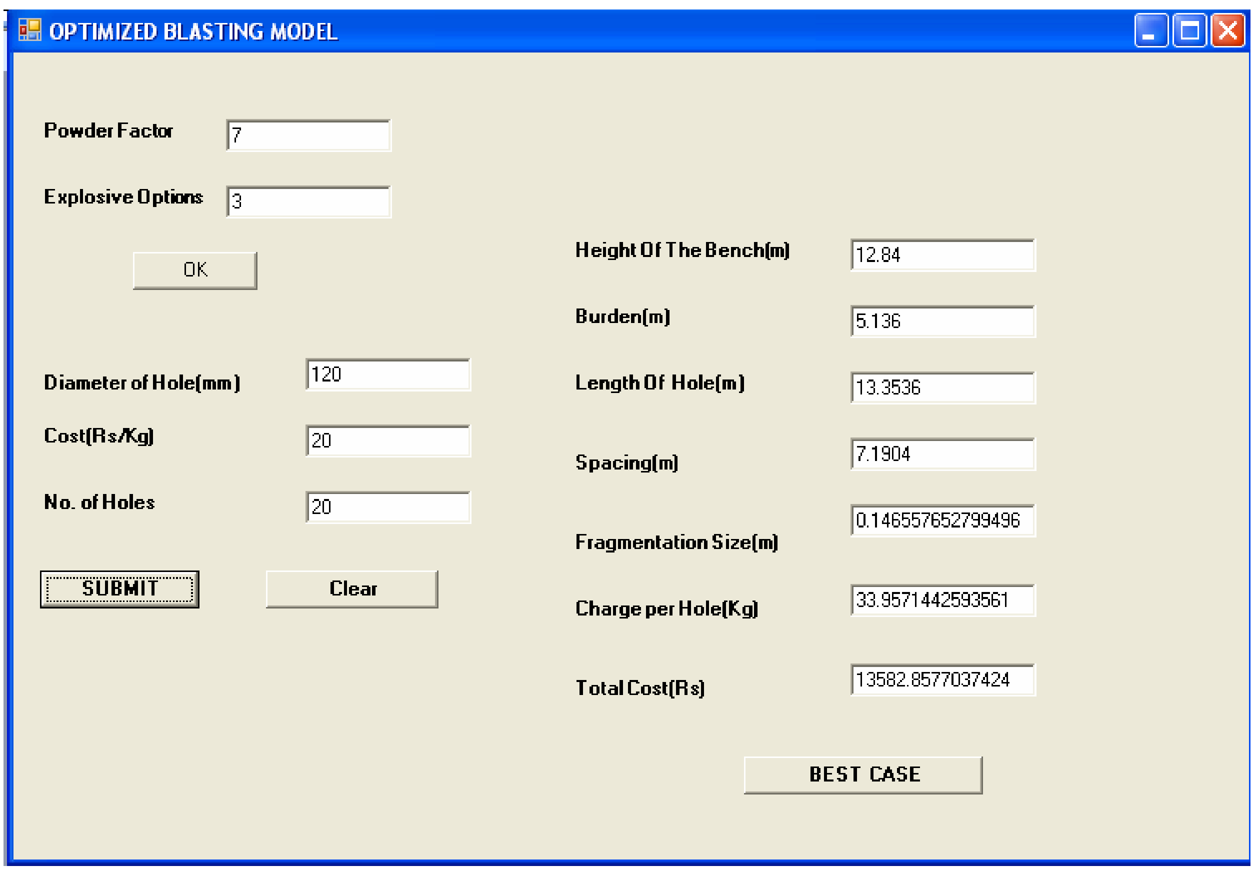
STEP 12: Stop.

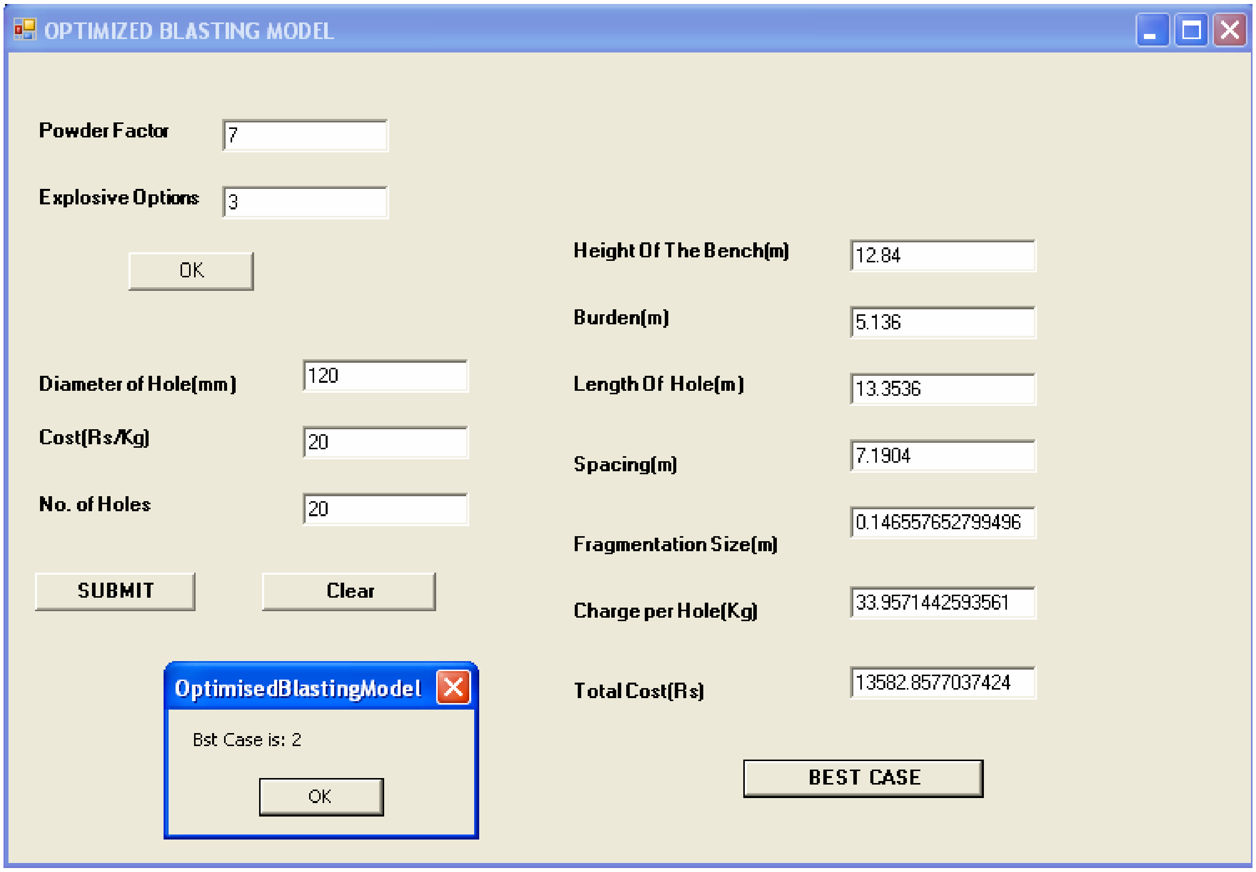
A computer program was written using visual basic on .NET platform. A sample presentation of the input and output parameters are as presented below:

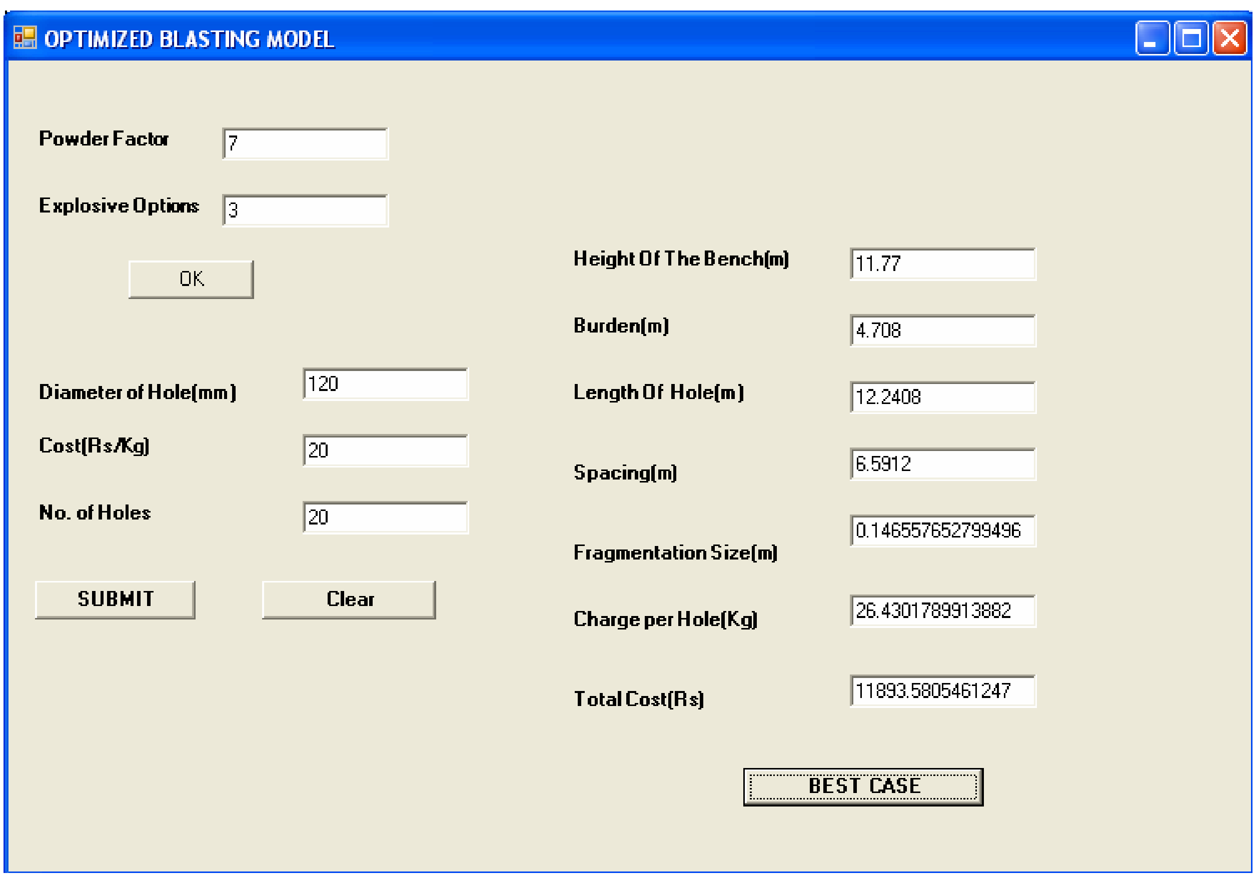

OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
100
Cost(Rs/Kg)
20
No. of Holes
30
SUBMIT
Clear


OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
100
Cost(Rs/Kg)
20
No. of Holes
30
SUBMIT
Clear
Height Of The Bench(m)
10.7
Burden(m)
4.28
Length Of Hole(m)
11.128
Spacing(m)
5.992
Fragmentation Size(m)
0.146557652799496
Charge per Hole(Kg)
20.0858017133268
Total Cost(Rs)
12051.4810279961


OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
110
Cost(Rs/Kg)
18
No. of Holes
25
SUBMIT
Clear
Height Of The Bench(m)
11.77
Burden(m)
4.708
Length Of Hole(m)
12.2408
Spacing(m)
6.5912
Fragmentation Size(m)
0.146557652799496
Charge per Hole(Kg)
26.4301789913882
Total Cost(Rs)
11893.5805461247
BEST CASE


OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
120
Cost(Rs/Kg)
20
No. of Holes
20
SUBMIT
Clear
Height Of The Bench(m)
12.84
Burden(m)
5.136
Length Of Hole(m)
13.3536
Spacing(m)
7.1904
Fragmentation Size(m)
0.146557652799496
Charge per Hole(Kg)
33.9571442593561
Total Cost(Rs)
13582.8577037424
BEST CASE


OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
120
Cost(Rs/Kg)
20
No. of Holes
20
SUBMIT
Clear
Height Of The Bench(m)
12.84
Burden(m)
5.136
Length Of Hole(m)
13.3536
Spacing(m)
7.1904
Fragmentation Size(m)
0.146557652799496
Charge per Hole(Kg)
33.9571442593561
Total Cost(Rs)
13582.8577037424
BEST CASE
OptimisedBlastingModel
Bst Case is: 2
OK


OPTIMIZED BLASTING MODEL
Powder Factor
7
Explosive Options
3
OK
Diameter of Hole(mm)
120
Cost(Rs/Kg)
20
No. of Holes
20
SUBMIT
Clear
Height Of The Bench(m)
11.77
Burden(m)
4.708
Length Of Hole(m)
12.2408
Spacing(m)
6.5912
Fragmentation Size(m)
0.146557652799496
Charge per Hole(Kg)
26.4301789913882
Total Cost(Rs)
11893.5805461247
BEST CASE

CHAPTER 7

DISCUSSION AND CONCLUSION

Discussion
Conclusion
Scope for Further Study

CHAPTER - 6

DISCUSSION AND CONCLUSION

6.1 DISCUSSION

The mining industry is heading towards a technology driven optimization process. It has been realized that the unit operations such as drilling, blasting, excavation, loading, hauling and crushing are interrelated variables in the total cost equation. The development, advancement and utilization of the innovative technologies are very important for the mining industry to be cost effective.

The last decade has seen dramatic progress in the advancement of blasting technology and the quality of performance of products. Monitoring instruments, measurement technologies and computing tools now have the capabilities to provide a bank of useful information that has previously been the subject of broad assumption. The performance and reliability of explosives and initiation systems are now at a level that allows the distribution and sequencing of explosives energy to be carefully controlled. The major developments in blasting technologies can be grouped according to the blast optimization pyramid (Fig. 6.1). Three main stages of this pyramid are planning, execution and output of a blast.

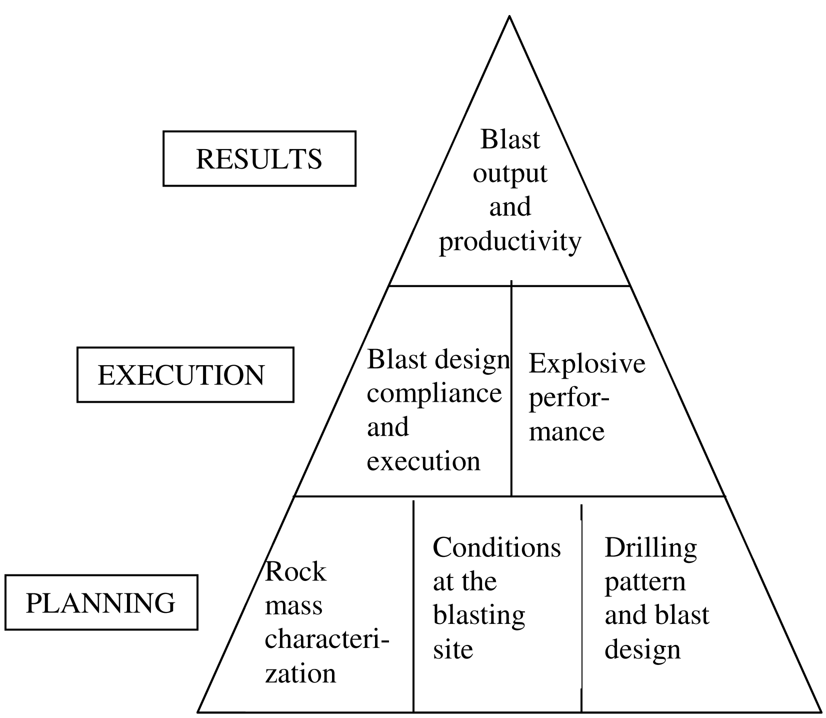


Fig. 6.1: Blast optimization pyramid

6.2 **CONCLUSION**

Efficiency of drilling and blasting operations can be defined in many ways, but the “bottom line’ is that they must contribute to the best overall economic result for the total mining operation. Thus decisions on drilling and blasting operations need to be considered in the overall context, and should not generally be based on short term economic factors. Drilling and blasting costs are always a significant part of the overall operating costs for surface coal mines. The cost of explosives may vary from 4.0% to 12% of the total operating costs and out of this about 20% of the costs are controlled by mine site management, and thus there should be a strong emphasis on reducing the explosive consumption without sacrificing performance.

In the present work a blast optimization model has been developed with simple methodologies which can be adopted by the mining industry to compare the explosive costs and achieve better blasting results and. The model takes into account the common explosives being utilized by large opencast mine at the moment which in turn is decided by the rock characteristics, density and other related parameters.

The model developed is a user friendly one, since by keeping the powder factor and number of choices of explosives available as constant and by varying the parameters like drill hole diameter, number of holes and cost of explosives one can compare the explosive performance and accordingly take a decision to select the proper type of explosives for blasting.

It may be noted here that the model has been developed based on case studies of three different mines of MCL, and it can be modified with collection of information from a large number of mines.

The model will definitely give some relief to the mine operators and blasting engineers to achieve a better output with a low cost of mining. By reducing the cost of explosives a considerable amount of expenditure can be saved, since modern mines require very large quantities of explosives throughout the life of the mine.

6.3 SCOPE FOR FURTHER STUDY

The present work is location specific as the data has been collected only from three different mines of MCL because of time constraints. However, by collecting information from a large number of mines across the country a large database can be created. Appropriate constant can then be obtained from this database which will help in deriving better results.

The work is an overview of the energy transmitted by the blasting; and has been considered by number of holes desired as the parameter due to unavailability of specific data. With availability of specific information about the explosives being used the explosive energy can be directly incorporated in the optimization model.

Optimization of drill hole parameters also contribute significantly towards the explosive consumption. Selection of appropriate drilling machine to match the desired output and optimization of drill hole parameters can also be incorporated in the model for better blasting results.

CHAPTER 7

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