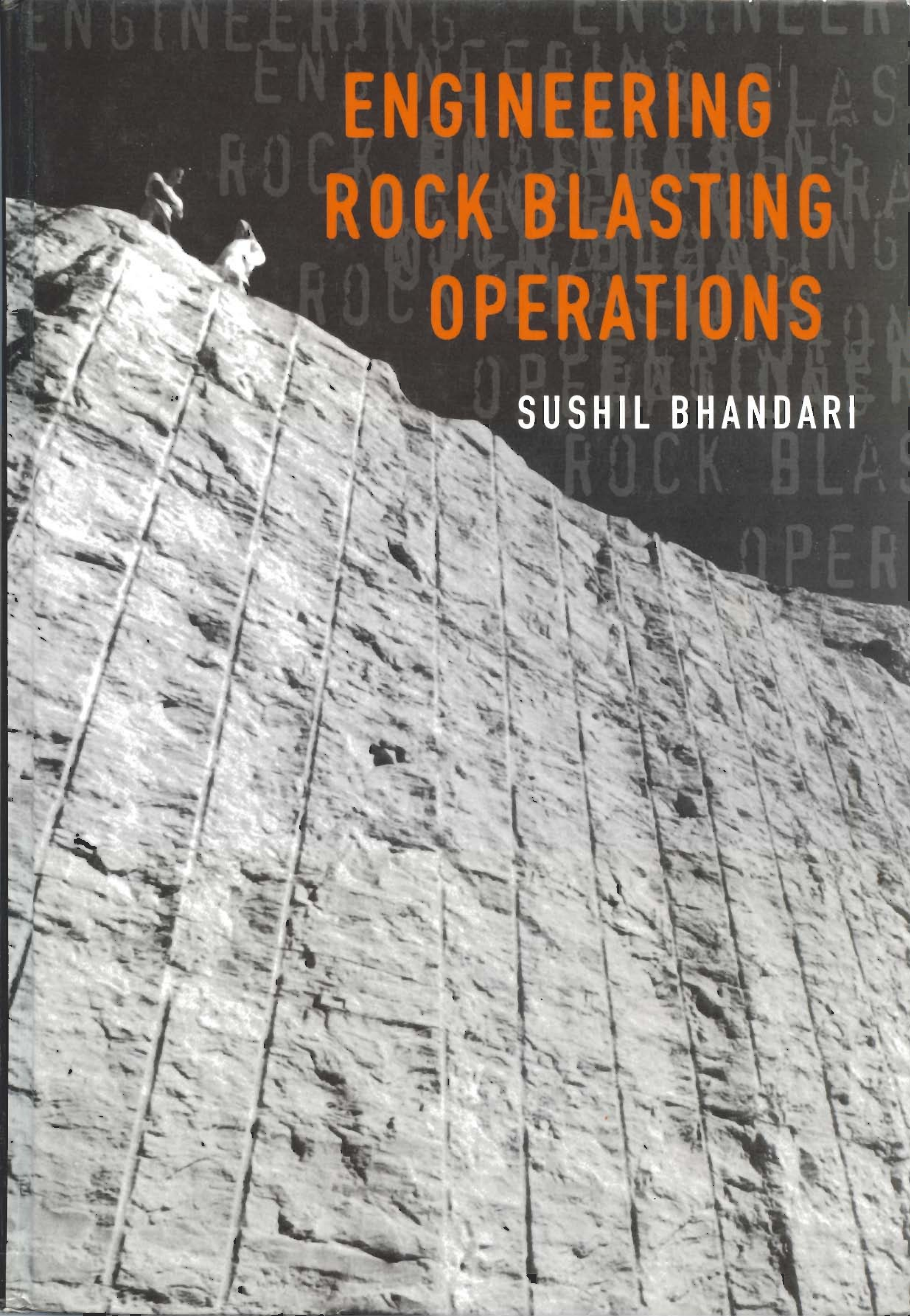
ENGINEERING ROCK BLASTING OPERATIONS

SUSHIL BHANDARI



Rock is blasted either to break ore or waste, or to create space. The engineering of blasting operations needs clearly defined objectives, materials, skilled techniques, necessary theoretical understanding of the rock fragmentation process and effect of conditions and experience in design as well as execution. The need to address such important aspects in a single book has been a long felt need.

The book provides extensive information about materials needed for carrying out blasting operations such as explosives and related accessories, understanding of the process of fragmentation, various techniques, design methods and applications including environmental aspects. Further, it contains recent techniques as well as research findings.

This book is meant to provide both the beginner and the professional with a better understanding of today's blasting technology which is continuously witnessing rapid changes.

Engineering Rock Blasting Operations

by

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Front cover: Controlled blasting results. Photograph provided by Nitro Nobel (recently: Dyno Nobel)

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Preface

Rock is blasted either to break ore or waste, or to create space. In mining and quarrying, the objective is to break rock into required size and extract the largest possible quantity of valuable resources from the ground at minimum cost. In civil engineering, rock is removed to create openings such as tunnels or caverns, or for deep excavations with the requirement of smoothwall and long-term stability. Similar requirements also exist for many mine openings. These different objectives give rise to different philosophies and techniques. The engineering of blasting operations needs clearly defined objectives, materials, skilled techniques, necessary theoretical understanding of the rock fragmentation process and effect of rock conditions and experience in combining them.

This book is meant to provide the beginner as well as the professional with a better understanding of today's blasting technology. Attempts have been made to include recent techniques, applications and research findings. The intent is to provide both a reference book for the working engineer and a textbook for the mining student.

This book is condensation of author's experience as mining engineer, explosives engineer, consultant, researcher and teacher in India, Australia and Oman over several years. It is an effort to transfer advances in explosives, mining, rock mechanics and computers to blasting operations.

The book has been organised into three parts. Chapters 1 to 7 provide knowledge about materials needed for carrying out blasting operations such as explosives and their properties, selection, loading, initiating devices and other accessories. Chapters 8 to 11 provide theoretical understanding of the process of fragmentation, rock mass conditions, timing and initiation sequence and evaluation of blasting results.

In the final part, Chapters 12 to 16 deal with techniques, design, and applications in surface and underground operations as well as specialised applications. Computer aided blast design details are provided in Chapter 17. In Chapters 18 to 20, environmental and safety aspects are considered.

The blasting industry is rapidly changing with new products, techniques and research findings but it is hoped that blasters and students will find this book equally useful.

This book has benefited from interaction with many individuals in the field of blasting: Professor William L. Fourney, University of Maryland; Dr Agne Rustan of Technical University, Lulea; Mr Roger Holmberg of Dyno-Nobel; Mr Frank Chiappetta of Blasting Analysis International Inc., Mr C. McKenzie of Australian Blasting

Consultants, Dr Finn Ouchterlony of Swedish Rock Engineering Research Foundation and many others who provided reports, papers and other material. Dr Piyush Pal Roy of Central Mining Research Institute, Dr. M.S. Sisodia of J.N.V. University and Dr N.C. Sacheti of Sultan Qaboos University reviewed and provided many useful suggestions. I have also drawn material from my research with Dr V.S. Vutukuri, University of New South Wales, my several students of Ph.D. and M.E. level. Several of my colleagues at J.N.V. University and Sultan Qaboos University have helped in preparation of the manuscript. I thank all of these persons, both named and unnamed. Acknowledgement is due to manufacturers who provided illustrations of their products and publications.

I am specially indebted to my wife Sudha and sons Amit and Akshat, who have shown great patience during the writing of this book.

Sushil Bhandari

CHAPTER 1

Engineering blasting operations

1.1 INTRODUCTION

Rock is blasted either to break it into smaller pieces such as in most mining and quarrying operations or large blocks for dimensional stone mining and some civil engineering applications, or to create space. The conditions under which blasting is carried out also affect the operations and the results. Precise engineering of blasting operations are needed to achieve the desired objectives. The engineering of blasting operations needs clearly defined objectives, materials, skilled techniques, the necessary theoretical background of the process of rock fragmentation and effect of rock conditions and experience in combining them.

1.2 BLASTING OBJECTIVES

In mining and quarrying, the main objective is to extract the largest possible quantity at minimum cost. The material may include ore, coal, aggregates for construction and also the waste rock required to remove the above useful material. The blasting operations must be carried out to provide quantity and quality requirements of production in such a way that overall profits of mining or quarrying operation are maximised. In-situ rock is reduced in size by blasting and crushing into the required size or with additional grinding, into a finer powder suitable for mineral processing. Large blocks needing secondary breakage or an excess of fines, can result from poorly designed blasts or due to adverse geological conditions. A well designed blast should produce shapes and sizes that can be accommodated by the available loading and hauling equipment and crushing plant with little or no need for secondary breakage. While optimising the fragmentation, it is also important, for safety and ease of loading, to control the throw and scatter of fragments. However, other times controlled displacement is provided, as in the case of casting of overburden or explosive mining, where part of the overburden is thrown to such a distance that it need not be handled again.

In civil engineering, rock is removed to create tunnels or caverns, or deep excavations at the ground surface for road cuts, foundations, or basements. The emphasis is not on high rates of production, although the job must be done as quickly and as cheaply as possible, but on creating space and leaving behind stable rock walls that are either self-supporting or require little reinforcement and lining. Requirements for smooth walls and long-term stability exist also in mine shafts, crusher stations and to a lesser extent, in mine development drifts that must remain open for moderate periods. Problems in the blasting of civil engineering works are most often associated with overbreak and underbreak. Special blasting techniques are applied in carrying out controlled blasting to produce smooth walled excavations. Often adjustments to conventional procedures are needed (Chapter 20).

In some civil engineering operations large blocks are needed for many operations such as dam construction, and break water construction. To meet these requirements techniques and explosive materials used are modified as practised for conventional blasting (Chapter 16). Underwater blasting is carried out to deepen harbours or river ways which also calls for specialised blasting operations. Similarly there are many other specialised operations like blasting in fire zones, and blasting of ice, both of which need materials and techniques different from the conventional blasting operations.

The size of blasting operations varies greatly from those needing a small charge to those requiring several hundreds of kilograms of explosives and also from those involving a few cubic meter rock to millions of cubic meter of rock. The equipment to charge holes is available for various sizes and types of explosives and initiating systems. For surface blasting operations such as quarries, foundations, trenches, the environment needs to be protected. Blast damage due to vibrations and noise needs to be avoided by careful blasting. Controlled blasting is often used to limit the projection of flyrock. For underground blasting operations care needs to be taken so that fumes and dust produced is minimal to protect persons working underground. Care needs to be taken of extraneous hazards while handling and during storing and also while carrying out blasting operations to prevent accidental explosions.

Mining and quarrying operations have production as a primary goal, but precautions must be also taken to avoid damaging the rock left behind, at least to an extent sufficient to preserve safe working conditions while the mine remains in production. In terms of mining economics, the optimum excavating method is one that maximises production and safety and minimises dilution, excavation costs and environmental costs. Long-term stability needs to be preserved, to avoid unnecessary subsidence or if the space is to be converted to other uses.

1.3 THE PROCESS OF FRAGMENTATION

The processes involved in rock fragmentation by blasting have been the subject of large amount of experimentation and studies. However, much of this is qualitative or semi-qualitative in nature. The difficulty is that at the time of fracture, fragmentation and displacement of rock, persons or sensors may not be close to the point of action. As a consequence, the results of various studies are useful but they do not represent definitive scientific results and provide only generalised directions for adopting techniques and materials for engineering blasts. Nevertheless, without basic theoretical considerations and understanding of the phenomena involved precise blasting results are difficult to achieve.

When an explosive detonates in a hole the pressures can exceed 10 GPa (100,000 atmospheres), sufficient to shatter the rock near the hole, and also generate a stress

wave that travels outward at a velocity 3000-5000 m/s. The leading front of the stress wave is compressive, but it is closely followed by the tensile stresses that are mainly responsible for rock fragmentation. A compressive wave reflects when it reaches a nearby exposed rock surface, and on reflection, becomes a tensile strain pulse. Rock breaks much more easily in tension than in compression, and fractures progress backward from the free surface. The gas pressures generated during the process also act to widen and extend stress-generated cracks or natural joints. The fragmentation process which takes place is the combined effect of the above two, the role of each is dependent on the rock conditions, blasting geometry, explosive materials and initiation systems. The relative role of the stress waves and gas pressure is not fully understood and hence an accepted process of rock fragmentation by blasting is lacking.

Much of the effort in recent years has been on understanding the vibration effects of blasting, determining damage criteria, developing techniques to reduce the damage to protect the structures near the blasting site (Chapter 18). Some efforts have also been made to understand the damage to the remaining rock and flyrock accidents with the aim of reducing these impacts.

The theoretical understanding has also developed from practising blasting personnel which allows one to adopt modifications in blasting parameters depending on rock and geological conditions as well as the desired results. Based on these, many empirical design calculations have been developed culminating in the use of computerised blasting design calculations for open pit blasting, tunnelling and stope blasting operations. Use of high speed photography and vibration monitoring equipment has been extensive as well as the monitoring of explosive and initiating system behaviour has been developed.

1.4 BLASTING MATERIALS

For carrying out blasting operations, explosives (in cartridged shape or free flowing form) initiating devices for these explosives, loading systems and techniques are needed. The use of black powder known to be used in the 13th Century has continued till now. NG (Nitroglycerine) based explosives developed a century ago dominated the usage till about 30 years ago. Recent years have seen developments with the discovery of new generation of explosives which offer greater flexibility in their range and application, and are safer in usage. With the increased popularity of new generation of explosives (like ANFO, slurries, emulsions) the demand for NG based explosives has declined. ANFO, consists of an oxidiser and a fuel. Ammonium Nitrate in prilled form (coated with an inert absorbent material, or treated with a surfactant to promote thorough mixing), sensitised with fuel oil is the least expensive, and when suitably employed performs as well as NG based explosives and is safer to handle and use. ANFO is supplied either in bulk or in waterproof polyethylene bags. The separate components, delivered for bulk mixing on site, are not classified as explosives and can be shipped without incurring the extra costs of transportation and storage precautions. However, it is susceptible to absorbing water, and therefore cannot be employed in wet conditions. In addition, the density of material is low, therefore, may not provide enough energy per unit rock as compared to similar other blasting agents.

To overcome these disadvantages and still retain the cheapness, performance and safety of ANFO specially designed Ammonium Nitrate based slurry and emulsion explosives have become widely used explosives. A combustible fuel is mixed with granular AN dispersed with a sensitiser and a thickener and with just enough of an aqueous solution of AN to give a semifluid mixture containing up to 20% water. The combustible fuel component can be material such as aluminium or even TNT. Thickeners include starch, water soluble vegetable gum, or oil with an emulsifier. Sensitisers include pigment grade aluminium, air bubbles, micro balloons, TNT or water soluble organic nitrates. The mixture can be sold in final thickened form, or a gelling agent is added before or during loading so that it forms a thick gel after charging into the hole. Thus the slurries and emulsions can be used in watery conditions. Recently the use of emulsified ANFO has increased. Sold as Heavy ANFO, this retains advantages of ANFO and increases their energy density.

Specially formulated explosives are used for blasting operations such as smooth blasting, dimensional stone blasting or for hot material blasting

Initiation of blasts needs devices (Chapter 6) which provide the necessary impetus such as flame or spark to the explosive at the desired time and place. There are basically two systems – electrical and non-electrical.

About 100 years ago it was the use of detonators and safety fuses. The safety fuse contains a core of black powder. When lit with hot flame the powder burns slowly at a standard rate (120 s/m). At the end of safety fuse is a crimped blasting detonator or blasting cap inserted in a cartridge of high explosive. This detonator provides a powerful localised shock to initiate detonation in a commercial explosive. Later on electric shotfiring was introduced and is now commonly used. Like plain detonators, electric detonators contain an initiating charge of primary explosive and a base charge of secondary explosive, but they also contain a bridge wire and an ignition charge that ignites when an electric current is passed through the wire. Either portable exploders or main power is utilised to provide an electric impulse. Later on to achieve the same objective, detonating cord consisting of a core of high explosives in a plastic sheath and protective wrapping started to be used and is most often attached to a primer or booster, which in turn initiates the charge. Unlike the safety fuse it has a high velocity of detonation (6 km/s), and is initiated by a plain or electric detonator. It is used for mass initiation of large blasts.

The detonating cord is much safer to handle than a detonator, is extremely water resistant and is comparatively safer. There are some problems, like discontinuity in the cord and others. To overcome these problems and that of electric blasting, use is made of other non-electric blasting systems which utilise about 1-2 g/m of initiating explosive or use hollow tube coated with initiating agents to provide initiation. These overcome many problems of electric detonation and the detonating cord, and also provide some of advantages of both the systems like delay and bottom hole initiation.

Delay blasting by use of delay electric detonators or the non-electric delay system, commonly used tends to give lower muckpile and longer throw, increased fragmentation, reduced vibration and reduced overbreak and back break. Electric delay detonators give a controlled time gap between pressing the plunger and initiating the charge. A large array of charges can be fired and in a controlled sequence. Electric delay detonators are manufactured with nominal half-second time intervals and also 8 to 100 ms.

Delay firing can also be achieved using non-electric millisecond delay connectors into a detonating cord trunkline. These are copper tubes about 75 mm long with an explosive charge at each end, separated by a delay mechanism.

Many tools are needed for checking electric circuits, locating extraneous electric hazards and for detecting current leakages, etc. (Chapter 7). Another major requirement is the modern use of high speed loading of explosives, which is specially applicable for large surface mine blasts or for awkwardly sited underground holes. Site-mixed and plant mixed explosive loading systems are frequently applied. High speed cartridge loaders are used for long holes and for achieving higher loading density per meter of hole.

1.5 BLASTING TECHNIQUES

Rock is in general blasted towards a free face. Bench blasting is often carried out in surface operations and even in large underground tunnels, caverns or stopes. When a bench face does not exist, a release cut (simply called cut) is made by drilling, cratering, or cutting. In bench blasting several holes are drilled in a pattern, loaded with explosives and initiating devices and then fired in a particular initiation sequence (see Chapter 12). Several variables are involved in bench blasting with respect to holes – diameter, depth & location, explosives and loading, initiation sequence. By making adjustments, the blasting engineer can obtain desired results in terms of fragmentation, reduced damage to the remaining rock walls, reduced costs, reduced vibration and airblast and throw.

Small diameter holes (32-35 mm) are used for very small operations whereas for larger operations hole diameters range from 100 mm to 400 mm. Bench blasting operations are usually accomplished by parallel rows of drill holes, detonating first the row nearest to the exposed face to give better release for successive rounds (see Chapter 9). Small diameter holes are generally loaded with explosive cartridges and larger holes are loaded with ANFO or other blasting agents. Many parameters which characterise surface blasting are related to burden. The burden is the distance between the exposed rock face and the nearest line of blastholes. The burden is kept about 30 times the hole diameter with averages between 20 and 40. The subdrilling is kept about 0.3 of the burden, except that no subgrade drilling is needed when joints run parallel to the floor or when blasting on coal benches. The length of stemming averages 0.7 times the burden and ranges from about 0.5 to 1.0 of the burden. Spacing is kept between 1 and 2 times the burden. These parameters need to be oriented and adjusted between the exposed rock face and the nearest line of blastholes.

Another important parameter is the sequence of firing, which includes the number of blastholes detonated in any one round and the time delay between successive rounds. The sequence can be varied using delay charges, to minimise vibration levels and unwanted rock damage and to give a more efficient pattern of rock removal.

In engineering projects, which are often in densely populated areas, special attention has to be given to prevent damage to nearby buildings and services, and to safeguard the public (see Chapter 18). Thus blast coverings are placed over the area being blasted to limit the throw of the flyrock (Chapter 19).

In engineering projects road cuts, trenches and foundation blasting are carried out.

In these, special attention is given to smoothwall blast designs to reduce damage to the surrounding rock. In forming road cuts, blasting is made difficult by continually varying height of bench, heights of more than 10 m are usually blasted in more than one lift. Trenches and ditches are blasted by two basic methods, blasting with the overburden still in place or with it removed. The trench is advanced by blasting towards a free face. For narrow trenches say up to 600 mm wide, a single row of holes may suffice, but better results are usually attained by staggered or paired holes in two rows. Usually 300-600 mm of subdrilling is needed. In wider and deeper trenches, the holes are drilled in rows and tend to be larger in diameter.

Casting of overburden (also referred as controlled trajectory blasting or explosive mining) not only breaks rock but also moves the rock and earth either to reduce handling of overburden (Chapter 16) or to build a dam across a river.

Underground blasting (Chapter 15) needs release of rock within the confines of underground rock excavations. The quantity of explosive required per unit volume of rock excavated because of the confinement of the blast, is much greater than in the case of open cut excavations. When blasting a rock tunnel or mine drift, the first holes to be detonated (those with shortest delays) create a cut, an opening toward which the rest of rock is successfully blasted. Subsequent delays blast the rock into the cut in a pattern of rings of increasing diameter until they reach the perimeter, which is outside the line of holes, often with reduced charge density. There are two main types of cut; firstly an angle cut, in which the blast holes are inclined inward to meet each other; and secondly a parallel cut, in which the holes are drilled parallel to each other in the direction of advance. Nowadays, for larger openings parallel hole drilling is often used in which large empty holes are drilled by jumbo. Typically these have diameters of 57, 76 or 125 mm. The smaller charged holes have diameters in the range of 30-50 mm and are loaded with a charge concentration ranging between 0.25 and 0.55 kg per meter. Tunnels and mine drifts of a diameter smaller than about 8 m and larger headings in good quality rock are excavated full face by drilling and blasting the entire face in a single round. As rock conditions deteriorate the heading becomes larger, an upper section (top heading) is often removed first, followed by supporting the roof and then removal of the remaining bench. The length of advance per round is limited by the quality of rock and by the diameter of the excavation. Pull varies between 0.5 m in very weak ground to as much as 3 m in massive and self supporting rock.

In shaft sinking two alternatives for blasting are available, benching and full face. In benching the two halves of the shaft bottom are blasted alternatively. In the full-face method angle cuts are most often employed. The average powder factor for a 3.0 by 4.5 m shaft is 3.25 kg of explosive per cubic meter of rock, varying from about 2 to 6 kg according to shaft size and rock strength.

Raises can be driven blind or can follow a pilot hole. Holes are drilled upwards in a regular pattern, most often in angle cut pattern. Use of the vertical crater retreat method, when holes are drilled downwards and blasting is in slices, has made raising a much easier operation in recent years.

In stope blasting, depending on the method of mining employed, either short hole blasting or long hole blasting is employed. In traditional short-hole blasting, one of the three standard types of rounds is employed – breasting, overhand stoping or underhand benching. Hand held drills are commonly employed, although jumbo drilling can be used to increase production where there is sufficient headroom, particularly in room and pillar mining operations. Benching and crater retreat methods of mining make use of long holes of large-diameter (150-200 mm). The production blast is detonated one or two rows at a time, using either full column loading, decking or bench slicing in which only the bottom of the slice is loaded.

1.6 CONTROL OF BLASTING RESULTS

In terms of mine and quarry blasting the optimum result is one that maximises production, fragmentation and safety, minimises dilution and excavation costs and environmental impact. Blasting techniques and designs in mines are optimised over months or years, till blasters gain knowledge of the rock conditions. Even though computer aided designs (see Chapter 17) are available, their adaptation to actual field conditions is necessary. Control is often needed for obtaining specific fragment size distribution, reduction of vibration, airblast and flyrock damage and damage to the remaining rock.

In rock excavation for civil engineering projects and in quarry or surface mines blast vibrations must be limited to minimise environmental impact, damage to nearby structures, and damage to the rock walls of the perimeter. In mining, high levels of vibrations can damage the open pit slopes or underground pillars and lead to subsequent problems of safety and subsequent recovery of ore from the blast affected areas. In stopes, damage to the hanging wall can lead to slabbing, high dilution, and need for secondary blasting to relieve ore passes that become blocked by large fragments of ore and waste rock.

Empirical predictions of blast vibration are often in error. This leads to a requirement for vibration monitoring and subsequent blast designs. Seismographs can be used to record the vibrations generated by blasting. Vibration levels can be reduced by limiting the charge weight per delay to an amount sufficient to achieve the required degree of fragmentation. Permissible vibration levels are specified in relation to the levels that can be tolerated by the wall rock, by different types of structures or by people. The most common criterion for prevention of structural damage at the surface, is that the peak particle velocity should not exceed 50 mm/s. In another practice structures are categorised and criteria is altered according to structure. The frequency spectrum of the transmitted vibration also plays a role in determining which blasts cause the most complaints. Frequencies in the range of 5-20 Hz are apparently most annoying.

Air transmitted vibrations (called airblast) also need to be kept within certain limits. Damage can occur, during above-ground demolition, during unstemmed blasting of tunnels and shafts, and where large quantities of detonating cord are exposed at the surface. Airblast over pressures greater than 0.7 kPa will almost certainly break all windows. Even in the absence of damage, complaints and legal actions resulting from annoying levels of noise and vibrations close operations down.

Damage to the remaining rock occurs when fracturing, including crushing and radial cracking of rock, around the blasthole takes place. It is caused by excessive explosion pressures, excessive burden, inadequate time between rows in multirow blasting, or unfavourable orientation of the blasting row relative to the jointing, or principal stress directions. Also overbreak occurs when rock is removed beyond a specified perimeter. It can result from overblasting, from inaccurate blasthole drilling, or from the action of gravity combined with geological conditions. Backbreak is the removal of rock beyond the line of blastholes. Underbreak is rock remaining within a specified excavation perimeter that should have been removed by the blast.

Blasthole pressures in most production blasting applications are in the range of 2-5 times the compressive strength. Blasthole pressures and consequently wall damage and backbreak, can be reduced either by reducing the density of the explosive (for example, by adding an inert filler to ANFO, which produces fracture radius one half to one-quarter of that of a dynamite), by reducing the charge diameter in the blasthole (decoupling), or by air decking.

Special methods for producing smooth surfaces are some of the controlled blasting techniques (Chapter 20). These include line drilling, pre-splitting, smooth and cushion blasting.

Blasting operations need to be carried out with the utmost safety of man, machine and environment. Though much safer explosives and accessories have become available, the explosives still need to be treated with the utmost respect while storing, transporting and handling them. Adequate safety precautions are essential to carry out blasting operations without accidents.

CHAPTER 2

Explosives

2.1 INTRODUCTION

Explosive is a solid or liquid substance or a mixture of substances which on application of a suitable stimulus is converted in a very short time interval into other more stable substances, largely or entirely gaseous, with the development of heat and high pressure.

Many types of modern industrial explosives are available to meet the varied requirements of the mining, quarrying and construction industries. A wide range of products are available. It is important to know about each type of explosive in order to select and use the explosive efficiently and safely.

2.2 CLASSIFICATION OF EXPLOSIVES

The explosives can be classified into the following types:

1. Low (or deflagrating) explosives;
2. High explosives, (a) Primary and (b) Secondary.

Low explosives were the earliest to be developed. These lead to an explosion which is really a rapid form of combustion in which the particles burn at their surfaces and expose more and more of the bulk until all has been consumed. Such an explosion is called deflagration and the reaction in this case moves slower than the speed of sound. Typical examples of this category are the blasting powder or gun powder, propellants in ammunition, rocket propellants and pyrotechnics.

High explosives, depending on their composition, explode at velocities of 1500-8000 m/s and produce large volumes of gases at considerable heat at extremely high pressures. High explosives themselves may be further subdivided into primary explosives and secondary explosives. Primary explosives are characterised by their sensitiveness to stimuli like weak mechanical shock, spark or flame, the application of which will take explosive compounds from state of deflagration to detonation easily. Examples of these explosives are Mercury Fulminate, Lead Azide, Lead Styphnate, Tetrazene and other mixtures. These explosives are used as initiating charges in the initiating devices such as detonators. Secondary explosives are capable of detonation only under the influence of a shock-wave, normally generated by the detonation of primary explosives. Secondary explosives of this type are military explosives like TNT, RDX, PETN, Tetryl and other combinations of these and indus-

Table 2.1. The significant developments in commercial explosives.

Year	Development
13th Century	Mention of saltpetre and other blasting powders in Arabian and Chinese writings
1242	Black powder formula by Bacon
1846	Sobrero discovered nitro-glycerine
1861	Alfred Nobel sets up nitro-glycerine plant
1866-75	Dynamite and blasting gelatine commercially manufactured
1880's	Permissible dynamites investigated
1950's	Ammonium Nitrate and combustibles as dry powders (ANFO)
1957	Watergel (Ammonium Nitrate, TNT and Aluminium powder)
1964	Emulsions (Emulsified Ammonium Nitrate and Nitric Acid)
1969-74	Watergel (Methyl Amine Nitrate, or Hexamine Nitrate or Aluminium powder)
1979-82	Heavy ANFO (Up to 50% emulsion and ANFO)

trial explosives like nitro-glycerine, emulsion, slurries, watergels, ANFO and other powder explosives. These explosives are normally set off with suitable initiating devices like detonators or detonating cords and in some cases there is need of initiation by another high explosive. The explosive needing another high explosive is called Blasting Agent such as ANFO, some slurries, some emulsions and mixtures of emulsions and ANFO.

2.3 NITRO-GLYCERINE BASED EXPLOSIVES

Nitro-glycerine (NG) has been in use for a long time as the most important sensitiser for commercial explosives. It is made by reacting a mixture of glycerine and glycol with a mixture of acids, during which the temperature must be carefully controlled. Discovered by Sobrero in 1846, it was developed to a commercial scale by Alfred Nobel. Nitro-glycerine is a viscous liquid which freezes at 13.2°C to a sensitive solid explosive. In both forms it is too sensitive to be handled safely. It is therefore converted into a more convenient gelatinous (plastic) solid by the addition of 8% gun cotton or nitro-cellulose (obtained by the nitration of cotton) to form Blasting Gelatine or by absorbing it in Kieselghur to give straight dynamite (containing about 75% NG), by admixture with other explosive agents and additives to form other types of dynamites. The properties of nitro-glycerine, and the way in which it is mixed with other ingredients, determines the type of explosive produced. Due to higher cost of production and pollution control involved it is expected that the usage of NG-based explosives will not increase.

A wide range of NG based explosives are produced. They are packed in cylindrical cartridges, 25 mm diameter and larger, with lengths ranging from 200 mm to 1000 mm (Fig. 2.1). Various paper shells or wrappers are used to package and protect it from moisture. The overall weight percentage quantity and type of wrapper have an important influence on explosives fume production, water resistance, tampability and loadability.

NG based explosives must be handled and stored according to all regulations. Stock levels must be consistent with the rate of use, the oldest stock should be used

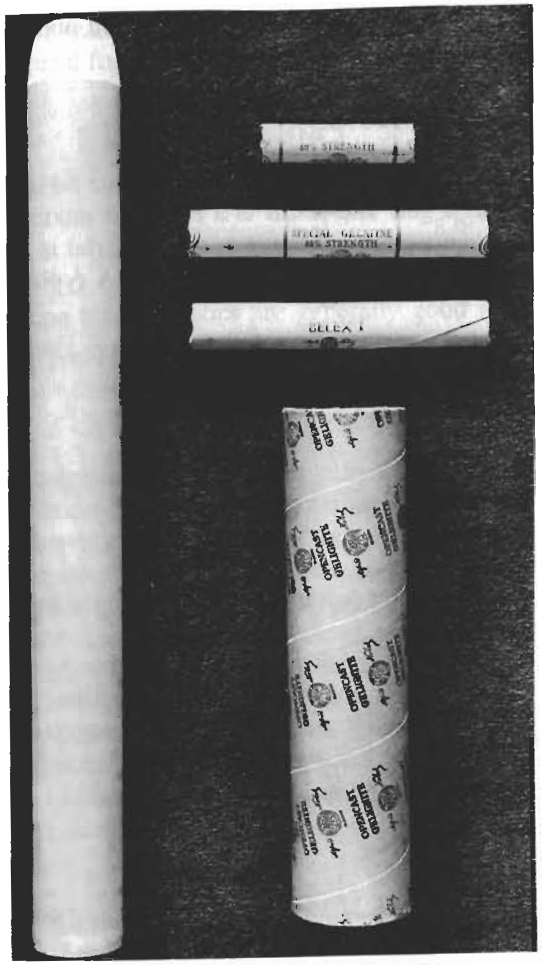


Figure 2.1 Different types of cartridges of explosives.

first. Cartridges which begin to deteriorate where the nitro-glycerine leaks should be destroyed in an approved manner by an experienced blaster in presence of officials.

2.3.1 *Straight dynamites*

Straight dynamites contain 15% to 60% explosive oil (nitro-glycerine plus ethylene glycol). In addition antacid, carbonaceous material and sodium nitrate is used. A typical percentage formulation for a 40% straight dynamite would be:

Nitro-glycerine (NG)	40%
Sodium Nitrate (SN)	44%
Antacid	2%
Carbonaceous Material	14%

Though they have high detonation velocities and good water resistance, they have high flammability, are highly sensitive to shock and friction and produce large noxious fumes. As such straight dynamite have only limited use in ditching and underwater blasting.

2.3.2 *Gelatine dynamites*

Gelatine dynamites are similar to straight dynamites, except that the explosive oil has to be colloided with nitro-cellulose to form a gel. The result is a cohesive mixture that has better water resistance properties than straight dynamites. The material is of higher density, rubbery and much less sensitive than straight dynamites. A typical gelatine explosive would be as below:

Nitro-glycerine(NG)	22.5%
Nitro-cotton(NC)	0.5%
Ammonium Nitrate(AN)	15.0%
Sodium Nitrate(SN)	47.2%
Woodmeal	7.2%
Sulphur	6.2%
Calcium Carbonate	1.4%

The composition is varied to alter the density, energy output, power, oxygen balance and fume characteristics.

Blasting gelatine is the strongest form of explosive used industrially and serves as a standard against which explosives are graded. It consists of nitro-glycerine, to which a temperature depressant (ethylene glycol dinitrate) is added plus approximately 7% nitrocellulose, to form the gel. The gelled or colloided mixture is totally water resistant, however, its poor fume characteristic make its application limited.

2.3.3 *Ammonia dynamites*

Ammonia dynamites are actually ammonium nitrate explosive sensitised with NG. Generally they supply the same blasting strength as most straight dynamites and are less costly. Typical constituents of 40% ammonia dynamite are:

Nitro-glycerine (NG)	14%
Ammonium Nitrate (AN)	36%
Sodium Nitrate (SN)	33%
Antacid	1%
Carbonaceous Material	10%

They have high heaving effect, are hygroscopic and are desensitised by water and therefore, their application is generally limited to soft rock and where water conditions are not a problem.

2.3.4 *Ammonia gelatine dynamites*

Ammonia gelatine dynamites are similar to the corresponding ammonia dynamites except for the addition of nitro-cellulose to the explosive oil to form a gel. These semigelatinous explosives are equal in most respects to gelatine dynamites except for somewhat lower velocity of detonation and slightly less resistance to water, however,

their water resistance properties are greater than conventional ammonia dynamites. A typical formulation for a 40% grade is:

Nitro-glycerine(NG)	26.2%
Nitro-cellulose(NC)	0.4%
Ammonium Nitrate(AN)	8.5%
Sodium Nitrate(SN)	49.6%
Carbonaceous fuel	8.9%
Antacid	0.8%

Fume characteristics are generally good in all strengths and for many uses an ammonia gelatine dynamite can replace a gelatine dynamite at a lower cost.

2.3.5 *Semigelatine and nitrostarch explosives*

Semigelatine dynamite is a hybrid formulation with properties between those of gelatine dynamite and granular dynamite (which contains no nitro-cellulose, has relatively low density and is susceptible to the effect of water). They combine the economy of ammonia dynamites with the water resistance and cohesiveness of ammonia gelatine explosives, contain less explosive oil, sodium nitrate and nitrocellulose, but more ammonium nitrate. They have low cost in comparison to comparable gelatine dynamite, offer good fume characteristics and water resistance and are suitable for underground use.

Nitrostarch explosive is a formulation in which the glycerine is replaced entirely by nitrostarch, eliminating certain objectionable characteristics such as NG headache and nausea. They contain no liquid sensitisers, cannot freeze and do not exude or leak during storage.

2.4 AMMONIUM NITRATE DRY MIXES (ANFO)

Prilled ammonium nitrate (AN) and fuel oil (FO) mixtures, known as ANFO, were introduced for blasting operations in mid 1950s. Ammonium nitrate in a proper form when mixed with carbonaceous or combustible material in appropriate proportion forms a blasting agent. Although, many forms of AN could be used with a solid or liquid fuel to form a blasting agent, the porous prilled forms are preferred for ANFO. Through the years, fuel oil has proved itself to be an almost ideal fuel for AN. It is readily available, is inexpensive and easily mixed with AN to produce a uniform mix commonly known as ANFO. Small porous spherical prills absorb oil readily giving a dry mix which has good pouring properties. The microprills in each prill allow the AN to absorb and retain the optimum amount of fuel oil. The prills are hard enough to withstand transport shocks without breaking down, yet soft enough to breakdown and give a high loading density when blow loaded into blastholes with equipment such as pneumatic blasthole charger.

AN is stable at ambient temperatures but can absorb moisture from the atmosphere if the humidity is above about 60%. To minimise moisture absorption and caking, the prills are lightly coated with anti-caking agents. While it is strong supporter of combustion, AN is not flammable. However it is an oxidiser. Proper mixing

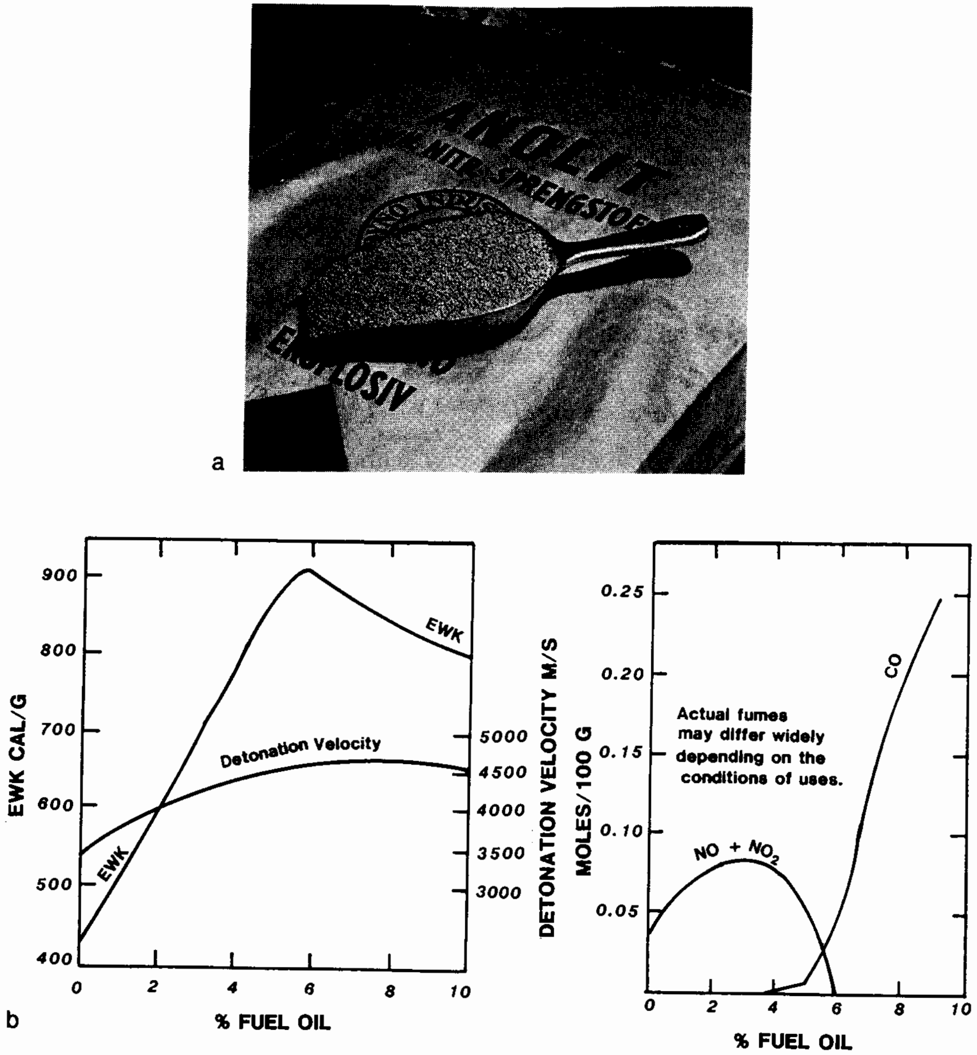


Figure 2.2 (a) Factory mixed ammonium nitrate fuel oil mixture and (b) E_{wk} and detonation velocity of ANFO versus percent of fuel oil and idealised ANFO fumes versus percent of fuel oil.

of AN and FO is important for predictable explosive performance.

When 94.5% AN and 5.5% FO are mixed the following reaction takes place:

$$3NH_4NO_3 + CH_2 = 3N_2 + 7H_2O + CO_2 + 930 \text{ kcal/kg}$$

This mixture represents the ideal reaction of an oxygen-balanced mixture. None of the detonation products are poisonous. Maximum energy output and VOD is obtained by mixing 7.5 litres of FO with 100 kg of AN prills (Fig. 2.2). With too much or too little FO, energy yield falls:

A mixture containing 92 % AN and 8% FO creates oxygen deficiency:

$$2NH_4NO_3 + CH_2 = 5H_2O + CO + 2N_2 + 810 \text{ kcal/kg}$$

As a result the carbon in the mixture is oxidised only to CO, a poisonous gas, rather than relatively harmless CO_2. Because of the lower heat of formation only about 810 kcal of heat is released for each kilogram of ANFO detonated.

A fuel short mixture containing 96.6% AN and 3.4% FO creates an oxygen excess condition.

$$5NH_4NO_3 + CH_2 = 11H_2O + CO_2 + 4N_2 + 2NO + 600 \text{ kcal/kg}$$

Some of the nitrogen from the ammonium nitrate combines with this excess oxygen to form nitrous oxide which, upon exposure to normal atmosphere, forms NO_2, an extremely toxic gas. An oxygen balanced mixture, thus, maximises energy release while minimising formation of toxic gases.

In each case, the energy release is obtained by calculating the difference of heats of formation of the ingredients and the products or product amounts formed, but the major penalty is in reduced energy release.

The appearance of distinctly orange coloured fumes after an ANFO blast may indicate too little FO in at least some of the ANFO. These fumes can also appear where properly mixed ANFO has become wet and has absorbed blasthole water. The field mixture commonly employed consists of 94% AN and 6% FO, which ensures a slight oxygen deficiency-fuel excess as a safety measure.

The most common method of mixing small quantities of ANFO is to pour the correct amounts of both ingredients into a hand-operated mixer (concrete type) (0.05-0.10 m^3 capacity) and after thorough mixing, tip the ANFO back into the empty bags, and from there into blastholes. Such mixing care is essential for small diameter blastholes and for maximum explosive energy output. Commonly, the bowl is lined with plastic or epoxy resin to prevent corrosion. Aluminium bowl is also used.

Any machine used for mixing should be designed so as to avoid the possibility of frictional heating and any bearings or greases must be protected from spillage of AN or ANFO. A petrol engine or diesel engine must not be used for power-operated mixer. Approved electric-operated motors can be used.

For the purpose of providing an immediate visual indication of the distribution of FO in the mix, and of distinguishing between a mixed product and straight AN, it is advisable to colour the FO with a vivid oil-soluble dye before FO is added to the AN. When added at the rate of 100 g per 500 l of FO, red dye should give adequate coloration of ANFO mixtures.

Where several users require relatively large amounts of ANFO, a central mixing plant is sometimes installed to meet this need. Mixing of AN and FO constitutes manufacture of an explosive. For this reason, a license or permit to mix ANFO must be obtained from the statutory authority.

The loose-poured density of ANFO is usually about 0.80 g/cm^3. Even when used in long vertical column in deep blastholes, ANFO is relatively incompressible, and its density varies very little, if at all, from the top to bottom of the charge. To obtain densities above the loose-poured value, ANFO must be physically compressed. In blastholes with diameters less than about 75 mm, the charge density can be increased to values as high as about 0.95 g/cm^3 by blow loading the ANFO through a suitable semi-conductive hose. If ANFO is compressed beyond a density of about 1.20 g/cm^3, it will not detonate as it becomes dead pressed.

The VOD and the energy yield increase with the degree of charge confinement.

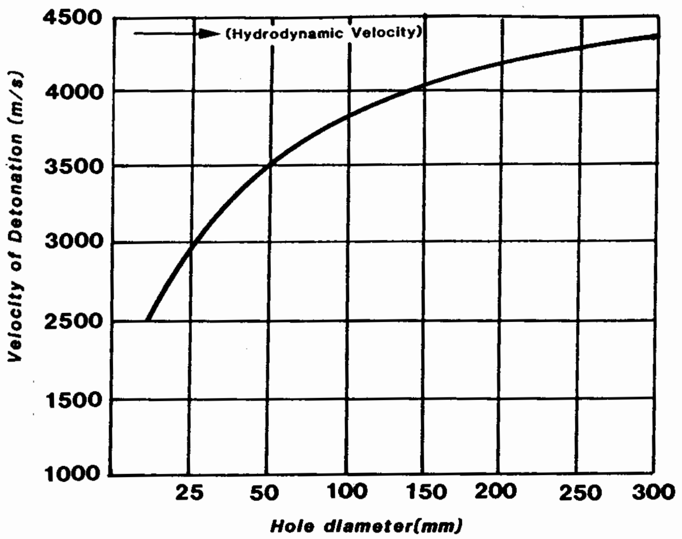


Figure 2.3 Detonation velocity of ANFO and the hole diameter.

VOD and energy release are smallest when the charge is fired completely unconfined (i.e. in the open air).

The blasthole diameter has a pronounced effect on the VOD of the ANFO (Fig. 2.3). As the diameter increases, so does the VOD. But the energy yield does not vary with blasthole diameter, because the initiation sensitivity of ANFO decreases with increasing blasthole diameter, a primer that is adequate for a 50 mm blasthole may be inadequate for a 125 mm blasthole. Well mixed loose-poured ANFO can be used successfully in blasthole diameters down to about 25 mm.

Although ANFO is used predominantly as loose poured charge, it is important to recognise that the available energy increases rapidly with coupling. Where the ANFO charge does not completely fill the cross-section of the blasthole, the charge confinement, the VOD and the peak pressure acting on the walls of the blasthole are reduced.

The sensitivity and VOD increase as the ANFO sizing becomes smaller (through either deliberate or uncontrolled breakdown of prills). When ANFO becomes very fine, a sensitive explosive may result. But it is difficult to take advantage of these improved explosive properties, because finer ANFO has a greater tendency to cake by absorbing moisture from the atmosphere. As the amount of fines in ANFO increases, the mix becomes more difficult to handle, it does not flow as freely, and when it is poured into small diameter blastholes, plugging can occur. An increase in the percentage of fines also causes a decrease in the loose-poured density of only about 0.60 g/cm^3.

Although ANFO mixtures are explosives, they are relatively insensitive and unless suitably primed, reliable detonation in large blastholes will not occur. The size and type of primer required depends upon blasthole diameter, degree of confinement, dryness of blastholes, etc. In general, the primer should have a high VOD and the maximum possible diameter. Needless to mention, the primer should be in intimate contact with the ANFO.

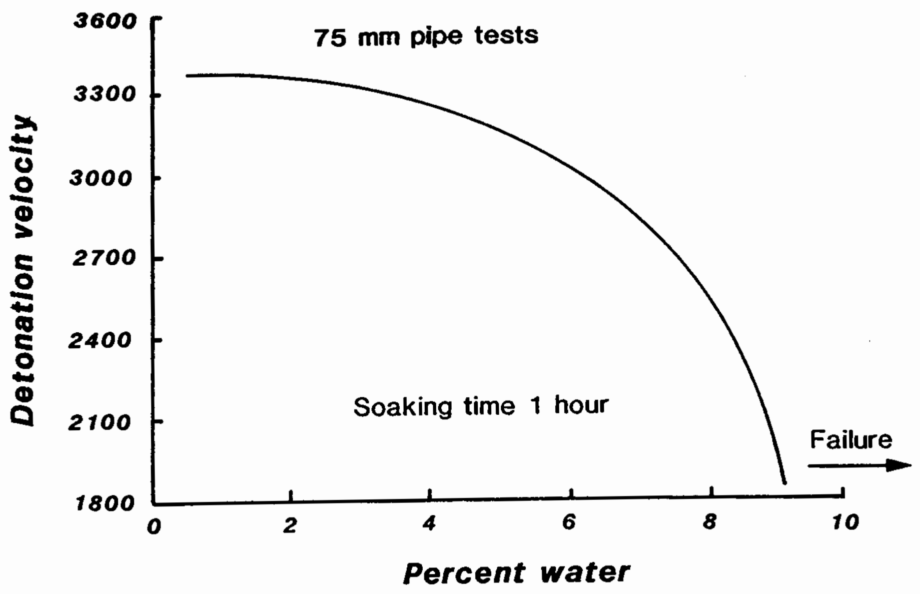


Figure 2.4 Influence of water on ANFO performance.

Lack of water resistance is the major limitation and disadvantage of ANFO. AN is readily dissolved by water, and, unfortunately the addition of 5.64% FO does little to reduce the rate of dissolution. Both the strength and VOD of ANFO are reduced by the addition of water (Fig. 2.4). ANFO which contains more than about 10% water usually fails to detonate. As one would expect, the longer ANFO is exposed to water, the greater is the damage, and the lower the blast effect. Wherever there is a chance of blasthole water desensitising ANFO and make it less effective, efforts should be made to minimise the period between charging and firing the blast.

The experience of thousands of the mine operators throughout the world provide ample warning that the blasting performance of the ANFO cannot be relied upon in the presence of water in blasthole. The efficient use of ANFO in wet blasthole depends almost entirely upon the protection given to it by polyethylene liners or by the packaging materials used to make up cartridges of ANFO. Dewatering of blastholes is also carried out (see Section 4.3). The use of Heavy ANFO has allowed operators to overcome this important shortcoming.

Most ANFO products can be purchased in four forms. In increasing order of cost they are as follows:

1. Site-mixing as bulk product obtained as separate ingredients;
2. Pre-mixed in bulk form for on-site storage or direct borehole delivery;
3. In paper, polyethylene or burlap packages;
4. Rigid cartridges (Fig. 2.5).

Rigid cartridges are the most expensive forms and eliminate the advantage of direct borehole coupling. These should be used only where borehole conditions dictate.

Figure 2.5 ANFO in bagged form.

Aluminised ANFO (ALANFO)
The addition of aluminium powder to ANFO increases the energy output quite considerably.

The effect of adding 9.9% aluminium to ANFO mixture containing 87.6% AN and 2.5% FO mixture is the following reaction:

$$6NH_4NO_3 + CH_2 + 2Al = Al_2O_3 + 6N_2 + 13H_2O + CO_2 + 1320 \text{ kcal/kg}$$

The higher heat of explosion is due to the high heat of formation of Al_2O_3. The increased density and increased energy per unit weight of an aluminised blasting agent must be weighed against its higher cost.

The use of aluminium with particle size in the -30 +150 mesh range ensures complete reaction of the aluminium without affecting either the performance sensitivity or hazard sensitivity significantly (Lansdale 1971; Thornely & Aldert 1981). The reactions of large aluminium particles (say +10 mesh) are quenched before the particles have burnt through completely. Very fine grades of aluminium rarely perform better than the -30 +150 mesh size range and for most mixing methods, increase the dust explosion hazard quite considerably. To overcome such hazards, and for reasons of increased composition flexibility, ALANFO is usually prepared and loaded by mix trucks.

The loose-poured density of ALANFO increases only slightly with an increase in aluminium content. The water resistance of ALANFO increases only slightly with an increase in aluminium content. The water resistance of ALANFO is not better than that of ANFO. The most commonly used compositions contain between 10% and 15% aluminium.

Other energetic fuels
In the past thought has been given to the use of energetic fuels such as nitromethane, though energy increases by the use of these materials but proportionately cost increases by about 10 times (Dick et al. 1983). Also there would be difficulty in holding all this nitromethane unless the AN is ultra fine. In some applications where the

smell of FO can be objectionable, other fuels such as glycol could be used. Fuels such as ferrosilicon FeSi, FeP and SiC though appear suitable but prolonged use has not found wide acceptance.

2.5 SLURRY EXPLOSIVES

Slurry explosives were first developed as a result of attempts to waterproof, improve density and strength of ammonium nitrate.

A slurry is a mixture of nitrates such as ammonium nitrate and sodium nitrate, a fuel sensitiser, either explosive or non-explosive, and varying amounts of water. Although they contain large amounts of ammonium nitrate, slurries are made water resistant through the use of gums, waxes, and cross linking agents. Most commonly used fuel sensitisers are carbonaceous fuels, aluminium, and amine nitrates. They are sensitised by air bubbles which are entrapped by churning the mixture. Even when none of the ingredients are in themselves explosive substances and it is only in the final stages of production that the compositions acquire explosive characteristics.

Slurry/watergel explosives are 'fuel' sensitised ammonium nitrate (AN) with or without other oxidisers such as sodium nitrate (SN) or sodium perchlorate (SPC) or ammonium perchlorate (APC) in which the solid fuel and the solid part of the oxide(s) are dispersed in continuous fluid medium generally an aqueous solution with or without other polar solvents and often more or less aerated. They have been designated differentially as slurry explosives (SE) when they are sensitised with explosives (e.g. TNT) and slurry blasting agent (SBA) when the fuel is not an explosive (e.g. aluminium, sulphur and sodium hydrocarbon). A further division of these water based explosives is – watergels and emulsion explosives. A watergel is essentially

Table 2.2. Some typical slurry compositions.

Ingredient	Weight %	Weight %
Large diameter TNT slurry		
AN	64	
TNT	20	
H_2O	15	
Gum + Crosslinker	1	
Basic Al NCN Slurry		
AN	77.0	68.8
H_2O	15.0	19.0
Gassing Agent	0.2	0.2
Gum + Crosslinker	2.0	2.0
Fuels	4.0	2.0
Al	1.0	8.0
Basic FO NCN Slurry		
AN	72.9	
H_2O	20.5	
FO	4.4	
Gassing Agent	0.2	
Gum + Dispersing Agent	0.2	

the same as a slurry and the two terms are frequently used interchangeably.

The watergel explosives have the oxygen donor in the form of nitrate salts fully or partly in a water solution which is gelled. In this gel, the fuel which is dissolved or distributed quite often consists of aluminium powder, methyl amine nitrate (MAN), glycol and urea, some of the fuel sensitises the mixture. Additional sensitising is achieved by the addition of small gas bubbles.

Slurries may be classified as either explosive or blasting agents. Those that are sensitive to a blasting cap are classified as explosives, even though they are less sensitive than NG-based explosives. It is important that slurries be stored in magazines appropriate to their classification.

Advantages of slurries over existing high explosives were quickly realised. Slurries could be manufactured without some of hazards usually present during explosives manufacture, and the user no longer suffers the discomfort of headache associated with nitro-glycerine based explosives. Except for their excellent water resistance and higher density and bulk strength slurries are similar in many ways to dry blasting agents. Good oxygen balance, decreased particle size and increased density, increased charge diameter, good confinement and coupling, and adequate priming all increase their efficiency. Although slurry blasting agents tend to loose sensitivity as their density increases, some explosive-based slurries function at density up to 1.6. The effect of charge diameter on the detonation of slurries is not pronounced as it is on ANFO.

Most non-capsensitive slurries depend on the entrapped air for their sensitivity and most capsensitive varieties are also dependent, to a lesser degree, on this entrapped air (Cook 1974). If this air is removed from a slurry through pressurisation

Figure 2.6 Cartridged slurry explosives (courtesy Karnataka Explosives).

from adjacent blast, prolonged periods of time in the borehole, or prolonged storage, the slurry may become desensitised. To overcome this loss entrapped air addition of 'perlite' or microballoons is several times resorted to.

Whereas slurries were originally developed with site mixing as the aim, factory produced cartridged slurries have advantage of being available in a wide range of packaging (Fig. 2.6). Cartridged slurries for use in small-diameter blastholes (50 mm or less) are normally made capsensitive. The sensitivity and performance of some grades of slurries are adversely affected by low temperatures. Slurries designed for use in large diameter blastholes (above 100 mm) are the least sensitive slurries. Slurries containing neither aluminium nor explosive sensitisers are the cheapest, but they are also least dense and powerful. In wet conditions where dewatering is not practical, and the rock is not extremely difficult to fragment, these low cost slurries offer competition to ANFO.

Aluminised slurries or those containing significant amounts of other high-energy sensitisers, develop sufficient energy for blasting in hard, dense rock.

Bulk slurries or watergels as in the case of packaged products consist of mixtures of combustible fuels and oxidisers dispersed with intimate mixtures of air to impart sensitivity. A typical formulation consists of (Gehrig 1982):

Ingredient	Weight %
Oxidisers	85
Fuels	5
Water	9
Others	1

Nitrate salts were first introduced in slurry explosives to improve the accessibility of oxidiser to fuel (Glynn 1984). However, due to the use of thickeners and gelling agents to stabilise the fluid phase for preventing segregation of solid ingredients and to increase water resistance, it is not possible to formulate a slurry in which all the oxidisers are in solution form. Hence, much of the oxidisers must be in solid form and this reduces the intimacy of contact between oxidiser and fuel and the rate of efficiency of reactions.

2.6 EMULSION EXPLOSIVES

An emulsion is a two phased system in which an inner or dispersed phase is distributed in an outer or continuous phase as shown in Figure 2.7. In simpler terms an emulsion is a mixture of two liquids that do not dissolve in one another. This unique feature coupled with the fact that minute size of the nitrate solution droplets are tightly compacted within the continuous fuel phase results in good intimacy between the oxidiser and fuel and increased reaction efficiency compared to other systems.

The emulsion matrix is obtained by emulsification of two immiscible liquids. By the process of emulsification two type of emulsions are obtained, one is oil-in-water and other is water-in-oil. Water is in discrete dispersed phase in water-in-oil emulsion in the form of fine droplets dispersed in continuous phase which is oil phase. Whereas in oil-in-water emulsion the reverse is true. A typical water-in-oil emulsion

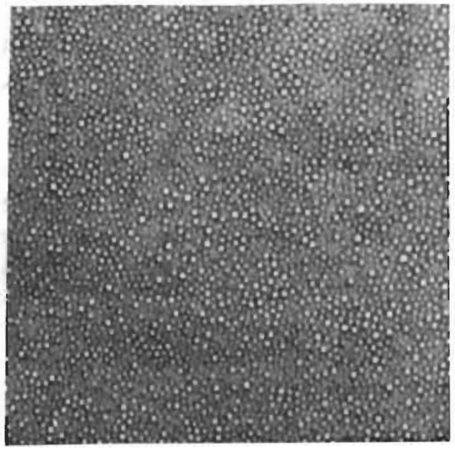


Figure 2.7 Structure of emulsion oxidiser surrounded by a thin coating of fuel.

blasting agent has the following composition (Gehrig 1982):

Ingredient	Weight %
Wax/oil	6
Emulsifier	2
Water	17
Ammonium nitrate	58
Sodium nitrate	15
Glass bubbles	2

A typical composition of oil-in-water emulsion explosive is as follows (Gehrig 1982):

Ingredient	Weight %
Diesel (fuel oil)	6
Water	15
Sodium perchlorate	5
Calcium nitrate	20
Ammonium nitrate	51
Glass bubbles	2
Guar	1

The emulsion explosives have the oxygen donor consisting of nitrates and perchlorates in an aqueous solution. The water phase is in the form of the small droplets in the continuous oil phase which constitutes the fuel. The fuel consist of mixtures of waxes and oils. The explosive in this case is sensitised by gas bubbles in the form of microspheres. Additional strength can be achieved by the addition of fuels such as aluminium powder.

Three products are available in emulsion category. These products are as given below:

1. Straight emulsions;
2. Doped emulsions;
3. Repumpables.

2.6.1 *Straight emulsions*

Basic formulation of an emulsion explosive can be represented as follows:

Ingredient	Weight %
Ammonium nitrate	60-70
Calcium nitrate/Sodium nitrate	0-20
Fuel oil	2- 6
Aluminium	1- 3
TNT & Water	Varies

Straight emulsions are normally hot mixed at temperature over 50°C and a suitable chemical gassing agent is incorporated to control density. Since the final product is straight emulsion, the explosive displays exceptionally high water resistance, stability and explosive reaction efficiency leading to superior velocity of detonation (VOD) characteristics.

2.6.2 *Doped emulsions*

This is a generic term for emulsion explosive to which varying percentage of ammonium nitrate are added to achieve a wide range of strengths. Pumpable characteristics as in the case of emulsions are maintained at all stages and so also the superior VOD and water resistance properties of emulsions.

2.6.3 *Repumpable emulsions*

These formulations are designed for small and intermediate diameter and apart from the normal advantages of emulsions are designed for pumpability at low temperatures. Products are available for a wide range of strengths and diameters (Hunsaker 1989).

These are low cost units for fast and accurate loading of small quantities per borehole. Like all bulk explosive all the components are non-explosive and are mixed on the mine bench and loaded directly into the borehole.

2.7 HEAVY ANFO

None of the above slurry and emulsion systems have cost competitiveness with ANFO as such their usage has been limited. The exception to this is densified or Heavy ANFO. The aim in Heavy ANFO is to have the advantages of both, the high density and water resistance of slurries and emulsions plus the low cost of ANFO (Poole 1987; Daubney 1988). In the mix of Ammonium Nitrate and Fuel Oil about 50% of air in total volume exists, roughly 30% in the prills and 70% between the

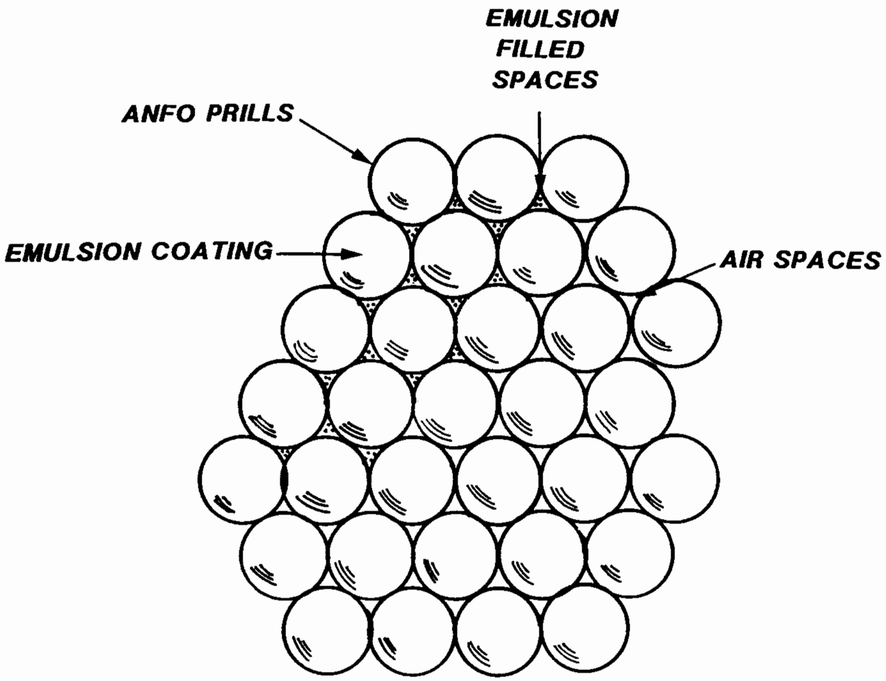


Figure 2.8 Heavy ANFO structure.

prills. The air in the prills provides the necessary sensitivity but the air between them is a waste volume (Fig. 2.8). This volume is filled with a high density, waterproof product such as emulsion (15-40%), the resultant mix is denser and more water resistant than ANFO and with very little added cost.

The water resistance and/or density of mix can be tailored by changing the emulsion level. In principle the emulsion occupies the interstices between prills of ANFO within the borehole. The denser product allows maximum use of borehole volume. Incremental amount of ammonium nitrate increases the density and bulk strength and a wide range of products can be obtained. However, water resistance and velocity of detonation (VOD) characteristics are inferior to emulsion products due to large percentage of ANFO but are superior to straight ANFO.

When plain ANFO is pumped into a blasthole, air spaces necessarily surround each prill. When emulsion is blended with ANFO in a 25/75 mix these air spaces are filled with emulsion matrix, to produce about 40% increase in explosive energy. The high percentage of emulsion maintains the energy and makes the mix more water resistant.

Blasting properties and economics of heavy ANFO depend upon the emulsion matrix level (Brulia 1985; Chironis 1985). Aluminium is also added to increase the strength of heavy ANFO.

Table 2.3 shows the effect of emulsion matrix level on physical properties. Typical properties of blasting agents are given in Table 2.4 and energy output of blasting agents is given in Table 2.5.

Table 2.3. Effect of emulsion matrix level on physical properties of heavy ANFO.

Emulsion matrix* (%)	Density g/cc	Relative bulk strength	Cohesiveness	Water resistance
0	0.84	100	None	None
10	0.93	100	None	None
20	1.04	125	Free flowing	Slight
30	1.15	125	Some	Fair
40	1.28	150	Good	Good
45	1.35	175	Good	V. Good
50	1.40	200	High	Excellent

*Water resistance increase with increased percentage of emulsion

Table 2.4. Properties of blasting agents.

Product	Density g/cc	REE WS % ANFO	REE BS % ANFO	VOD m/s	Water resistance
Bulk ANFO	0.80	100	100	4700	Nil
Bulk Watergel	1.80-1.20	80-100	110-135	4500/5000	Excellent
Doped Emulsion	1.15-1.25	100-120	135-170	5000/5500	Excellent
Heavy ANFO	1.00-1.30	100-120	135-170	4500/5500	Moderate

WS= weight strength, BS=bulk strength

Table 2.5. Energy output of blasting agents (in kilo calories per metre of borehole).

Hole dia -meter (mm)	Bulk ANFO	Packaged ANFO	Packaged 25/75 HANFO mix*	Bulk 50/50 HANFO mix	Bulk75/25 HANFO mix	Bulk 100/0 HANFO mix
165	1467	1136	1479	2055	1788	1668
224	2812	2114	2735	3939	3432	3233
311	5209	3758	4862	7295	6358	5991

* 25/75 mix is 25% ANFO, 75% emulsion.

2.8 LIQUID OXYGEN EXPLOSIVES

Explosive compositions utilising liquid oxygen have sometimes been employed. Oxygen gas liquefies at -183°C. A given volume of liquid, when gaseous, is equivalent to 840 times at NTP i.e. it has as much oxygen as would be available from 4000 times its volume of atmospheric air. If a combustible ingredient, made in the form of a cartridge, is soaked in liquid oxygen and then subjected to combustion reaction, the rate of reaction is extremely high and a large volume of gas is instantaneously released at high temperature and pressure. The velocity of detonation under suitable confinement can be more than 5000 m/s. This is the principle behind the use of liquid oxygen explosive (LOX). The fuel cartridges are dipped into the liquid oxygen immediately before being lowered into the borehole, and the cartridges are fired by the initiation system. A minimum delay after loading is imperative, since LOX rapidly loses its power through evaporation of oxygen. Confined detonation velocities of 3700-5800 m/s are attained.

Usage of LOX is confined to some countries only, and that too in limited areas.

LOX is available in two sizes of cartridges – small 25 mm to 90 mm and large cartridges of diameter over 100 mm. A LOX cartridge ready for blasting is prepared at the depot by soaking an absorbent cartridge in liquid oxygen. The basic ingredient of an absorbent cartridge is a cellulosic substance like crushed jute stalk or other agricultural product though other substances such as hydrocarbons or metallic powders, which are used to impart to the soaked cartridge the properties of industrial explosive.

LOX cartridges are inflammable and flow of gaseous oxygen emanating from a cartridge will cause smouldering material, glowing coals, and cigarette stubs to burst into flames. LOX should, therefore, be kept away from such burning and smouldering material.

Characteristics of LOX are not constant. It depends much on the time that has elapsed between removal from the soaking vessels and firing. The LOX cartridges should be used in the field without delay (within half an hour in the case of a large cartridge) to prevent loss of absorbed oxygen.

2.9 PRODUCTS FOR SPECIALISED BLASTING OPERATIONS

Blasting is carried out for many specialised requirements or the regular blasting operations need to be carried out in such a manner that it does not cause damage and performs specified skilled operation. Such example can be controlled blasting, underwater blasting, blasting of high temperature explosive. In general, either the available products are used or specialised products are formulated.

2.9.1 *Pipe charges*

Pipe charges in rigid plastic tubes are used which can be screwed together by means of extension sleeves. These are suitable for underground work. The cartridge dimensions range between 25 mm and 50 mm in diameter and 600 and 700 mm in length (Fig. 2.9).

While carrying out blasting which requires minimum damage to the remaining rock or the blasted material in which development of cracks needs to be reduced, then one way of achieving this is the use of pipe charges which are explosives having lesser diameter (11 to 19 mm) than the borehole (Fig. 2.9). The reduced diameter thus provides decoupling between the hole walls and the explosives and thus the peak pressure of the liberated gases reduces, causing reduced crushing and the intensity of cracks developed in the rock around the blasthole gets reduced (see Chapter 20).

Some examples of pipe charges are K-pipe of Finland, Gurit and Nabit of Nitro Nobel of Sweden, Gelatine Donorit 2E of Austria (Langefors & Kihlstrom 1963; Gustafsson, 1981). These types of explosives are used in the last rows of a blast while carrying out smooth blasting operation, or for blasting to obtain dimensional stone blocks.

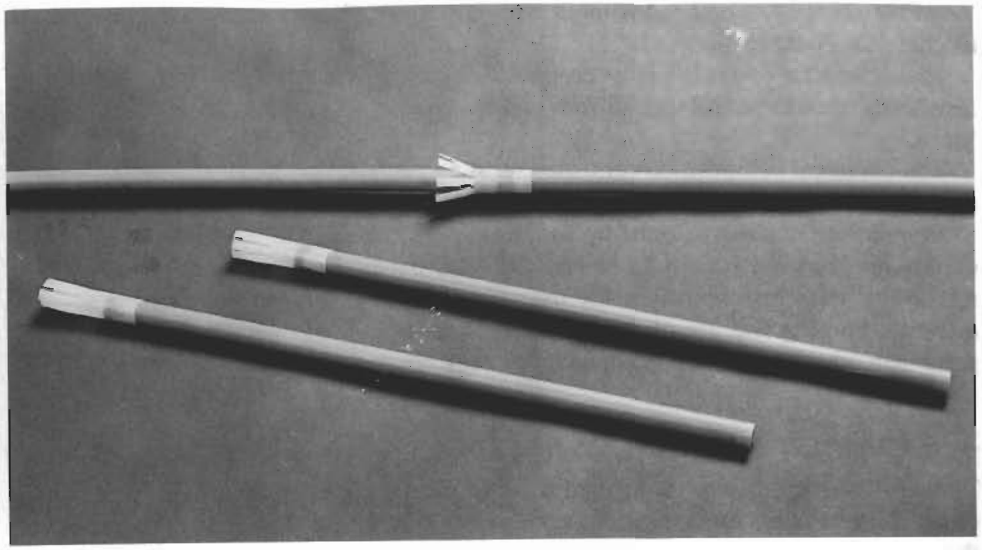


Figure 2.9 Pipe charges (courtesy Nitro Nobel).

2.9.2 *Powder explosives*

Comparatively weaker explosives are also used in many situations to blast very weak rocks. This can be done by using explosives with comparatively less energy. Polystyrene granules have been mixed with ANFO to obtain reduced energy explosives thus the crushing of rock and intensity and extent of cracking get reduced.

2.9.3 *Permissible explosives*

Blasting in coal mines is potentially more hazardous than in metal mines due to the presence of methane gas and coal dust, both of which are explosives themselves under certain conditions of concentration. These are susceptible to initiation from blasting charges in coal mines; therefore in these mines it is necessary to use those explosives which produce heat and flame of such a nature that it is incapable of igniting the gas or coal dust (Gregory 1984). Both nitro-glycerine-based permissible and slurry, watergel and emulsion permissible are available.

In coal mines only those explosives (called 'permitted explosives') which pass certain tests, are used. The safety of 'Permitted Explosive' depends primarily upon 1) low temperature, and 2) duration of flame produced, the flame lasting only 1/1000th second. A cooling agent like sodium chloride, potassium chloride, etc. is an essential constituent while the main constituents are NG (only in some explosives), ammonium nitrate in some of permitted explosives. Though types of tests to be passed depend from country to country, in general they must pass the following tests:

– The explosive when fired must not ignite a mixture of methane and ethane gases;

– A sequence of ten shots must be fired without causing ignition in bituminous coal dust; and

– Five shots must be fired without a mixture of methane and ethane gases as well as coal dust being ignited.

In addition the explosive must propagate completely in a series of tests, the air gap sensitivity should be at least 75 mm and noxious gases should be within specified limits.

The chemical composition furnished must agree, within tolerance, with that of the statutory authority.

Permissible explosives must be used in a permissible manner. Different types of explosives are used for cut faces and for solid blasting (see Chapter 14). The permissibility tests for explosives for usage in solid blasting are more stringent. The quantity, priming and use of delays is specified. Permitted explosives are less energetic than other explosives and have lower rock breaking capability.

2.9.4 *Shaped charges*

The term 'shaped-charge' is applied to cylindrical charges of high explosive with a cavity formed at the end opposite to the point of initiation (Fig. 2.10). The effect of the cavity is to produce an intensified pressure which is projected as a jet in the direction of initiation. If the cavity is lined by metal, then this is fragmented by the detonation to produce a high velocity jet.

Shaped charges have been applied to penetrate the casing of oil and gas well, to tap furnaces, to break rock underwater without carrying out drilling operations, and for penetration of frozen ground.

The type of explosive depends on the usage: RDX for metal cutting, slurry explosive or NG-based explosive which will not get desensitised by the high hydrostatic pressure when underwater blasting is carried out.

Figure 2.10 Shaped charges.

Figure 2.11 Cast boosters.

2.9.5 *Cast boosters*

Cast boosters are explosive units designed to act as primers, comprising a mixture of PETN, TNT and other minor ingredients (Fig. 2.11). Their high strength, high density and very high velocity of detonation (7000 m/s) make them suitable for priming ANFO mixtures and slurries. An additional advantage is their lower sensitivity to shock, friction and impact than gelignites and other NG explosives. Cast boosters are manufactured in the range of 100, 200 and 250 g and above. Holes provided in the boosters permit the insertion of electric detonators and the threading of boosters on to downlines of detonating cord.

2.9.6 *Seismic explosives*

For seismic prospecting, a special range of explosives are manufactured. These are compact, flexible and high powered explosives used in seismic reflection and refraction survey work. They are cap sensitive, highly water resistant as well as not affected by long sleeping times in water filled holes. In most cases they are couplable plastic cartridges.

CHAPTER 3

Properties of explosives

3.1 INTRODUCTION

Blasting performance of explosives are based on the ability of explosives to function properly in the specific conditions. Since many different types of explosives are available and explosives differ in many ways such as ability to resist water and water pressure, generation of fumes, input energy needed to start detonation, ability to fragment and displace, it is appropriate that knowledge about properties of explosives is obtained so that type and quantity of explosives used can be decided based on its ability to function under the specified conditions and achieve the objectives efficiently.

In this chapter important properties of explosives and some of the methods of measuring the above mentioned effects are described.

3.2 EXPLOSIVE PROPERTIES

Each explosive has certain specific characteristics or properties. Some of the principal properties of explosives which influence the ultimate choice are:

Detonation velocity; strength – energy; detonation pressure; density; safety in handling; storage qualities; water resistance; sensitivity; sensitiveness; medical aspects; inflammability; resistance to freezing; permissibility.

3.3 DETONATION VELOCITY

The detonation velocity is a measure, in meters per second or feet-per-second, of the speed at which the detonation wave travels through a column of explosives. Many factors affect the detonation velocity such as explosive type, diameter, confinement, temperature, and priming.

3.3.1 *Type*

The detonation velocities of today's commercial explosives range from about 1500 m/s e.g. ANFO in small diameter holes and certain permissibles to more than 6700 m/s e.g. detonating cords and cast-primers. Velocity of detonation (VOD) for com-

mon explosives fall within the range of 3000 to 5000 m/s. Every explosive has an ultimate or ideal velocity known as hydrodynamic velocity, which is the steady-state velocity of the explosive. As a general rule, the higher the velocity the greater the shattering effect; thus when harder rock is to be blasted, a higher velocity explosive should be chosen. It is to be noted that velocity is not the solitary property which influences the power of an explosive.

3.3.2 *Diameter*

Depending upon the type of explosive up to a certain diameter the velocity of detonation of an explosive is influenced by the diameter. In general, the larger the diameter the higher the velocity until the steady state velocity of the explosive is reached.

Every explosive also has a 'critical diameter' which is the minimum diameter at which the detonation process, once initiated, will support itself in the column. In diameters smaller than the 'critical' the detonation of the explosive will not be supported and will be extinguished. The detonation process once initiated will support itself for greater than the critical diameters and the critical diameter in confinement is usually smaller than that for the same explosive unconfined. Figure 3.1 shows some typical explosive products and how diameters affect velocity.

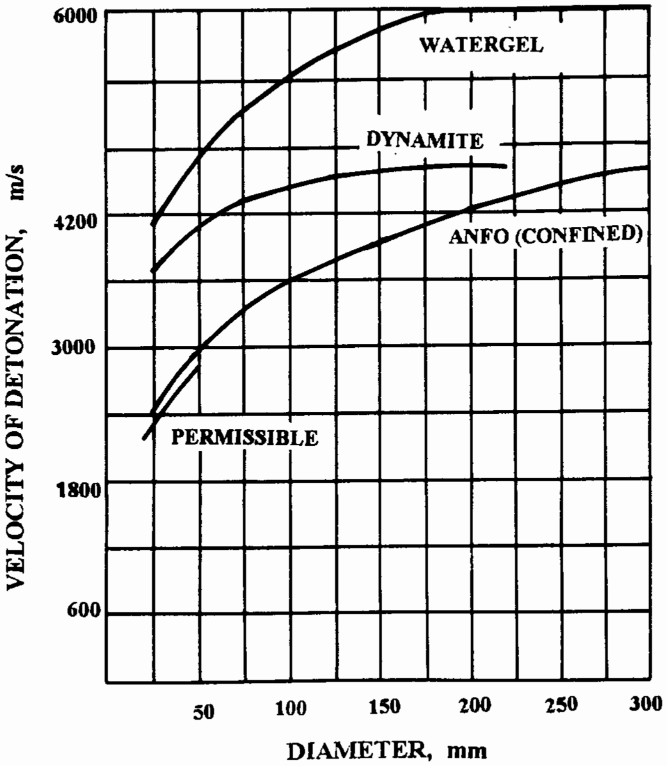


Figure 3.1 Effect of explosive diameter on velocity of detonation.

3.3.3 *Degree of confinement*

Generally, the greater the confinement of an explosive, the higher the detonation velocity. This is particularly true for products such as ANFO and some watergels in small diameter boreholes. Depending on the type, the degree of confinement has less effect on the velocity as the diameter of the explosive increases. In a compressible medium as the expanding gases compress the material such as air water or soft-porous rocks, the energy is rapidly lost and the pressure and temperature drop sharply in the reaction products. These losses reach the reaction zone as a refraction wave, i.e. low pressure area that removes support from the detonation front. This results in a detonation velocity lower than the hydrodynamic or ideal velocity. If the diameter is small enough, the detonation may ultimately decay and fail. With the increase in the diameter of the explosive, however, the degree of confinement gets less and less effective.

If the confining burden is relatively incompressible (hard massive rock), the refraction wave is weaker and a larger primary reaction zone at high pressure and temperature supports the shock front. The minimum diameter for stable detonation will be smaller under such confinement.

3.3.4 *Temperature*

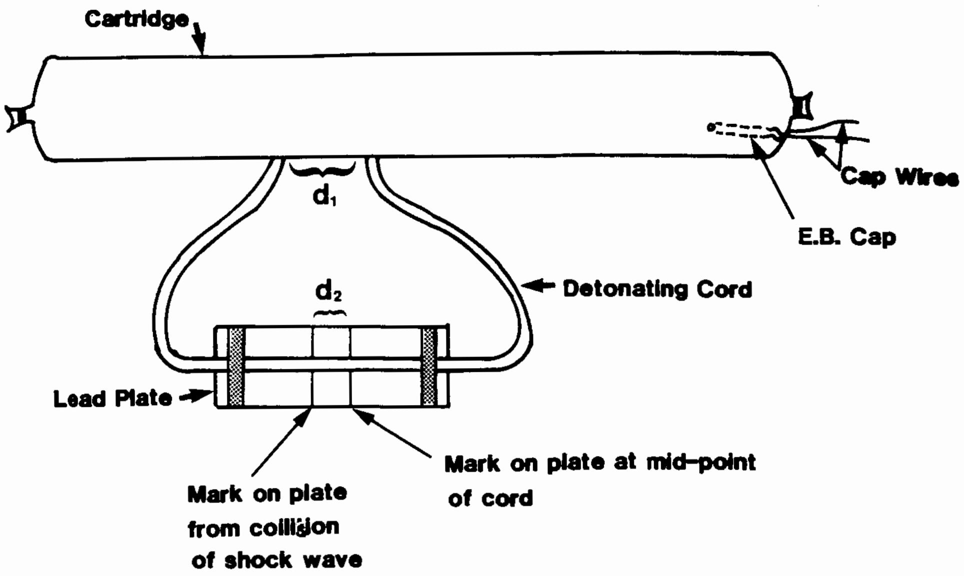
Depending on the type of explosive, changes in its temperature affect the velocity of the detonation. A decrease in temperature will decrease the sensitivity of any explosive. Typically the explosives that are solids at normal temperatures and contain little or no liquid are relatively unaffected at the normal low temperatures experienced in commercial blasting e.g. ANFO. The velocity of explosives which contain liquid in some quantity, such as watergels, is more affected by temperature, although, formulations are available to minimise this effect in practical applications.

3.3.5 *Priming*

It is essential that adequate priming is ensured so that the explosive may reach its maximum velocity as quickly as possible. Inadequate priming can result in the failure of the explosive to detonate, a slow build-up to final velocity, or a low velocity detonation (which amounts to deflagration). An example of the later is deflagration of ANFO by a detonating cord in small diameter holes.

3.3.6 *Methods of measuring detonation velocity*

Velocity determinations are made by measuring the time required for the detonation wave to travel a measured distance longitudinally through a column of the explosive. There are a number of methods available for measuring detonation velocity. Often figures given by the manufacturers are in unconfined conditions. Measurements can be made for an explosive confined in a borehole and even underwater at high pressure.



$$\text{Approx. velocity of the explosive (V) in m/s} = \frac{\text{(Vel. of det. cord(m/s))} \times d\ \text{(mm)}}{2d_2\text{(mm)}}$$

Figure 3.2 Dautriche's method of velocity of detonation determination.

The Dautriche method
When electronic equipment is not available, the velocity can be approximated by the Dautriche method (Fig. 3.2). In this method continuous column, one cartridge or several cartridges are joined firmly together at least 0.4 m long, is punched with two holes. The first hole should be at least 120 mm from the end to be primed, and for the second, a measured distance – 150 mm or more is preferred – along the length of the cartridge from the first. The ends of 1 m length of 10 g/m cord are inserted into the product through the punched holes, and the exact middle of the detonating cord is placed and marked on a lead plate usually about 250 mm in length. When the explosive is initiated, its detonation wave initiates the end of the detonating cord of the first hole near the primed end, and then the other end of the detonating cord in the second hole. The collision of two detonation waves in the detonating cord distinctly marks the lead plate. The approximate explosive velocity can be calculated from the detonating cord velocity by comparing the distance between the marked middle and the mark from the shock wave collision.

The standard velocity test
In order to measure the detonation velocity over a reasonable short column of explosive, it is necessary to measure a very short time interval with high accuracy. Counter-chronograph capable of measuring time intervals with an accuracy of 1/10 of one millionth of a second (0.1 microsecond) is used in this test.

'Velocity targets' (these look like electric blasting caps but have only two separated wires held in a copper shell by a rubber plug), are inserted into the explosive

Figure 3.3 Data Trap for continuous velocity of detonation monitoring (courtesey MREL).

cartridge at a predetermined spacing. The detonation wave from the explosive crushes the shell, closes the circuit by smashing the separated wires together. The first target starts the chronograph and the second stops it. The distance between the targets is divided by the elapsed time between the crushing of the targets to give the velocity of detonation.

Continuous velocity test

The test is carried out with a sensing probe made of nylon-covered nichrome wire with a known resistance per meter which is placed along the length of the explosive column to be measured. As the detonation proceeds along the length of explosive column, the resistance of the wire decreases with the decrease in wire length because of the ionisation at the shock front, which effectively short circuits the two wires. This rate of change in the resistance results in change of voltage that is monitored on the oscilloscope and from it the continuous velocity of explosive can be measured.

Digital velocity measurement instruments have become popular and allow determination of in-the-hole velocity measurements. Data Trap (Fig. 3.3) is one such example. This multi purpose data recorder monitors through its VOD module a continuous probe of accurate resistance per unit length, placed axially in the explosive.

3.4 ENERGY-STRENGTH

A large number of tests and various calculations have been made which refer to energy content to predict the performance of explosives. However, the term 'strength', traditionally associated with the 'strength markings' of different dynamite grades, has little correlation with the effectiveness of an explosive in blasting and has no meaningful relation to modern commercial products like ANFO, emulsions or watergels.

The factors which predict more accurately the ability of an explosive to fragment and move the material efficiently have been recently re-examined. These include such calculated properties as theoretical energy (Q) and expansion work (EWK), as well as the measured properties such as underwater shock and bubble energy, shock wave impulse, and stress wave measurements in the material to be blasted itself.

Even these new tests and calculations, when considered independently of one another, do not predict an explosive's effectiveness in all cases. To date, no single test

or calculation can predict the blasting action of a commercial explosive, principally because of the complex nature of the materials being blasted.

3.4.1 *Explosive strength ratings*

The nitro-glycerine or straight dynamites are rated or graded according to percentage by weight of nitro-glycerine they contain. A 60% straight dynamite contains 60% nitro-glycerine, the 30% grade strength contains 30% nitro-glycerine, etc. An erroneous concept is that the actual blasting power developed by the different grades is in direct proportion to the strength ratings, for example, an explosive marked 60% is twice as strong as one marked 30%. Such simple ratios unfortunately do not exist because nitro-glycerine is not the only energy-producing ingredient in their formulation.

New method for measuring strength became necessary to compare an explosive's ability to do work with a straight dynamite's ability to do work.

The 'work' test chosen was with the ballistic mortar test, (Fig. 3.4) in this test a 10 g of test explosive was loaded in a small cannon suspended on a pendulum and was detonated. The swing of the pendulum was compared to that produced by 10 g of the 'standard' straight dynamite or blasting gelatine. A swing equal to that of the standard meant that the test explosive was given the same percent rating as the standard explosive. This method of measuring 'work' ignored many variables of explosive properties and rock conditions, and did not correlate well with the actual rock blasting results

In lead block test, a small amount of explosive is detonated in a hole in a lead cylinder. The volume of the cavity produced by the detonation gives an indication of the blasting effect.

Since most explosives are different in density from straight dynamite used as the standard, a similar test was used to compare equal volumes of test explosive and straight dynamite. Thus explosives were rated with weight strength and/or volume strength.

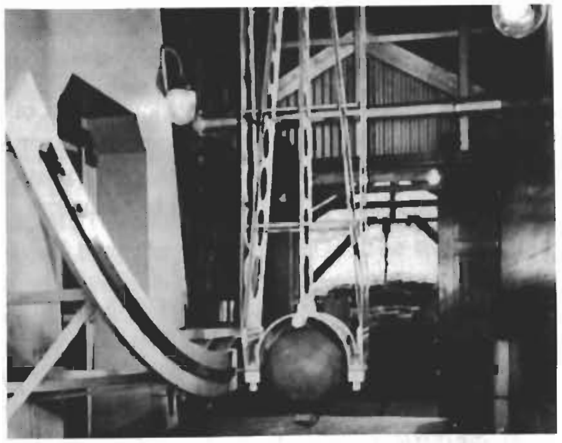


Figure 3.4 Ballistic mortar.

With the introduction of ammonium nitrate prilled blasting agents and watergels new methods of calculating and testing performance became necessary because these blasting materials do not react fully in the small quantities used in the ballistic mortar test.

Two types of explosives having the same measured or calculated energy rating do not necessarily produce the same blasting action when used under field conditions. Also, relating product safety to strength markings is not correct.

3.4.2 *Explosive energy*

The performance of an explosive is not determined simply by knowing the total energy released by the explosive. It depends also upon the rate of energy release and how effectively the energy is utilised in fragmenting and moving the material being blasted. Both the explosive properties and the properties of the material being blasted influence the effectiveness of an explosive.

Some of the current tests or calculations to measure and characterise an explosive energy are the various underwater tests, the pressure time measuring technique in the rock, and the theoretical calculation techniques.

Theoretical energy

The calculated or theoretical energy of an explosive is the difference between the heat of formation of the products of the explosion and the heat of formation of the ingredients of the explosive. This energy, known as the heat of explosion, 'Q', represents the total thermal energy and includes the heat retained by products of detonation at atmospheric pressure. When the energy of the detonation products is examined at different temperature and pressure states of expansion, ending with gas expansion to atmospheric pressure, another measure of energy evolves. This concept, defined as the expansion work (EWK) is made possible by a computer model based on the change of energy for different equilibrium states in an isentropic, adiabatic expansion. EWK is one of the most realistic measures of explosive power as it approximates the amount of work the gaseous products of the explosive can accomplish as they expand from the initial detonation conditions to atmospheric condition. The EWK consists of the change in internal energy and utilises such parameters as gas volume, pressure, and temperature in its calculations.

Although the calculated results for Q and EWK are similar for many explosives, the expansion work is lower for compositions which yield significant quantities of solid, high-temperature products at atmospheric pressure. In some cases, however, EWK may be higher than Q because of the change in reaction products with expansion. Energy tied up in the expanded detonation products is not a useful blast energy.

Underwater test method

Underwater tests are used as a means of comparing the relative effectiveness of various explosives. This is based on the hypothesis that 'shock energy' from an explosion underwater measures the explosive's shattering action in other material, such as rock, and that 'bubble energy' from the underwater explosion is the 'heaving action' of the explosive. The shock energy in the tests is the compressional energy radiated from an underwater detonation and is derived by measuring the area under the pressure squared-time curve at a known distance from the explosion. The bubble energy is the potential energy of the displaced water at the maximum size of the bubble. It is derived by measuring the elapsed time of first collapse of the gas bubble, knowing the ambient hydrostatic and atmospheric pressure energies, underwater tests can also measure the shock wave impulse, another indicator of an explosive's strength. The shock wave impulse is derived by measuring the area under pressure time curve for a selected integration time interval at a known distance from the explosion.

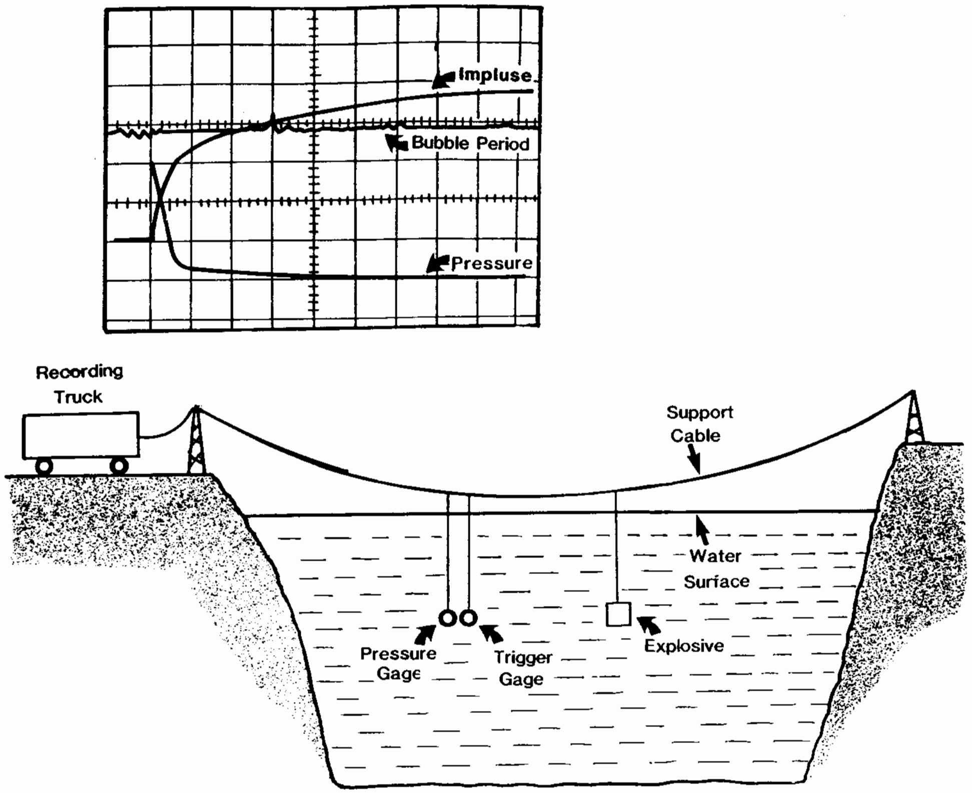


Figure 3.5 Underwater test configuration.

A schematic diagram Figure 3.5 shows a typical underwater test configuration and oscilloscope record used to determine shock wave impulse. The pressure v/s, time is displayed both at fast and slow scope speeds and impulse v/s, time at the faster scope speed. The impulse v/s, time is electronically integrated from the pressure v/s, time signal from the pressure gauge. The slower scope speed record is used to measure the time interval between the shock wave and the first bubble pulse. The integration time of the shock wave impulse is taken as a fixed percentage of the bubble period.

Underwater tests have been found to be useful tool in evaluating relative strengths of various explosives provided that these tests are carefully interpreted in conjunction with theoretical calculations and field performance. Experience has shown that bubble energy values frequently overrate an explosive's ability to fragment hard rock, but are more closely related to the capability of displacing weak rock.

A crater test amongst field tests was considered for indicating field strength ratings. However, this has not proved very successful- because of a number of shortcomings such as difficulties in calculating volume/quantity of rock broken, difficulty in getting similar test rock.

3.5 DENSITY

The density of explosives is expressed in g/cm^3. Densities of commonly used explosives are:

ANFO free poured	0.80 g/cm^3
ANFO charged pneumatically from pressure vessel	up to 0.95 g/cm^3
ANFO charged pneumatically with a ventury type charger	up to 1.10 g/cm^3
Watergels and emulsions	0.80 to 1.50 g/cm^3
Semigelatine and gelatine	1.00 to 1.60 g/cm^3
Cast boosters	1.60 g/cm^3

In general commercial watergels, emulsions and nitro-glycerine based explosives commonly used are in the range of 1.10 to 1.35 g/cm^3. Densities of free flowing blasting agents can vary widely upon compaction achieved in use, the amount of compaction affecting both the sensitivity and performance of these products, some have a density of 0.8 g/cm^3 and can achieve a density of 1.30 g/cm^3 when bulk-loaded with pneumatic loading equipment. While using pneumatic cartridge loader, density up to 1.6 g/cm^3 can be achieved.

The prime purpose in varying the density of commercial explosives is to enable the total energy charge in a borehole to meet particular field conditions. In many cases, such as in mining hard ore and driving tunnels through hard rock it is necessary to use dense gelatines or high density blasting agents in order to break the burden. In other instances, as in production of certain ore or stone where a high percentage of lump is desired, the charges are distributed in the borehole with low density gelatines or blasting agents. In still others, as in quarrying, a high density explosive is sometimes used in the bottom of hole to ensure pulling the toe, while a bulkier one is used to obtain proper distribution of the charge.

A useful guide in designing a blast is to know approximately how many kilograms of explosive can be loaded in one meter of borehole. Because the density of water is 1.0 g/cm^3, products loaded into holes containing water must have a density greater than 1.0 g/cm^3 in order to sink.

Critical density

An explosive sensitivity can be reduced or destroyed by too much increase in density. If the density becomes too high and the 'critical density' is exceeded then even a good primer may not detonate the blasting material. In field use this phenomena is illustrated when ANFO is 'dead pressed'. This usually occurs when the stress wave from a previous detonation compacts ANFO in an adjacent borehole beyond its

'critical density' prior to its initiation by the primer. High hydrostatic pressures also can desensitise an explosive by increasing its density. For example, the water-filled, deep boreholes used in seismic work require specially formulated products.

In underground mines usually hole diameters are close specially for drive blasting. When pattern like burn cut is used and explosive type is water based then dead pressing or precompression may occur and result in failure of some of the holes in a round. This causes not only loss of production but also creates safety problems. In newer products pre-compression resistance has been substantially improved.

When using product densities to calculate borehole loads, it is to be remembered that variations in borehole size from worn bits, and soft strata of rock and variations in product compaction (tamping and slumping) mean that calculated loads are only good approximations.

3.6 DETONATION PRESSURE AND EXPLOSION PRESSURE

The detonation pressure, usually measured in kilobars, is generally considered as the pressure in the shock zone ahead of the reaction zone. When an explosive detonates, this tremendous pressure is released, practically instantaneously, in a shock wave which exists for only a fraction of second at any given place. The sudden pressure thus created will shatter rather than displace objects and is generally accepted as providing to an explosive an ability called brisance. This brisance or shattering effect is dependent upon the suddenness with which the gaseous products of an explosive are liberated. The detonation pressure is a function of the density, the detonation velocity, and the particle velocity of the explosive. For condensed explosives, the particle velocity is about ¼ of the detonation velocity. The detonation pressure can be approximated as follows:

$$P = 2.5\,\rho.V^2 \times 10^{-6}$$

where P = Detonation pressure (kilobars), ρ = Density (g/cm^3), V = Velocity (m/s).

This pressure is important in that it is related to the stress level in the material to be blasted which may be an important factor in fragmentation. It is also important in priming for effective and reliable initiation that the detonation pressure of the primer exceeds the detonation pressure of the main explosive charge.

The detonation pressure is different from the explosion pressure, which is the pressure after adiabatic expansion back to the original explosive volume. The explosion pressure is theoretically about 45% of the detonation pressure. The effective explosion pressure in blasting depends on how well the explosive fills the holes.

3.7 WATER RESISTANCE

The ability of an explosive to withstand water penetration, termed as water resistance, vary widely for the various explosives. In many blasting situations explosives have to remain under water for long periods of time. Even in case of common rock blasting operations the drill holes are full of water. Water resistance is generally expressed as the number of hours a product may be submerged in static water and still be detonated reliably. Explosive products penetrated by water have their efficiency impaired first and, upon prolonged exposure or in severe water conditions, they may be desensitised to a point where they will not detonate.

Commercial explosives vary widely in their ability to resist the effect of water penetration. Ammonium nitrate/fuel oil have no inherent water resistance. If ANFO is poured into water filled drill holes, it will quickly desensitise. Packaged ANFO products, if used in wet work, depend entirely on their packaging to resist water penetration. Polyethylene tubes or plastic bags are used for packaging but such ANFO or other powder explosives offer very little water resistance when loaded into water-filled boreholes.

Also, low density products can easily become separated by floating in dirty water or mud in the borehole. Frequent priming, often every two cartridges, satisfactorily overcomes substandard performance for this problem.

Watergels (slurries and emulsions) and many nitro-glycerine based products have water resistance. The straight dynamites and straight gelatines are the most water proof, especially in the larger cartridge diameter and in higher strengths. Some of the ammonia gelatines and high density ammonia dynamites have good water resistance but generally speaking the bulkier ammonia dynamites, including 'permitteds' of this type, have little or no water resistance, particularly if the wrappers are torn in loading operations. Watergels, and emulsions, of-course are based on water proofing ANFO. Despite inherent water resistance, some severe field conditions can desensitise these products. In general, in severe water conditions it is recommended that the products should not be removed from the packaging and should not be cut into small sections. The packaging does improve the water resistance of the product.

The water resistance of a product depends not only on the packaging and the inherent ability of explosive to withstand water, but also on the water conditions. Static water at low pressure will not affect the explosive as quickly as dynamic fast-moving water specially at high pressure. Consequently, results of water-resistance tests must be considered in relation to the particular conditions of each blasting operation.

3.8 RESISTANCE TO FREEZING

Where the temperature fall below freezing point it is important that explosives are resistant to freezing. Normally the explosives become stiffer at low temperatures and some explosives will then also be less sensitive to initiation. Gelatines tend to stiffen and may become firmer after prolonged exposure to low temperatures. Those containing appreciable quantities of ammonium nitrate may become set as a result of picking up moisture during storage and temperature changes. It is sometimes desirable, therefore, to place explosives which have been in storage at very low temperatures, in a warm room for several hours before use since this will soften them and make them easier to prime, as well as to tamp in the boreholes.

ANFO and slurry explosives become progressively stiffer and less pliable as the temperature falls. Under severe conditions slurries become hard and less sensitive to normal levels of initiation.

3.9 SENSITIVITY

Sensitivity is the measure of ease of initiation. There are numerous measures of sensitivity, including cap sensitivity, drop tests, friction test and others. These tests are often carried out to measure an explosive's ease of initiation through accidental means and thereby measure safety in handling.

Cap sensitivity. One of the most frequently referred tests is the cap sensitivity test. This not only characterises an explosive's ease of initiation with a blasting cap, but also is used to classify products. Either a No. 6 strength or No. 8 strength detonator is used as standard by the explosive industry.

To compare the sensitivity between different products various strength caps are used in the same manner. Some explosives are sensitive to caps with very small base charges, while others are only sensitive to a booster initiated by a cap.

Normally the explosive is initiated by the use of a detonator but some explosives need more powerful initiation. As example it can be mentioned of ANFO explosives and some slurry explosives which are normally initiated by using some cap sensitive explosive primers or special primer consisting of an explosive with a high detonation velocity.

The blasting procedure depends to a great extent on a satisfactory initiation and maintenance of stable detonation.

3.10 SAFETY IN HANDLING

Safety in handling is very important and one obvious requirement of an explosive is that it can be transported, stored and used under normal conditions without any risk to people carrying out blasting operation. Explosives are subjected to many tests before they are approved.

Drop hammer test. In a standard test for dynamite a specified weight is dropped from varying heights until a drop distance is reached at which the product, usually a thin section on a metal plate detonated 50% of time. With many slurry and emulsion explosives and ANFO such method do not show any detonation within standard test limits.

The friction tests. In this test friction under increased pressure is applied upon a small amount of explosive. When a reaction in the explosive is obtained the pressure is noted.

The riffle bullet test. A determination is made of the bullet velocity needed in order to create a detonation of the explosive.

Heat test. A determination is made of how much heat an explosive can withstand before a reaction starts and what happens when an explosive starts to decompose with heat.

Despite lower sensitivity and greater safety with newer explosives like slurries, emulsions and also ANFO, nevertheless they are used as explosives. They may be initiated under more severe impacts or shocks. Explosives, regardless of their degree of safety, should never be abused in any way.

3.11 STORAGE QUALITIES

Most explosives are perishable, and both climate and magazine conditions are factors of great importance in their storage. Some types have better storage qualities and, as a general rule, those which do not contain ammonium nitrate store the best. Thus straight gelatine is superior to ammonia gelatine and straight dynamite is superior to ammonia dynamite.

Explosive such as slurries undergo a normal ageing process during storage since air bubbles entrapped during manufacture disappear partly or wholly. This implies that there may be some disruptions in the detonation stability. Such explosives should not be stored in or subjected to high temperatures since they can soften and the salts in the explosive substance can penetrate through the paper round the cartridges. The cartridges then become deformed and difficult to use.

Powder type explosives in the cartridge form are usually more sensitive to moisture in the storage environment. In the case of humid atmosphere, the salts in the explosive can form deposits on the cartridge which then harden. The ageing phenomenon, however, does not affect the powder explosives in the same way as other explosives.

3.12 SENSITIVENESS

Sensitiveness of an explosive is a measure of its propagating ability. For NG-based explosives and some slurries it is the distance in inches or centimetres over which one half of 25 mm × 200 mm cartridge would propagate to another one-half of a 25 mm × 200 mm cartridge when both halves with the cut-ends facing were enclosed in a paper tube and shot unconfined. This is termed as 'gap sensitivity'.

This is an important consideration since in case of low sensitiveness, there can easily be interruptions in the detonation if the column of explosive in the charged drill hole is not continuous or if something is obstructing various units. Sensitiveness or flashover tendency decreases considerably at low temperatures.

An explosive with high flash-over tendency can cause flash-over between adjacent drill holes if the holes are closely spaced. Particularly in the case of rock types which have many cracks, and under moist conditions, there is a risk of flashover. Under most conditions it is important that the individual charges do not propagate, but rather detonate independently with a predetermined delay interval as in tunnelling, shaft sinking, trenching, etc.

3.13 INFLAMMABILITY

This is a measure of the ease with which an explosive or blasting agent can be ignited. Gelignites ignite readily and burn violently. This burning can transform into a detonation. Watergels are more difficult to ignite than dynamites and in many cases an outside source of flame must be applied continuously. After most of water is evaporated by this outside flame source, however, watergels will support combustion. Both ammonium nitrate products and watergels have a lower tendency to convert the burning to a detonation.

3.14 MEDICAL ASPECTS

More and more consideration is given to medical aspects. Today it is desired that toxic gases are minimised and other effects such as headache is rendered least when handling explosive containing nitro-glycerine.

For blasting in quarries and open conditions the toxic gases seldom create problems but for underground operations it is essential that toxic gases such as: carbon monoxide, oxides of nitrogen, ammonia and fumes of NG, are kept at an acceptable level. Headache from NG-explosive is a side effect that can cause a lot of inconvenience for many people since it can be rather difficult to prevent NG-fumes entering the blood circulatory system either via the respiratory organ or by direct contact with skin and cause lowering of the blood pressure. Slurries, emulsions, ANFO type explosive have an advantage from medical point of view since they do not cause headache. These explosives have, however, been reported to cause skin irritation and in some cases even eczema. Some slurries based on organic sensitiser give bad smell.

3.15 FUMES

The gases resulting from the detonation of commercial explosives are principally carbon dioxide, nitrogen and water vapour, all are non-toxic in the ordinary sense. However, poisonous or toxic gases including carbon monoxide and nitrogen oxides, also result from any detonation. In the explosives industry these toxic gases are called fumes. Fumes are different than smoke, which is composed mainly of steam and the solid products of combustion. Although smoke is non-toxic, excessive exposure to smoke, especially that produced by dynamite, can cause severe headache and should be avoided. Both the nature and the total quantity of poisonous gases and smoke vary according to conditions of use. Regardless of the type used, some toxic gases will be produced on detonation since, unfortunately, there is no such thing as a 'fumeless' explosive.

The composition of an explosive is said to be balanced when the oxygen contained by the ingredients combines with the carbon and hydrogen content to form carbon dioxide and water. If there is insufficient oxygen (a negative oxygen balance), the tendency to form carbon monoxide is increased. If there is an excess of oxygen (a positive oxygen balance), oxides of nitrogen are formed. The weight of paper and wax per cartridge affects oxygen balance and this must be considered in the calculations. Because oxygen has such an important effect on the types of gases evolved, it is closely controlled in formulation. Oxygen balance is kept within specific limits to give the lowest practical content of toxic gases.

In open blasting operation fumes cause little concern if they can be quickly dispersed by air movement, but in underground work the type and amount of explosive, the blast conditions, ventilation and other factors must be considered. Where fumes can be problem, properly formulated and manufactured explosives and blasting agents will give minimum quantities of toxic gases. However, condition of use can drastically shift the type of gases produced.

Some factors that increase fumes are: poor product formulations, inadequate priming, insufficient water resistance, lack of confinement, reactivity of the product with the rock or other material being blasted. An explosive that has acquired an excessive moisture content because of exposure to unfavourable storage conditions or to water in borehole will produce a greater percentage of toxic fumes than the same weight of explosive with normal moisture content detonated in a dry hole. A burning explosive gives off a high percentage of toxic fumes.

The better the confinement under which an explosive is detonated, the more complete the reaction and the better fumes. An adequate quantity of incombustible stemming in the borehole prevents the release of gases and permits them to do useful work on burden. An excessive charge that expends most of its energy in making noise instead of being confined behind the burden will produce excessive fumes.

The ventilation of a mine is something over which the manufacturer of explosive exercises no control. Inadequate ventilation is frequently the true cause of complaints on the fumes from the explosives.

Control over the fumes produced by an explosive depends not only on the type of explosive and the care exercised during manufacture, but also on conditions of storage and use. Since all explosives produce some gases while large volumes are dangerous to health, it is not advisable to rush into a mine or tunnel until the gasses have had ample time to dissipate.

There are two prevailing concepts of expressing safe concentrations of toxic substances: 1) Maximum Acceptable Concentrations (MAC) and 2) Threshold Limit Values (TLV).

The Maximum Acceptable Concentrations are peak concentrations, not average and should not be exceeded during a normal work day. Any variation should be below the recommended level.

The Threshold Limit Values are expressed in parts of vapour or gas per million parts of air by volume at 25°C and 760 mm pressure. Most of the averaged TLV values may fluctuate a reasonable amount above the TLV providing an equivalent fluctuation below the limit occurs.

Maximum Allowable Concentrations acceptable in general are:
- Carbon Monoxide, 50 ppm (parts per million);
- Nitrogen Dioxide, 5 ppm.

These are peak values.

After the blast the ventilation system should be inspected for damages, since maximum protection against toxic fumes may be obtained by allowing sufficient time for ventilation to disperse the fumes before returning to the scene of a blast. If ventilation is considered inadequate, necessary steps need to be taken for improving the same.

Classification. In USA there are two different fume classifications for explosives. The type of classification is based on whether the explosive is a permissible grade or a non-permissible grade. No explosive is approved as permissible if it generates more than 71 l of toxic gases per 456 g (2.5 cu. ft of toxic gases per pound). Permissible grades of explosive do not carry fume markings. US Bureau of Mines gives classification for permissibles as below:
- Class A: 0 to 53 l (0 to 1.87 cu. ft) noxious gases/681 g (1.5 lb) explosive;
- Class B: 53 to 106 l (1.87 to 3.74 cu. ft) noxious gases/681 g (1.5 lb) explosive.

All non-permissible grades are classified according to the Institute of Makers of Explosives (IME) classification. Fume classification standard based upon the volume of poisonous gases emitted by cartridge of 200 g is as below:
- Fume Class 1: 0.00 to 0.16 cu. ft noxious gas/200 g of explosive;
- Fume Class 2: 0.16 to 0.33 cu. ft noxious gas/200 g of explosives;
- Fume Class 3: 0.33 to 0.67 cu. ft noxious gas/200 g of explosives.

Explosives in Fume Class 2 or Class 3 must not be used underground unless special permission has been obtained. However, the methods of detecting these noxious gases are still not standardised.

Tests. There are several ways to determine fume concentrations. These include: measurements in the Bichel gauge, the Crawshaw-Jones Apparatus, the Ardeer Tank, field tests and theoretical calculations.

Bichel Gauge is an apparatus equipped with a 15 l chamber in which the explosive is fired. A pressure gauge is provided for recording the pressure in the chamber and a stopcock valve for controlled release of gasses after the charge has been fired. The charge is 200 g with a proportionate amount of wrapper. Detonation is by a No. 6 strength instantaneous electric blasting detonator. The gauge is evacuated to about a 50 mm vacuum prior to firing. Pressure and temperature readings allow calculation of the volume of gasses in the gauge to 0°C and 760 mm of pressure, the standard for this determination. Five minutes after firing, 200 cc samples of the gases are withdrawn and analysed to determine their nature and volume.

Crashaw Jones Method uses an apparatus consisting of a heavy steel cannon with a 50 mm diameter by 540 mm long borehole, which can be attached to a closed chamber with a capacity of 90 l. The closed chamber contains access parts for evacuation, for recording the temperature and pressure and for withdrawing gas samples. The procedure involves the use of charges consisting of 300 g of the explosive ingredients and its proportionate amount of wrapper, as found in a 32 mm diameter cartridge. A No. 6 electric detonator is used to initiate charge. After the chamber is closed and evacuated to 1 mm and the shot is fired. After a 5-minute interval allowed for temperature equilibrium, the temperature and pressure are noted. Samples are drawn for analyses in previously evacuated bottles. The volume of gas is reduced to 0°C and 760 mm of Hg and volumes of carbon monoxide and oxides of nitrogen are calculated. For permissible explosives, the sum of carbon monoxide from the Bichel method and the oxides of nitrogen from the Crawshaw Jones method is taken as the total fumes.

There are several problems associated with extrapolating laboratory fume data to actual field use. Some watergels and ANFO will not always completely detonate in such small quantities. Also, the explosive is not confined and the explosive gases readily expand into a relatively large volume, unlike the borehole confinement typical of field use.

The most efficient method is to take on-site measurements after the blast. There are two methods, 1) remote drawing of samples, 2) by trained personnel protected against noxious gases. In the first case in one test facility actual blasting is carried out in a tunnel and through boreholes drilled from surface, samples are drawn at 5 minute intervals. By gas chromatography these samples are analysed. In the second method samples are collected by vacuum bottles or water/liquid replacement methods when persons enter the atmosphere. A rather simplified approach is that of using spot samplers for detecting concentration of gases.

Also, theoretical estimates of fume production is carried out using computers. In general these theoretical results correlate quite well with measured results, but are not designed and should not be substituted for on-site measurements.

3.16 PERMISSIBILITY

All explosives when fired give a flame which varies in volume, duration and temperature. Permissible explosives are designed to produce flame of the least volume, lowest temperature, and shortest duration possible. Certain salts are incorporated in their make-up which have a cooling or quenching effect on the flame. Standardised tests are carried out by regulatory authorities and after they have passed these tests, then they are classed as safe for blasting in gassy or dusty mines provided it is used in a specified or 'permitted' manner.

P-1 explosives are recommended to be used in Degree I gassy mines. P-3 explosives are recommended to be used in Degree II and Degree III mines and P-5 are allowed to be used for blasting off-the-solid in underground coal mines.

CHAPTER 4

Explosive loading and priming systems

4.1 INTRODUCTION

Blast hole loading consists of placing all the necessary components into the blasthole, including the main explosive charge, deck charges, initiation systems, primers and stemming. Choice of blast hole loading system depends on the type of explosive, diameter and inclination of hole and size of the blast. Loading of holes is either carried out manually or by mechanised means. Large blasts need mechanised loading operations whereas small blasts can be charged by manual methods. Small diameter holes are predominantly manually charged but several mechanised means are available. Depending on the technology employed in related operations the loading system is employed. Small diameter holes are drilled at various angles – vertical, horizontal and inclined in any direction. Large diameter holes are generally drilled vertically downwards, but in some cases are angled or are horizontal. The explosive system need to be tailored according to the technology employed in related operations.

To achieve proper energy release from the explosive column adequate priming need to be incorporated at the time of loading of explosive in the hole. During the period in which virtually all commercial explosives were NG-based explosives, initiation (and priming) was a relatively simple matter. However, now a days a variety of explosives, their initiation and loading systems have become available and to be able to maximise energy utilisation it has become important to place emphasis on priming systems, for which necessary steps are taken at the time of loading the explosives. The type of primer explosive and quantity used should be suitable. Poor priming can cause poor fragmentation, excessive ground vibration, airblast, flyrock and/or damage to the remaining rock.

In this chapter consideration has been given to various available explosive loading and priming systems.

4.2 CHECKING THE BLASTHOLES

After drilling operations have been completed checking of blastholes should be carried out. Firstly, it is necessary to ensure that drilling has been carried out according to planned pattern, depth and inclination. It is not uncommon to find holes drilled up to 50% error on burden and spacing. These large errors change the energy distribution and energy density in the rock mass and cause boulders, flyrock, high ground vibrations, as well as other problems associated with blasting. Best results occur if blastholes are drilled within one-hole diameter of their hole location. Generally, a driller will find it difficult while collaring hole at a specific location and will have to move over. This problem can specially come on the top benches where the ground may be uneven with depressions and humps. The first row of holes on the bench may be affected by the backbreak of the previous blast thus causing difficulty in collaring the holes and in turn may lead to smaller burden in the collar region and higher burden in the toe region leading to unsatisfactory results.

Depending on the depth a tape to which a small weight has been attached or a tamping pole can be used for measuring hole depth. For long uphole it is rather difficult to measure drilled distance. Instruments for measuring hole depths have also become available. Holes shorter than the planned ones should be cleaned out with the drill or with compressed air or redrilled. Redrilling is also resorted to when it is found that the holes have been clogged by a falling stone. Obstructions in small holes may sometime be dislodged with a tamping pole. In large diameter vertical holes, a heavy weight suspended on a rope and dropped repeatedly on the obstruction may clear the hole. Sometimes economics or equipment limitation may dictate that the blast be carried out with a few short holes.

If it is necessary to redrill a hole adjacent to blocked, then the new hole should be placed sufficiently away so that it does not drill into the blocked hole. A hole should not be drilled if it is loaded with explosive or when there is danger of intersecting a loaded hole.

The inclination of the hole is important for many stope or underground blasting situations and in many surface blasting situations. Mechanical and electrical devices are available which help the driller to position the drill mast so that each hole enters the ground at the designed angle. Often visual estimation is resorted to which is a poor procedure. For a 20 m bench an error of 10° in alignment will result in burden being 3 m less or more than the planned burden. The blasthole will be short and the subdrilling will not be deep enough.

Holes should also be checked for the presence of water. Estimation of the level of water in a borehole can be made by visually checking the tamping pole or weighted tape for wetness after the borehole depth check has been made.

A blasthole may pass through or bottom into an opening. Where this opening is not unduly large, it may be filled with stemming material. Where the opening is too large for this to be practical, the hole must either be left unloaded, redrilled in a nearby location, or plugged.

In some coal mines or pyrites mines hot holes may be encountered. If there is reason to suspect a hot hole, the hole can be checked by suspending a thermometer in it for a few minutes. Explosive material should not be loaded into holes hotter than 65°C. Specially formulated explosives and initiating devices are to be used to meet with such a situation.

4.3 EXPLOSIVE LOADING PROCEDURES

Blastholes may be loaded with bulk or packaged explosive products. Bulk products

are either poured into the hole, augured, pumped, or blown through a loading hose. Packaged products are either dropped into the hole, pushed in with a tamping pole or other loading device, or loaded through a pneumatic charger. The rise of explosive column be checked frequently as loading progresses, using a tamping pole, weighted tape, or loading hose. This gives warning of a cavity or oversized hole that is causing a serious overcharge of explosive to be overloaded, and will also assure that sufficient room is left at the top of the hole for proper amount of stemming. When the explosive column has reached its proper location, the primer is loaded into the blasthole. It is important that the wires, tubes, or detonating cord leading from the primer are properly secured at the borehole collar.

When cartridged products are used, coupling is improved by slitting the cartridge and tamping them firmly. There are some situations where cartridges or packages of explosives should not be tamped:

1. Nitro-glycerine based explosive cartridge should not be cut or slit;
2. In permissible coal mines, the cartridge should not be deformed, slit or cut;
3. In controlled blasting, where string loads or even gaps between cartridges are used to reduce the charge load in the perimeter holes to prevent shattering;
4. In water, where the package serves as protection for non-water resistant explosive product;
5. A primed cartridge is never to be tamped.

Blasthole stemming

Stemming is used in order that a high-explosive charge functions properly and releases the maximum energy, the charge must be confined in the borehole. Adequate confinement is also necessary to control airblast and flyrock. The common material used for stemming is drill cuttings. However, very fine cuttings make poor stemming material. Selection of the proper size of stemming material is important if one wants to minimise the stemming depth in order to break cap rock. Very fine drill cuttings will not hold into the blasthole. Very coarse materials have the tendency to bridge the hole when loading and may be ejected. The optimum size of stemming material would be material that has an average diameter of approximately 0.05 times the diameter of the blasthole. Material must be angular to function properly. Use of stemming machines in surface mines is recommended when large blasts are to be carried out.

Blasthole dewatering

While blasting in watery holes it is advisable to use a water-resistant explosive. However economics may some times dictate that hole be dewatered. For the purpose of borehole dewatering several types of dewatering units are available. Generally, these units are hydraulically driven though electrically driven units are available. The unit may be mounted on a pickup or attached to the front of an ANFO bulk truck. The former is preferable as ANFO truck mounting reduces the flexibility of the unit. The unit is a hydraulically driven submersible pump attached to a hose and reel assembly which is mounted on a ½-¾ ton pickup. The pump is connected to 25 m hydraulically driven telescopic boom provided to position the pump over the borehole. The hydraulic motor which drives the pump, the boom and the hose reel, is itself driven by a take off from the engine of the pickup. A variety of pump sizes and ratings are available depending on application. Generally these pumps are of rugged stainless steel construction, however they are affected by slurries in the hole or by adverse pH (+9 or –5) conditions.

For the purposes of preventing the influx of water into a blasthole loaded with a dry mix blasting agent, dry liners could be used. These generally consist of polyethylene tubing of 8-12 mil. thickness, sealed at the lower end and made to correct depth and diameter of borehole. Pre-manufactured units are available with a reinforced bottom and double heat sealed pocket for weight insertion to aid in loading, or of double wall 4 mil. ply, co-extended polyethylene construction.

Units may also be manufactured at the blast site by blasting crews from layflat polyethylene tubing. The cost of such units is lower, though the chances of pinholing and imperfection are increased. The units should be of construction such that they are waterproof, flexible and will not crack (especially the seams) when exposed to low temperatures.

4.4 EXPLOSIVE LOADING IN SMALL DIAMETER HOLES

Cartridged NG-based and slurries (watergel and emulsion) are commonly used in small diameter holes. These cartridges are, usually manually loaded, and tamped to provide maximum coupling and loading density. One or two cartridges should be loaded after the primer before tamping begins. Tamping should be done firmly, but not excessively. Using the largest diameter cartridge compatible with borehole diameter will increase coupling and loading density.

When small-diameter holes are loaded, the primer cartridge is normally loaded at

Figure 4.1 Pneumatic cartridge loader (courtesy Nitro Nobel).

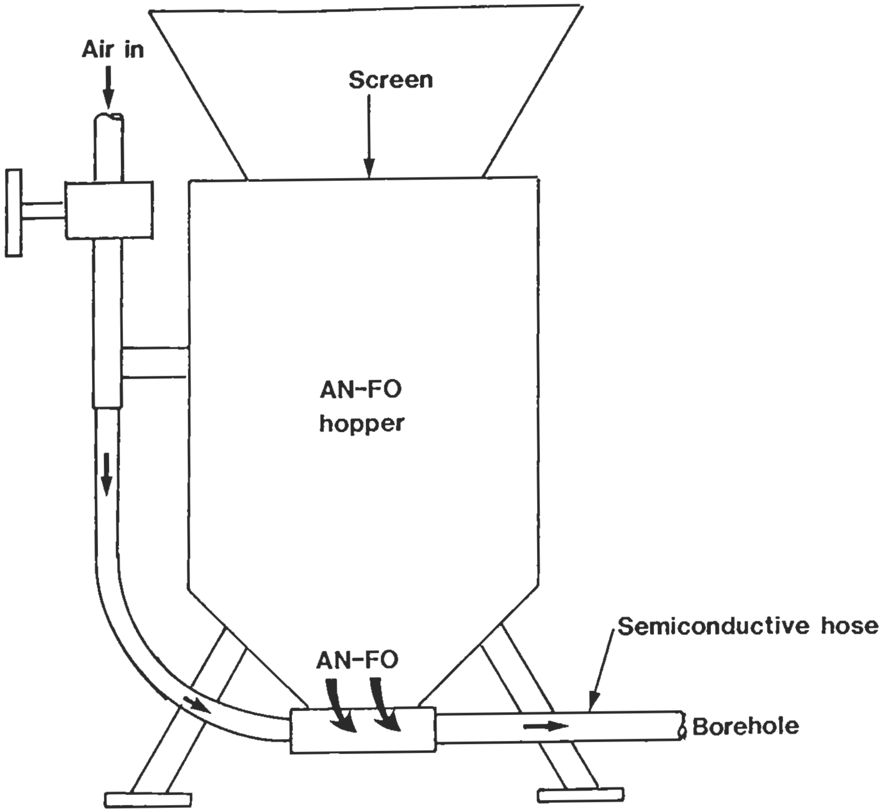


Figure 4.2 Ejector type pneumatic ANFO loader.

the bottom of the hole. This gives maximum confinement at the point of initiation and also guards against leaving undetonated explosive in the bottom of the hole if it should become plugged or cut off during the blasting process.

Pneumatic cartridge loaders are used to load cartridged explosives into long and awkwardly cited holes (Fig. 4.1). Upholes in ring blasting are suitably charged with the use of these loaders. This charging equipment is found to provide good charging capacity and high charge density. The chargers are compressed air operated and use anti-static material. The cartridges are propelled through a loading hose at a high velocity at a rate of up to one cartridge per second. The cartridges are automatically slit as they enter upon impact. Automatic chargers are also available.

For placement of ANFO in small diameter underground holes, pneumatic loaders using standard mine air pressures have been in use. Two types are available:

1. The ejector type consisting of a hopper, ejector assembly and control valve (Fig. 4.2). Its operation is based upon the venturi system whereby the premixed ANFO mixture is drawn from the hopper into the centre of an annular air jet within the ejector assembly and is then ejected into the borehole. Loading rates vary from 3 to 10 kg/min and by varying the loading hose dimensions it is possible to vary the particle size and thus the density of the loaded ANFO from 0.9 to 1.0 g/cm^3;

Figure 4.3 Pressure vessel type pneumatic ANFO loader (courtesy Nitro Nobel).

2. The pressure vessel type consists of a pressure vessel, regulator and control panel (Fig. 4.3). The compartment containing ANFO is maintained under pressure during loading should have a pressure regulator so that the tank pressure does not exceed the manufacturer's recommendation usually 200 kPa. The ANFO is then passed via a control valve into the discharge hose which may be up to 60 m in length. Using this method, discharge rates of 40 kg/min are possible, with hopper capacity of up to 450 kg. Smaller loaders have loading rates of 7 to 20 kg/min and ANFO capacity of 35 to 80 kg.

4.5 EXPLOSIVE LOADING SYSTEMS FOR LARGE DIAMETER HOLES

Large diameter nitro-glycerine based explosives are mainly used now a days as primer or bottom charge. Slurries, emulsions and ANFO give better economy in large diameter blastholes. Slurries and emulsions are available in polyethylene packaging in diameters up to 200 mm. Some of these products are semi-rigid and others are in dimensionless bags that will slump to fit the borehole diameter. With the semi-rigid cartridges, the advantages of borehole coupling is lost.

Mechanised (also called Bulk) ANFO loading systems have been in use for over three decades the world over. The use of on site mixed slurries since their introduction in mid-sixties have also continued. Further, with the development of emulsion explosives and combinations of various products other bulk loading systems have become available. The late 1980s have also seen introduction of emulsion explosives and Heavy ANFO. The bulk loaded explosives systems have many advantages over the conventional cartridge loading system:

– It provides versatility to change explosive energy as required;

– Since mostly these types of explosives are in a pumpable form at the time of delivery, the loading rates are much higher than conventional packaged slurries or emulsions;

– Since the explosive is pumped into the borehole, a better coupling is ensured with the borehole walls;

– The system provides flexibility to change the density of explosives and ability to charge three or more explosives formulations in predetermined quantities in the same blasthole.

When compared with cartridged system considerable savings are apparent, in addition one can consider savings in storage, transportation and charging costs. Appendix 1 compares cartridge and bulk explosive system in a mine using 5000 tonnes per annum.

Various mechanised systems of manufacturing and charging of explosives suitable for surface mines are as below:

1. Dry blasting agents and ANFO mixtures;
2. Bulk slurries/watergel explosives;
3. Emulsion products, a) Bulk emulsions, b) Doped emulsions, c) Repumpable emulsions;
4. Heavy ANFO.

The term bulk delivered explosive is used to denote the explosive product that is delivered directly into the blast hole without being cartridged and packaged. The delivery of the explosive into the blast hole is affected by use of pump or auger from storage tanks mounted on truck chassis. A variety of delivery systems are available whose characteristics vary depending on the type of product delivered, charging rate, capacity and size, support plant facilities provided on the truck etc. Some of the common delivery systems deployed are as follows:

1. Site Mixed ANFO and Heavy ANFO Systems
2. Site Mixed Slurry (SMS) or Emulsion (SME) Delivery System
3. Plant Mixed Slurry(PMS) or Emulsion (PME) Delivery System

4.5.1 *Site mixed ANFO and Heavy ANFO systems*

There are two variations of the basic ANFO truck. The simple one is a concrete mixer where ANFO is batch mixed and chute loaded into the hole. This has the advantage of delivering a uniform product at a very high rate but the disadvantage of being able to load a single product at a time. The other variation extends the flexibility of the system. A second pump and a small mixing bowl are added and this allows either a straight or doped emulsion to be added through a hose. This gives Heavy ANFO product with the wet hole loading characteristics and water resistance of standard bulk slurries or emulsions. Thus one truck can load both a completely waterproof Heavy ANFO product or ANFO as well as all variations in between.

These trucks deliver one of the widest range of products of ANFO, Heavy ANFO, emulsion and doped emulsion products. The arrangement essentially consists of an AN storage bin and containers for fuel oil and emulsion matrix storage (Fig. 4.4). ANFO and emulsion are blended in the mix hopper for producing blended products or to deliver individually as ANFO or a gassed emulsion explosive.

The advantage of the system is that it delivers a very wide range of products. The system is expensive and needs high maintenance.

The second type of system can deliver only ANFO and Heavy ANFO products. These units contain a bin for ammonium nitrate, a fuel oil tank and an emulsion bin (Fig. 4.5). The fuel oil and ammonium nitrate are mixed in an auger to make ANFO. The emulsion is added in varying percentage in the boom auger of the truck to make

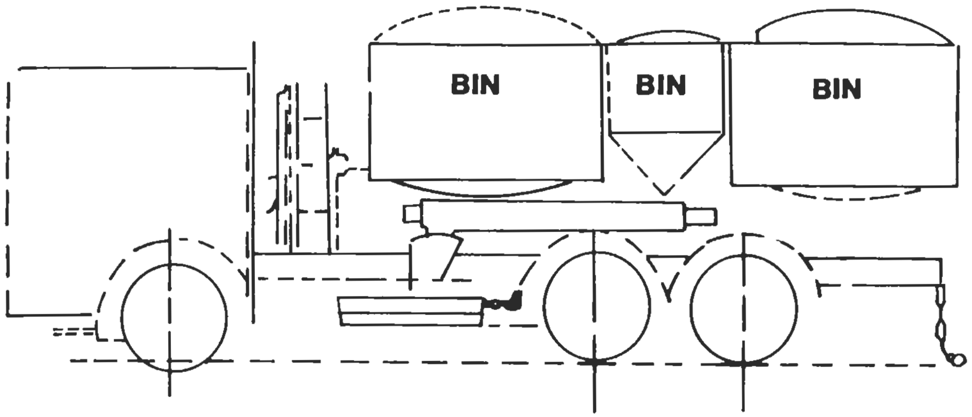


Figure 4.4 Multi product explosive loading truck.

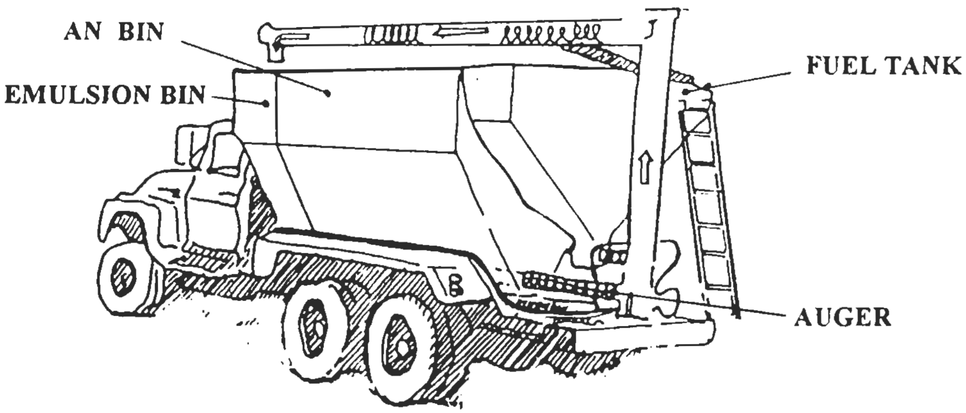


Figure 4.5 ANFO – Heavy ANFO loading truck.

the desired product (Evans & Taylor 1987). The standard Heavy ANFO truck is either a converted ANFO truck or essentially identical purpose built Heavy ANFO truck. It consists of an ammonium nitrate bin, an emulsion bin, probably an aluminium bin and a fuel oil tank. The fuel oil is added to the AN as usual (as is the aluminium if needed) and the matrix is pumped into the ANFO in the loading auger. Even a relatively short length of auger provides good mixing and the product drops into the hole. The system is relatively inexpensive and possesses adequate capacity. The disadvantage of the system is that it is used only for augerable product.

4.5.2 *Site mixed slurry/emulsion system*

The Site Mixed Slurry (SMS) or Emulsion (SME) System consists of support plant and pump trucks. The support plant is located near the mine or at centrally located place if it caters to a group of mines. Non-explosive ingredients are stored in it. Certain intermediates are prepared from some of these ingredients and kept ready. When

Figure 4.6 Site Mixed slurry/emulsion explosive loading truck (courtesy IBP Co.).

the blastholes are to be loaded, the ingredients are loaded in specially designed pump trucks.

The pump trucks are specially designed to carry all the ingredients and to pump the blended slurry into boreholes through a delivery hose carried on the truck (Fig. 4.6). The blending operation is controlled by a sophisticated control system. A predetermined quantity can be pumped into a hole through the control system. The various ingredients are continuously metered and passed through a hose into the borehole. In case high energy bottom load and low energy top load is required, the desired quantity of each is set on the control system.

At the site, the calibrated quantities of ingredients for a particular product are mixed and the product is pumped into the blast hole through a delivery hose. The mix becomes sensitive after five minutes into the hole after certain reactions are completed. In this system there is a flexibility to charge three different product formulations of various energies and densities in predetermined quantities in the same hole.

The pump truck is a vital part of SMS system. These trucks are specially designed to carry all the ingredients required for blasting and to pump the blended slurry into boreholes through a delivery hose carried on the truck. At the blasting site the above ingredients along with a suitable cross linking agent like guar gum (to form gel) and gassing agent (for controlling density of slurry) are all mixed together just before commencing the loading itself so as to form a pumpable slurry of the required composition.

The blending operation is controlled by a sophisticated control system which is reliable and simple in operation. A pre-determined quantity to be pumped into a hole can be set through the control system by the operator and then a 'start button' is pressed to pump the required slurry into the borehole. The various ingredients are continuously metered and passed through a hose into the borehole. When the quantity of slurry is delivered the unit automatically shuts off. In case high energy bottom load and low energy top load is required, the desired quantity of each is set on the control system. After the desired quantity of bottom load is delivered, the unit automatically shifts to the low energy composition. When the desired quantity of both bottom and top loads are delivered, the unit automatically shuts off. This ensures that there is neither excess explosive getting into the borehole, nor stored in the pump trucks.

There is facility of charging three types of explosives having varied strengths into the blast holes depending on the nature of strata and sequence as desired.

4.5.3 *Plant mixed slurry/emulsion system*

In this system premixed slurry is made at a captive plant called satellite plant located in close proximity to the mine site. These explosives are then loaded into a pump truck with two or more compartments and carried to the site. The explosive is prepared under controlled conditions and exactly tested for its properties and is directly pumped into borehole with addition of cross linkers and thickeners. In this system fi-

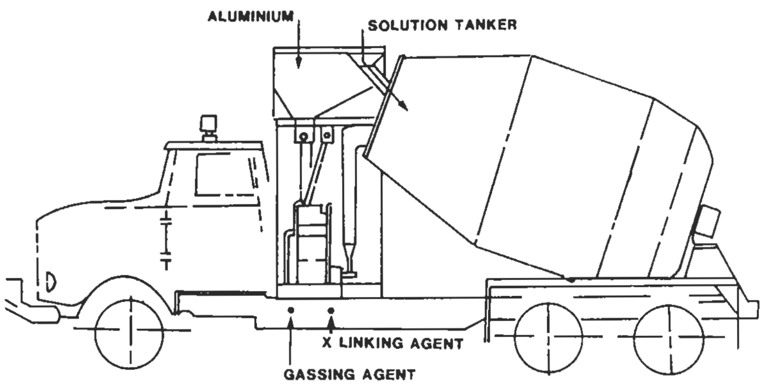


Figure 4.7 Plant mixed slurry explosive loading truck (courtesy Indo-Gulf).

nal rheological properties are given to explosive at site and not in plant.

The truck consisting of a rotating mixer bowl feeds a slurry (or emulsion) pump via a blender if necessary for aluminium dosing (Fig. 4.7). The pump transfers the product through a flexible loading hose mounted on a retractable, mechanised hydraulic boom and reel into the borehole.

Explosive delivery is controlled from inside the truck cab. The hose puller and boom extension position the hose in the borehole and retract it during the loading cycle. During the pumping sequence aluminium and other reagents can be added to vary the specific energy and also to control the slurry density and gel structure.

In a modified delivery system of PMS pump trucks the non explosive premix prepared under controlled supervision at the plant or on the gelmaster truck is carried to the mine site along with other ingredients. The explosive is mixed in batches and pumped down the borehole where the mix attains its explosive properties after a short period of time.

The main part of the system is the truck consisting of a rotating mixer bowl mounted on a truck chassis. The discharge end of the mixer bowl feeds a slurry pump via a blender for aluminium dosing. The pump transfers the product through a flexible loading hose mounted on a retractable, mechanised, hydraulic boom reel into the bore hole. All controls are located inside the cab on a panel beside the driver.

4.5.4 *Consideration while selecting loading system*

With the availability of a number of bulk explosive systems, the choice is a difficult one. Several factors need to be considered:

1. Normally for large operations bulk loading systems are suitable. Problems may arise when one comes across limited size of individual mines and quarries. For smaller operations the shared services of a pump truck and its support facilities may be necessary. In such situations to operate this system effectively a strictly prearranged schedule of hole loading may be necessary;

2. Not only blasting must be arranged to a given schedule, but also the size of the shots be tailored to suit the capacities of pump trucks and support facility;

3. Rock and blasting conditions have very important effect. For soft to medium strength rock such as in coal measure strata most of the systems work satisfactorily. But in hard rock it seems emulsions work better. In watery holes ANFO usage is precluded and even with Heavy ANFO (with up to 25% emulsion) reliable blast results are not achieved. When hard rock and watery holes are encountered emulsions are found to be much better;

4. Whenever, any mine is planning for a bulk loaded explosive system it must be considered that variable rock conditions will be encountered hence any system, which offers more than one type of explosive with variable composition, be chosen. This is an essential consideration since mine will then be dependent on one plant only, hence that plant should be able to supply all the required types and compositions of explosives;

5. An important advantage of bulk loading system is that explosive companies are responsible for carrying out charging and assisting in blasting operations and hence they provide technical service. A company with good technical background is an obvious choice;

6. The system should have check on the weight and quality of explosive delivered. It is difficult to have foolproof system hence it is advisable that double checking system including electronic records be adopted;

7. The system should have adequate alternative possibilities so that impact of breakdown of the system is reduced to minimum;

8. Another relevant consideration is about the usage of increased explosive charge per hole (up to 20%) because of the better coupling afforded by these type of explosives. This may pose vibration problems if the mine blasting is near an urban area.

4.6 EXPLOSIVE LOADING FOR UNDERGROUND MINES

Broadly, the excavations in underground mines are divided into development and production mining. Mostly cartridged explosives (varying from 25 mm diameter up to 125 mm diameter for 38 mm to 150 mm holes) are used. Small diameter cartridges are generally 200 mm to 250 mm length. In mechanised mining there is need to consider increasing the hole diameter and corresponding increase in explosive cartridge diameter. To reduce loading time long tube charges can be used while manually loading horizontal holes in drive blasting.

Bulk ANFO loading systems for underground usage have been developed. The system uses the same principles as ejector type unit and may be equipped either with

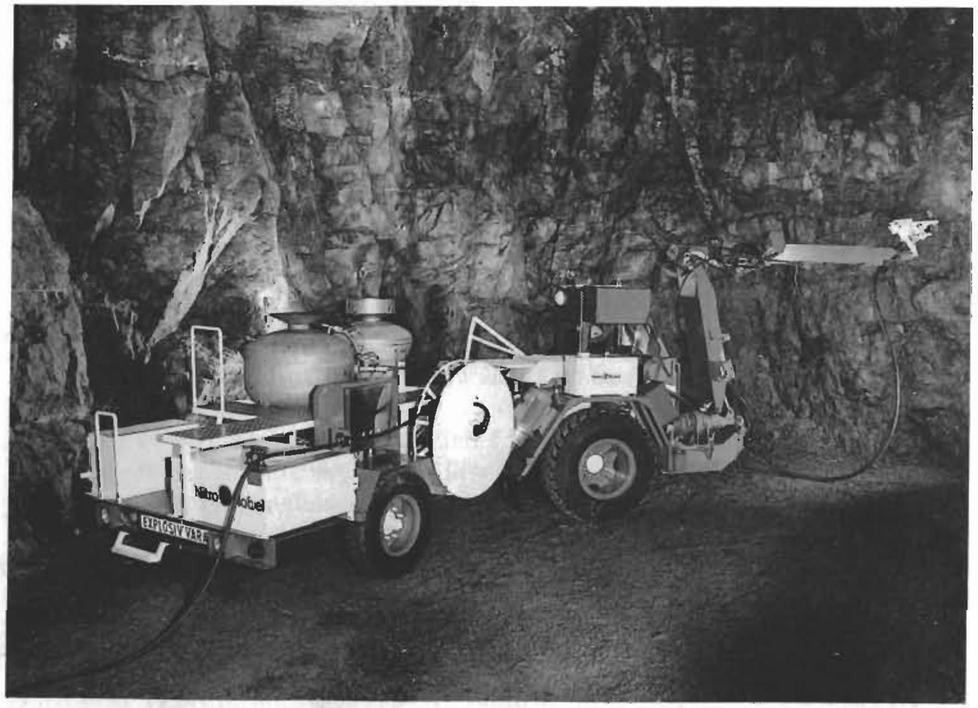


Figure 4.8 Bulk loading pump truck for underground usage (courtesy Nitro Nobel).

one or two replaceable 400 kg hoppers. The whole unit is truck mounted and avails to a highly flexible bulk handling for small diameter ANFO loading (Fig. 4.8).

Bulk SMS/SME pump trucks are also used for large diameter holes in underground mines. The capacity ranges from 0.5 tonnes for development headings to 12 tonnes for ring blasting. The advantages are:

– Higher loading densities and improved borehole coupling provide major cost saving benefits as hole spacing are widened and the required drilled meterage is reduced. Loading densities greatly exceed competitive methods such as pneumatic loaded ANFO and pneumatic cartridge loaders;

– Productivity also improves with high speed loading (for example in VCR mining and in pillar blasting);

– It alleviates the need for extensive magazine storage areas in underground mining. Safety is greatly improved as the need for large quantities of flammable packaging is removed from the usually congested underground environment. Shelf life may be varied as required. Charging into vertical upholes of up to 125 mm in diameter by use of a special homogenising valve is possible.

4.7 PRIMING SYSTEMS

Cap sensitive explosives are initiated by energy delivered from the initiator like a detonator. Whereas blasting agents such as ANFO products and many watergel and emulsion which are non-capsensitive products do not reliably initiate with the energy released from a detonator. These products need additional energy from a primer or a booster.

Essentially, the term primer is used to describe a unit of cap-sensitive explosive that contains a detonator, while the term booster describes a unit of explosive that may or may not be cap sensitive, and is used to increase explosive energy in the remaining explosive charge. Although a primer is generally thought of as containing a blasting cap, the primer cartridge may also be detonated by a downline of detonating cord.

To achieve proper energy release from the explosive column adequate priming need to be incorporated in the explosive system. The type of primer explosive and quantity used should be suitable. Poor priming can cause poor fragmentation, excessive ground vibration, airblast, flyrock and/or damage to the remaining rock.

4.7.1 *Primer type*

Primers are of many sizes and are of varying compositions. Primers may be as small as 200 grams or may consist of a 20 kg cartridge of explosive. Primer diameters can vary from a few mm to well over 250 mm. In blastholes of less than 100 mm diameter capsensitive cartridged products are used. In larger diameter blastholes, cast primers are used, although some operators prefer to use cartridged high explosives. Various grades of nitro-glycerine based explosive cartridges are used as primers. Now a days capsensitive watergel and emulsion slurry cartridges are used. These are used when both cartridged or bulk explosives are charged in small or large diameter holes. When the holes are bulk explosive loaded then cast boosters of non-nitro-glycerine explosives can also be adopted as primers. Each of these is designed to satisfy requirements for a given condition of pressure, energy, velocity, size, cost or convenience. Each of many types of primers requires an assembly technique ingenious to particular booster and initiator being used.

Cast boosters (see Section 2.9.5) are capsensitive primers, which produce a high detonation pressure for initiation of ANFO products and other bulk loaded watergels or emulsion explosives in borehole diameters 50 mm and larger. Cast boosters vary considerably in make up. Some are made up with a less sensitive outer portion with a more sensitive inner core to accept initiation from a detonator or a detonating cord. These inner cores may be made of pentolite (50/50 mixture of TNT/PETN), PETN in a small plastic sock, a small section of high strength detonating cord, or plasticised PETN composition. Other cast boosters are made completely of Pentolite (50/50 mixture of TNT/PETN). Most of these cast boosters have high densities (about 1.5-1.6 g/cc) and high velocities (about 7000 m/s) resulting in detonation pressures exceeding 240 kb.

4.7.2 *Primer properties*

Important properties of a primer which are needed to meet the diverse conditions of application to ensure both reliability and performance include detonation pressure, size, water resistance, initiation sensitivity, and physical strength.

It has been suggested that a primer should have a minimum detonation pressure of 80 kb and match the diameter of the borehole as closely as possible.

The selection of a primer must include the ability to perform at full potential under both high water pressure and long sleeping times. Unexpected rains or equipment breakdown resulting in higher water pressures or longer sleeping times than originally can be disastrous if primers with inadequate water resistance are selected.

The initiation sensitivity and performance of a primer are dependent on sensitiser, particle size of ingredients, density, diameter, temperature and static water pressure encountered in use. Extreme low temperatures can reduce the initiation sensitivity of water based watergels or emulsions. When low temperature applications are anticipated specially formulated products are recommended. High water pressures can adversely affect the reliability and performance of water gel slurries and most ammonia gelatine dynamites which rely on or in part, entrapped air for sensitivity. However due to varying composition and physical make-up cast boosters do vary in water resistance and ability to accept initiation under various conditions.

The physical strength (the ability to remain intact during handling and use) of a primer must be such that the detonator or detonating cord remains securely embedded in the primer until the shot is fired. The primer can be physically damaged if large diameter cartridges are dropped on them in dry holes. Dropping on primers in large diameter dry holes should never be allowed which is extremely hazardous. This practice can result in premature detonations.

Watergels and emulsions have semi-rigid characteristics and usually do not present problems when used as primer in small or medium diameter holes where the cartridge diameter is fairly close to the hole diameter and the column weight is small.

Small diameter water gel or emulsion cartridges can be used in large diameter dry holes with bulk ANFO with little deformation. However, small diameter water gels

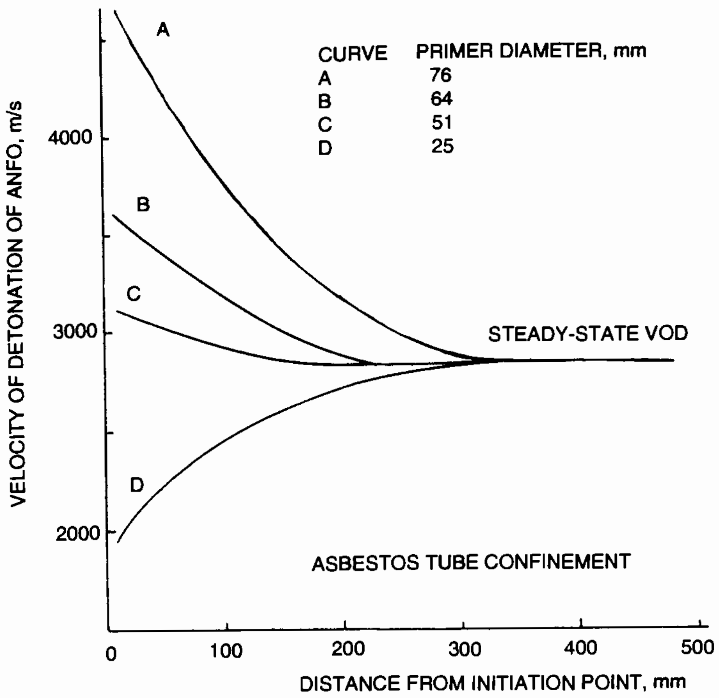


Figure 4.9 Effect of diameter of the booster on initial velocity of a 76 mm column of ANFO.

or emulsion primers should not be used in large diameter wet holes when a column of cartridge explosives is loaded to get out of water. The weight of the column can force the detonator out of the cartridge, force the watergel or emulsion out of the cartridge, or deform the cartridge below the critical diameter of the product. Any one of these or a combination of these can result in misfired holes.

Mixtures of ammonium nitrate and fuel oil (ANFO) are not very sensitive explosives. Performance can be influenced by hole diameter, ingredient particle size, density, degree of confinement, moisture, drill cuttings and disruption from detonation cord.

Studies have been undertaken to determine the primer properties which affect the VOD of the ANFO in the vicinity of the primer (Fig. 4.9). The VOD can start at low order (a velocity lower than steady state) and then change to steady state or they can start at high order (a velocity higher than the steady state velocity) and then return to steady state velocity. Figure 4.10 shows effect of detonation pressure on the initial velocity of ANFO. It shows the initial velocities to be very low when primers with low detonation pressure were used. The initial velocities of the ANFO increased progressively as the detonation pressure of the primer increased. In all cases the VOD of the ANFO returned to the steady state VOD of ANFO within a length of about four charge diameters.

In another series of tests where the detonation pressure of the primer was held constant and the only size or diameter of the primer was varied, showed that the initial VOD of the ANFO was very high when a primer with high detonation pressure matched the diameter of the ANFO column. However, as the mismatch between the

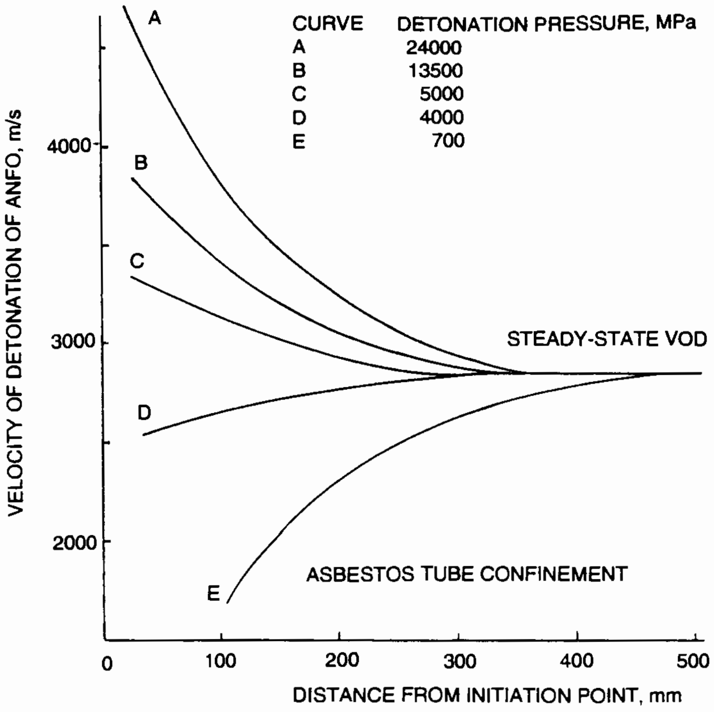


Figure 4.10 Effect of detonation pressure on the initial velocity of ANFO in a 75 mm diameter test column.

diameter of the primer and the diameter of the ANFO column increased, the initial VOD of the ANFO decreased. This test established the importance of closely matching primer to borehole diameter to obtain performance from ANFO.

The primer must be of sufficient length to reach its rated VOD when the ANFO is initiated. Many explosives do not reach its rated VOD in the sizes normally associated with primers.

4.7.3 *Primer Location*

Proper location of the primer is important from the standpoint of both safety and efficiency. Normally, the rock which needs explosive energy is at the bottom of the hole (in vertical holes) and at the back of the hole in headings and drifts. The rock in this region must not only be sheared, but also be shattered and moved out. In quarry work, proper priming and bottom hole initiation results in a clean even floor without a prominent toe. In tunnel work or headings it means there is little or no bootleg with more advance per round. When using cartridged products in small-diameter blastholes, the primer should be the first cartridge placed into the hole, with cap pointing toward collar. In small size holes 75 mm diameter or less, the practice of using high strength detonators or 2 to 18 g mini-primers to initiate ANFO can result in false economies. The use of a good primer results in either greater advance and better fragmentation, or in drilling of fewer holes.

If bulk products are being loaded, the primer may be raised slightly from the bottom of the hole. In bench blasting with a bulk loaded product, where subdrilling is

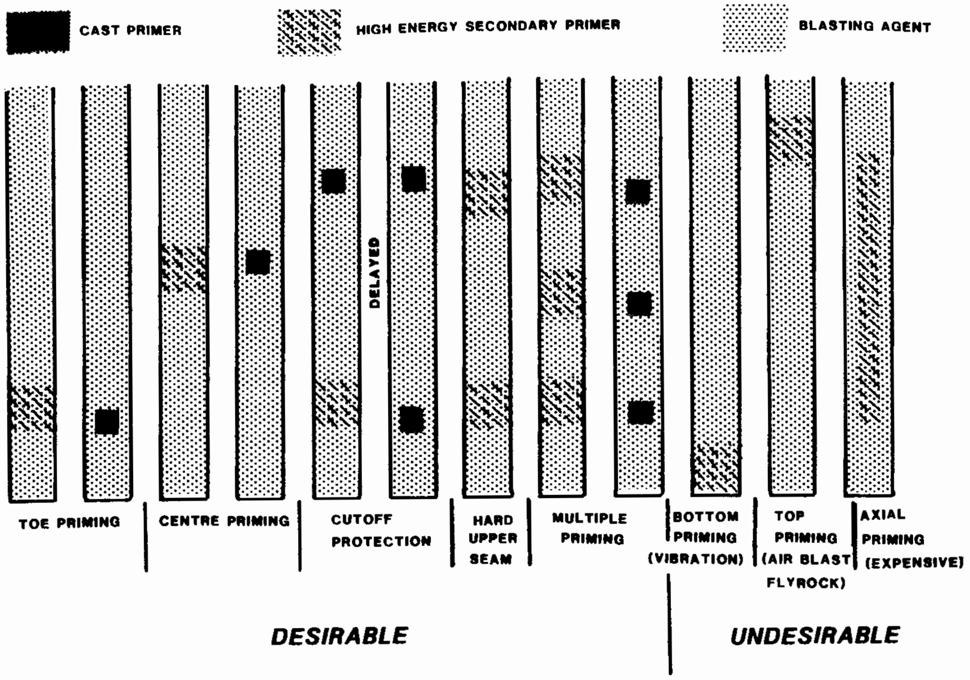


Figure 4.11 Priming in large diameter blastholes.

used, the primer should be placed at toe level. If there is some compelling reason to place the primer at the collar of the hole the detonator should be pointed toward the bottom of the hole.

4.7.4 *Single priming*

In large diameter blastholes, the recommended location of the primer is the bottom of the hole although many different locations are practised (Fig. 4.11). To help reduce the vibrations a primer should be at toe level rather than in the bottom of the hole where subdrilling is used. Bottom initiated holes tend to produce less flyrock and airblast than top-initiated holes assuming that all other blast dimensions are equal. Top priming is seldom recommended except where the only fragmentation difficulty is a hard band of rock in the upper portion of bench.

4.7.5 *Multiple priming*

In many blasting situations, single-point priming may be adequate. However there are some situations in which multiple primers in a single borehole may be needed. The first is where deck charges are used. Deck charges are used (1) to reduce the powder factor in a blast while still maintaining satisfactory charge distribution, (2) to break-up boulder-prone caprock in the stemming area of the blast, or (3) to reduce the charge weight per delay to reduce vibrations. Each deck requires separate primer.

Another reason for multiple priming is a safety factor to assure total column detonation. There is possibility of getting a poor blasting cap, which may not fire, or cut-offs of the hole may be caused by shifting rock. This application is also referred to as alternate velocity. The concept is to use an additional cartridge of a high energy – high VOD product in the ANFO column to obtain a faster, more efficient release of energy from the ANFO.

Cast primers have also been developed which incorporate an internal millisecond delay. The cast primers and the delay devices are supplied separately. These delay primers are slipped on to a detonating cord downline and are especially useful in providing multiple delays in the blasthole on single downline.

The possible undesirable effect of the cord on blasting agents must be considered. With cap-sensitive explosives, continuous axial initiation will occur with any cord containing 5 g or more of PETN per meter of cord. Lower strength cords may also cause axial initiation. With blasting agents the effect of detonating cord is less predictable. The blasting agent may be desensitised or it may be marginally initiated. A low energy downline or alternative, non-disruptive initiation systems are recommended. If the column charge is cap sensitive, the detonating cord will cause initiation to proceed from the top down.

4.7.6 *Axial priming*

Axial priming employs a central core of primer throughout a bulk loaded explosive column. This technique gives good results specially if ANFO is not of uniform quality. It is time consuming and tedious when small diameter cartridges are taped on to detonating cord and then it is suspended in the holes. However, one can use a reel of small diameter slurry or emulsion explosives. This reduces cost of priming.

APPENDIX 1: COMPARISON OF CARTRIDGED AND BULK EXPLOSIVES

Given below is the comparison between cartridged and bulk explosives for a mine using about 5000 tonnes of explosive per year. The comparison is drawn considering safety, requirement of magazines, vans and manpower etc.

	Cartridges	Bulk explosives
1. Safety	Explosive is handled, stored and transported. Thus potential hazard is always there.	Explosive handling, storage and transportation totally eliminated. The ingredients become explosive on being charged into the hole.
2. Explosive magazine size	For the consumption level of 5000 tons/year at least 200 tonnes storage magazines are needed.	Storage facility for only cast boosters needed. Cast booster used is 0.1% or 500 kg/year. Thus 500 kg magazine is adequate for cast booster and small capacity for DF and detonators.
Cost	The cost of magazine of 200 tonnes would be around 20 times.	The cost of magazine would be small.
Safety zone	Large safety zone is required which would be difficult to get. The safety zone for 4(50t) magazine has to be 820 m radius with 90 m distance between magazines. Thus for 4 magazines a safety zone of 1 km radius is needed.	A very small safety zone of 95 m radius is needed.
3. Explosive vans requirement	For charging 400 tonnes of explosives per month at least 3 vans of 10 tonnes capacity are needed for transportation of explosives from magazine to site.	A small explosive van of 500 kg capacity would be sufficient.
Cost	The capital cost of 3 vans is nearly ten times.	The capital cost of 500 kg van would be small.
Manpower	Three van drivers and 3 helpers would need to be employed.	One driver and one helper would be needed.
4. Blasting crew	The blasting crew comprising of at least 12 labourers and two supervisors would be needed for charging and stemming.	The blast crew would comprise of 2-3 workers and one supervisor.
5. Charging rate	For charging/stemming of 20 tonnes of explosives a full shift would be needed using 12 labourers.	For charging 20 tonnes about 3 hours are needed using 2-3 workers for stemming. The operation is fully mechanised.
6. Explosive	Explosive of fixed energy is available. The quality available in stock has to be used. Inventory of each product is required to be maintained.	A wide range of explosive energies are available. There are over 20 formulations to choose from. These can be made with minor adjustments on pump truck.
7. Explosive	The density of product can not be varied.	The density of product can be varied to suit the requirement over a wide range.
8. Coupling	The coupling is not full.	The coupling is full resulting in maximum transfer of energy from explosive to rock.
9. Cost	The cost is fixed.	The cost decreases as off take increases.
10. Pilferage	There is hazard of explosive pilferage.	Pilferage is not possible.

CHAPTER 5

Selecting suitable explosives

5.1 INTRODUCTION

Selecting proper explosive is an important part of blast design to assure a successful blasting operation. Explosive selection is dictated by economic consideration, by rock type and by blasting results required. A product be selected and used in such a manner that optimum fragmentation and displacement is obtained while giving lowest overall cost with adequate safety. Several factors need to be taken into consideration while choosing a suitable explosive for a given blasting operation. The objective of blasting operation will have important influence, whether blasting is for surface mining/quarrying or for underground metallic or non-metallic operations. The equipment available, size of fragmentation, amount of displacement, allowable damage to the remaining rock, water condition, adequacy of ventilation, atmospheric temperature, propagating ground, storage conditions, sensitivity considerations and explosive atmospheres, are to be considered along with cost of explosives and drilling. Besides these, initiating systems and charging techniques available are also to be considered. Adequate attention will be needed to match the explosives to the rock and its structural pattern. These all factors can be divided as:

- Economic factors;
- Explosives;
- Rock and blasting conditions;
- Blasting results.

5.2 ECONOMIC FACTORS

All operations try to use a blasting system where the cost of blasting operation is the least. Often many operators believe that the cheapest explosive be used. However, the choice of explosive will determine costs in the drilling, secondary breaking, loading, hauling and crushing subsystems, as well as blasting subsystem involving cost of explosives and their loading and initiating systems.

5.2.1 *Cost of explosives*

Generally ANFO has the lowest cost per unit of energy. Slurry costs range from slightly more than ANFO to about four times the cost of ANFO. Cheaper slurries or

emulsions are designed for use in large-diameter blast-holes and contain no high cost, high energy ingredients. They are low in energy per unit weight basis. The more expensive slurries are: a) those designed to be used in small diameters, and b) high-energy products containing large amounts of aluminium or other high-energy ingredients.

Cost of nitro-glycerine based explosives range from three to five times that of ANFO depending largely on the proportion of nitro-glycerine or other ingredients. Despite its excellent economics, ANFO is not always the best product for the job, because it has several shortcomings. ANFO has no water resistance, it has low specific gravity and under adverse field conditions it tends to detonate inefficiently. However, site conditions may make the use of higher energy explosives more attractive.

5.2.2 *Cost of drilling*

Under normal drilling conditions, the blaster should select the lowest cost explosives that will give adequate, dependable fragmentation. However, when drilling costs increase, typically in hard dense rock, the cost of explosive and the cost of drilling should be optimised through controlled experimentation. When drilling is expensive the blaster will want to increase the energy density of the explosive, even though explosives with high energy density tend to be more expensive. The energy density of a slurry/emulsion depends on the density and the proportion of high-energy ingredients such as aluminium, used in formulation. Because of the diverse varieties of slurries and emulsions careful consideration should be made for the use of high-energy products.

In small diameter blastholes, the density of ANFO may be increased up to 20% by high velocity pneumatic loading. The loading density (weight per meter of borehole) of densified ANFO cartridges is about the same as bulk ANFO because of the void space between the cartridge and the borehole wall. The energy density of ANFO can be increased by the addition of finely divided aluminium. Charge density of cartridged explosive per meter can be increased by slitting cartridges before dropping in the holes. The economics of higher energy density explosives slurry, or emulsion improve where the rock is more difficult to drill and blast.

5.3 EXPLOSIVES

With the availability of nitro-glycerine based explosives, ANFO, slurries, watergels, emulsions and heavy ANFO, it is really necessary to consider relative properties of all these types. In addition, various types of these products, further, make choice rather difficult.

5.3.1 *Explosive energy*

Explosive energy is the most commonly considered property when explosives are compared. It is generally agreed that most explosive energy is released in the form of Shock Energy (SE), Bubble Energy (BE) and Heat Energy. SE and BE being the useful components of the blast process. Many methods such as cratering, underwater shock and bubble energy determination, seismic method, and ballistic mortar, to name a few, have been proposed for the purpose of determining the blast potential of different explosives (see Section 3.4). Some of these tests favour the measurement of one particular component of energy released and can give a false impression of the true performance of the explosives in rock blasting situation. All these methods have difficulties associated with them, for example, in cratering, where small charges are fired in rock and crater of broken rock is measured, it is necessary to compare craters produced. Rock structural variations from shot to shot often are such that they mask differences between the explosives, especially in jointed/fractured rock.

In underwater test a known mass of explosive is detonated underwater in a large pond. The resulting seismic wave-forms are measured and on analysis the partitioning of SE-BE and the total energy released can be determined. However, the question is how they are partitioned to the rock?

The seismic method consists of shooting different weights of a standard explosive (ANFO) buried in the ground and recording the particle velocity versus time in three mutually perpendicular directions at different known distances from the shots. A calibration curve of peak resultant particle velocity versus standard charge weight of ANFO at a given distance can be established and one can determine the equivalent ANFO weight which is called the seismic strength. The material often used is sand and because of this the rock effects tend to be played down and bubble expansion effects predominate. This is viewed as a drawback by many.

Ballistic mortar involves firing a small known mass (10 g) of explosives in a 'cannon' that is able to swing like a pendulum. The extent of displacement of the cannon gives an indication of the kinetic energy imparted from the explosives. The small size of the sample used is a disadvantage. The large size of sample holder permits considerable decoupling of the test charge and for oxygen deficient explosives this results in considerable shift in the gas equilibria producing a different energy output. Slow reacting materials will not have time to react fully even if finely ground in some instances, and for blasting agent this method has little if any value.

Commonly used other methods such as double pipe, plate dent, Trauzl block, cylinder compression, copper tube, all suffer from one difficulty or the other.

Careful observation of blasting results indicates that they can be corrected with the total energy input/ton of rock, provided the blasts are properly designed. The most important characteristics of explosives for blasting are their useful energy outputs per unit weight and per unit volume i.e. their weight and bulk strength. In some instances these values can be calculated from the thermochemical energies and densities especially if the explosive is to be used in large diameter boreholes. By expressing the results relative to a standard explosive such as ANFO a relative weight strength and bulk strength can be obtained. However, in explosives containing high aluminium percentage such as AL/ANFO dry mixes and some of the slurries there is a problem of the energy trapped in the solid products of detonation and how it can be treated. Can it be considered to be available to do the useful work? In slurries there is the additional question of how one can consider water which is present? Different methods available for theoretical energies prediction, give results which may differ by 20%. Moreover, one method may rank a series of explosives in a different order from another method to the great confusion of the blaster.

If the thermochemical energies are discounted appropriately (such as computer models with non-ideal detonation) and modified then these values agree reasonably and closely with values obtained by the seismic method. Such values are found useful in comparing explosives.

5.3.2 *Storage consideration*

Storage is an important consideration in the selection process of explosives. Legal requirements for magazine construction are less stringent for blasting agents than for high explosives. Magazines for the storage of high explosives must be well ventilated and must be resistant to bullets, fire, weather and theft, whereas a blasting agent needs protection from theft only. Although this is not an overriding reason for selecting a blasting agent rather than an explosive, it is an additional point in favour of blasting agents. The shelf life of explosives is an important consideration. Slurries compared to NG based explosives have poor shelf life specially in some of the types escape of bubbles may reduce sensitivity.

5.3.3 *Sensitivity consideration*

Sensitivity reflects on safety and the dependability of an explosive. More sensitive explosives such as dynamites are somewhat more vulnerable to accidental initiation by impact or spark than blasting agents. Slurries are generally less sensitive to impact than nitro-glycerine based explosives. However, less sensitive explosives, all conditions being equal, are less likely to fire in the blasthole, upon accidental impact from a drill bit, e.g. a blasting agent. This is an important criterion while selecting explosives for small diameter explosives for construction, small quarries and underground mines.

Some explosives are affected by detonating cord used in the hole, therefore, sensitivity to cord also needs to be considered. Low energy detonating cords can be used while initiating these explosives.

5.4 ROCK AND BLASTING CONDITIONS

The type of rock being blasted and conditions in which explosives will be detonated during blasting will have considerable influence on the selection of explosives. The charge diameter, temperature of rock in the hole, water conditions all affect the selection.

5.4.1 *Charge diameter*

The dependability and efficiency of ANFO are sometimes reduced at smaller charge diameters, especially in damp conditions or with inadequate confinement. In diameters under 50 mm ANFO functions best when pneumatically loaded into a dry blasthole. When using charge diameter smaller than 50 mm, many blasters prefer the greater dependability of a slurry or nitro-glycerine based cartridge despite the higher cost because such economy on cost by ANFO can be lost.

At intermediate charge diameters, between 50 mm to 100 mm the use of nitro-glycerine based explosive is seldom justified because ANFO and slurries function quite well at these diameters. Slurries designed for use in intermediate charge diameters are somewhat cheaper than small diameter slurries and are more economical than nitro-glycerine based explosives. The performance of ANFO in a 100 mm diameter blasthole is substantially better than at 50 mm. When practical, bulk loading in intermediate charge diameter offers attractive economics.

In blasthole diameter larger than 100 mm, a bulk loaded ANFO, slurry or emulsion should be used unless there is some compelling reason to use a cartridged product. ANFO efficiency and dependability increase as the charge diameter increases. When the use of an emulsion or a slurry is indicated, many varieties function well in large diameters.

5.4.2 *Rock conditions*

Both drilling and fragmentation difficulties are experienced in hard, dense rock and jointed rock. Despite the controversy as to the importance of detonation velocity in rock fragmentation, there is evidence that a high velocity does help in fragmenting hard, massive rock. With cartridged dynamites, the detonation velocity increases as the nitro-glycerine content increases, with gelatine dynamites having higher velocity than their granular counterparts. Several varieties of slurries, and particularly emulsions have high velocities. However, it has been experienced that explosives having velocity of detonation of about 6000 m/s does not yield acceptable results in average conditions. In general for hard granites, quartzites and similar rocks, explosives equivalent to 90% gelatines yield better results. Whereas for medium-soft rock ANFO type explosives are more suitable.

In many operations with expensive drilling and fragmentation it is advantageous to use dense, high velocity explosive in bottom of the borehole and ANFO as top load.

5.4.3 *Water conditions*

A consideration in the selection of explosive is the ability of that explosive to be water resistant. The water resistance of an explosive can be provided by either the inherent chemical and physical properties of the explosive or its packaging. ANFO has no water resistance. It may, however, be used in blastholes containing water if some technique is followed to overcome this drawback. First, the ANFO may be packaged in a water-resistant polyethylene container. To enable the ANFO cartridge to sink in water, part of prills are pulverised and the mixture is vibrated to a density of about 1.1 g/cm^3. Of course, if a cartridge ruptures during the loading process, the ANFO will quickly become desensitised. In the second technique, the blasthole is dewatered by using a down-the-hole submersible pump. A waterproof liner is then placed into the blasthole and ANFO is loaded inside the liner before the water re-enters the hole. Again, the ANFO will quickly become desensitised if the borehole liner ruptures. The appearance of orange brown nitrogen oxide fumes upon detonation is a sign of water deterioration, and an indication that a water-resistant product or better external protection should be used. Emulsions are more water proof and do not depend on a package for water-resistance, this is due to the protective nature of the oil and wax membrane surrounding each particle. Slurries are gelled and cross-linked to provide a barrier against water intrusion and as a result, exhibit good water resistance.

Some explosives sink easily in muddy waters while others do not sink. Since slurries use air bubbles and manufacturers use methods which entrap large sized bubbles also, such slurries do not sink easily. Therefore, if water is encountered in the holes the products which sink easily be selected.

Some slurries/watergels which rely upon entrapped air in whole or in part, for sensitivity begin to lose detonation velocity under 15 m of water and may fail to detonate at deeper depths. Similar problems can occur with heavy ANFO. However, some emulsions can withstand high static pressure and can detonate satisfactorily. When nitro-glycerine based products are used in wet holes, gelatinous varieties are preferred, though, gelatines are slightly costlier. The higher cost however, is more than justified because of their increased reliability in wet blastholes.

5.4.4 *Propagating ground*

Explosive propagation is defined as the sympathetic transfer of detonation from one point to another. Although, propagation normally occurs within an explosive charge in blasthole, it may occur between blasthole through the ground. The probability of cross-propagation between blastholes is related to water saturation, highly jointed rocks, the type of explosive and with small burden/spacing dimensions. The effect of sequential delays are negated when propagation between holes occur resulting in poor fragmentation, failure to properly pull the round as well as in excessive vibration, airblast and flyrock. For instance in underground blasting, the entire round may fail to pull. The problem is more serious when using small blastholes loaded with nitro-glycerine based explosives. Small blastholes require small burdens and spacings, increasing the chance of hole-to-hole propagation, particularly when sensitive explosives are used. Water saturation and blasthole deviation compound the problem. When propagation is suspected the use of less sensitive product usually solves the problem. Straight nitro-glycerine dynamite is most sensitive commercial explosive followed by other granular dynamites, gelatines, capsensitive slurries and blasting agents, in decreasing order of sensitivity.

Another problem can occur when ANFO or slurry blasting agents are used at close spacings in soft ground. The shock from adjacent charge may dead press a blasting agent and cause it to misfire.

5.4.5 *Temperature*

Until the development of slurries, atmospheric temperatures were not an important factor in selecting an explosive. Many nitro-glycerine based explosives use low freezing oils which permit their use in the low temperatures. ANFO and slurries are not affected by low temperatures if priming is adequate. A potential problem exists with slurries that are designed to be cap sensitive. At low temperatures many of these products may loose their cap sensitivity, although they will still function well if adequately primed.

The effect of low temperature is alleviated if explosives are stored in a heated magazine or if they are in the borehole long enough to achieve the ambient borehole temperature. Except in permafrost or in extremely cold weather, borehole temperatures are seldom low enough to render slurries insensitive. If explosives are used near a fire zone, then special products are needed.

Under hot storage conditions above 32°C, many compounds will slowly decompose or change properties and shelf life will be decreased. Storage of ammonium nitrate blasting agents in temperatures above 32°C can result in cycling, which will affect the performance and safety of the products. Most dynamites, slurries and emulsions contain ammonium nitrate. Products which are stored over the winter or for a period during the summer will likely undergo some cycling. The effect of cycling on AN when isolated from the humidity in the air is that the prills breakdown into finer particles. The process of breakdown can continue until the density is no longer near 0.8 g/cm^3, but can reach density near 1.2 g/cm^3. The density increase can make the product more sensitive and it will contain more energy per unit volume. When the density is above 1.2 g/cm^3, ammonium nitrate will no longer detonate.

To further complicate the situation some cartridged blasting agents or those stored in bins may not sufficiently exclude humidity. After the ammonium nitrate has undergone cycling, the water resistant coating is broken and water vapour in the air condenses on the particles. As cycling continues water collects on the particles and the mass starts to dissolve. Recrystallisation into large crystals can occur when temperatures drop.

5.4.6 *Medical aspects*

Although, most explosives are oxygen balanced to maximise energy and minimise toxic detonation, some are bad from the stand point of fumes. Even with oxygen balanced products, unfavourable field conditions may increase the generation of toxic fumes, particularly when explosive without water resistance get wet. The use of plastic borehole liners, inadequate charge diameters, removal of a cartridged explosive from its wrapper, inadequate priming, or an improper explosive ingredient mix may cause excessive fumes. In areas where the efficient removal of detonation gases can not be assured (normally underground) ANFO should be used only in absolutely dry conditions. Most small-diameter slurries have very good fume qualities. Large diameter slurries have variable fume qualities.

Storage and use of NG-based explosives have headache producing effect. In the present day environment conscious society this also needs to be considered.

5.4.7 *Ability to maintain shape*

The ability of the explosive to maintain its original shape requirement changes for different applications. Sometimes explosive cartridges must maintain their original shape and at other times the cartridges must slump. For example, when blasting in cracked or broken ground, the explosive should not flow into the cracks causing holes to be overloaded, or when building of column is needed. On the other hand, in other applications such as bulk loading, explosive should flow freely and not bridge the borehole nor form gaps in the explosive column and sometime the slurry explosives cartridges are slit. Therefore, ability to maintain shape also becomes an important criterion in the selection.

5.4.8 *Explosive atmosphere*

Blasting in gassy atmosphere can be catastrophic if the atmosphere is ignited by the flame from the explosive. All underground coal mines are classified as gassy; some metal/ non-metal mines may contain methane or other explosive gases and many construction projects encounter methane. In such situation permissible explosives offer protection against gas explosions. Depending on the degree of gassiness of mines, corresponding class of explosives is to be used.

5.5 BLASTING RESULTS

Blasting results are measured in terms of volume of rock broken, degree of fragmentation, flyrock, airblast, ground vibration, displacement and damage to the remaining rock.

5.5.1 *Fragmentation required*

Production costs are directly related to the degree of fragmentation achieved. In hard, dense rock, the difficulties of achieving adequate fragmentation increase along with the expense of drilling. The importance of matching sonic velocity of the rock with the velocity of detonation of the explosive used is the subject of ongoing debate. However, there is evidence that velocity matching does increase fragmentation in hard, massive rock. With nitro-glycerine based explosives, the detonation velocity increases as the nitro-glycerine content increases, thus 90% gelatine having higher velocities than 60% gelatine. The 90% gelatine is more suitable for hard massive granite than 60% gelatine and gives adequate fragmentation. Several varieties of slurry and emulsions have high velocities.

The detonation velocity of ANFO is highly dependent on its charge diameter and particle size. In diameters of 225 mm or more ANFO detonation velocity will exceed 4000 m/s, reaching nearly 5000 m/s in a 375 mm diameter hole. These velocities compare favourably with velocities of most other explosive products. In smaller diameters below 50 mm the velocity is less than 2500 m/s. In these small diameters, the velocity may be increased to nearly 3000 m/s by high velocity pneumatic loading, which pulverises the ANFO and gives it a higher loading density.

In many operations with expensive drilling and difficult fragmentation, it may be advantageous for the blaster to compromise and use a dense, high velocity explosive in the bottom of the borehole and lower energy product as column charge.

5.5.2 *Special consideration*

Additional special considerations may be needed at the time of selecting explosives, depending on special nature of blasting operation. For wall control work decoupling properties may need to be considered. In bulk operations reliability and quality con-

Table 5.1 Comparison of explosives characteristics.

Characteristic	NG	Slurries/ Watergels	Emulsions	Cartridged powder	F.F. power	Lox
Safety	Fair	Very good	Very good	Good	Very good	Medium low
Weight strength	All ranges	Desired ranges	Desired ranges	Medium	Medium low	Medium low
Life	Long	Adequate	Long	Adequate	Short	Very short
Water resistance	Good	Very good	Excellent	Poor	Poor	Poor
Density	High (1.4)	Medium (1.1-1.2)	Medium (1.1-1.2)	Low	Low	Low
Low temperature	Not affected	Affected	Not affected	Not affected	Not affected	–
High temperature	Not recom-mended	Not recom-mended	Recom-mended	–	–	Not rec-ommended
Coupling	Poor	Good	Good	Poor	Excellent	Poor
Ingredients	Explosive	Non-explosive sometimes explosive	Non-explosive	Non-explosive	Non explosive till mixed	–
Medical aspects	Poor	Fair	Fair	Good	Fair	Fair

trol may be another factor to be considered. Loading procedure, gelling rate, etc. may also influence the results. Loading procedure such as for bulk products and for pneumatic loading may also influence performance.

Products for dimensional stone blasting are the ones which provide decoupling and which are specially formulated. These charges are 11 mm, 17 mm and 19 mm diameter and are formulated with special ingredients as there is possibility of these explosives becoming desensitised due to channel effect. Products for hotholes need to have characteristics so that in high temperature and in confinement detonation does not take place.

Products for overcasting of overburden by use of explosives should have higher bubble energy so that more displacement can take place.

Controlled in-the-mine research with careful cost analysis is the best way to determine the optimum explosive product to use. Table 5.1 gives a comparison of the characteristics of different explosives.

CHAPTER 6

Initiating devices

6.1 INTRODUCTION

Commercial explosives (and blasting agents) are designed to be relatively stable for safe usage, transport, storage and manufacture. A powerful localised shock or detonation is required to initiate commercial explosives. This is achieved by use of an initiating device such as a detonator. In this chapter various initiating devices are described.

An initiation system consists of three basic parts:

1. An initial energy source;
2. An energy distribution network that conveys energy into the individual blastholes;
3. An in-the hole component that uses energy from the distribution network to initiate a cap-sensitive explosive.

The initial energy source may be electrical, such as a generator or condenser-discharge blasting machine or a power line used to energise an electric blasting cap, or a heat source such as a spark generator or a match. The energy conveyed to and into the individual blastholes may be electricity, a burning fuse, a high-energy explosive detonation or a low energy dust or gas detonation.

There are basically two methods of initiation, electrical and non-electrical (Fig. 6.1). Electrical initiation systems utilise an electrical power source with associated circuit wiring to convey electrical energy to the detonators. Non-electric initiation systems utilise various types of chemical reactions ranging from deflagration to detonation as a means of conveying the impulse to non-electric detonators, or as in the case of detonating cord, it is the initiator. Different types of devices are available for surface initiation or for in the hole initiation. Several different types are needed depending on the type of blasting operation.

6.2 ELECTRICAL INITIATION SYSTEMS

An electrical initiation utilises an electrical power source with an associated circuit to convey the impulse to the electric detonator which in turn fires and initiates the explosive charge.

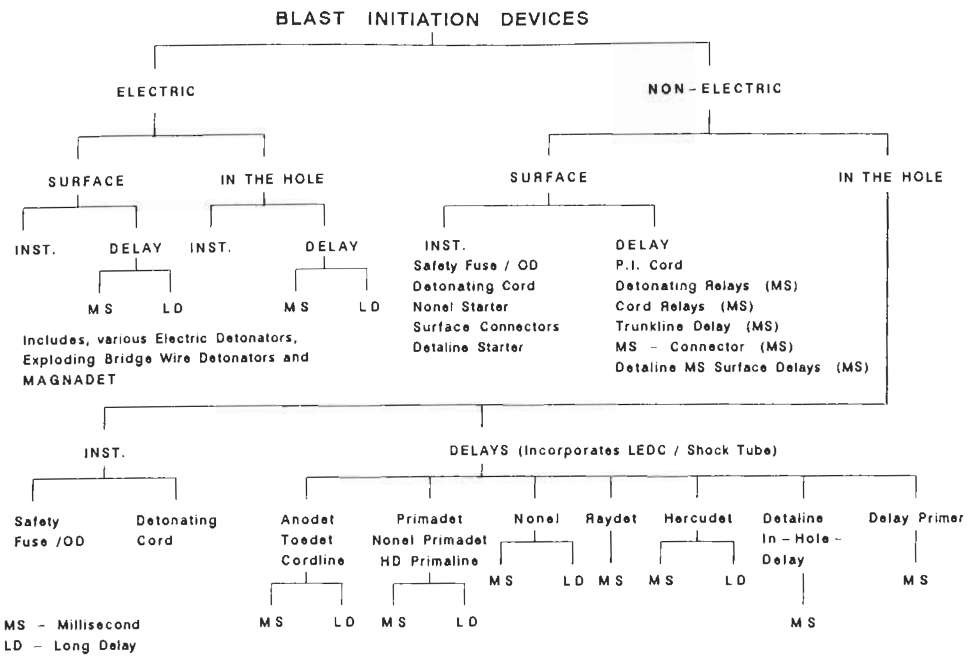


Figure 6.1 Various methods of initiation.

6.2.1 *Detonators*

All detonators (also called blasting caps) consist of a metal tube or shell 6.5 to 7.5 mm in diameter (outer) of varying length. Normally detonator shells are made of aluminium. In case detonators are to be used in coal mines, a copper tube or a steel tube (copper or zinc coated) is used. At the closed end of the tube an explosive charge of either a single initiating explosive (as mercury fulminate) or a combination of secondary explosive (base charge) and an initiating explosive charge (top charge) is placed. The charges are compacted to give the desired strength and also ensure that they do not fall out while handling. The quantity of charge used must be adequate to reliably initiate the high explosive. Either No. 6 strength or No. 8 strength detonators are used. No. 6 strength detonator was originally the strength obtained from 1 g of a mixture containing 80% mercury fulminate and 20% potassium chlorate. This standard has been transformed into its equivalent in lead azide, lead styphnate and aluminium powder (A.S.A.) used in conjunction with PETN, RDX, Tetryl and other similar explosives. No. 8 strength detonators have explosive mixture equivalent to 2 g of 80% mercury fulminate and 20% potassium chlorate. It is necessary to ensure that the detonators are capable of being stored over long periods under varying climatic conditions and they should function reliably and safely. Detonators are classified as plain detonators (see Section 6.3.1) or electric detonators. In the electric detonators (Fig. 6.2) the electrical current to the detonator is supplied from the power source through the circuit wiring to the detonator by means of two leg-wires that are internally connected by a small length of high resistance bridge wire. The electrical energy is converted into heat energy on passing the firing current through bridge

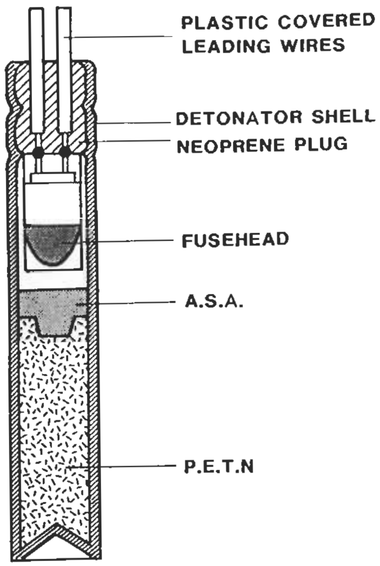


Figure 6.2 Electric detonator.

wire. The heat energy ignites the pyrotechnic that surrounds the bridge wire on the match head assembly. The resulting flash or flame ignites initiating charge or the delay element, these in turn set off base charge. The leg wires on electric detonators are made of either iron or copper. The leg wires enter the detonator through the open end. A rubber or neoprene plug seals the opening and only the leg wires pass through the plug. This prevents contamination by foreign material or water. Electrical detonators may again be classified as instantaneous and delay detonators.

6.2.2 *Instantaneous detonators*

Instantaneous detonators fire within a few millisecond (less than 5 ms) after they receive the current. The construction of these detonators has been described above. These are used when all the holes are to be fired simultaneously.

6.2.3 *Delay detonators*

For most blasting operations it is an advantage to have the various holes fired in a predetermined sequence with specific time intervals between detonators. The most notable advantages of delay detonators are:

- Reduced vibration, airblast and flyrock;
- More predictable throw (amount and direction);
- Reduced backbreak and overbreak, with working faces left in an improved condition.

In delay detonators, a delay element is inserted between the electric fusehead and the ASA/PETN charge in the detonator (Fig. 6.3). This delay element consists of a column of slow burning composition contained in a thick-walled metal tube. The length and composition determine the amount of delay-time introduced into the detonator.

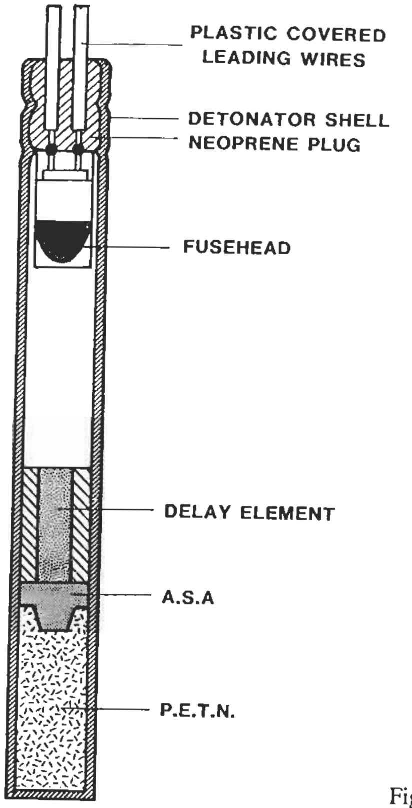


Figure 6.3 Delay detonator.

There are three basic delay series:

1. Slow or those which have half-second interval between successive numbers (also called long delay detonators);

2. Fast or those which have millisecond interval between successive numbers (also called short delay detonators);

3. Coal mine delays for use in underground coal mines (also called coal delay detonators).

The long delay detonators are used mainly in tunnelling, shaft sinking, and in underground metal mines. There is standard time increment of 0.5 to 1.0 seconds introduced between explosions of successive numbers in the range.

The short or millisecond delay detonators are the most commonly used delays. In millisecond detonators the delay intervals are 25 to 50 ms in the lower period and longer in the higher periods.

Coal mine delays are specially designed and manufactured for use with permitted explosives, they must not cause ignition of the methane gas and coal dust present in the mines. Since only electric initiation systems are permissible in underground coal mines delays are available only with electric initiators. Delay intervals are from 50 to 100 ms, with instantaneous caps being prohibited in some countries. Coal mine delay detonators must pass the statutory tests for using it in methane gas and coal dust atmospheres.

Hazards encountered when using detonators

Detonators necessarily contain small quantities of very sensitive explosives as well as the means of initiation of these explosives. Improper handling and use can present totally unacceptable hazards and it is necessary to know the extent of hazard and its methods to eliminate it. All detonators whether electric or otherwise, are susceptible to heat, friction and shock, and it is therefore necessary to avoid taking detonators to places where they may be submitted to such effects. Precautions must be taken to ensure that detonators are carried from storage to the place of use by routes which do not include workplaces where the detonators may be exposed to heat, impact or friction. It is essential that they be carried in locked containers which are sufficiently firm and robust to prevent crushing, should they be involved in an accident.

There is a risk of accidental initiation of detonators wherever extraneous electricity is encountered, and special precautions have to be taken. The risks are from lightning, stray electricity from machinery or cables; induced electric currents from high voltage power lines, static electricity from non-conductive clothing, pneumatically charged ammonium nitrate, radio or other magnetic signals. The special types of detonators are available to overcome or reduce these hazards.

6.2.4 *High intensity detonators*

High intensity detonators are similar in construction to standard electric detonators except that these detonators have fuse heads which are activated only by very high electric currents. The use of such detonators requires special exploders, and operators need special training in the techniques of use. These are known by various names in different countries and the characteristics of some of these detonators are given below:

Type 1. 'U' Type of Germany or 'SF' type of Austria. Electric energy required to set them off is as high as 16 millijoules and is about eight times that required for normal instantaneous detonators.

Type 2. 'HU' Type of Germany or 'VA' Type of Sweden. Generally the energy required is of the order of 100 to 120 millijoules and will not fire at anytime whenever the energy supplied is less than 80 millijoules.

Type 3. Some detonators like the polex developed by Schaffler in Austria require as high as 3600 millijoules and are used under conditions where extremely high electrostatic fields are met with.

6.2.5 *Electronic detonators*

Electric delay detonators use pyrotechnic delay elements and such delays have limitation in terms of accuracy, all the manufacturers indicate that variation of about 10% from the nominal delay period leading to possibility of overlap. Electronic delay detonators have been developed and successfully field tested recently. These detonators are based on micro-chip technology. The electronic detonator is basically a device which stores electrical energy for a certain time and then delivers that energy as a sharp pulse at precise time to a conventional fusehead.

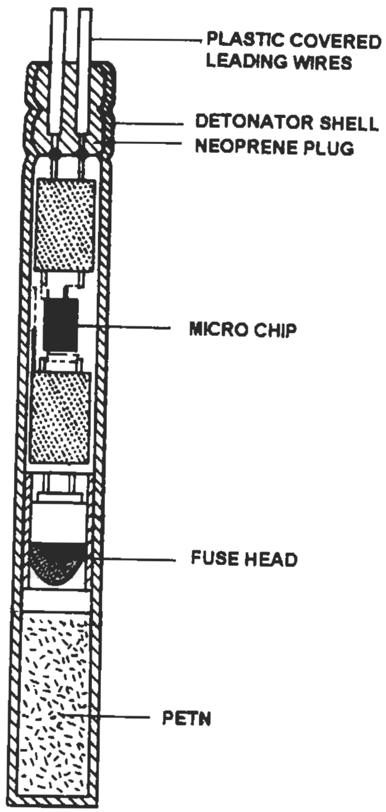


Figure 6.4 Electronic detonator.

6.2.6 *The Magna system*

In the Magna system, the lead wires of conventional electric detonators are coupled to ferrite rings (torroids) in such a way as to act as secondary windings of a transformer (Fig. 6.4). The primary windings of the transformer coupling is effected by passing a length of plastic covered connecting wire once through the centre of each torroid.

The Magnadet transformer-coupled detonator assembly provides significantly improved safety properties to the basic conventional electric detonator assembly. The system is frequency selective since the torroid can only be activated by AC power of 15,000 Hz or greater, a frequency range which is not often encountered. Special exploders tuned to suitable frequencies are required, to produce in the circuits, currents which will fire the detonators.

The Magna system is therefore immune to mains currents and to stray direct currents. There is enhanced protection against RF pick-up in the primary circuit. The way in which the detonators are independently coupled makes the system immune to earth leakage since the circuit is completely insulated, offering also added protection against static electricity hazards.

The concept of Magnadet has also been used in the design of the Magna Primer for open pit and large underground blasting.

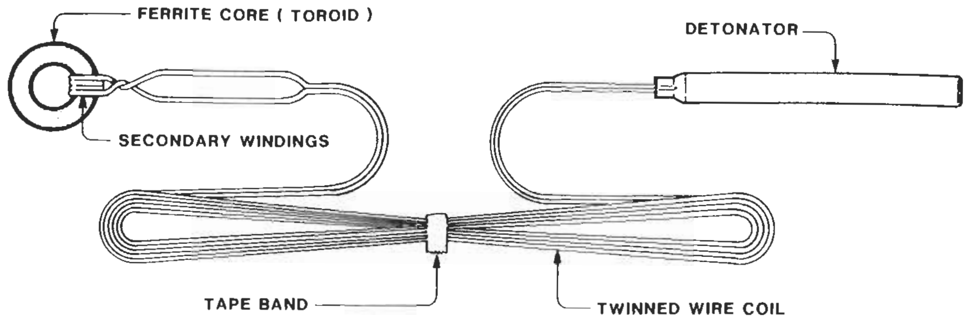


Figure 6.5 Magnadet assembly.

6.2.7 *Saf-t-Det*

The Saf-t-Det resembles a standard electric detonator but has no base charge (Fig. 6.5). A length of 10 g/m or less detonating cord is inserted into a well to act as base charge just before the primer is made up. The device is similar to an electric blasting cap in regard to required firing currents and extraneous electricity hazards. The system dependably initiates detonating fuse in the range of 1 to 10 g/m, nonel, etc.

6.2.8 *Seismic detonators*

In seismic exploration work, electric detonators of special type are needed. The main characteristics, which make the seismic detonators different from the normal instantaneous electric detonators, is the extremely small reaction time, which the electro-explosive device (bridge wire and match head composition) has. These detonators are expected to have a reaction of less than 1 ms i.e. they must go off within one millisecond from the time the electrical energy is supplied to the fusehead. The firing current supplied to the detonators is therefore very high in the region of 10 amp. As a result of use of seismic detonators highly improved signals are obtained and significant lessening of ghost echo problem is experienced. Seismic electric detonators withstand high hydrostatic pressures.

6.2.9 *Submarine electric detonators*

Submarine electric detonators are specially manufactured to prevent water under high pressures from reaching the detonating composition. They are tested to withstand a pressure of about 1.37 MPa i.e. 135 m head of water, for four hours.

6.2.10 *Electric blasting circuits*

In order to fire electric detonators, they must be connected together in a firing circuit and energised by a power source. There are three basic types of blasting circuits (Fig. 6.6):

Series: commonly used for small number of detonators,

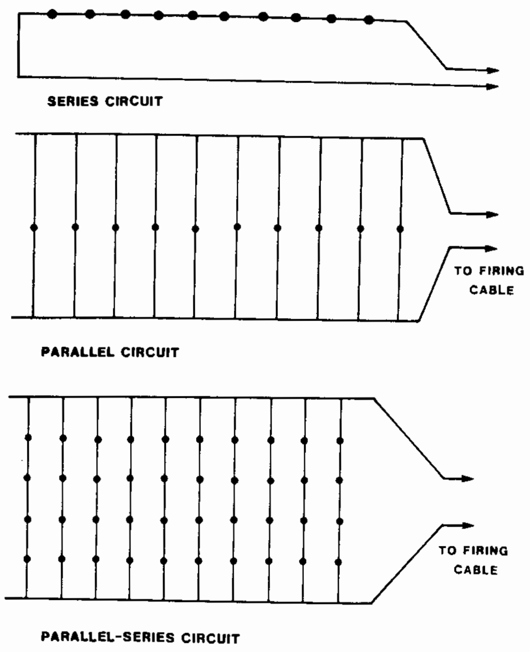


Figure 6.6 Types of blasting circuits.

Parallel: most infrequent, most difficult,
Series/parallel: most common.

In *series* circuits all the detonators are connected consequently to provide a single path for the current; one leg of one detonator is connected to one wire from another detonator, the other from the second detonator to one from the third and so on. The two free ends are connected by lead wires to a blasting machine or other power source. The series circuit is recommended because of its simplicity.

In *parallel* circuit one wire from each detonator is connected to one side of the circuit and the other wire to the other side of the circuit.

Parallel circuits are not recommended for surface blasting but are often used for large development heading, tunnels or shafts where charges are close together and the leg wires may be connected with minimal splicing of extra wire to bus bar. Power lines are usually employed to fire parallel circuits because of high amperage required and the time for which it must be sustained.

In this type of circuit it is difficult to check the circuit for broken wires or faulty connections. Secondly because the available current is divided by the number of caps in the circuit, power line must often be used to provide adequate current for large parallel circuits, which has its own problems.

In *series/parallel* circuits the caps are divided into a number of individual series. Each series should contain the same number of caps or the same resistance to assure even current distribution. The leg wires of the caps in each series are connected consequently. It has the advantage that a number of electric detonators can be fired with a reasonable power requirement and also the whole circuit, as well as individual series, can be tested with a blasting circuit tester.

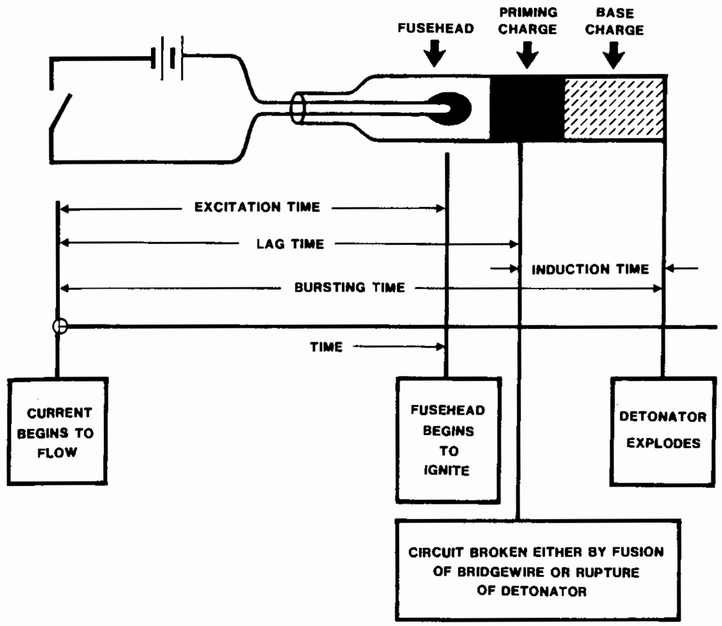


Figure 6.7 The sequence of events in firing electric detonators.

6.2.11 *Firing characteristics of electric blasting circuits*

As it is often difficult to understand why the current needed to fire series of detonators should be greater than that required for a single electric detonator, the principles underlying the firing of a number of electric detonators simultaneously and in series will be briefly described. Consider a series of electric detonators to which a uniform direct current is suddenly applied. Before any one detonator can fire, the fusehead must be traversed by the current for a certain minimum period of time – usually of the order of a few milliseconds – during which the bridge wire heats up to a temperature at which the sensitive composition of the fusehead ignites and so fires the detonator. This minimum time is termed the ‘excitation time’ (Fig. 6.7). In practice the excitation time will vary slightly from one detonator to another, owing to inevitable small manufacturing variations.

After the fusehead has received this minimum dosage of current there will normally be a further small time lapse before the ignition has spread through the fusehead and communicated to the detonator composition itself. Thus, the total time taken before the electric circuit through the detonator is finally broken will be rather longer than the excitation time. The time from the first application of current to the rupture of the circuit (whether caused by the bursting of the detonator or by fusion of bridge wire) is referred to as the ‘lag time’.

The interval between the first application of the current and the detonation firing is called the ‘bursting time’. The bursting time is either equal to or slightly greater than the lag time. The difference between the lag time and the bursting time is known as the ‘induction time’.

For all the electric detonators in the circuit to fire successfully, the shortest lag time of any of them must exceed the longest excitation time of any one of them. In

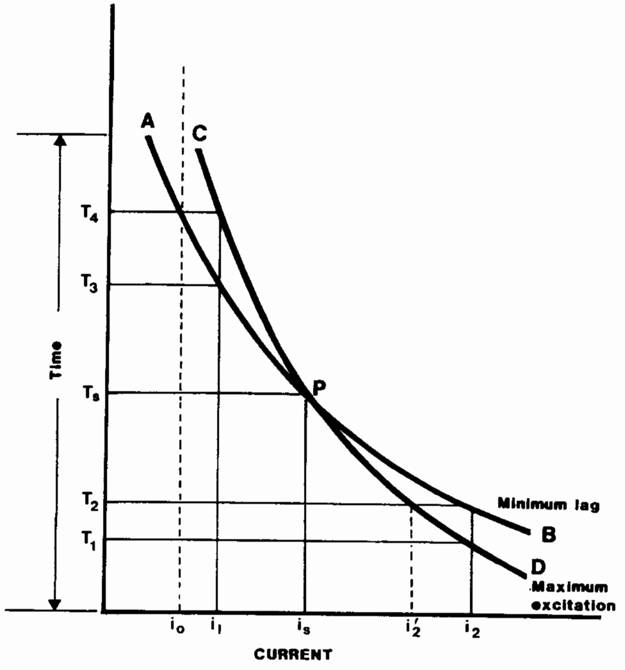


Figure 6.8 Theory of series firing.

other words, every one of the electric detonators must receive the full quota of electric energy to cause its ignition before any single one of them has completed its lag time and so broken the circuit.

The firing characteristics of electric detonators can be shown on a diagram as in a Figure 6.8. Here the line APB shows the relation between current and minimum lag time, while CPD shows the corresponding relation between current and maximum excitation time for a particular type of commercial electric detonator. Now consider the current i_1. From the curve it will be seen that the most rapid detonator will break the circuit in T_3 ms whereas the least sensitive detonator requires the current to flow for at least T_4 ms to enable it to fire. At a higher current i_2, however, the excitation time T_1 is less than the time T_2 which is allowed by the most rapid detonator and hence the least sensitive detonator in the circuit will have ample time to receive its full quota of current before the circuit is broken at T_2.

Somewhere between the two currents values i_1 and i_2 corresponding to the cross-over point P, there is a value i_s which will be minimum series firing current for the series considered. In practice i_s is always higher than minimum firing current i_o for a single electric detonator of the same type and this is the reason why series firing requires a higher current than single firing.

The diagram also makes clear why it is essential to avoid shunting of any of the detonators in a round. In particular, for a series firing current i_2, it would only require the current in one of the detonators in the round to be decreased to i'_2 in virtue of its shunt, for its excitation time to rise to T_2, the minimum lag time of the remaining detonators; under these conditions the risk of misfires is very high.

6.3 NON-ELECTRIC INITIATION SYSTEMS

The first initiation systems used were safety fuse and ordinary detonators and over the years it was considered that electric detonators were much safer, had better control on the timing of blast and were less hazardous. However, electrical firing involves problems of detonators and extraneous electricity. In addition there are severe demands on the exploders. Several non-electrical firing systems have become popular during recent years.

6.3.1 *Safety fuse and plain detonators*

Use of ordinary detonator and safety fuse is the oldest explosive initiation system. Safety fuse has a core consisting of granulated gunpowder. The continuous core is covered with molten bitumen and in some kinds of fuse an extra coating of high polymeric material is given to make it highly waterproof.

The gunpowder used in safety fuse is broadly of two types: one with low nitrate content and the other with high nitrate content; the former contains about 65% of potassium nitrate while the later contains about 75%. The rest is made up of sulphur and charcoal properly balanced to get a uniform and reliable rate of burning.

The most important requirement of the safety fuse is that it should have a uniform rate of burning. The burning speed of the most common type of safety fuse is 100 s/m. The tolerance allowed generally is about ±10%. It should also show no side sparking and the gases evolved during process of burning should vent through the textile layers uniformly.

Both dampness and high altitude will cause the fuse to burn more slowly. Fuse should be test burned periodically so that the blaster can keep a record of its actual burning rate. 'Fast fuse' has been blamed for blasting accidents but the fact is that

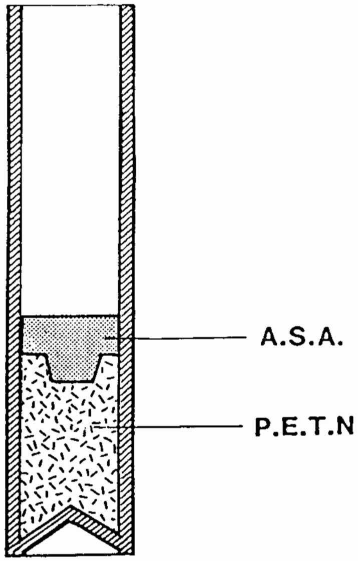


Figure 6.9 Ordinary detonator.

this rarely if ever occurs. However pressure on the fuse may increase its burning rate. To guard against water deterioration, it is good idea to cut-off a short length of fuse immediately before making cap and fuse assemblies.

Plain or ordinary detonator is the earliest of the modern blasting detonators which provide non-electric method of initiating explosive charges, when used in conjunction with safety fuse. Figure 6.9 shows the general construction of such a detonator. The detonator contains two types of charges (sometime three types): the igniting charges and the base charge. The igniting charge ensures flame pickup from the safety fuse, which in turn detonates the base charge and thus detonates the explosive charge being primed with the detonator. Plain detonators store well for long periods.

To assemble an ordinary detonator and fuse, the fuse is carefully cut squarely and inserted into the cap until it abuts against the explosive charge in the cap. The cap is attached to the fuse with an approved bench or hand crimper, and never with teeth or pliers. When crimping the cap, care should be taken so as not to crimp the zone containing the powder. Matches, cigarette lighters, carbide lamps or other open flames are not suitable for lighting fuse. Hot-wire lighters, lead splitters or igniter cord are controllable ignition systems.

6.3.2 *Plastic igniter cord*

Plastic igniter cord is an incendiary cord developed for lighting a number of safety fuses in a series via beanhole connectors. Two types are available – fast and slow. When cord is ignited an intense flame (which will ignite the black powder core of safety fuse) passes at a uniform rate along the cord. Figure 6.10 shows a typical cap, fuse, beanhole and igniter cord assembly. The igniter cord is available 'fast' or 'slow' type and have excellent water resistance and their burning speeds are reliable and consistent even under adverse conditions. Beanhole connector for plastic igniter cord consist of aluminium tube closed at one end and containing a plug of incendiary composition. An oval aperture is cut through the tube and the incendiary composi-

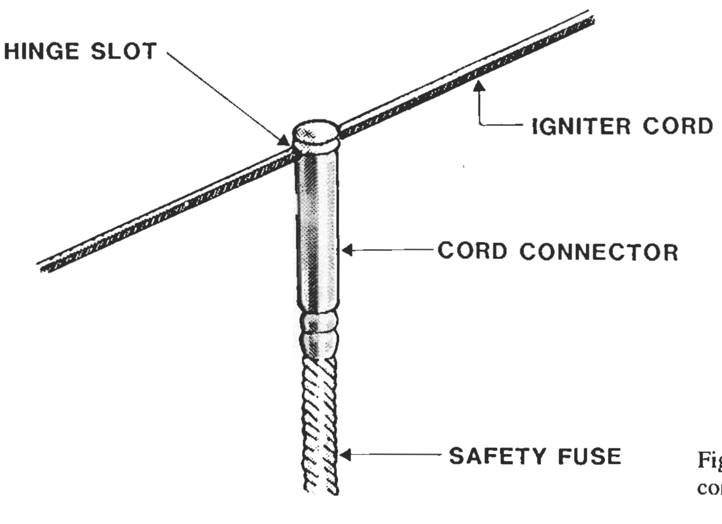


Figure 6.10 Method of connecting igniter cord.

tion, so that a loop of plastic igniter cord can be inserted. The safety fuse is inserted into the open end of the tube.

The primary hazard of using safety fuse is the tendency of blasters to linger too long at the face, making sure that all the fuses are lit. To guard against this, regulations specify minimum burning times for fuses, depending on how many fuses one person lights. Two persons are required to be at the face while lighting fuse rounds.

If a person lights only one fuse, the minimum burning time is 2 min; for 2 to 5 fuses minimum time is 2-2\3 min; for 6 to 10 fuses the minimum time is 3-1\3 min; and for 11 to 15 fuses the minimum time is 5 min. One person may not light more fuses in a round. Kinks and sharp bends in the fuse should be avoided because they may cut off the powder train and cause a misfire.

The cap and fuse has two major inadequacies; inaccurate timing and poor safety. The first one results in poor fragmentation, a higher incidence of cut off and less efficient pull of the round. All of these factors nullify the small cost advantage derived from slightly lower cost of the system components.

The poor safety record attained by cap and fuse is even more serious drawback. It is the only system that requires the blaster to activate the blast from hazardous location and then retreat to safety. The use of igniter cord rather than the individual fuse alleviates this problem.

Cap and fuse does have the advantage of lack of airblast, no charge disruption, somewhat lower components costs and protection from electrical hazards.

6.3.3 *Detonating cord initiation*

As an alternative to electric blasting, detonating cord initiation has been used for many years. It is mainly used in multihole blasting and when detonated, has the initiating energy of a detonator at all points. Generally in mining practice, one line of detonating fuse is used as a trunkline, from which a number of branches are drawn, each of which leads into a hole containing explosive to be blasted.

The most commonly used detonating cord (Fig. 6.11) which has an explosive charge of nominally 10 g of PETN per meter run. The diameter of this cord is nominally 4.65 mm and it has a breaking strength in excess of 100 kg. Depending upon the end use, the tensile strength can be altered.

Other detonating cords containing explosive charges in the range of 3.5 g/m to 80 g/m are available. The labels 'high' and 'low' in this context mean containing a higher or lower explosive content than the standard detonating cord.

The most commonly used High Energy Detonating Cord (HEDC) is one containing 40 g/m explosive charge having a diameter of 7.0 mm. The breaking strength is in excess of 100 kg. HEDC is used to initiate insensitive explosive such as ANFO by side initiation. Cords of 80 g/m to 150 g/m are occasionally used as a substitute for explosive cartridges in very sensitive or small, controlled blasting jobs. The Low Energy Detonating Cord (LEDC) is used where surface initiation is desired but where side initiation is not desired. The explosive column has to be chosen carefully to produce this effect. Low energy cord is not to be used for surface trunklines, as it will not readily propagate across joints in the cord. Cords having nominal explosive charge of 3.5 g/m up to 6 g/m may be used in small diameter holes, for secondary blasting and for initiating LEDC systems described later.

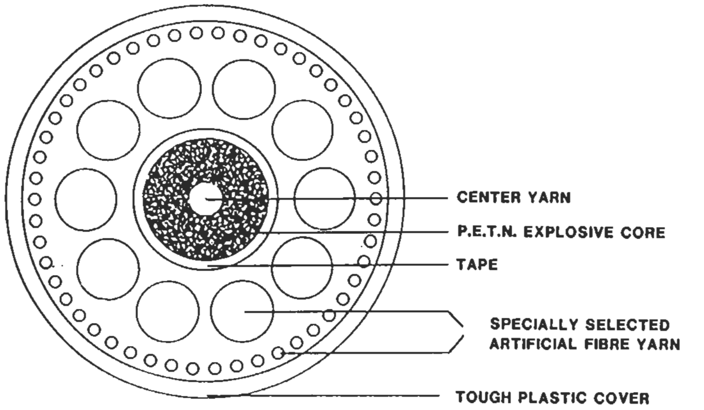


Figure 6.11 Structure of detonating cord.

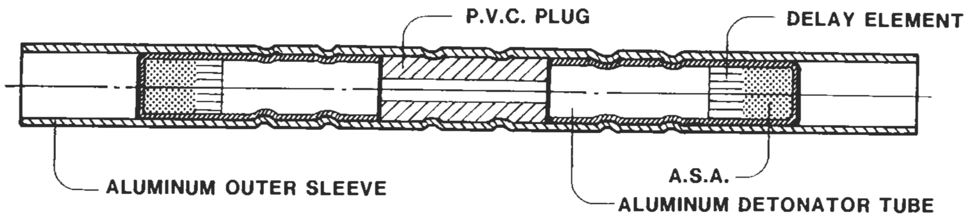


Figure 6.12 Detonating relay.

6.3.4 *Delay connectors*

Millisecond delay surface connectors are used for delaying detonating cord blasts. To place a delay between two holes, the trunkline between holes is cut and the ends are joined with a delay connector. In one type, called Detonating Relays, consist of a long aluminium tube with two mini delay detonators on either side and having an attenuator in the centre. Openings at either end are provided to receive and crimp the cut ends of the detonating fuse. The construction of one such connector is shown in Figure 6.12. Another type of delay connector is a plastic assembly containing a delay element. At each end of the element is an opening into which a loop of trunkline can be inserted. A tapered pin is used to lock the trunkline cord into place. Delays of 15 ms, 17 ms and 25 ms are available.

6.3.5 *Delay primers*

Delay primers have been used in the USA. The delay primer (Fig. 6.13a) basically consists of a cast explosive booster within a rugged plastic housing. The plastic housing on one side contains a sliding tunnel/hole for threading the detonating fuse. The hole is placed in such a way that the detonating cord does not initiate the cast explosive. A delay insert is provided by means of which the detonation of the detonating fuse is transferred to explosive booster after providing a delay. The insert consists of a heavy walled tube with a sensor on one end and an aluminium shell de-

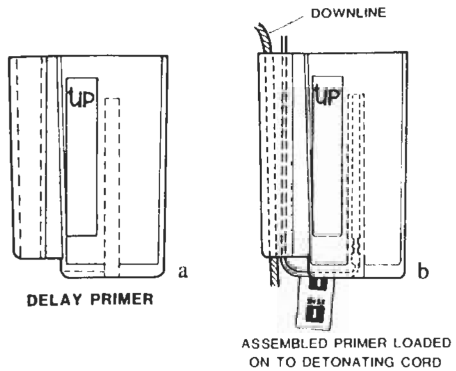


Figure 6.13 Delay primer.

lay cap at the other. The plastic housing is designed for fast easy connections and loading. Figure 6.13a gives the constructional details.

For loading boreholes an assembled delay primer is slid on a single detonating cord downline at the required position. One can load several primers on a single downline in a borehole. In case of deck charging also there is no need for separate downline for each deck. Figure 6.13b gives the charging method while using delay primers.

6.3.6 *Field applications of detonating cord*

After the primer has been lowered to its proper location in the blasthole, the detonating cord is cut from the spool. About 0.75 to 1 m cord should extend from the hole to allow for charge settlement and tying into the trunkline. When the entire shot has been loaded and stemmed, the trunkline is laid out along the path of desired initiation progression. Trunkline-to-trunkline connections are usually made with a square knot. A tight knot, usually a clove hitch, a half hitch or a double-wrap half hitch is used to connect the downline to the trunkline. Any excess cord from the downline should be cut off and disposed. The cord lines should be slack, but not excessively so. If too much slack is present, the cord may cross itself and possibly cause a cut off. Also, if the lines are too tight and form an acute angle, the downline may be cut off without detonating.

Two of the primary advantages of detonating cord initiation systems are their ruggedness and their insensitivity. They function well under severe conditions such as in hard, abrasive rock, in wet holes and in deep, large diameter holes.

The system is not susceptible to electrical hazards, although lightning is always a hazard while loading any blast. Detonating cord is quite safe from accidental initiation while initiating cap or delay connectors are attached. Available delay systems are extremely flexible and reasonably accurate.

There are several disadvantages that may be significant in certain situations. Systems employing only surface connectors for delays present the hazard of accidental initiation by impact.

Detonating cord trunklines create a considerable amount of irritating, high-frequency airblast (noise). In populated areas the cord should be covered with fine material such as drill cuttings.

The means of checking the system is visual examination. In hot holes standard cord containing PETN explosive be replaced by cord containing explosives such as RDX. Detonating cord downlines present the problem of charge or stemming disruption.

6.3.7 *Redundant system*

Redundant system is a non-electric blast initiation system which employs multiple signal paths to blast holes in order to accomplish reliable hole firing and avoid surface delay shut downs. Redundant system provides a bi-directional system with two or more paths of initiation to each borehole.

A redundant trunkline delay (RTD) consists of a length of signal tubing with delay detonators inside block connectors (RTD blocks) at both ends. This permits firing from either end. The RTD block accepts the signal from in-hole delays. The system provides technique to the blast designers to maintain desired delay sequence even though a cut off of path requires the initiation signal to follow another path to fire the hole.

6.3.8 *Low energy detonating cord (LEDC) delay systems*

This is an integral system consisting of two devices combined in such a way that the detonation from the low energy cord starts a delay detonator. Thus, a sequential firing mechanism is introduced into the non-electric circuit while retaining the advantages of the precise short delay blasting of electric circuits.

The advantages of the system can be summarised as follows:

- Eliminates concern for electric hazards of premature or extraneous electrical currents;
- It is immune to premature initiation from static electricity that generates from pneumatic loading operations or from radio frequency energy.

The following LEDC systems are available:

- Primadet system of Ensign Bickford;
- Detaline system of DuPont;
- Anodet and Toedet System of Canadian Industries Limited.

Primadet delay system

The system uses a LEDC called Primaline containing 0.65 g/m PETN. An assembly called Primaline Primadet consists of a length of the above cord crimped to a standard instantaneous or delay blasting cap (Fig. 6.14). It is available in both millisecond and long delay periods. In principle, the system utilises a detonation impulse via the Primaline to ignite the delay element which in turn activates the explosive charge in the detonator. The primaline which conveys the initiation impulse to the delay element is itself initiated by a detonating cord trunkline in the surface.

It is important to avoid using the primaline as a downline for sensitive explosive charges such as NG based types, since it initiates such a column of explosive – perhaps imperfectly and cause it to detonate instantaneously rather than in delayed order. Primadet delays can therefore be used effectively only in charges of low initiation sensitivity such as blasting agents (e.g. ANFO type or non-cap-sensitive slurries/emulsions).

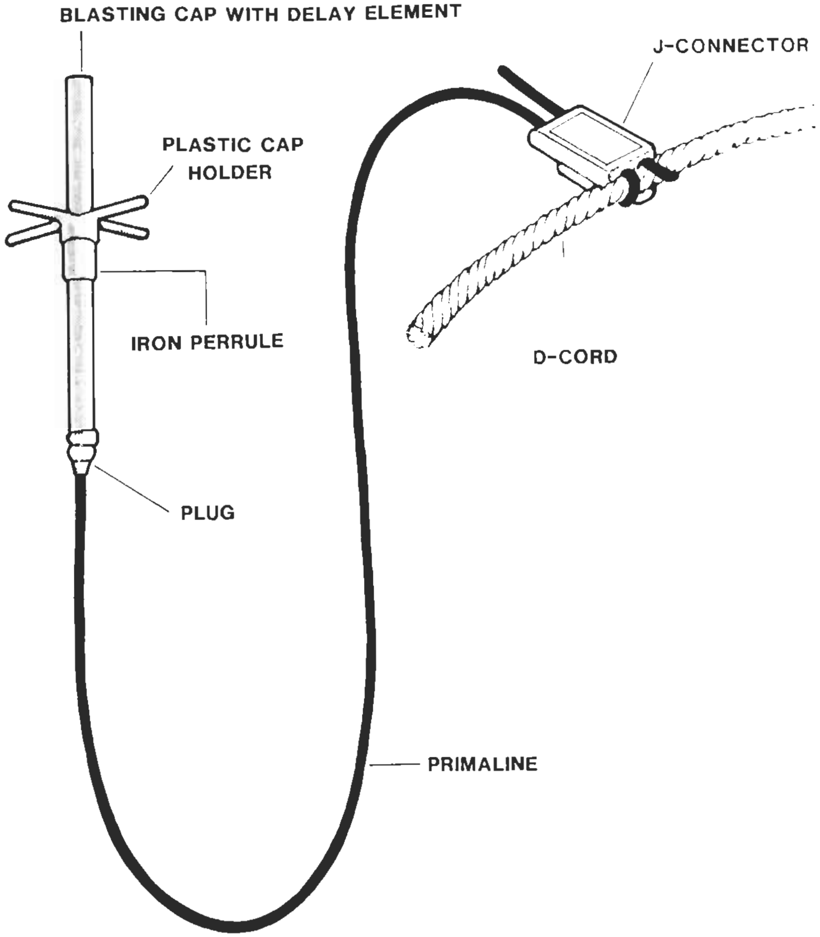


Figure 6.14 Primadet delay.

Detaline system

DuPont's Detaline system utilises low energy detonating cord. It functions similarly to conventional detonating cord systems except that the trunkline is low in noise, downlines will not disrupt column of explosive, it will not disrupt the column of explosive except NG based, and all connections are made of connectors rather than knots. The system consists of Detaline cord, Detaline starter, Detaline MS surface delay, Detaline MS in-hole delay.

Detaline cord has 0.4 g/m PETN core protected within textile fibres and covered by a seamless, tough, abrasion resistant outer plastic jacket. This is extremely insensitive to mechanical impact. A No. 8 strength detonator will not reliably side initiate the cord and also detonation will not propagate through the knot, which is why connectors are required. To splice a line or to make a non delayed connection a Detaline starter is required. The body of the starter is shaped much like a clip-on detonating cord millisecond delay connector, except that the starter is shaped like an arrow to show the direction of detonation.

Anoline and Anodet system
This system is particularly designed for underground blastholes 25 to 64 mm in diameter where ANFO is pneumatically loaded. The Anodet system consists of a special high strength detonator, factory crimped to a length of Anoline, which is connected to the detonating cord trunk line by a J-connector. Where necessary Anodet booster can be used. To locate the primer centrally in the blasthole, a plastic cup holder is available. Anodet delays are available in short or long periods. Anodet short delays are of the range of 30 delays with intervals from 20 to 250 ms. Anodet long delays range from 275 to 1050 ms over 20 delay periods.

Also in Canada, Canadian Industries Limited provides Toe-det Delay system using a similar special delay cap linked permanently to a reinforced Anoline low strength detonating cord. This is in turn connected by a J-shaped plastic connector to a specially designed high strength trunkline detonating cord. It is essentially a short-delay system with 30 delay periods ranging from 25 to 250 ms.

6.3.9 *Non-electric systems not utilising detonating cord*

Beginning about in 1970, efforts were devoted toward developing new initiation systems that combined the advantages of electric detonator and detonating cord systems. Basically, these systems consist of a cap similar to an electric blasting cap, with one or two small tubes extending from the cap in a manner similar to leg wires. Inside these tubes is an explosive material that propagates a mild detonation which activates the cap. These initiation systems are not susceptible to extraneous electricity, create little or no airblast, do not disrupt the charge in the blasthole, and have delay accuracy similar to those of electric cap systems. Several systems are under development or in completion stage. Four systems are in the market:

1. Hercudet,
2. Nonel,
3. Exel,
4. Raydet.

Hercudet
Hercudet system (Fig. 6.15) of Hercules Incorporated in USA. utilises gas detonation in hollow tubes to initiate special type of blasting caps. The Hercudet cap resembles a regular electric cap in outward appearance except that there are two hollow plastic tubes in place of the two lead wires. In the space where a fusehead is normally present in the conventional electric detonator, the Hercudet cap has only an open chamber, which connects the two plastic tubes together and is also in contact with ignition charge leading to the delay element. Except for the above, the Hercudet cap is similar to an electric cap in all other respects though the Hercudet blasting cap has higher strength. The tubing in Hercudet has a relatively small internal diameter. The small diameter, combined with the low density Hercudet gas results in the tubing remaining undamaged by the passage of the detonation wave. This characteristic leads to an essentially silent transmission of energy which does not have any adverse effect upon the main explosive charge in the borehole. In a factory assembled stage the two tubes are filled with inert nitrogen gas to keep the system free of contamination and condensation.

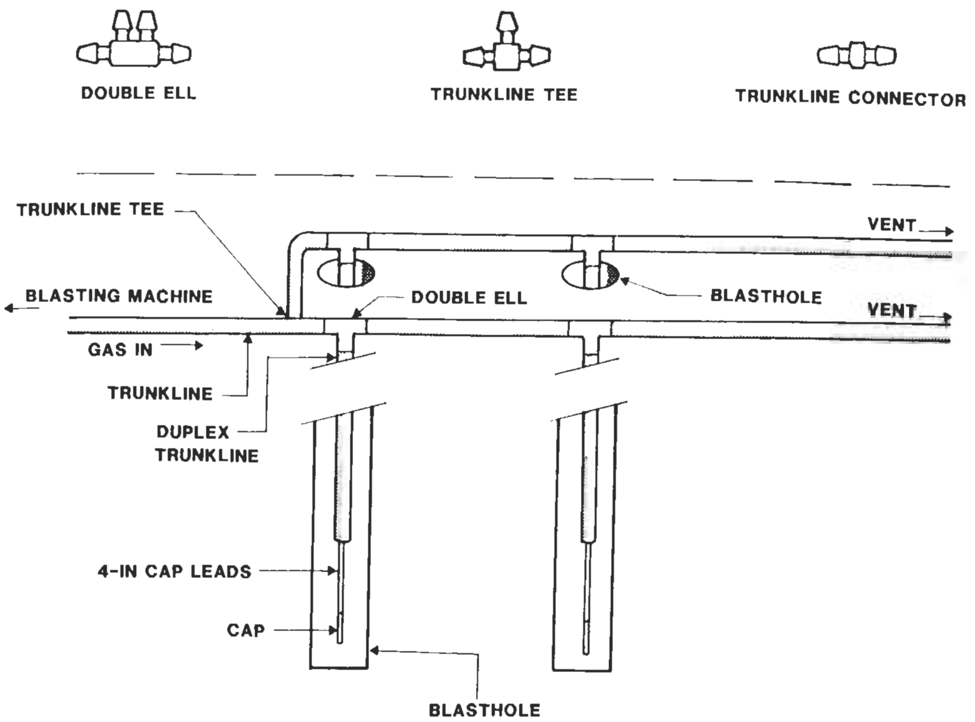


Figure 6.15 Hercudet connections for surface blasting.

The caps are inter-connected to one another through a series of extra length of tubing and suitable connectors. After all the holes are connected, a single trunkline of hollow plastic tube is taken to the Hercudet 'Pressure Test Module' by which leaks, bad and loose connections are checked. After testing, the trunkline is connected to the Hercudet blasting machine cum bottle box. The bottle box contains three bottles viz. oxidiser, fuel and nitrogen. When the blasting machine is turned on the charge position, a mixture of oxygen and fuel in proper proportion is fed into the tubing which in turn displaces all the air/nitrogen in the entire circuit. Thus the circuit is 'inert until charged'. It should become necessary or desirable for some reason to de-activate the system, it can be purged with nitrogen gas, thus making the entire circuit inert once again.

After charging is complete, the shot is fired by pressing the 'fire' button in the blasting machine. By pressing this button a piezoelectric crystal is activated producing a spark in the mixing chamber. This spark initiates the detonation of the mixed gas. The detonation travels at a velocity of 2500 m/s through the entire circuit initiating all the detonators.

The Hercudet system has the advantages of no airblast, no charge disruption, no electrical hazards, versatile delay capability, and system checkout capability. The inert nature of the system until the gas is introduced is a safety benefit. Specific crew training by a representative of the manufacturer is necessary because the system is somewhat different in principle than the more prevalent systems. Care must be taken not to get foreign material such as dirt or water inside the tubing or connector while hooking up the shot, and to avoid knots or kinks in the tubing.

The Nonel system

Nonel is the common trade name of a series of blast initiation accessories developed by Nitro Nobel of Sweden, which uses shock tube principle. It is now manufactured and marketed by many countries. Nonel is a safe, multipurpose firing system which combines the simplicity of fuse and igniter cord with the precision achieved with electric firing. The system is based on a plastic tube, the inside of which is coated with a reactive substance that maintains the propagation of a shock wave at a rate of approximately 2000 m/s. This shockwave has sufficient energy to initiate the primary explosive or delay element in a detonator. Since the reaction leaves the tube intact there is no lateral shock effect and the tube acts merely as a signal conductor.

The Nonel tube is made of flexible plastic with an outside diameter of 3 mm and inside diameter of 1.5 mm. In its standard form it is transparent and meets with most requirements in the blasting field. For specially tough conditions a heavy duty (HD) tube quality is available; this has much greater resistance to wear, and has higher tensile strength. In its standard form Nonel tube is adequate for ambient temperatures of up to 50°C. The tube is crimped to a delay detonator. The nonel detonator (Fig. 6.16) is made up as follows:

1. The visible part is an aluminium shell, the length of which may vary with the length of the delay element;
2. Base charge: a high explosive giving No. 8 strength detonator;
3. Primer charge: a flame-sensitive explosive;

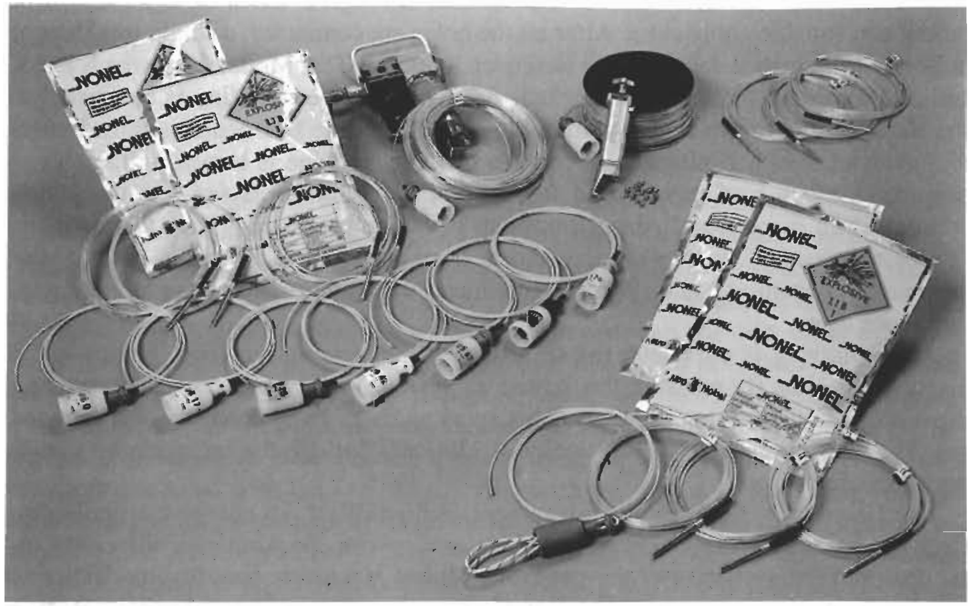


Figure 6.16 Nonel system, the non-electric initiation system created and developed by Nitro Nobel.

4. The desired delay interval is provided by an aluminium tube delay element filled with a pyrotechnic composition, part of which is pressed directly into the shell;
5. The detonator is crimped against a rubber sealing plug which also protects a portion of the Nonel tube against wear;
6. A specific length of Nonel tube with its free end sealed.

Special variants of Nonel detonators. Nonel GT-HD detonators have heavy duty quality tubes. Nonel GT-OD and Nonel unidet-OD detonators have, in addition to the reinforced tube, an outer aluminium shell for added rigidity and extra protection at the detonator/tube crimping point. The detonator is thus better protected during the charging work.

Connection accessories. A UB-O connector unit acts as a relay, the shock wave impulse from the Nonel tube is received, amplified and distributed to a number of receptor tubes. For this purpose the connector block holds a miniature detonator (transmitter cap), the strength of which is approximately one third that of normal detonator. The connector block is designed to: a) give mechanical protection to the transmitter cap, b) slow down the aluminium splinters from the transmitter cap.

Nonel GT/MS (millisecond series) is an all-round initiation system for millisecond delay blasting. This could be for surface, underground or underwater blasting applications. The Nonel GT/MS system is made up in a conventional manner; the detonators have individual delays and the activation is simultaneous. The range of delay periods is 3-20 with 25 ms between each period starting at 75 ms.

Nonel Unidet, the latest of the Nonel system has been developed to give maximum simplicity in both handling and storage of detonators. This has been accomplished by a newly developed detonator with a uniform delay. With nonel Unidet the timing sequence is built up on the surface. The detonators have an extremely accurate delay of either 475 ms or 500 ms. For the connection of Nonel Unidet detonators there are four types of connector unit. The delay connectors UB17, UB25 and UB42 have delays of 17, 25 and 42 ms respectively, whilst the UB0 is instantaneous.

When using Nonel Unidet the timing sequence does not have to be defined beforehand. In the first instance each borehole receives a detonator with a basic delay of 500 ms. When all the holes have been charged, surface connectors are used to provide the optimum blasting sequence and times between adjacent holes in rows and columns.

Nonel GT/T (Tunnel Series) is built up conventionally, the individual delays and activation occurs simultaneously. In order to obtain adequate swelling volume the shortest interval chosen is 100 ms (75 ms between 0 and 1). Nonel GT comprises 25 delay periods. The easiest method of connecting up a tunnel round is by bunch connection.

The Nonel system can be initiated by an ordinary detonator or an electric detonator or special starter caps. Manually operated blasting machines are available, as are pneumatically remote-control blasting machines both which utilise proprietary percussion caps.

Exel

The original shock tube of late 1970s was composed of an ionomer resin which was well-suited for its purpose on several accounts but deficient in others; namely strength, toughness, and resistance to contamination by water and diesel fuel. Current shock tubes are an improved type introduced in the early 1980s are overcoated with polyethylene. However with improper loading of heavy cartridges in deep holes, even this outer layer of polyethylene may not prevent stressing of the tube beyond its breaking point resulting in a hole cut-off. A new type of shock tube has been introduced called Exel, which consists of a plastic tube with an outside diameter of approximately three millimetres, the inside walls of which are coated with a fine layer of explosive dust. The dust is a typically a mixture of HMX (Octogen) with a small percentage of flake aluminium. Exel tubing is able to withstand rough handling, low temperatures up to –3.5°C and high surface temperatures of 65°C and is impermeable to diesel oil used in ANFO-type explosive and water/ammonium nitrate solutions in wet holes.

Raydet

The product, developed by IDL Industries Ltd of India, is similar to Nonel using the shock tube principle. Raydet consists of a Raytube made of plastic having an internal diameter of 1.5 mm and external diameter of 3.0 mm. The tube is thinly coated inside with explosive about 20 mg/m. The detonation velocity passing through the shock tube is of the order of 2000 m/s. In view of the low charge inside the tube, even after detonation is complete, the tube remains intact. Hence, unlike detonating cord, this does not disrupt the explosive column in the borehole.

On one end of the Raytube, a delay detonator is crimped with a special protection sleeve. This sleeve prevents damage to the Raytube at the crimped end while in usage. The other end is sealed to prevent ingress of moisture and other foreign matter. Presently up to 10 delays are available starting with 50 ms and with an interval of 25/50 ms.

In field usage, Raydet can be initiated with individual detonators or with detonating cord trunkline or with special starter caps.

6.4 COMPARISON OF INITIATING SYSTEMS

The advantages and disadvantages of electrical and non-electrical methods of initiation are well recognised (Table 6.1). Electrical methods provide a relatively accurate method of initiating a blast circuit that may readily be tested prior to initiation. Primary disadvantages however are that electrical detonators are more susceptible than standard explosives to accidental initiation by heat, impact, friction or extraneous current from static, stray current, lightning or radio waves.

Non-electrical methods, on the other hand are relatively insensitive to premature or accidental initiation from these causes. However, they have associated disadvantages such as: fume emission in underground applications from safety fuse and igniter cord, cross ignition of trunklines, noise from detonating cord surface lines, effected by water and oil infiltration. The introduction of low-energy detonating cords and the shock tube system has eliminated some of these problems but these have low

Table 6.1 Comparison of various initiating systems.

Feature	Nonel	HEDC + delay elements	Electric detonators	Fuse and igniter cord	LEDC	Hercudet
Timing ability	Precise	Limited	Precise	Limited	Precise	Precise
Blasting capacity	Unlimited	Unlimited	Unlimited	Limited	Unlimited	Limited
Blasting adaptability	Most aspects	Mainly open pit blasting	Mainly U/G blasting	U/G and smaller and surface blasting	ANFO and large diameter slurries	Most aspects
External electric hazard	None	None	Electricity and radio energy	None	None	Water and dirt contamination
Airblast	Non-existent	Significant	Non-existent	Non-existent	Limited	Non-existent
Fire risk	Non-existent	Non-existent	Non-existent	Existent	Non-existent	Non-existent
Means of checking	Visual	Visual	Instrumental	Visual	Visual	Instrumental
Craftmanship	Unskilled	Unskilled	Skilled	Unskilled	Unskilled	Skilled

HEDC = High-energy detonating cord, LEDC = Low-energy detonating cord.

tensile strengths and in case of the latter require careful design due to the low burning speed compared with conventional detonating cords.

In hole non-electric initiation schemes have tremendous potential for improved rock pile movement by means of longer delays without the fear of cut-offs. However the reliability of the overall systems needs to be improved upon.

6.5 SELECTING SUITABLE INITIATING SYSTEM

Many conditions exist at an operation which influence the selection of the appropriate initiation system. These may include:

– *Type of explosive.* Initiating systems employing detonating cord downlines may initiate high explosive or cause disruption of less sensitive explosive.

– *Borehole temperature.* Special explosives and initiating system need to be used when the borehole temperature exceeds 60°C.

– *Geology.* Initiation system should be fully activated before rock movement occurs to prevent cut-offs.

– *Hydrostatic pressure.* All systems are generally tested to a prescribed hydrostatic pressure, therefore, necessary information be obtained regarding the minimum hydrostatic head to which these will be able to withstand.

– *Extraneous electricity.* Consideration must be given to potential hazard from extraneous electricity when using electric detonators, since electric detonators are designed to be fired by a pulse of electrical energy which can come from many other sources and may lead to accidental ignition.

– *Environment and other constraints.* Proper delay sequence and delay period need to be selected depending on the vibration and airblast constraints and also flyrock control needs. The damage to the remaining rock is also controlled to a large extent on the delay sequence. Rock displacement and fragmentation requirements also dictate the choice of delay period and the delay sequence.

CHAPTER 7

Blasting accessories

7.1 INTRODUCTION

Various products are used in connection with blasting, in addition to explosives and initiating devices. These are termed blasting accessories, and while a few are consumed in the blast, most of them are designed to be used in blasting operations repeatedly. It is imperative that the blasting crew have the necessary tools and equipment to safely and efficiently load the explosive materials into the blasting holes. Tools and equipment normally required include:

– Power source (exploders, sequential blaster or appropriate mains firing instrument);
– Blasting circuit testers;
– Non-metallic measuring tapes equipped with lead or non-sparking weights;
– Lowering ropes;
– Non-sparking lowering and retrieving hooks;
– Tamping poles (wooden or non-sparking);
– Blasting knives;
– Connecting wire (new – not reclaimed);
– Lead line;
– Blasting covering material (when applicable);
– Crimper (when blasting with safety fuse);
– Lightning detector.

The equipment need to be first quality and be kept well maintained. The substitution of steel nuts or bolts for weights, steel pipe or length of drill steel for tamping poles, lengths of detonating cord as measuring tapes or lowering ropes, reclaimed legwire as connecting wire, improper test instruments must not be permitted.

7.2 POWER SOURCES FOR ELECTRICAL FIRING

Electric blasting circuits can be energised by blasting exploder or by powerlines through special arrangements. Storage and dry cell batteries are not recommended for blasting operations because they cannot be depended on for a consistent output.

a

b

Figure 7.1 a) Twist generator type exploder, b) Hand driven generator powered capacitor discharge exploder (courtesy Schaffler).

7.2.1 *Exploders*

Exploders are machines which provide the required electrical power to fire a series of electrical detonators (Fig. 7.1). Usually there are two types of exploders: generator type and condenser discharge type.

Generator type
The rack bar exploders are operated by vigorously pushing down a rack bar which, through a set of gears, spins the rotor in a DC generator. The electrical energy from the generator is connected to output terminals when the rack bar reaches the bottom in its downward travel where it closes a switch. These are usually of large capacity capable of blasting 30-50 caps in single series.

Another type is where a spring is first wound and with the help of key, it is suddenly released. This through a set of gears, spins the rotor in a DC generator. This type has very large capacity of 250 to 400 shots.

The key-twist type exploders are relatively small, hand held units that are operated by a quick twist of the handle with one hand while the machine is held firmly in the other hand. This type of machine initiates 10 to 25 shots of 4 ohms each. The actual current put out by these machines depends on the condition of the machine and the effort exerted by the shot firer. Both the rack-bar and twist machines should be operated vigorously to the end of the stroke because the current flows only at the end of the stroke. Because the condition of a generator blasting machine deteriorates with time, it is important that the machine be periodically checked. In some machines internal mechanical or electronic switch cuts off the output voltage at the end of 4 ms. There is also a built-in provision which does not connect the voltage to the output terminals, unless it is above minimum predetermined value. This ensures that unless the shot firer operates the exploder properly, and unless it has attained its rated output, no output is delivered to the external circuit and hence avoids the chances of misfire due to improper operation.

Condenser discharge type
Although the generator machine has been a dependable blasting tool, its limited capacity and variable output has caused it to be replaced, by the condenser (capacitor) discharge type machine. As the name implies, in the capacitor discharge (CD) machine a condenser(s) is charged to a high voltage and discharged later through the external firing circuit. The basic source of power may be a low voltage dry battery or an electromagnetic generator. CD machines are available in a variety of designs and capacities with some capable of firing over 1000 caps in a parallel series circuit.

All CD machines operate in basically the same manner. One button or switch is activated to charge the capacitors and a second button or switch is activated to fire the blast. An indicator light or dial indicates when the capacitor is charged to its rated capacity.

Considerable care must be exercised while selecting the proper exploder for series firing. While the capacity of the exploder must be more than sufficient to fire all the detonators, it should not be too much higher than what is needed. A good rule of thumb is to use an exploder of 1.5 times to 2 times the needed capacity. It is important that exploders should be properly maintained and regularly tested as to their firing capacity. Even if well-maintained, they may develop faults in time that reduce their capacity. The overall condition of a CD blasting machine should be checked by tester or by a set-up combining a rheostat and a resistor. Maintenance of exploders should be done only by qualified persons and when in doubt, the manufacturer must be consulted.

7.2.2 *Sequential blasting machine*

A sequential blasting machine is a unit containing 10 capacitor discharge machines that will fire up to 10 separate circuits with a preselected time interval between the individual circuits. When used in conjunction with millisecond delay electric blasting caps, the sequential machine provides a very large number of separate delay intervals

Figure 7.2 Sequential blasting machine.

(Fig. 7.2). This can be useful in improving fragmentation and in controlling ground vibrations and airblast. Because blast pattern design and hook-up can be quite complex. The sequential blasting machine should be used only by well trained persons or under the guidance of a consultant.

7.2.3 *Mains firing*

Another alternative for energising electric blasting circuits is the power line. Power line blasting is often carried out with parallel circuits where the capacity of available blasting machines is inadequate. When firing off a power line, the line should be dedicated to blasting alone, should be visually checked for damage and for resistance on a regular basis (Fig. 7.3). Power line blasting should not be carried out unless precautions are taken to prevent arcing. Arcing can result in erratic timing or a misfire.

Arcing in a cap results from excessive heat build-up, which is caused by too much current applied for too long a period of time. A current of 10 amp or more continuously applied for a second or more can cause arcing. To guard against arcing the blasters may use a blasting switch in conjunction with the power line.

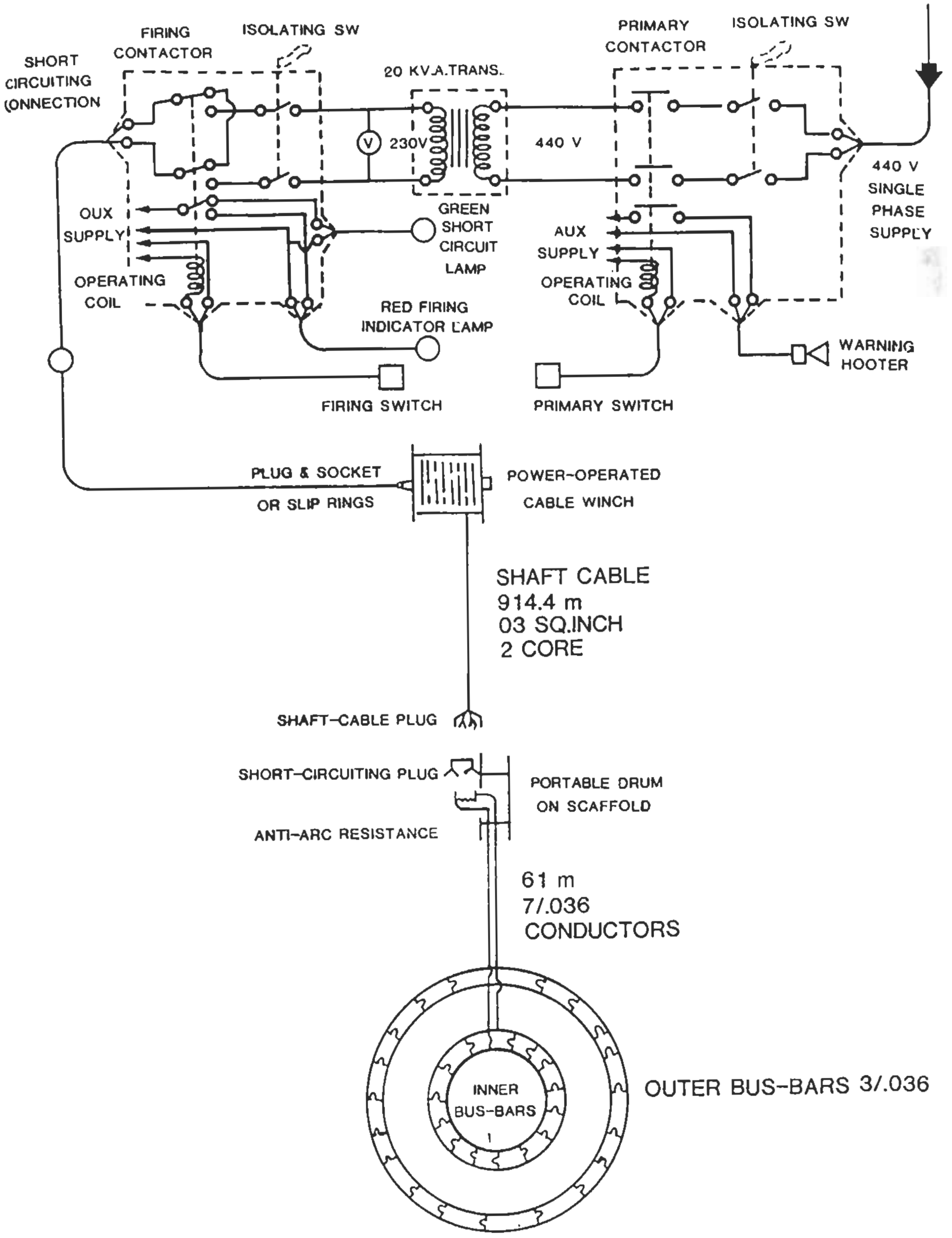


Figure 7.3 Typical arrangement for electric mains firing.

7.3 BLASTING CIRCUIT TESTING

Electric blasting procedure can be successfully executed only if proper care is exercised in planning and connecting the blasting circuit. A list of possible circuit weaknesses is given below.

Figure 7.4 Blasting circuit tester.

1. *Discontinuities or shorts* in: a) the detonator circuits, and b) the total or any part of the blasting circuit;

2. *Current leakage* can occur when damage to leg wire or connecting wire insulation allows the bare wire to make a contact with the rock (ground) specially under damp conditions or conducting ore body;

3. *Stray electricity* (current leakage from extraneous sources).

Instruments are available to locate faults, thus they provide opportunity to remove faults and hence contribute to safety and efficiency of blasting operations. There are two types of blasting circuit testers: a blasting galvanometer (actually an ohmmeter) shown in (Fig. 7.4) and a blasting multimeter. The blasting galvanometer is used only to check the circuit resistance, whereas a blasting multimeter can be used to check resistance, ac and dc voltage, stray currents, and current leakage. Only a meter specifically designed for blasting should be used to check blasting circuits. The output of such meters is limited to 0.05 amp, which will not initiate an electric blasting cap, by the use of a silver chloride battery and/or internal current-limiting circuitry.

It is generally recommended that each component of the circuit be checked as hook-up progresses. After each component is tested, it should be shunted. To begin with all electric detonators be tested for continuity before incorporating them in the blasting circuit. A total deflection of the circuit tester needle (no resistance) indicates a short circuit. Zero deflection of the needle (infinite resistance) indicates a broken wire. Either conditions will prevent a blasting cap, and possibly the whole circuit, from firing.

Before testing the blasting circuit, its resistance should be calculated. After the caps have been connected into a circuit the resistance of the circuit is checked and

compared with a calculated value. A zero deflection at this time indicates a broken wire or missed connection and an excessive deflection indicates a short circuit between two wires.

After the circuit resistance has been checked and compared, the connecting wire is then added and then circuit is checked again. If a parallel series circuit is used, the change in resistance should be checked as each series is added to the bus wire. In a straight parallel circuit, a break in the bus wire can sometimes be detected. However, a broken or a shorted cap wire cannot be detected in a straight parallel circuit because it will not affect the resistance significantly.

A final check of the circuit is made at the blaster's location after the firing line has been connected. If a problem is found in a completed circuit, the circuit should be broken up into separate parts and checked to isolate the problem. The firing line should be checked for a break or short after each blast, or at the end of each shift, as a minimum.

To check for a break in the firing line, the two wires at one end of the line are shunted and the other end is checked with a blasting meter. A large deflection indicates that the firing line is not broken, a zero deflection indicates a broken wire. To test for a short, the wires at one end of the lead line are separated and the other end is checked with the meter.

Certain conditions such as damaged insulation, damp ground, conductive ore body, water in a borehole, detonator wires touching the ground, or bulk slurry in the borehole may cause current to leak from a charged circuit, some detonators may not then get adequate current and may lead to misfires.

Presently digital circuit testers (Fig. 7.5) have become popular as they do not have any moving parts.

Besides, the blasting multimeter instruments are used for earth leakage and insulation testers. Measures for combating current leakage include using fewer caps per circuit, using heavier gage lead lines and connecting wires, keeping base wire connections from touching the ground or using a non-electric initiation system.

Electric blasting is a safe, dependable system when used properly under the proper conditions. Advantages of the system are its reasonably accurate delays, ease of circuit testing, control of blast initiation time, and lack of airblast or disruptive effect on the explosive charge. In addition to extraneous electricity, one should guard against kinks in the cap leg wires, which can cause broken wires, especially in deep holes. Detonators of different manufacturers may vary in electric properties, therefore only one manufacturers detonators per blast should be used. It is recommended that the

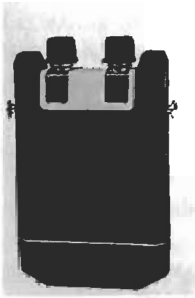


Figure 7.5 Digital blasting ohmmeter 'Digi ohm' (courtesy Schaffler).

blasters carry the key or handle so the power source cannot be inadvertently fired while checking of the shot is going on.

7.4 EXTRANEOUS ELECTRICITY HAZARD

The principal hazard associated with electric blasting systems is lightning. Extraneous electricity in the form of stray currents, static electricity and radio frequency energy, and from high-voltage powerlines can also be hazard. Electric blasting caps should not be used in the presence of stray currents of 0.05 amp or more. Stray currents usually come from heavy equipment or power systems in the area and are often carried by metal conductors or high-voltage powerlines. Techniques are available for stray currents, which continuously monitor ground currents and sound an alarm when an excess current is detected. The supplier should be consulted as to the availability of these units.

Static electricity may be generated by pneumatic loading, particles carried by highwinds, particularly in a dry atmosphere and by rubbing of a person's clothes. When pneumatically loading blasting agents with loaders, a semi-conductive loading hose must be used, a plastic borehole liner should not be used, and the loading vessel should be grounded. Electrical storms are a hazard regardless of the type of initiation system being used. Even underground mines are susceptible to lightning hazards. Upon the approach of an electrical storm, loading operations must cease and all personnel must retreat to a safe location. Lightning detectors are available which can be used to detect approaching storms. Meteorological reports are also helpful.

Broadcasting stations, mobile radio transmitters and radar installations present the hazard of radio frequency energy.

High-voltage powerlines present the hazards of capacitive and inductive coupling, stray current and conduction of lightning. A specific hazard with powerlines is throwing part of the blasting wire onto the power line. This shorts the power line to the ground and has been responsible for several deaths. Care should be exercised in laying out the circuit so that the wires can not be thrown on a power line. Other alternatives are to weigh down the wires so they cannot be thrown.

7.4.1 *Blasting cables*

The main requisite of a blasting cable (Fig. 7.6) are low resistance, good waterproof insulation with suitable tensile strength and flexibility. Actual choice must depend upon working conditions and statutory regulations.

The following types could be used with portable exploders:

General duty. 23/0076 parallel twin core cable with PVC insulation at 5.8 ohms per 100 m of cable or 24/0.20 mm at 4.6 ohms per 100 m.

Heavy duty. 70/0076 twin core cable with PVC outer sheath at 1.8 ohms per 100 m of cable or 50/0.25 mm at 1.4 ohms per 100 m.

Connecting wire. This is used to lengthen the lead wires of electric detonators. They are not to be reused. For routine work under generally dry conditions 25 SWG copper wire with PVC insulation is available.

For more arduous work such as extending lead wires down wet or ragged blastholes, twin bell wire is recommended in 21 SWG copper twin twisted.

Figure 7.6 Blasting cables.

7.4.2 *Insulation testers*

When misfire occur with electric firing the cause is frequently due to the leakage of current to earth through faulty insulation with consequent starvation of current to the detonators. Leakage to earth can take place at unprotected joints and points of damaged insulation in contact with water or conductive rock. An instrument to measure the effectiveness of the insulation is the insulation tester with a maximum test current of 12 mA.

The insulation tester is especially useful for checking large blasts incorporating many detonators where misfires or a partial failure could seriously hinder production and make recovery slow and expensive, e.g. pillar blasts, mass blasts, multi-row surface blasts.

To obtain accurate earth fault resistance reading a good earth connection for the earthing terminal is essential. Satisfactory insulation exists in a full round where the earth fault resistance is not less than four times the resistance of the circuit and, in any level less than 200 ohms.

7.4.3 *Lightning detector*

Snow, dust and electrical storms with high levels of static electricity in the atmosphere, and the possibility of lightning are very real hazards for blasting operations. Lightning detectors measures atmospheric voltage gradient and operate both flashing lights and warning siren when the gradient approaches the levels at which a lightning discharge could occur.

7.4.4 *Warning system*

A proper warning system against possible entry of persons and equipment in the danger zone is essential when blasting operations are being carried out. They are either portable or permanent type mounted on top of the site office and/or on a vehicle which is deployed for guarding the blast area. A portable hand held loudspeaker is also used to warn the residents and workmen posted inside the mine. Besides residents and workmen should be instructed about the warning codes adopted in the mine for different phases of operation.

7.5 STEMMING MATERIAL

Commonly used stemming material in surface mines is drill cuttings. However, research has shown that drill cuttings will not hold into the drill hole. Very coarse materials will have tendency to bridge the hole when loading and may be ejected. The optimum size of stemming material would be material that has an average diameter of approximately 0.05 times the diameter of the blasthole. Material must be angular to function properly. River gravel which has become rounded, will not function as well. Upon detonation of the explosive in the blasthole, stemming particle will be compressed to mortar consistency for a short distance above the charge.

In many cases stemming is not used, specially in case of development headings in hard rock mining. Gases generated by the detonation of explosives is assumed to act as self stemming. In underground coal mines clay plugs are used as stemming plugs. The clay plugs should be compact but not hard. It is a mixture containing 70% fine sand and 30% clay. PVC ampoules (water filled or gel filled) are used as safe stemming material. Their use significantly reduces the risk of methane ignition or coal dust explosion and also dust and fumes from blasting are reduced.

Stemming of large diameter blastholes in surface mining is laborious and time consuming process. Blasthole stemming machines have the capability of stemming a 10 to 20 m deep blasthole of 312 mm diameter in less than 15 minutes time. On a front end loader the fitting consisting of hydraulic cylinders are attached. Besides the loader is equipped with an extra control valve for operating stemming and hydraulic hoses running down the boom arms to the attachments.

7.6 OTHER TOOLS

Many tools are required for efficient and safe execution of blasting operation.

Crimping of safety fuse to plain detonator is done either with a bench mounted detonator crimper or with an approved type of hand crimping pliers. In both, a fuse cutter and crimping jaws are incorporated. When a very large number of detonators are to be crimped, bench crimpers are used in a central crimping/capping station on surface. When a small number of detonators are to be crimped hand crimping pliers are used.

Pricker is a pointed tool, made of wood or non-ferrous metal for pricking hole in the cartridge to insert detonator for priming.

Knife or cutting pliers are needed to cut safety fuse in proper manner. When use is made of the detonating cord, special cutter for detonating cord is used.

Pliers for detonating relays are needed for crimping detonating relays to detonating cord.

Scraper is used to clean the holes and crack detector is used to detect transverse crack. This tool is useful in underground coal mining.

CHAPTER 8

Rock fragmentation process by blasting

8.1 INTRODUCTION

When an explosive charge, confined within a blasthole is initiated, reactions take place resulting in production of large amount of gases at very high temperature and pressure in a very short time. An important characteristic of high explosive is the production of very large amount of energy per unit of time. The gas pressure acts on the walls of the hole and thus subjects the media beyond the hole to vast stresses and strains.

More than 300 years ago Vauban indicated that rock was broken due to lifting action of gases produced in blasting. His views with some modifications were believed to be true till later part of 1950's. Hino (1956a and b) and Duvall & Atchison (1957) indicated that rock is broken mainly by reflection of shock wave that travels outward spherically from the charge.

However, later workers indicated that it was the joint action of gaseous pressure and shock waves which was responsible for rock fracture and fragmentation under the action of explosives. A single widely accepted mechanism of rock fragmentation is lacking which can explain the fragmentation phenomena in all the situations in various formations. Many theories of rock blasting and resultant fracture and fragmentation have been proposed. Many of these were for specific environment, conditions and assumptions. More than one mechanism is needed to explain the fragmentation phenomena.

In this chapter the present understanding of the process of fragmentation by blasting is described. Attention has been mainly given to those aspects which are relevant to most of fragmentation in blasting, i.e. the process taking place between hole and the free face. Other aspects such as the process of detonation of an explosive charge in a hole and the phenomena occurring around the hole are briefly described in order to better understand the subsequent phenomena. The processes involved are explained first for homogeneous rock and subsequently for the jointed rocks and other blasting situations. The effects other than the useful part of explosive reaction, such as seismic waves, airblast, flyrock and damage to the remaining rock are considered later in Chapters 18 to 20.

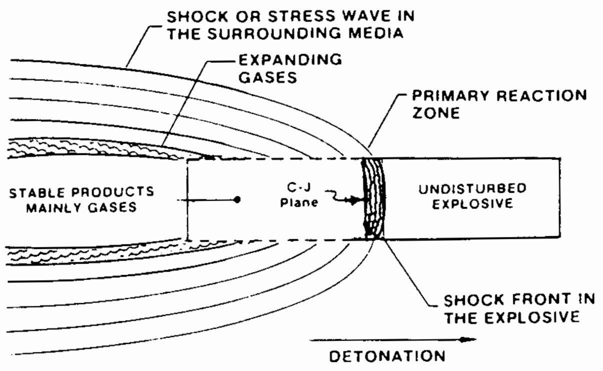


Figure 8.1 Detonation reactions in a cartridge.

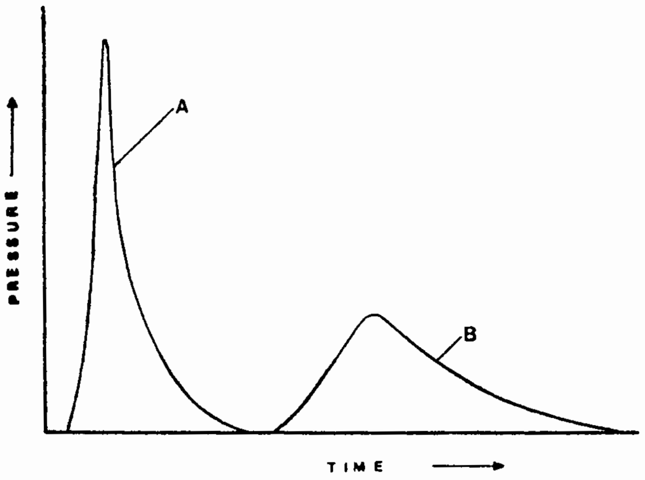


Figure 8.2 Pressure pulse shapes for high explosive (A) and a commercial explosive containing high gas volume (B) (after Aimone 1992).

8.2 THE PROCESS OF DETONATION OF AN EXPLOSIVE CHARGE IN A HOLE

For accomplishment of rock breakage the process begins with the detonation of high explosive which results in a very rapid chemical reaction at the velocity of 2000 to 7000 m/s or more normally between 4000 to 6000 m/s of a thermodynamically unstable substance producing gases at high temperature and pressure in a very short time (Fig. 8.1). The initial pressure developed in the detonation zone may be between 0.5 to 50 GPa or more and temperature may be between 3000 to 4000°K.

The actual pressure exerted on the walls of the hole containing the explosive charge is affected by many factors among which are the loading density of the explosive, the space and type of material between the charge and the surrounding rock (coupling) and the properties of the explosion products and of the rock.

The first interaction between the rapidly expanding high pressure gases and the surrounding rock occurs at the moment the explosion products impact the surface of the hole. As a result of the impact of the explosion gases on the rock a high pressure is suddenly exerted on the surface of hole. This pressure on the hole almost instantaneously rises to its peak which probably is of the order of one quarter to one-half of the detonation pressure, and then decays roughly exponentially due to the cooling of the gases and their outward expansion (Fig. 8.2). The more the gaseous products are able to expand before they encounter the rock, lower will be the peak pressure at the moment of impact. Initially the transmission of the pressure pulse is in the form of a high pressure shock wave immediately around the hole, decreasing to stress wave farther from the hole as the stress decreases (Rinehart 1975; Mohanty 1985). By the time the pressure pulse has been transmitted into the rock, the initial high pressure in the gas-filled cavity has decreased, although it is still high enough to continue expansion of the cavity and to exert a quasi-static pressure on the rock for a considerable time.

A number of reports and monographs are available which deal with the science of the explosive reaction and detonation processes (Cook 1958; Johansson & Persson 1970; Clark 1987) reference to these can be made for details.

8.3 PROCESSES IN THE ROCK AROUND THE HOLE

The pressure of the explosion pulse greatly exceeds the dynamic compressive strength of the rock, causing crushing and fracturing of the rock immediately around the hole. As the rate of attenuation of the pressure pulse is high, the crushed zone around a drill hole is of limited extent and probably does not exceed two to four hole radii, while fracture zone averages 20 borehole radii away and extends to 50 radii

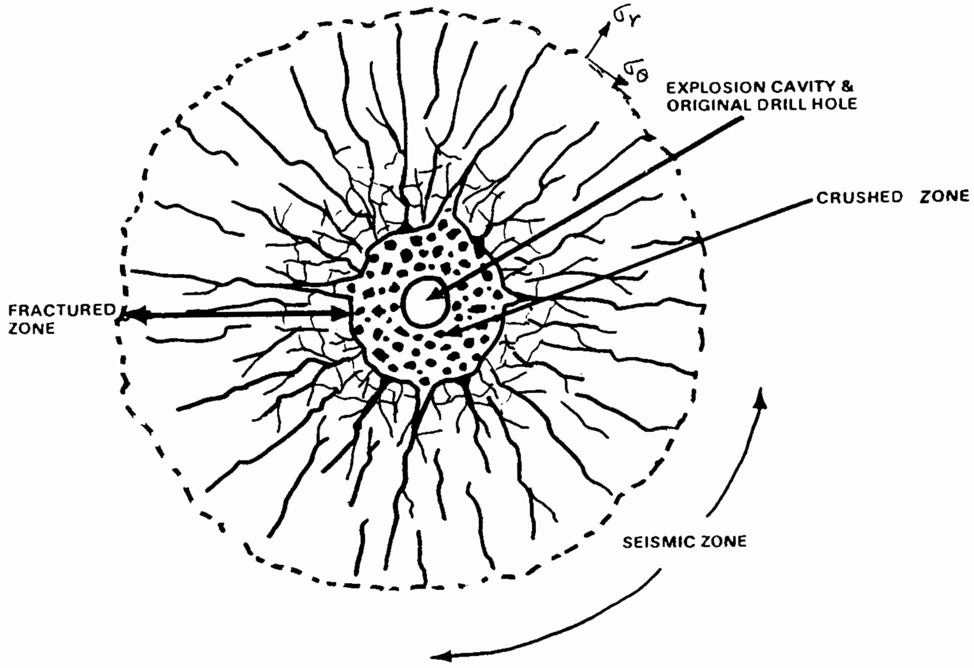


Figure 8.3 Fragmentation zones around an explosion in rock.

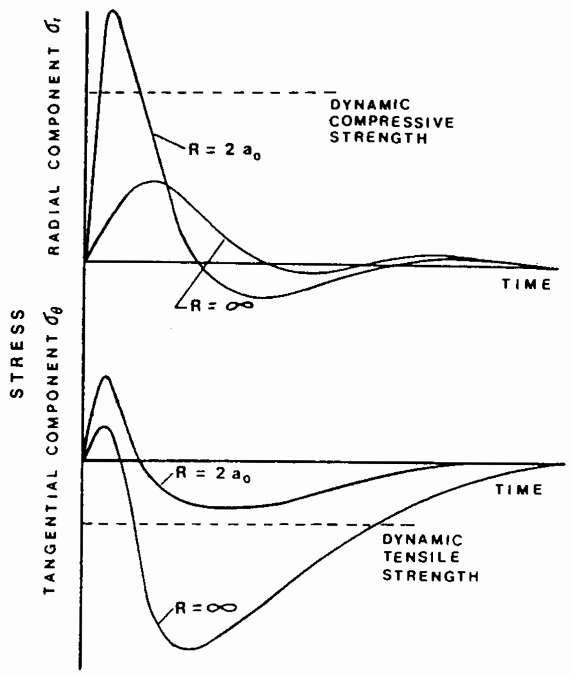


Figure 8.4 Generalised stress versus time for radial and tangential components of stress at two distances from the borehole centre (after Aimone 1992).

(Siskind & Fumanti 1974). Since the intensity of the stresses generated by the explosion falls off rapidly with distance from the borehole, the rock behaviour will range from plastic deformation to brittle elastic fracturing and the particle size will increase rapidly with distance from the borehole wall.

Atchison (1968) indicates that there exists a fractured zone between the crushed zone and zone of initial breakage (Fig. 8.3). In this fractured transition zone the strength of the rock becomes more significant but the pressure of the pulse is initially still greater than the strength of the rock. In the inner region of this transition zone, therefore, one might expect further crushing of the rock – although not complete disintegration as in the crushed zone.

As the amplitude of the pressure pulse decreases because of the spreading and energy absorption so also there is corresponding decrease in the intensity of crushing. The rock outside the innermost part of this zone responds in the form of oscillatory stress transients having both radial and tangential components (Fig. 8.4). The radial component is essentially compressive, the stress decreasing from greater to less than the compressive strength of the rock, so that in this zone it takes very little part in the rock fracture. The stress components in the tangential directions are tensile in nature. Since, the strength of rock in tension is much less than in compression these tensile stresses give rise to radial fractures which extend out from the crushed zone. The amplitude of these tangential tensile stresses decay very rapidly with increasing distance from the hole, so that the extent of radial fracturing caused by them is limited to between two and six hole diameters.

8.4 FRAGMENTATION PROCESSES BEYOND THE TRANSITION ZONE

A number of ways have been suggested by which the explosive energy imparted to the rock accomplishes the task of fragmentation beyond the transition zone. Based on the present understanding, the important principles can be divided into two broad areas. This grouping is based upon the relative importance accorded either to:

a) The role of stress waves imparted to the rock by the rapid release of energy during the detonation of explosive in blast hole; or to

b) The role of gases released by detonation, in creating a quasi-static stress field around the blasthole, in extending radial cracks and in moving the fragmented rock.

8.4.1 *Role of stress waves*

Early researchers which included Hino (1956a) and Duvall & Atchison (1957) felt that the stress waves were responsible for most of the fracture and fragmentation.

At the edge of the transition zone the stress transients degenerate into a conventional strain wave (here after referred to as stress wave) which radiates outward into the solid mass, and in the absence of a free face, can take no further part in the fracture process. However, when a free face is available nearby, then at the free face, the incident wave is reflected back towards the borehole as tensile wave (Fig. 8.5). This happens in the case when the impedance (product of density and velocity of propagation) of the medium in which the wave travels is greater than the medium at which it is reflected. The rocks being weaker in tension than in compression, fracturing of the rocks occur from the face inward where the effective tension (i.e. the effective difference between intensity of tension wave and remaining compression wave) exceeds the dynamic tensile strength of the rock. This phenomena under the influence of stress wave reflection is called scabbing, spalling or slabbing. This process is multiple if the incoming compressive wave is strong enough after reflection from the new face to cause breaking.

The effect of reflected stress waves from free face was also realised by other workers. Field & Ladegaard-Pederson (1971) showed that the reflected waves modified outgoing radial cracks and the function of stress waves was to precondition the rock.

Barker et al. 1975 and Barker & Fourney 1978 demonstrated from their experiments on Homolite models, the development of fracture pattern due to stress waves only. These waves propagate in the rockmass and initiate the fractures up to a significant distance. The number of fractures was eight to twelve which form large pie shaped fragments. These pie-shaped fragments are reloaded with the reflected waves (primary and shear) which have sufficient strength for further fracturing of rock in tension.

Many workers consider that the role of reflected stress waves in blasting is not important since the reflected stress waves for the conventional burdens used in field blasting (50 to 100 times the charge radii) become too weak to cause any breakage (Noren 1956; Bergmann et al. 1973; Langefors & Kihlstrom 1963; Persson et al. 1970; Porter & Fairhurst 1970; Harries 1973). Evidences from high speed photography and other methods have shown that the period between detonation of the explosive charge and the beginning of the movement of the bench face is between three

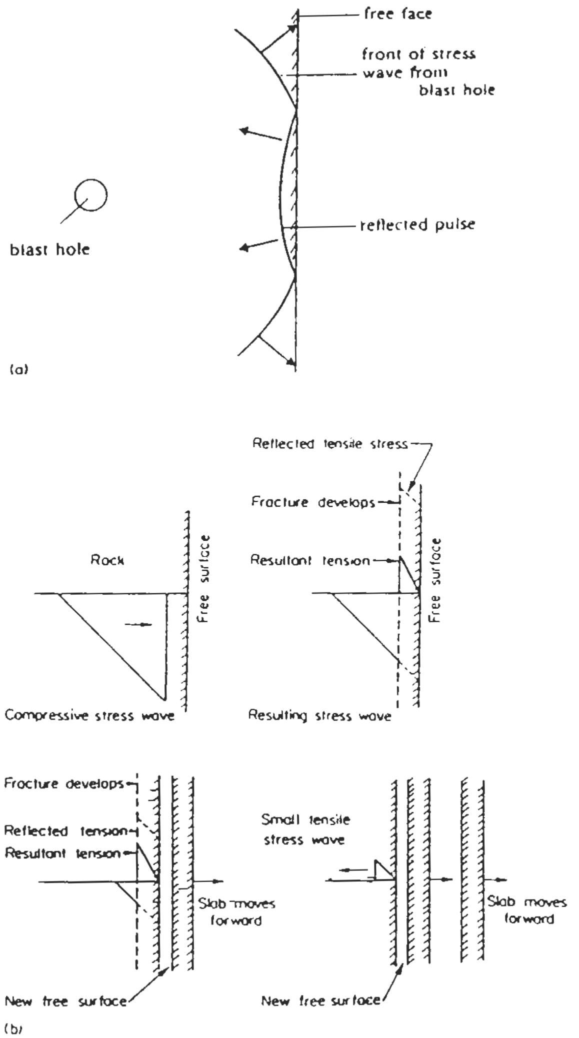


Figure 8.5 a) Reflection of stress wave and b) Mechanism of scabbing.

and ten times that taken for a stress wave to pass from the hole to the free face and back indicating thereby that breakage is by effects other than that of stress waves (Turata et al. 1966; Bergmann et al. 1974).

In an investigation to understand the role of stress waves, fragmentation variation and fragments were studied by carrying out small scale blasting in bench shaped blocks of cement mortar or granite (Bhandari & Vutukuri 1974; Bhandari 1975a; Bhandari et al. 1975; Bhandari 1979a). The fracture surfaces of large fragments showed that two distinct fracture mechanisms participated in fragmentation. One fracture surface parallel to the bench face was rough and had a bubbly appearance showing that fracture due to direct tension had occurred. Reflected stress waves and subsequent scabbing cause fracture by direct tension and fracture is parallel to the bench face. On the other hand, the fragment surface containing the hole had fracture surface with heckle marks. The origin of the fracture appeared to be near the hole.

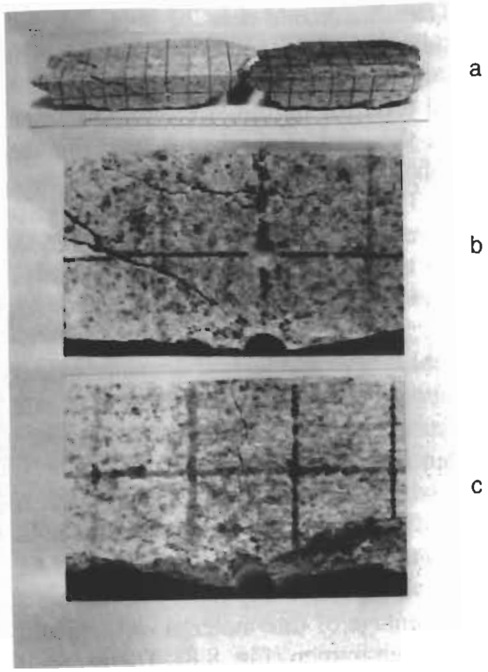


Figure 8.6 Two fragments shown in (a) obtained from tests without and with wave trapping experiments. Enlarged view in (b) of area opposite the holes shows presence of scabbing crack parallel to bench face in tests without wave trapping. In wave trapping test scabbing crack is absent (c).

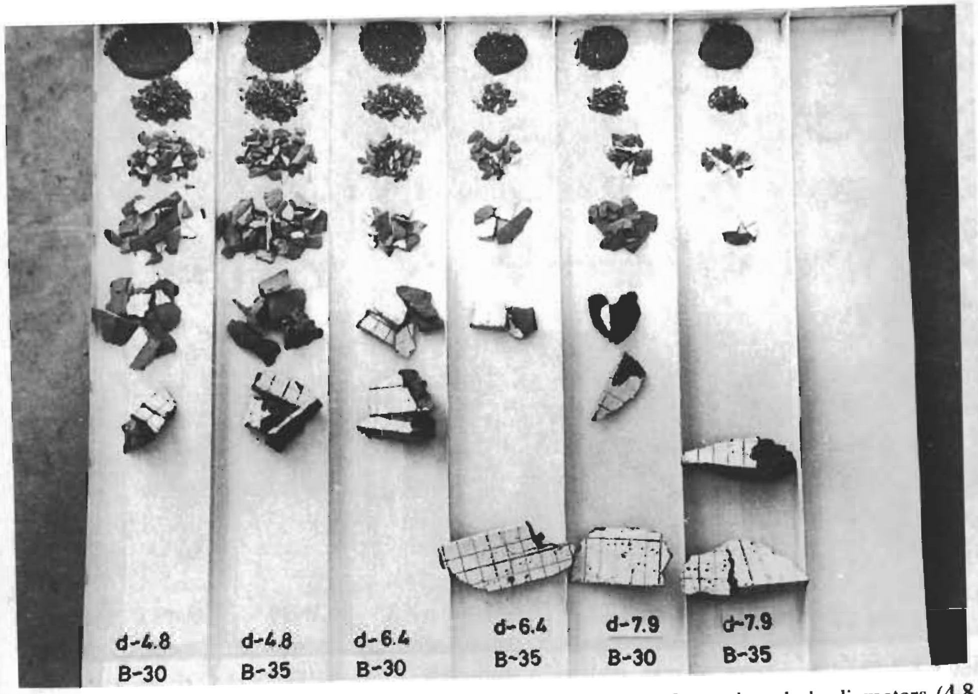


Figure 8.7 Sized fragments, from single hole cement mortar tests for various hole diameters (4.8, 6.4 and 7.9 mm).

The latter surface, therefore, indicated that the fracture started near the hole wall and progressed towards free surface and was not the result of stress wave action. Thus in the latter instance the breakage was by gaseous action.

The large fragments obtained in single hole tests for optimum breakage burden and larger burdens appeared to originate from the area of the bench opposite the hole. The large fragments in single hole tests showed that in each fragment a crack parallel to the bench face was present (Fig. 8.6). The crack parallel to the face is generally accepted as due to the scabbing action of reflected stress waves (Saluja 1962). Multiple scabbing was absent. From the above evidence, it was concluded that in large fragments, the stress wave reflection and scabbing actions were weak.

Indirect tests were carried out to show the role of the stress waves by reducing or eliminating the stress wave reflection and scabbing. For a constant diameter of explosive charge, three different hole diameters were used to show the decreased role of stress waves as in decoupling. Tests showed that the increased decoupling produced coarser fragments and for a given burden fragment size increased (Fig. 8.7). The percentage of fine fragments also reduced with the increase in hole diameter. It was stated that one reason for the coarser size of fragmentation with increased hole diameter (and thus the decoupled charge) was the reduction in the role of stress waves. In another set of experiments when the waves were trapped by attaching a plate of same material to the bench face, the percentage of fine material reduced and the average size of fragments increased for any given burden (Fig. 8.8). Therefore, it

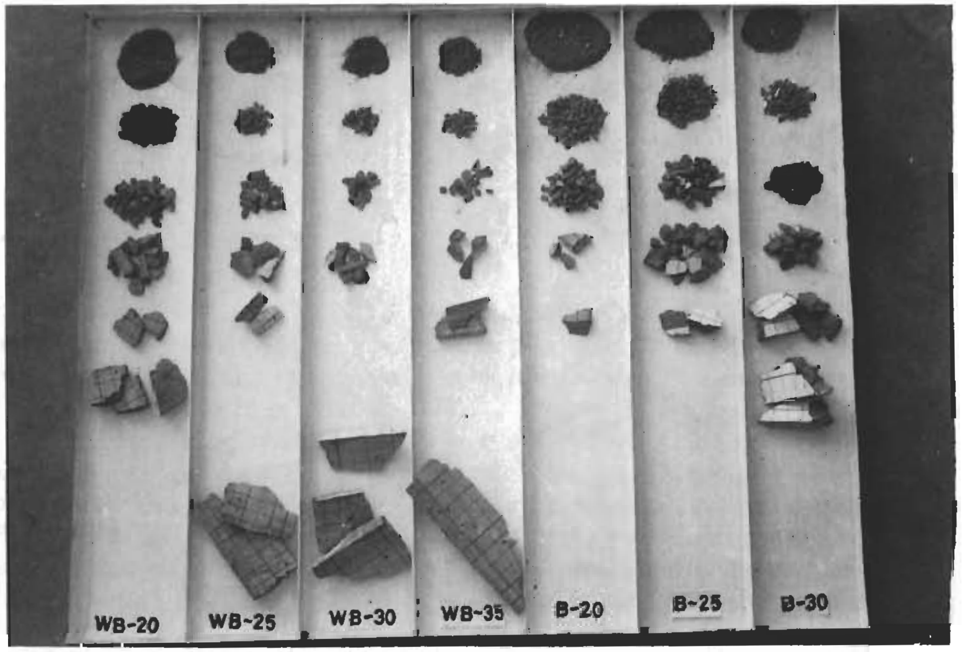


Figure 8.8 Sized fragments, from single hole cement-mortar, wave trapping tests for burdens, WB = 20, 25, 30, 35 mm, and without wave trapping for burdens, B = 20, 25, 30 mm, placed in vertical rows. Particles arranged in vertical rows for small to the largest in the sizes.

was suggested that in the absence of stress wave effect the fragment size obtained was coarser and stress waves helped by scabbing and by weakening of the rock. Increased role of the stress waves helped in increasing the subsequent quasi-static gas pressure role.

8.4.2 *Role of quasi-static gas pressure*

Saluja (1962), Langefors & Kihlstrom (1963), Sadwin & Duvall (1965), Cook et al. (1966), and others have shown that shock wave is not the only agency, which is responsible for rock breakage under the action of explosives. Langefors & Kihlstrom (1963), Persson et al. (1970), Porter & Fairhurst (1970) and others concluded that fragmentation by blasting could be solved as a quasi-static gas pressure problem. It was suggested that radial cracks initially developed in the transition zone were extended by gas pressurisation until they intersected the free face and thus resulted in fragmentation.

Fogelson et al. (1965) concluded that energy in the form of shock wave was about 9% of the total energy of high explosives. Langefors & Kihlstrom (1963) took shock wave energy to be between 5 to 15% of the total theoretical energy of the explosive and explained effect of shock wave by considering a charge to be blasted in a hole

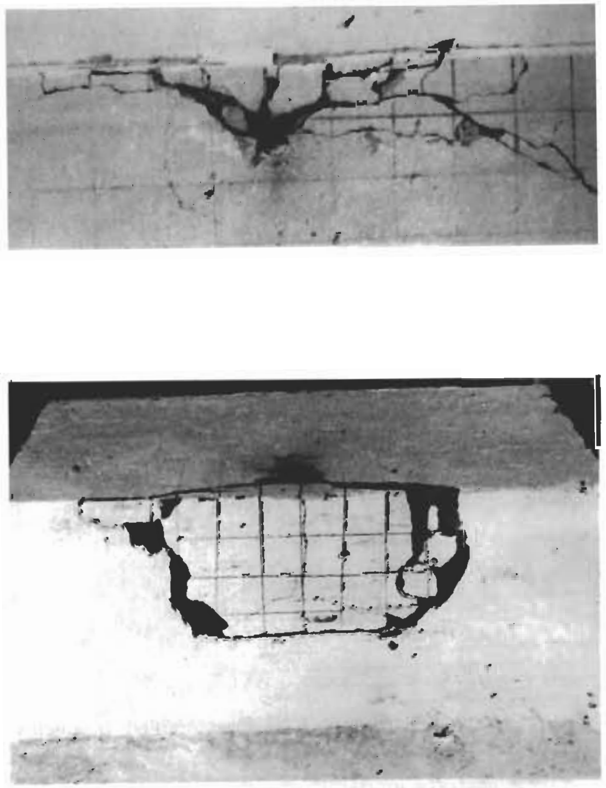


Figure 8.9 Reassembled particles from single-hole tests in cement-mortar blocks.

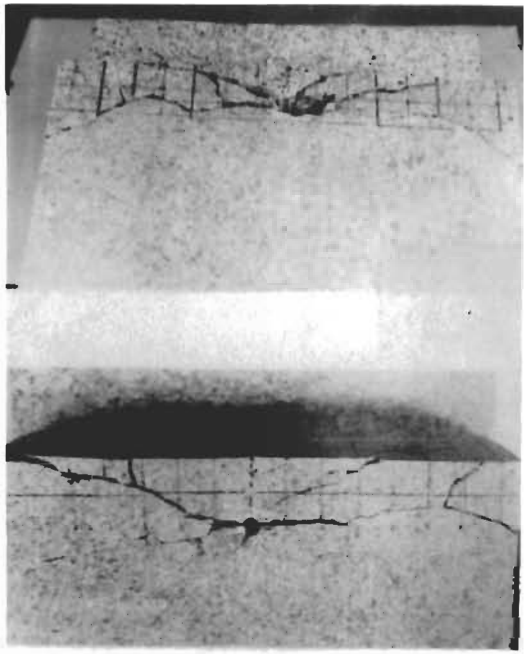
with one free face. They say as the shock waves are distributed all around the charge, at least two-third of this shock wave energy will disappear without affecting the breakage for a single hole with an angle of breakage less than 120°. This would imply that only 3% of the total energy of the explosive is distributed by the shock waves within the angle of breakage, hence the shock waves are not responsible for the actual breakage but provide only the basic conditions for rock fracture to begin.

Sadwin & Duvall (1965), while observing cratering results noted that change in depth had effect on observed stemming velocity and air-blast which did not decrease monotonically with the charge depth and as such these indicated that the explosion gases were involved in rock blasting. In the case of air-blast data they observed that as more rock was broken less air-blast energy was evolved and this meant gaseous products were better utilised, which in turn broke more rock.

Similar observations were made by Livingston (1956) while carrying out crater tests. He observed that as the depth of a constant charge increases, the amount of rock broken first increases and then it decreases till it reaches a depth where no fragmentation takes place. At this later depth, termed as critical depth, all the energy of the explosive is consumed in local crushing and seismic waves. When the charge is near the surface, gases form a bubble and due to this rock bulges out and extreme fibres get fragmented.

Kutter (1967) visualises that subsequent to outgoing pressure pulse establishing a system of primary radial fractures, the expanding gases penetrate these cracks and develop high forces in the radial and tangential directions. The resulting tensile stresses cause further extension of primary radial cracks.

Porter & Fairhurst (1970) suggested that quasi-static gas pressure in the hole gen-



a

b

Figure 8.10 Reassembled fragments from (a) 35 mm and (b) 40 mm burden tests in granite blocks.

erates stresses in the rockmass and assuming strain-energy-density criterion according to which cracks would be expected to grow as long as strain-energy at a point exceeds the critical strain-energy level for that particular material. The direction of crack extension would be along a path requiring the least amount of work and crack would follow one of a family of principal stress trajectories, or isostatics. An iterative process was used to calculate the stress trajectories or isostatics.

In a small scale investigation (Bhandari 1975a, 1979a) reassembly of fragmented material as in Figures 8.9 and 8.10 showed that radiating cracks originated from the holes progressed up to the free face. Tests at critical burdens also showed radiating cracks originated near the hole and progressed to the free face. Even at larger scale in quarry blast rock fragments showed similar development of cracks (Fig. 8.11). The study when extended to the stress distribution in rock due to gas pressure can be used to obtain stress trajectories. A comparison between the experimental results and the stress trajectories plots from the simulation study showed that the directions of radial cracks in single hole experiments seemed to follow some of those stress trajectories

Figure 8.11 Development of cracks in large fragment in quarry test.

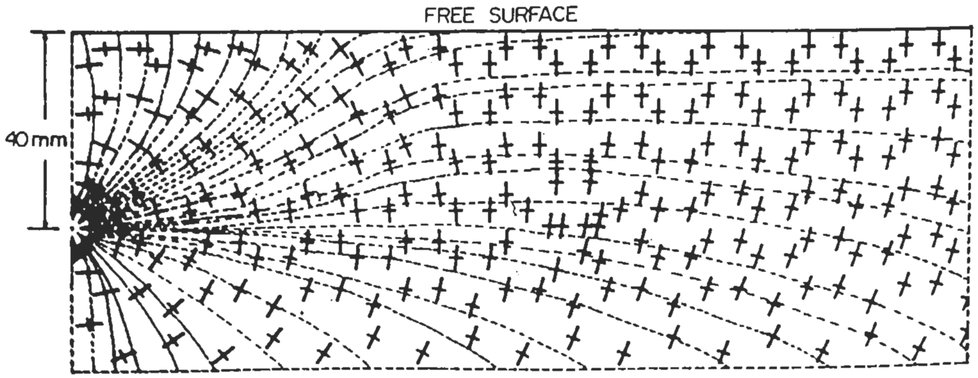


Figure 8.12 Stress trajectories plotted for a burden of 40 mm, **with single hole.**

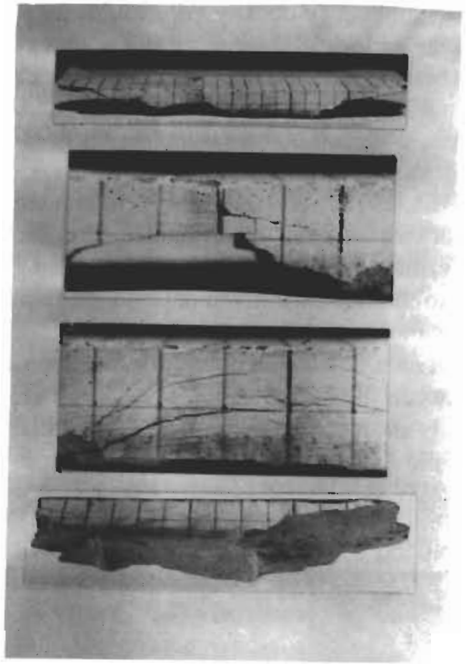


Figure 8.13 Reassembled fragments from test with 40 mm burden and 120 mm spacing. Enlarged views on either side of middle hole shown in (b) and (c).

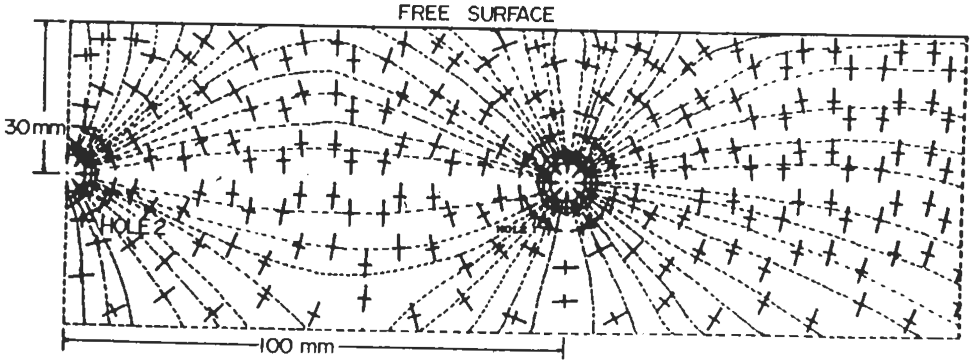


Figure 8.14 Stress trajectories plotted for 30 mm burden and 100 mm spacing with multihole.

which ran towards the free face (Fig. 8.12). At smaller burdens, the number of cracks originating and propagating appeared to increase. Similarly for multihole blasts as observed in small scale tests radiating cracks were found to follow similar path as some of the stress trajectories (Figs 8.13 and 8.14).

Brost (1971) presented evidence from dynamic photo-elastic observations that the gas penetration was limited to small distance from the hole and that the cracks can extend under the influence of the stress field generated by the gas action alone, without penetration of gases into the cracks. High speed photography of blasting has shown additional fragmentation occurring during gas-driven heaving of rock thrown from the bench (Hagan 1977; Chiappetta et al. 1983).

Figure 8.15 Reduced scale tests showing bending of the burden rock.

According to Kovazhenkov (1958) major work in blasting is done by energy left in gases after shock waves have initiated rock fracture process. Kovazhenkov says that after initial rock breakage at the free face by the shock waves, the remaining 60 to 70% energy of the explosives is transferred to the rock between the crushed zone and initially fragmented rock at the free face due to compression by pressure of gaseous products. The energy is stored in the form of potential energy. Kovazhenkov (1958) postulated that because of relief of this energy rock fragmentation takes place. The period of over-all relief involved in a blast is 5 to 10 times the period between detonation and emergence of the shock wave at the free surface.

Ash (1973) suggests that in bench blasting with long cylindrical charges, transverse fractures should form due to tangential stress components generated by highly compressed rock as it bends on being displaced outward by expanding gas pressure. A cantilever beam model was proposed with the thickness equal to the burden. Borehole pressure was considered to act as a distributed load along the length of the blasthole containing burden. In a reduced scale test evidence (Fig. 8.15) of such bending was observed (Bhandari et al. 1979). The degree of fragmentation is controlled by the stiffness property of the burden-rock mass. This stiffness depends on existing restraints to the movement, rock (Young's modulus), radially – cracked block's geometric shape as defined by its average thickness, width, and length. To achieve adequate flexture rupture, the burden to length ratio becomes critical because stiffness varies with the third power of this ratio. For a given explosive diameter and burden value, decreasing the bench height has the effect of stiffening the burden rock and reducing the fragmentation. Reducing burden for a given bench height has the opposite effect.

Thus it can be seen a number of ways have been suggested through which rock is fragmented by the action of quasi-static gas pressure.

It was concluded and supported by many investigators (Kutter & Fairhurst 1971;

Bhandari 1975a, 1979a; Hagan & Just 1974) that rock fragmentation in homogeneous rock results from joint action of stress waves and quasistatic gas pressure but still it is not clear how both combine and how much is the part played by each.

8.5 FRAGMENTATION PROCESSES IN JOINTED MEDIA

In the preceding paragraphs rock fragmentation has been considered without taking into account rock mass weaknesses or local inhomogeneities. In practice blasting has to be accomplished in rock mass having fractures, beddings and/or joints. Rock formations as they occur are not homogeneous and isotropic and even on small scale the homogeneity varies. These variations occur due to difference in origin of formations, textural features and structural control, etc. All the depositional discontinuities, textural features and deformation structures such as bedding planes, joint planes, fault planes, fracture planes, cleavage, foliation can be termed as 'discontinuity or joint'. The structural control has a considerable influence on the geomechanical and dynamic properties of the rock formations. The strength of rock mass decreases with the increase in frequency of joints and the deformability of rocks depend on their orientation. It is the interaction between the rock mass and stresses generated due to explosive detonation which may produce favourable or harmful blasting results.

The influence of discontinuities and joints on blasting of rockmass has been studied by a number of researchers either on full scale blasting in field or on reduced scale blasting in quarry (Ash 1973; Bhandari et al. 1973; Larson & Pugliese 1974). It has been shown that structural discontinuities have greater influence than the explosive properties and blast geometry (Ash 1973; Bhandari 1974; Hagan 1973). Barker & Fourney (1978) and Fourney et al. (1983) stated that the fragmentation is considerably different in jointed material. Hagan (1980) has discussed in general terms effect of joints, bedding planes, hard bands, vughs on the blasting results. Sometimes the joint planes add to the performance of explosive induced fragmentation mechanism. If the propagating transient stress waves encounter discontinuity in a favourable direction then it will increase the performance of the stress waves, whereas the adversely oriented discontinuity will cause adverse results.

Kutter & Kulozik (1990) attempted to explain the possible role of stress waves and quasistatic gas pressure on the joint interface for explaining the fracture process involved.

In studying free surface fracturing in limestone, Winzer & Ritter (1980) noted surface crack formed independently of radial crack development. Timed photography showed the effects of bedding and joint (inherent flaw) influence on cracking during burden acceleration. Trapped waves reflecting in detached rock particles contributed to continued particle size reduction.

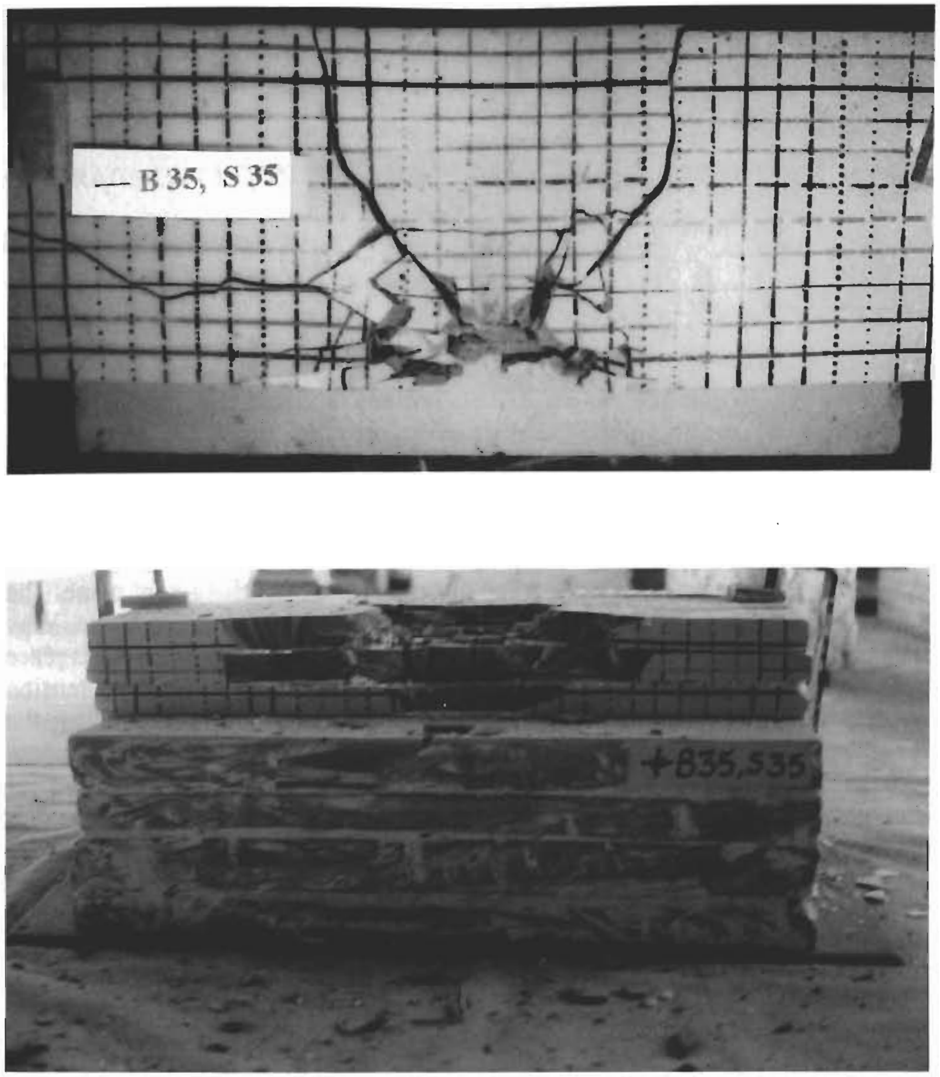
For the simple case of crater formation by a single contained explosion with horizontal bedding, rock breakage primarily occurs along significant bedding planes and secondarily across the beds themselves (Gnirk & Pfleider 1967; Bhandari et al. 1973; Bhandari 1974). As a consequence, the wall of the crater, as a whole is rather steeply inclined, definitely exhibits a discontinuous stepped profile owing to breakage across the individual beds (Fig. 8.16). The depth of the crater is influenced by major bedding planes above or adjacent to the explosion source. Where rock mass has both

Figure 8.16 Stepped crater profile.

horizontal and vertical joint systems, the crater formed by a single contained explosion is affected by the both. If the joints are closely spaced the walls of the breakage will quite probably correspond to the joint planes. If the joints are widely spaced, the breakage may take place across a sequence of beds rather than along joint plane. The bottom of breakage may be flat and may correspond to a major bedding plane. Why does this happen? Explanation given was as follows: When the explosion takes place the reflected transient wave at the free surface causes the rock to break in tension across major bedding planes. If the joints are present then the breakage along the bedding planes is effectively terminated at the joint planes. The simultaneous expansion of the residual gaseous explosion product cause tensile fracturing along major bedding planes in the vicinity of the explosion source. As a consequence, the gaseous products can escape into these fractures and in so doing exert a more widely distributed thrust on the rock above. The net effect is that the areal extent of influence of the gaseous products is sufficiently great to cause increased rock breakage above and laterally to the explosive source.

Problems are created when bedding and joints make up large preformed blocks. As explained above, stress wave utilisation is inefficient because of poor transmission from one block to another. Some rocks have so many natural planes of weaknesses that they require just a little displacement to affect good shovel loading. Others require large displacement. The displacement work is mainly achieved by gaseous products. Several factors are important for good displacement, the first being adequate mass confinement. The gaseous products should not be allowed to vent before movement is imparted to the rock mass. Another effect is not to have too much confinement. If it is assumed that the rock has been broken properly in its place, the only job left to do is to displace it. If the same explosive in the same diameter drill hole is too deep, the gaseous effect energy will become too much confined and high bottom will develop.

Bhandari (1983), Bhandari & Badal (1990a), Badal (1991), Badal & Bhandari (1992) and others have studied on reduced scale and on production scale, the relationship between orientation of joints and some blast parameters. Unlike in field blasting situations where variations of rock and its discontinuities are not always apparent either on the top or at the face of the bench, in the laboratory tests joints were

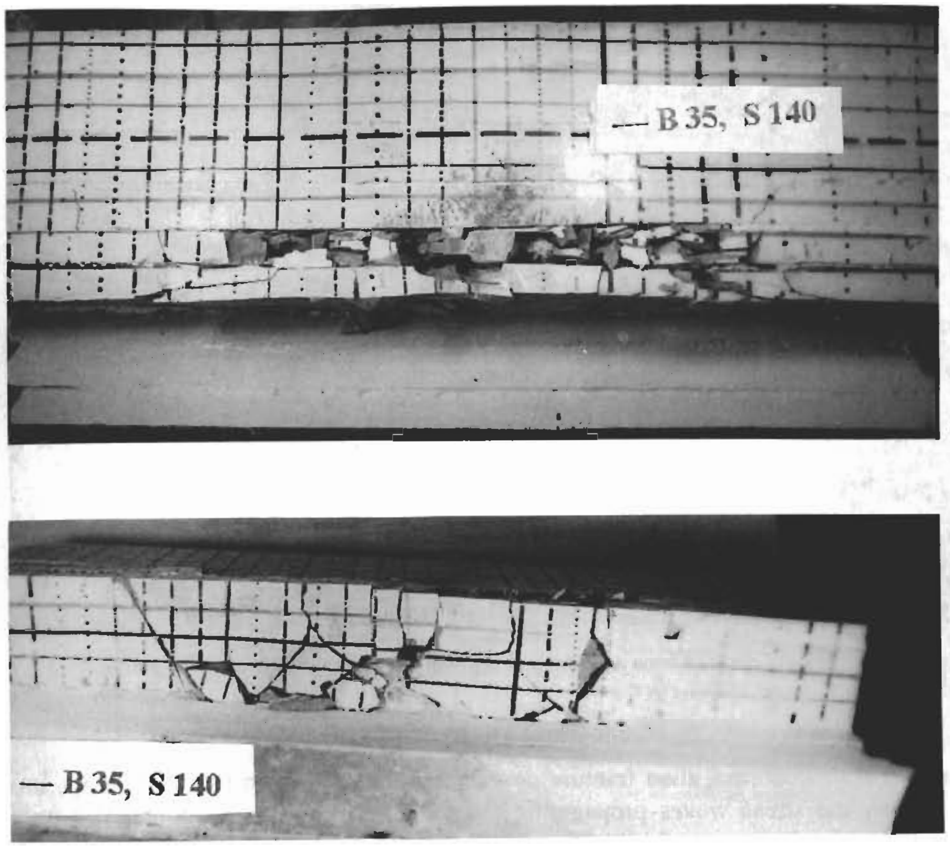


a

b

Figure 8.17 Blasted block with horizontal joints. a) Top view of reassembled fragments for horizontal joints (showing fracture pattern and b) Front view of reassembled fragments for horizontal joints.

placed in known directions. Figure 8.17 show the front and top view of a blasted block with horizontal joints. Based on the fractures present, the fragmentation process can be explained as: The detonation of explosive inside the hole, formed a crushed zone at the periphery and compressive wave propagated in all directions which resulted in formation of radial fracture pattern starting from the blasthole walls. The propagating stress waves, were responsible for extension of radial fractures to greater length, which also developed in the block behind the hole. In all slabs forming bench, the development of fracture pattern was not identical because of the



a

b

Figure 8.18 View of the blasted block with vertical joints parallel to the face. a) Top view of reassembled fragments for vertical joints parallel to the face and b) Front view of reassembled fragments.

different stress wave systems. It is believed that the propagation of the stress waves arrived at the free face after crossing the burden distance from where these waves got reflected and travelled back towards the borehole axis. During this interval the newly reflected waves have sufficient strength for further fracturing and extension of fractures which were formed in the earlier stages of stress wave motion resulting in fine fragmentation between blasthole and the free face. The fracture network was so dense that the reassembly of the broken fragments became very difficult for this study as can be seen in Figure 8.17. And the stress waves which travelled towards the side and in back of the block, attenuated and weakened gradually on reaching the ends of the block, so that the reflected waves do not carry sufficient energy for further fracturing of rock on the bench, only large pie shaped fragments could be created in the adjoining area of the blasthole.

Figure 8.18a and b show front view of a block with vertical parallel joints. Based on the presence of fractures the fragmentation process can be visualised: After detonation of the charge, the stress waves propagate in all directions. First formation of

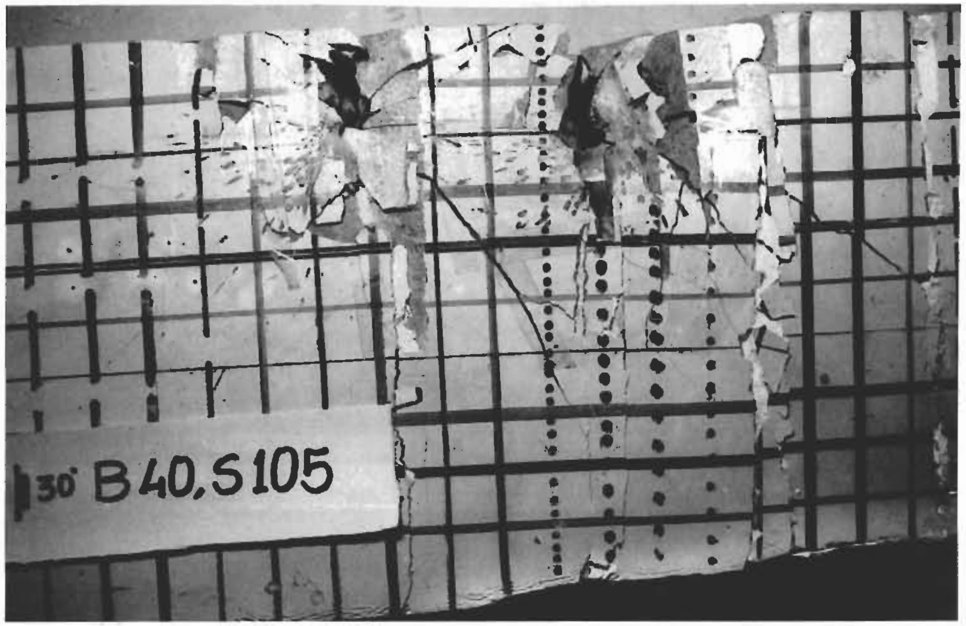


Figure 8.19 Intensely cracked zone near the hole and very few cracks beyond joint planes.

crushed zone occurred, then fracture development started from the blast holes, immediately the stress waves propagating towards the free face, encountered a joint plane. At this plane, a considerable amount of attenuation of stress waves took place, some part of it reflected and travelled back to the blasthole, and remaining part of the waves which propagated towards the free face, was weakened and were unable to create hardly any fracture. Thus the stress waves caused small fracture in the slab at free face and reflected back in the form of tensile waves towards the place of origin. Since these waves were too weak to create any new fractures, they could only add in the extension of fractures, causing a highly fractured zone in the slab containing explosive as shown in Figure 8.19. No significant fracture could grow from the blastholes. The slab containing explosive was subjected to the repeated loading of stress waves, which had enough strength in the reflected tensile stress waves to propagate the fractures and further growth of the fractures resulted in a dense fracture network.

The propagation of stress waves towards the side ends of the block created parallel fractures in the direction of propagation. The reflected waves which travelled back from the side end of the block were too weak to create any useful fragmentation except the possibility of increase in fracture width.

Figure 8.20 illustrates the front and top of blasted block containing vertical joints perpendicular to the face. In this orientation of joints the vertical blastholes were not crossing any of the joints and the slab containing hole and explosive charge extended up to the free face. On detonation of explosive inside the hole stress waves propa-

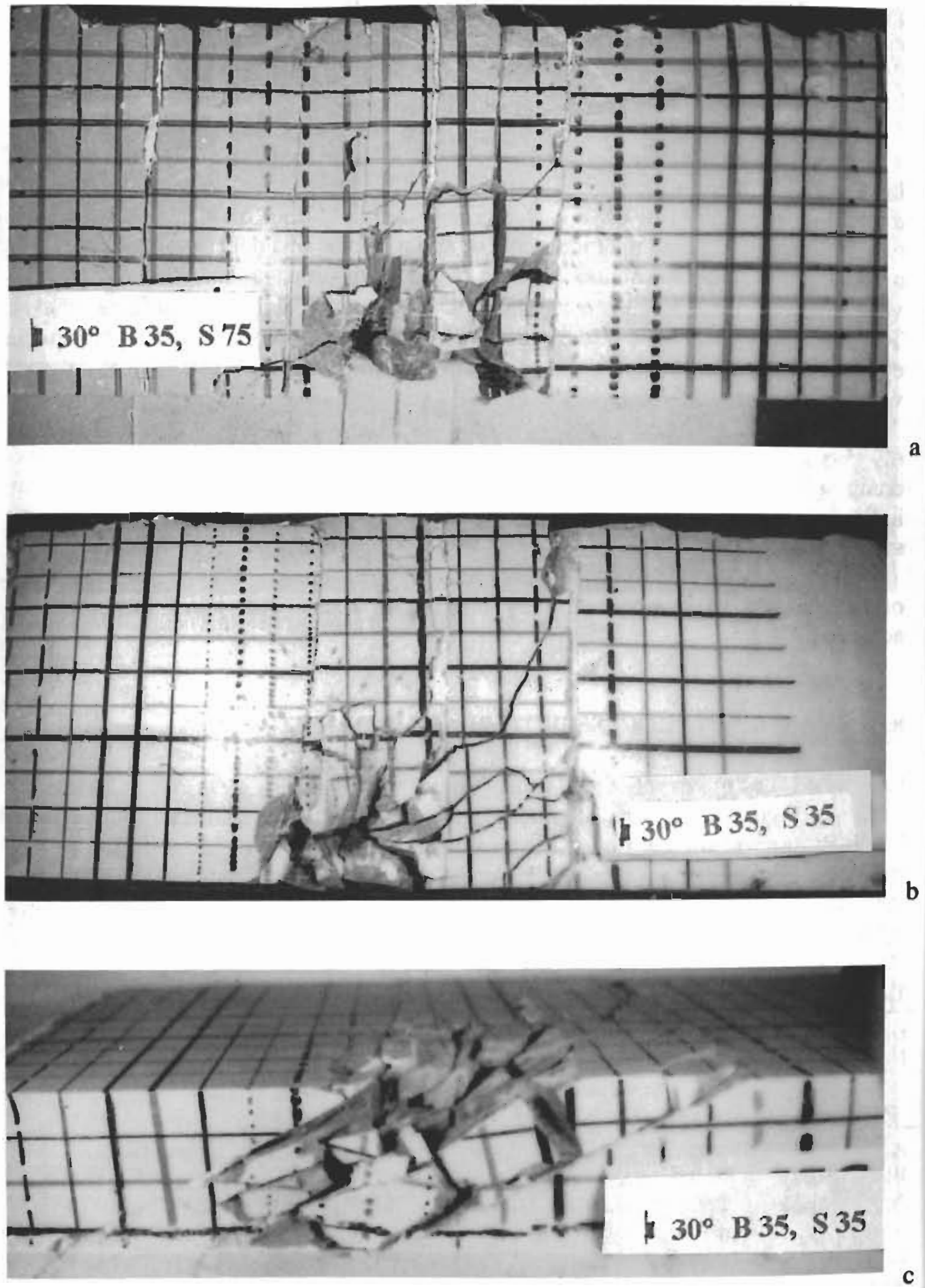


Figure 8.20 a) Top view of reassembled fragments for vertical joints perpendicular to the face, b) Top view of reassembled fragments for vertical joints perpendicular to the face, c) Front view of reassembled fragments for vertical joints perpendicular to the face.

gated in all the directions towards free face. When these come across inherent micro-cracks and weakness planes then initiation of the cracks takes place which grow further. The reflected waves from free surface intensify the fracture network. The breakage extent is reduced and terminated at joint planes.

Similar principles were applied for the blast holes fired simultaneously in a row. The waves propagated in all directions from one hole to another, but in between a large number of joint planes had to be crossed by these stress waves. A significant amount of attenuation in these waves occurred, due to high frequency of joints between the holes. The attenuation of these stress waves was dependent upon the frequency of the joints encountered in the direction of propagation. Initiation and development of dense fracture network took place in the slab containing blastholes. The repeatedly attenuated stress waves were reflected partly from each joint plane encountered in the direction of propagation, and hence form a dense fracture network which resulted in fine fragmentation.

The filler material of the joints also have influence on the fragmentation. Bhandari (1975a) showed that the fine fragmentation increased in case of cemented joints compared to joints which were filled with weaker material or were open joints. Thus indicating that participation of stress waves was better in case of joints filled with strong filler material.

However, as yet due to involvement of numerous factors complete understanding of the process of fragmentation by blasting in rocks with joints has not been achieved.

8.6 CHANGES IN PROCESS WITH BLAST PARAMETERS

Controlled blasting experiments (Bhandari 1990b, Bhandari, 1996) on a 1.0 m high bench showed that the smaller burden gave smaller volume of rock in small sized boulders but maximum throw distance was very large with comparatively less ground vibration levels (Table 8.1). At the largest burdens volume of rock broken was very large, with large size boulders thrown to a small distance with large vibra-

Table 8.1: Results of blasting tests with variable burden/spacing.

Blast No.	Burden (m)	Spacing (m)	Total breakage volume (m^3)	Vol.of ten largest boulders (m^3)	Max. throw (m)	Σ throw distance (m)	Σ throw index	Peak particle velocity (mm/s)
AF 3	0.50	0.50	2.34	1.31	200	164.5	12.90	42.5
BF 2	0.50	1.00	2.78	0.84	75	156.0	14.50	47.5
AF 10	0.50	1.50	3.80	0.63	40	149.0	12.54	60.0
AF 4	0.75	0.75	2.81	1.75	100	257.0	40.86	75.0
BF 1	0.75	1.50	5.10	3.61	35	136.5	65.50	62.5
AF 6	0.75	2.25	6.75	1.64	17	98.5	22.62	47.5
AF 5	1.00	1.00	4.40	3.74	20	89.0	34.01	70.0
BF 3	1.00	2.00	8.80	7.74	3	27.0	22.08	87.5
AF 1	1.00	3.00	12.50	10.99	3	12.5	22.67	100.0

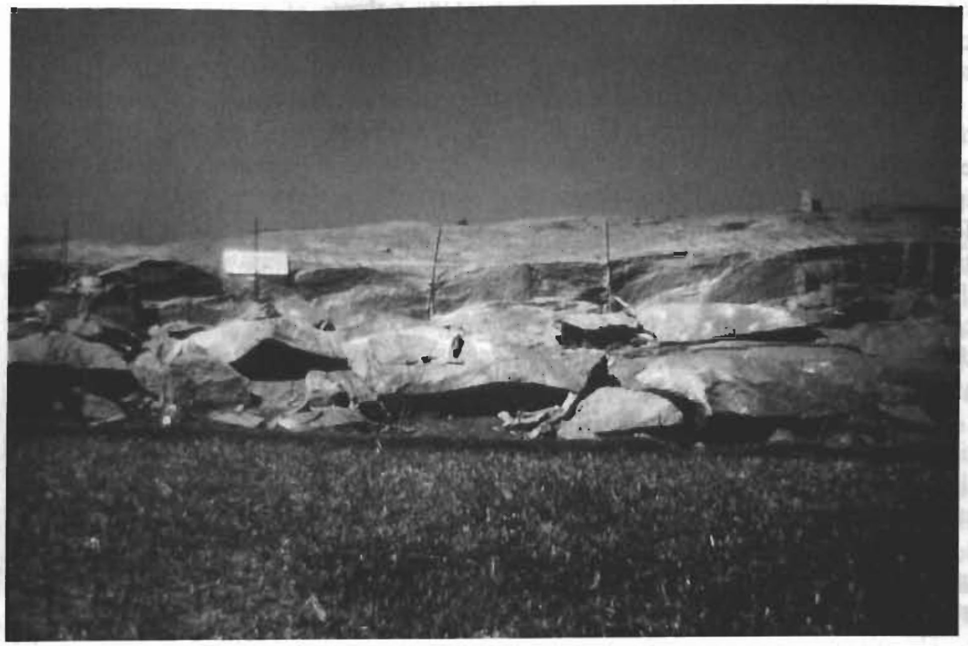


Figure 8.21 Reduced scale test showing results with large burden.

Figure 8.22 Reduced scale tests at intermediate burdens.

tion levels (Fig. 8.21). At intermediate burdens volume of rock broken was large only at larger spacing, size of large boulders was small, the sum of throw distances of ten largest boulders was fairly large but vibration levels were less (Fig. 8.22).

These tests show that with respect to fragmentation, the results are similar to as that for small scale tests (Bhandari 1975a, b). It can be deduced that the utilisation of explosive energy changes for the same rock and the same explosive with the change in blasting parameters. This is on account of quantitative changes in the role of different mechanisms involved in blasting. At smaller burdens stress wave energy is better utilised in causing scabbing fractures as well as in increasing microfractures in the rock. At small burdens gas energy does not get enough time to cause radial cracks and their bifurcation, before it is vented. It results in greater distances of throw. Further, its transfer as seismic energy is lesser thereby causing reduced amount of vibrations.

At larger burdens the role of stress waves is considerably decreased in fragmentation but the waves are transferred to the surrounding rock as seismic wave and thereby causing increased vibrations. Rock broken is coarser because the role of stress waves is decreased. The amount of throw is considerably reduced.

At the intermediate burdens the boulder size is not very large and the throw distance is very large. Vibration levels are lesser. This suggests that more energy is utilised in fragmentation.

The effect of increased spacing is that each hole is breaking separately and the closer spacing results in enhancement of seismic energy resulting in increased vibration and airblast. One must utilise that part of energy which is needed for the type of fragmentation and other blasting results by altering blast design. The concept could be further extended to the selection of explosives and initiation pattern.

8.7 INFLUENCE OF ROCK TYPES

The influence of rock types can be explained by classifying rocks into two main categories, elastic and plastic. Rocks such as granite or quartzite are examples of elastic rocks as they can transmit stress waves and have high compressive strength and high Young's modulus. Since elastic rocks transmit stress wave well, comparatively greater fragmentation results from stress wave scabbing and other effects. Development of cracks due to stress waves would enhance gas effect. Rocks such as some limestones, sandstones and porphyries are of plastic acting type which are relatively low in compressive strength and absorb stress wave energy at a much faster rate, thereby making poor use of the stress wave energy by not developing extensive cracked zone for gases to work. Because the ratio of gas effect to stress wave energy varies in different explosives it is easy to understand why some explosives perform well in elastic rock and poorly in plastic acting rock and vice-versa. In competent dense rock, excellent fragmentation occurs when using explosives that produce high shock energy. In general, explosives formulation which produce large volumes of gas contribute to good heaving and throwing. Their use in well-bedded and jointed material such as sedimentary rock or highly altered ore, promote good fragmentation.

Bauer et al. (1965) observed that brittle rocks are fragmented mainly by scabbing,

whereas in plastic rocks material is pushed out by expansion cavity and outward deformation.

8.8 PRESENCE OF STRESS

In deep underground high static stresses are usually present (Obert 1962). On account of these stress conditions two effects occur. The first is that of pre-existing cracks crossing the borehole. This alters the growth and development of fractures under the action of explosives. The second effect is due to stress concentration around the borehole itself. The fracture pattern is influenced by the presence of non-uniform stress field. The difference between two principal stresses in the plane normal to the blasthole axis causes non-radial fracture pattern. The cracks from a blasthole, particularly in massive rocks, start to propagate radially but tend to develop parallel to the direction of the major principal stresses (Fig. 8.23).

In order to move the rock from the face, the heave of the explosive, to a first approximation, has to work against the friction along the hanging wall and footwall contact. This resisting force F, will be proportional to the coefficient of friction of the contact μ and the vertical stress σ_V that is: F increases with distance ahead of the face in approximately linear fashion over small distances involved. Thus the work required to heave the rock from a face increases as does the burden.

Considering first the stress wave, only that fraction of energy radiating towards

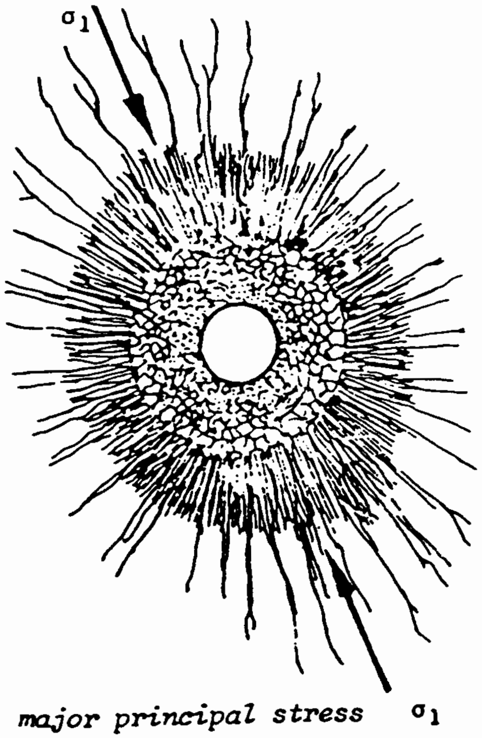


Figure 8.23 Non-uniform growth of fractures due to presence of stress.

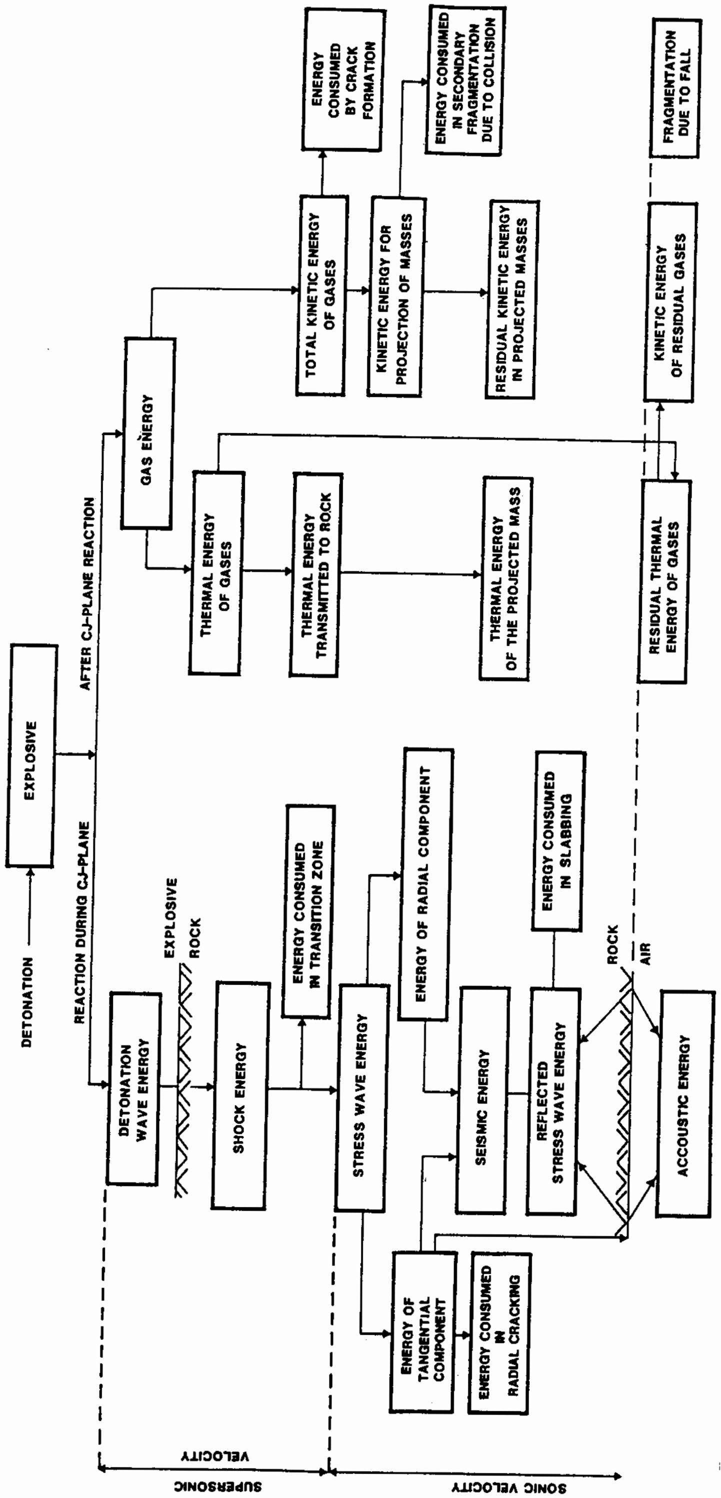


Figure 8.24 Explosive energy utilisation in fragmentation processes.

the free face is able to accomplish breaking. This fraction is proportional to the angle subtended by the free face at the hole. Thus the useful work which can be extracted from the strain wave energy decreases with increasing burden. Considering gas pressure in a near vertical crack the net force it produces to overcome friction depends on the difference between the gas pressure P and the horizontal stress σ across such a crack. The effectiveness of gas pressure is therefore proportional to the difference $(P - \sigma)$. Since σ increases with burden, the effectiveness of the gas pressure decreases with the burden.

8.9 ENERGY UTILISATION AND PRESENT UNDERSTANDING OF THE FRAGMENTATION PROCESS

As a result of detonation of explosive energy is released some of which is gainfully utilised and much of it is wasted. Useful part of the energy is capable of doing the work of fragmentation and displacement and wasteful part of energy can be put to no work, which causes many harmful effects. Useful energy is in the form of stress waves and gaseous pressure.

Any one mechanism, as to how the useful energy is utilised is not adequate to explain the whole process of fragmentation involved in various conditions and material types. The energy evolved on detonation of explosives is utilised in the fragmentation process by two groups of mechanisms (Fig. 8.24). First, a stress wave of extremely short duration results from the detonation of the explosive; it is followed by quasi-static gas pressure generated by the gas products of explosion. A small zone of crushed rock is created immediately surrounding the hole, on detonation of explosive. Intensity of crushing and fracturing decreases as the distance from the hole walls increases till it reaches the transition zone beyond which other effects occur. The stress pulse propagates as cylindrical or spherical detonation wave into the surrounding rock and induces besides the radial compressive stress, a circumferential tensile stress around the borehole. As this stress exceeds the tensile strength of rocks a pattern of radial fractures is created. As the stress wave travels outwards from the borehole, its amplitude is rapidly attenuated, so that after some distance no further crack initiation and eventually no crack propagation can occur. If, however, this stress pulse reaches a free surface, it is reflected from there and its originally compressive radial component is reflected as tensile stress. This newly generated tensile stress may be of sufficient magnitude to exceed the tensile strength of the rock, and this results in surface parallel scabbing or spalling of the rock. Multiple reflections of outgoing and reflected waves occur while fracturing takes place, dictating flaw initiation sites.

As a result of quasi-static gas under high pressure acting in the widened borehole and on the surfaces of the radial fractures, further propagation of the cracks occurs. The gases also find their way into the stress induced radial fractures. In addition, flextural failure may occur at the surface, when the layers between cavity and free surface are bent outwards by the expanding gases. The resulting rock fragments are finally pushed outwards and ejected. During ejection process there is consumption of energy in the collision of fragments and further fragmentation takes place.

In the presence of joints, interaction between joints and related mechanisms for rock breakage take place and separation of rock mass loosening also occurs. In some situations the role of transient stress waves in jointed rocks dominates the fragmentation phenomena. In some other situations, the role of quasi-static pressure is to aid to the fragmentation of weak jointed formations. The orientation of joints changes the growth of fractures.

The explosive detonation also produces energy which does not in itself, lead to fragmentation and does no useful work during blasting operations. This energy can be called as waste energy which finally yields acoustic energy, thermal energy in the fragmented mass and released gases, light energy and seismic energy.

Suitable blasting designs and systems are needed which are adjusted according to blast conditions and desired results, to utilise explosive energy efficiently in useful work rather than part of it being wasted. Adequate understanding of the fragmentation process is needed to utilise greater percentage of energy in doing useful work.

CHAPTER 9

Influence of rockmass characteristics on blasting

9.1 INTRODUCTION

To carry out blasting operations in an optimum manner it is essential that the influence of rock properties on the blasting process, operations and the results is fully understood. Unfortunately no single property characterises a rockmass. The quantitative classification system correlating fragmentation with various geomechanical properties have been only moderately successful. None of the conventional geomechanical rockmass classification schemes predict blasting performance. In addition, in many mines, the nature and properties of the rock mass and/or ore body change considerably over short distances. Such variability can occur in lateral direction as well as with increasing depth. A thorough understanding of the influence of the rockmass parameters along with the other parameters involved in blasting and fragmentation is essential.

Important geomechanical characteristics of rockmass which affect the blasting results are the strength properties, elastic properties like Young's modulus, longitudinal wave velocity, porosity, density, specific damping capacity, ground water and moisture content, conductivity, the presence of static stress field (Gnirk & Pfleider 1967; Grant 1970; Hagan 1973). Considerations to the presence of various geological discontinuities like joints, bedding planes, faults, their filler material and orientations, presence of hard or soft bands or cavities are also very important for achieving desired blasting results (Bhandari et al. 1973; Bhandari 1974, Bhandari 1975a and b; Bhandari & Badal 1990a; Badal 1991). Continuous efforts are being made to relate the rock mass characteristics with the blast results. This chapter provides details of information available with respect to important rockmass characteristics influencing blasting operations and results.

9.2 STRENGTH PROPERTIES

It has always been recognised that the strength of rocks is an important criteria for selecting explosive type and charge amount but quantitative values relating them are lacking. At constant energy levels in high compressive strength rocks, fragmentation results obtained are poor. Also the major influential factor on the blastability (resistance of the rock to blasting) of the rock is strength of the rock. Sassa & Ito (1974) indicated that the blastability was affected by its strength and also by its brit-

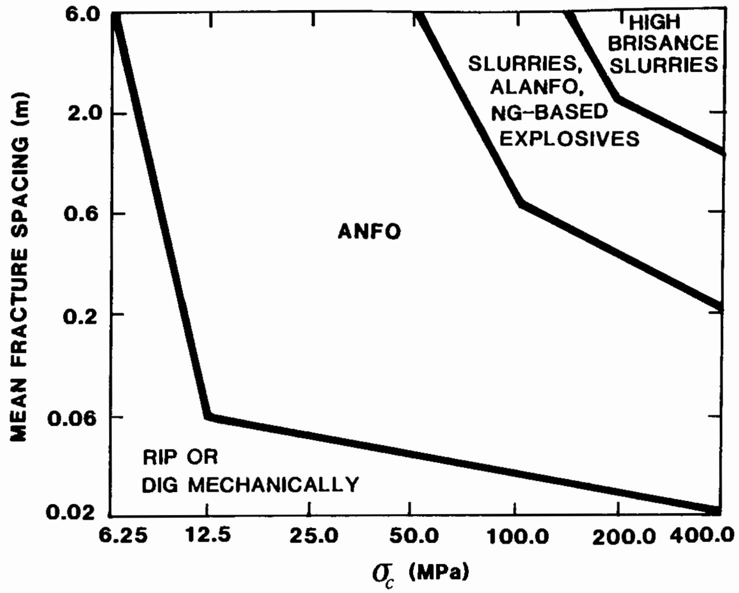


Figure 9.1 An empirical integration of explosive type and rock mass properties (Brady & Brown 1985).

tle index. Brittle index is the ratio of the uniaxial compressive strength to uniaxial tensile strength and was found to vary between 10 and 100. At higher ratios rocks are easier to blast. Brady & Brown (1985) suggested that uniaxial compressive strength (σ_c) may be used to represent the ease of generating new fractures in the medium. An empirical relationship between uniaxial compressive strength and explosive type was given as in Figure 9.1. An important feature is that ANFO is suitable explosive for use in wide range of rock mass conditions. The use of high brisance explosives is justified only in the strongest and more massive rock formations.

In the erstwhile Soviet Union a scale Soyuzvzryprom (Table 9.1) is adopted for blasting purposes, in which rocks are grouped into categories. The lowest category has minimum strength. This is also based on the unit consumption of Ammonite No. 9 (AN based blasting agent) which produces crater of standard dimensions (Rzhevsky & Novik 1971).

Such generalised classification does provide guidance but an explosive which has an excellent effect in a hard and homogeneous type of rock is possibly not so effective in soft rock types with natural crack systems and cavities. The strengths of rocks increase with the rate of loading (Rinehart 1958). The rate at which strengths increase varies with rock type; the weaker the rock and lower the tensile strength, the greater is the relative increase in strength with increase in loading rate.

One of the major problems besetting blasting optimisation lies in the uncertainty of the properties and behaviour of rocks under the extremely high loading rates associated with explosive attack. Strength-loading rate relationships have not yet been accurately defined, especially for loading periods less than 1 ms. (The loading period at the blasthole wall is usually less than 0.5 ms). There is a need to determine the insitu strengths of rocks for such loading periods. Rinehart (1960) observed that the dynamic fracture strengths were 5 to 13 times greater than the corresponding static strengths.

Table 9.1 Scale of rocks on the basis of blastability.

Sl. no.	Rock	Category	Explosive consumption kg/m^3 for charge of normal crushing
1.	Fat soft clay, heavy clay, morainic clay, slate clay, heavy loam, coarse grit	III-IV	0.3-0.5
2.	Marl, brown coal, gypsum, tuff, pumice stone, anthracite, soft limestone, diatomite	V-VI	0.35-0.55
3.	Clayey sandstone, conglomerate, hard clay shale, marly limestone, anhydrite, micaceous shale	VII-VIII	0.45-0.6
4.	Granites, gneisses, synites, limestone, sandstone, siderite, magnesite, dolomite, marble	IX-X	0.6-0.7
5.	Coarse-grained granite, serpentine, audisite and basalt, weathered gneiss, trachyte	XI-XII	0.7-0.75
6.	Hard gneiss, diabase, porphyrite, trachyte, granite-gneiss, diorite, quartz	XIII-XIV	0.85
7.	Andesite, basalt, hornfels, hard diabase, diorite, gabbro, gabbro diabase	XV-XVI	0.9

In general, static compressive strength and tensile strength values are usually available for mine rock as these are required for mine design purposes. These properties if recorded will help in analysis of blasting designs and results after a period of time. However, since blasting is a dynamic phenomenon, there is a need for using dynamic strength properties. Specialised laboratory facilities are needed for determination of dynamic properties.

If on detonation of an explosive charge, the peak of the outgoing stress wave exceeds the dynamic compressive strength, a zone of intensely crushed rock is formed immediately around the charge (Chapter 8). Further, excessive crushing is also associated with high absorption of stress energy. Thus, the explosive which generates peak stresses higher than the dynamic compressive strength of rock, will have no advantage as it will only enhance crushing accompanied by greater absorption of energy and thus the useful work carried out by explosives get reduced. Therefore, there is a need to determine dynamic in situ compressive strength of the rock and relate it to the peak stresses generated from a given explosive.

Because the dynamic compressive strengths of rocks are greater than the dynamic tensile strength, the latter are more important as tensile fracturing usually accounts for considerable fragmentation such as by spalling (Chapter 8). Specially in rocks with higher strength values spalling phenomena is predominant. In case of softer rocks the attenuation of the outgoing compressive wave itself is greater, hence even though tensile strength is low the scabbing effect is not significant. Further, gas pressure present in the borehole subjects rock to intense radial compression and tangential strains. If these strains exceed the dynamic tensile strength of the rock, radial cracking beyond the crushed zone occurs. Rocks with lower dynamic tensile strength, should therefore, develop radial cracking in a larger zone.

9.3 ELASTIC PROPERTIES

The nature of proportionality between elastic strains and stresses depends on the bonding between the particles and is evaluated by the elastic properties of the rock. Young's modulus, Poisson's ratio, longitudinal wave velocity, bulk modulus are some of the properties which have to be considered in blasting.

The modulus of elasticity characterises the rigidity of rock and its capacity to resist external influences. It is difficult for the explosive gases to compress and stretch the rock if Young's modulus of the rock is high. It is found that gas pressure should be less than 5% of Young's modulus for efficient blasting (Anon 1980a). If this is not the case then the crushed zone increases and explosive energy is wasted.

Rocks possessing the lowest Poisson's ratio (υ) fail directly by a brittle failure and those having a high υ fail by plastic means. Rocks in general have υ in the range of 0.2-0.3. For dry and highly weathered material close to the surface υ may be around 0.15. Being generally anisotropic and inhomogeneous rocks exhibit a considerable variation in the measured value of υ. Water content, rock fabric and the nature of voids play a significant role in determining the effective Poisson's ratio. For a given explosive energy level, rocks having lower υ values are likely to give better fragmentation (Sassa & Ito 1974). If acceptable fragmentation is to be achieved, the VOD of the explosive and the peak blasthole pressure should be increased as υ decreases in the range of 0.5-0.2 (Hagan & Harries 1977).

Rocks may be generally classified into two main categories: elastic and plastic acting. Generally, elastic rocks are those having relatively higher compressive strength and plastic acting rocks are those having lower compressive strength. Stress energy is better utilised in elastic rocks whereas it gets absorbed in plastic-acting rocks. In plastic rocks, it is the gas energy which does most of the work (Grant 1970; Hagan 1973). Very difficult situations arise when mixtures of plastic acting and elastic rocks are encountered. This situation often results in poor fragmentation of rock (Grant 1970). Consider a case of granite boulders embedded in plastic-acting matrix (Fig. 9.2). The loss of stress wave due to its absorption by the plastic acting matrix and reduced stress wave reflection at the surface results in poor breakage of the boulders which do not have drill holes in them.

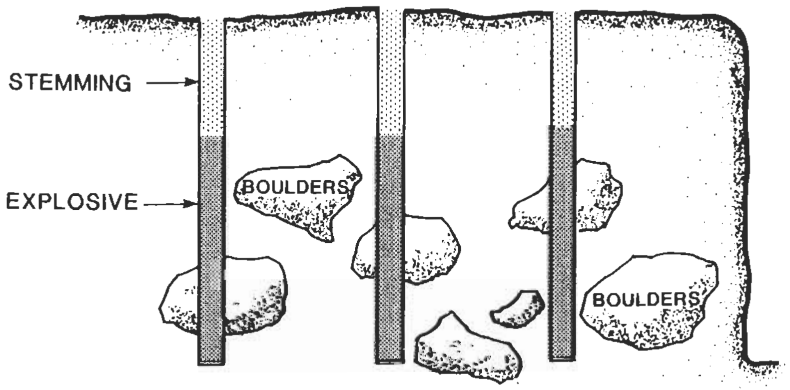


Figure 9.2 Boulders of granite embedded in plastic acting rock.

9.4 WAVE VELOCITY

The distribution and time of stress imposed on the rock by detonating explosive is affected by the velocity of propagation of stress waves in rocks (Johansson & Persson 1970). Longitudinal wave velocity in the rock indicates the velocity at which the rock can propagate the compressive waves. Based on this and the density of the rock other elastic parameters are determined. The higher the longitudinal wave velocity of rock, higher the required velocity of detonation of the explosive (Hemphill 1981). Although, it is recommended that in situ measurements are more relevant, wave measurements have been successfully correlated with blasting results in only isolated cases. Broadbent (1974) made seismic wave velocity measurements and concluded from visual inspection of muckpiles that the fragmentation decreased as velocity increased. Hagan & Harries (1977) made a generalised conclusion that within geological consistent areas, there was a direct relationship between seismic velocity and the powder factor (or, more correctly, energy factor) required to achieve satisfactory fragmentation. As the velocity increases, it is desirable to select explosives with higher detonation velocities.

An indication of ability of rock to attenuate the explosion-generated stress wave has been termed as 'specific damping capacity' (SDC) which in turn indicates the relative significance of the shock energy and gas energy. SDC was found to vary with rock properties. Considerable variations in SDC values were recorded for granites and sandstones (Windes 1950; Blair 1956). SDC increases with jointing, porosity, permeability and water content of the rock.

Unfortunately, the SDC values were determined in laboratory tests using rock cores (Windes 1950). As the SDC values depend upon the amplitude and velocity (Kolsky 1953), direct measurements of stress wave attenuation in rock would be more useful (Hagan 1973; Hagan & Just 1974).

Explosives capable of producing high initial peak stress, e.g. some slurry explosives, were found to be much less effective in soft, decomposed or frozen iron ore than in hard, brittle granite (Cook 1961; Lang 1966). In softer rocks like limestones, explosives of lower stress energy such as ANFO were found to perform better (Cook 1961).

9.5 ROCK DENSITY

In general, the ease or difficulty in breaking the rock is dictated by the density of the rock. It indicates the energy needed to deform and displace the rock and affects the energy propagation properties of the rock. Generally low density rocks undergo deformation easily and need relatively low explosive energy for good fragmentation. Denser rocks need explosives with higher detonation velocity and detonation pressure. However, the porous rocks having lower density absorb energy and make fragmentation difficult.

Characteristic impedance, which is the product of density and longitudinal wave velocity is useful in analysing the transfer of energy from the explosive to the rock mass. Knowing the impedance of the rock, explosives with similar impedance values are often selected. Fracture frequency of the rock and its competency **is not included** in impedance correlation and perhaps the concept of impedance as an index in the selection of explosives has not been successful on this account.

9.6 POROSITY

Porosity of rocks also affects the blasting performance. During blasting of highly porous rocks, greater dissipation of energy takes place and considerable crushing and production of fines occur. The work of fragmenting highly porous rock therefore, is performed almost entirely by the heave energy component of an explosive's total energy output. Consequently, it is important to retain the explosion gases at high pressure until they have completed all the work. This situation is best realised where stemming lengths and burden distances prevent the premature release of explosion gases. Also it would be prudent to provide decoupling or low borehole pressure explosive so that unnecessary crushing and loss of energy is prevented. Porous rocks are susceptible to influence of pore water pressure which reduces the compressive and shear strength considerably (Obert & Duvall 1967). When such rocks become saturated, blast effects are intensified (Ash 1968).

9.7 MINERAL COMPOSITION, GRAIN SIZE AND INTERNAL FRICTION

Mineral composition and grain size also has some influence on blasting performance. They affect density, porosity and strength of rocks. The strength of rocks decreases with the increase in the grain size, other conditions remaining constant (Vutukuri et al. 1974). The cohesive strength between grains in most rocks is less than the strength of the grains themselves. Thus when a rock breaks, rupture takes place between crystals (Rzhevsky & Novik 1971). Therefore, the degree of fragmentation near a charge is influenced by grain size because fractures tend to propagate along grain boundaries rather than through the grains themselves. Rocks with fine and interlocked grains need high energy factors. The larger grained igneous rocks such as granite and diorite usually shatter when blasted.

The value of internal friction depends on texture and moisture content. The internal friction results in considerable attenuation or damping of explosive generated waves. Due to this, in rock with higher internal friction, higher energy factor and large number of blast holes are required for good fragmentation

9.8 STRUCTURAL DISCONTINUITIES

Rock formations are generally inhomogeneous and anisotropic and homogeneity varies even on very small scale. These variations occur due to difference in the origin of formations, textural features, structural control, etc. All the depositional discontinuities, textural features and structural features such as bedding planes, joint planes, fault planes, fracture planes, cleavage, foliation have considerable influence on the geomechanical and dynamic properties of the rockmass and therefore on the blasting results. To simplify, all the above structural features are referred hereinafter as discontinuities or joints. The discontinuities are responsible for making the rock body deformable under the dynamic loading. These discontinuities are inherent in rock either in macro or micro form or both.

The presence of joints influences the behaviour of rockmass in response to various forms of applied loads. The strength of rockmass, its deformation characteristics and strain wave propagation through the rockmass are dependent on the nature of the joints such as their location, their properties and orientations. Sometimes the joint planes add to the performance of explosive induced fragmentation mechanism. If the propagating transient stress waves encounter discontinuity in a favourable direction then it will increase the performance of the stress waves, whereas the adversely oriented discontinuity will cause deterioration of results.

The influence of discontinuities/joints on blasting of rockmass has been studied by a number of researchers on full scale, on reduced scale and on small scale blasting (Ash 1973; Bhandari et al. 1973; Larson & Pugliese 1974). Many have concluded that structural discontinuities have greater influence than the explosive properties and blast geometry. Bhandari (1975a) concluded from small scale blasting tests on bench shaped granite that the effect of joints on average fragment size was significant at 5% level compared to granite without joints. However, this is true only if some predominant joints/discontinuities are present. Hagan (1980) discussed in general terms effect of joints, bedding planes, hard bands, cavities on the blasting results. Several authors have studied on reduced scale and on production scale the relationship between orientation of joints and some blast parameters (Belland 1966; Bhandari 1974; Bhandari 1975a; Bhandari 1983; Singh & Sarma 1983; Singh & Sastry 1986; Bhandari & Badal 1990b; Badal 1991, Badal & Bhandari 1992).

Small scale blasting tests in bench shaped blocks showed influence of orientation of joints (Bhandari 1983; Bhandari & Badal 1990b; Badal 1991). In these tests, joints were placed in the known directions, unlike in field blasting situations where variations of rock and its discontinuities are not always apparent either on top or at the face of the bench.

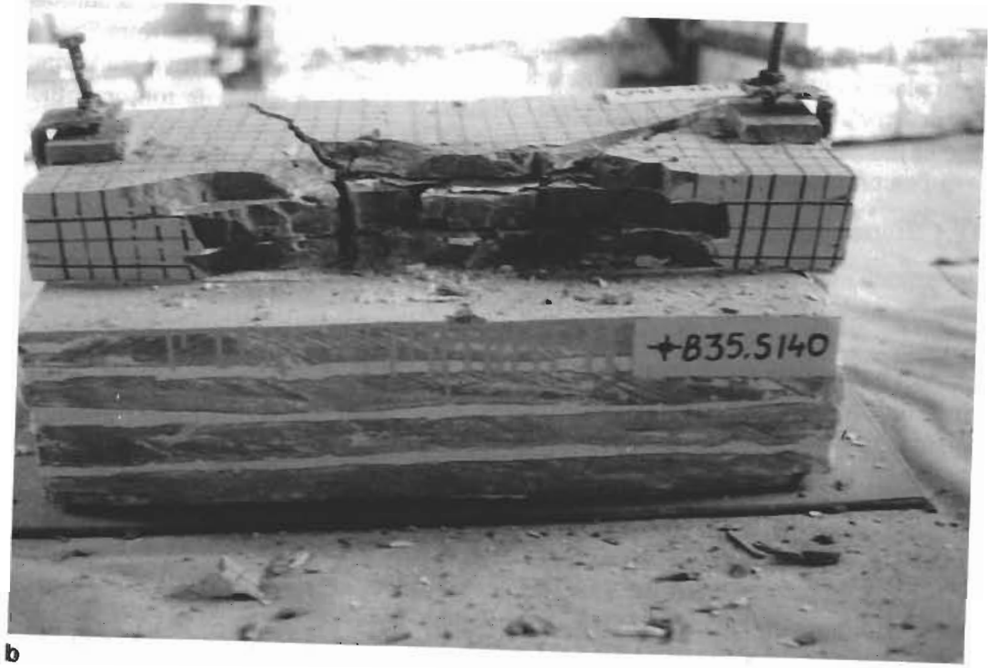
9.8.1 *Horizontal joints*

Horizontal joints are most common types of joints (bedding planes) found in sedimentary rock formations as depicted in Figure 9.3a. Results after blasting are shown in Figure 9.3b. Rocks with predominant horizontal joints give high powder factor (m^3/kg) but uneven fragmentation with some overbreak occurring on the sides of the blast and in the collar zone. Backbreak in beds on top is expected, if large collar heights are used. Fragmentation from the upper part of the bench will be determined by the bedding interval. Structures with this orientation seldom affect the results while carrying out controlled blasting.

9.8.2 *Joints parallel to face*

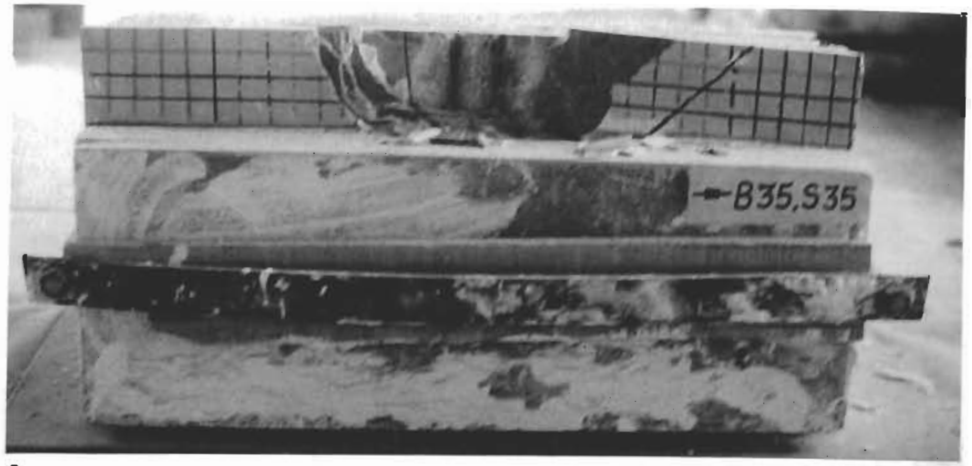
Vertical or dipping joints parallel to the bench face are the other types of joints which occur as shown in Figure 9.4a. Some typical blasting results are shown in Figure 9.4b. Fragmentation obtained is uniform. Backbreak is absent but overbreak at the sides of breakage appears, in the field situation these appear as boulders. Quan-

a

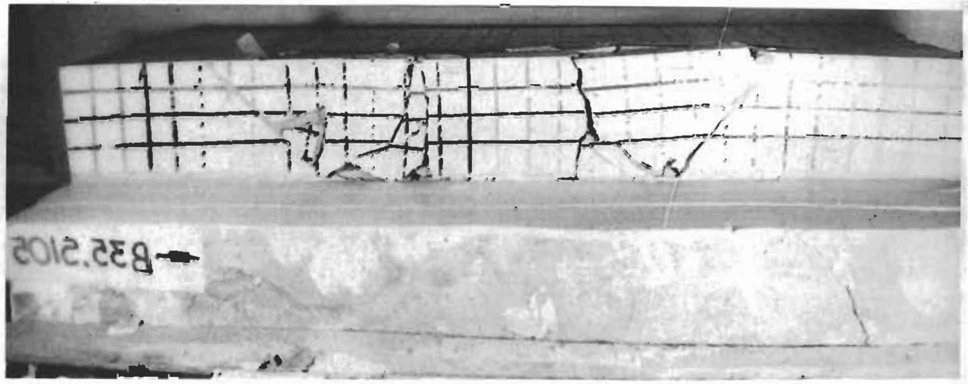


b

Figure 9.3 a) Horizontal bedding (joints) in bench face and b) View of blasted block with horizontal joints.



a



b

Figure 9.4 a) View of block with vertical parallel joints and b) Front view of reassembled fragments for vertical parallel joints.

tity of rock broken for the same conditions is large, thus in general powder factors obtained are large. Fragments are slabby. The face condition is clean. This result has significance for production blasting in that by orienting the face parallel to dominant joints better fragmentation results are obtained.

When the face has joints parallel and the dip of joints is away from the face excessive sliding occurs creating significant overbreak problems (Fig. 9.5a). When the joints are parallel and the joints are dipping into the face there may be overhang problems and excessive toe may be formed (Fig. 9.5b).

9.8.3 *Joints perpendicular to face*

Vertical or dipping joints perpendicular to the face are another type of joints. Figure 9.6a shows such orientation. Figures 9.6b and c show results of blasting. When the joints are perpendicular to the face and the blasting direction is parallel to the strike

a

b

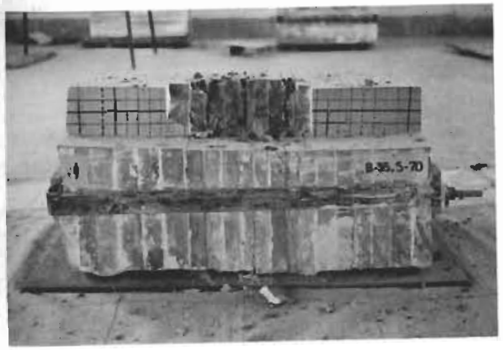
Figure 9.5 a) Block after blasting, with joints dipping away from the face and joints are parallel to face and b) Blocks after blasting with joints dipping into the face and joints are parallel to face.

of joints the powder factor (kg/m^3) is less with uniform fragmentation. Overbreak is absent, though, some backbreak is observed. Floor is uneven and face conditions are not clean. The significance of results is that when a row of holes are oblique to the joint directions, poor fragmentation results.

It is important to know the joint orientation in the pit, so that either the faces can be reoriented or delay pattern can be so arranged that the favourable direction of blasting is chosen. If a record of the blasting results in a mine are kept then it is generally known which direction give good blasting results and which direction give poor blasting results. At the pit wall there is usually no possibility of changing face direction but there is always choice of blasting patterns and initiation direction.

a

b



c

Figure 9.6 a) Perpendicular bedding planes steeply dipping up direction in a limestone quarry, b) Top view of reassembled fragments for perpendicular vertical joints, c) Front view of reassembled fragments for perpendicular steeply dipping jointed blocks.

9.8.4 *Joint spacing*

The spacing or frequency of joints plays a vital role in fragmentation of the rock mass. The resistance to blasting increases as block size increases or joint frequency diminishes (Da Gama 1983). The strength of rockmass decreases with the increase in the frequency of the joints. In highly jointed rocks with weak filler material it is often

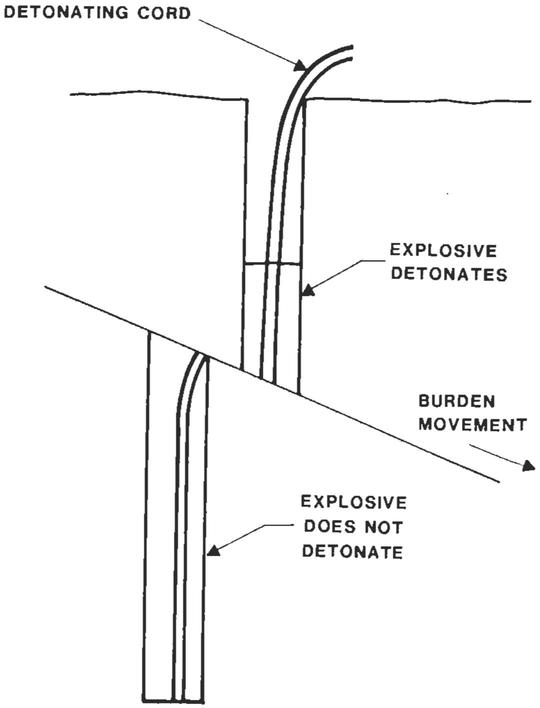


Figure 9.7 Movement of ground along the weak plane can cut the trunkline or downlines.

enough to move the rock and get good fragmentation. Overbreak and boulder formation as a result of blasting in widely spaced jointed rock mass are common features. It is advantageous to use smaller diameter holes with closer burden and spacing so as to have a hole in each block (Fig. 9.7). Coates (1981) suggested that a detailed study of the burden, joint spacing, and maximum allowable size of blocks helps in overcoming the problems encountered in blocky formation.

9.8.5 *Joint filler material*

The joint filling material, is also a factor influencing blasting results. The type of filling material changes the rock mass behaviour due to varying properties of deformability, wave transmission, etc. The cohesion and frictional characteristics of a joint depends upon the properties of filling material, nature of host surfaces, whether smooth or rough, and the fill thickness or joint width.

Bhandari (1975a) observed that if a weakness plane is open and filled with air it has a strong control on the fragmentation as the stress wave utilisation is better. Even cement filled joints gave better fragmentation than weak material filled joints. This was also observed by Singh & Sastry (1986) that in open joints the reflected strain wave from joint face led to better fragmentation in the zone between joint and blasthole.

In jointed rock another problem which is experienced is the cut off of down line or trunk line due to ground movement along the plane of weakness. As shown in Figure 9.7, upward movement along a weak plane, due to holes fired on delay Number 1 can cut the down lines or trunk lines for holes on delay Number 2. This often results in a misfire.

9.9 PRESENCE OF CAVITIES/VUGHS

Cavities (vughs) resulting from dissolution of the primary rock structure by ground water are found in many ore deposits and are up to 150 mm in many sulphide ores and even larger in some limestones and iron ores. Cavities tend to reduce blasting efficiency.

When intersected, cavities can cause drill steels to jam. Cavities can also cause the following charging problems, especially where bulk ANFO or pumped water gel blasting agents are used:

1. Where a standard weight of explosive is charged into each blasthole, large cavities can result in: a) excessive charge concentration within the cavity; and b) a corresponding lack of explosive energy in the upper part of the blast hole.

2. When it is possible to obtain stemming rise within that part of the blasthole immediately above the vugh, a separate column charge should be used. If stemming difficulties prevent this procedure, efforts should be made to increase the energy yields of upper parts of charges in surrounding blastholes. Such measures are most easily carried out with bulk loaded explosive systems.

3. When all blastholes are charged to give a constant stemming length, large cavities may allow very heavy charge weights per blasthole with consequent risks of cut-off, flyrock and/or overbreak. If the charged section of the blasthole lies near a sizeable vugh, blasting effectiveness is reduced as a result of: a) the premature termination of outward-propagating cracks at the wall of the cavities, and b) the more rapid

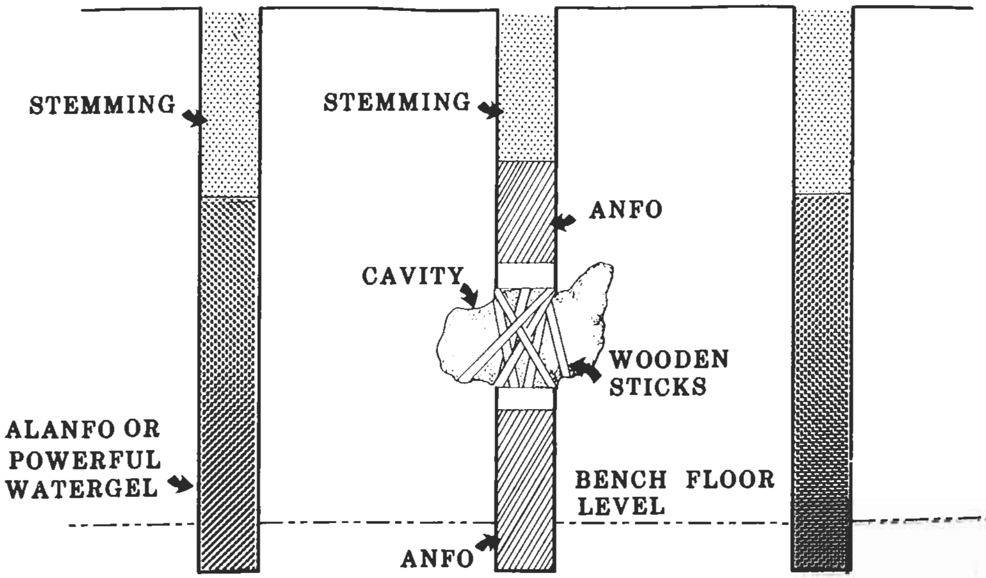


Figure 9.8 Charging a hole containing a cavity (Hagan & Reid 1983).

drop in blasthole pressure as explosion gases jet into the vugh via discontinuities and strain-wave generated cracks.

Once the explosion gases start to stream through a radial crack or a combination of discontinuities into a nearby cavity, they cease to fully pressurise other radial cracks. For this reason, radial cracks in directions other than towards the cavity then tend to stop propagating.

Presence of cavities can also cause propagation between charged holes resulting in undesirable initiation out of sequence with adverse effects on blasting results (Chapter 5).

Presence of voids and cavities are also cause of flyrock if adequate precautions are not taken (Chapter 19).

Figure 9.8 shows a method of charging ANFO in a hole which has indicated presence of cavity.

9.10 FLOATERS

Mixtures of elastic and plastic acting rocks can cause formidable blasting problems. Where 'floaters' (i.e. boulders of a relatively elastic rock embedded in a much softer

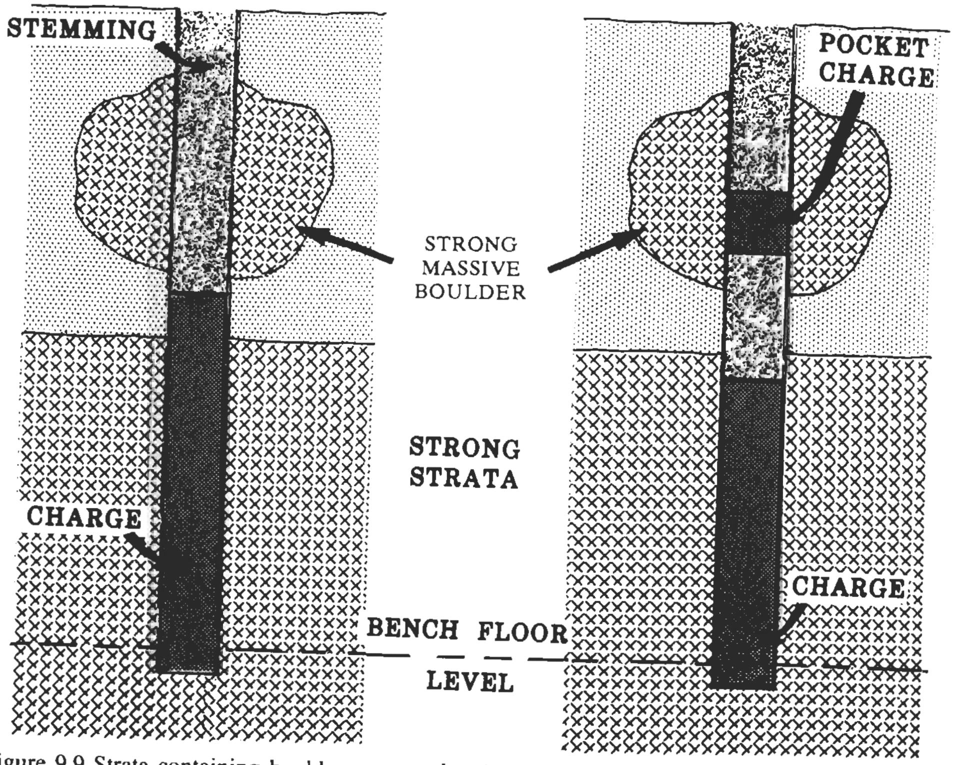


Figure 9.9 Strata containing boulder may not break in (a) which may be broken with the use of pocket charge in (b).

plastic – acting matrix) are encountered, the strain wave propagates with little attenuation in the boulders, but its energy is rapidly dissipated in the matrix. Floaters which do not contain part of the explosive charge receive very little strain energy and, often, are simply pushed out intact into the muckpile (Fig. 9.9a). When floaters contain some of the charge, the degree of breakage can range from inadequate to excessive depending on the size of floater, charge location and matrix characteristics.

If the floater is large, the charge is located only within the outer shell of floaters and the matrix is highly compressible, then breakage will be poor. On the other hand, high degrees of breakage result when a small floater contains a comparatively long charge and the overlying stemming material is relatively efficient,

The combination of inefficient stemming and a highly compressible matrix reduces the contribution of heave energy breakage, since the rapid expansion of the blasthole in the matrix allows an impulsive drop in blasthole pressure. In these conditions, a pocket charge within the floater should be fully coupled (Fig. 9.9b) and preferably, should exhibit a high detonation velocity and high strain energy: heave energy ratio.

9.11 VARIABILITY OF STRATA

Rock properties often vary in different sections of the mine. In surface coal mining operations, overburden strata often consist of adjacent beds which exhibit totally different strengths. Only in a few metal mines and quarries, variations in rock mass properties are as pronounced as those encountered in coal mines.

If maximum blasting efficiency is to be maintained, any significant variation in rock properties will demand a corresponding change in blast design. In principle with change in rock characteristics each hole loading need to be changed but in practice this is possible only if one knows specific change needed in the blasthole loading and other parameters with respect to rock property change. This is very cumbersome and not found to be practicable for the change to be made for each hole. However, there are many steps which can be taken to alter charge loading based on general variation in rock properties.

Where competent strata are overlain by extensively weathered equivalents and/or unconsolidated sediments, it is usually found that explosives need to be charged only into that part of the blasthole below the contact (Fig. 9.10). If the competent strata are well fragmented and displaced by the blast, the weak overlying material will be sufficiently disrupted and bulked to give rapid trouble-free digging.

Where a thick layer of weak or highly deformable material lies between beds of stronger and more massive rocks, lowest overall mining costs are usually achieved by charging only those parts of the blasthole within the stronger beds, such as in example (Fig. 9.11), in which a thick bed of loose sand or thinly bedded shale lies between a strong massive rock. If a single continuous column of explosives were to be charged up to weak horizon, the following sequence of events would take place:

1. Explosion gases would rapidly expand into the (laterally compressed) sand.
2. High-pressure explosion gasses within the stronger beds would stream along the blasthole towards the section of low pressure.
3. The abnormally high rate of decay of gas pressure within the stronger strata

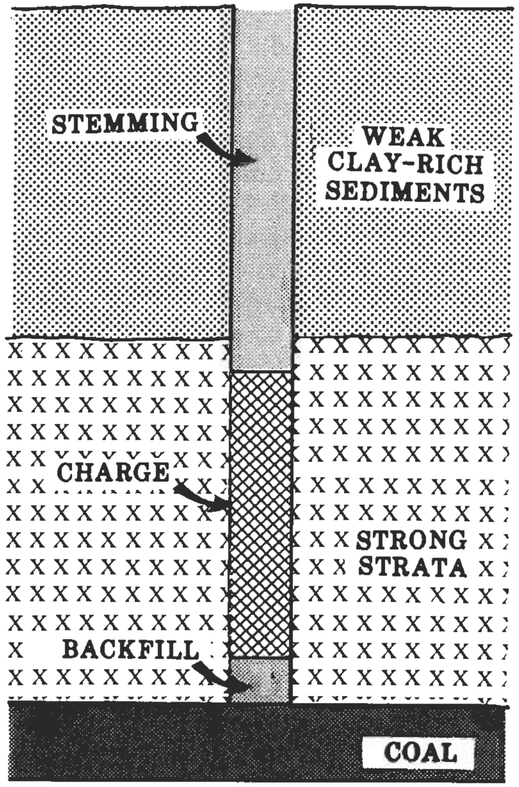


Figure 9.10 Charging of blasthole passing through a weak upper strata.

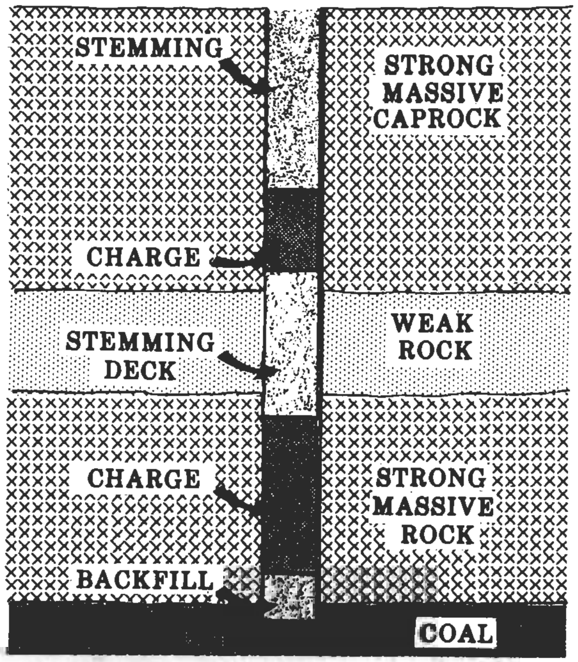


Figure 9.11 Charging of hole passing through variable strata.

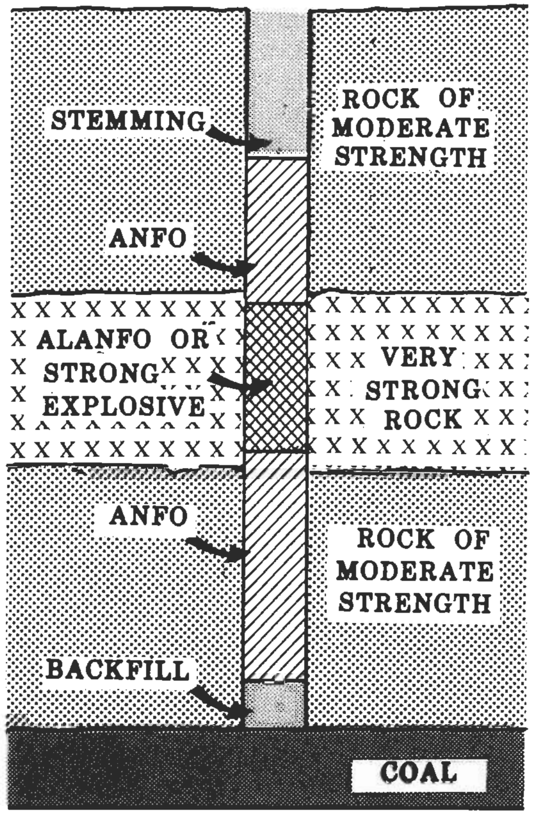


Figure 9.12 Charging of hole which passes through a strong bed.

would result in reduced fragmentation, displacement and muckpile looseness.

The careful placing of drill cuttings within the loose sand prevents this occurrence and, importantly, avoids the wastage of explosion energy.

In the event that a very strong bed lies within the competent strata that necessitate the use of continuous column charges, it may be profitable to locate a charge of aluminised ANFO or high-energy watergel within the stronger bed, whilst, using ANFO in the remainder of the hole (Fig. 9.12).

Considering a blast block (Fig. 9.13a) which contains both ore (two types, one highly friable and the other strong and massive) and waste of uniform and moderate strength it is generally difficult to use different blasting patterns in ore and waste. It is possible to do so if during drilling information is made available about various parameters of the material. Alternative is to standardise the blasthole pattern, the burden distance and blasthole spacing being such that long heavy charges give good blasting results in the strong massive ore. Shorter lighter charge would then be used in the friable ore and waste (Fig. 9.13b).

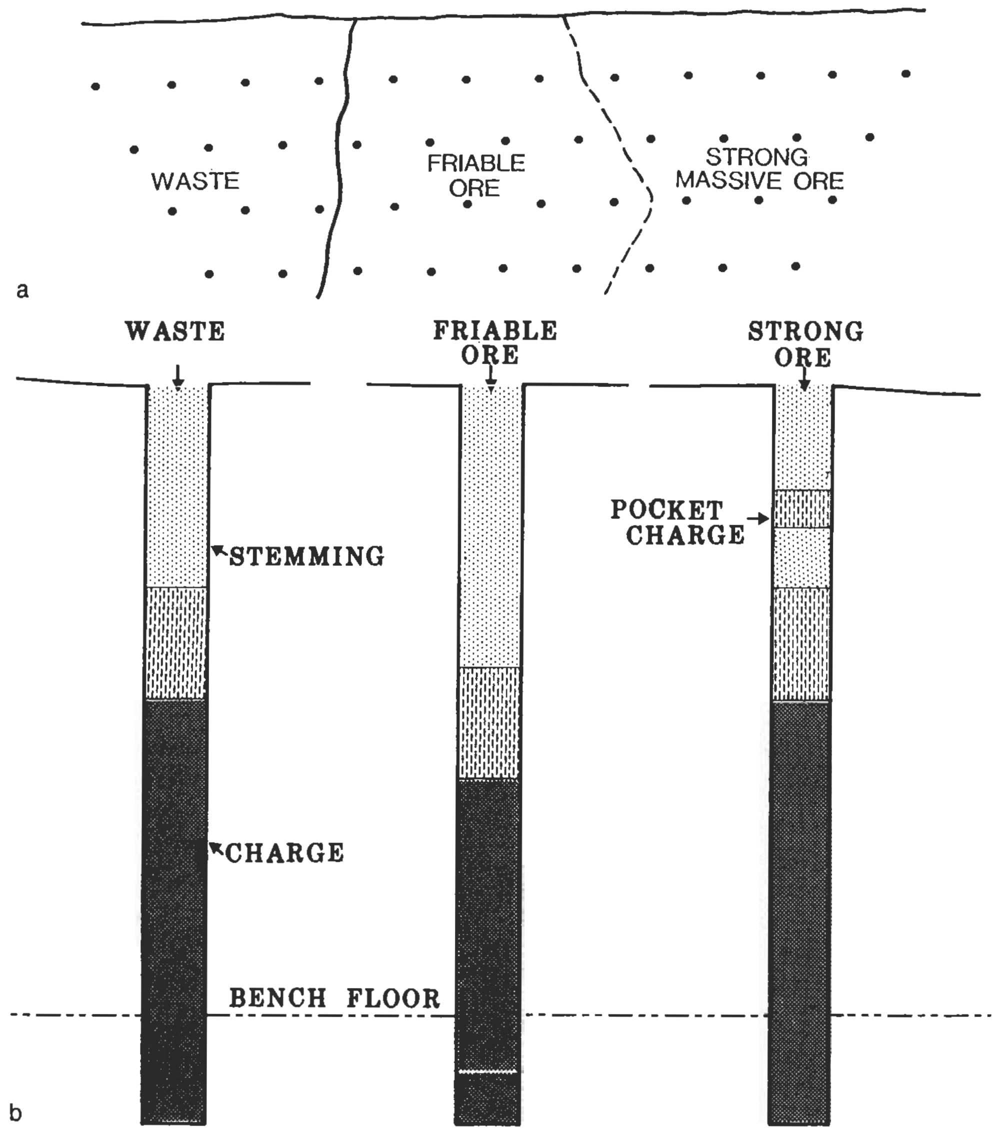


Figure 9.13 a) Blast containing highly variable strata and b) Charging of holes for different rock types.

9.12 GROUND WATER AND MOISTURE CONTENT

Pore water pressures in the rocks reduce the compressive and shear strength considerably. When such rocks become saturated blast effects are intensified (Ash 1968). Due to high degree of saturation less energy absorption and scattering of strain energy leads to better transmission resulting in greater fracturing. The water content of rocks influences both the selection of the explosives and primers (Chapters 4 and 5). Blasting agents like ANFO give poor results in watery conditions, since they are hygroscopic in nature and have low density. Slurries or emulsions (preferably pumped) are more effective in such situations because of full coupling.

When a decoupled charge is surrounded by water, its effective strength upon detonation increases. Field studies have indicated that the presence of water around decoupled charges can increase the level of ground vibrations several times. If blastholes cannot be kept dry, this becomes a consideration while designing a blast to protect buildings or tunnels from ground vibrations.

9.13 CONDUCTIVITY

In ore bodies such as sulphides and magnetites, which are relatively conductive, current leakage and subsequent misfires and poor blasting result is often experienced. In such cases appropriate lead wire and covering of leadwire joints is often recommended. Also it has been noticed that series-parallel circuits often reduce misfire complaints. Non-electric blast initiation is often resorted to overcome such problems.

9.14 IN-SITU STRESS

The high static stresses in deep underground operations have a two fold effect on blasting. The first effect is that of pre-existing cracks crossing the borehole. The second effect is due to stress concentration around the blasthole itself. The fracture pattern is influenced by the non-uniform stress field (not hydrostatic) acting. The difference between two principal stresses in the plane normal to the blasthole axis causes non-radial fracture pattern. The cracks from a blasthole, particularly in massive rocks, start to propagate radially but tend to curve off into the direction of the major principal stress (Fig. 8.23). In this way, in situ stresses play a significant role in the formation and development of crack patterns and thus have influence on the method to be adopted for controlling damage to the remaining rock.

CHAPTER 10

Evaluation of blasting results

10.1 INTRODUCTION

Blasting follows drilling and precedes loading, hauling and crushing. It is important that blasting operations be carried out in an efficient manner as results of blasting influence subsequent operations. The influence of better blasting results can usually be found in:

– Increased shovel production rates, improved bucket fill factors, enhanced bucket loading time, reduced lost time in moving boulders, reduced lost time in cleaning hard toe, etc.

– Reduced shovel maintenance, increased rope and teeth life.

– Reduced truck maintenance, specially suspensions, tyres, etc.

– Reduced secondary breakage costs.

– Reduced delays elsewhere in the mining system such as ore pass problems and crusher blockages due to oversize.

– Improved overall crusher production, reduced energy consumption, etc.

– Higher productivity rates reduce all the unit contributions of fixed overhead.

– Greater safety as better stability is achieved leading to savings in support and reinforcement of rock.

– Improved relations with neighbours as vibrations, airblast and flyrock damages and complaints are reduced.

In view of the above it is imperative to attempt to achieve better blasting results. However, better blasting results may mean different to different operations. It is rather difficult to quantify blasting results in a definite measure and only generalised assessments are made.

10.2 BLASTING RESULTS

The following aspects are looked into for assessing the blasting results.

1. Fragmentation, what percentage oversize;
2. Muckpile profile and displacement;
3. Hard toe and undiggable areas;
4. Backbreak and overbreak;
5. Vibrations and airblast;
6. Flyrock;

7. Damage to wall or slope stability.

In a generalised way all these can be termed good, fair or poor. Desired results will depend on the subsequent usage of blasting material, excavation equipment, crusher size and so on.

To properly evaluate blasting results the operation should be considered as a whole and the evaluation should consider:

1. Total cost breakdown for the operation, what proportion is of drilling and blasting?

2. What influence does the drilling and blasting operations have on the other cost centres?

3. If drilling and blasting costs are increased (or decreased) by a certain amount what benefits can be expected elsewhere in the operation?

4. The blasting should consider influence of rock mass properties and influence of frequency, type and orientation of joints.

5. The type of loading equipment, crusher size and subsequent utilisation of blast material or the remaining excavation space.

6. What environmental constraints have been placed on the blasting operation such as nearby houses, flyrock restrictions etc.?

10.3 BLASTING MEASUREMENTS

The primary measurements include explosive and initiation system performance, fragmentation, blasting damage and condition of the remaining rock. It is useful to keep some form of record such as shown in the Table 10.1 along with photographic record of the blast results. In order to be able to analyse these records it is important to have complete details of the blast itself (Table 10.2) along with blast layouts.

10.3.1 *Explosive performance monitoring*

Performance of explosives need to be measured in the hole so that it is known that the explosive is doing the desired task. Some explosives in particular have been observed to vary considerably and their performance may be reduced to the point where they are no longer cost effective. Causes of this may be related to charging practice, variations in composition or mixing, inadequate priming, or interaction with contaminants such as water in the borehole.

In general, it is difficult to measure explosive performance in the field. Primarily it is measured through velocity of detonation measurement in blastholes. Maximum fragmentation and muckpile displacement requires maximum energy release from the explosive during detonation. Maximum energy release requires the explosive to detonate at its maximum rate, or velocity of detonation (VOD). There are a number of methods for measuring VODs but in general the methods can be split into two categories: point to point and continuous systems.

Point to point systems rely on detecting the detonation front at discreet points using resistive or fibre optic targets or ribbon wire techniques. Such systems are generally suitable for quality control checks but often lack the resolution required to assess accurately explosives performance.

Table 10.1 Blast assessment (overburden).

Mine	Oversize	Flyrock	Horizontal displacement	Vertical displacement	Fragmentation	Diggability characteristics	Noise/vibration
DD	Primarily in sandstone parting	Negligible in highwall	Moderate	3-4 m	1-1.5 m max. except in bench No. 4 which can be quite hard	Muckpile difficult to penetrate with bucket	No problem
CC	Some in interbedded layers	None-control by increased stemming	Minimal	Minimal	Good to excellent	Tight muckpile	

Table 10.2. Blasting details.

Mine	Hole diameter (mm)	Hole depth (m)	Burden (m)	Spacing (m)	Explosive type	Priming		Collar heights (m)	No. of decks	Remarks
						Type	Location			
DD	269	28	12	12	ANFO	25% slurry	0.75 m from bottom of deck	10-12	6	Cartridge slurry used in holes

Continuous systems employ a probe which is consumed by the detonation front which is monitored as a voltage drop or by an electronic pulse system. The resolution on these systems depends on the equipment used to detect the change in length of the probe, with some systems capable of detecting the front 10,000 or more times along the charge.

The basic system utilises a high-resistance probe that is inserted axially into the explosive charge. The probe is fed by constant current at a known initial voltage. As the probe is consumed by detonation head, the voltage drops with time and this event is captured on a digital storage oscilloscope. Using Ohms law, the drop in voltage with time will determine the reduction in probe length with time and hence the VOD.

The advantage of such an accurate yet practical field system is that it can be used not only for VOD measurement as velocity in meters/second, but the data produced give a full time history of the explosive, allowing the following analysis (Vassie & Bonneau 1992).

1. Assess whether detonation or deflagration occurs;
2. Compare results against explosive manufacturers specification to check if product is up to standard;
3. Determine the minimum primer weight of an explosive;
4. Determine the critical diameter for an explosive;
5. Assess gap sensitivity between cartridged explosive or different types of explosives;
6. Determine the explosive efficiency by comparing the field data with theoretical computer code predictions;
7. Investigate the effectiveness of stemming decks within a charged hole;
8. Investigate the effect of water and contamination in the hole;
9. Detect side initiation or degradation of the explosive due to detonation fuse down-lines;
10. Use on test charges or in production holes either on surface or underground;
11. Determine the blasthole detonation pressure profile throughout a hole.

A typical VOD oscilloscope trace is shown in Figure 10.1a. This trace shows the effect of the pentolite primer as a steep section at the start of the voltage-drop slope, followed by the transient velocities building to the steady-state VOD as a steeper slope at the end of the trace. In some cases the trace shows many spikes and a slope flattens out, indicating that a decelerating detonation head and incomplete consumption of the probe; such a trace is shown in Figure 10.1b and illustrates poor explosive performance.

Figure 10.1c shows a trace resulting from a charge failure as a slow beginning which drops to horizontal line, indicating that there has been no change in voltage and, therefore, detonation has stopped and the charge has failed.

The 'continuous velocity' system can be used for tube samples of bulk explosives as a quality check and also to characterise the explosive in terms of the effects of confinement, priming, charge diameter, density and so on. Down-the-hole measurements provide information on performance in actual production holes which may be affected by certain site specific conditions.

Other measures of explosive performance relate to energy of vibration (shock energy) produced by the explosive and using gauges located close to the blasthole along with monitoring of environmental effects of blasting (Chapter 18).

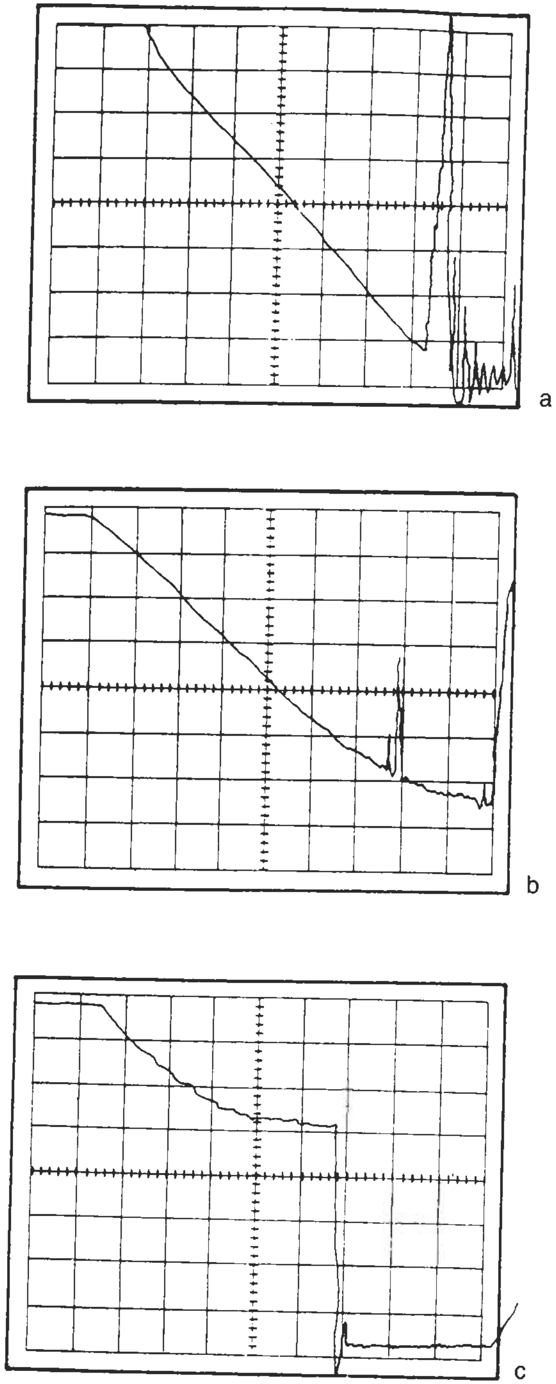


Figure 10.1 Typical velocity of detonation trace results (Vassie & Bonneau 1992). a) Steady-state VOD trace, b) incomplete detonation, c) failed charge.

Figure 10.2 Blast monitor (courtesy Australian Blasting Consultants).

By using computer based monitors (Fig. 10.2) and analysis techniques full vibration wave form describing bulk rock mass response is obtained. Modern monitors permit the operator to monitor either the far field effects of blasting, or the near-field events which cause the far-field disturbances (McKenzie 1993). Both over pressure and ground vibration can be monitored close to the blast to provide the operator with a greatly enhanced understanding of the blast sequence and aspects of blast design which require particular attention and modification.

Vibration monitoring is used to evaluate the effectiveness of delay intervals in separating the individual vibration from blastholes, and to determine the precise cause of high vibrations which may be responsible for structural damage or environmental disturbance. The monitoring may involve particle acceleration to establish influence on stability, or over pressure to assess disturbance levels to residents.

Figure 10.3 shows the vibration response to a normal blast at varying distances. Whereas the discrimination of individual blasthole detonations is possible at close distances, the duration of vibration disturbances from each blasthole increases with distance from the blasthole, making identification impossible at large distances. At sufficient distance from the blast (several hundred meters), the whole nature of the disturbance changes as the effect of the surface wave dominates the body wave. A closer examination of the nearest trace of Figure 10.3 reveals additional useful information, as shown in Figure 10.4. The expanded view shows that two blastholes have initiated almost simultaneously (within 4 ms). This can be caused by excessive explosive usage (interaction between charges in adjacent holes), or by scatter of delays. Thus monitoring helps in careful evaluation of delays, powder factors, blasthole

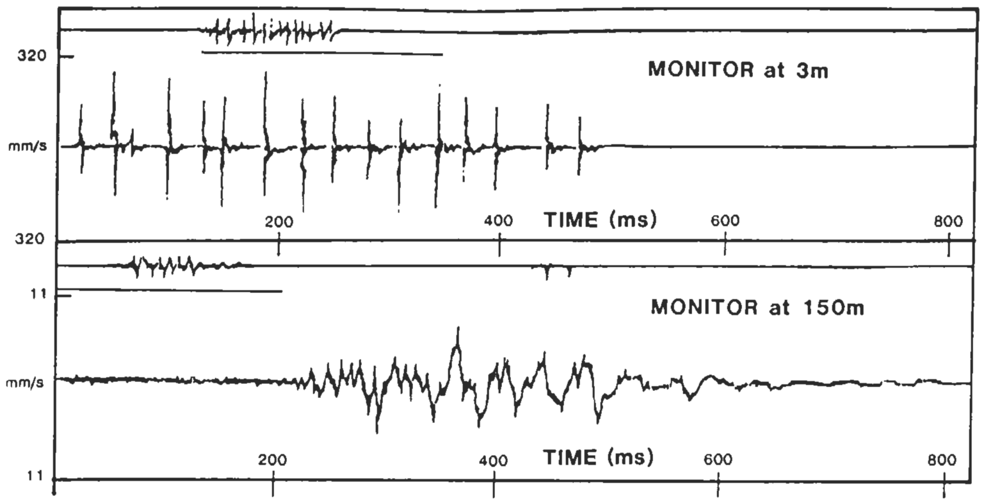


Figure 10.3 Ground vibrations at different distances (McKenzie 1993).

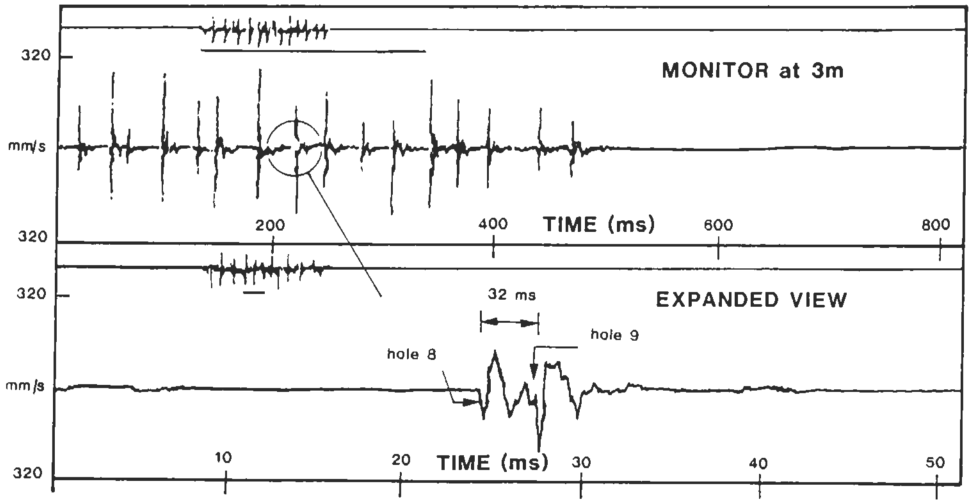
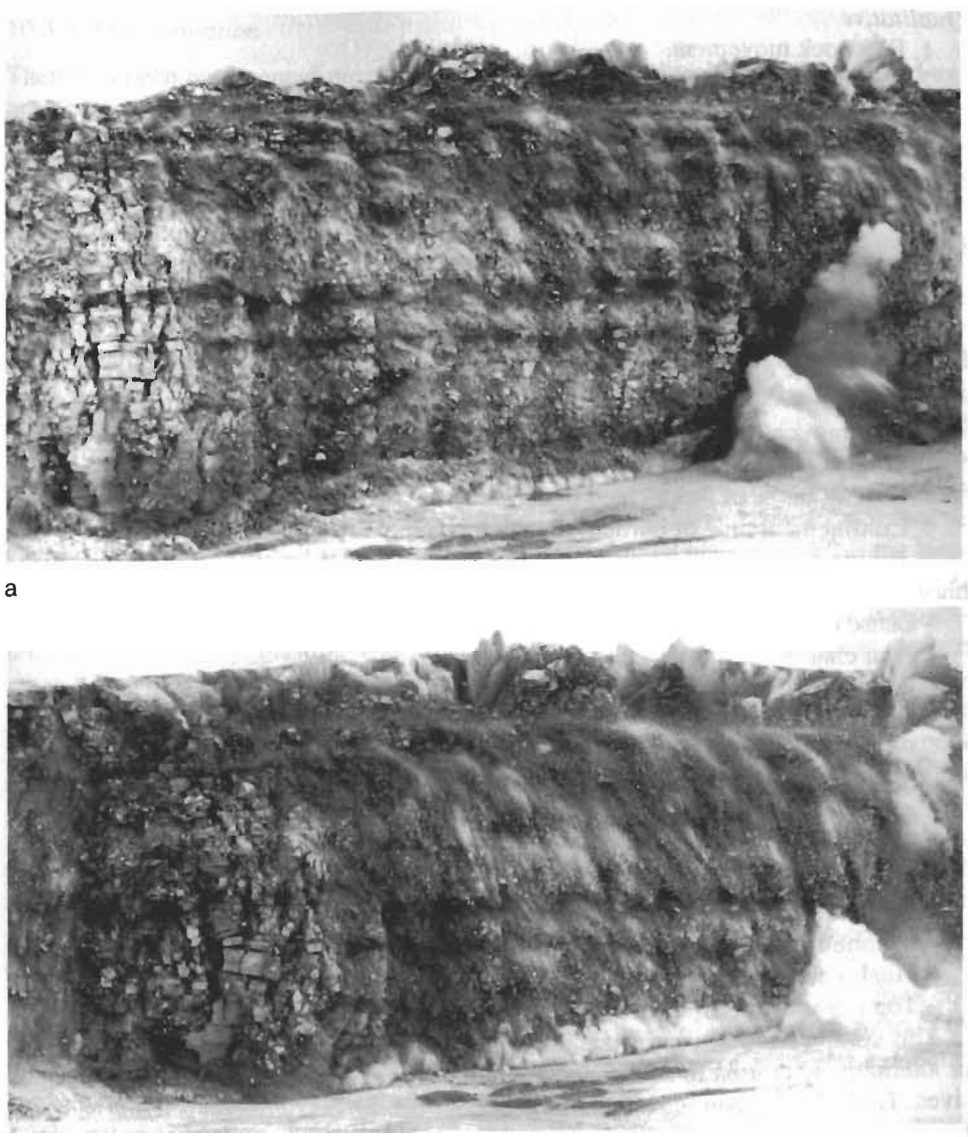


Figure 10.4 Expanded view of near-field blast record (McKenzie 1993).

burdens and spacings. Advanced blast monitoring allows use of additional instrumentation to evaluate performance of the explosives itself in the hole.

10.3.2 *High speed photography*

In recent years, high-speed motion picture photography has become a powerful diagnostic tool and technique to study, analyse, evaluate and aid in blast designs (Chiappetta & Borg 1983). The interaction of energy, time and mass movement following the blast initiation is observed and quantified with the use of high speed cam-



a

b

Figure 10.5 High speed photographs of a blast.

eras (300-500 frames/sec). High speed photography is carried out to determine the magnitude, direction and timing of rock face motion, together with the response of rock surface and blasthole stemming (Fig. 10.5).

Information obtained from high-speed photographic analysis is categorised into two main groups.

Qualitative

1. First rock movement,
2. Firing sequence of holes in blast,
3. Confinement and/or blow-out of stemming,
4. Shape or form of primary movement,
5. Confirming function of explosive accessories.

Quantitative

1. Firing time of initiators,
2. Time and duration of escaping gases,
3. Acceleration, direction and velocity of projectiles,
4. Initial time of detonation,
5. Ground-swelling velocity,
6. Stemming ejection velocity,
7. Total flight time of a projectile,
8. Casting range of fragmented material,
9. Confinement time of burdens, bench top and gases after hole detonation.

A comprehensive collection of this data along with pertinent information on the blast design data and drilling-loading-mucking operation is useful for determining:

- Cause of misfire;
- Poor charge loading practices;
- Poor firing practices;
- Effects of redrills, water, gas venting and buffers;
- Proper delay interval between holes and row of holes to obtain adequate forward burden relief;
- Optimum initiating system;
- Massive ground movement;
- Source of oversize;
- Optimum cast of fragmented material;
- Optimum explosive-rock combination;
- Final rock pile geometry;
- Top movement characteristics (i.e. cut-off potential).

This study describes the efficiency of utilisation of heave energy from explosives, or alternatively is used to compare the degree of heave produced by different explosives. Timing of burden movement largely controls the delay intervals required to effect unrestricted burden motion and minimum backbreak. The timing and velocity of rock movement, and therefore the delay interval, are dictated by the rock and its structure, and will be different for each rock type and its structural zone. High speed photography is also useful in the detection of some initiation functions such as instantaneous initiation of explosive charges by detonating cord downlines, and for evaluating the performance of toe charges by considering the shape of the face during explosive gas expansion. This can give valuable insight into the effectiveness of different sizes or types of primers or initiators in producing high order detonation of the explosive.

High speed photography does have limitations, similar to most photographic techniques. The greatest problem and limitation is when gas venting, smoke and dust obscure crack formation and muck displacement.

10.3.3 *Fragmentation*

There has been no accepted measure of fragmentation. It has been measured and expressed in numerous ways, most important among these are: a) Screen sizing, b) Average size, c) Specific surface, d) Granulometry, e) Fragmentation coefficient, f) Crusher monitoring, g) Boulder count and secondary breakage, h) Muck assessment, i) Productivity studies with work and time studies, j) Photographic analysis, k) High-speed photography, and l) Image analysis.

Screen sizing is a reliable, accurate and unbiased method of evaluating fragmentation quantitatively. A complete analysis requires that the entire muckpile be screened. For small work areas such as in underground headings, this technique may be well within the economic constraints of the operation. In laboratory and reduced scale studies this technique is frequently adopted (Just & Henderson 1971; Bhandari 1975a and b; Bergmann et al. 1975; Bhandari & Badal 1990a). Dick et al. (1973) and Ash (1973) studied fragmentation on a reduced scale quantitatively. Unfortunately in production blasting the screen sizing technique is impractical because of enormous cost and time involved as a result such studies have been very limited (Mohanty & Chung 1990; Bhandari & Tanwar 1993). Sometimes fragmentation refers to the largest pieces resulting from a round. There exists the possibility of measuring the volume of the largest boulder and thereby obtain a relative measurement of the fragmentation. In laboratory blasting fragment size analysis the concept of new surface created has been utilised. For a given weight of particle of one shape the specific surface is inversely proportional to the diameter of fragment. Utilising this knowledge of fragment size distribution the concept of mass surface that is product of the mass and surface area created has been utilised in determining the optimum fragmentation burden (Bhandari 1975a and b).

The size analysis of fragmentation has been utilised to quantify the fragmentation by plotting these as fragmentation gradient. Unlike that in communition the method for plotting size distribution curves was modified by using logarithmic scales on both axis to obtain a straight line relationship between the cumulative percentage passing, and size ratio (Just & Henderson 1971).

The definition of fragmentation is also given in the form of the fragmentation coefficient. The relative fragmentation coefficient is defined as the mesh in relation to bucket opening through which 50% of the material passes. The relative fragmentation coefficient indicates the fragmentation in relation to a certain bucket size.

Granulometry of muckpile is done by the string technique. A decametre is spread along a line at the surface of the muckpile and then measurements of the block lengths cut by the string and of the greatest length visible at the surface are carried out. The first measure is used to calculate the cumulative percentage of blocks situated in granulometric class defined by the second measure. The granulometric data are processed according to Rosin-Rammler curves (Chiappetta et al. 1983).

The coarseness of muck can be gauged as a function of steel plate wear, crushing delays and power consumption because bridging delays at a crusher are mainly attributed to oversize. This technique of evaluating fragmentation is preferred in smaller operations where the origin of the ore is known. In large operations where numerous muckpiles are worked simultaneously, the evaluation of blast design variability and resulting fragmentation is not possible.

Crusher energy consumption, type, strength and size of feed material, size of crushed product and crusher through input is monitored to investigate the effects of fragment size on crusher performance (Mol et al. 1987). This provides only a generalised idea about fragment size.

Any material produced from blasting which cannot be adequately handled with the mining equipment is generally referred to as oversize. Owing to their magnitudes, boulders of this size can easily be distinguished and counted during digging or after they have been set aside for secondary breakage.

The fragmentation qualitatively can also be gauged on the basis of mucking operations. When fragmentation is fine or good, mucking is easy and the ore rills smoothly from the draw point. If fragmentation is coarse, mucking is a little more difficult, some large rocks have to be put aside for secondary breakage and the draw point hangs up occasionally. If fragmentation is rough, mucking is difficult, rocks frequently require secondary breakage and the draw point frequently hangs up. The loading times are related to both fragmentation and expansion of the muckpile. Time to fill the bucket of loader and a dumper are recorded several times.

The technique of assessing the degree of fragmentation by use of productivity work and time studies assumes that digging ratios can be gauged directly to the coarseness of a muckpile. Any oversize, misfires and poor toe conditions encountered in digging are reflected in evaluation. When technique is properly implemented, an accurate assessment can be made.

The fragmentation of a blast can also be measured by means of a set of photographs. The photographs are taken of the blasting muckpile using a scaling object, such as a ball of known diameter, and these are compared with the standard photographs to find one which matches both in texture and size. The known scaling of both prints can then be used to deduce the fragmentation in the blast.

The modern photographic analysis of fragmentation involves taking camera pictures of muckpiles, digitising the image and computer analysis for size distribution (Hunter et al. 1990l; Vogt & Aβbrock 1993).

The measurements, which have to go on over a long time while the loader takes away fragments from the heap, is done by using a TV camera as sensor (Carlsson & Nyberg 1983). The picture from TV camera is digitised and evaluated in a microcomputer and the size distribution from this analysis will be thereafter listed on paper. To determine the scale factor between heap and digitised image, something with known dimensions is laid on the surface of heap. Some known length is then marked by moving a cursor on the TV monitor and then the system calculates the scale factor to be used. One way to measure the size is to estimate the area of each fragment by following their contours. Such an algorithm is very time consuming and is therefore not suitable. Instead of this a typical diameter can be used for each fragment to estimate the sizes. A further improvement has been to measure size distribution of the muck in dumpers and then to rebuild distribution for the whole muck pile distribution (Cheimanoff et al. 1993).

All the photographic procedures are found to be time consuming, fail to quantify fines and are biased towards the large end of the size range.

Conventional photogrametric methods of fragmentation analysis can provide more accuracy, less equipment cost and lead itself to 3-D measurements compared to high-speed photography. High-speed photography is still in its infancy but once developed may prove very valuable for the field use.

10.3.4 *Muckpile displacement*

Displacement of fragmented rock is necessary and desirable but excessive displacement may cause serious trouble. Displacement increases with increased charge. In multirow blasts selection of suitable delay between rows is very important. Proper burden relief has to be provided to each row for effective and efficient horizontal movement. Instantaneous delay in the row gives more displacement than when the holes are delayed in the row. Depending on the machinery employed displacement is required. If the benches are relatively low and power shovel is used for digging the blasted material, the muckpile should not be scattered to ensure a high fill factor. On the other hand, if benches are high and a front end loader is used for digging, scattering is desirable.

10.3.5 *Oversized material and/or fines*

A well designed blast should produce sizes and shapes that can be accommodated by the available loading and hauling equipment and crushing plant, with little or no need for secondary breakage. When fragmentation is fine or good, mucking is easy and ore rills smoothly from draw points. When fragmentation is poor, mucking is little more difficult, rocks frequently require secondary breakage and the draw points frequently hang up (in underground mining) also crusher blockages are reported. Therefore, it is important to locate large boulders or fines, and to from where they arise, so that necessary remedial action can be taken.

10.3.6 *Face conditions*

In civil and mining engineering excavations it is important that the blast does not create backbreak, overbreak or underbreak and remains within specified limits. A survey of wall conditions should be made of slope angle and a visual assessment should be made after each blast. Underbreak, the rock remaining within a specified excavation, interfere with the construction or long term use of the excavation, and must be removed by secondary blasting or breakage. Underbreak can be caused by inaccurate blasthole surveying or drilling, by a charge that is insufficient or poorly distributed within the hole, by misfire or low order detonations or by an inappropriate choice of hole spacing or burden. Backbreak can be caused due to excessive burden on the holes thereby causing the explosive to break and crack radially further behind the last row of holes. Improper delay timing from row to row can cause backbreak when the timing is too short because there is excessive confinement on the last rows in the shot. Long stemming lengths on short benches produce backbreak. Overbreak at the end of the blast site can be produced sometime by the geological structure of the strata which may promote extension of cracks off the end of the shot. The overbreak can also be caused by improper timing on the perimeter holes. If the timing is too short then much larger than the normal burden will be felt thereby resulting in cracking back into the formation. The problem in such situations can be tackled by using longer delay times on the end holes, allowing time for the central portion of the blast to begin to move out, thereby producing additional relief before the end holes fire.

10.4 SOLVING BLASTING PROBLEMS

Evaluation of blasting results will indicate if they are satisfactory and evaluation will help in correlating the design parameters with the actual results. Problem areas could be identified and necessary modifications in the blasting procedure could then be made to overcome the problems.

While designing blasts it is assumed that all holes release energy nearly ideally, that is, all the energy at the desired time and sequence. However several studies indicate that this is not so (Winzer et al. 1983; McKenzie 1993). Blast holes sometime fire with excessive violence, while at other times, they may not fire at all. Some blasts result in excessive vibration while at other times blast noise complaints are minimal. Some blasts produce excessive overbreak and toe. Therefore, it is essential to identify the problems and solve them by necessary modifications.

The problems which can arise as a result of blasting could be grouped as:
1. Malfunctioning of holes;
2. Poor fragmentation and muckpile displacement;
3. Backbreak, overbreak at sides and back shatter;
4. Environmental hazards such as vibration, flyrock and air blast.

10.4.1 *Malfunctioning of holes*

Malfunctioning holes are a common occurrence in blasting operations. Malfunctioning holes cause either no energy release, insufficient energy release or energy is released at improper time. Malfunctioning holes also affect the performance of adjacent holes. Malfunctioning holes are those which do not function as intended and release little useful energy. These could be in several ways.

A *misfire* means that explosive did not fire. *Cut-off* is a general term used to either describe a breakage of the initiator leadwire before it has a chance to energise, the detonating cord has been cut by flying rock, movement of strata or cutting off the explosive column by shifting rock beds. *Missed holes* are those where the initiator was not connected into the firing circuit or those in which the explosive column did not receive detonation signal.

Malfunctioning holes result from many causes. These causes can be broken down into four general categories.

1. *Inefficient explosive energy release.* Explosive conditions; insufficient priming; detonating cord incompatibility; ageing during explosive storage; dead pressing; insufficient confinement.

2. *Initiation errors or incompatibility.* Improper initiation timing; cap scattering; water hammer.

3. *Poor execution of blasting plan.* Propagation; drilling errors; timing errors; incomplete connections; borehole loading errors.

4. *Geological conditions.* Wet holes; lost holes; cut off.

Inefficient explosive energy release

There are many reasons why explosives do not detonate or partially detonate or low order detonation occurs releasing only a fraction of their potential energy. Products of same type can vary from one batch to another. Field mixed explosives such as ANFO, some slurry or emulsion blends and HANFO, can have a highly variable energy release depending on the composition of the mixture. Field mixed explosives are more prone to variations in mixtures because of human error and less stringent quality control procedure.

Selection of improper primer type, size and numbers may adversely affect performance of explosives. Performance monitoring steps can help in overcoming the problems caused.

The behaviour of the explosive is also controlled by the environment in the blasthole and the effect of outside influences from the adjacent blastholes. In blasting patterns using delays, adjacent holes might be affected by compression of rock which in turn can cause precompression or dead pressing of explosive. Dead pressing in some types of explosives can lead to inefficient energy release. Propagation between holes can also occur if there are cracks present or if stresses are too high and this may lead to blastholes firing out of sequence. In burn cut patterns and in holes using insufficient delay period possibility of such situation arising are high.

Cap sensitive explosives such as those containing NG are initiated by detonating cord. Non-cap-sensitive explosives such as ammonium nitrate, emulsions, and water gels can be affected adversely in many different ways by a detonating cord passing through the explosive column. If the detonating cord has sufficient energy and size some explosives may detonate or burn. A burning reaction rather than a detonation releases only a small fraction of the available energy. If the detonating cord is not of the sufficient size to cause a reaction in the explosive, yet it may cause dead pressing. Dead pressing increases the explosive density and the explosive will not detonate. This occurs as the detonating cord is of sufficient energy to crush out the air spaces within the explosive or to break the air filled microspheres placed in some products which also provide air to form hot spots for detonation.

Water conditions in blastholes range from totally dry holes to those which are filled with water up to the collar which may sometime be flowing or may be under pressure. Water conditions influence the choice of explosive. Problems arise when the explosives have insufficient water resistance and on absorbing water do not release energy efficiently. Some explosives produce low-order detonation when the water head is high as a result of precompression. Water flowing through holes loaded with water-proof explosives such as slurry or emulsion explosives may cut through a part of slurry or watergel rendering it useless thus this portion is unable to pass detonation throughout the column. It is advisable to use cartridged explosives. Some explosives do not sink properly and can cause discontinuity in the column. Therefore necessary remedial measures need to be taken. Instrumentation suitable for explosive performance assessment can greatly help in overcoming problems of inefficient energy release.

Initiation errors or incompatibility

Initiators are placed in the blasthole to transfer the detonation signal to the explosive at a precise time. Choosing the proper initiator is critical to blasting performance.

Whenever blastholes are in close proximity to one another, initiator can get damaged by firing of the adjacent holes. This damage may cause some initiators to fire improperly or may cause the initiators to misfire. Some initiators may withstand the effect whereas others may not. Some initiators may be affected by water as well as water head even when they are embedded in the explosive. One needs to check if the malfunctioning of holes is occurring because of one of these reasons. Proper strength detonator and/or detonating cord is necessary for use with the explosive column. If adequate strength is not there than explosive does not initiate properly, resulting either in low order detonation or deflagration. Detonating cord downlines that are too energetic can cause large energy losses in the explosive column. The cord can also cause the explosive to prematurely burn, releasing little energy. When explosives are prematurely initiated by detonating cord downlines, the cord acts as an inefficient primer. The charge may deflagrate or go into a low-order detonation causing a large energy loss.

In multihole delay firing the selection of the precise number of milliseconds of time between initiator periods is important in rock breakage. Initiators in general have large timing errors, both in nominal firing times and in cap scatter (Winzer 1978). In order for the blastholes to properly function, the design of blasting pattern should consider both the nominal firing time and the cap scatter. In absence of this, holes commonly fire out of sequence resulting in poor fragmentation and increased backbreak, vibration, flyrock etc.

The use of high speed photography led to realisation that initiators do not function as designated by the manufacturers. The use of blast monitors has made it possible to detect and correct the problem. The availability of precise electronic detonators and sequential timers has also helped in providing accurate initiation timing.

Poor execution of blasting plan

A blast needs to be carried out according to pre-determined plan wherein all decisions are taken beforehand about the blasting pattern, blasthole loading, initiation sequence and timing to achieve the desired results. Poor drilling which deviates from the desired location of holes can cause problems such as dead pressing, precompression, and propagation between holes and therefore, proper drilling is important so that the holes function as desired. The manner in which holes are loaded has effect on the performance. For example, insufficient stemming between decks within a hole invites propagation from deck to deck. Cartridge hang up or bridging of explosive charge can lead to loss of energy or unexploded explosive within the blasthole.

Some of the most important parameters for execution of efficient blasting are the true burden on the blastholes and the real bench height. More often than not these differ significantly from those planned. Measuring true height and burden has, until recently, been a difficult and tedious operation. One way is to use a long stick and string hanging over the face in order to establish precisely where the rock face lies in relation to his planned position of the front line of blastholes. Even a qualified surveyor, using a standard surveying equipment would find it difficult to obtain an accurate profile of the rock face.

Laser surveying equipment has been available for some years now, but a relatively new development is laser equipment which can receive a beam reflected directly off the rock (as opposed to using specially designed reflectors).

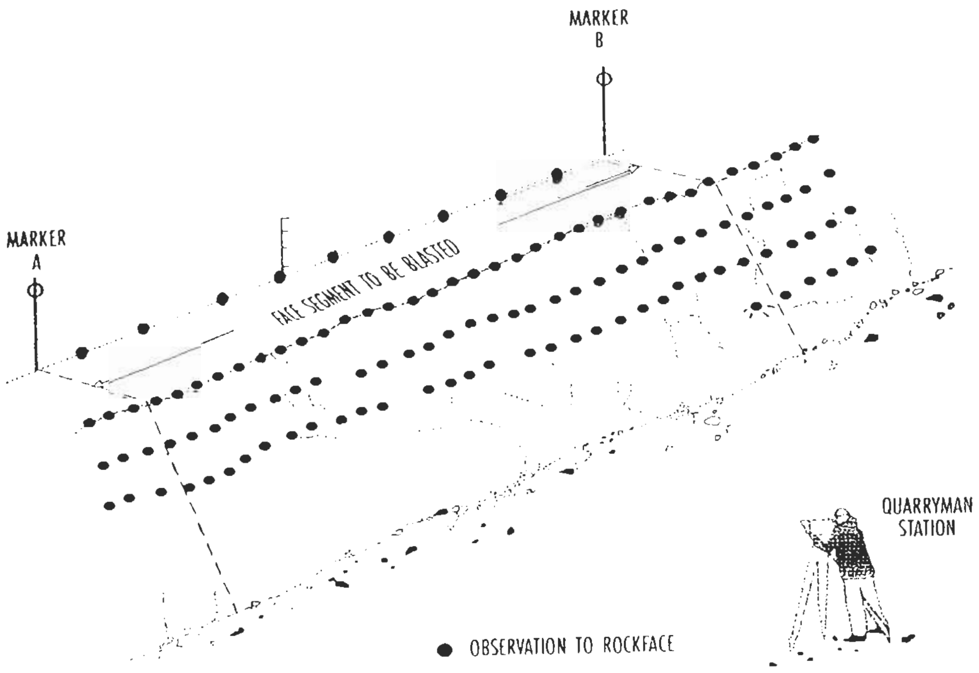


Figure 10.6 Laser surveying equipment used for profiling the face (courtesy Measurement Devices Ltd.).

Figure 10.6 shows how the system is set and the shots taken. The rock face is covered by a grid of points which are stored in the instrument's on board memory. Once down loaded into a computer the software calculates the co-ordinates for each point and a 3-D isometric view is obtained. It is possible to locate collars of the holes already drilled or to be drilled.

Where the blast holes have been already drilled the software gives the true profile for each blasthole. This enables explosives charging details to be planned for each hole to cater for over or under burden (e.g. positioning of decks and adjustments to the stemming lengths.

Profile of the face in this case shows that there is a large burden at the toe of the hole and substantial backbreak at the top of the face. Both of these factors will aggravate the problem of correctly positioning the first line of holes in the next blast.

Many times blasters forget to tie initiator in the circuit which then result in missed hole. Therefore, it is important to check the hook-up visually in case of non-electric initiation and by use of circuit testers in case of electric firing. Sometime initiators are tied in improper manner resulting in some holes firing out of sequence. Many initiator systems today require that the initiators be hooked together in a designated order to provide the proper detonation path from hole to hole.

Geological conditions

Blasting engineer has very little control on the geological conditions of the rock to be blasted. It is required that blast be tailored according to the geological conditions and desired results. In general, it is very difficult to properly assess the geological conditions and predict the blasting results. There are times when geological conditions have detrimental effect on the blasting results and there are other times when the apparently same conditions have little if any influence. The jointing systems, dip and strike of bedding planes, mud or soft seams can have influence on the blasting and safety.

Mud or soft seams can cause considerable problems. Mud seams result in loss of confinement and can change the detonation characteristics of the explosives. Mud can be thrown far distances with flyrock. These seams have the additional effect of allowing the gas energy to be distributed in a totally different manner than intended around the borehole. This can lead to large boulders and little breakage. Seams can also be responsible for the 'cut-offs' or shifting of one portion of the powder column prematurely. In such situations decking or stemming across the mud seam is essential to obtain good blasting results.

Water in the blasthole changes the energy release for some explosives and can cause non ideal detonation.

Blasting parallel with the strike can produce results which are difficult to predict since many different rock layers can be intersected by a single pattern. Since burden and spacing are uniform in a given pattern, layers respond differently. Fragmentation will be different in each rock layer. Floor conditions and backbreak will differ in various sections of the blast.

10.4.2 *Improving fragmentation and displacement*

To control fragmentation the proper amount of energy must be applied at appropriate sites taking into account rock mass characteristics. The energy release must be at precise time to allow the proper interaction to occur. Sufficient explosive energy must be available in a geometric configuration, whereby the energy is maximised for fragmentation. The energy release at wrong time can change the end result, even though the proper amount of explosive energy is suitably placed in the rock mass in the proper pattern. A proper blast design considering rock mass characteristics, timing and initiation sequence is desirable.

General principles for considering muckpile displacement are as below:

1. Instantaneous initiation along a row causes more displacement than delayed initiation.
2. Shots delayed row by row scatter the rock more than shots arranged in V-cut.
3. Shots designed in a V-cut produce maximum piling close to the face.
4. Rock movement will be parallel to the effective burden.

10.4.3 *Reducing damage to remaining rock*

Blasthole pressure, and consequently wall damage and backbreak, can be reduced either by reducing the density of the explosives (for example, by adding an inert filler to ANFO), by reducing the charge diameter in the blasthole (decoupling), or by decking (Chapter 20). If adequate delays and initiation sequence and blast direction are maintained then wall stability improves (Chapter 11). In general, it can be said that the better the fragmentation and displacement is obtained, the better the wall

control. If insufficient energy is available to break rock properly in the burden, the added burden resistance placed against the borehole causes increased confinement and will cause more fracturing (back scatter) behind the blast. If large boulders are produced from the stemming area rather than from the burden, increased backbreak especially at the top of the bench will result, thereby causing problems with subsequent drilling of patterns and the final wall will be less stable. Those orientations of rock discontinuities which produce more stability of the remaining rock should be taken advantage of while planning the blast (Chapter 9).

10.4.4 *Control of environmental hazards*

Reduction of vibrations, airblast damage and flyrock accident hazards are required and can be achieved by understanding the nature of these effects and taking remedial steps (Chapters 18 and 19). Even in absence of damage, complaints and legal action resulting from annoying levels of noise and vibration can close an operation down. All blasts should be precalculated for acceptable vibration and noise levels and monitored. Vibration levels can be reduced by limiting the charge weight per delay to an amount sufficient to achieve the degree of fragmentation. However, drilling costs tend to increase because more holes are needed. Vibration levels are slightly reduced if there is no stemming, but problems of flyrock and airblast increase. Another effective way is to use delayed detonation and sequential blasting. Ground vibration levels are related more to the weight of explosive in any one delay-period than to the total explosive in the blast. In general because of timing errors of most initiation systems, some of the charges not separated by at least a nominal delay of 8 ms may act together in causing vibration. If flyrock is produced from the face and flying to far distances, it could be indication that too little burden is used or that soft rock or clay band is present hence necessary remedial action like decking at that band and control of burden is practised. When vertical cratering is the cause of flyrock then either column height being too high in the hole or too small stemming length is used. Use of proper covering can overcome problems of flyrock to a great extent.

CHAPTER 11

Initiation system, sequence and delay timing

11.1 INTRODUCTION

Blasting operations are often carried out with a number of holes in a single row or multiple rows in which large quantity of explosives is distributed. An important blast design requirement is that a free face should exist in front of the holes when they are fired. The rock is broken so as to provide free faces to which subsequent holes can break. This is achieved by using time delay between initiation of different holes. This allows blast to be divided into smaller charges which are detonated in a predesigned sequence at predetermined time interval. The delay sequence and timing interval at which holes detonate are important factors influencing blasting results. The delay pattern used in a blast determine the detonation sequence of holes or deck initiation, and thus controls the overall direction of movement of the blasted rock and resulting fragmentation. Delay interval controls the time of explosive energy release, which in turn affects the fracture and fragmentation process, the maximum vibration levels, potential for vibration damage, and movement of broken rock. Depending on other blast parameters, the actual timing between detonating charges determine muckpile displacement and height. In many cases, a change in the initiation system – sequence and interval can lead to significant increase in blasting efficiency.

In this chapter initiating systems used during different blasting conditions, various initiation sequences and delay timing are considered.

11.2 INITIATION OF A BLAST

Various initiating devices are used for blasting operations such as ordinary detonators and safety fuse, electric delay detonators, detonating cord and delay connectors, non-electric delay system, sequential blasting machine and electric detonators (Chapter 6). New devices such as electronic detonators have been introduced.

An initiation system is a combination of explosive devices and component accessories specifically designed to convey a signal and initiate an explosive charge from a safe distance when properly configured and activated. The signal function may be either electric or non-electric. Electric initiation systems utilise an electric power source with associated circuit wiring to convey electrical energy to the detonators. Non-electric initiation systems utilise various types of chemical reactions ranging from deflagration to detonation as means of conveying the impulse to non-electric

detonators, or as in the case of detonating cord it is the initiator.

For most surface operations millisecond (ms) delay blasting using vertical holes drilled to a predetermined pattern either in grid or staggered formation is preferred, though single rows of holes fired either simultaneously or with ms delays can be used. Detonating relays or ms delay detonators are used to provide the delay sequence. When blasting in surface mines, the following initiation systems are used:

– All electric detonators (to NG based cartridges or cap sensitive slurry or emulsion);

– Electric detonators to detonating cord (to NG based cartridges, to cap sensitive slurry/emulsion or to primers in the hole for blasting agents);

– Above systems + sequential timers;

– Detonating cord trunk lines, downlines plus non-electric surface delays. Single detonator initiation. This can be electric or capped safety fuse;

– Detonating cord trunk lines, LEDC downlines, down hole delays. Surface non-electric delay systems may or may not be used;

– Non-electric systems such as Nonel, Anodet.

Electric detonators are fragile for use in deep blastholes. They suffer from accidental initiation due to static or induced current. Detonating cord is much safer with non-electric delays. Cut off problems, desensitisation of explosive charge and difficulty in down the hole initiation are experienced when detonating cord is used as downline. Down the hole non-electric delays are used where cut-offs are a problem. Non-electric systems have become increasingly popular. Electronic delay detonators have been introduced.

In *underground mines and tunnelling*, short small diameter holes are often used. The following initiation systems are used depending on the explosive:

NG based explosives

- All electric detonators, short delay as well as long delay,
- Capped safety fuse and igniter cord,
- Non-electric system like Nonel, Raydet.

Pneumatically charged ANFO

- Electric delay detonator + cast boosters or cap sensitive slurry or NG based cartridge,
- Non-electric systems such as Nonel, Anodet system.

Slurry or emulsion:

- In case of cartridges similar to as that for NG based explosives,
- In case of pumpable products electric detonators + primer either a castbooster or a capsensitive product,
- Nonel + castbooster.

Where *large diameter holes are used in underground* they may be up holes or vertical down holes. The following initiation systems are used.

– Detonating cord down the hole with cap-sensitive explosive distributed intermittently along the column. On the surface, down lines are connected by a detonating cord trunk line. Successive delay detonators are connected to trunk lines and fired;

– Electric detonators are connected to downlines in predesigned pattern. The detonators are then connected in a circuit and fired.

Free poured ANFO

– If deck charges, Anodet to the decks;
– With many decks, need the delays in the primers and a single down line. Non-electric delays between holes;
– Detonating cord and castboosters. Non-electric delays between holes.

Slurry/emulsion

– Detonating cord and cast boosters. Non-electric surface delays. Initiate with electric detonators.
– Anodet + primer. Non-electric surface delays. Initiate with electric detonator;
– Nonel + primer. Non-electric surface delays;
– Nonel of different periods + castboosters.

11.3 DELAY TYPES

Blasts are generally delayed using combinations of three delay types – inter-hole surface delay elements, inter-row surface delay elements and in-hole delay elements. The functions of three types of delays are as below:

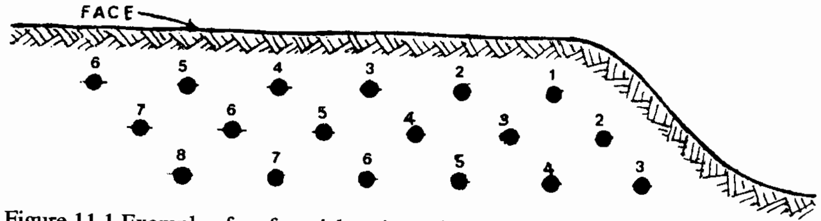


Figure 11.1 Example of surface delays in surface blasting.

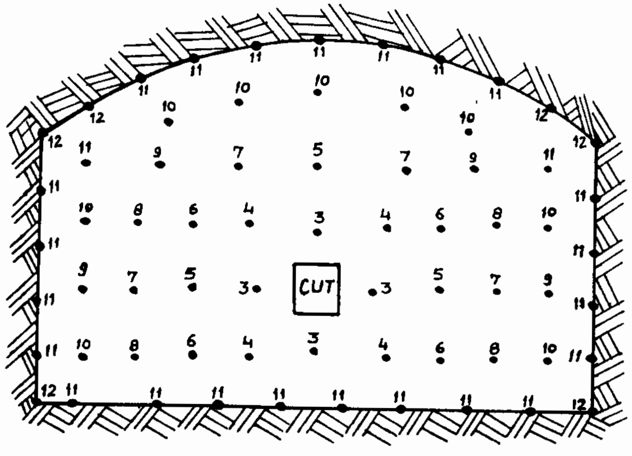


Figure 11.2 Example of use of delays in case of underground blasting.

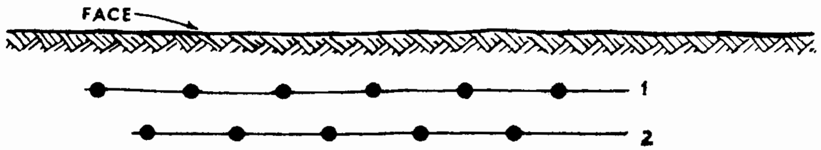


Figure 11.3 Inter row delay in surface mine blasting.

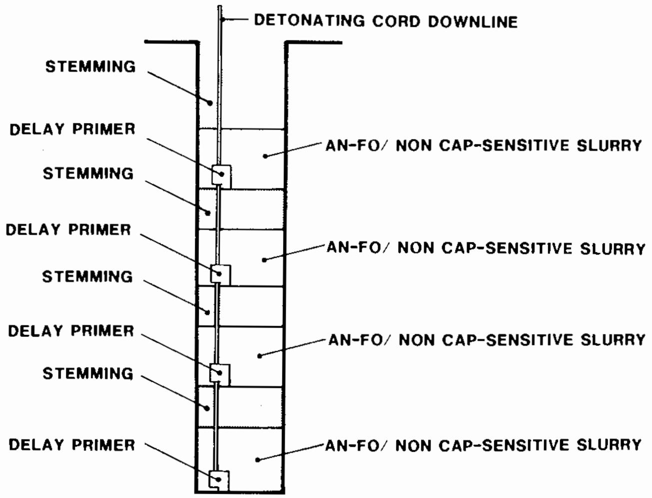


Figure 11.4 Use of in hole delays.

1. *Inter-hole surface delays* provide the timing to create a second free face for the next blasthole to break towards, as well as acting to reduce the degree of vibration enhancement between successive blastholes (Figs 11.1 and 11.2).

2. *Inter-row surface delays* act to provide the major free face for the entire row to break towards, enabling the previous burden to move away from the blasthole behind. Inter-row delays are generally much greater than inter-hole delays, and should be carefully matched with inter-hole delays (Fig. 11.3).

3. *In-hole delays* act to minimise the probability of cut-off by ensuring that all the blastholes have been primed before any blasthole detonates. Where it is not possible to prime all blastholes before any detonation, the down hole delays at least ensure that the initiation front is well ahead of the detonation front (Fig. 11.4).

11.4 INITIATION SEQUENCE

One of the fundamental requirements of blasting is that a free face should exist in front of the blasthole when they are fired. When excavation has to be made into solid rock from the surface, then the rock is broken so as to provide free faces at which subsequent breakage can take place.

Initiation should commence at that point in a blast which gives best possible progressive relief for the maximum number of blastholes. If there is no free end, initiation should commence near, but not at, one end of the blast block (Fig. 11.5a). This

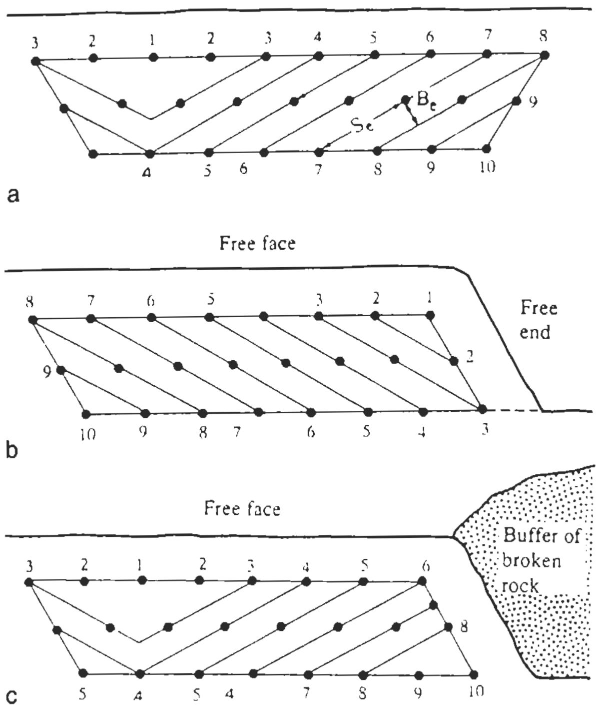


Figure 11.5 Blasting to a) a free face b) a free end and c) a buffered end.

produces less overbreak and lower ground vibrations. If there is a free end initiation should commence at the end (Fig. 11.5b). If this end is choked by a previous blast, initiation should commence near the end of the blast which is remote from the buffer (Fig. 11.5c).

Staggered blasthole patterns are more effective than square or rectangular patterns. Small scale tests (Bhandari 1975a) show that with staggered holes toe between holes is removed even when the spacing is excessive (Fig. 11.6). Hagan (1986) maintained that in hard rocks blastholes drilled on equilateral triangular patterns produce the best fragmentation. However, when blasting weaker rocks, good fragmentation is achieved by using square or rectangular patterns.

A row of holes with the same interval number has better breakage possibilities since the drill holes co-ordinate with each other and enhance the mutual effect. However, increased back break is experienced in simultaneously detonated holes in a row. Fragmentation, on the other hand can be poorer since there is little delay between the adjacent holes.

Some typical arrangements are shown in Figures 11.7 to 11.10. Figure 11.7 shows the arrangement for the simultaneous firing of a single row of holes with detonating cord. This pattern gives spread out muck profile but increased overbreak and vibrations.

Figure 11.8a shows a single row of holes initiated with detonating cord and detonating relays to fire in succession and Figure 11.8b shows how the same sequence can be achieved with short-delay detonators. This arrangement will give minimum vibrations since charge per delay is limited to charge per hole.

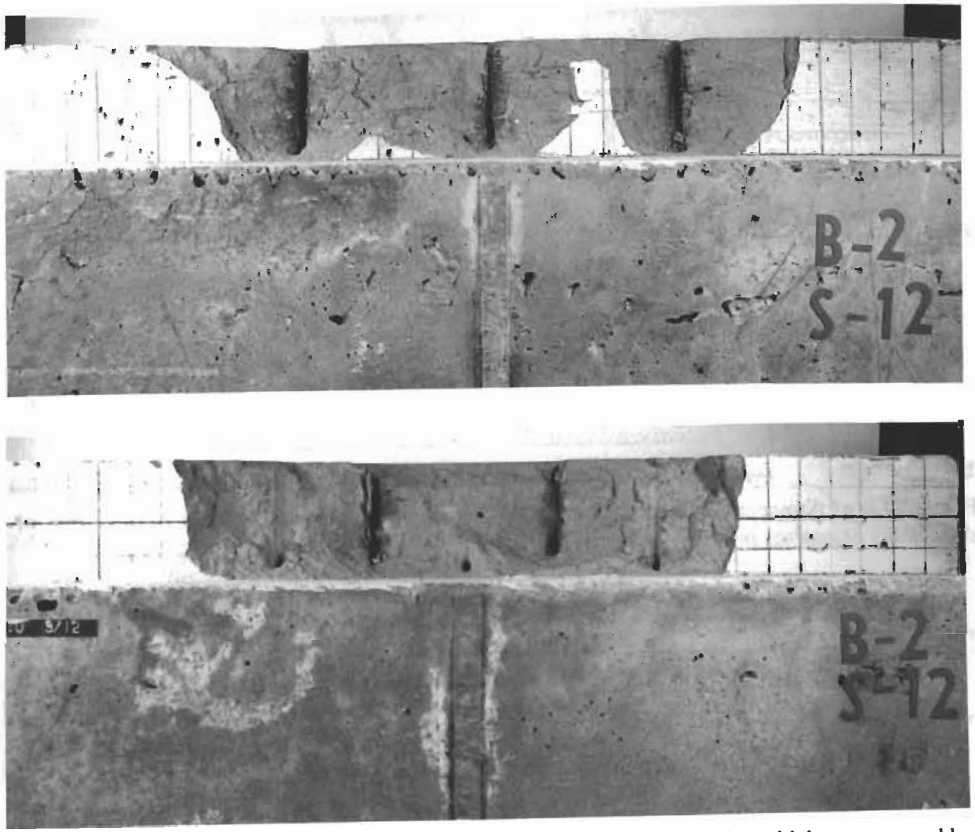


Figure 11.6 Small scale tests show presence of unbroken rock between holes which are removed by using staggered holes in the next row.

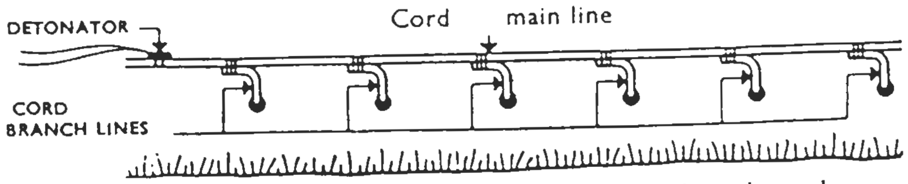


Figure 11.7 Connecting on row of holes for simultaneous firing with detonating cord.

Figure 11.9 illustrates a delay sequence for a single row of holes where there is no open end. This arrangement allows the number of shots fired in the range of short delay detonators.

Multihole blasting is illustrated in Figures 11.10a and b. Only two rows are shown but these blasts can be expanded either up to the range of electric delay detonators or to any range with detonating relay connectors. In Figures 11.10c and d patterns are shown with electric delay detonators and delay connectors.

Depending on the initiation sequence, an effective burden B_e and effective spacing S_e result as shown in Figure 11.11. The effective spacing is the distance in the direc-

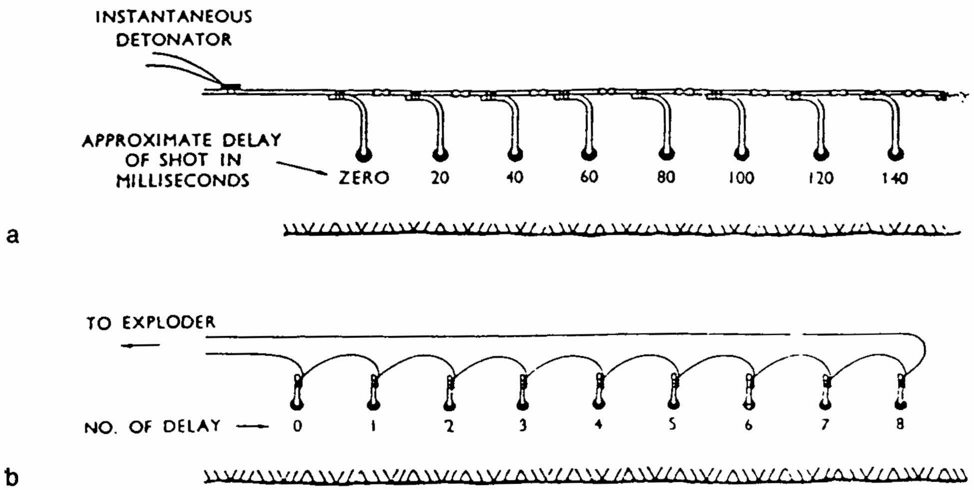


Figure 11.8 a) Arrangement of detonating relays in single-row blasting and b) Arrangement of short-delay detonators in single-row blasting.

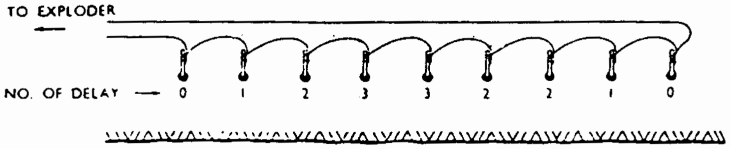


Figure 11.9 Use of delay detonators for single row.

tion of resultant rock mass movement. The *V* and echelon (diagonal) patterns are used when the rock displacement is restricted. Designs using two free faces usually provide improved fragmentation and throw control over those using a single face.

If the blast is to be shot row on row then alternating rows should be staggered to minimise overlap and premature arresting of the radial fractures propagated by each blasthole. This drill pattern can be tied in using a V_1, V_2 or V_3 configuration where angle to the crest is decreased accordingly. The V_1 tie-in is at 64° to the crest and the *S:B* ratio is approximately 1:1.3.

The V_2 tie-in is at 34° to the crest and the *S:B* ratio is 1:3. This tie gives improved results. The V_3 has very shallow angle to the crest and very large *S:B* ratio. It is felt that this tie-in would give excessive displacement of the muckpile, similar to the row on row, and the possibility of hard bottom pockets is considerable based on the large *S:B* ratio. In addition, an irregularly shaped face is expected after the blast unless fill in holes are drilled along the perimeter. Hagan (1986) expresses the opinion that the best results would be obtained with V_2 tie-in for the staggered drill pattern. As in the case of square pattern, a diagonal pattern shot from one end of the blast would increase the amount of free face and most probably improve the fragmentation process (Fig. 11.12). In addition, the V_2 and V_3 tie-in eliminates the tight *V* at the point of initiation.

In general, and in the absence of any unique geological structure, the V_1 and V_2

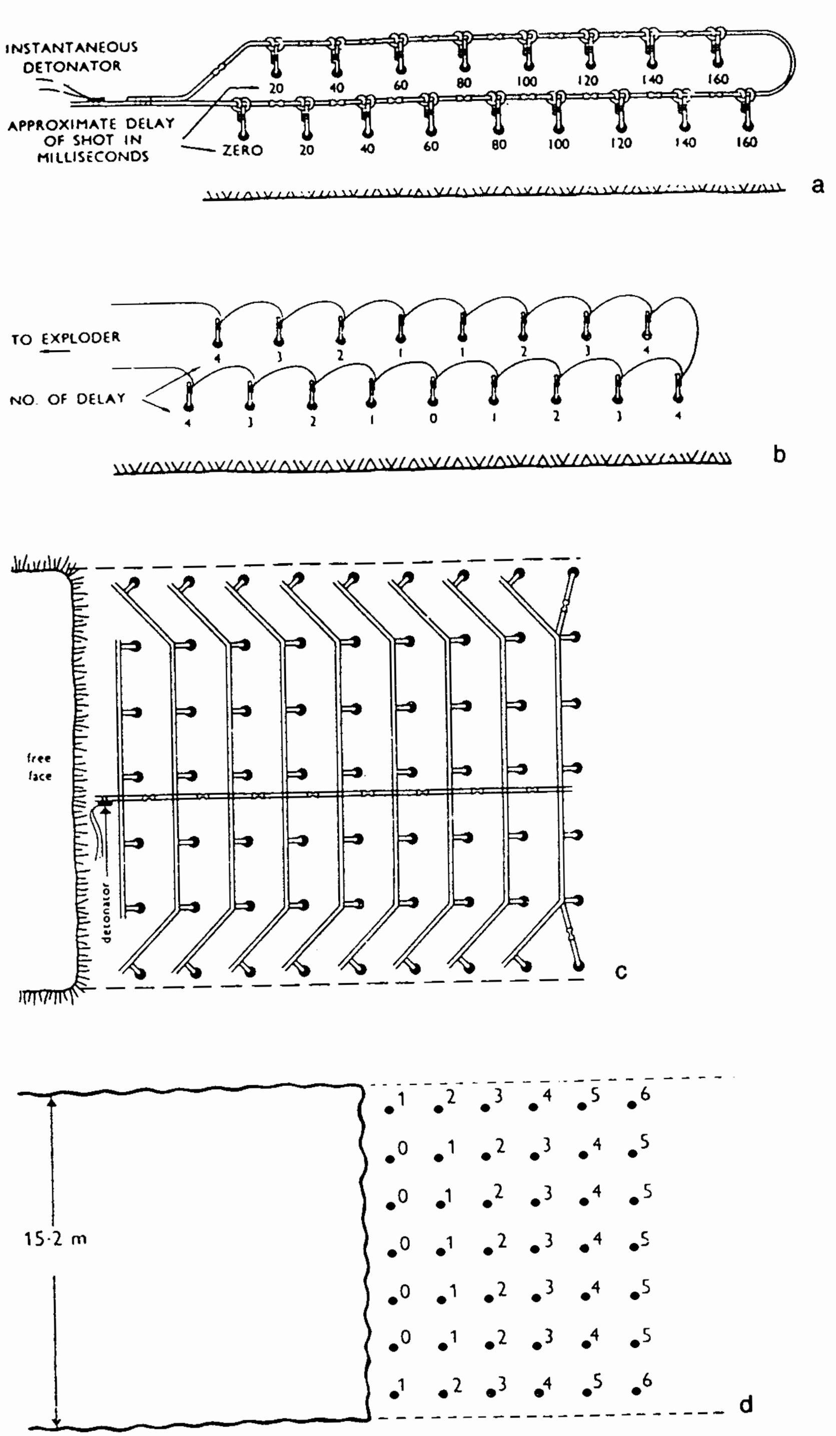


Figure 11.10 a) Use of detonating relays in multirow blasting, b) Use of ms-delay detonators in multirow blasting, c) Arrangement of detonating cord and relay connectors for wide cuttings and d) Arrangement of ms-delay detonators multirow firing for a trench.

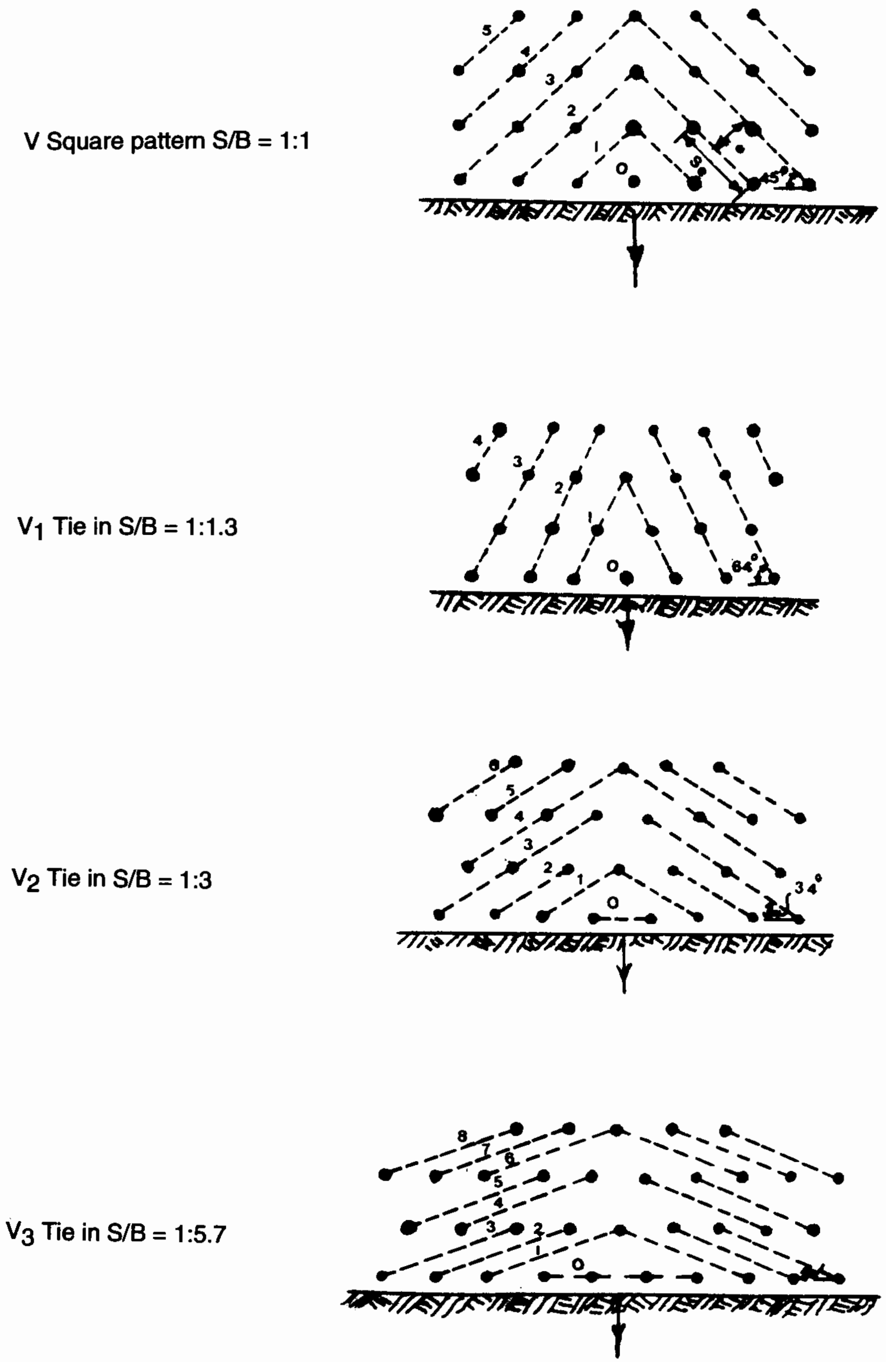


Figure 11.11 Multihole/multirow blasting with various delay patterns.

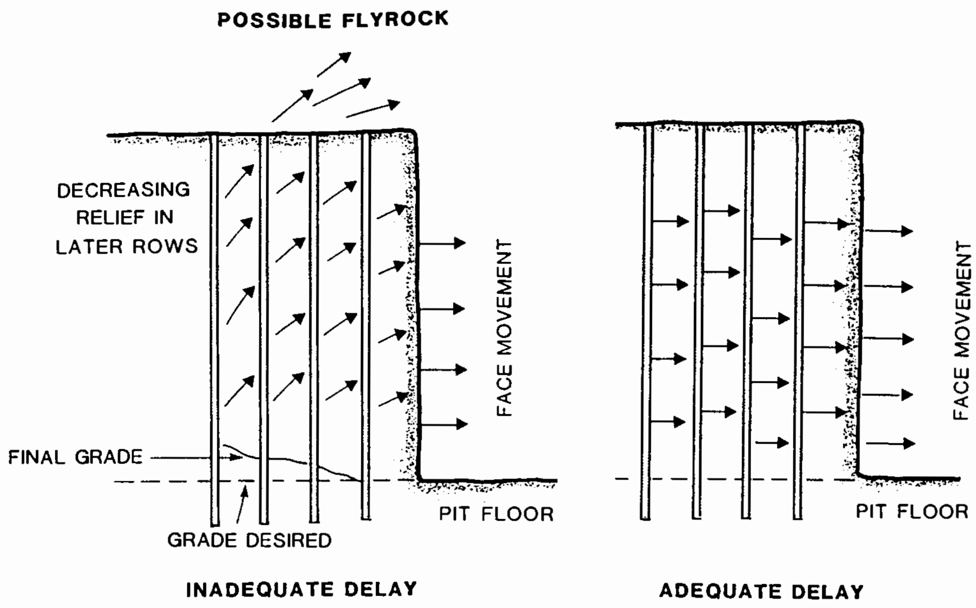


Figure 11.12 Effects of insufficient delay time and that of adequate delay time.

tie-ins with the square and staggered drill patterns respectively give best results in terms of *B:S* geometry and horizontal displacement (e.g. excessive horizontal displacement is not desirable from the productivity point of view). In both cases, the amount of free face for blasting is improved by employing diagonal tie in starting from the free face created by the previous blast. It should be noted, however, that the staggered drill pattern is more difficult to layout and set up with the drill. In addition, where staggered rectangular patterns are employed there is loss of face-hole frequency which is important with regards to moving the toe burden created by the previous blast.

11.5 DELAY TIMING

In multihole/multirow blasting it is important to have adequate delay time between successive holes and/or rows. The delay is provided such that each hole breaks the burden rock in front of it and the broken rock moves out before the hole behind it detonates. If the inter-row delay is too short, the movement of row burdens is restricted which causes excessive burden in the second and subsequent rows and the rock tends to move vertically because of insufficient relief (Fig. 11.12). This causes poor fragmentation, tight muckpiles and high ground vibrations and flyrock, and backbreak along the new face may persist, jeopardising the stability of the slope or opening. If inter-hole delays are too long, cut-offs of surface delays may occur.

The minimum time for design is controlled by the stress wave travel distance in order for radial cracking to begin to develop, contributing to the detachment of the rock mass in the vicinity of the hole. This detachment forms an internal free face (or relief) to which successive detonations will interact with the reflection of stress waves. The minimum timing is, therefore,

$$t = 2\,B_e/C$$

where t is stress wave travel time in ms, B_e is effective burden or distance from the hole to free face in meters, and C is velocity of sound for the rock in m/s. The maximum timing is that at which the burden is fully detached and accelerating as gas pressures build. Research by Barker & Fourney (1978), Barker et al. (1978), Winzer & Ritter (1980), and others, has shown that stress wave travel time is fraction of the time required to develop radial cracks. Furthermore, studies using high-speed photography indicate that the burden moves within a time frame which is between 2 to 10 times the wave travel time to the face. Hagan (1977) noted the time to burden movement ranges from 5 to 50 ms and suggests an optimum range of timing for design between 4.5 to 7.5 ms/m of B_e. Bauer et al. (1979) observed high speed photographic movement of burden rock and suggested that it takes 1 ms/ft of burden before the burden starts to move. It was suggested that to ensure adequate movement of a delay time of 1 to 2 ms/ft of burden would be appropriate in most instances.

Of particular importance when selecting surface delay combinations is the average delay interval between successive hole detonations. It is frequently observed that inter-row delay intervals are such that the second row of blastholes commenced detonation before the first row detonation has completed. This invariably leads to the situation where the delay interval between successive delays is reduced to a period less than the nominal inter-hole delay period. It is common for average inter-row delay intervals to be in the range 5 to 10 ms and inter-hole delay intervals to be in the range of 17 to 50 ms. Where the sum of inter-hole delay exceeds the inter-row delay, then there is no obvious pause in the detonation of blastholes, and far field or remote surface vibrations will most likely increase, together with human perception.

The selection of the down-hole period should also be carefully matched to the surface delay intervals and the size of the blast. Large downhole delays provide a high safety factor against cut-offs, but may offset the value of the surface delays by virtue of the delay scatter. Delay scatter, as a percentage of the nominal firing time has been found to be present in most commercial detonators (Winzer 1978). A delay period for a 10% scatter for a 25 ms delay reduces to an error of ±2.5 ms, and to an error of ±25 ms for a 250 ms delay. Large down-hole delays (>500 ms) therefore, produce scatter which can easily exceed the surface delay intervals, promoting sequence reversals and simultaneous hole initiation.

In general, scatter of the down-hole delay element offsets the advantage of the inter-hole delay and, if sufficiently large, will offset the advantage of the inter-row delay. Computer programmes are used to optimise the selection of delay periods for different blasting requirements. Delay scatter can easily be measured using the blast monitor (Chapter 10).

Studies have been performed to investigate effect of delay timing on resulting fragmentation and muckpile shapes. Reduced-scale research using a variation in delay ratios suggests improved fragmentation for timing between 11 to 17 ms/ft of B_e (Stagg & Nutting 1987), while Bergmann et al. (1974) demonstrated improved fragmentation for *S:B* ratios of two at timing ratios of 3.3 ms/m to 6.6 ms/m of B_e or greater.

Delay intervals recommended by different researchers.

1. Langefors & Kihlstrom (1973)	2 to 5 ms/m of burden (from experience in production blasting).
2. Bergmann et al. (1974)	1 to 2 ms/ft (3.3 to 6.6 ms/m) of burden (from model blasting experiments)
3. Hagan (1977)	8 ms/m of burden for long collars, low powder factor (kg/m^3), soft, densely fissured, low density rock; and 4 ms/m of burden for short collars, high powder factor (kg/m^3), dense, tough, massive rock
4. Lang & Favreau (1972)	1.5 to 2.5 ms/ft (5 to 8.3 ms/m) of burden (from high speed photography in iron ore mines)
5. Andrews (1981)	1 to 5 ms/ft (3.3 to 17 ms/m) of spacing between adjacent holes in a row, and 6.6 to 50 ms/m between rows
6. Winzer (1978)	3.4 ms/ft (11 ms/m) of relief between holes and about 8.6 ms/ft (28.7 ms/m) along the echelon
7. Anderson et al. (1981)	8.4 ms/ft of B_e recommendation for an optimum breakage and forward movement

Many production-scale, multiple-row blasting studies have resulted in recommended timing to improve fragmentation. Andrews (1981) suggests delays of 3.3 to 17 ms/m within rows and 6.6 to 50 ms/m of B_e between rows (or on echelon). Anderson et al. (1981) measured flyrock velocity, or gas venting, through the collar stemming to establish a 3.4 ms/ft of hole spacing and 8.4 ms/ft of B_e recommendation for optimum breakage and forward movement. Similar work in which muckpile profiles were mapped indicates that optimum forward throw and muckpile height reduction occur for delay ratios of 4.2 ms/ft of S_e and 10 ms/ft of B_e, while forward throw is minimised, resulting in high muck piles, with ratios of 1.5 to 2 ms/ft of S_e and 5 to 6 ms/ft of B_e (Winzer et al. 1981). Hagan (1977) has shown for single row production shooting and *S:B* of 1.2 to 1.6 that timing ratios greater than 1.2 ms/ft of B_e are ideal. Hagan recommended 1.2 ms/ft for multiple-row production blasting in hard rock, while using high powder factors and short stemming lengths. A 2.4 ms/ft was recommended for soft rock with long stemming lengths and low powder factors. To control ground vibrations, Kopp (1987) recommended that timing ratios of 1.3 ms/ft of *S* and 1.2 to 4.3 ms/ft of B_e be used.

The timing ratios cited are found to vary over a wide range. A great deal of research on the effects of initiation timing cannot be compared due to lack of similar variables such as geology, scale, and explosive type.

Winzer et al. (1983) recognised the need to qualify delay ratios, in a general way, based on existing fracture density. Competent dense rock requires lower delay ratios to achieve fine fragmentation, while weak fractured rock fragments best with higher delays.

In underground mines delays used are either short (millisecond) or long (half second) period electric or non-electric. In general, long periods provide coarse fragmentation, and muck pile placement is high and close to the blasted face. These delays are necessary in tight headings and raises where more time is necessary to displace rock for individual delays. Short-period delays generate finer fragmentation and a long, low muck pile profile.

Today, vibrations are major concern in blasting. It means the size of blasts is reduced and hence productivity is adversely affected. The only way to overcome vibrations is to use more and more delays. If two charges fire at any interval greater than 8 ms, vibrations do not accumulate. Using combination of delay systems, large blasts can be planned in such a way that only small charges fire at intervals of 8 ms. When vibrations are a problem, steps involved in selecting suitable time intervals are as below:

1. Maximum allowable charge per delay should be determined if vibrations impose such limit. Depending on the charge per hole, this will fix the maximum number of holes that can be blasted.

2. Delay interval between holes in a row should be decided. A useful guide line for this purpose is 1 to 5 ms/ft (3 to 15 ms/m) of burden (depending on the rock type-lower for harder rocks and higher for soft formations). A delay ratio of about 3 ms/ft (9 ms/m) of burden gives good results in many kinds of rock (Andrews 1981).

3. Delay between rows should be determined. A proposed guide line is 2 to three times that between holes in a row (Andrews 1981). A long delay time between rows is needed so that the burden from previously fired holes has enough time to move forward to accommodate broken rock from subsequent rows. The long delay time also reduces the tendency of later firing rows to move upwards and provides added relief for the last row so that backbreak is reduced.

4. Due to the longer timing intervals required in such designs, only surface delays may result in initiation cut-offs. Therefore, in addition to the surface delay system downhole system should be used to ensure that all or most of the charges are energised before the first one fires. The first in-the hole delay should fire only after all or most of the surface delays have been consumed.

CHAPTER 12

Designing surface blasting rounds

12.1 DESIGN OBJECTIVES

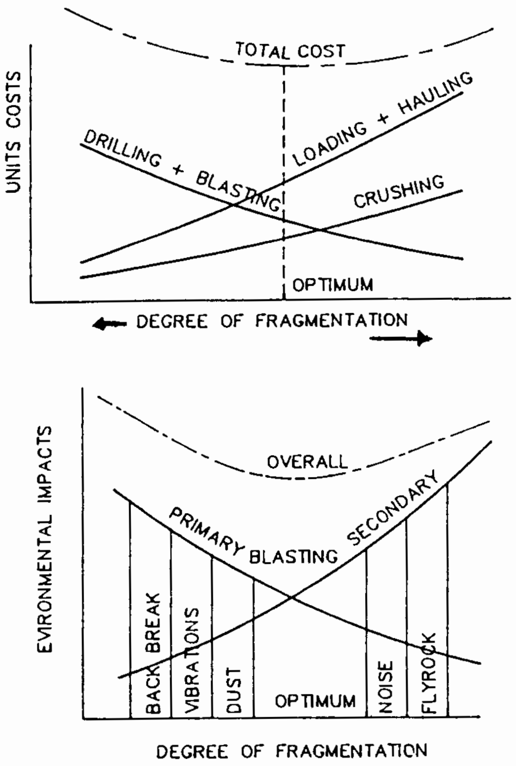
Good primary blasts result from the correct choice of explosives, proper blast design parameters and procedures. The primary requisites for any blasting round are that it ensure optimum results for existing operating conditions, possess adequate flexibility, and be relatively simple to employ. It is important that the relative arrangement of blastholes within a round be properly balanced to take advantage of the energy released by the explosives and the properties of the material being blasted. There are also environmental and operational factors peculiar to each mine that will limit the choice of blasting patterns.

The purpose of this chapter is to outline the basic principles, significant variables, and procedures for their integration in the design of rounds that are applicable to various conditions of surface mining. In case of blasting agents particular attention must be given to priming, control of confinement and selection of charge diameters that ensure complete explosive-reaction execution. Proper fragmentation being the primary objective is discussed first, followed by other blasting parameters which play important roles in designing surface blasting rounds.

12.2 FRAGMENTATION

Fragmentation refers to the size distribution of the rock obtained after blasting. It is rather difficult to define fragmentation since there is no generally accepted way of measuring fragmentation (Chapter 10). In many cases reference is made to the average size of the resulting broken rock and sometime fragmentation refers to the largest boulders resulting from a blast.

In most cases the term 'better fragmentation' is used in the sense of smaller average size with uniform fragments. Poor fragmentation refers to the case when large boulders requiring secondary blasting are in greater proportion. In some other operations such as construction work, 'better fragmentation' is one when proper size of boulders are produced. In most situations the optimum fragment size depends on the requirements of the operation including those of loading equipment and crusher size. In production blasting most often the number of boulders per unit mass of rock is considered as an index of fragmentation. Shovel loading and crusher performances are also used as indicators of fragmentation.



a

b

Figure 12.1 a) Optimum fragmentation and b) correlation between fragmentation and adverse impacts and costs.

'Optimum blasting' (Fig. 12.1a) has been suggested as obtaining the proper degree of fragmentation to achieve the lowest combined cost of drilling, blasting, loading, hauling and crushing (McKenzie 1966). Production can be increased from a given combination of shovels and trucks if fragmentation is suitable. With poor fragmentation even additional loading and hauling units can not increase the production. However, the actual range of optimum fragmentation size will depend upon the individual requirements of any mining operation and also on several design parameters. Da Gama & Jimeno (1993) indicated that at optimum fragmentation environmental impacts are also optimum (Fig. 12.1b). Bhandari (1975b) had shown that energy utilisation of explosives was much better when optimum fragmentation was obtained.

12.3 BENCH GEOMETRY

Figures 12.2 and 12.3 show some parameters used in bench drilling and blasting. Figure 12.2 shows those dimensions that are used for two holes blasting while Figure 12.3 gives dimensions for multi hole blasting.

D_e = diameter of the explosive in the borehole;
D = diameter of borehole;
B = burden, or the distance from a charge axis to the nearest free face;

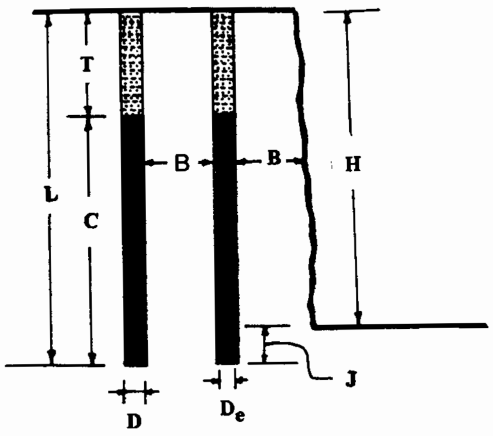


Figure 12.2 Parameters in two holes blasting.

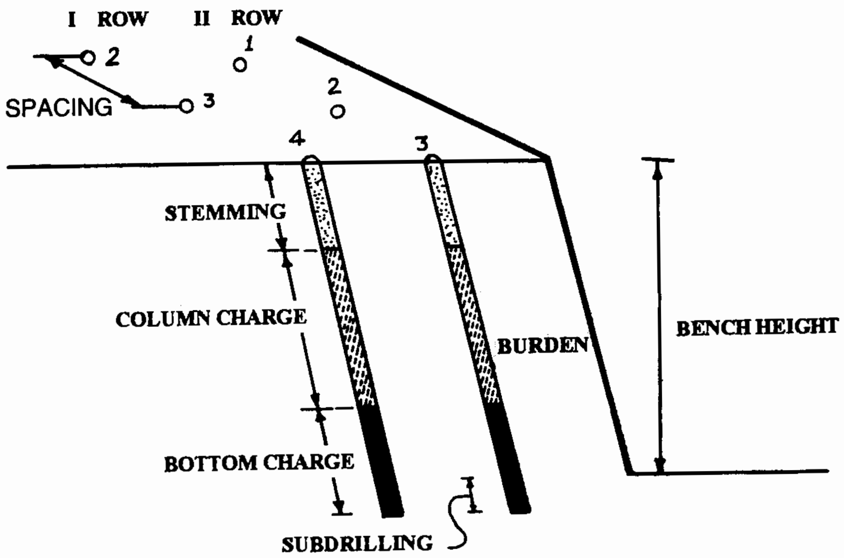


Figure 12.3 Parameters in multihole bench blasting.

S = spacing or distance between adjacent holes measured perpendicular to the burden;
L = depth of the hole;
J = subdrilling, or the hole depth drilled below the bench;
T = the part of the hole that does not contain explosive, or collar length;
H = bench height;
C = charge length.

It may be mentioned that B and S are burden and spacing that are measured with reference to free faces. Drilled burden and spacing may not be same as the effective parameters, the latter may be referred as effective burden B_e, and effective spacing S_e.

12.3.1 *Bench height*

One of the primary factors that controls the design of a blast is that of bench geometry. Usually, the bench height, H, is relatively constant for most multi-level pits (Fig. 12.2) and its value is set to conform with the working specifications of loading equipment.

Bench heights vary within wide limits. In large open pits from which stone or minerals are mined, bench heights of 15-20 m are common, although benches with heights up to 30 m are occasionally encountered. In many places, bench heights are limited as safety precaution. In road building and other construction projects, the same work sites may have benches varying from a few decimetres to several meters.

The bench height is related to the degree of heaping and spreading of material broken by blasting, thus, directly affecting displacement requirements to be accomplished by the round design. The height also limits the maximum and minimum charge diameters that should be used, and it influences drill selection. The maximum bench height is dictated by the appropriate statutory authority. In general, faces with heights of about 10-18 m have been considered the most economical and least hazardous to work. Where it is necessary to practice selective mining/quarrying, the face height may be dictated by the thickness of ore/rock of a certain quality. The most economical face height may also be determined by the drill penetration rate; whenever the penetration rate decreases significantly, it is generally uneconomical to drill deeper. High faces pose the problem of considerable bit wander or drift, especially with smaller diameter blastholes. The deviation of blasthole places a limit on the maximum allowable bench height.

In tests to determine limiting burden and spacing, without any consideration of the degree of fragmentation, charge length (and hence bench height) was found to have a strong influence on the limiting spacing between simultaneously initiated blastholes. The maximum spacing to burden ratio for a given rock can not be achieved unless a minimum charge length to burden value is exceeded. When the charge is short (and the bench low), small changes in burden, B and/or spacing S, have major effects. As the bench height and charge length increase, for a given burden, the limiting spacing increases rapidly to begin with, then increases at a slower rate and finally attains a constant value. The charge length should not be less than 3 times the burden if the maximum limiting spacing is to be achieved with simultaneously-fired blast holes.

In recent years, there has been a considerable trend towards larger diameter blastholes. Because bench heights have either remained unchanged or decreased slightly, a considerable increase in the stemming length to bench height ratio has been brought about. At some open pit operation, for example, the use of large diameter blastholes (310 or 380 mm) and relatively shallow benches (12-14 m) prevent efficient charge distribution in the rock to be fragmented; the rock alongside the stemming column (relatively remote from the charge) can be up to 40% of the total rock volume to be removed. From the view point of increasing blasting efficiency (through reduced collar rock and greater charge length to burden ratios), there are good technical reasons, where blasthole diameters are large, for increasing bench heights.

12.3.2 *Bench width*

There is a minimum bench width, measured horizontally in a direction perpendicular to the pit wall, for each bench height and set of pit operating conditions, whose value is established by the working requirements of the loading and haulage equipment. The width also must be such as to ensure stability of the excavation, both before and after blasting, because each blast effectively reduces the restraint sustaining the pit walls at higher elevations. Because of the limits set by requirements for equipment operating room and bank stability, there is a maximum bench width that should not be exceeded by any blast.

12.3.3 *Broken characteristics of materials*

All materials expand when broken, and each has its own characteristic swell factor, S_f. The factor is defined as the ratio of the volume of a unit weight of solid material to that when broken. The importance of swell in the design of blasts is that it directly affects the choice of an initiation-timing system and limits the number of rows in the blast. For the box cut, expansion will occur in two directions, while for the corner or side cut it will be in three directions.

If swell is assumed to be uniform, each solid dimension towards an open face could be expected to expand to an amount equal in value to the original solid dimension divided by either the square root or cube root of the S_f ratio characteristic of the material. The increase will vary from 5-30% with 15% being considered average.

Loose material will not normally stand vertically of its own accord. Horizontal spread towards all open ends beyond that due to swell will be a function of the natural angle of repose of the material. Moisture and greater than normal throw from blasting will effectively lower the repose angle and increase the spread.

It may be necessary to consider the effects of weathering on broken material. Some materials deteriorate rapidly where exposed. In other instances, serious difficulties arise during crushing, grinding, and screening of certain materials when they are wet. Therefore, total volume and frequency of firing blasts will be limited by the maximum permissible exposure time of the material.

12.3.4 *Operating restrictions*

Each blasting pattern has distinctive effect on the heaping or scattering of broken material, thus strongly, affecting loading. In the event shovels are used, the broken pile is best loaded when its height is approximately equal to the maximum cutting height of shovel. Lower heights require excessive shovel movement, while higher piles present the hazard of having to undercut boulders or slabs that might be left on the pile top. If the ledge is lower than optimum, heaping is necessary to provide good loading. In addition to height restrictions imposed by operating characteristics of the loading equipment, there will be limitations as to spread of broken material over the bench floor.

The haulage method must also be considered in blast design. For example, conveyor and rail haulage generally require long benches with material displacement during blasting being directed parallel to the pit walls for a parallel loading approach.

Blasts necessarily should contain only a few rows of holes because loading will be restricted quite often to only one side of the loader. Except for very high ledges, truck haulage favours blasts with several rows due to greater manoeuvrability of the haulage units. For draglines and high lifts, greater displacement of broken material is desired than that required for shovels.

12.4 BLASTHOLE DIAMETER

The hole diameter is selected such that in combination with appropriate positioning of the holes, will give fragmentation suitable for the loading and transportation equipment and crusher used. Additional factors to be considered in the determination of the hole diameter are: the size of operation, the bench height, the type of explosive used and rock characteristics. Occasionally it may be necessary to use two different hole sizes even within the same blast.

As long as poor fragmentation does not create problems for the mine operator, generally the larger the blasthole, the more economical drilling and blasting become.

The larger diameter holes could be drilled with only a small increase in the cost per meter of drilling and larger holes could be put further apart. However, when the blasthole diameter is increased and powder factor or energy remains constant, the larger blasthole pattern generally gives coarser fragmentation. Perhaps by keeping burden unchanged and elongating spacing alone the problem of coarser fragmentation can be over come.

Where joints or pronounced bedding planes divide the burden into large blocks, or where hard boulders lie in a matrix of softer strata acceptable fragmentation is often achieved only when each block or boulder has a blasthole (Fig. 12.4). This usually

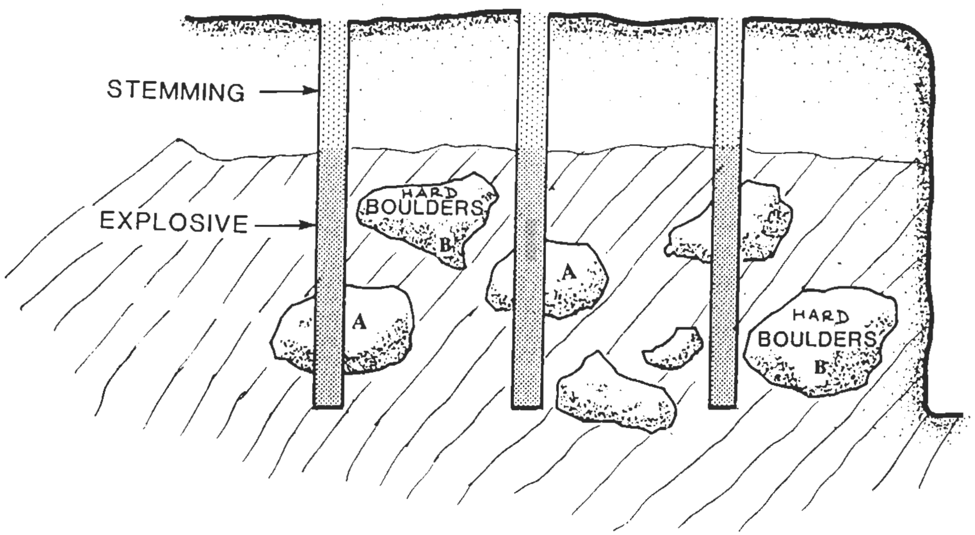


Figure 12.4 Boulders in soft material. Boulders marked *A* will be broken while boulders marked *B* will not be broken.

necessitates the use of smaller diameter blastholes and correspondingly smaller blast size. In a strata which exhibit a dense network of (natural) fissures, on the other hand, increases in blasthole diameter cause relatively small change in fragmentation.

The maximum possible charge diameter is rarely utilised since fragmentation may be too large for efficient handling. The primary crusher and loader bucket sizes will limit the largest size of material desired. The effects of rock structure sizing, however, must be also considered. Another factor that may restrict the maximum charge diameter may be when the loading density of the explosive must be limited for ground vibrations or other adverse blast effects. In addition, drilling time for the necessary meterage that satisfies production requirements may limit the size of blastholes for proper equipment operating balance.

The minimum charge diameter must be somewhat greater than the critical diameter of the explosive because velocity of detonation increases as charge diameter enlarges, but this is up to a certain finite value beyond which no further velocity increase results. It is thus advisable to utilise that diameter where maximum velocity and energy yield per kilogram are attained.

Hole diameters vary from 35 mm (13/8 in.) in small benches to up to 440 mm (approx. 17 in.) in large benches. With small diameter holes the blast size is small. Large blasts need large diameter deeper holes. In Europe and Scandinavia where rock is more competent, smaller diameter holes 64-125 mm, are used on very high benches. In USA, Australia, Canada and India trend is towards large diameter holes with lower bench heights. A combination of two conditions is practised in different countries. Further, depending on type of rock different trends exist. For example in India limestone quarries use 100-150 mm diameter holes, in coal mines hole diameters vary from 150-269 mm, and in iron ore mines hole diameters used are generally 200-269 mm and above.

12.5 BLASTHOLE INCLINATION

In recent years, increasing attention has been given by open pit and quarry operators to the drilling of blasthole at angles up to 20° from the vertical. The benefits from inclined charges are the reduction of collar and toe regions, less subdrilling requirement, and (usually) increased throw. But airblast and flyrock may occur more easily due to the smaller volume of material surrounding the collar. Most often it is advisable to reduce loading density in the collar zone to minimise those effects. Inclined holes are successfully used in Europe and Scandinavia where high benches and smaller diameter holes in medium to high strength rock are practised.

The use of vertical blastholes usually produces a considerable variation in burden between the top and bottom of the face (Fig. 12.5). This is particularly the case where the face is high and/or highly inclined. Front row blastholes are often collared near the crest so as to remove the heavy toe burden. But then, of course, explosion gases may blow out prematurely in the upper face, causing high levels of noise, air blast and flyrock. The rate at which such venting reduces blasthole pressure near floor level may be sufficiently great to prevent adequate breakage or movement of the toe. This effect is more pronounced for top-primed than the bottom-primed charges. If a vertical blasthole is drilled at the nominal burden distance back from the

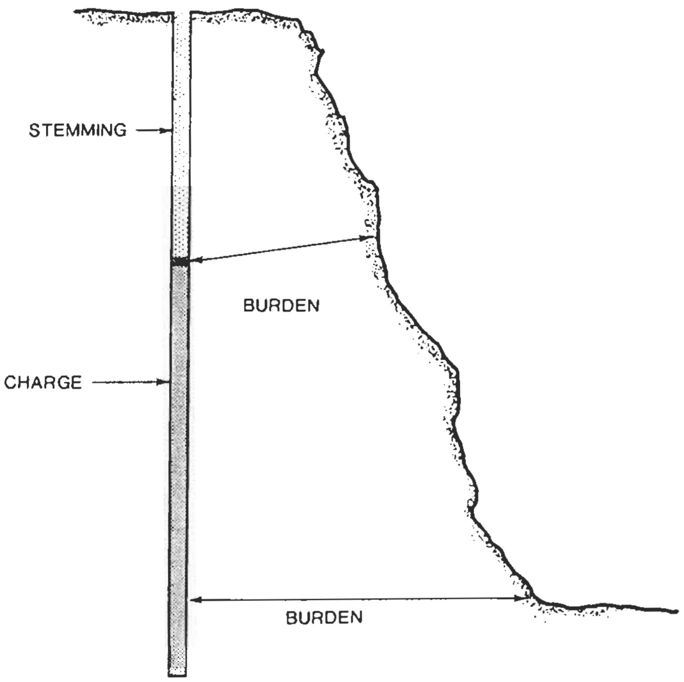


Figure 12.5 Variation of burden in case of vertical hole on an inclined face.

crest, on the other hand, hard immovable toe can be expected. One of the major advantages of inclined blastholes, therefore, is the greater uniformity of burden throughout the length of the blasthole. Ideally, the blasthole should parallel the face.

For constant fragmentation, inclined drilling allows the use of greater blasthole spacing. It also results in greater operating safety for men and machines due to the cleaner face that is obtained. When blastholes are inclined, less subdrilling is necessary and so less damage to the area beneath the pit floor is caused; drilling of the next bench therefore, becomes easier. In some circumstances, there, may be very good reasons to go to the extra trouble of drilling inclined blastholes. Where benches are high, angles of 20-25° are recommended. Angles greater than 25° are seldom used because of difficulty in maintaining blasthole alignments, excessive bit wear and/or difficulty in charging blastholes.

If the blasting operation is to be successful, it is essential that the holes are correctly drilled. To assure this, the holes must be properly aligned and straight. Deviation tends to increase with increasing depth. Compensation for this deviation demands increased drilling.

If the amount of deviation is to be kept to minimum it is essential for the rod to be rigid. Full-section round rods are more rigid than light rods with the same thread diameter. Deviation can be minimised by the use of guide tubes. DTH drills using drill pipes and low feed force assure best hole straightness.

12.6 THE BURDEN

This is one of the most critical parameters in the design of blasts. The burden B, is the distance from a charge axis to the nearest free face at the time charge detonates. With multiple row blasts, the burden may not necessarily be given as the distance to the nearest free face. As boreholes with lower delay periods detonate, they too create new free faces. As a result, the true or effective burden will depend on the selection of the delay pattern.

There are many relationships available for obtaining the approximate value of the burden for various explosive and rock combinations. Most relationships utilise either charge volume, charge weight or hole diameter as the basic parameter, with the burden being a function of the cube-root or square-root of the independent variable. The cube-root law is stated as:

$$B = K\,Q^{1/3}$$

where, B is the burden (ft or m), Q is the charge weight (lb or kg) and K is an empirical factor depending on the explosive and rock properties.

Of the many formulae using the charge diameter, D_e, as the independent variable, that proposed by Anderson (1952) was the simplest. The expression did not consider either the properties of explosives or the rock. To account for the characteristics of the explosives and the strengths of the materials, Pearse (1955) and later Allsman (1960) and Speath (1960), proposed similar formulae. The difficulty in applying these relationships lies in the selection of appropriate values for the properties of the explosive and rock. A modified formula was given by Ash (1968) for the first approximation for burden determination dependent on the borehole diameter:

$$\text{Burden } (B) = K\,D_e/12$$

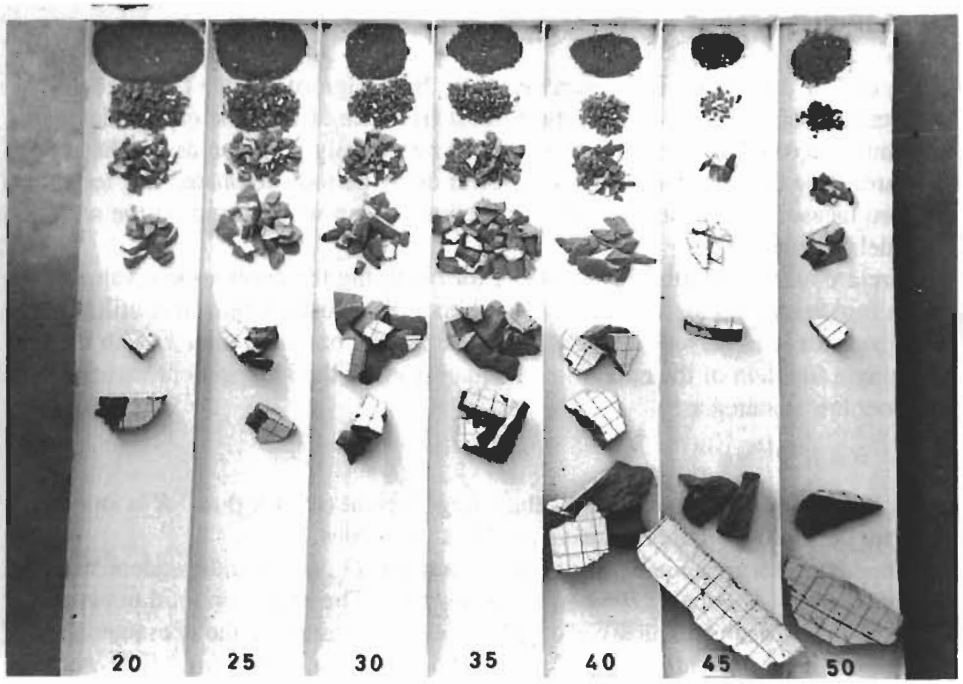
where B = Burden (ft), D_e = Explosive diameter (in), K = between 25 and 35, equal to 30 in average conditions, in rock with density around 2.7 g/cc. If the rock density is higher than this, then K would be reduced and vice versa.

Several other formulae are available but in general for any given explosive-rock combination, in most instances, approximately the same result is obtained. In an analysis of data for 100 opencast mines (Vutukuri & Bhandari 1973), it was found that burdens used were proportional to hole diameters according to the relationship

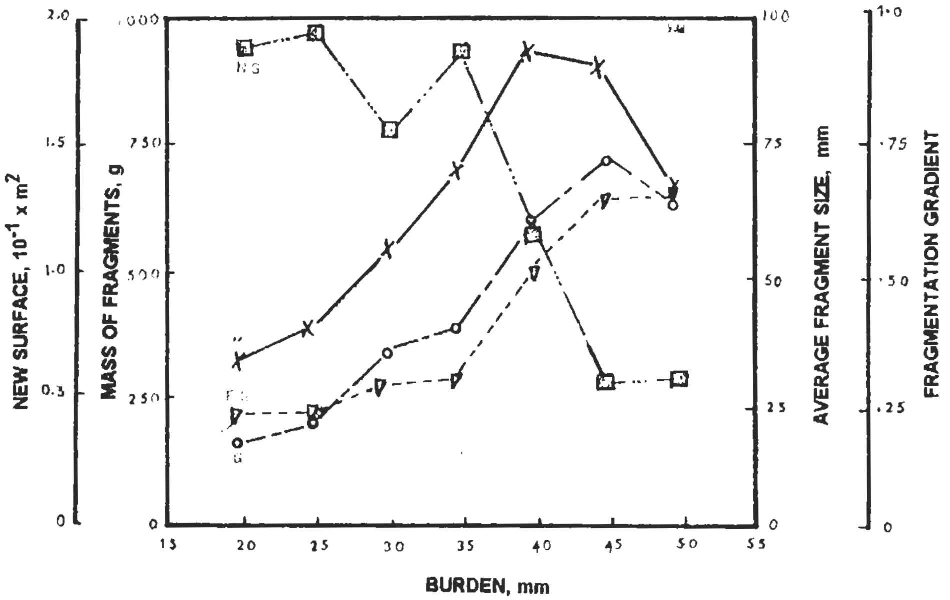
$$B = 0.024\,D + 0.85$$

where B is in meters, and D is the hole diameter in mm.

For a given charge, rock type and spacing, there is an optimum burden (for which the volume of suitably fragmented and loosened rock is maximum and toe conditions are acceptable). Bhandari (1975a, b) distinguished between optimum breakage burden and optimum fragmentation burden. At the optimum breakage burden volume or mass of rock broken is maximum but the fragmentation obtained is not essentially acceptable as it has greater fines and some large boulders. This was shown by laboratory scale blasting (Fig. 12.6) where it was found that optimum fragmentation burden was 30-40% less than the optimum breakage burden. At the optimum fragmentation burden, quantity of rock broken is less but the fragmentation obtained is uniform and large boulders are absent. At optimum fragmentation burdens larger spacing can be



a



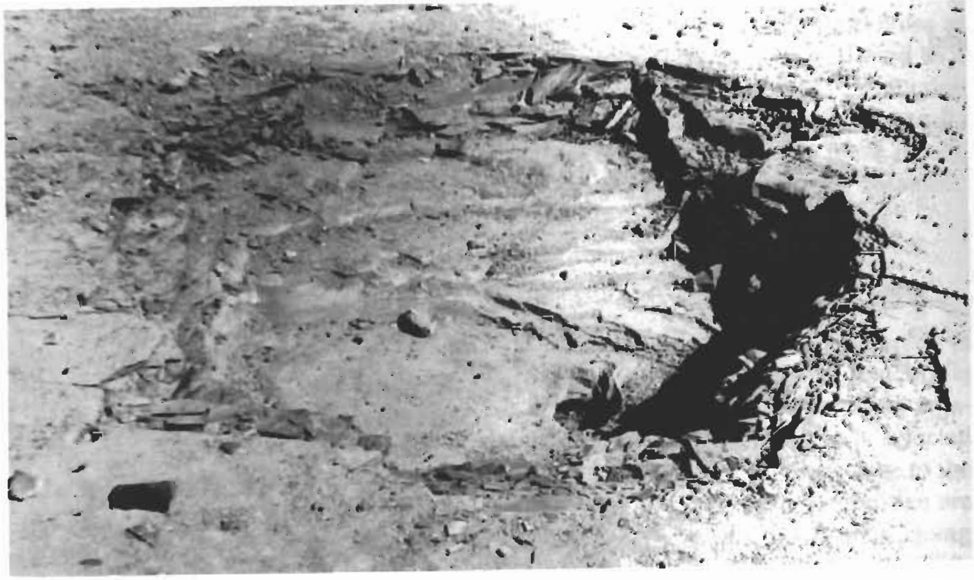
b

Figure 12.6 a) Sized fragments for single hole small scale tests for 20, 25, 30, 35, 40, 45 and 50 mm burdens placed in vertical rows. Particles arranged in vertical to rows for small to the largest sizes. b) Variation of measured mass, M: average fragment size, FS: fragmentation gradient, G: and new surface, NS: created for single hole tests in cement mortar blocks.

utilised as compared to that for optimum breakage burdens hence there is no loss of total quantity of rock breakage.

If the burden is too small, detonation gases escape into the atmosphere in the form of noise and airblast therefore less energy is available for the fragmentation. Aside from the objectionable aspects, noise and airblast are outward signs of the inefficient use of explosives energy. Where the burden value is too large, gases are confined for a time interval longer than desired; which can result in higher ground vibrations, excessive backbreak, toe and uneven floor.

In a ground, where good fragmentation is required and which is tough and/or blocky, the burden and also spacing should be conservative. When good fragmentation is less important, or when blasting ground tends to break easily, satisfactory re-



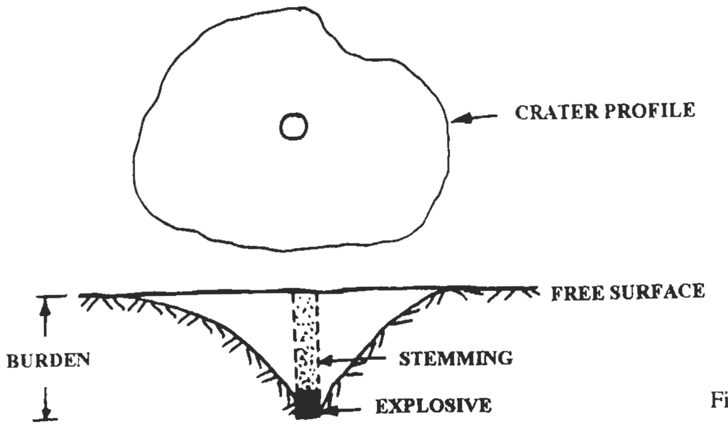


Figure 12.7 Cratering tests.

sults may be obtained by drilling larger diameter blastholes on a correspondingly larger pattern.

In instantaneous single row blast, the effect of adjacent charges superimpose and, where fragmentation is unimportant or is structurally controlled, burdens can be 25-30% greater than those fixed on long delay.

Besides various empirical formulae, some field determination procedure for determination of optimum burden is needed. In the early 1960's the application of crater theory was used for designing blasts (Livingston 1956). This procedure is based on the concepts of 'energy factor' and 'critical depth'. To determine these parameters, charges of the same weight are fired at various depths in the material concerned and volume broken by each blast is measured (Fig. 12.7). The critical depth, N is the depth of charge when surface rock failure is first apparent.

The energy factor E is calculated from

$$N = E\,W^{1/3}$$

and,

$$\Delta = B/N$$

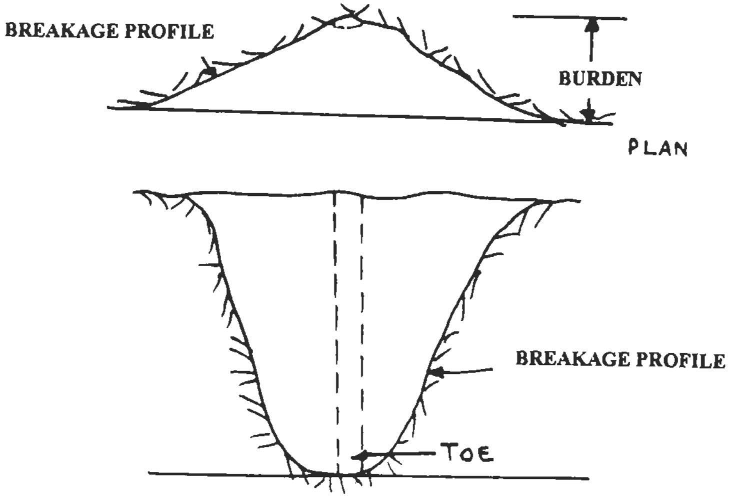


Figure 12.8 Bench cratering tests.

thus,

$$B = \Delta N = \Delta E\,W^{1/3}$$

Therefore, for any given burden B the required charge weight may be determined. Values for E range from 1.8-4.6, the larger magnitudes pertaining to the harder brittle rocks. The depth ratio coefficient ranges between 0.45 and 1.0. Livingston (1956) suggested that a transition in the mode of failure, from shear in the softer rock to tensile in the harder rock, takes place when E is around 3.3. Bauer (1961) and Grant (1970) used these spherical charges to relate with bench blast design. Many claims of success by this method have been made, though, there is very little similarity between the cratering geometry and bench geometry which uses long cylindrical charges breaking to two free faces.

Further, a semi-infinite, joint free rock area is required for the experimental work. A modified system called Bench Fragmentation method could be useful (Bhandari 1977 and Bhandari et al. 1979). In this method (Fig. 12.8) separate single holes at different burdens with constant explosive charge (mass or volume) are blasted to a bench geometry. Preferably, the same explosive and hole diameter as in production blasting are used with same method of priming and blasting. After each blast, results in terms of volume of breakage and fragmentation are recorded. From these results the optimum breakage burden or the optimum fragmentation burden can be obtained.

12.7 SPACING

The distance between adjacent blast holes, measured perpendicular to the burden is defined as the spacing which controls the mutual effect between holes. Spacing is calculated as a function of burden, hole depth, relative primer location between adjacent charges and also depends upon initiation time interval. The spacing is selected according to widely held concept that since the break angle made by the charge to the bench face is near 90° (Fig. 12.9) hence spacing larger than two times the burden are not possible (Gregory 1973). Over the decades, in most mining operations, spacing to burden ratios used have been between one and two (Fraenkel 1957; Speath 1960; Kochanowski 1963; Ash & Pearse 1961; Lewis & Clark 1964). Commonly ratio suggested is 1.2 to 1.3. Vutukuri & Bhandari (1973) found from a survey of 100 operations that drilled spacing to burden ratio practised was:

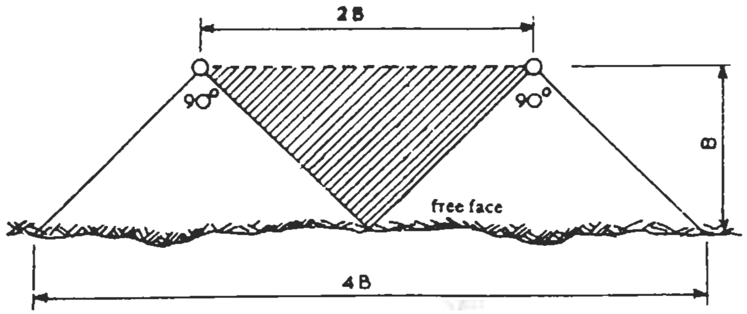


Figure 12.9 Break angle near to 90° in case of large burden.

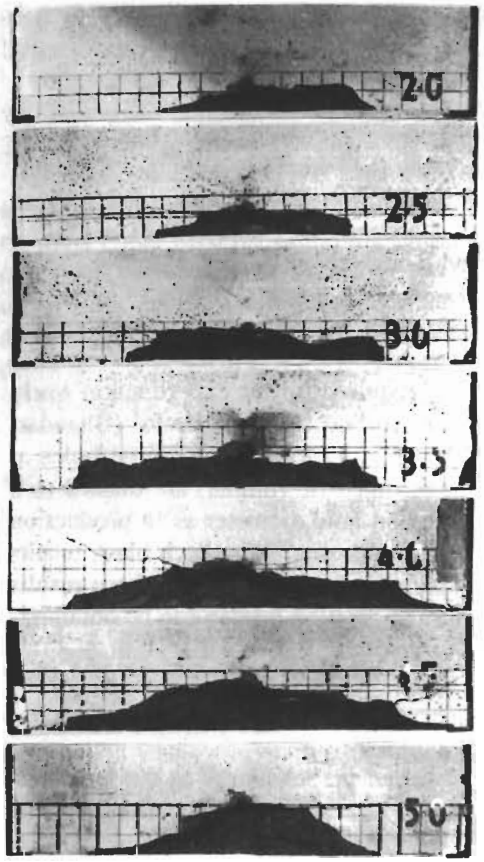


Figure 12.10 Variation in break angles with burden as shown in small scale tests.

$$S = 0.9\,B + 0.91$$

where both the burden and spacing were in meters.

In contrast to general practice of adopting spacing to burden ratio ranging from one to two, Langefors et al. (1965) and Langefors (1966) demonstrated from laboratory model scale tests that ratio exceeding three for simultaneously fired charges in a single row gave better fragmentation. This was observed by reducing the conventionally used burden. For the same model tests with multiple rows of charges fired together, but rows of holes delayed relatively resulted in good fragmentation up to spacing to burden ratio of eight. Kihlstrom et al. (1973) verified successfully wide hole spacing technique on production scale blasting. The method suggested became popular in early 1970's and is known as Swedish Wide Spacing Technique. Bhandari (1975a, b, c) showed that the break angles were much larger up to 150° for optimum fragmentation burden which allowed spacing to burden ratios greater than two (Fig. 12.10). It was explained that reduced burdens allowed much better utilisation of stress waves and gas energy resulting in improved fragmentation. A burden to spacing ratio up to 3 or 4 was recommended with reduced burden.

The usefulness of laboratory test results have also been proved in production scale

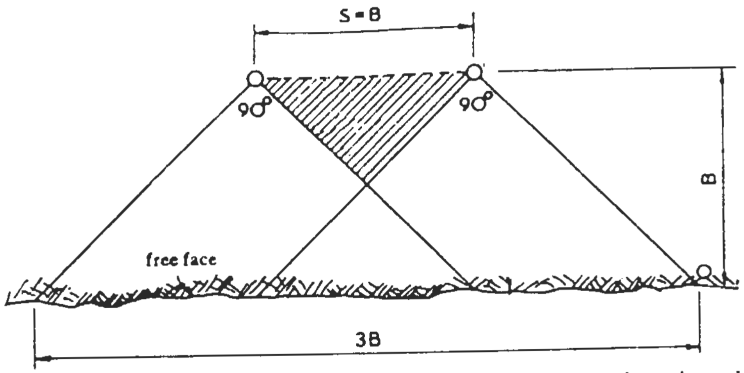


Figure 12.11 Adjacent charges breaking on separate delays need spacing ~ burden.

blasting (Lambooy & Espley Jones 1970; Lang & Favreau 1972; Kihlstrom 1973; Brown 1973; Johansson & Persson 1974; Mikkelborg 1974; Bhandari 1975a, b, c; 1983; Winzer et al. 1983; Rustan & Lin 1987). These applications were mostly in open pit mines. Ratios used varied in different operations. Since then the wide spacing technique has been used in several mines, sometimes this technique works successfully whereas it fails on many other occasions. Sometimes variation is noticed in the different parts of a quarry. Works of Bhandari & Badal (1990a) and Badal (1991) indicate that the variation is on account of geological structural orientation. When the joints are across the face, a close spacing is needed and when the joints are parallel to the free face larger spacing ratios are possible.

Whenever adjacent charges are initiated separately with a time-delay interval of sufficient duration to permit each charge to break separately, there is no interaction between the holes. In such cases spacing should approximate the burden i.e. $S \sim B$ (Fig. 12.11). When the time interval for initiating adjacent charges is reduced, there may be reinforcement of the stresses generated by the explosives in the zone between blast holes.

Adequate results can normally be obtained when spacing and burden are about equal forming a grid pattern. But the elongated pattern, where spacing exceeds the burden, is more effective, particularly in massive, hard-breaking formations. A large spacing and small burden tend to cause more twisting and tearing of the rocks, less splitting along the line of the blast holes, and less back break. Where short delay initiation methods are used, greater spacing are of advantage in that there is less chance of cut-offs.

Spacing appreciably less than the burden tend to cause premature splitting between blast holes and early loosening of the stemming. Both these effects encourage rapid release of gases to the atmosphere; back break is usually considerable. The loss of bubble energy detracts from overall breakage in the burden, and large slabs are often found in the muckpile. The production of large size fragmentation such as for break water armour stone utilises this principle of reduced spacing (Bhandari & Tanwar 1993).

Where spacing to burden ratio is too high (Fig. 12.12), each charge fragments and detaches a prismatic section of rock; the initial face midway between blast holes may

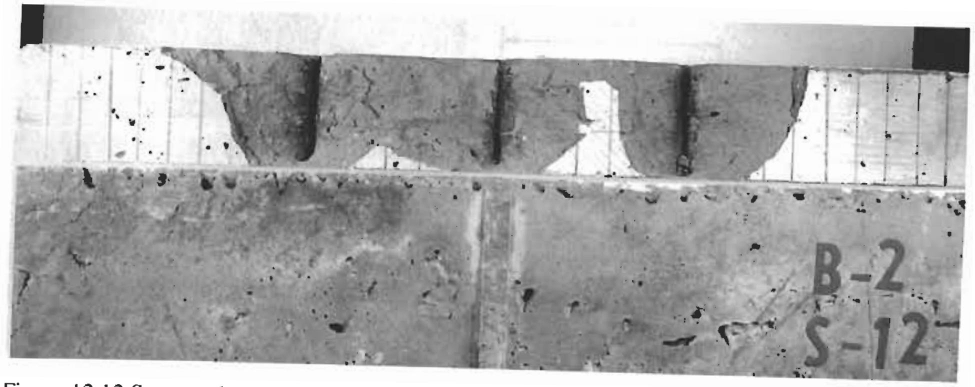


Figure 12.12 Separate breakage of charge if spacing is too high.

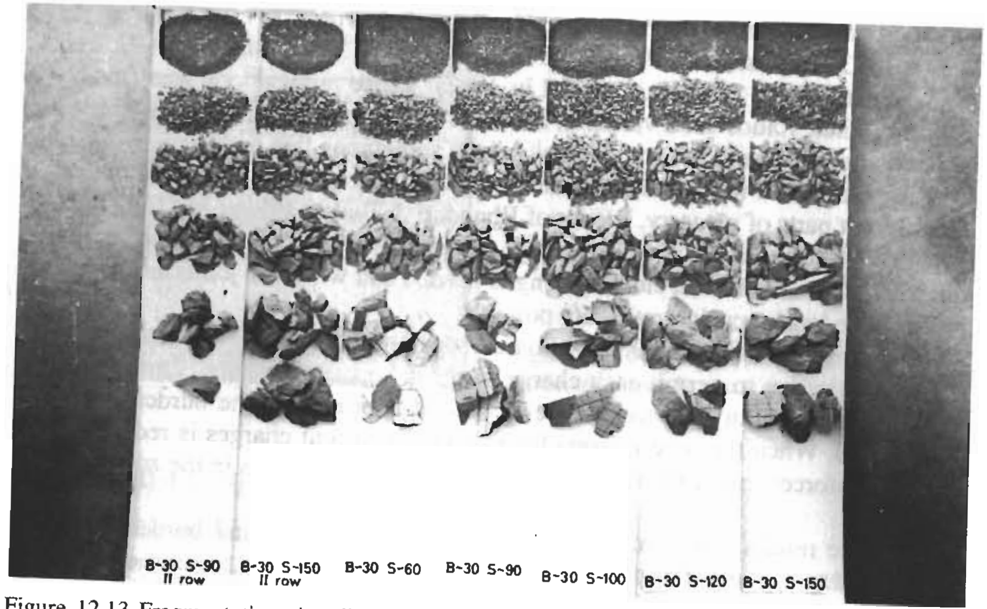


Figure 12.13 Fragmentation size distribution for small scale tests with different spacing (S) and burden (B) ratio.

remain intact, especially near grade level, where the unsuitability of spacing would be manifested as bootlegs or toes.

In delay blasting of multirow shots, best results with rectangular patterns are obtained at spacing two times the burden; with further increase in spacing, fragmentation declines. For staggered rows, fragmentation improves as S increases up to about $4\ B$ (Fig. 12.13). The difference between straight and staggered patterns is considerable, the latter giving generally superior results, especially where the rock is massive and difficult to break. Production increase up to 30% have been reported by decreasing B and increasing S, values of blast hole length and charge weight/m^3 rock remaining constant. With large S/B values, the number of blast holes can be reduced

and/or loading efficiency increased as a result of the improved, more uniform fragmentation.

Normally, spacing and burden are related to blast hole depth and more particularly to charge length. In multirow shots in a massive sandstone, for example, adequate fragmentation with 230 mm diameter blast holes may necessitate powder factors of 0.65 and 0.57 kg/m^3 for bench heights of 12 and 25 m respectively.

Changes in burden tend to affect the overall degree of fragmentation and presence of toe or high bottom much more rapidly than changes in spacing. If blasting results are more than adequate and it is decided to try a larger pattern, the spacing should usually be increased rather than the burden. If the current pattern is already quite elongated, however, it may be necessary to keep the spacing constant and increase the burden marginally. With any trial blast, it is important that muck pile is completely removed and toe conditions assessed before the next blast is drilled out.

If, at any time, large chunks of rock are actually desired, the burden should be made about twice the spacing and if possible single row rather than multirow blasts should be fired. If ground vibration levels permit, the row of blast holes, should be fired instantaneously (i.e. without delays). This view is supported by experimental studies by Dick et al. (1973) and Smith (1976).

In thin tabular deposits (e.g. coal), the burden and spacing are usually made equal so that blast holes are drilled on a square grid pattern. The largest blast hole pattern that can be used successfully is generally controlled by the thickness of the seam. For a seam 5 m thick, for example, patterns larger than about 5 m^2 rarely give satisfactory results. For the same energy factor, the firing of a 10 m^2 pattern in this 5 m thick seam would cause the material around each blast hole to be cratered upwards, but midway between the bottom of blast holes, the material would move very little, if at all, and would remain in its solid unblasted condition. The resulting solid plugs, of course, will retard the loading operations. For efficient blasting, therefore, the burden and spacing in the pattern should not normally be greater than the thickness of the seam. The use of large diameter blast holes (and a correspondingly larger pattern) in a thin seam is prevented by the fact that the size of the blast hole pattern is restricted by seam thickness. When the blast hole diameter becomes too large, there is also the problem of poor charge distribution; the explosive energy would then be concentrated down the bottom of the blast holes. This could well result in excessive fines in the material around the charge, with large lumps from the middle and top of the seam. To summarise, the following could be stated as basic principles for spacing:

1. For sequence delays in the same row, $S \sim B$;
2. For simultaneous timing in the same row, $S \sim 2\ B$;
3. For multiple rows with sequence timing between charges in the same row, the entire round should be drilled in a square arrangement, particularly if identical timing is used for charges located laterally with one another in adjacent rows; and
4. Staggered patterns are preferred between rows where all holes in a single row are fired simultaneously but timing between rows is delayed.

It is important that once the pattern has been designed, the holes are drilled in the correct place, at correct angle and to the correct depth.

12.8 SUBDRILLING

To avoid leaving a hump, bootleg or toe in bench blasting, the blastholes are drilled below the floor (grade) level. This is termed as subgrade drilling, underdrilling or subdrilling. The optimum effective subdrilling depends on:

- The structural formation and density of the rock,
- The type of explosive (and more particularly the energy generated per meter of blasthole),
- The blasthole diameter,
- The blasthole inclination,
- The effective burden, and
- The location of initiators in the charge.

In practice subdrilling (J) should be roughly 0.3 times the burden, B. If a pronounced discontinuity, or parting is located at grade level often no subdrilling is required. Unless such a condition exists subdrilling should be at least one-third the value of the burden. The required subdrilling is also expressed in terms of the blasthole diameter, d. In dipping or massive rocks, subdrilling of about $8d$ is usually found to be satisfactory. But where vertical blastholes are drilled in relatively high and/or highly inclined faces, sub-drilling of $10d$ or even $12d$ may be necessary in front-row blastholes because of the heavy toe burden. Subdrilling less than 8d can often be used satisfactorily when:

1. A very high energy per meter of blasthole can be generated, and/or
2. Bottom priming is done.

Too much subdrilling must be avoided, since it:

- Wastes drilling and explosive expenditure,
- Increases ground vibration levels appreciably,
- Causes undesirable shattering of the pit floor (which may create drilling problems on the next bench down), and
- Increases the vertical movement of the blast.

In coal strip mines, subdrilling may in fact, have a negative value. Where the overburden is shallow, blastholes are often stopped sufficiently short of the coal to avoid shattering it in the blast. If soft beds lie immediately above the coal, good blasting results may be obtained by drilling down only to the bottom of the lowest hard band in the overburden. But in most cases, and especially where the overburden is deep, drilling is stopped as soon as coal appears in the drill cuttings. If the coal is relatively weak, it may be necessary to backfill 1 m or so with drill cuttings to minimise shattering of coal by the overburden blast. Such shattering is particularly to be avoided where coal seam is relatively thin.

12.9 STEMMING

The primary function of stemming is to confine the gases produced by the explosive until they have adequate time to fracture and move the ground. The type and length of stemming have no significant effect on the characteristics of the explosion generated strain-waves and hence does not increase stress-wave effect. By reducing premature venting of high pressure explosion gases to atmosphere, however, a stemming column of suitable length and consistency enhances fracture and displacement by gas energy. Experiments have shown that the critical burden can be increased significantly when a suitable type and quantity of stemming is used. For long charges, the effect of stemming variations on the limiting burden is most significant for top primed charges.

The amount of unloaded collar required for stemming is generally from one-half to two thirds of the burden dimension. This length of stemming usually maintains sufficient control over the generation of objectionable air blast and flyrock from the collar zone. When the burden has a high frequency of natural cracks and planes of weakness, relatively long stemming columns can be used. When the rock is hard and massive, the stemming should be shortest which prevents excessive noise, airblast and/or backbreak.

The resistance to ejection of water, mud and wet clay, stemming depends almost entirely on the inertia of the column. Dry granular materials, on the other hand, exhibit both inertial and high frictional effects. For this reason, dry granular stemming is much more efficient than materials which behave plastically or which tend to flow.

For blasthole diameters in the 230-380 mm range, angular crushed rock in the approximate size range of 20-30 mm makes a very effective stemming column. Larger fragments tend to damage the detonating cord downline and/or the detonator leadwires. When impulsively compressed at its lower end by the tremendous pressure of the explosion gases, crushed stone 'arches' and provides better interlocking and confinement of the gases than (relatively fine) drill cuttings. This helps to maintain peak pressure in the blasthole for a longer period of time, and the longer these gases can be 'bottled up', the more they will be able to fragment, heave and loosen the ground. Where blasthole water exists above the charge, crushed angular rock occupies its intended position rapidly rather than form a soupy suspension in case of drill cuttings. In coal blasts in strip mines, inert stemming material should be used rather than coal cuttings.

Where rock is hard and massive, the stemming column should be the shortest which prevents excessive noise, airblast, flyrock and/or backbreak. The use of small 'pocket' charges may be considered.

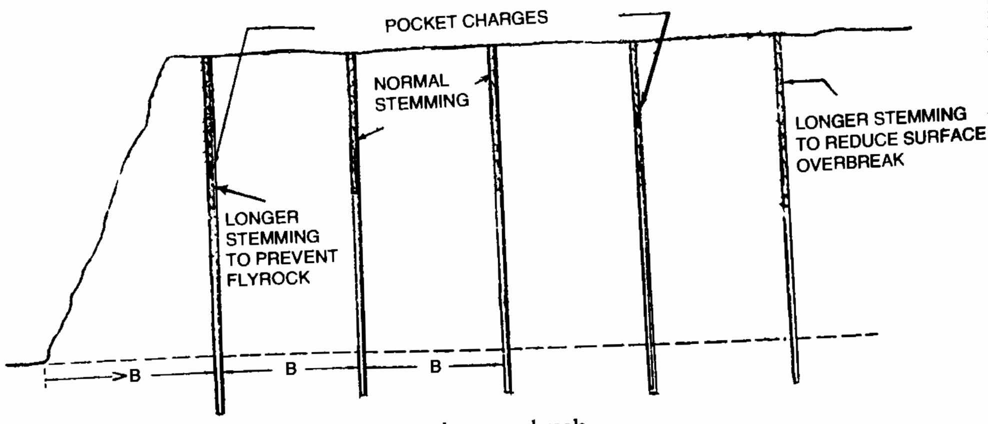


Figure 12.14 Longer stemming column to reduce overbreak.

In multirow blasts, where the mean direction of rock movement tends more and more towards the vertical with successive rows a longer stemming column is often used in the last row to reduce overbreak (Fig. 12.14).

In some mining fields for small diameter shallow hole blasting, the availability of cheap ANFO blasting agents has led to the complete filling of a hole with ANFO and no stemming is used. In this case, it is presumed that gases have self-stemming effects.

12.10 POWDER FACTOR/SPECIFIC CHARGE

Two terms are often used to relate explosive mass and consequent rock broken: Powder factor and specific charge, q. Observation of blast designs based on empirical relations has been often indicated in terms of powder factors. Powder factor (sometime also referred as charge factor) is the ratio between the total weight of explosive detonated in a blast divided by the amount of rock that is broken. It is usually expressed as kilogram per ton or kilogram per meter. In some cases reciprocal of these are referred as powder factor but correctly these are termed as specific charge. These are some times termed as charge ratio also.

As the powder factor in kg/m^3 increases, the average fragment size decreases when the burden, B, remains constant. According to Gustafsson (1981), when B exceeds 3 m, the fragment size becomes uncontrollable, especially in the upper portions of the blasthole, where charge densities are the lowest.

The powder factor varies between 0.1 kg/m^3 and 0.53 kg/m^3 for bench blasting. Powder factor for tunnelling should be larger by 1.25-1.5 times then bench blasting because of the larger fixity.

The following Table 12.1 compares various powder factors for surface mines when vibration is not a consideration. Table 12.1 also relates powder factor to muck removal equipment and geology for surface coal mining with ANFO. The smaller the muck removal equipment, the larger the powder factor necessary to obtain desired fragmentation.

Table 12.1 Powder factors for surface coal mining.

Major removal equipment	Predominent geological unit fragmented	Powder factor (kg/m^3)	Bench height (m)
Large dragline	Shale	0.30	15.0
	Shale	0.35	23.0
	Shale	0.40	–
	Sandstone	0.35	18.0
Small dragline	Shale	0.20	6.0
	Shale	0.50	7.6
	Shale	0.35	18.0
	Sandstone	0.65	26.0
Front-end loader	Shale	0.35	9.0
	Sandstone	0.65	16.0
	Sandstone	0.40	20.0
	Sandstone	0.95	15.0

Table 12.2 Powder factors for surface metal mining.

Geology	Sonic velocity range (m/s × 10)	Powder factor (kg/m^3)
Weathered limestone	0.3-0.6	0.30
Weathered porphyry	0.6-0.9	0.35
Rhyolite breccia	0.9-1.2	0.45
Monazite porphyry	1.2-1.5	0.60
Quartz sericite porphyry	1.5-1.8	0.60
Fresh limestone	1.5-1.8	0.60
Massive jasperoid	1.8-3.6	0.77

Table 12.2 shows the range of powder factors employed in surface metal mining with 12 m benches and 228 mm diameter holes loaded with ANFO. Obviously with the change of explosives, powder factor values would change.

In order to obtain good fragmentation and thereby ease in loading operation, the explosive consumption in excavation is somewhat greater than in quarrying. When firing is confined to single row of blastholes in soft laminated strata, the charging ratios may be as low as 0.15-0.25 kg/m^3. In harder sedimentary strata the charging ratios generally are around 0.45 kg/m^3 while they may be about 0.6 kg/m^3 in jointed igneous rock. Even higher charging ratios may be necessary in order to obtain satisfactory results in some metamorphic rocks such as mica schist, which absorb much of the energy of the blast. Generally 1 kg of explosive will bring down about 8-12 tons of rock.

12.11 ENERGY FACTOR

Till the introduction of slurries, watergels and emulsions; the powder factor was a good indicator of the amount of energy used to break a quantity of rock. However, with slurries, watergels, and emulsions, the energy can vary greatly when small charges are used in the strength tests and also when density remains constant. Therefore, many use a different method to relate the amount of explosive energy required to fragment a given quantity of rock. This is termed as the Energy Factor. The amount of theoretical energy is used as an index of effectiveness of an explosive to break rock. Many different computer programmes are available which determine the theoretical energy yield. Thermochemical energy is expressed in terms of calories per unit volume or weight of an explosive. The Absolute Bulk Strength (ABS) of a given explosive is the amount of thermochemical heat energy expressed in terms of calories per cubic centimetre. The Absolute Weight Strength (AWS) of a given explosive is the amount of thermochemical heat energy expressed in terms of calories per gram. The Energy Factor (EF) can be defined as the amount of explosive energy kilo calories and its distribution relationship in a given quantity of rock. Therefore,

$$\text{Energy factor} = \frac{\text{kilo calories}}{\text{quantity of rock}}$$

The energy factor can be related to either cubic meters or tons. There is a difficulty, however, that different computer programs using exactly the same input and output characteristics, give significantly different strengths on the same composition. Thus one has to be careful while using these theoretical calculations given by different manufacturers.

12.12 BLAST DESIGN PROCEDURE

In designing surface blasting rounds, the following guidelines are suggested for optimum results. The proposed values will need to be modified after conducting some trials.

Variable/determining factors and remarks	Typical parameters	
	Large scale operation	Medium scale operation
1. *Draw a plan to scale*		
Showing the original rock face and the proposed new face with the desired direction or rock movement. Major geological features such as joints, cleavage, and faults should be indicated and their possible influences be considered. Include conventional rock classification information and other rock characteristics.		
2. *Bench height, H*		
Normally dictated by site parameters which include type and dimensions of loading equipment. If the height is not predetermined then it should be greater than or equal to 2.5 × burden distance. While drilling bench heights greater than 4 × *B*, care to be taken due to hole deviation.	10-15 m	6-12 m
3. *Hole diameter, D*		
Depends on drilling equipment. For optimum fragmentation use, hole diameters equal to the bench height divided by 120. The maximum hole diameter should be equal to the bench height divided by 60. When using charge diameters that are less than the hole diameter, the effect of decoupling must be taken into account. Smaller holes distribute the explosive energy better than larger holes. Cost of drilling, loading, and explosives per hole volume generally decrease with increase in hole diameter.	250-350 mm	83-200 mm
4. *Explosive type*		
ANFO: Holes must be dry or lined. Low cost. Simple to mix, safe to handle components. Safe in use. Low bulk strength. Good for low powder factor situations. May be expensive in high powder factor situations owing to high drilling costs.	Site mixed	Site mixed
Slurry/emulsion: Used in dry or wet holes. Usually mixed and pumped by contractors. Higher bulk strength and hence lower drilling costs than for ANFO, but lower weight strength and higher cost than ANFO.	Site mixed	Site mixed
Heavy ANFO: Advantages of both ANFO and emulsion. Cost advantages and can be used in wet or dry conditions. Possibility of varying strength characteristics.	Site mixed	Site mixed
Cartridged explosives: Wherever quality of ANFO is poor		

Variable/determining factors and remarks	Typical parameters	
	Large scale operation	Medium scale operation
or site mixed products are not available cartridged explosives are used. Labour intensive. Costly products. Cartridges of NG based, AN based or slurry/emulsion are used. Needed for controlled blasts.		
5. *The Burden, B*		
Based on rock characteristics and explosive to be used burden is determined by using appropriate relationship or experimentally determined from bench crater method.	5-10 m	3-7 m
$B = 0.024\,D + 0.85$ m		
It is generally kept from 20-35 × explosive diameter, D_e. Existing rock hardness, fractures, explosives used and the required fragmentation dictate the burden.		
Minimum allowable distance from the crest to front row: Determined by safety requirements. Drill dimensions and bench crest conditions determine safe distance from front row to bench crest.	3 m	2 m
Front row mean burden: Crest to front row distance + ½ bench height × cotangent face angle	4-10 m	3-7 m
Front row mean burden volume: Mean front row burden × spacing × height.		
6. *Burden/spacing ratio*		
Square pattern: Easy to layout, poor charge distribution. More difficult to delay.	1-2	
Staggered pattern: Even charge distribution, easy delaying. Good in large patterns for low powder factors.	1.1-1.4	
Wide spacing: Good for high powder factor in competent rock. Not suitable for joint orientation across the face.	1:>1 1:<8	
Delay sequence will alter drilled burden and spacing. Recommended effective ratio.	1 to 3-4	
7. *Spacing determination from burden*	6-11 m	4-7 m
$S = 0.9\,B + 0.91$ m		
8. *Stemming, T*		
Inert material placed in the hole in collar to confine gasses. Crushed rock confines explosive energy better than drill cuttings. Usually equal to 0.7 to 1.5 times the burden dimension. If the stemming is less than ≈ *B* then flyrock and premature venting may occur.	2.8-7 m	2-5 m
9. *Subdrilling, J*		
The distance drilled below grade level	1-3 m	1-2 m
$J = 0.3$ to $0.5 \times$ burden		
If 0.5 × burden still leaves an excessive toe then the burden distance should be reduced. No subdrilling is used if hole through the overburden ends at the top of coal seam.		
10. *Hole depth, L*		
Check the hole depth in relation to the rock burden and bench height.	12-17 m	8.12 m
$L = H + J$		
11. *Broken volume/hole*		
Burden × spacing × bench height	500-2000 m^3	

Variable/determining factors and remarks	Typical parameters	
	Large scale operation	Medium scale operation
12. *Hole inclination* Blast hole should be in conformity with the face. Vertical to 20° uniform burden and need less subdrilling.		
13. *Estimation of powder factor* Controlled by rock conditions. May be determined by empirical relationship. Design powder factor should be expressed in terms of weight or volume of standard explosive.	0.25-1.2 kg/m^3 0.1-0.4 kg/t	
14. *Actual powder factor* Measured for whole blast or total production – less than design powder factor due to overbreak and free digging.		
15. *Explosive charge/hole, Q* Density of explosive × available hole volume (cross sectional area of hole) × (bench height + subdrill – stemming).	350-700 kg	
16. *Charge distribution* Bottom charge is the amount of explosive required to fragment the toe region.	20-40%	
Column charge is the amount of explosive required to fragment the remaining rock associated with the hole.	80-60%	
17. *Drill pattern* Select a pattern of drill holes that is estimated to be able to provide the desired rock movement. Take into account any possible environmental problems that may limit the maximum permissible charge weight or the number of holes fired per delay interval. Mark the sequence of charge initiation on the plan, select an appropriate number of rows and the number of holes per row.		
18. *Delay interval* 3 to 6 ms/m of effective burden. Place delays 1 m from delayed holes.	17, 25, 35, 45 ms	
19. *Delay sequence* Delay to free face. Delays must be used to avoid choked blast. Delays must be used to modify spacing: burden ratios. Delays can be used to reduce maximum instantaneous charge level.		
20. *Controlled blasting* Apply necessary controlled blasting techniques. Back row to final wall distance should be equal to anticipated back break. Diggable back-break approximately equal to burden. Varies with rock conditions, explosive strength and powder factor of back row.		
21. *Maximum instantaneous charge* Depends upon charge/delay allowed according to risk of damage due to ground vibrations and distance to surrounding structure.	450-4500 kg	
22. *Number of holes per row* Depends on quantity of rock to be broken, length of face and spacing.		
23. *Number of rows* Depends on the delay time, type of initiation system and rock structure.		

CHAPTER 13

Drift and tunnel blasting

13.1 INTRODUCTION

Tunnelling is a highly specialised field of endeavour within the construction and mining industry. The rock removed from the tunnel excavation is usually waste material in contrast with ore recovered during mining. The emphasis is on advancing the face with greatest possible speed compatible with safety and efficiency. At times cautious blasting is carried out to prevent overbreak and/or keep ground vibrations low. Drifts or headings are used to drive access tunnels, levels, drives or crosscuts into and through the ore body. These headings vary in shape and cross-sectional area depending on their ultimate use such as transportation of ore, ventilation passage, or manways. These are generally smaller and are blasted somewhat differently from the larger vehicular tunnels used for highways, trains or water diversion. Smaller tunnels with cross-sectional area less than 25 m^2 are driven with the same drilling and blasting techniques in principle as used in mining.

A far greater explosive consumption per cubic meter of rock blasted is used in tunnelling/drifting than most other work. When tunnels/drifts are being blasted, the only free surface is the tunnel face and this means very constricted rounds. The smaller the area is, the more constricted the rock is and this means that the specific charge increases as the area decreases. The effective and economic use of explosive, therefore, is a major factor in the cost. The size of tunnel, the type of rock and the method of working all influence the cost, but without a carefully planned drilling and charging pattern, cost optimisation becomes difficult, as all other factors, such as the capacity of the loader required, the number and type of drilling machines and the number of men required at the tunnel face, are a function of advance.

In this chapter, blasting techniques for various methods of tunnel driving are given. Details are provided for the full face round design and for top heading and benching rounds.

13.2 DRIVING METHODS

There are several methods which can be used for driving tunnels. For tunnels of small cross-section the full face method is adopted (Fig 13.1), in which blasting round is designed to pull the full cross-sectional area of the face in one blast. This method is adopted even up to a height of 10 m, unless extremely bad ground requires

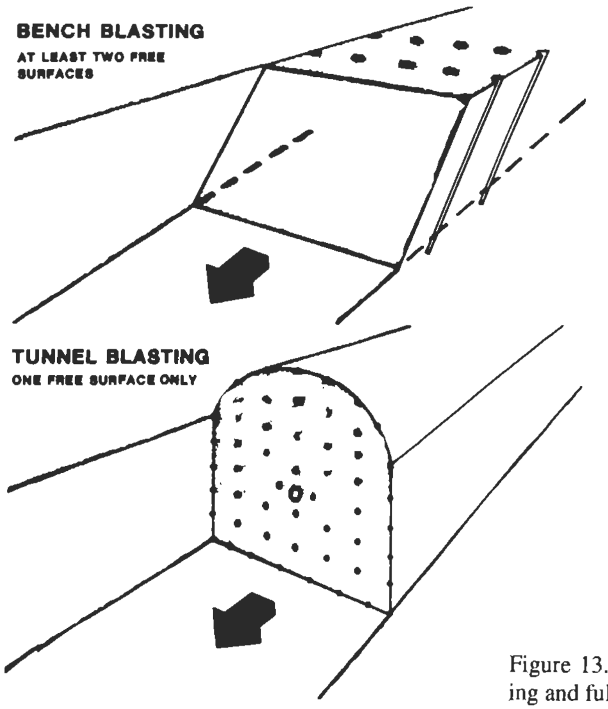


Figure 13.1 Comparison between bench blasting and full face tunnelling method.

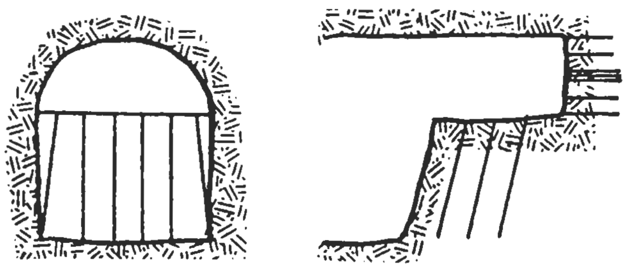


Figure 13.2 Top heading and benching for tunnelling.

short rounds. When rock conditions are good, it may even be used for large tunnels (face area of 80-100 m^2). For other conditions, an initial full face top heading may be enlarged to final shape and size using split section driving technique. The top heading and bench arrangement is perhaps the most common (Fig. 13.2). Generally, the excavation of top heading is completed prior to the driving of the bench. It consists of driving a heading at the top of the tunnel. The lower portion is then removed in one or more benches with either vertical or horizontal holes. In horizontal drilling (Fig. 13.3a), an entire cycle of drilling, charging, blasting, loading, etc. must be completed before the drilling of the next round can begin. In case of vertical drilling of the benches (Fig. 13.3b) the drilling and loading operations can be carried out independently of each other.

The top heading may be divided into a pilot heading plus one or more slashes. In

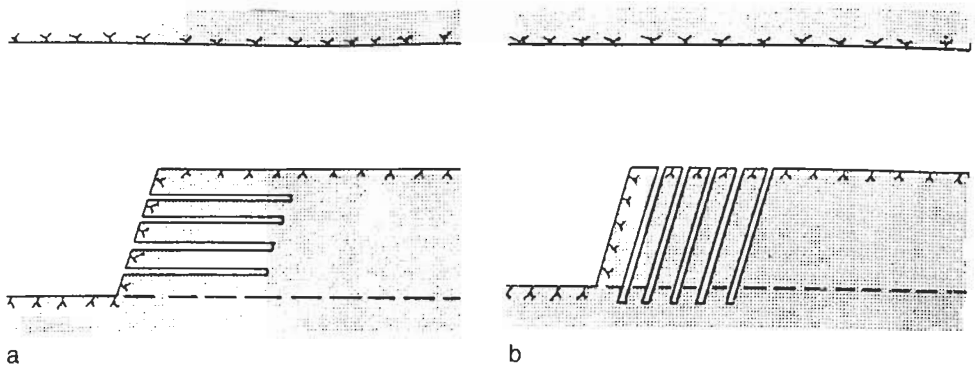


Figure 13.3 a) Horizontal drilling for the bench and b) vertical drilling for the bench.

this way reinforcement can be installed in the roof of the pilot tunnel before widening to full span.

Combining pilot headings, slashes and benching, tunnels of any shape and size can be excavated. Where possible, it is generally less expensive to use a full face method.

13.3 HEADING ROUND

Blastholes drilled in the face of the tunnel are collectively known as the round. They must be drilled and positioned efficiently (Fig. 13.4). The round is arranged in a pattern. The initial step in advancing the headings/tunnels is making an opening into the solid face to produce a void and to create a face or plane of relief, as deep as it is practical. This plane of relief, or void, is called the 'cut'. The advance that can be achieved by any round is a function of the advance obtained by the cut hole(s), all other holes in the round being blasted to this second free face formed initially in the cut area. It is a requisite, therefore, that the cut should break clean to as great a depth as possible. An accurately drilled and charged cut can mean the difference between breaking a full round or obtaining only a small part of the planned advance.

The lineal advance of the face for a round is the pull, and the ratio of pull to depth of holes is a measure of efficiency. An 80% pull means that 20% of all drilling in the tunnel has been unproductive.

The round must be as long as is practicable, have high ratio of pull to depth of holes drilled, have no unnecessary holes, have a low explosive requirement, and give a profile without excessive over break or under break. Underbreak requires additional man-hours for scaling whereas overbreak implies wasted labour and explosive, additional rock to be loaded and additional concrete requirements.

A round consists of several classes of holes (Fig. 13.5). The cut is formed of those holes which are grouped in such a manner as to produce a longitudinal free face being blasted and the remaining holes called stoping holes are those which extend the cut to nearly the full area of the face; and the wall, roof and floor holes are the peripheral holes.

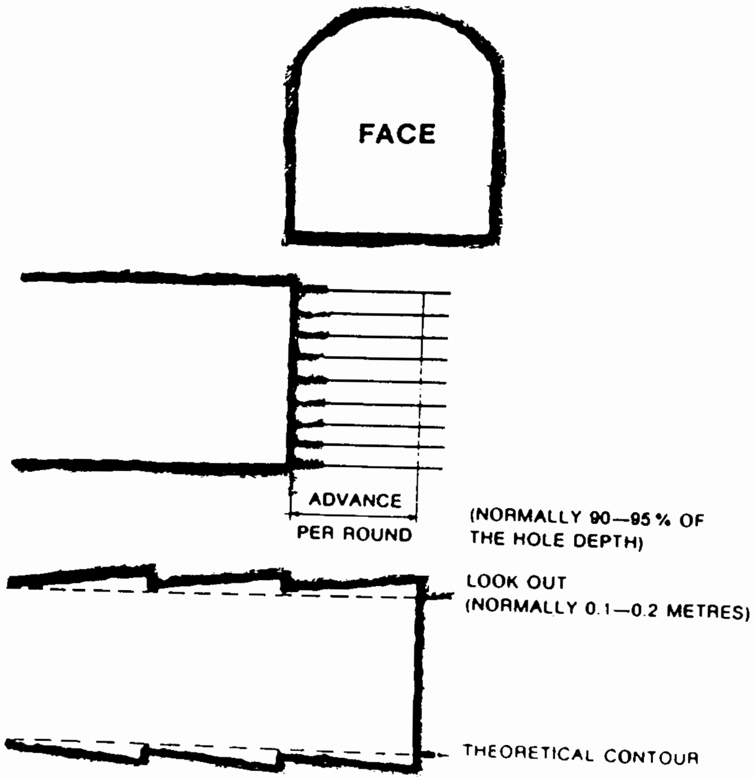


Figure 13.4 Nomenclature used in tunnelling.

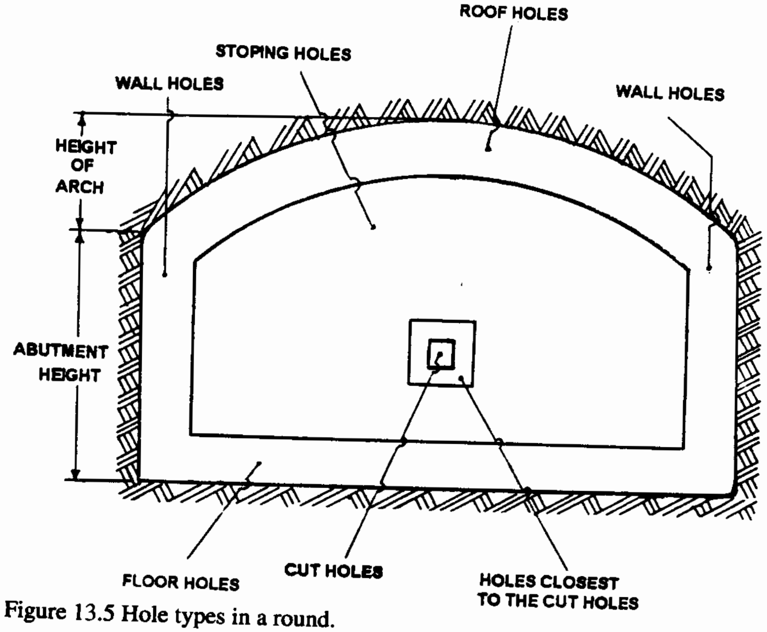


Figure 13.5 Hole types in a round.

13.4 ROUND DESIGN

The design of heading round depends on:

1. The desired utilisation of the heading;
2. The cross-sectional area;
3. The rock and drilling parameters;
4. The skill of miners.

13.4.1 *Utilisation and cross-sectional area*

Where the headings are being driven in waste rock, they are usually driven as small as possible while remaining compatible with haulage and ventilation requirements. If the heading is in usable ore, it may be expanded in cross-sectional area to increase production. Also, when the heading is driven in rock to be expanded later, then larger sizes are used.

13.4.2 *Rock*

Obviously, the type of rock being excavated is a major factor. To a large extent the type of rock will determine the shape of the heading cross-section. If the back is solid and can be maintained in a safe condition without an excessive amount of external support, the design may be with a flat back (a rectangular cross-section). If the flat back is not stable, an arched back horseshoe cross-section may be necessary.

Bedding planes may be a major factor in the round design. Frequently, the bedding planes can be used as a plane of relief.

13.4.3 *Drill hole diameter*

The diameter of the boreholes should be the largest size compatible with drilling efficiency and acceptable fragmentation. The diameter of the hole determines how much explosive can be loaded into the borehole. In most formations the critical problem is pulling the full depth of the borehole. Therefore, the larger the diameter of the borehole, the more explosive can be loaded in the back of the hole where they are most needed to remove the back of the hole leaving a smooth face. With larger diameter holes the number of holes can be reduced. Depending on the formation, reducing the number of holes may reduce the overall drilling time. However, in some conditions because of the reduced penetration rate of the larger diameter, the overall drilling time may not be reduced. The decision on the diameter of the hole must be evaluated in relation to the formation.

The diameter of the borehole must allow for explosives distribution that will produce the necessary fragmentation. In most formations this is not of major concern and can be controlled by the type of explosives being used. Holes are generally drilled with a diameter ranging from 26-50 mm. In Sweden, generally holes of 32-38 mm in diameter are used whereas in the USA tunnelling is carried out mainly with the use of holes in the diameter range of 42-46 mm.

13.4.4 *Hole depth*

The hole depth used depends on the type of rock to be blasted, in a rock material of low strength and which is readily loosened, shallower boreholes and smaller explosive charges are generally used. The depth of the holes is also controlled by the time available for completing the driving cycle, in particular for drilling the rock and loading it. The round pulled by holes is generally shorter by about 300-400 mm than the hole depth.

However, attempts have continued to drill deeper rounds as the advance per round depend on the depth of holes. Advance per round varies from about 0.5 m in very broken ground that requires immediate support, to as much as 3 m in massive and self supporting rock in a large diameter excavation. The tunnel advance as a percentage of depth drilled varies from 50-90%. In Sweden practical advance per round is normally 90-95% of the drilled depth (Holmberg 1982). There is no point in drilling deeper than can be pulled. However, attempts are continuing so that deeper holes are drilled and higher advance is obtained from a blast. If the average advance of a round could be increased substantially, fewer drilling rigs would be needed, lead times to production could be cut and blasting costs would decrease.

Ouchterlony (1992) provides details of research to drill 4-5 m deep rounds with 48 mm diameter blastholes or larger, using parallel hole cuts with one or two large empty holes. Niklasson et al. (1988) and Niklasson and Keisu (1993) indicate that opening cuts with only 64 mm diameter holes work well in ore and rock for rounds with 4.0-4.4 m drilled depths, as well as standard cuts with large diameter empty centre holes. Long rounds, with holes drilled 7.5 m deep, are also being used and with 300 mm empty hole, advance has increased to 100% (Fjellborg & Olsson, 1996).

13.4.5 *Number of holes*

The number of holes required in the face depends on the area of the face, the shape of the drift, the rock type, the required fragmentation and the size of hole. The number of holes needed per unit face area is greater in the smaller drifts. The highest concentration of holes is in the cut and in a small drift the area of the cut forms a high percentage of the total; the other factor involved is the number of perimeter holes which increases only as the square root of the area of the face. Figure 13.6 gives relationship between the number of holes and cross-sectional face area (Gustafsson 1981). Guidance (Whittaker & Frith 1990) based on US experience is as below:

Tunnel cross-sectional area (m^2)	Required number of holes per round	
	Weak rock	Strong rock
10	23-27	35-50
25	45-50	60-70
50	75-85	95-110

Weak represents soft or highly fractured rock types; strong represents hard or massive rock types.

Pokrovosky (1980) indicates that total number of holes N, in a round are given as:

$$N = N_b + N_1 = \frac{qS}{\tau} + \left\{\frac{c\sqrt{S} - B}{b} + 1\right\}\left\{1 - \frac{\tau_0}{\tau}\right\}$$

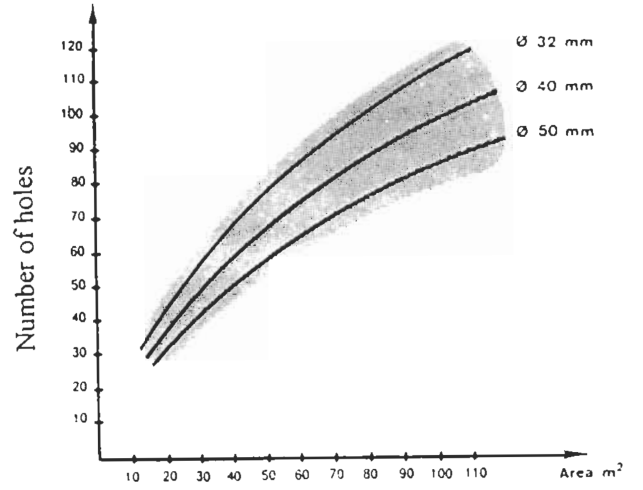


Figure 13.6 Number of holes needed in a tunnel face (after Gustafsson, 1981).

where N_b = number of cut and stoping holes, N_1 = number of perimeter holes, c = coefficient depending on shape of working cross-section. For square cross-sectio workings, c = 4, for trapezoidal cross-section, c = 4.2, and for arch shaped cross section, c = 3.86, S = the working cross-section of the opening m^2, b = packing facto as charged in the hole, for NG based explosives it is equal to 1.2 and for AN based is 1.0, B = the width of working in m, τ = the explosive consumption per m lengt which is a function of explosive density for cut, stoping and floor holes. For exampl explosive consumption with ANFO and NG based cartridge in 32 mm holes con sumption varies between 0.62-0.75 and 0.8-1.0 kg per m, τ_0 = the explosive con sumption in perimeter holes. Explosive packing density is lesser in perimeter hole than in other holes, and q = explosive consumption in kg/m^3.

13.4.6 *Drill deviation*

Blasting results are deleteriously affected in any operation where drilling accuracy i lacking. In the cut, drill hole deviations is one factor that limits the advance of tunne round (Langefors & Kihlstrom 1978). Deviations from the planned spacing and bu den of drill holes may occur due to inaccurate positioning at the hole collar, errors i alignment and deviations along the hole length due to the effects of gravity and roc nonhomogeneity.

Errors in hole collaring can be eliminated by adopting a system of marking hol positions on the face of the tunnel. Such errors, where they do occur, are a consta factor, and unlike the other sources of drill deviations, do not increase with hol depth.

Errors due to faulty alignment and rock nonhomogeneity increase with the dept of the hole being drilled. Errors in alignment are mainly a problem of supervisio and experience. Nonhomogeneity of the rock and the drilling system are the remain ing factors which affect drill scatter, and to large extent these are uncontrollable, an allowance must be made for them in deciding the hole spacing and burden. The dri deviation is mainly assessed on the basis of past experience, and may have a bearin on the drilling system to be employed. A higher accuracy can be obtained with air-leg mounted machines than by hand held drills, and an even higher accuracy can be achieved by drill rigs.

The effect of drill scatter is to reduce the spacing and burden at the tunnel face, so that including the deviation inherent in drilling the hole, it will bottom in a position such that its burden and spacing is not greater than that which the hole can break.

As a general rule, for hand held machines a deviation of about 6.5 mm/m of hole depth should be allowed for in planning hole spacing and burden, and about 2.5 mm for 1.8 m of hole depth with air leg mounted machines. The above deviation values can be reduced somewhat if short drill steels are used for collaring the holes. This reduces the amount of sway which is inevitable when collaring with long steels. When airlegs are used, careful control of feed pressure is necessary.

Inability to collar a blasthole at the desired location (perhaps as a result of protrusions and indentations in the face) can lead to drilling errors which are relatively much greater than those commonly experienced in bench blasting operations. The relatively pronounced effect of face irregularities has promoted some operators to use templates for drilling the cut (these were first introduced for drilling the coromant cut). The (very limited) use of templates would appear to be completely incommensurate with their value. Templates have the greatest potential benefits, of course, where experienced drillers are not available.

13.5 THE CUT

There are three basic types of cuts.

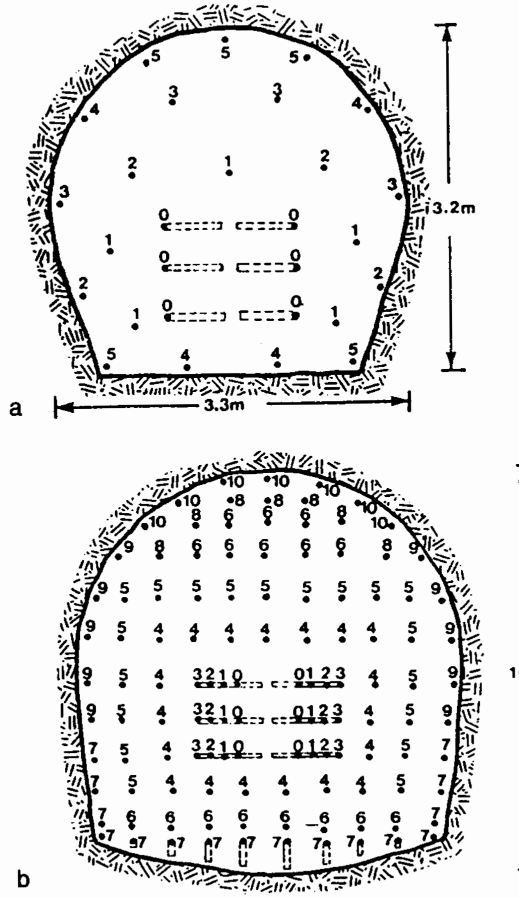
1. Parallel hole,
2. Angle, and
3. Machine.

Each type of cut has many variations of designs to make them adaptable to a particular formation. Various rounds derive their name from the type of cut used. The primary function of the cut remains the same regardless of the type of cut or its variations. The cut must be broken and completely removed to create a void for expansion and relief. The remaining holes in the round are timed to fire in sequence breaking to nearby created free face and allow most of the rock in a section to move forward into the open tunnel excavation. The simplest way to obtain required direction of throw is to drill cut holes at an angle to the face. These cuts are known as angled cuts. However, if a group of closely spaced parallel holes are drilled (of which some are uncharged and, preferably, of larger diameter), the rock within the area bordered by these holes, when blasted, will be able to move laterally to a limited extent in place of the rock removed as spoil in drilling, while expansion of gases at the back will expel a proportion from the face. Cuts based on this principle are known as parallel hole cuts. Where machine can be used to cut a cavity, remaining holes are blasted to this cavity and therefore are known as machine cut faces. Such methods are used in coal mining and are described in Chapter 14.

13.5.1 *Angle cut*

An angle cut is a group of holes drilled at various angles inclined to the free face to provide as much freedom of movement for the rock as possible. They require fewer holes per round and less explosive per meter of advance than parallel hole cuts. However, their use is restricted by the width of the heading. The heading must be wide enough to allow the drills to be placed at relatively small angles to the face. They are generally more difficult to drill and require more experienced miners. Usually the angle cut will move the rock further down the heading and produce coarser fragmentation out of the cut area. Longer drill steel is required in the cut holes to achieve the same depth as the surrounding boreholes in the round. The various types of cuts being used are:

V or Wedge cut. The V or Wedge cut (Fig. 13.7) in its simplest form is one in which two rows of holes are angled in to a common line. This single V-cut is symmetrical; the drilling can consequently be evenly distributed among the workers. It needs few blastholes and is satisfactory in easily blasted rocks. If rocks contain fissures, the



Numbers = half sec. delay period.

Type of rock	Massive granite
Face area	8·3 m^2
No. holes drilled	28
Rock broken	13·9 m^3
Rock drilled	3·8 m/m^3
Depth of pull	1·7 m
Explosive type	HS gelatine 25 x 200 mm
Explosive quantity	21·3 kg
Rock: explosive	0·65 m^3/kg
Explosive: rock	1·52 kg/m^3

Numbers = 0.5 sec. delay periods

Type of rock	Granite
Face area	76·5 m^2
No. holes drilled	118
Rock broken	302·7 m^3
Rock drilled	1·5 m/m^3
Depth of pull	4 m
Explosive type	HS gelatine 32 x 200 mm
Explosive quantity	272·1 kg
Rock: explosive	1·11 m^3/kg
Explosive: rock	0·88 kg/m^3

HS = High strength

Figure 13.7 a) V or Wedge cut and b) series of wedges in cut.

cuts are made accordingly (e.g. horizontal cut when horizontal fissures occur). The disadvantage is that only a comparatively small advance per round can be achieved.

In hard rocks, more complex forms are necessary, in which a series of wedges with an increasingly obtuse apical angle are arranged one within the other (horizontal V-cut) (Fig. 13.7) or better still, the more sophisticated version in which the cut is not symmetrical about the centre line, the holes on each side alternating (shearing V-cut). Vertical V-cut can also be used and is particularly appropriate for those tunnels which have a greater width than height, and in which jumbo mountings are used. The upper row of holes, directed downwards, can be drilled by a line of men on one level of the platform, and the lower holes from the invert.

These cuts are normally designed so that the holes on each level originate from one point on each side (i.e. the drill position) and no more than one drill at each side can be used to drill the cut.

Pyramid cut. In this cut (Fig. 13.8) 4 or more holes are angled so as to meet at common point just to the rear of the general round and they are fired simultaneously. Concentration of explosive is good, and the cut is normally expelled well provided that the angle forming apex of the pyramid is not too acute. These cuts are suitable for hard rock but produce a long throw; they are regular in form and are therefore easy to drill, but have down holes which may present difficulty with feedleg-mounted drilling. They are sensitive to hole deviation and are not satisfactory for long rounds. The variants include forms based on truncated pyramids and cones.

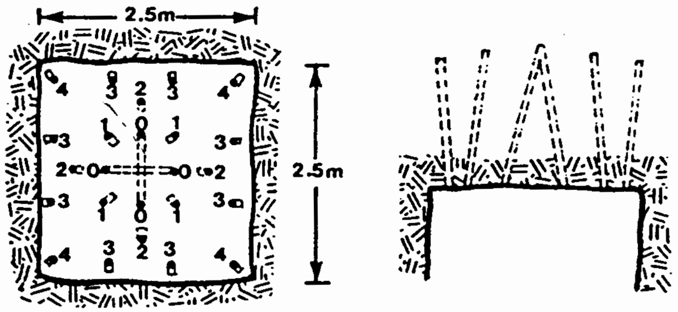


Figure 13.8 Pyramid cut.

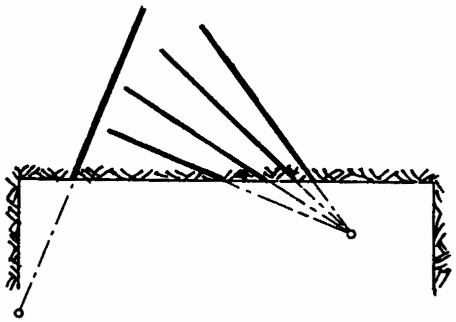


Figure 13.9 Horizontal fan cut.

Fan cut. Horizontal fan cut is shown in Figure 13.9. This cut typically consists of 2 or 3 horizontal rows of holes. In fan cut the first hole is drilled at a steep angle to the face and fired first breaking to the free face. The next holes, sequenced to fire into this initial cut are drilled at increasingly steep angles and increasingly greater depths. It is useful provided that the tunnel width permits it to be employed and the rock is easily broken. This cut is efficient in that the burden on the initial holes is evenly balanced. From the drilling point of view, the pattern is not symmetrical so that it is not always practicable to allocate the meterage evenly. Such an uneven distribution adds to the time required for total meterage. In plan the fan cut in a wide tunnel consists, in reality, of two overlapping fans, and this leaves a small central wedge of rock at the back of the round which cannot efficiently be blasted. To overcome this, the easers are usually shortened slightly thus reducing the pull.

Vertical fan cuts are also practicable and are useful in much narrower drifts. Holes are inclined in an upward direction when feedleg drills are used or downwards if the heading is very small and the holes are drilled with sinker drills.

Draw or hammer cut. The draw or hammer cut (Fig. 13.10) is a modified V-cut. It is

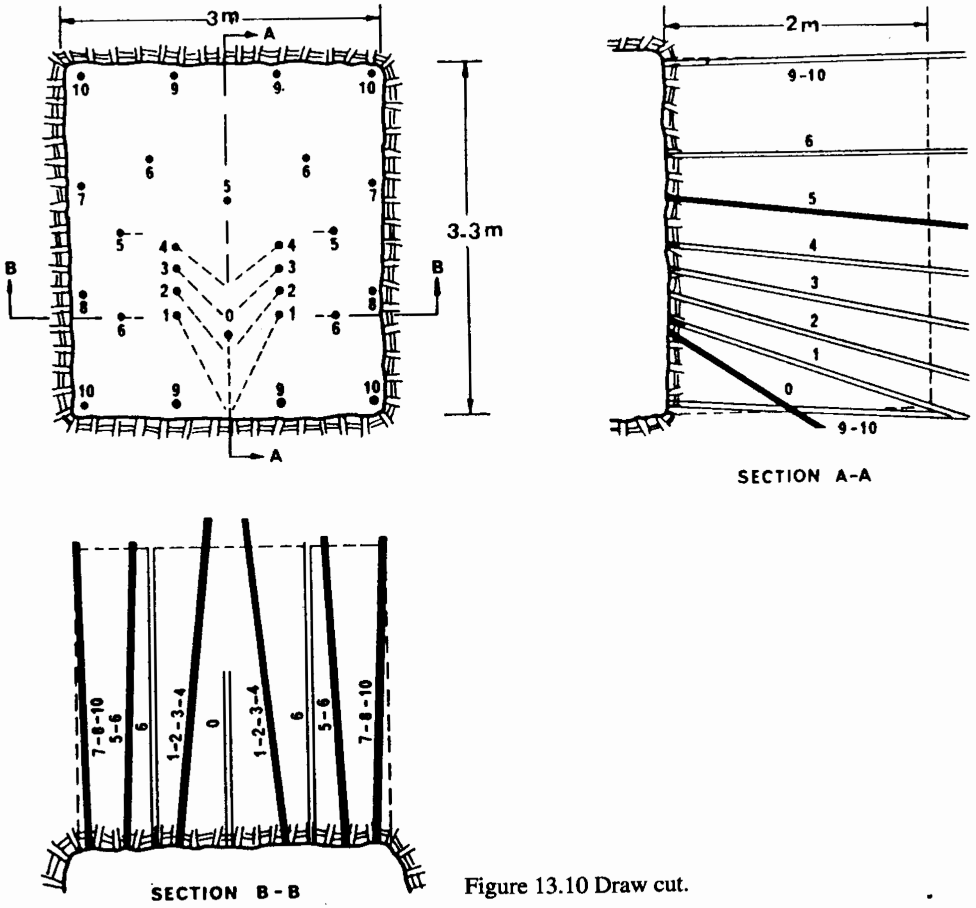


Figure 13.10 Draw cut.

generally located away from the centre of the face and the holes are drilled so that they do not meet. This cut requires careful drilling; but when properly drilled, loaded, and delayed, can perform very well, especially when a crack or parting is present at the bottom or side of wedge. This method is frequently used to drive in small headings where the jackleg drill or some other unmounted drill does not have enough room to drill a cut in the centre of the face.

General remarks on angled cuts. In modern tunnelling the emphasis is on advance, and to achieve this longer pulls are required. To keep the drilling time to a minimum there is also a tendency to increase the number of machines working at the face. Angle hole cuts as a basis for rounds have limitation particularly in tunnels of small cross-sectional area. The advance that can be achieved is limited due to the need to angle the holes, and as the cut holes tend to be concentrated on either side of the tunnel there is inevitably interference between machines due to the various angles required for the cut, easers and other holes.

Unless precautions are taken, the material from the cut can also be blown from the face as a single or several large fragments of rock which may require secondary breakage. If this is a problem it can generally be overcome by drilling one or two short 'burster' holes near the centre of the cut and firing them with the cut. In addition, it may be necessary to prime the cut holes with spaced delay detonators. If the latter method is adopted the holes should not bottom closer together about 15 mm to reduce the risk of flashover.

The collar positions must be marked on the face and the holes drilled each at its own set of angle on its plane. It is not too difficult to drill with mounted drifters, but great skill is required when drilled freehand. Even then considerable deviation occurs.

In small tunnels, the clearance required for drill steels when angled may make angled cuts inconvenient. If feedleg drills are to be used there should be a sufficient distance between the side wall and the hole collars for the drill length, the steel change, and a proportion of the feedleg length. In small to medium size tunnels, cut may have to be drilled offset. Another point with feedleg drilling is that it is difficult to drill downward holes and this should be kept in mind while choosing cut. Often drilling of the angled cuts interferes with the drilling of the other holes in the round and, as these form the major portion of the total meterage the time required is likely to be longer than with parallel hole cuts. Also, fewer drillers can be accommodated on a face of a given size if angled cuts are used.

The advance per round which can be obtained with angled cuts is approximately 60% of the tunnel width (or height in case a vertically angled cut is used) although in practice the results may be something under this (say 45-55%) due to inaccurate drilling or deviation if the tunnel is large and the holes long, or an unsuitable choice of cut. Once a pattern has been established and is working well, it is not possible to increase the length without radical changes in angles, planes and locations.

On the credit side angled cuts tend, on the whole, to require fewer holes and less explosive than many parallel hole cuts, and from a blasting point of view, must be regarded as more efficient. Also they can be used in rock of any type.

13.5.2 *Parallel hole cuts*

Unlike the angled cuts which are designed to break out a wedge or pyramid of rock, the parallel hole cut and its variants are designed to break out a roughly cylindrical or rectangular area of rock of constant cross-section on a line parallel to the advance of the tunnel and perpendicular to the face (Fig. 13.12).

Parallel hole cuts are of two main types.

1. Large hole cuts: incorporate one or more large diameter holes, these are used as relief holes and generally contain no explosive charge;

2. Burn cuts: in which all the holes are of normal size, of which one or more are left uncharged.

In some parallel hole rounds all the holes are charged and these are known as no-cut rounds.

Burn cuts (which do not have large diameter holes) have one great advantage over other parallel hole cuts of neither requiring special heavy drills nor of delaying the completion of drilling cycle. However, the ratio of pull to blasthole depth cannot be as high as with large hole parallel cuts.

A greater number of holes is required and the weight of the explosive charge is relatively high. As a matter of fact the rock in the vicinity of the cut is literally burnt. Both loaded and unloaded holes must be perfectly parallel otherwise blown-out shots or break through at the position of the charges may occur. The cuts which make use of large diameter holes do not require the high drilling accuracy. The large diameter holes used vary in size, but holes up to 255 mm diameter have been used. Evidently, with holes of this size they practically form the cut themselves. For large diameter holes special drilling equipment is required and the speed of drilling is slow. For holes of about 75 mm diameter it is possible to drill a small diameter pilot hole and ream this hole with another bit mounted on the same steel. Large diameter hole/holes act as effective second free face and less explosive is needed.

The burn cut round is used almost exclusively in small cross-sectional area tunnels because all the holes are drilled parallel to the centre line of the drift and permit 'pulling' deeper rounds.

It is most important that all holes in a burn cut be drilled exactly parallel and at the proper distance from one another (Hagan, 1979). The burn cut is usually located close to the centre of the face. However, to avoid drilling into the most shattered area of the face, it is recommended that the location of the burn be relocated on alternate blasts. The burn area has the highest potential for undetonated explosives and is the most difficult to identify.

Therefore, alternating the location of the burn for each blast is considered a good safety practice.

The diameter of the holes used in the burn cut varies from 32-70 mm depending on the individual operations. The diameter of the holes should be the largest compatible with the drilling equipment for maximum drill efficiency and explosive distribution necessary to achieve the desired fragmentation.

The design of the burn will depend on the rock being blasted, the type of explosive being used and the diameter of the holes. All rocks when broken have an expansion factor that varies with the particle size of the broken material. The burden design must allow room for this expansion factor. Usually, a minimum of 15% of the area within the burn cut that will be influenced by the first holes to fire is essential to successful shattering and clearing of the burn. The percentage will vary with the rock formation. However, the more room that is provided for expansion, the more successful the round will be in pulling the full depth of the drilled boreholes. The necessity for expansion room cannot be over emphasised as it is the essential factor in successfully blasting burn cuts. Time spent in drilling additional holes for expansion of the broken rock will be more than compensated by the increased depth of round that can be pulled.

Where large hole cuts are used, the number of expansion holes required is reduced.

To shatter a hole its entire length, the explosive column must extend almost to the collar of the hole to throw the shattered material creating the desired void. The back of the hole must be loaded heavier than the front. It is usually necessary to tamp explosive in the back of the burn cut holes to achieve maximum borehole density near the bottom of the hole and to reduce the column load near the collar of the hole. The primer cartridge should not be tamped. It is desirable to have half of the total load-per-hole concentrated in the back one-third of the hole. To achieve the void desired by the cut, it is necessary to shatter the burn area throughout the complete length of the hole and to displace the broken material away from the face.

If a 40 mm diameter burn cut, 3 m deep hole, is to be loaded with 32 mm diameter watergel explosive (density = 1.1 g/cc), the back 1 m of the hole can be loaded with 1.25 kg of explosive. The next 1.7 m can be column loaded with 1.25 kg, which leave a collar of 0.3 m.

It is possible to overload a burn hole if the explosive column load is not reduced in the front of the hole. Excessive shattering of the rock will result in the compaction of the material to the extent that the load in the back of the hole cannot displace the broken material. This is commonly referred to as 'freezing the burn'. When a burn partially freezes, the round can only break to the depth to which the burn cleaned itself.

The type of burn used is determined by the miner's experience and the type of the ground. Generally, hard, solid rock responds well to a smaller burn 150-200 mm wide. Softer laminated rock is better pulled by a burn 250-350 mm wide. In many operations 150-200 mm wooden spacers between the cartridges are loaded in the burn to lighten the explosives load. In some areas this is still done to reduce the tendency of the burn to freeze from the extremely high powder factor in the burn area. The most expedient method, however, is to switch to a lower density explosive, spread the burn holes, use more or larger relief holes, or use a different delay system. Testing of several burn varieties will establish which is best for existing conditions.

Typical burn cuts are shown in Figure 13.11 which represent some of the many burn cuts used today in underground mining.

When these burn cuts are supplemented by helper or reliever holes (next holes to fire) that are accurately located near then they will succeed in most types of ore and rock. If these holes are not satisfactorily cleaning out the burn, angled reliever holes or raker holes often correct the problem. Angle holes used in conjunction with burn cuts usually provide the most successful drift or heading rounds.

Parallel hole cuts with one or more large diameter, unloaded relief holes are being

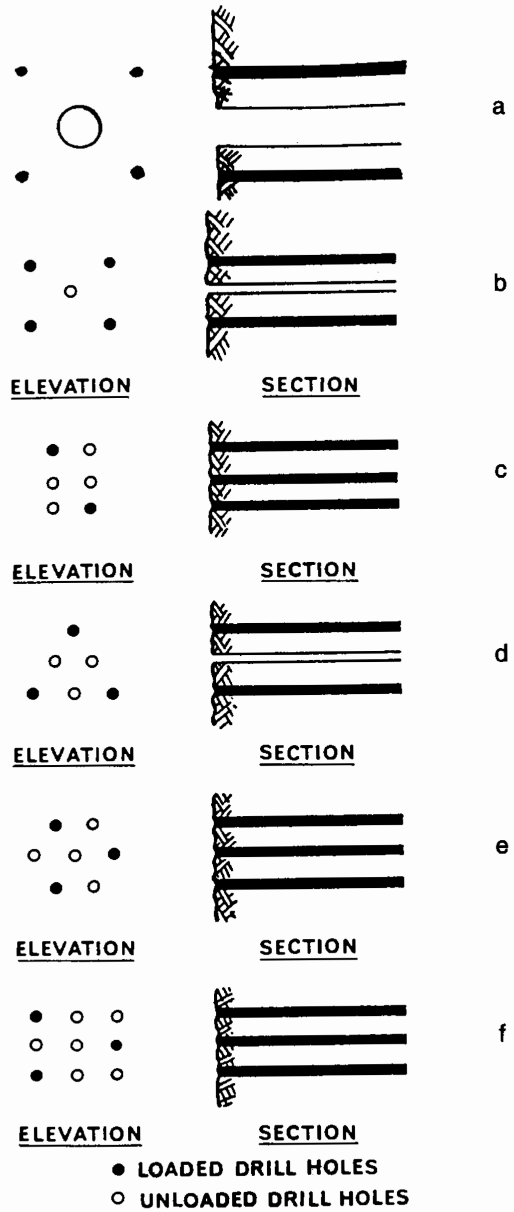


Figure 13.11 a) Large diameter empty hole parallel cut and b to d small diameter burn cuts.

widely used in drift or heading rounds. These cuts are associated with deep advance-per-round, and freezing is minimised because of the extra relief provided.

Failure of burn cut rounds

Sometimes a dependable burn suddenly fails to pull depth and begins to break short rounds. The resulting lost meterage is not only extremely wasteful, but it also slows down the progress of the entire operation. Immediate effort to identify the change or changes causing the trouble is necessary to resume satisfactory results.

Change of rock. Type, hardness, fracture, and orientation can render a 'working' burn ineffective.

Change in drilling. Miners occasionally cut down excessively on the number of holes, or a new driller or the crew-drills in different pattern.

Change of explosives. A different grade of explosive with a different density, diameter, or tampability can result in different loading of the burn and possible poor result.

Identifying the problem

Assuming that a change in explosives has not occurred, the following should be considered.

Symptom	Indicated cause
Trouble on one shift	Change in drilling
Trouble on all shifts	Rock change
Burn pulls full depth, but the remainder of round bootlegs.	Relievers too far from burn or improperly delayed.
Burn 'Freezes'	Rock may have become softer. Widen the burn spacing. Try angled relievers or raker holes. Rock may have become harder. Reduce burn spacing.

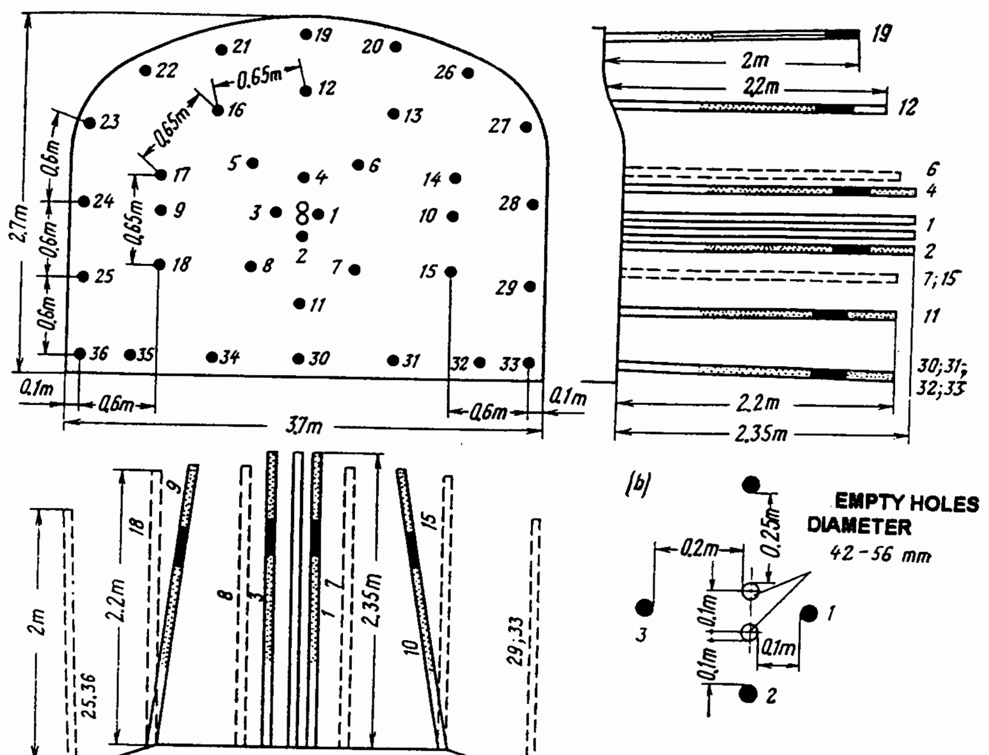


Figure 13.12 Double spiral cut.

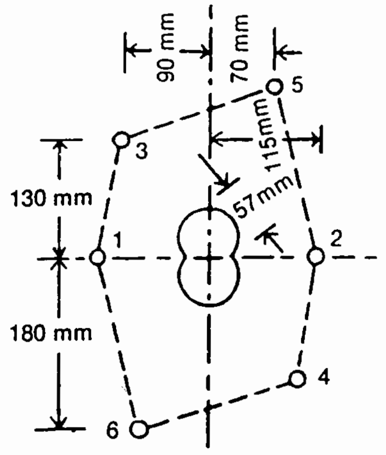


Figure 13.13 Coromont cut.

A new burn cut may be needed when short rounds begin to occur. A few shifts devoted to drilling or blasting to only modify burns are the most efficient way to determine the burn that will give the best results.

Spiral cut. A spiral hole pattern gives the widest spacing. When high advances are required, however, a double spiral cut (Fig. 13.12) is selected. In the double spiral cut, opposite holes are fired successively. This gives the best cleaning of the opening. Also safety increases, since one section of double spiral can give breakage irrespective of the other.

Coromant cut. Can also easily be drilled with hand-held equipment. It is in principle, a double spiral cut (Fig. 13.13). Two 57 mm holes drilled together to one common figure of 8-shaped hole are used as an empty hole. Approximately the same advance is obtained with this cut as with a double spiral with a 75 mm empty hole. A ready-made template is used to drill the holes in the cut. In this a 57 mm hole is drilled in the centre of the cut position. On completion a guide-tube having an invaginated groove is inserted, and a further hole drilled alongside. The bit is permitted to overlap the first hole but cannot break into it fully because of the guide tube. In the conjoined holes, which resemble the numeral 8, a template is inserted giving the location of the nearby cut holes, and these are completed.

13.6 CALCULATION FOR HOLES

In tunnel blasting principle is to create an opening using a cut and then stoping is carried out towards the opening. When calculating drilling pattern and charges, it is suitable to calculate in two steps, the cut and remaining part of the round. Stoping can be compared with bench blasting but requires considerably higher specific charges. The stoping holes can be up, down or side ways. Langefors and Kihlstrom (1978), Pokrovsky (1980), Gustafsson (1981) and Holmberg (1982) give calculations for the cut and other holes. The following is based on their work:

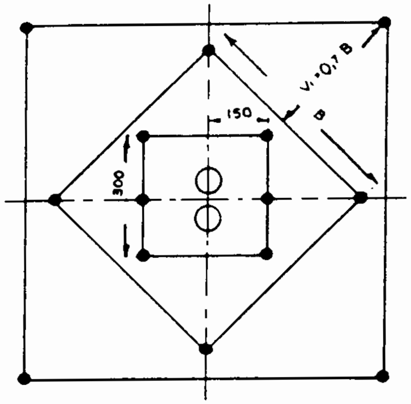


Figure 13.14 Burden for cut holes related to empty hole in parallel cuts.

13.6.1 *Calculations for parallel hole cut*

For the holes closest to the empty hole, the following relationship can normally be applied when calculating the burden,

$$V = 0.7\,d$$

where d is the diameter of large empty hole. In case where two large holes are used the formula is changed to:

$$V = 0.7 \times 2 \times d$$

Depending on the deviation, the maximum burden V_1 for the spreader holes in the cut must be less than the maximum burden ($V = 1.7\,d$). It is recommended that $V_1 = 1.5\,d$.

If holes in the first quadrangle has been drilled without deviation, then the geometry of the firing would be as shown in Figure 13.14. The side length of the quadrangle would be equal to B.

$$B = \sqrt{2V_1}$$

Increasing quadrangles are obtained till the burden for each quadrangle reaches $V = 0.7\,B$. Gustafsson (1981) suggests stoppage of quadrangles when the area of cut becomes 2 m^2.

When charging the holes closest to the empty large holes, it is important to select a charge concentration which is well balanced. In case of a too weak charge there is a complete failure of the round, since the cut holes will remain unblasted. On the other hand if the charge is too strong, a similar result is obtained due to burning of cut holes. Breakage conditions differ very much depending upon explosive type, structure of rock, and the distance between the charged hole and the empty hole. As a guideline, if the hole diameters are 32, 35, 38, 45, 48 and 51, charge concentration for NG-based explosives with strength 80% is correspondingly 0.25, 0.30, 0.36, 0.45, 0.55 kg/m.

13.6.2 *Calculation of holes for V-cut*

V-cut calculation requires the height of the cut and the burden and subsequently charge concentration. The charge calculated in this section assumes that the angle at the inner point of V is at least 60°. If the angle is less than this, then the charge per hole must be increased or yet another V is to be applied in height or depth. The height of cut for three V's vertically is also required.

For all the holes the bottom charge (h_b) is equal to at least 1/3 of hole depth, H. A sharper angle requires a more powerful charge. The column charge concentration, l_p is to be 0.5 × bottom charge, h_b. Uncharged part of the cut holes $h_o = 0.3\,V_1$.

13.6.3 *Stoping holes with breakage upwards or horizontal*

The drill holes are charged with a concentrated bottom charge to one-third of the hole depth. The burden is taken not larger than

$$\frac{\text{Hole depth} - 0.40}{2}$$

Hole spacing is taken to be 1.1 × burden. The following specific charge is needed in the bottom section of the blast hole.

Drill hole diameter (mm)	Specific charge kg/m^3
Approx. 30	1.1
40	1.3
50	1.5

The concentration of column charge section of the hole is taken to be 0.50 × bottom charge in kg/m.

13.6.4 *Floor holes*

The floor need higher specific charge due to the gravitational effect and also greater time interval between holes. The burden and spacing for the floor holes can be calculated in same way as for stoping holes above.

While floor holes are located, it is important to consider 'look out' which need to be included in the burden (Fig. 13.15). The magnitude of the look out angle is dependent upon the available drilling equipment and hole depth. For an advance of about 3 m a lookout angle equal to 0.05 rad (3°) (corresponding to 5 cm/m) should be enough to provide room for drilling the next round.

Hole spacing should be equal to burden, V. However it will vary upon the tunnel width. The number of floor holes is given by

$$N = \text{the integer part of} \left[\left\{\frac{\text{Tunnel width} + 2H\sin\tau}{V}\right\} + 2\right]$$

The spacing S for the holes is given by

$$S = \frac{\text{Tunnel width} + 2H\sin\tau}{N - 1}$$

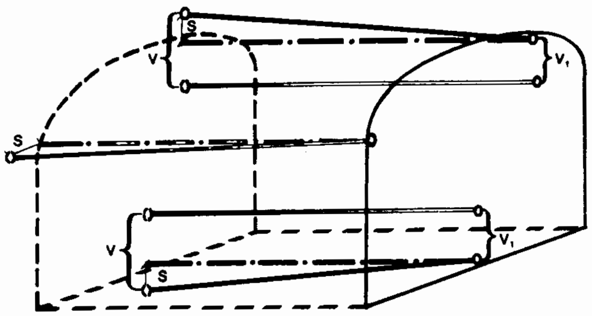


Figure 13.15 'Look out' of roof, wall and floor holes.

13.6.5 *Stoping holes with breakage downwards*

Since these holes require less effort for breaking and also are helped by gravity, the specific charge in the bottom section can be reduced to:

Drill hole diameter (mm)	Specific charge (kg/m^3)
Approx. 30	1.0
40	1.2
50	1.4

In most cases burden can be increased by 10%. The hole spacing can be increased to 1.2 × burden. In the case of small cross-section areas, burden and spacing are decreased according to the geometrical conditions.

13.6.6 *Wall/roof holes*

If smooth blasting is not necessary, the burden including the 'look out' is selected as being 0.9 × the burden for stoping holes. Hole spacing is taken as 1.2 × burden. The charge length in the bottom is reduced to 1/6 × depth of hole.

Uncharged section = 0.5 × burden. The concentration of the column charge is reduced to 0.40 × concentration of bottom charge.

13.7 HEADING AND BENCH METHOD

In large tunnels with dimensions over 10 m (100 m^2) the *Top Heading and Bench Method* of excavation is frequently used because it is adaptable to a high speed drilling, blasting and mucking cycle. It is preferred method in weak ground. Occasionally advantage is taken of lower drilling cost and powder factors with medium diameter vertical drill holes. The top heading and bench arrangement is most common. First, the heading at the top of the tunnel takes out a part of the finished height and its full width (Fig. 13.2). Then the lower portion is taken out in one or more benches. In horizontal drilling (Fig. 13.3a), the same equipment employed for the top heading can be used. Horizontal holes are generally inclined slightly towards the free face because this increases the forward movement of the rock. An entire cycle of

drilling, charging, blasting, loading etc. must be completed before the drilling of the next round can begin. Figure 13.3b shows a bench round using vertical medium diameter holes. A different type of drilling equipment is generally required from that used to drive the top heading. In this case, however, the drilling and loading can proceed independently of each other.

The top heading may be divided into a pilot heading plus one or more slashes (Fig. 13.2). In this way reinforcement can be installed in the roof of the tunnel before widening to full span.

Pilot Heading Method has also been used to drive a large number of tunnels. A pilot heading is driven down the centre line of the proposed tunnel by conventional methods and is usually completed portal to portal. The pilot tunnel is enlarged to full size by means of long holes parallel to the centre line. This method is advantageous where overbreak and shattering of walls must be avoided. With this method slots or stations are cut in the side of the pilot tunnel to the full diameter of the finished tunnel. These slots are spaced at predetermined intervals and used for drilling the long parallel holes for the enlargement blasting operation. *Pioneer Tunnel Method* consists of driving a small tunnel, parallel to the line of the main tunnel about 15-25 m on one side. The pioneer heading allows exploration of the main tunnel area ahead of its construction, it gives sufficient warning of ground changes.

13.8 EXPLOSIVE CHARGES

On completion of drilling, the holes are charged with explosives, primed and stemmed. Drilling equipment is removed from the tunnel face and men withdrawn to a place of safety whilst blasting is carried out.

Explosives used in tunnelling should have high density, high energy and be free from objectionable fumes. The high energy ensures good fragmentation for easy loading, and the high density is required to give concentration of the explosive force at the back of the boreholes to give maximum pull in each round. Good water resistance is required in wet conditions and often in floor holes. However, with certain cut patterns such as parallel hole cuts, when low charge density is essential for the success of the cut, a medium density explosive with good sensitivity is preferable. It was common to space out the charges with short pieces of timber, but now low density semi-gelatinous explosives or emulsion/slurry type explosives are used. Often slightly weaker explosives need to be used in the perimeter holes to minimise damage to the remaining rock. ANFO is often used in areas which are not wet and can be charged by means of special pneumatic equipment. In Sweden special pipe charges are used in tunnelling operation, these are 700 mm long and need lesser time in loading a round.

Because of the greater confinement of the rock, small tunnels generally require a larger amount explosive per unit volume of rock broken than large tunnels. Indian experience (Anon 1988a) shows that using 80% NG based explosive, the consumption is of the order of:

3.9-4.3 kg/m^3 for granite and other hard igneous rocks
2.9-4.1 kg/m^3 for hard shale and limestone
1.2-2.4 kg/m^3 for soft stratified rock

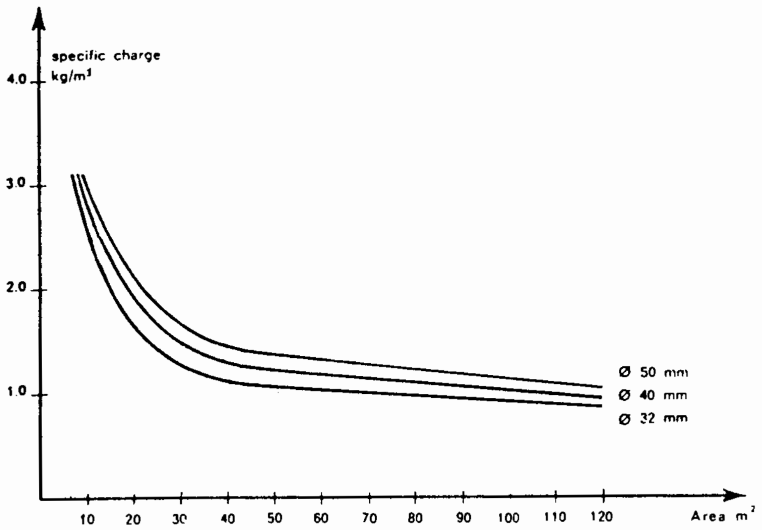


Figure 13.16 Explosive consumption related to tunnel size (after Gustafsson 1981).

The tightest area is of course the cut and the charging ratio may be as high as 10.25 kg/ m^3 but as the distance increases, the beneficial effect of the free face developed also increases and the ratio of charge to broken rock approaches the conditions required for minimum explosive consumption.

Fig. 13.16 gives a good guide for explosive consumption for varying tunnel sizes (Gustafsson 1981).

US experience in connection with explosives requirements as related to tunnel cross-section are given by Whittaker & Frith (1990) as:

Tunnel cross-sectional area (m^2)	Quantity of explosives (kg/m^3)	
	Weak rocks	Strong rocks
10	1-4	5–7
25	1-2.5	4-5
50	1-2	3-4

In benching operation in a tunnel using the top heading system explosive consumption may be as high as 0.56-0.85 kg per m^3. Conditions under which explosives break the rock in the bottom bench of the tunnel differ, owing to a second free face and the tunnel wall constraint, from blasting conditions in full face tunnelling and open pit mining. All conditions being equal, the consumption of explosives for the bottom bench will be less than that for the heading, but greater than that in open pit mining.

Often explosive cartridges are placed as continuous column or are filled as column charge with ANFO or other similar products. In a column charge, the effectiveness of blasting is greatly affected by the location of the primer and the material and the size of stemming. The primer can be placed at the blast hole collar or on the blast

hole bottom to effect a direct or an inverse initiation of the blast. In the inverse initiation, the energy of the blast is utilised to greater degree than in the direct initiation owing to longer action of explosive products upon the surrounding rock. Because of this rock fragmentation is increased and the throw of broken rock is reduced.

Stemming is used to enhance the effectiveness of blasting, and its use is always must in drifting operations. Generally, the length of stemming may be taken equal to 0.6-0.8 m for blast holes of 40-42 mm in diameter.

A widely used stemming material is a mixture of clay and sand mixed in a ratio of 1-4, clay being the binder for sand.

Along with the use of sand-clay stemming, water stemming of polyethylene ampoules of 300-400 mm long, 37-38 mm in diameter have also been introduced. These improve fragmentation and reduce dust production. In coal mines this also reduces risk of methane gas ignition. In addition, water stemming absorbs some of the blasting fumes. Use of gel ampoules has also been advocated.

Due regard should be given to the effect of blasting on surrounding structures. Chapter 18 gives methods which allow determination of the weight of the explosive to ensure that the blasting vibrations are kept below the threshold of damage to surrounding structures. Chapter 20 describes technique to control damage to walls of tunnels.

CHAPTER 14

Blasting in underground coal mines

14.1 INTRODUCTION

Blasting in underground coal mines is a specialised operation. The proper use of explosives is vital to safety, influences the cost of production, and in many cases, has a direct bearing on the quality of the coal produced. The possible occurrence of highly inflammable mixtures of methane and air and explosible dusts has led to special statutory requirements for explosives and their use in such mines.

The geological nature of coal seams shows a considerable variation such as height of the seam, type and amount of impurities, cleavage, brittleness and type of formations surrounding the seam. These factors greatly influence the blasting methods and type of explosive required.

Basically two methods of working are followed in coal mines – Room (also called Bord) and Pillar, and Longwall Methods. Different blasting practices are followed in the development stage and in removing pillars so formed. In this chapter use of explosives and blasting practices in underground coal mines are discussed.

14.2 RESTRICTED CHOICE OF EXPLOSIVE

In underground coal mines, highly inflammable mixtures of methane and air and explosible dust has led to special requirements for explosives.

In many coal mines methane is emitted regularly or at intervals and it can ignite when mixed with air in proportions between 5 and 14%. The ignition starts when it comes in contact with a body at 650°C temperature or higher. The time required for ignition depends on the temperature. At 650°C, 10 seconds of contact between the hot body and gas are necessary before the gas ignites, but at higher temperatures the time shortens rapidly.

Very fine coal dust, suspended in the air in quantities of not less than 30-40 g/m, is also capable of exploding if in contact with a body at 700-800°C.

The reaction temperature of explosive is much higher than the minimum temperatures needed to ignite mine gas or coal dust. Therefore, to avoid gas or dust ignitions in coal mines, only those explosives should be used in which the reaction is both extremely fast, and produces a short flame. In addition, the reaction should produce a large quantity of inert gases which are unable to enter into further reaction with the oxygen of the air.

Explosives with these properties, allowed to be used underground in gassy or dusty mines, are called permitted (permissible) explosives. The protection is achieved by flame- quenching additives or substances which absorb some of the heat of the explosion.

Each country has own regulatory authority and requirements for granting permission and testing explosives for use in coal mines. In the United States the explosives that have passed the tests by MSHA are considered safe for use in coal mines. In the Great Britain the Ministry of Energy tests and permits use of explosives in coal mines. In Canada, a 'permitted' explosive is one approved by MSHA or the UK, Ministry of Energy. In Great Britain and in India mostly similar tests are adopted:

The permitted explosives for blasting in coal mines are sub-divided into five groups, each group being designed for a particular mining operation to give maximum safety.

Group P_1 explosives may be used for instantaneous blasting in undercut coal or relieved rock (ripping) near a coal face. They can also be used for delay blasting in shafts and drives away from sources of gas. These are for use in Degree I mines.

Group P_2 explosives are sheathed by sodium bicarbonate This method is obsolete.

Group P_3 (previously known as 'Equivalent to sheathed'), used mainly for blasting undercut coal and rock ripping by single shot firing or instantaneous firing of up to six shots. These explosives are used in Degree II and Degree III mines.

Group P_4 explosives were developed for use in rock ripping with delay firing.

Group P_5 explosives were designed for delay blasting in solid coal i.e. when the coal face has not been undercut.

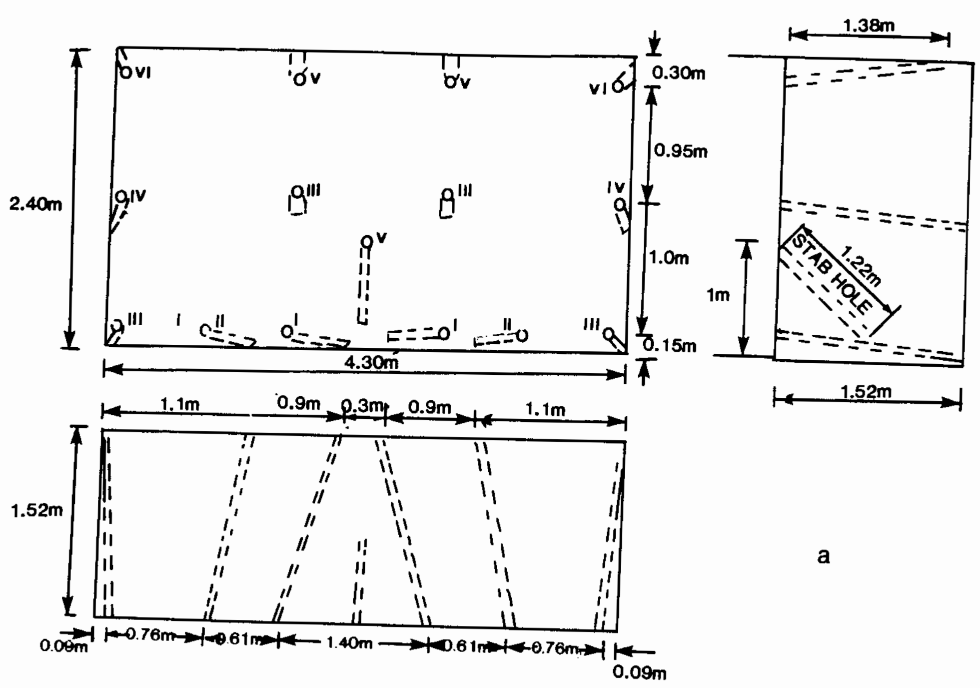
Non-incendive detonators are only permitted to be used in coal mines. Usually tests are specified for these detonators also. Only electric detonators which have copper or copper coated steel shells which meet with the requirements of statutory authorities can be used to initiate explosives in underground coal. Cap and fuse or detonating cord are not permitted. An exception is use of non-incendive detonating cord for blasting gallery method in France and in India. Every blasting machine, every instrument, every explosive cartridge and every detonator case must be clearly marked as being approved permissible.

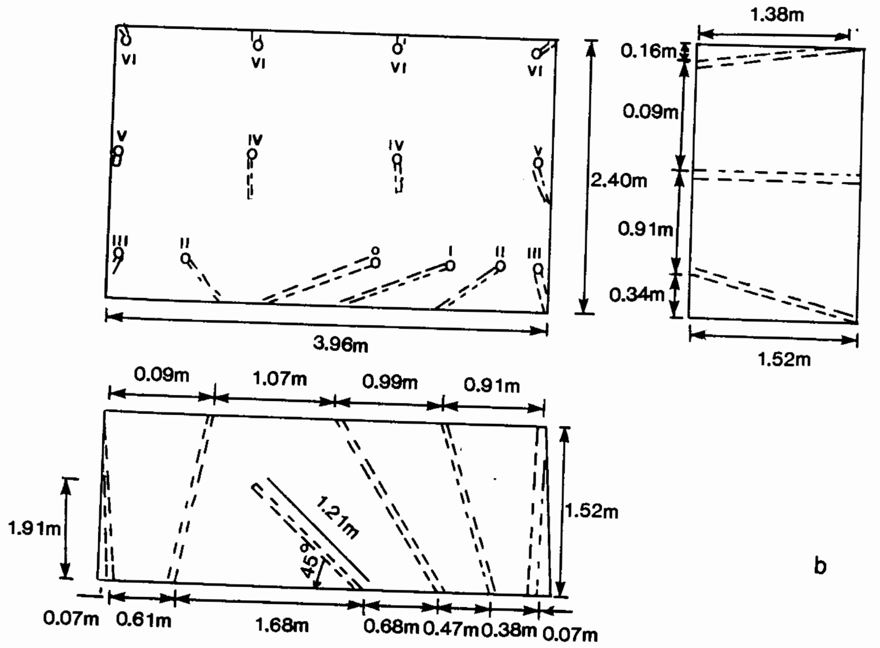
14.3 BLASTING METHODS

In drives or in mining out a room, the coal is either blasted 'off the solid' where the coal is blasted from the solid seam which has but one free face, or 'cut coal' is blasted in which the seam has previously been cut to provide at least one more free face.

14.3.1 *Blasting off the solid*

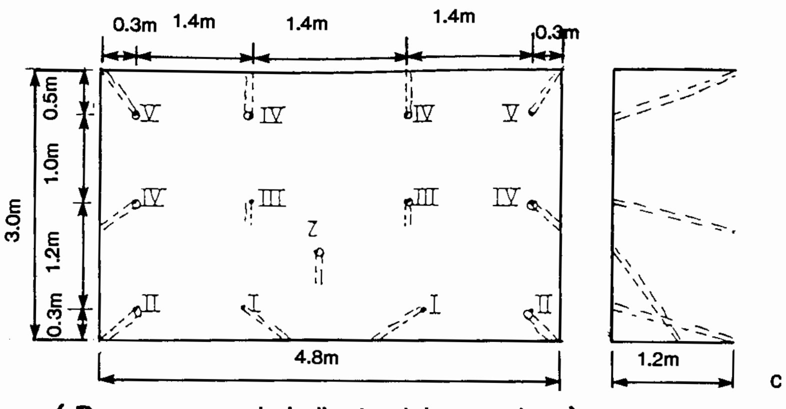
The first essential in the technique of drilling and blasting solid coal development headings is to create a free face for the explosive to blast into. The holes (which will form the CUT) are drilled so that relief of burden is towards the free face. The remaining holes in the round are drilled so that they break towards the cut. Short delay detonators are used to control the sequence of firing of holes. The maximum burden





(Roman numerals indicate delay sequance)

Figure 14.1 Blasting off the solid coal. a) Wedge cut pattern, b) Fan cut pattern.



(Roman numerals indicate delay number)

Figure 14.1 Continued. c) Drag cut pattern.

placed on shot holes with this method should not exceed half their depth and normally, is established by field trials since breakage characteristics vary from seam to seam. Holes should be spaced to ensure that one does not rob the other and that the explosive charges will be adequately confined for maximum efficiency and safety.

Blasting off the solid offers several advantages over other methods of blasting in that cutting machines are not necessary and in many locations, better advance per manshift can be obtained which facilitates the simultaneous use of high speed mechanised mining methods. Development of permitted explosives both NG-based and slurry type have greatly improved safety and efficiency during recent years, permitting increased application of this method in longwall, bord and pillar and in development work.

Three methods of solid blasting off the development headings have been successfully used. The method used depends on the mining conditions and individual choice.

Wedge cut pattern

In wedge cut or V-cut pattern (Fig. 14.1a) 'breaking in', 'cut' or 'opening' holes are drilled towards each other to form a wedge, the angle and depth of which depends on the prevailing conditions, hardness of coal etc. The wedge cut pattern will reduce the number of delays needed. In hard coals it may be necessary to use an inner wedge in addition to the main wedge for pulls of 1.8 m (6 ft) or over. In some mines (Degree I) first the cut holes are fired and in the second round remaining holes are blasted using instantaneous detonators for all the holes. This later practice is unsafe, therefore, not recommended.

Fan cut pattern

In fan cut (Fig. 14.1b) method, the holes are drilled in a fan like pattern initiating the shortest holes first with the delay sequence rising towards the longest holes. This system is advantageous in that there is less liability to dislodge supports than when using the wedge cut. Holes are drilled progressively at approximately 30, 50 and 70° and at increasing lengths. Holes are normally drilled in pairs on two horizons.

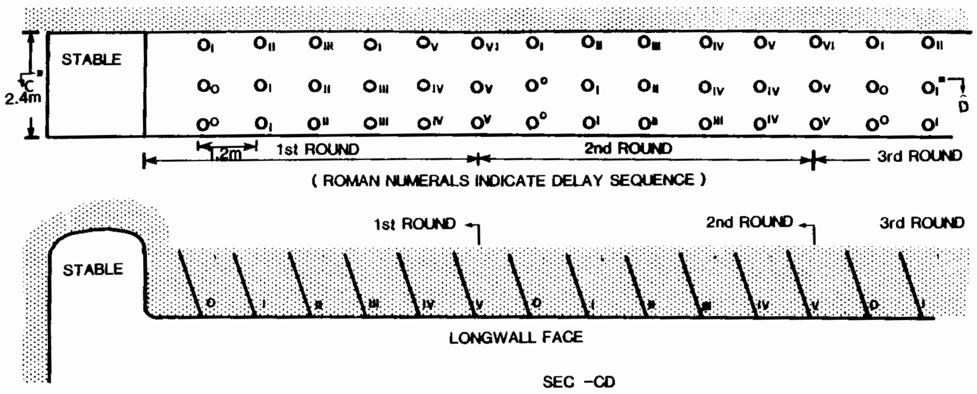


Figure 14.2 Blasting off the solid on longwall face.

Drag cut pattern

The drag cut (Fig. 14.1c) is in effect a type of 'vertical' fan cut in which pairs of holes are drilled towards either the floor or the roof of the excavation. This pattern is less commonly used. The angle of the hole is somewhat similar to the angles used in the fan cut and again depend principally on the hardness and character of the coal being blasted.

After the initial free face has been excavated, the remaining round can either be fired with delay detonators in one sequence, provided that sufficient delay is available to do this or alternatively, several rounds may be used to pull the necessary advance.

Every mine has to take permission for solid blasting from the statutory authority for mines safety.

14.3.2 *Blasting on longwall face*

On a longwall face, the shotholes are drilled at an angle of 45°-60° to the face; a coal face 2.4 m high requires 3 rows of holes and distance between holes in the same rows of holes should be nearly 1.2 m. A typical layout of shotholes on a longwall coal face for blasting off the solid is shown in Figure 14.2.

14.3.3 *Blasting in cut coal*

With this method the solid coal is relieved by cutting and thus an additional free face is provided to a shot. This permits a more equal distribution of the total explosives energy and consequently decreases the amount of explosive needed to blast the coal. The normal cut or kerf is 125-150 mm wide and 1.3-2 m deep, and is placed either horizontally at floor level or near the roof, or at some intermediate position. Vertical cuts are also used. In longwall operations the cut should extend for full length of the working face. Since the cut provides an additional free face and a chance for the coal to be displaced, it follows that the wider it is, the more effective it will be. It is essential that a machine cut be made accurately. If the cut is irregular drill holes cannot be properly located and as a result poor blasting results occur. In addition to estab-

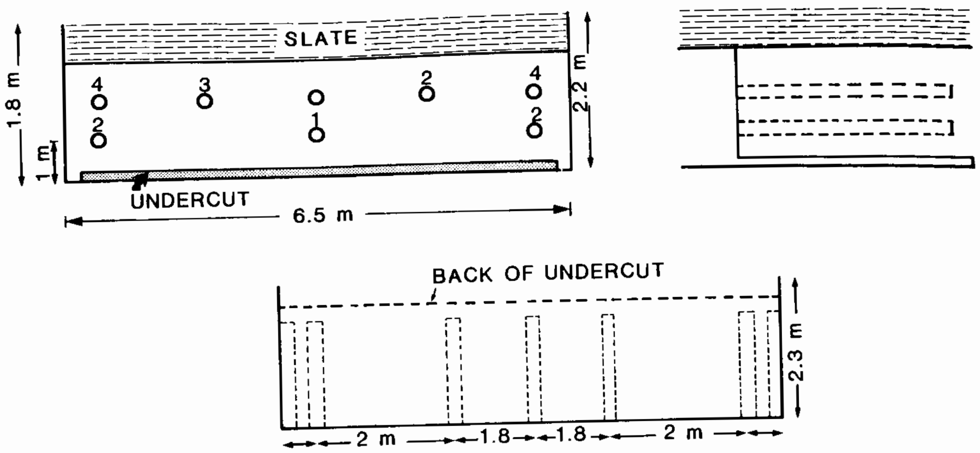


Figure 14.3 Blasting with an undercut.

lishing free faces, the cut provides an additional area for expansion of the broken coal. A given volume of solid coal will 'swell' approximately 30% when broken. Therefore, it is essential that all 'cuttings' be removed to provide additional room for expansion. The cut provides only a small portion of the volume required for expansion. Therefore, the coal must also be moved forward into the heading.

The proper location for the cut is normally determined by the physical characteristic of the coal seam, the stability of the roof and the firmness of the floor. Bottom cutting is the most common type of cut. It is used in areas where floor is soft. The machine cut will leave relatively smooth surface. A middle side or top cut is preferred sometimes. The middle cut is preferred where heavy horizontal rock bands are present in the seam. The top cut offers maximum confinement for all the explosive charges with minimum disturbance to the roof, thus for weak roof, this cut is preferred.

Centre, off centre cuts are used to provide additional relief in seams that are difficult to blast.

The position of holes for blasting cut coal depends on thickness of the seam, position of the cut, presence of hard dirt bands, type of parting with the roof, cleat of coal and methods of working. The depth of holes should be such that there is ample clearance at the back of the cut. Under average conditions where a seam 1.75-2.5 m thick is undercut, in headings 3-5 m wide, six holes are used (Fig. 14.3). In middlecut, the holes are placed as shown in Figure 14.4. Some coal seams, due to extreme roof weakness, are overcut with this method, the shot holes are normally placed as in heavy shooting. More than one row of holes will probably be necessary where the seam is 1.7 m thick or over. Closer spacing is usually required or they may be loaded more heavily than when blasting undercut coal, since the coal has to be lifted as well as broken.

Proper drilling with accurate measurement is essential for good blasting. The holes should never be drilled beyond the point of relief. They should never be drilled deeper than the cut nor gripped into the ribs. The placement of holes should be so chosen taking into consideration:

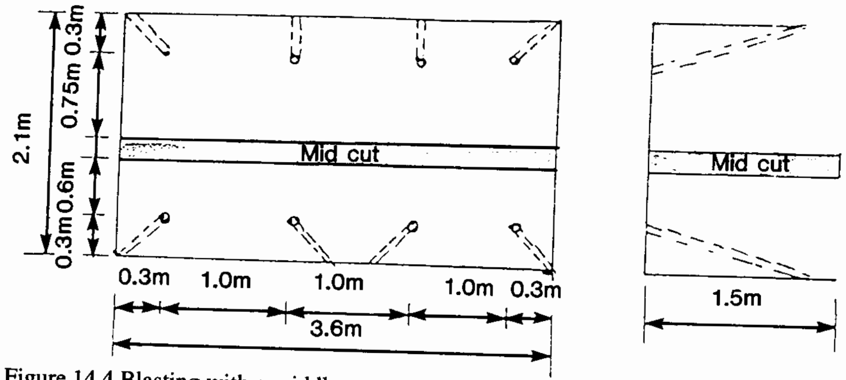


Figure 14.4 Blasting with a middle cut.

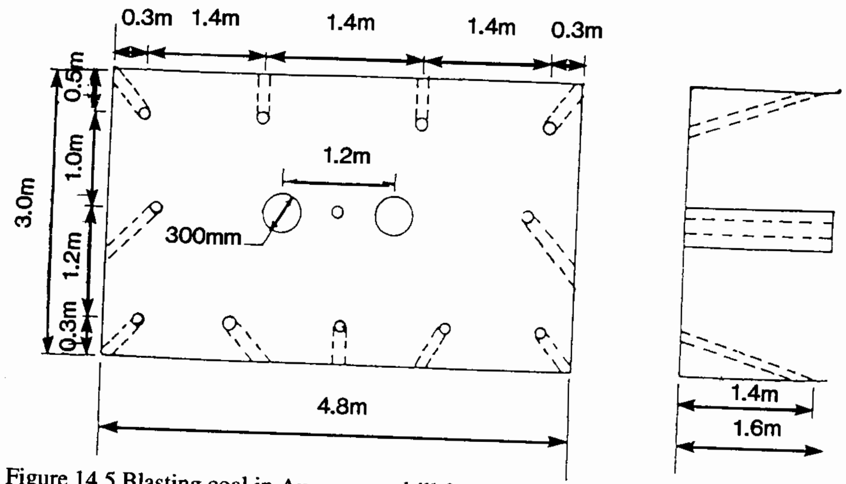


Figure 14.5 Blasting coal in Auger-cum-drill faces.

1. The maximum distance from the ribs at which the explosive charge will shear the ribs – usually 150-200 mm.

2. Holes not deeper than is necessary to break the cut (a depth of 150 mm less than the cut) usually pulls full depth.

Before drilling cut coal, it is essential to measure the depth of the cut. The holes should not be drilled into the solid beyond depth of the cut or into solid coal along the ribs. This would cause the explosive charge to fire in the solid. Usually, the depth of the hole is statutorily required to be 150 mm less than the depth of the cut.

The location of the holes will vary with the individual coal seam. In general, the holes should be drilled as level as possible and 150-450 mm off the rib lines.

Each hole must have a minimum burden of 0.45 m. In most coal seams holes can be located with 1-1.3 m of burden. The holes should be spaced to ensure that one hole will not rob the other holes and that each hole will have adequate confinement to achieve maximum efficiency and safety. All holes should be of sufficient size to accommodate the explosive cartridges without having to use force to charge them.

Blasting in Auger-cum-Drill faces utilises instead of cut, one or several holes in bigger dimension than the blastholes, and apply the concept of blasting on a burn cut. The diameter of the auger is 250-300 mm. The larger diameter holes provide greater free faces (Fig. 14.5).

14.3.4 *Blasting in depillaring operations*

Blasting in depillaring operation is the same as that of the cut face blasting. There are two free faces present. Here P_1 and P_3 explosives (where ever applicable) are used along with instantaneous electric detonators. Blasting is carried out towards the free face parallel to the plane of the holes .

14.3.5 *Blasting gallery method*

Blasting gallery method developed in France and also utilised in India involves drivage of level galleries along the coal seam. Depillaring is done by drilling a series of holes in a ring pattern from these galleries (Fig. 14.6). The full thickness the seam is extracted by retreating along the galleries and blasting and loading out of successive slabs. Shotholes of 42 mm diameter are drilled using extension rod of 1 m. The shotholes are loaded with a succession of explosive charges and PVC spacers (used to distribute the charge in the shotholes, provides 500 mm distance between the cartridges) with detonating fuse along with them from the first loaded cartridge to the detonator inside the last loaded cartridge. The detonating cord transmits initiation to the cartridge. The end of the cord is inserted by pricking a suitable hole, in the first cartridge. The last cartridge is primed by making a hole at an angle of 45° in the cartridge and inserting the detonating fuse along with the detonator. A clay plug is inserted before charging. Stemming is done with wet clay plug to the mouth of the

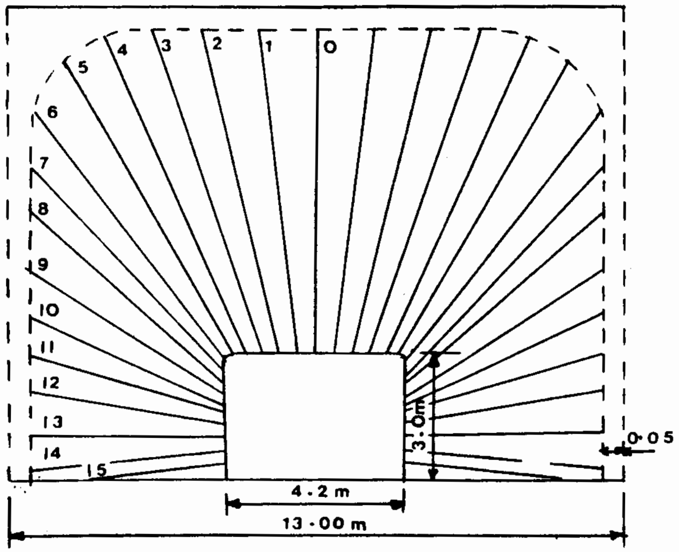


Figure 14.6 Blasting gallery method with ring of holes.

hole. All holes are fired simultaneously in a ring. The explosive used is specially formulated of 32 mm diameter and 200 mm diameter and 200 g in weight. The detonating cord used is non-incendive type.

14.4 BLASTING GUIDELINES

All phases of a cycle of production such as face preparation, cutting, drilling and blasting must be considered as an intrinsic unit completely dependent on the other for success of the cycle. As underground coal blasting is specialised operation wherein additional safety precautions are to be taken, a detailed drilling and blasting procedure is described here.

– The preparation of the face is accomplished at the beginning of the cycle on repetitive basis. Besides support checking it is essential to test the face for the methane concentration and assuring that the ventilation is adequate; and

– To reduce dust problems it is mandatory to sprinkle water also. The face should be examined carefully for cracks and crevices.

All cuts should be well cleaned. Partially cleaned cuts result in decreased blasting efficiency as there will not be room for normal expansion of the broken coal. A water filled kerf will result in the same effect. If the water cannot be removed, additional drill holes will be necessary.

Proper drilling of holes with accurate alignment is fundamental to good blasting. When holes are properly placed, diameter is correctly chosen, alignment and depth are accurately controlled, fewer holes and less explosive will be required to adequately fragment and displace the coal for maximum loadability.

Permitted explosives only may be used in coal mines. The size of charge detonated in any one shothole, shall be within the specific charge limit for that particular explosive.

Before placing explosive in any shothole, the shot firer should clean it thoroughly and examine it carefully using a proper tool designed for this purpose. If loose cuttings were allowed to remain in the hole, they might be set on fire by the shot or they might come between the cartridges, causing part of the charge to fail to blast or possibly to burn.

The existence of cracks, breaks or partings is one of the most common causes of flame shots because such openings are usually filled with gas. If they extend across or along any hole, the firing of a charge may result in flame being projected into the openings, with consistent danger of an ignition. Any shothole which is not thoroughly clean, or which intersects a known crack or opening, should not be charged or fired.

There is a tendency among shot-firers to put extra charge to properly fragment and displace coal for maximum loadability. This practice is not only an unnecessary expense, but is hazardous because:

1. More than the normal coal dust is produced and thrown into suspension;
2. Over-charged hole will rob normal burden from adjacent holes and allow subsequent charges to shoot partially unconfined;
3. Flying fragments are thrown farther at higher velocity causing injuries; and
4. Excessive volume of smoke and fumes are created.

An overcharged hole may leave dependent or adjacent holes with insufficient burden, permitting subsequent charges to break through the confining material more readily and project flame and hot particles into the atmosphere of the working place. It is also important that the shot not be under-charged because it may not break but blow out, projecting flame into the atmosphere.

The average yield of coal usually runs from 4-10 tonnes per kg with maximum and minimum varying within great limits. The amount of coal produced from a kilogram of explosive varies from less than a tonne to 25 tonnes. On an average the following yield has been suggested.

– Cut face 5.0 t/kg;
– Solid blasting 2.5 t/kg;
– Depillaring 4.5-5.5 t/kg.

Normally, loading one-third to one-half of the depth will be sufficient to produce good loadability. The maximum energy is required in the back of the hole to shear the cut away from the solid coal.

Primer location has received a great deal of study over the years. Gallery tests have seemed to indicate that with single shotfiring, a charge primed in the direct manner, that is with the primer cartridge placed last in the hole and the cap in the outer end pointing toward the bulk of the charge, is less likely to ignite gas or dust in the atmosphere than is the charge primed in some other manner. It has been found in practice, however, that indirect priming with the primer cartridge placed first in the hole and the cap in the inner end pointing outward toward the charge is preferable. Indirect priming is strongly recommended where multiple shot firing with millisecond delay detonators are employed. The most desirable location for the primer with this method of firing is the first cartridge in the hole since the danger of cut off is reduced to a minimum when it is loaded in this position. There are several other points in favour of indirect priming. For instance, in cases where a seam is undercut there is sometimes a tendency, for coal to break away from the roof which results in cracks and crevices opening up above the back of the mining, extending from the roof toward the floor. When this occurs, the bottom of the shothole might be close to such crevices. Under such conditions a charge primed in the indirect manner is generally less likely to ignite gas in the crevices. With indirect priming, there is no possibility of the primer becoming separated from the remainder of the charge since any accidental pull on the cap wires would cause the entire charge to move. Also the entire charge can be withdrawn should this prove necessary. This would be impossible with direct priming as only the primer could be withdrawn or moved.

It is required by law that all holes be stemmed with an incombustible material. Stemming confines the explosive charge, forcing it to do its work effectively and guards against blowing out and the projection of flame and hot particles into the working place. The stemming material should be damp stone dust, clay, sand or a clay-sand mixture. Combustible stemming (such as coal dust) should never be used, as it might be ignited by the shot. When stemming the holes, the first few pieces should be pushed gently back to the charge; as subsequent material is added to fill the holes, tamping may become progressively harder. Water-filled plastic bags are permitted devices for stemming. Water stemming bags have proven very effective in providing confinement and reducing dust.

CHAPTER 15

Underground hard rock blasting

15.1 INTRODUCTION

Underground mining in hard-rock is highly dependent on the successful execution of blasting procedures. Although rock borers, shearers are occasionally used, the majority of mine development and production is still undertaken with well established and efficient methods of drilling and blasting. Ore body rock is liberated from its natural surroundings and is subjected to fragmentation by primary blasting. Other minor blasts in stope draw points or ore passes, for example, may be required to maintain the free flow in the ore handling system. Adequate release is particularly important within the confines of underground rock excavations. The quantity of explosives required per unit volume of rock excavated (the specific charge), because of the confinement of the blast, is much greater than in the case of open-cut excavations. Powder factors range from 0.9-6 kg/m^3. The lower values are used in large open rooms in soft weak strata while the higher values are used in confined raises and shafts for hard competent rock. A particular concern with blasting is its effect on the rock in the immediate periphery of an excavation. Intense local fracturing, and disruption of the integrity of the interlocked, jointed assembly, can be produced in the near field rock by poor blast design. More extensive adverse effects can be induced by the transmission to the far field of energy input to the rock by explosive action. In high-stress environments, such as occuring at depth, or in pillars supporting panels mined to a high extraction ratio, stress waves associated with blasting may trigger extensive instability in the mine structures. The gases produced by blasting are also of concern in underground applications.

There are many different methods of ore extraction practised whose choice mainly depends on thickness of deposit, angle of inclination, characteristics of ore and country rock etc. Many metalliferous deposits are located in veins or irregular masses, often steeply inclined. These call for blasting techniques different from those employed for minerals such as iron ore, limestone, bauxite, etc. which usually occur in stratified deposits of uniform thickness.

In this chapter blasting techniques as adopted for underground hard rock mining are described. There are many different types of blasting because the conditions under which they are carried out and results required are also varied. Basically blasting can be categorised as development blasting and stope blasting. Similar techniques are also applied for construction of underground chambers.

15.2 EXPLOSIVES FOR UNDERGROUND BLASTING

Suitable explosives for underground work are those which are easy to handle and charge and are chosen according to rock and blasting conditions. Explosive with fume class I are required and in gassy mines permitted explosives must be used. Both cartridges of NG explosives and emulsion/slurry and bulk loaded ANFO, emulsion/slurry explosives are used. Blasting conditions vary in size, depth, hole inclinations etc., and hence the explosives have also to be tailored accordingly. Atmospheric conditions present high humidity and temperatures, therefore, storage of explosives be made for a limited period of time. Usual precaution of rotation of explosive need to be taken. During blasted muck removal, a constant watch for unblasted explosive must be made. Prior to drilling the subsequent round, the face must be closely inspected for evidence of unshot explosives from the previous round. The holes are then blown with compressed air to clear blockages and remove water from the holes.

While loading explosive cartridges in short holes, the charge is tamped with non-sparking poles. The cartridges containing the initiator should never be tamped, but rather pushed gently into the holes. In recent years, advances in hole loading have been achieved with mechanised ANFO loaders, pneumatic cartridged loaders, bulk loading system uses pumpable products. Pneumatic loading uses air under pressure to inject conventional cartridge as well as dry bulk explosives (Ljung 1978; Smith 1982; Day & Joyce 1988). In loading dry bulk ANFO, care must be taken to adjust the pressure level of the vessel and the line by use of pressure regulator. Static electricity build up is a problem with this method, and non-electric or anti-static initiating devices must be used with pneumatically loaded ANFO. The success of bulk loading ANFO and wet blasting agents in upholes, using pneumatic ANFO loaders and pumpable slurry/emulsion explosives has allowed the application of large diameter holes 150-165 mm to stoping techniques and the development of the vertical crater retreat (VCR) method of stoping (Lang 1978).

Priming methods used in underground holes are a matter of choice. Large diameter holes are primed at the base of the hole. Blasting devices are inserted with the detonating end facing the charge column. Holes should be primed at the hole bottom to ensure maximum confinement. If the primer containing the initiator is placed at the hole collar, it could be expelled upon detonation if stemming length is too short. A misfire could result with unshot explosives remaining in the hole.

Blasting may be carried out during the shift or shift change time, depending on the size of mine and size of blast.

Post blast procedures must include a safe waiting time for fumes to disperse from the working area. This time variation is based on explosive and rock type, as well as ventilation system.

15.3 DEVELOPMENT BLASTING

In underground mining, an extensive and carefully planned network of excavations are needed, recognised as development workings. Four basic components of rock excavation are involved in normal mine development, shaft sinking, drifting, raising and ramp or incline.

Development workings are conventionally and still predominantly excavated by drilling and blasting the full cross-section of the face to the depth drilled. The broken rock is then mucked out and the cycle of operation is repeated. Often there is only one free face available and therefore, an important factor in the successful blasting is an effective cut to provide a further free face towards which the rest of rock is successively blasted.

Subsequent holes called helper or reliever holes, blast the rock into the cut, spreading outward from the cut in a pattern of rings of increasing diameter until they reach the perimeter. The advance obtained in each round is generally slightly less than the depth of the cut, and the cut holes are therefore drilled about 150-250 mm deeper than the surrounding holes. The choice of the cut is determined by the depth of pull required, the dimensions of the opening, the nature of the rock and the type of drilling equipment. There are two main types of cuts which are commonly used in development openings.

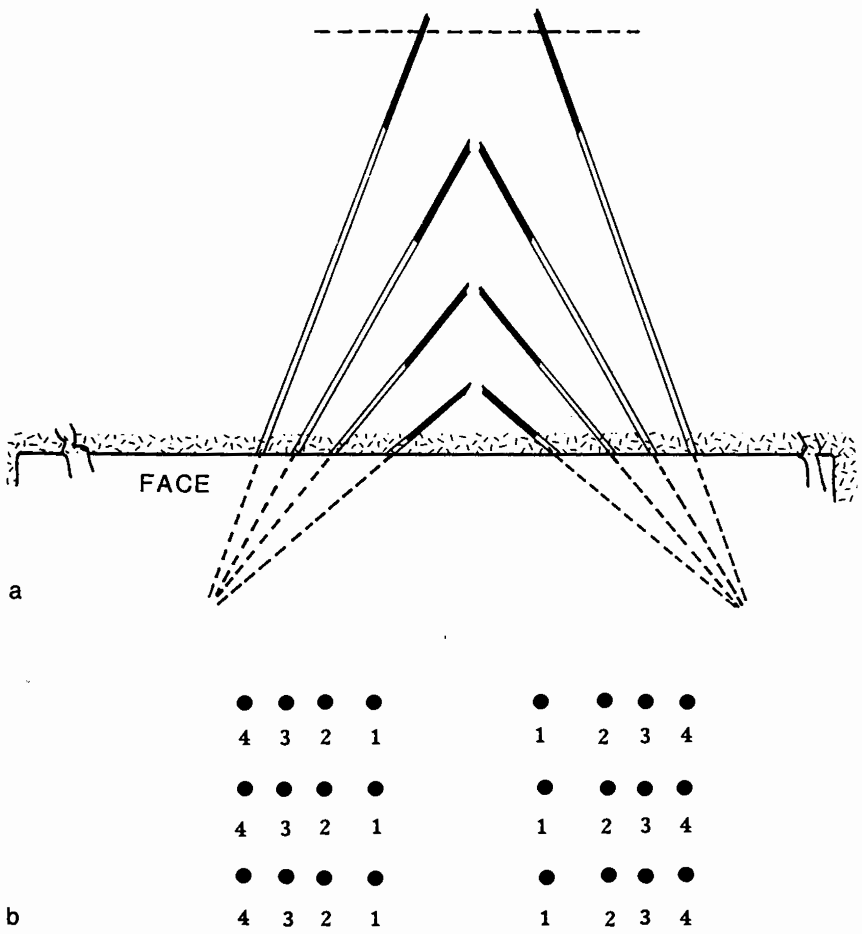


Figure 15.1 Angle cut. a) Plan view, b) face view and initiation sequence.

Angle cut

Many different types of angle cuts are employed. The most common type is the wedge cut or V cut which can be used for almost all types of rock. The cut is a wedge placed either vertically or horizontally and formed by pairs of holes drilled at about 60° to the face and meeting or finishing close together at the back of the cut (Fig. 15.1). With wedge cut there is a tendency for the rock in the centre to be blasted out in a large mass which could damage the supports and equipment; where this occurs one or more smaller wedges with shorter holes can be drilled inside the main wedge so that this mass is well broken up on blasting.

Angle cuts require fewer holes and less explosives, but are more difficult to drill accurately. They tend to give a more erratic throw of broken muck. Because advance is limited to 65% of the opening width, angle cuts are used most often in wide openings.

The pyramid cut consists of a group of shotholes drilled to meet at a point which

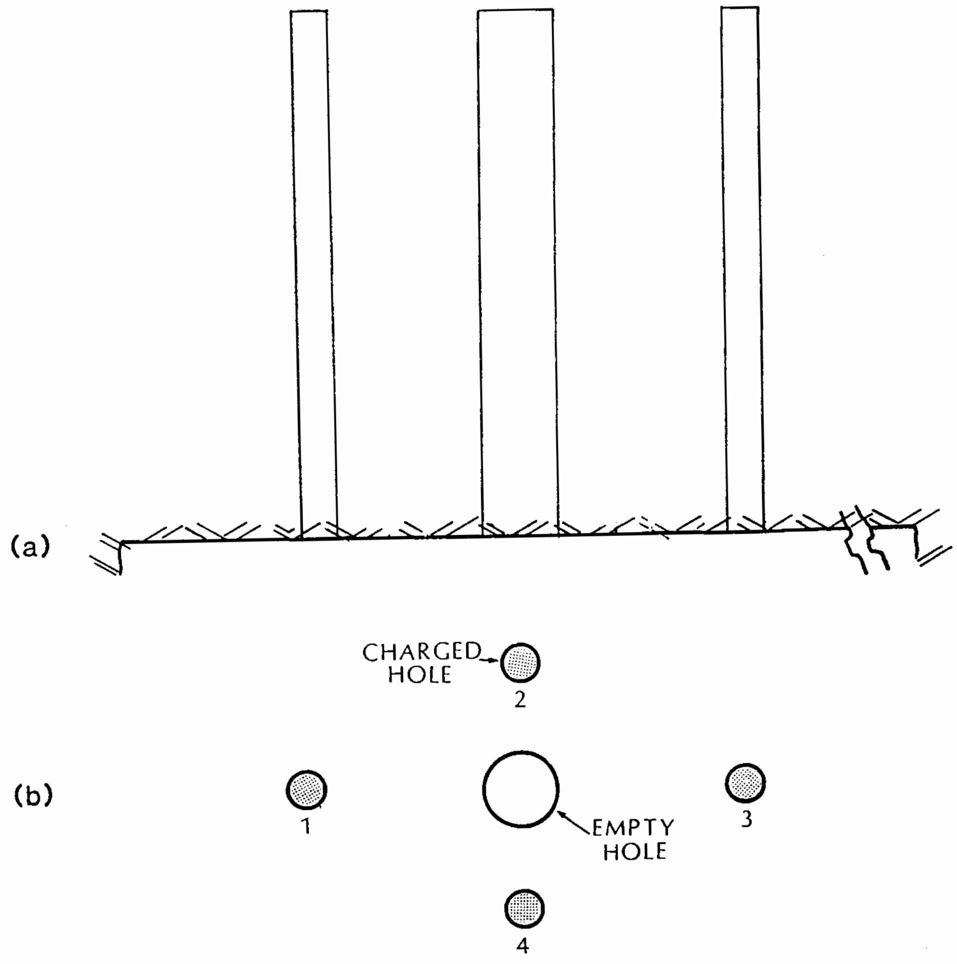


Figure 15.2 Parallel hole cut. a) Plan view, b) face view and initiation sequence.

forms the back of the cut. Usually, four holes are drilled in the shape of a pyramid but in tougher ground the holes may be increased to six. This arrangement gives a high concentration of explosives and is therefore suitable for very hard ground.

Parallel holes cut

With the angle cuts although the number of holes on the face and the explosives consumption/meter advance are less, the advance per round is limited as it is normally difficult to drill deeper than half the width of the drivage because of the angle of the drill. Since the size of openings in hard rock mines is generally small for reasons of economy, it would thus be desirable that, for achieving a higher rate of progress, a hole pattern independent of the size of the opening should be employed. The technique of 'parallel holes' blasting offers such an advantage.

A cluster of parallel shotholes are drilled at right angles to the face to blast out a cavity in the centre (Fig. 15.2). Some of the holes are charged while the others are kept empty and the blasting takes place towards these empty holes. The function of the empty holes is to provide a free face for rock and subsequent ejection of broken rock.

Modern trend includes use of larger diameter holes rather than a cluster of holes which are kept empty. Modern drill jumbos are capable of drilling 75-165 mm diameter holes in addition to smaller diameter holes. This has allowed change in drilling and blasting practices for drifting and raising practices. Chapter 13 provides fuller details of drive/heading blasts.

15.3.1 *Shaft blasting*

The purpose of shaft is to provide access to underground workings. The shaft can be used for various services, hoisting ore and rock, materials and personnel, to ventilate the mine etc.

The mine shaft is usually vertical or some time inclined to near vertical. The profile can be rectangular, circular or elliptical. Circular shafts are more frequently used. Shaft sinking is, in comparison with other development operations, a complex procedure. The sinking requires special machinery and specialised drilling and blasting techniques.

Drilling and blasting in shafts should provide:

1) Correct size and shape of planned excavation,
2) Even surface of the face that allows easier mucking and drilling the next round,
3) Safe and economical operation, and
4) Uniform fragmentation for effective mucking operation.

The locations of the holes drilled in the face depend on the shape of the shaft cross section and the bedding of the rock. In circular shafts, holes are placed on concentric circles, drawn from the centre of the shaft. The number of circles is three to five depending on the shaft diameter. Typical patterns of placing a round of holes in a circular shaft are shown in Figure 15.3.

In rectangular shafts, holes are distributed similarly to horizontal drifts. Typical patterns for rectangular shafts are shown in Figure 15.4, which are most often the wedge or pyramid type.

In the full-face method angle cuts are most often employed, although parallel cuts

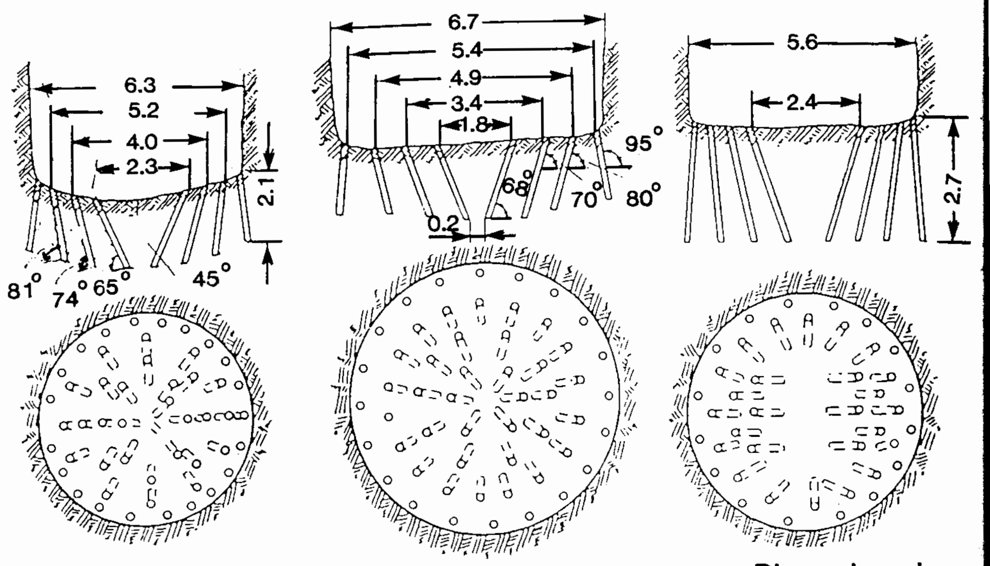


Figure 15.3 Patterns for circular shafts.

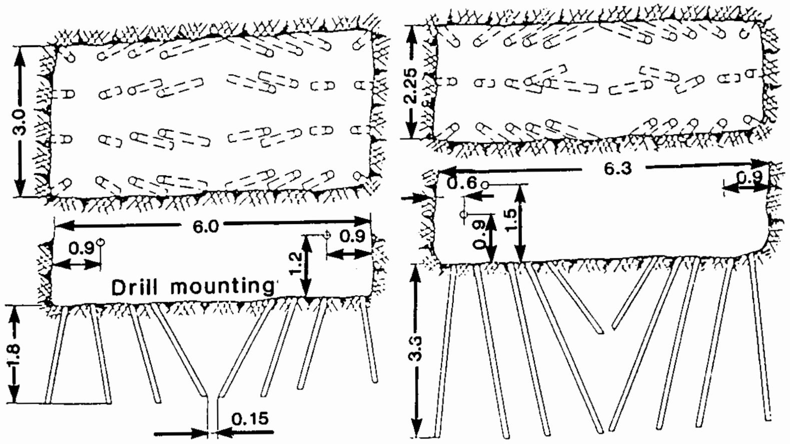


Figure 15.4 Patterns for rectangular shafts.

may be needed to limit flyrock and damage to timber supports and shaft fittings. Angle cuts permit deeper rounds, but usually with more overbreak. Increasing the depth of a round means a more rapid advance, but overbreak means more mucking, greater difficulty with accurate alignment, and a greater volume of concrete to be poured if the shaft is to be lined. Normally the cut is drilled in the centre of the round but the centre of the cut may have to be shifted to suit any strata weakness for better results.

The number of holes depends on the rock and varies from 1-2 per m^2 of the face area, more holes being used for stronger rock. The depth recently has been from 1.5-4 m, the deeper holes being drilled in the weaker rocks.

Example

Total number of holes: 52, total charge per round: 85 kg, pull per round: 2 m

Ring No.	Delay No.	No. of holes	Charge/hole (kg)	Total charge (kg)
1	0	04	1.0	04.0
2	1	06	2.0	12.0
3	2	10	1.8	18.0
4	3	14	1.6	22.4
5	4	18	1.6	28.8

Another alternative for shaft drilling and blasting available is benching rather than full face (Fig. 15.5). In benching the two halves of the shaft bottom are blasted alternatively. No cut is required. The method is suitable for wet conditions because a sump is automatically provided.

The advance per round is normally dependent on the facilities available for loading out the muck. Where loading is done manually, it may be convenient to blast shortholes and clear frequently. With mechanical loading, it is more convenient to pull as deep a round as possible, and advances up to 2 m are common. Generally speaking, the best results are obtained when the same cycle of operations is repeated in each shift.

In a round, it is advisable to drill the final trimmer holes on slightly smaller burden and more closely together than the preceding holes, and to load them relatively lightly. This reduces overbreak and the wall is not weakened by shock. Although the method of full face blasting has the advantage of offering a high rate of advance per round (up to 2 m), it is not particularly suitable where the condition of working is very wet. The benching technique of blasting has, therefore, become very popular in

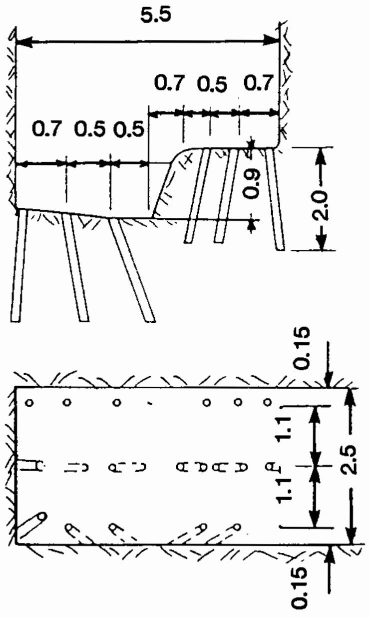


Figure 15.5 Benching for shaft drilling and blasting.

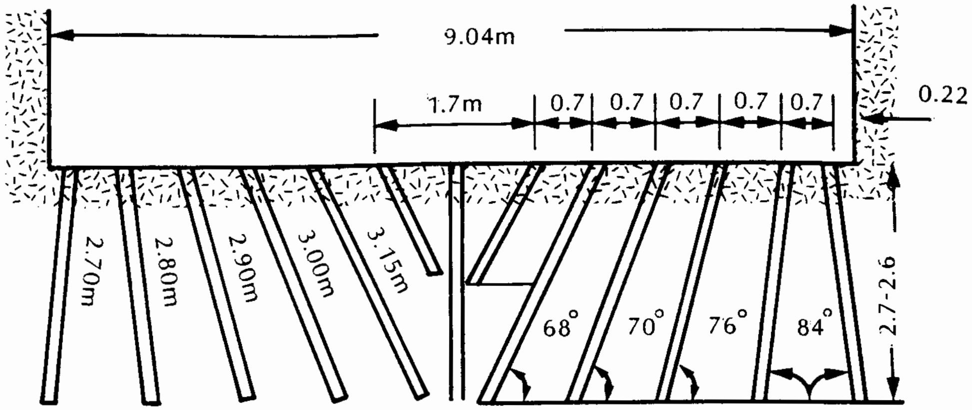


Figure 15.6 Pattern with central large diameter hole for shafts.

recent years and can be successfully applied where the working conditions are wet and also in case of shafts of large cross sectional area.

A recent development, originating in Scandinavian countries, has been to use a central larger diameter hole (holes) drilled parallel to the axis of the shaft. This is only possible with the application of drilling jumbo. Such a relief (burn cut) hole provides an additional free surface and increases the efficiency of blasting, reducing the consumption of explosives. An example of the drilling pattern has been provided by Unrug (1992) for a shaft of 9 m in diameter in high strength siltstone (Fig. 15.6).

ANFO is almost never used because shafts are usually wet. High explosives such as NG based explosives or more often, slurries are used instead. The average powder factor for a 3.0 by 4.5 m shaft is 3.25 kg of explosive per m^3 rock, varying from about 2 to 6 kg/ m^3 according to shaft size and rock strength.

15.3.2 *Raise blasting*

Raises in a mine connect levels at different vertical elevations with each other. Raises are driven either by conventional method using short holes and by use of long holes or by use of raise borers. Raises are prepared at angles from vertical to 55° which allows the broken rock to fall or roll down without assistance. The normal cross section ranges between 4 and 6 m^2, with rectangular or round profile.

With conventional raises height is kept to a maximum around 50 m and advance from each round up to 2.0-2.5 m is obtained though the use of conventional raising has decreased considerably. Hand held or airleg mounted drill machines or stopers are used for drilling upholes. Same machines are also used with more mechanised methods such as with the raise climbers (Bhandari 1967). Drilling upholes is a tedious and dangerous task. Falling rock and fumes are often a problem. Blasting in conventional raises is electric because the safety fuse may not allow the miner to get out of the raise before the round goes off, nor does it permit the precise timing demanded by modern blasting techniques. A poor round that shatters the face and walls is both hazardous and inefficient. In a drill round, the burden should be even on all holes and less than with headings. In timbered raises, the cut should be placed so that the rock

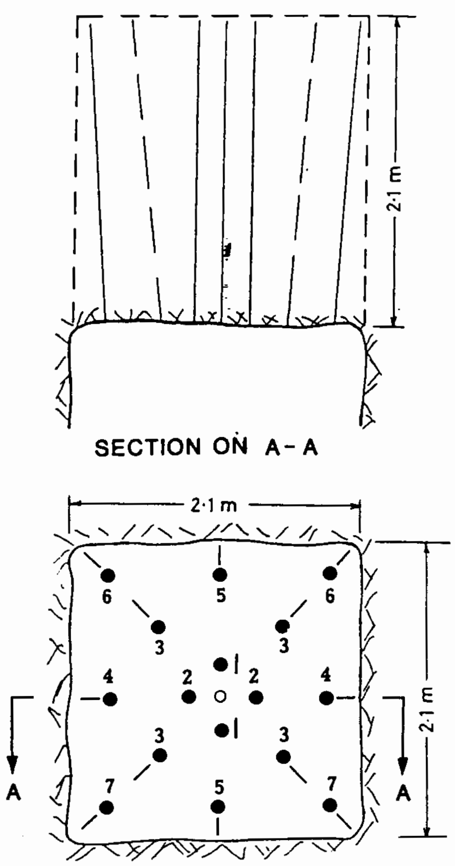


Figure 15.7 Pattern for a raise round.

is directed to the chute compartment. If not possible, the timber should be heavily bulkheaded. Both angled hole cut pattern or parallel holes cut pattern are used. A typical raise round is illustrated in Figure 15.7. This round, drilled to break 2.1 × 2.1 m, contains 17 holes of which 16 are loaded. The average depth of holes is 2.1 m and approximate advance is 2 m per round. Cartridged NG based explosives or slurry explosives could be used. Pneumatically loaded ANFO or cartridged explosive is occasionally used. Explosive consumption approximates 25 kg/m advance or 3 kg/m^3. The cut is placed to one side in order to protect the timbered manway which will normally be carried in the opposite end of the raise. Raises can be driven blind or can follow a pilot hole.

Raises using long holes 50-165 mm in diameter have become more common. Drill hole deviations limit raise drilling to less than 45 m. Drilling and loading upholes from down below can be dangerous and time consuming. Down-hole drilling and loading is safer and provides higher productivity. This is usually done with a central relief hole or using the vertical crater retreat (VCR) method.

Use of large diameter long holes started with blasting towards large holes. The drilling of the holes is usually carried out by means of relatively large drills of 51-75 mm. The wide hole is enlarged to 102-203 mm by use of special drill steels. Special guide sleeves are used to obtain greater precision in drilling. A pattern for 3 m^2 raise

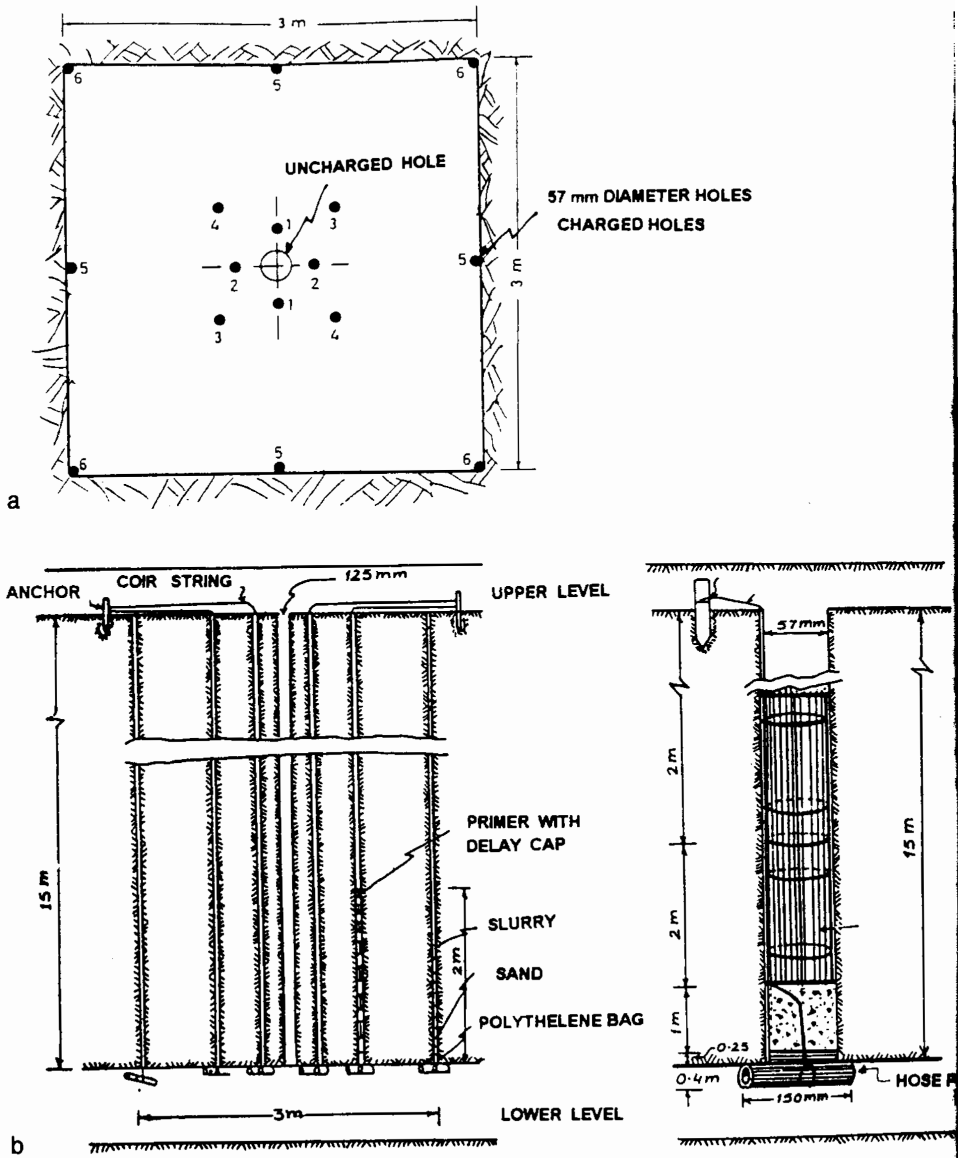


Figure 15.8 Pattern for a raise round with large empty hole.

with 57 mm drill holes and a 125 mm large diameter is shown in Figure 15.8. Blas ing is carried out in stages. If all the drill holes are blasted to obtain an advance p round which is normal for this area, there is risk of the drill holes becoming blocke and this complicates continued blasting. Figure 15.8b shows the sequence of blas ing. Charging is carried out by lowering the charges from the upper surface.

The VCR method has been adopted to drilling large diameter down-holes an blasting towards the free lower surface, providing a safer and more efficient mea of advancing raises from bottom taking advantage of the gravity, while equipme

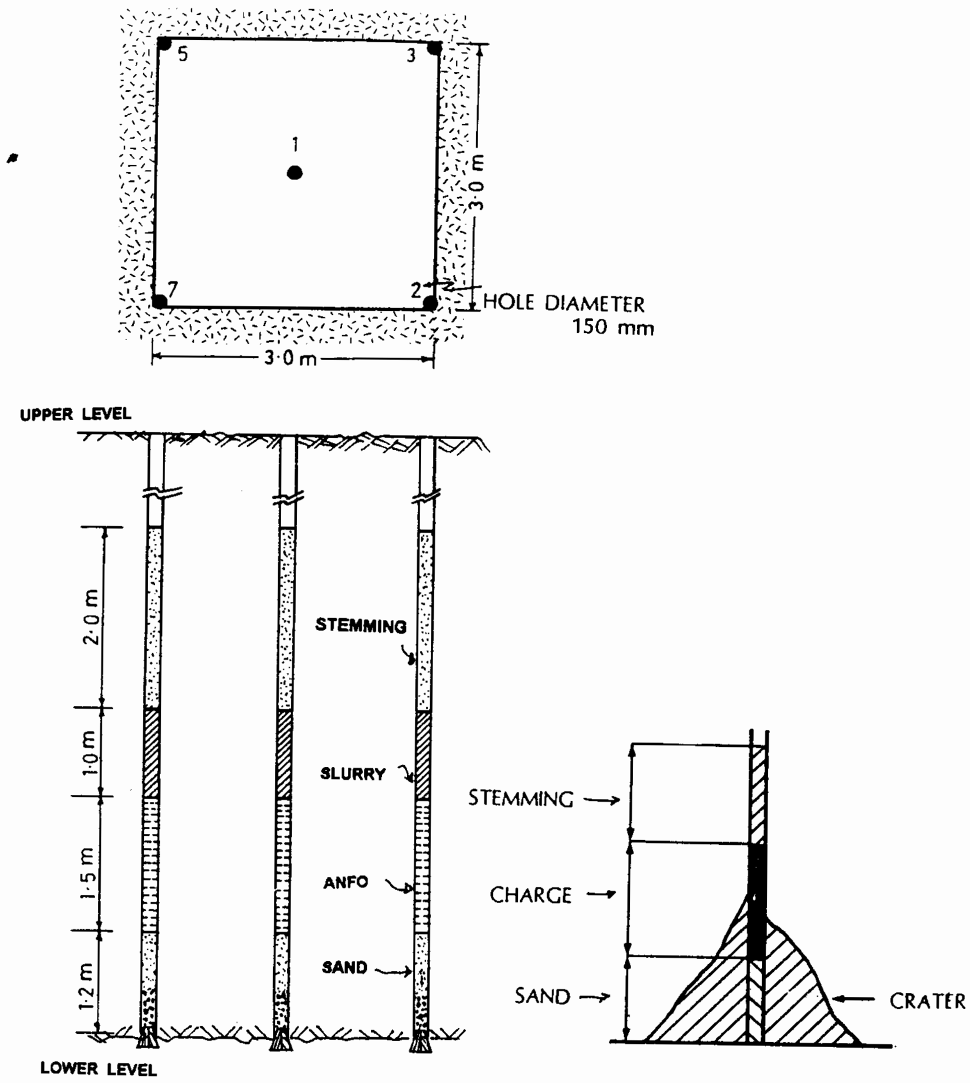


Figure 15.9 Pattern for raising with VCR.

and man remain at the top sill (Fig. 15.9). No large diameter centre hole is needed and there is less demand for drilling precision.

15.4 STOPE BLASTING

The initial development openings (drifts and ramps, shafts) are driven in much the same manner as small tunnels, with mobile drilling equipment, small holes, and short rounds. Subsequent production blasting of the mine stopes, the openings from which the bulk of ore is extracted, calls for much bigger rounds and often larger and longer

blastholes. A free face is prepared, and the ore is broken so that it falls downward, letting gravity assist in the flow and fragmentation.

The main considerations are safety, economy and selectivity. Dilution by waste lowers the grade of the ore and increases the unit cost, whereas leaving ore behind lowers the recovery and shortens the life of the mine.

Methods such as room and pillar stoping, shrinkage stoping, cut and fill stoping, employ shorter lengths and smaller diameter holes. Increasingly popular, particularly for large ore bodies in relatively stable ground conditions, are longer and larger blastholes (diameter 100-200 mm) drilled in a fan pattern or as in bench blasting.

15.4.1 *Stoping with short hole blasting*

Depending upon the method of stoping, parallel holes are drilled (Fig. 15.10) in row as follows.

– Holes drilled off the breast at an angle to break to a vertical face; for example, in stoping of narrow gently dipping ore bodies;

– Holes drilled upwards at an angle to the stope back to break to a vertical face; for example, in shrinkage stoping, cut and fill flat-back stoping;

– Holes drilled in horizontal planes to break to a horizontal face; for example, in shrinkage stopes.

These include horizontal cut and fill, inclined cut and fill, square set, shrinkage, open stope and combination of these methods. They each involve drilling of 29-33 mm diameter holes of 2.0-3.5 m depth. Holes are normally drilled horizontally, inclined, or vertically downwards. Horizontal (inclined) slices about 2.5 m high are progressively advanced along the face. In mechanised cut and fill stopes jumbos are used to drill 33-48 mm diameter holes up to depth of 3.0-4.0 m.

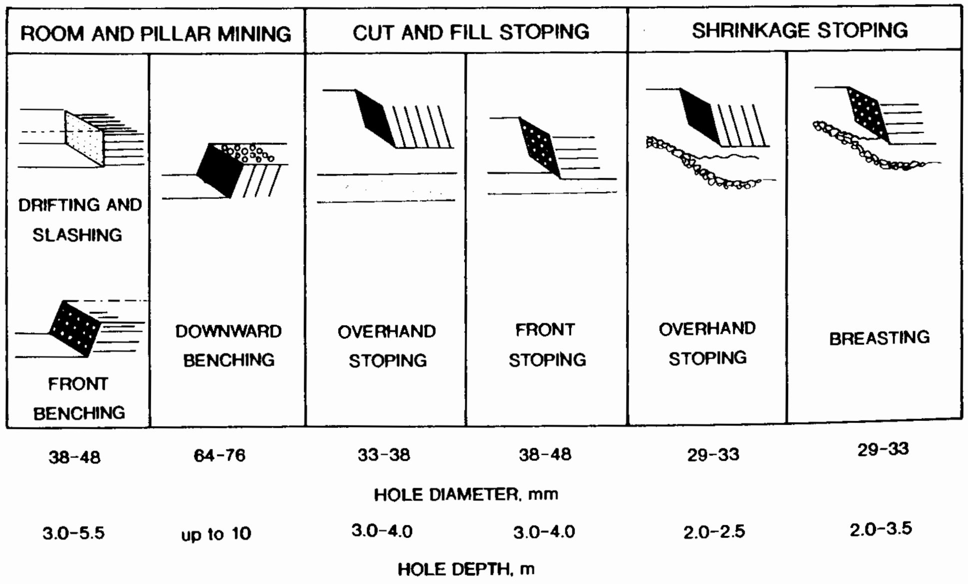


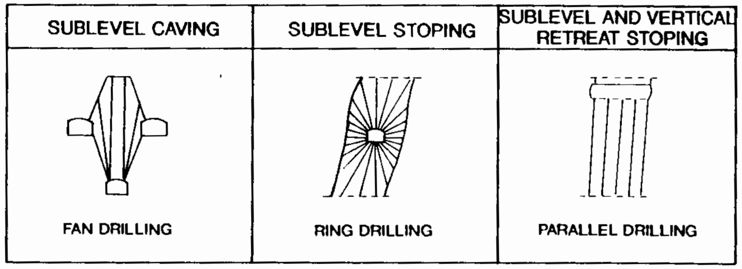
Figure 15.10 Drilling and blasting for stoping operations with short holes.

The blasting of vertical 'up' holes is also practised with mechanised light weight wagon drills, which drill 33-38 mm diameter holes up to 3.0-4.0 m depth.

15.4.2 *Stoping with long holes blasting*

The demand for increased rate of production coupled with improvement in drilling techniques has led to the wider use of long hole blasting for ore extraction. This technique renders mining of even low grade ores economically possible in certain cases. The cost of drilling long holes is no doubt much higher than for short holes but in most cases it is more than offset by savings in meterage per tonne drilled and savings in blasting cost. Extension steel drills with tungsten carbide bits are employed. By this means holes may be drilled up to 76 mm in diameter. Using 50 mm diameter cartridges, burdens of 2-2.5 m can be blasted; with 32 mm cartridges, the burden should be of the order of 1.25-1.8 m. The type of explosives used ranges from free flowing ANFO bulk loaded in dry holes to high strength gelatinous explosives for loading in wet condition. The explosive consumption in this type of blasting is of the order of 0.1-0.27 kg/t of ore blasted. With the radial system, rings of holes, usually in a vertical plane, are drilled from sub-levels. These holes are blasted successively into an initial slot formed by widening a winze or raise.

Sublevel or blasthole stoping methods include small diameter hole ring and fan drilling techniques or large diameter parallel holes drilled the entire stope length. Sublevels stopes are developed with small diameter holes approximately 76 mm in diameter. Fan upholes, angled 45-88° are drilled 12-20 m in length (Fig. 15.11). Stope width ranges 6-45 m and heights up to 80 m are common. An end slot raise is initially blasted from wall to wall to which successive slab rounds break. Fan or ring spacing along the drill drifts, or the burden distance between holes, vary 1.5-3 m, and hole bottom, or toe spacing between holes in a fan or ring ranges from 3-6 m. Hagan (1988) recommends the use of staggered drilling patterns between fans or rings with a spacing 3.5-4 times the burden distance. The use of this ratio should minimise sympathetic detonation, improve fragmentation, lower ground vibrations and provide good distribution of explosive energy. ANFO or water gels are commonly used with



HOLE DIAMETER	SUBLEVEL CAVING	SUBLEVEL STOPING	SUBLEVEL AND VERTICAL RETREAT STOPING
PNEUMATIC ROCKDRILL, mm	48-64	48-57	105-165
HYDRAULIC ROCKDRILL, mm	48-75	-	-
NORMAL HOLE LENGTH, m	10-20	10-25	30-60

Figure 15.11 Drilling and blasting for stoping operations with long holes.

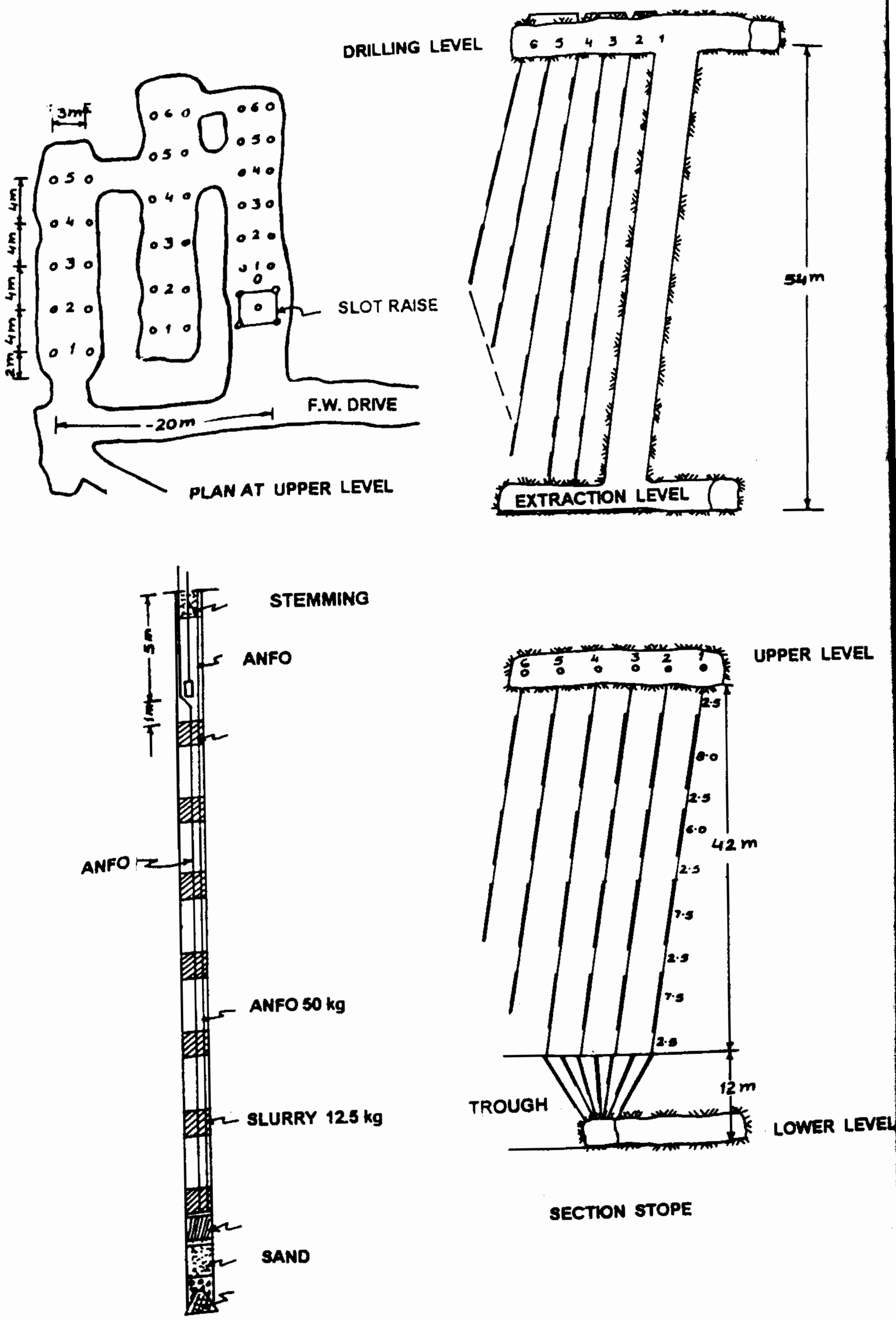


Figure 15.12 Large diameter holes drilling for stoping.

powder factors ranging from 1.2-3.6 kg/m^3. Short period electric or non-electric blasting caps with delay intervals of 25 ms are generally used.

Large diameter parallel holes 102-170 mm in diameter are often used in blast hole stopes without sublevels to height of 75 m (Fig. 15.12). The vertical limits of the stope are defined by an upper drill drift or top sill and an undercut drift. The upper sill is developed to the width of the ore body. Down the hole (DTH) drilling equipment is used to drill down holes 45-65 m in length. Two methods of blasting are adopted, one similar to bench blasting in surface mines and the other one adopts inverse crater blasting also called vertical crater retreat (VCR). The long hole blasting requires a raise and slot to be developed in addition to an undercut fan beneath each slab round. A 1.5 kg/t powder factor is typically used.

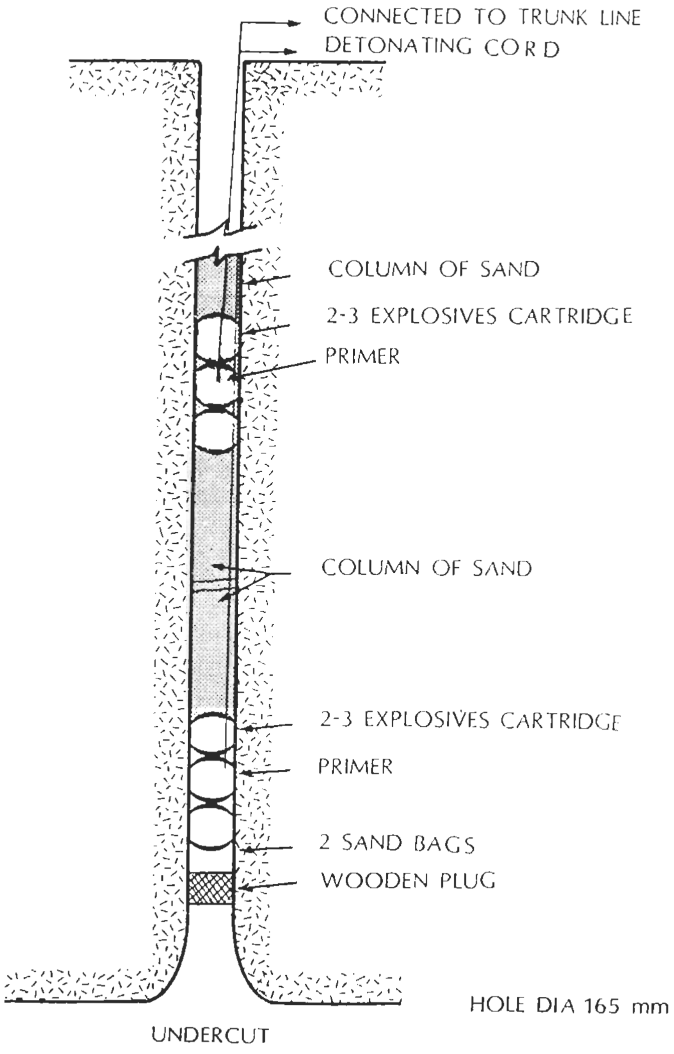


Figure 15.13 Charging a vertical crater retreat hole.

The application of crater blasting requires minimum development in the ore block. Its use depends on the size of the ore body and stability of wall rock. The VCR method has been shown to reduce pillar damage and overbreak, resulting in less ore dilution (Lang et al. 1977). Each round is loaded with a charge length to diameter ratio of 6 or less with parallel hole spacing designed 20 times the hole diameter dimension. Spacing and burdens range 2.4-3.0 m square or staggered on a 2.1 x 2.7 m pattern for typical hole diameter of 165 mm. Prior to blasting, each hole is measured and recorded. The next step is determining where to block the hole. There are several methods of blocking the hole at predetermined level. One of which is the process of securing two wooden wedges at the desired location near the bottom of the hole. The explosives are placed on the top of the block. Angles of blasthole determine where the hole is to be blocked. For 80-90° hole angle blocking height 1.2 m above the hole bottom is considered adequate. Holes are then loaded with explosives. Of the several methods in use, initiation with detonating cord in case of single deck of explosives is most common. Figure 15.13 gives two typical examples of loading of holes. Powder factors are ideally less than 0.5 kg/t. A vertical advance of 3.0-4.6 m is typical. After

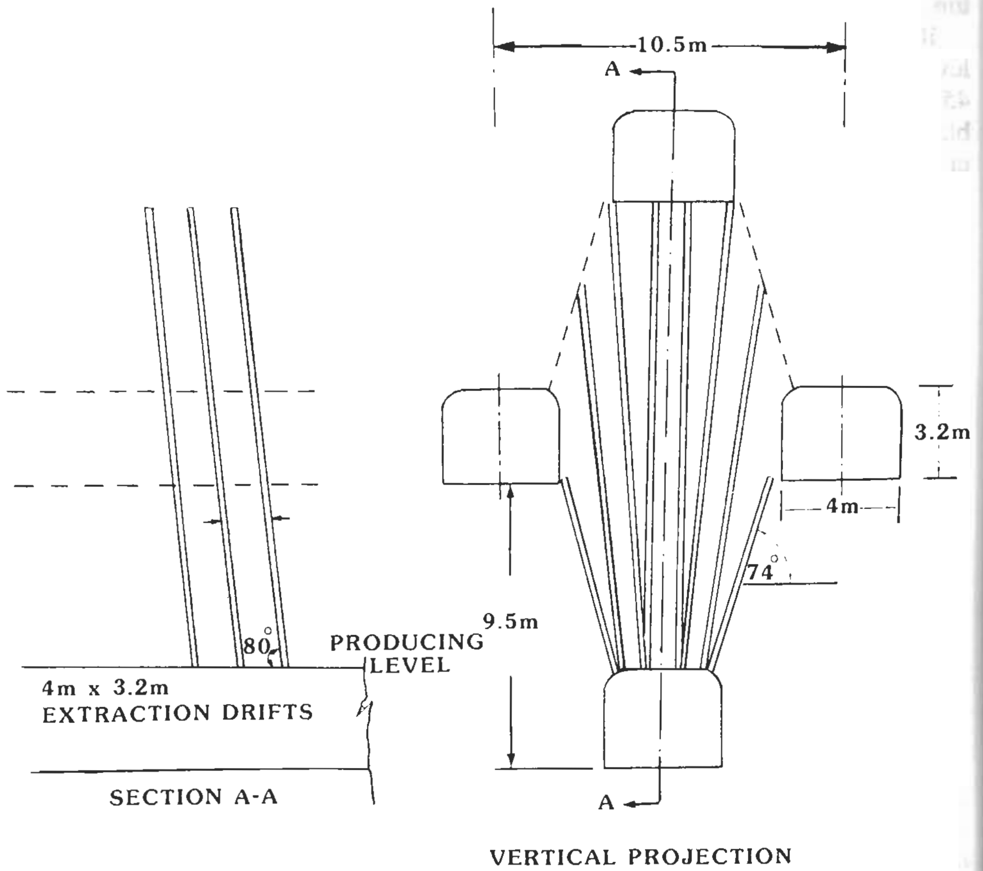


Figure 15.14 Fan drilling for sublevel caving.

determining the number of decks and column height of explosive, proper down hole delays are designed. All surface and down hole delays are often non-electric delays. Where multiple decks are used, a delay of 50 ms is used between decks. It is essential to use a delay pattern to ensure a free face for each hole to break to. Proper selection of delay times is essential to minimise overlapping of delay times and reduce vibrations. Explosive charge is to be kept within maximum per delay requirement. A major draw back with the use of this method is the difficulty in keeping holes open during blasting cycles in fractured and faulted ground.

Sublevel caving method employs the use of fan drilling using long, small diameter holes between sublevels or drill drifts to undercut and blast the ore zone. An initial slot is developed at the wall rock, and vertical uphole fans are drilled in a diamond pattern from sublevels in sequence (Fig. 15.14). Generally, eight holes, inclined between 70-85° toward the slot, are drilled. Hole diameters average 51 mm. Burdens and spacing vary from 1.2-1.8 m and 1.5-1.8 m respectively. Ring burdens affect the sublevel interval, pillar width and draw height. Just et al. (1972) suggest that ring burden depends on the balance between metal recovery and waste dilution from the previous ring. A large burden causes ore loss whereas a small burden causes dilution. Blasting is performed against broken waste rock as the wall rock caves. Powder factors range from 0.3-0.4 kg/t.

Block caving techniques require an initial development blast above the undercut level to start caving. Jumbos with 51-76 mm diameter holes drill fan rounds oriented 45° to vertical and 4.6-11.6 m in length. Fan spacing is generally 1.5 m. Secondary blasting is often required to dislodge oversize material or broken muck that has hung up within the raises.

CHAPTER 16

Specialised blasting operations

16.1 INTRODUCTION

Apart from primary production applications in mining and quarrying, blasting is carried out for many other specialised applications. These special applications need additional knowledge and changes in conventional blasting techniques. Each type of application requires different technique according to the objectives of the operations. In this chapter some examples of application of specialised blasting operations are given which illustrate changes needed according to the requirement.

16.2 BLASTING FOR DIMENSION STONE BLOCKS

In conventional blasting operations rock is fragmented to various sizes with a few large boulders which need to be further reduced in size by secondary breakage. However, there are many situations where large sized stones (boulders) are specifically required. Examples are dimension stones from marble and granite quarries, breakwater construction, dam construction and many other civil engineering operations.

When quarrying dimension stones, using drilling and blasting method, specialised blasting techniques need to be employed preventing damage to the block or to the remaining rock. If properly carried out this technique can improve productivity with considerable economy.

In conventional blasting as practised in quarries (even for removal of overburden in stone quarries) objective is to reduce rock to small size and the damage to the remaining rock is not of much significance. Whereas, in case of blasting for dimension stone significant difference is that the stone being extracted must not develop cracks and the surrounding rock must not be damaged.

The technique used is based on conventional blasting but needs knowledge about how rocks develop cracks when blasting is carried out and also use is made of special blasting materials and tools.

When an explosive charge is detonated in a hole, the explosive is converted into high pressure gases at a very high temperature in a few micro-seconds. As a result, rock near the hole walls is crushed, melted and cracked, and at some distance there is intense cracking and further away the number of cracks reduce but these can go to several hole diameters distances in any direction. The aim of blasting operations for

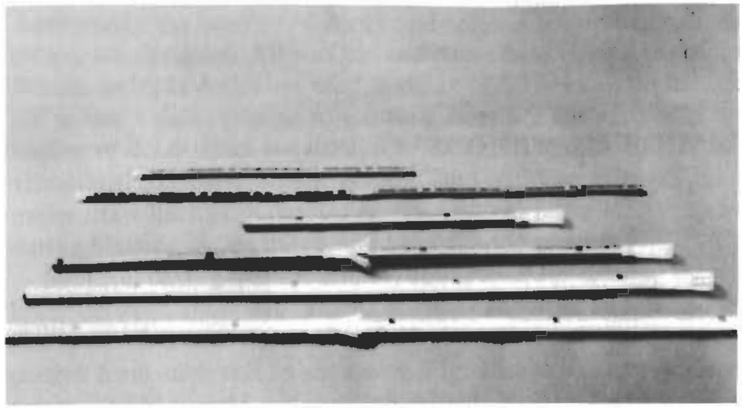


Figure 16.1 Pipe charges.

obtaining large sized stone blocks, therefore, should be to reduce the number and length of the cracks and if possible the cracks should develop only in the desired directions. If suitable rockmass is available then this is possible to achieve by the use of appropriate explosive materials, blasting parameters and/or by correct drilling.

16.2.1 *Explosive material*

Blasting powder/gun powder has been used in quarries since it has low detonating velocity and thereby has a mild effect on the rock (Anon 1976). However usage of gun powder has many difficulties. General purpose explosives such as NG-based, slurry explosives or ANFO which are used in mines and quarries produce greater stress waves and higher gas pressures which damage rocks. Modified compositions are available in various countries, which are used to reduce explosive energy generated for controlling the damage.

In Austria, Sweden and Finland, NG-based gelatine explosives formulated in special compositions and in pipe shape cartridges are used for controlled blasting (Fig. 16.1). They can detonate in 11, 17 or 19 mm diameter sizes in holes of 24-35 mm diameter. In Austria this explosive is called Donarit 2E, in Sweden it is sold as Gurit and Nabit and in Finland it is sold as K-pipe. These waterproof cartridges are automatically centralised by their specially designed connectors, making the charging operation quick and simple.

It is also possible to use ANFO mixed with hollow polystyrene granules. In USA special slurry explosives developed by Atlas Chemicals are used.

In place of such special explosives 40 g or 80 g/m detonating cord can also be used.

16.2.2 *Blasting parameters*

In Italy, large rock blocks (1000-3000 t) are separated from the main rock mass by using 25-35 mm diameter holes, 2-8 m in length with a hole spacing of 0.2-0.4 m (Mancini et al. 1993). Specific charge for PETN in the cord form is 18-50 g/m^3 with specific drilling being 0.6-3.8 m/m^3. Detonating strands in the hole vary between one

Table 16.1 Hole spacing for dimension stone blasting.

Type of rock	Charge	Hole spacing (m)
Granite	17 mm gelatine pipe	0.40-0.50
Granite	80 g detonating fuse	0.20-0.40
Diabase	17 mm gelatine pipe	0.40-0.50
Diabase	80 g detonating fuse	0.20-0.40
Marble	40 g detonating fuse	0.15-0.40

and three. Even 10 or 20 g/m cord can be used while using more than one strand.

A typical hole spacing for dimension stone blasting is given by Gustafsson (1981) Table 16.1.

Hole spacing and charges must be adopted to the result of test blasting in each particular case. Drill holes are charged with the pipe charges up to 75% of the hole depth. The detonator must be attached directly to the charge as no other primer charge can be used. No stemming is to be used except for the locks of the pipe charges. Millisecond delay detonators with the shortest possible delay between the drill holes provides the mildest effect on the surrounding rock, however, this means that in tough rock types it may be necessary to have rather closer hole spacing than that of shown in Table 16.1. While using blasting powder the distance between 35 mm holes is between 0.5-0.7 m. The depth of holes varies from 0.8-1.5 m.

It is important to observe the following precautions.

1. The burden in front of drill holes must be small;
2. If a block of the rock is to be blasted loose, it must not be constricted in front or in sides;
3. Drilling precision must be good;
4. In carrying out test blasting it is advisable to start in sections where the rock is less valuable.

Only after calculating the suitable hole spacing and the charge values, blasting must be planned so that the geometrical conditions are correct for breakage with the least possible constriction.

A well developed method of quarrying of granite blocks practised in Finland (Smith 1987) uses the above mentioned techniques as detailed below:

Before production commences overburden and loose material is removed by the excavator. In the first stage of production, the sides of the block, which may be as large as 4000 cm are freed by making a slot. A slot drill is used to drill holes up to 6 m depth. This operation can also be done by flame torch cutting. Flame cutting has a disadvantage of high energy consumption and poor dust and noise control. Also flame cutting can be applied to rock with a high quartz content only.

The slot drill uses a pneumatic drifter to drill a series of pilot holes of 64 mm to a depth of 6 m at intervals of 114 mm using a special guide. This row of holes is opened to a slot by drilling the spaces between the pilot holes using the same diameter bit. The slot achieved is both smooth and straight. The drilling equipment travels along the frame without need for separate measurement of hole spacing this is taken care, by manual mechanism. Power for jacking up the frame and for machine movements is provided by pneumatically driven hydraulic power pack.

The back line vertical holes and horizontal lift holes are drilled full depth at a pre-

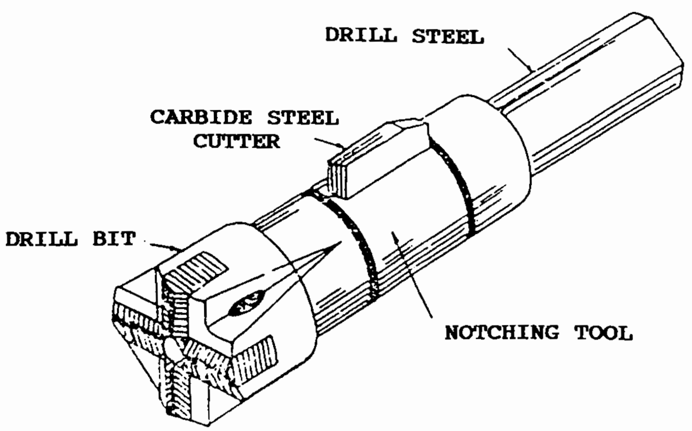


Figure 16.2 Simultaneous drill and notch tool.

determined spacing using a track drill. Hole diameter is from 27-32 mm according to rock density, hole spacing and burden. Absolute drilling accuracy is fundamental to the success of this method.

The holes are charged with K-pipe charges. Explosive charge used is about 60 g/m^3. Detonation of the vertical back line and horizontal lift holes is simultaneous moving the block about 150 mm of the face.

In the secondary drilling stage, the loosened block is subdivided into smaller blocks of about 30 m^3 again using the drill hole guide to achieve a hole spacing of 250 mm. These holes are charged with K-pipe charges with a charge density of 30-80 g/m^3 and are blasted and tipped down on to a sand bed which prevents damage. These 30 m^3 blocks are again subdivided into smaller blocks according to the loader capacity. The operation is carried out by using a guide for drilling followed by splitting using wedges or detonating cord. The blocks are then taken by fork equipped wheel loaders to the dressing station for squaring to the final shape.

If smaller sized blocks are to be produced in stone quarries then jack hammer with suitable drill guide with predetermined spaced pipe guides are used with a detachable drill bit (Bhandari, 1990a). This ensures accuracy of drilling.

To control growth of unwanted cracks in blasting it has been suggested that eye shaped holes be used, that is a hole with a notch (Bjarnholt et al. 1982). The notch acts as stress raiser and cracks first originate from these notches and as gases penetrate, the cracks progress up to greater distance. Additional notching tool is attached for making notch while drilling or after drilling is over (Fig. 16.2).

16.3 PRODUCTION OF ARMOUR STONES

In many situations such as breakwaters and dam construction large sized fragments (boulders) need to be obtained by blasting operations. In break water construction stone boulders (Armour stones) along with a certain proportion of medium and small sized fragments are needed. Often a million cubic meter or more rock is required in a pre-fixed size distribution.

The blast design for obtaining large sized rock is not an easy task. Blasting for

large sized fragmentation can be just as difficult to obtain as small size fragmentation. It requires the skills of an experienced person to blast lightly enough to maximise generation of large sized fragments but strongly enough to permit the rock to be separated. Further, the design of blasts for obtaining prefixed fragment distribution is also a difficult task. So far no conclusive methods for the designing blasts for specific size distribution and evaluation of blasting results have been devised.

There are several approaches for determining the specific charge and required charge distribution. However, once the necessary explosive charge is established, these tests require the blast design to be generally standardised throughout the programme. Several changes in conventional blasting designs need to be tried to obtain coarse fragmentation. Some changes suggested by Gustafsson (1981) and Wang et al. (1992) are:

- Low specific charge;
- Hole spacing less than the burden;
- Instantaneous firing;
- Firing one row at a time;
- Combining above depending on the circumstances.

Low specific charge at the limit, needed to tear the burden, produces large sized rock. It is advantageous for the column section not to be too short since, if it is, the bottom charge shatters the rock. If the round is fired with limiting value charges of 0.20-0.25 kg/m^3 it is advisable to fire one row in each round.

If the burden is chosen to be considerably larger than the conventional, then the fragments will be coarser (Bhandari 1975b; Bhandari & Badal 1990a). The stress waves are not allowed to participate and hence the fragmentation obtained will be coarser (Bhandari 1979a). Instantaneous or short delay firing normally results in coarser fragments than normal millisecond delay initiation. This is due to less tearing between the drill holes (Bergmann et al. 1975).

Tanwar (1990) and Bhandari & Tanwar (1993) describe a series of test blasts carried out on 6-9 m high bench in granite to find out if 25% fragmentation in the size range of Armour stone of 10-20 tons, 25% stones in the range of 1-10 tons and 50% quarry run smaller than 1 tonne can be obtained.

Test site in granite rock was of category I according to Deere's rock classification and geomechanical properties of the rock were: Density 2.65 kg/m^3, compressive strength 116.7 MPa and tensile strength 8.41 MPa. The rock had a few visible joints.

To produce desired size of Armour stone (up to 2 m^3), a low specific charge was required to facilitate minimum cracking in the rock formation. However, the explosive charge was to be enough to crack and move the bottom portion of the rock mass. Various specific charges from 0.25-0.35 kg/m^3 were used for judging what would break the rock suitably. While using 100 mm diameter holes for the hard granite the burden and spacing were planned to be 3.0-4.5 m and 3.0-5.0 m respectively. These were slightly different in practice. Explosive used for the blasts was a MMAN based watergel in 83 mm cartridge. Fragmentation obtained from one of the blast is shown in Figure 16.3.

The main conclusions which could be drawn.

1. By reducing the specific charge for the same spacing/burden ratio the percentage of coarse fragmentation also increased considerably.
2. By increasing the burden, the percentage of coarse fragmentation also increased.

Figure 16.3 Fragmentation during armour stone production.

3. By decreasing the spacing/burden ratio then also the percentage of coarse fragmentation increased.

The recommended design parameters based on the study were that the spacing/burden ratio should be less than 1 and it was suggested to be 0.80 and specific charge should be 0.32 kg/m^3. The burden was recommended to be around 4.5 m. For the second site the spacing to burden ratio was suggested to be 0.85 and specific charge should be 0.35 kg/m^3. The recommended burden was 3.5 m.

16.4 CASTING OVERBURDEN BY BLASTING

When overburden is stripped above a coal seam or a shallow tabular ore body, it is the normal practice to remove the fragmented material by mechanical means. Blasts are generally designed to give good fragmentation for handling and sufficient displacement to give a loose, highly diggable muckpile of the required profile.

A method of overburden displacement has been known and practised for sometime termed as overburden casting by blasting, throw blasting, explosive mining or control trajectory blasting (CTB). The method may be defined as the use of explosive for the purpose of fragmenting and providing displacement of the overburden to the final spoil pile. Maximum forward displacement is the primary requirement in the blasts. The purpose of casting by blasting is to utilise explosive energy to dislodge and move as much of the overburden as possible across the pit to the spoil dump so that reduced rehandling is required (Hagan & Aus 1979). Principally this is achieved by significantly increasing the powder factor. The increase in powder factor is achieved by using a combination of larger diameter holes, more holes and more explosive, use of higher density and energy products or increased explosive volume.

In opencast mining, the costliest operation is excavation. By adopting explosive mining savings are achieved by decreasing excavation cost by the reduced amount of material to be excavated though marginally increasing the drilling and blasting costs. Only after optimising the cast blasting it will be possible to compare cost effective-

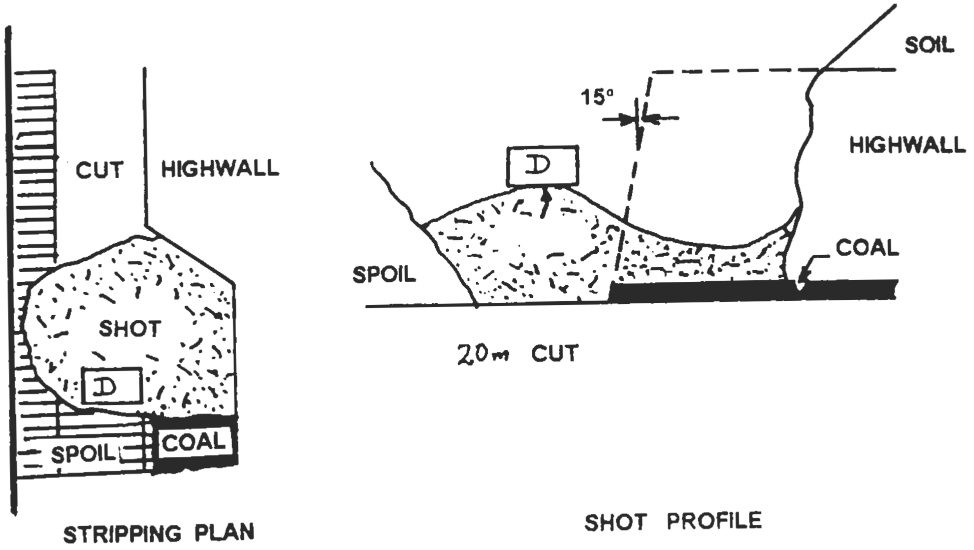


Figure 16.4 Typical cast blasting in a strip mine.

ness data with those for conventional blasting. Figure 16.4 represents typical cross-section of a strip mine illustrating how cast blasting decreases the amount of work required by stripping equipment.

16.4.1 *Advantage*

Advantages of cast blasting over conventional blasting and mechanical handling are as follows:

1. Casts up to 35-60% of the overburden directly on to the dump-pile without the use of stripping equipment and thus increasing a stripping capacity by similar amount.
2. The increase in capacity can be obtained with no large outlay of cash if a blasthole drill is already available. The comparatively small amount of additional money that is spent on explosives and drilling is responsible for the increased stripping capacity.
3. As an added advantage, the increased fragmentation resulting from the heavy blasting produces faster, easier stripping with less downtime and longer equipment life.
4. By using larger diameter drill holes and less stemming, the required powder factor can be obtained with minimum increase in drilling cost.
5. Less capital investment is required for excavators. This is one of the strong advantages for a new mine or a mine going for major expansion.
6. Less berm width is required for the benches because of smaller size machinery.

16.4.2 *Factors affecting application*

The factors which determine if cast blasting is feasible or not, can be classified in two general groups, natural conditions and methods of operation and equipment used.

1. Natural conditions

Favourable

1. Most advantageous in heavy overburden, but applicable in any depth.
2. Most desirable in moderately hard and consolidated overburden where heavy shooting is required for fragmentation.
3. Best in contour stripping, but also applicable in level stripping.
4. Like any type of blasting, most successful where ground conditions are dry.

Not favourable

1. Not practical in soft, unconsolidated overburden where no shooting is required.
2. Not practical where deposits are such that mining must be done by open pit method (i.e. where overburden is trucked away or moved by scraper).

2. Method of operation and equipment used

Favourable

1. Most successful where deep narrow pits can be worked and spoil can be stacked high (such as found in dragline operations).
2. Best in small to medium sized dragline operations, or generally, most successful where small to medium sized equipment is used.
3. Less advantageous, but still of value in small shovel and dozer stripping.
4. Can be used in any size operation except where pits must be kept wide (large shovels for example require wide benches).
5. Most advantageous where long, narrow benches can be used.

Not favourable

1. Not applicable in scraper operations, where wheel excavators are used, or where overburden is trucked away (eliminates open pit mining).
2. Where large shovels are used, thus necessitating wide benches for manoeuvring.
3. Where no shooting is required or where heavy shooting may cause damage.

16.4.3 *Design parameters of blast casting*

Blast design areas which need to be considered are: energy, time and mass. Each of these areas need to be considered as the blast casting results are greatly influenced by the blast design parameters. The parameters which are important in the design are:

- Explosive energy;
- Highwall height to width ratios;
- Hole diameter and inclination;
- Burden to spacing ratios;
- Burden velocities;
- Initiation timing relations.

Explosive energy

A number of different explosives can be used. An explosive with high gas energy is more effective than explosive having higher shock energy. Chiappetta et al. (1990) give an example to show influence of increase in powder factor on a 23 m highwall, overlying on a 7 m coal seam in a pit of width 50 m, on casting. Cast refers to the percentage of overburden that does not require rehandling. Powder factor between 0.25 and 0.4 kg/m^3 essentially fragments the material in place with very little cast. Increasing the powder factor to between 0.45 and 0.60 kg/m^3 increases casting to about 20 and 35%. Further increases in the powder factor between 0.65 and 0.80 kg/m^3 will result in casts of about 42-46%. Denser or higher energy explosive will be advantageous as less number of holes will be needed.

Highwall to pit width ratio

Through the use of projectile motion equations, the actual cast distance (i.e. where the material hits the pit floor) can be calculated for each marker location on the bench face using the following equation:

$$R = V_o \cos\theta \left[\frac{V_o \sin\theta + \sqrt{[(V_o \sin\theta)^2 + 2gH]}}{g} \right]$$

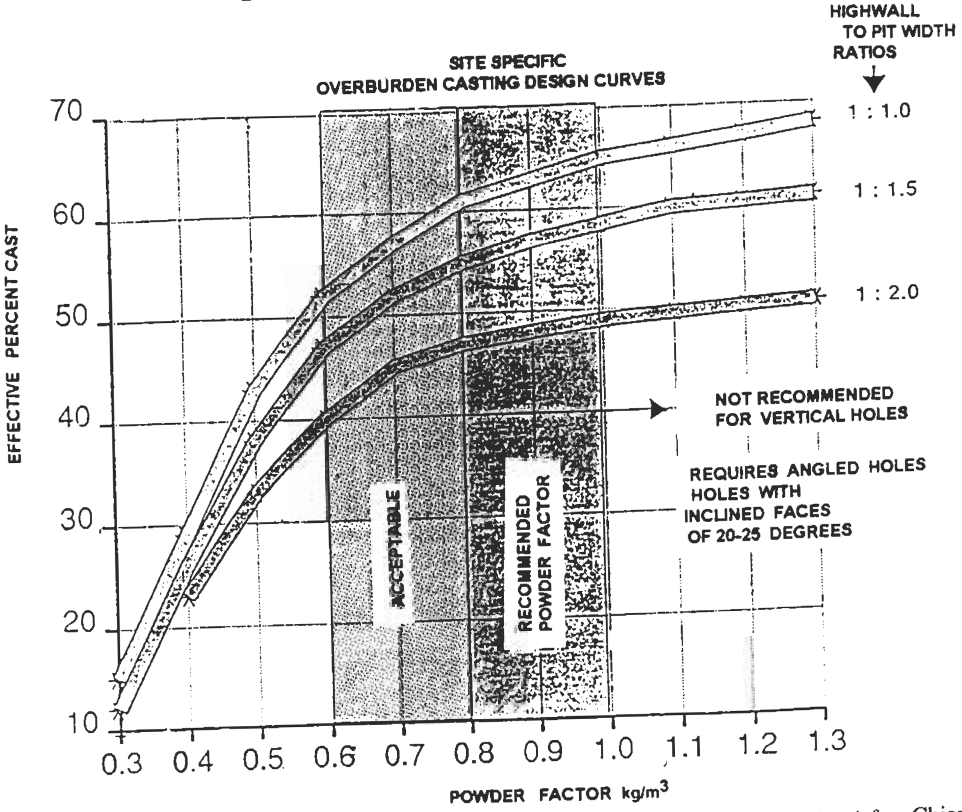


Figure 16.5 Effective percent cast expected for different highwall to pit width ratios (after Chiappetta et al. 1990).

where R = cast distance in meters, V_o= Initial burden velocity in meters per second, θ = Ejection angle, H = Height of burden above floor in meters, g = Acceleration due to gravity = 9.8 m/s^2.

According to Petrunyak & Postupack (1983) minimum highwall height that can be cast economically is twelve meters. For shorter highwalls the cost of drilling the larger number of smaller diameter holes needed to maintain required powder factor outweighs any cost savings. Smaller highwalls in the range of 12-18 m need more energy in the range of 0.8-1.1 kg/m^3.

Chiappetta et al. (1990) showed the effect of highwall height to width ratios. For constant pit width as the highwall to pit width ratios increased from 1:1.0-1.20 the resulting cast dropped from 63 to 47% and with further increase in ratio expected cast dropped below 30%. Figure 16.5 illustrates the effective percent cast for highwall to pit width ratios of 1:10, 1:1.15 and 1:1.20 for powder factors ranging from 0.30-1.3 kg/m^3.

Hole diameter and inclination

Larger diameter holes are advantageous as greater volume is available for placing the explosives and thus there will be reduction in drill meterage needed. Of course, larger diameters produce greater size fragments but as long as shovel or dragline employed is able to handle the material this consideration is not of importance. Chiappetta et al. (1983) recommend a borehole diameter corresponding to 10.5 mm for every 1 m of overburden. Worsey & Giltner (1987) suggest large diameters of up to 15 mm for every 1 m of overburden may be used successfully.

Commonly vertical boreholes are used on vertical faces. With inclined faces it may be necessary to use high bulk products in the toe and/or use inclined holes up to 20° or more off the vertical, depending on bench height and angle. Boreholes be kept parallel to the highwall face, thus providing a uniform burden along the hole, uniform burden movement and reduction in the occurrence of vents which adversely affect the casting potential (Giltner & Worsey 1986). Angled holes give greater throw, with a theoretical maximum at 45°. The residual highwall angle will depend on the excavation equipment and how it is operated. For example, a shovel will produce a steeper highwall than a dragline. As the highwall angle changes, the drilling angle should be altered accordingly. It is usual to maintain a highwall angle that is stable and allows the stripping equipment to be operated easily.

Burden to spacing ratio

The burden to spacing ratios used in practice have been either those used in conventional bench blasting in which spacing is generally greater than the burden or those used for some overcasting operations where spacing is lesser than the burden. Chironis (1981) reports burden to spacing ratio varying between 1:0.4-1:0.9 for successful casting operations. Giltner & Worsey (1986) studied blast casting operations with six different operations using burden to spacing ratios of 0.53-1.0. Those who prefer to follow conventional bench blasting ratios, a number of observations and/or recommendations for burden to spacing ratio fall between 1:1.2 and 1:1.7 for satisfactory casting (Crosby et al. 1982; Chiappetta et al. 1983; Petrunyak & Postupack 1983; Dupree 1986). Burden to spacing ratio between 1:1.2-1.5 seems to be more appropriate with staggered hole between rows. A larger ratio will cause blastholes to fire independently.

Table 16.2 Drill pattern designs for overburden casting for 50 m bench width-staggered pattern burden × spacing (m × m).

Hole depth (m)	Powder factor kg/m^3				
	0.6	0.7	0.8	0.9	1.0
15	6.6 × 8.5	5.8 × 8.0	5.8 × 7.0	5.5 × 6.6	5.2 × 6.3
23	6.7 × 8.7	6.1 × 8.2	5.8 × 7.5	5.5 × 7.1	5.2 × 6.7
28	6.7 × 8.8	6.1 × 8.3	5.8 × 7.6	5.5 × 7.1	5.5 × 6.8
33	7.0 × 8.5	6.4 × 8.0	6.1 × 7.3	5.8 × 6.8	5.5 × 6.5
38	7.0 × 8.5	6.4 × 8.0	6.1 × 7.3	5.8 × 6.9	5.5 × 6.5
43	7.0 × 8.6	6.4 × 8.1	6.1 × 7.4	5.8 × 6.9	5.5 × 6.6

Burden velocities

Greater burden velocity means higher percentage of cast. In order to adequately displace and move material to the spoil pile burden velocity recommended is between 12-27 m/s. Burden velocity higher than 27 m/s leads to energy losses through gas venting at discontinuities and results in air blast and flyrock problems. To achieve suggested burden velocity, depending on the highwall height and powder factor used, burdens should be designed between a minimum of 5.2 m and a maximum of 7.0 m (Chiappetta et al. 1990). As an example, assuming use of ANFO and maintaining burden to spacing ratios between 1:1.2-1:1.5 the suggested drill pattern has been given as in Table 16.2. With other explosives such as aluminised ANFO, it may be possible to increase burdens up to about 20% without unduly increasing burden velocities. With other higher energy explosives necessary adjustments need to be made.

Initiation timing relations

For optimum cast results, all holes in a row must be fired simultaneously. It is believed that simultaneous initiation give enhancement effect. Delays of 15 ms or less between holes in a row have very little effect on cast results. In situations where there are ground vibration restrictions, delays up to 25 ms between holes in rows can be utilised. Utilising delays greater than 25 ms between holes in a row can create differential burden displacements.

Timing between rows have substantial effect on the casting results. Shorter timing can cause back break and thus create uneven front row burdens for the next blast and, in many cases, cause severe coal damage. Timing required to displace the burden is dependent on the burden mass, explosive energy and dynamic material response to the explosive simultaneously. This response time will increase with the increasing burdens, with less energetic explosives and when blasting in soft unconsolidated materials. Delay times between rows need not be kept constant. The effective delay should be incrementally increased starting from the first row and going towards the last row.

It is important to have accurate delay caps to maintain desired timings. Alternately one should anticipate inaccuracies and scatter and accommodate it in the delay design.

16.4.4 *Practical problems*

The increase in energy in cast blasting shots may lead to several problems: The high degree of blast disturbance associated with the use of high explosive charges can create stability problems, specially when using vertical faces. It is useful to use angled holes so that the final face will be approximately of the same angle. Also controlled blasting techniques such as pre-splitting, air deck techniques can alleviate the problem to some extent.

In addition to stability problems the next blast may also face problems of providing consistent front row burden. While digging with shovel or dragline care needs to be taken in this regard, as the modern powerful equipment can dig into unblasted portions also.

Since higher powder factors, low spacing to burden ratios and smaller delay intervals are used ground vibrations, air blast and flyrock problems may be faced. Adequate controls are required to overcome these problems (Chapters 18 and 19).

16.5 PILLAR BLASTING

Pillar recovery operations in underground mining involves blasting large quantities of rock underground and some blasts have been over 1,000,000 tons. During mining operations a lattice of pillars is left behind in underground mines where sublevel open stoping, shrinkage stoping or other methods not involving filling or caving operations are practised (Fig. 16.6). As mining progresses with time, in depth and extent, these pillars have to bear more load. In due course of time these pillars weaken and cracks develop in the pillars. Self caving of such pillars can not be allowed as it may result in devastating air blast and its consequences. Uncontrolled failure of pillars would cause a sequence of failure similar to removal of one card in a house of cards situation. A systematic recovery of pillars in a controlled manner is essential to deal with unstable mine openings and slabbing of the rock mass. The interdependence of these pillars usually necessitate that several such pillars – rib and sill pillars be blasted simultaneously involving very large quantities of rock and explosives. Some of the recent mining methods have involved extraction of these pillars simultaneously along with the stoping operations. Since the quantities of rock and explosives are very large, meticulous planning and execution is needed.

Once the decision to recover a pillar or a group of pillars has been taken, question arises as to which pillars are to be selected for drilling and blasting? When and where to start depillaring? How the recovery operation should be carried out and lastly the safety precautions to be observed while conducting the operations and also immediately after the blasting.

Normally those pillars are selected which are about to fail or where appreciable cracking of rock has weakened the pillars. Another vital factor in selecting pillars for recovery is that blasting operations thereof must not affect the workings of the rest of the mine. Adequate free faces on the pillar sides must be available. A careful examination of presence of voids to receive expanded volume of blasted ore must be made before the recovery operations are taken in hand. Usually the exact survey of pillar outline is not available and the entire blast is designed on the approximate profile.

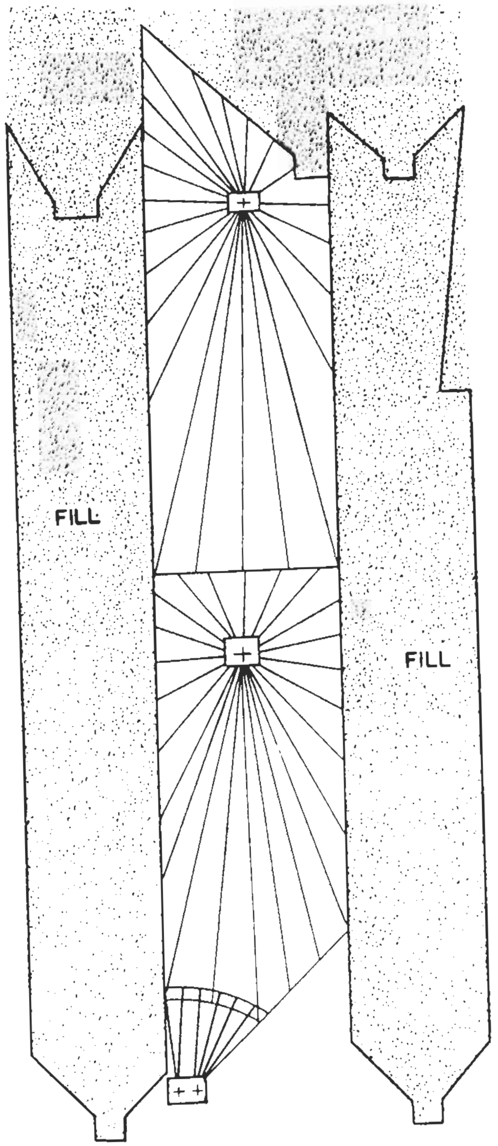


Figure 16.6 Drilling pattern for pillar extraction in open stoping.

Rock mechanics studies are essential to consider and see effect of leaving the pillars in situ vis-à-vis extracting particular pillar. Stress analysis of mine openings and structures is an essential step. The pillars extracted may cause stress accumulation in some other sections of the mine leading to deleterious conditions. A pillar must also be extracted before the rock starts flowing plastically. This would pose technical problems while drilling the pillar, resulting in frequent jamming of drill steels, increased breakage and loss of bits and steels.

The interdependence of these influences usually necessitate that several of such pillars be blasted simultaneously, making large blasts the only practical approach.

Depending on the technology used in the mine production relevant drilling equipment are employed.

16.5.1 *Design of pillar blasts*

Considerations involved in the blast design are pillar geometry, voids available, drilling parameters, maximum charge allowed per delay, charging technique, etc.

Availability of free faces
The geometry of rib and crown pillars under consideration is an important factor in designing the blasts. These should take into account the availability of ample free spaces for accommodating broken muck and thereafter, possibility of withdrawal of muck.

Choice of explosives
In pillar blasting three basic charging and loading techniques are employed.

In holes angled above horizontal, loading with pneumatic cartridge loader is often practised.

Holes 30° below horizontal are generally larger diameter which can be loaded with pumpable watergel/slurry or emulsions. Pillar holes 65° to vertical are hand loaded with a stiff watergel. Often these holes are the deepest in the pillar recovery programme, ranging from 30-40 m deep. They are also subjected to ground failure and cracking due to shifting of the pillar structure. For this reason often stiff, dense watergel or emulsion need to be utilised.

Though ANFO usage is common but separation of diesel from AN and dissolving of ANFO in watery holes is common.

Charging and connecting up periods could be very long. Thus explosives and initiating system could remain charged up for very long time.

Initiating system
Use of initiating system such as non-electric system will be needed for pneumatically charged holes. The effect of vibration levels need to be controlled as underground structures are exposed to high vibration level which will decide amount of explosive per delay.

Mains firing is often practised as blasting is often carried out from the surface. Dual circuits are provided as a safety measure against misfire-lest a fault develops. Chances of misfires and cut-offs exist and hence necessary precautions are needed.

Precautions
It is essential to prepare detailed pre-blast checklist and circulate amongst the staff for compliance. This may include some activities as follows.

1. Clear all machines and material from all levels to avoid danger due to air blast.
2. Stop all the fans including auxiliary fans.
3. Open all the doors in all the levels.
4. Stop the compressor (no pressure in the pipe lines). All the pumps to be stopped.
5. Keep the production shaft cage on keps.

6. Keep the skips at loading and unloading places.
7. Keep all the sumps empty.
8. All underground power to be cut off before blasting.
9. All diesel vehicles to be parked at confined places.
10. Special guards be posted at all entries to prevent entry.
11. Rescue teams are kept ready for any eventualities.

Activities after blasting operations

1. Close the doors of openings which are kept closed during normal workings.
2. Start the main ventilator.
3. Samples of blasting fumes be taken for assessing the concentration of toxic gases.
4. Inspect all the workings with section incharges and do not allow work to proceed till all the workings have been found free of any spalling or activity. The delayed failure can be expected after mass blasting operation and should be considered as possibility and precautionary steps be taken accordingly.
5. Inspection team to provide final clearance before work resumes.
6. Complete blast records be kept for future guidance.

16.6 CANAL/TRENCH BLASTING

The excavation of trenches is necessary for laying oil pipelines, cables, and water and sewage systems. Pipeline trench blasting generally refers to bench width of not more than 2 meters. Trenching in rock differs from normal benching in that it involves more drilling with closely spaced holes and higher explosives consumption. The reason for the increased consumption of explosives is that the rock is more constricted (difficult to blast free) at the bottom and sides of the trench, where there is no free surface. A typical drilling pattern is given in Figure 16.7. When pipeline trench blasting is carried out, the inclination of the drill holes is of vital importance. In deep holes proper inclination of holes allows tearing of the bottom rock and swelling of broken rock. Small diameter holes are normally used. The problems of ground vibrations must be taken into consideration when trenching in urban areas.

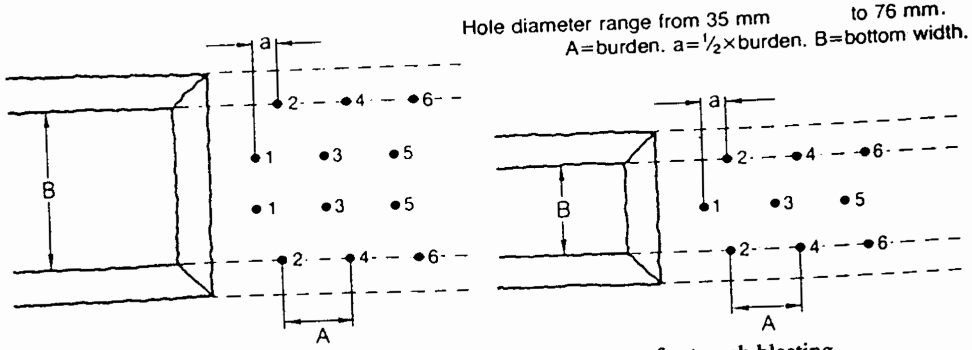


Figure 16.7 Drilling pattern, burden distances and delay sequence for trench blasting.

For narrow pipeline and cable trenches up to 0.6 m wide a single row of holes is drilled up to the grade level. Spacing is determined experimentally in conjunction with type and quantity of explosive selected. Holes are fired sequentially towards a free face with short delay detonators.

In harder ground or with wider trenches up to 2 m wide, two or more rows of holes are drilled, sometimes in a staggered pattern.

The explosive charge in kg/m^3 is higher than in bench blasting and it needs to be well distributed up the hole at a low density to avoid excessive overbreak. Holes should be well stemmed.

Near built up areas necessary covering need to be used. In sites away from built up areas hole diameters can be increased.

While carrying out pipeline trench blasting, it is important that drilling and blasting is carried out such that smallest possible overbreak results. Overbreak means not only more material to be mucked out but it also involves greater cost of refilling. Conventional trench blasting is often seen to be resulting in 100% overbreak. By using controlled blasting techniques considerable reduction in overbreak can be achieved.

16.7 SECONDARY BLASTING

Secondary blasting includes blasting carried out during bench toe levelling and bench sloping, oversize boulder breakage and also other auxiliary blasting operations. The handling of boulders (stone blocks which are too large for mucking or crushing) is often an expensive operation and for this reason, primary blasting objective is such that not too many boulders should result. Here, methods other than blasting for breaking of boulders are not described.

Two methods are used for blasting of boulders.

- Using plaster charges: Plaster shooting;
- Using charges in drill holes: Pop shooting.

16.7.1 *Plaster shooting*

The simplest application of plaster charges consist in that the charge is placed directly on the surface of the boulder as a flat layer of thickness of about 35-50 mm

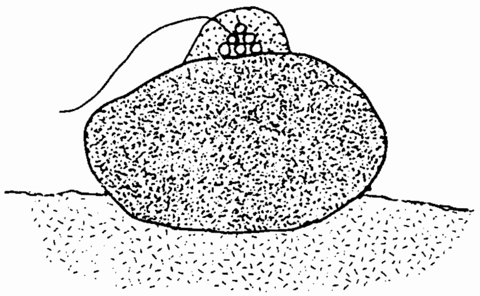


Figure 16.8 Plaster shooting.

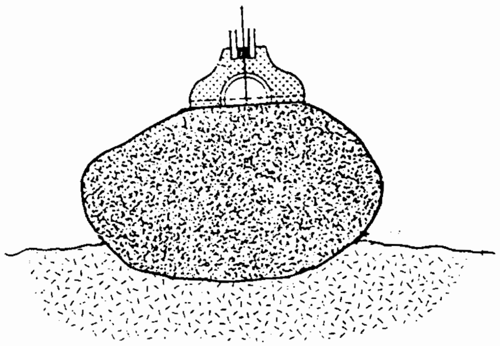


Figure 16.9 Special jet or shaped charge for plaster shooting.

(Fig. 16.8). The plaster charge is covered with a layer of clay or sand, hence it is also called mud capping. The method is efficient with brittle rocks and also when possibilities of drilling holes is not there such as in small scale mining. The method can only be used if buildings and other structures are far away. The explosive consumption may be more than 1.0 kg/m^3.

The charge is applied in such a way that the explosive comes in good contact with the face of the rock. The charge can vary with the type and shape of the boulder. A round boulder is more difficult to break with a charge of this type than a thin boulder with a large area. Since boulders resulting from previous round have been subjected to stresses and very powerful forces, they are often easy to break up by blasting compared with naturally occurring boulders. An explosive which has high velocity of detonation is suitable as blasting is taking place due to stress wave reflection and subsequent formation of fractures.

The efficiency of the method of plaster shooting is much improved if special jet charges or shaped charges are used (Fig. 16.9). Presently 0.1-4 kg charges are available.

16.7.2 *Pop shooting*

Oversize boulders are more commonly broken with the aid of the blasthole charges placed in a drilled hole (Fig. 16.10). The method is called pop shooting or block holing. The advantages are less explosive consumption and less noise than in plaster shooting.

The hole is drilled to depth of about (0.25-0.5) h_b, where h_b is the thickness of the boulder. Large sized boulder may need more than one hole, spacing between the holes is about (0.5-0.9) h_b. The amount of drilling length is about 0.2-1.0 m/m^3 of the boulder size. The hole is drilled so that the burden is not too small in any direction otherwise the result of the blasting deteriorates by large parts not being split. The explosive consumption is about 0.1-0.3 kg/m.^{3} The hole is stemmed.

The spreading of fragments is reduced and the consumption of the explosive is decreased by filling hole with water and then placing the explosive in the hole. The amount of explosive charge needed then is 8-12 times less than that of conventional explosives.

If several drill holes are used in a boulder, then initiation is carried out by using

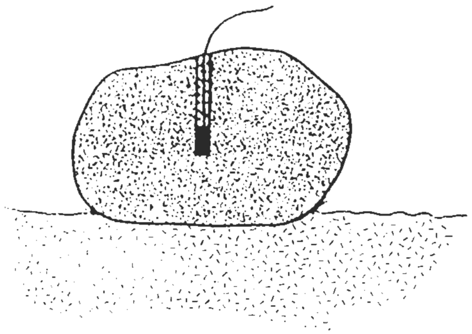


Figure 16.10 Pop shooting.

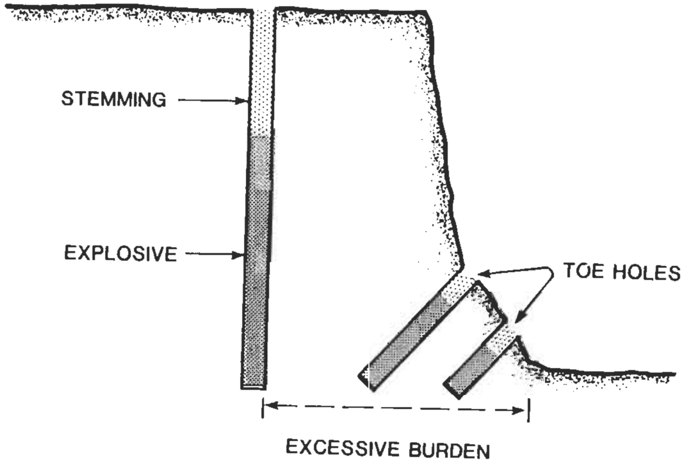


Figure 16.11 Toe blasting.

instantaneous detonators. In the case of built up areas it is better to use several drill holes with a small charge. Thorough covering is essential for built up areas.

16.7.3 *Toe blasting*

Bench toe blasting is usually accomplished with charges placed in inclined small diameter hole or holes of medium diameter (Fig. 16.11). The inclination and the design of the charge suit the blasting conditions under particular set of conditions.

CHAPTER 17

Computer aided blast design

17.1 INTRODUCTION

Computer simulation of blasting is making impressive improvements in blasting efficiency and steadily replacing the trial and error approach to fragmentation. The computer aided programmes allow mine operators to examine many different combinations of explosives and blast parameters in a very short time. This permits rapid optimisation of drilling and blasting designs and costs and can affect operational procedures, such as reduction of mucking costs through finer fragmentation or examining the effect of different blasting parameters on casting blast contours. Several types of programmes are available for simulation of blasting operations in open pit, quarrying and underground mining operations. In this chapter an attempt has been made to explain different computer aided blasting design concepts and procedures.

17.2 AIMS

The computer aided blast design programme considers as input all aspects of the blasting process.

- Rock properties;
- Blast parameters;
- Explosive parameters;
- Fragmentation parameters.

Based on the above, the computer aided blasts aim at the following.

1. Design blasting patterns to maximise blasting efficiency while lowering overall operational costs;
2. Select effective explosives and provide comparison to help in choosing optimum product according to needs;
3. Select proper borehole (size, inclination and depth) depending on the rock and after blast requirements;
4. Select initiation timing and delay sequence according to face conditions;
5. Predict fragmentation size and distribution and provide muckpile profile and muckpile displacement;
6. Calculate and optimise drilling and blasting costs for any given blasting pattern.

In addition, ground vibrations and charge mass calculations, detonator timing and circuit calculations and similar other calculations can be separately done. The computer can be used to store blast records and results for later retrieval and analysis.

17.3 INPUTS

A computer model requires maximum data input to provide a design for a specific operation. This data is divided into three key categories with some typical examples as follows.

1. *Rockmass parameters*
   - Rock density,
   - Tensile and compressive strength of rock,
   - Young's modulus of rocks,
   - Longitudinal wave velocity,
   - Poisson's ratio,
   - Strike and dip values,
   - Joint structure and frequency.

2. *Explosives*
   - Type of explosives,
   - Bubble energy,
   - Strength properties,
   - Velocity of detonation,
   - True hole density,
   - Mass/meter of blasthole,
   - Delay sequence within the blast,
   - Distribution of charge within hole,
   - Initiating system,
   - Initiation and explosive cost.

3. *Site factors*
   - Face height,
   - Bench width/length,
   - Wet or dry holes,
   - Production requirements,
   - Drilling diameter and equipment information,
   - Primary crusher dimensions,
   - Loading and hauling equipment information,
   - Rock displacement requirements,
   - Fly rock range,
   - Environmental constraints e.g. ground vibrations,
   - Allowed damage to remaining rock,
   - Drilling cost/meter.

Based on the input information blast design is provided to achieve the aims. Mini-computers, Micro-computers or programmable hand calculators are used for designing blasts. Many programmes are available are being used, some of which are described in Section 17.4.

17.4 COMPUTER PROGRAMMES

Several different types of computer programmes are available.

1. *Simple programmes*
Simple programmes use statistical data analysis of past and current practices in the mines from which empirical relationships are developed (Bhandari 1979b, 1991a; Daniel 1984; Ghose & Samaddar 1984). Using these empirical data, programme provides parameters like burden, spacing, sub-grade drilling, stemming, initiation pattern and calculate powder factors, drill factor, average fragment size, displacement and flyrock. All the parameters or any one parameter can be studied. This provides possibilities of comparison and optimisation of blasting parameters. SARO-BLAST programme covers the design of the fragmentation and muckpile profile (Kau & Rustan 1993). Swedish Detonic Research Foundation bench blasting programme includes damage zone and vibration particle velocity analysis (Anon. 1988b). Many programmes using similar relationships are available commercially (Kennedy 1994).

2. *'BLASPA'*
'BLASPA' (Favreau et al. 1987) is a mathematical model containing many aspects of blasting mechanisms. A general solution is found for each mathematical description and all solutions are finally brought together in the memory of the computer to create the complete model of the phenomenon.

The principle mechanisms in the model are as follows. Immediately after the explosive column has detonated two types of energies are released during an explosion, the brisance (shock energy) is associated with actual detonation while the heave is produced by subsequent expansion of high pressure gas. The magnitude of these energies participating in the blast is determined by the explosive used and the competency of the enclosing rock.

Three criteria affect the results of a blast: the explosive type, rock type and blast geometry. The explosive type determines the shock energy and gas volume. The rock type influences blasting in two ways, the degree to which burden rock is physically detached from the bed rock and simultaneously reduced to convenient sized blocks. Rock types also governs the energy distribution throughout the burden. Computer simulation techniques can assess the effects of all these variables on the blasting programme. During simulation sensitivity analysis is performed, altering one parameter at a time to determine its effect on the other variables. Output normally gives data on grade lift, grade rupture, mass and grade movement, flyrock range, backbreak, effect of V.O.D., minimum delays, fragmentation size and distribution, drilling and blasting costs and numerical estimates of the brisance and heave, not only as average values but also as local values at key areas of the blast such as at the toe and the collar zones.

3. *Scientific approach to breaking rock with explosives (SABREX)*
SABREX is a modular computer program which is used for surface and underground blast simulation (Jorgenson & Chung 1987). SABREX uses data on the detailed geometry of the drilled and loaded pattern, detonation characteristics of the explosives and the dynamic properties of the rock to generate blast predictions. The programme

can be run on a micro-computer and provides predicted blast results as dynamic colour graphics and in tabular form.

Variables input

1. 'CPEX' non-ideal detonation code and ideal detonation code 'BLEND' provide explosives property inputs. Extensive field and laboratory testing provides necessary input data.

2. Rock properties determined by laboratory testing that are normally used as input to 'SABREX' determination are: density, Poisson's Ratio, Young's Modulus, unconfined compressive strength, tensile strength, dynamic tensile strength, and shock attenuation. In-situ measurements of the rock to be blasted may also be required and have generally consisted of seismic determination and structural orientation mapping.

Once the explosives and rock properties have been assembled and input, the following controllable parameters may be varied and tested on SABREX: blast hole diameter, pattern (burden-spacing-special face condition), blasthole inclination, subgrade drilling, collar distance, initiation pattern, delay timing, coal seam depth and spoil pile location (for cast blasting in coal operations) and costs for explosives and drilling.

The operational modules of SABREX computer programme are as follows:

Fragmentation modules

KUZ-RAM and CRACK modules offer independent absolute predictions of fragmentation resulting from a given blast.

The heave module

This module uses calculations based on gas expansion to determine the velocity and time of initial movement of blocks within the burden and then tracks the motion of the blocks until they come to rest. Factors in the calculation include limitations of subsequent rows by the first row movement, swell factor and angle of repose.

Damage module

The rupture envelope module – The ability to model damage effects from blasting is an important tool in design and maintenance of mine structures. In this study the rock properties are determined and then damage envelopes run for spherical charges of several explosives types. The radius of damage behind the charge give the resultant crater radius and the sub-grade crack penetration.

4. *Drilling parameters and blast design*

Scheck & Mack (1989) approach is based on the study of variable nature of the material being drilled and based on these data blast is designed. The following parameters are obtained from the drilling operations.

1. Power required to drill: Rotary torque; Rotary speed; Down pressure; Penetration rate.
2. Vibration: Vertical; Horizontal.
3. Air pressure.

Electrical signals proportional to the actual values of these parameters are monitored on the drill. The first three parameters e.g. rotary torque, rotary speed and down pressure are related to energy consumption and can be measured directly. The penetration rate can be calculated from the bit position and drilling time. Air pressure depends on the flow rate and the friction losses in the hole.

For correcting the drilling parameters to overburden structure the procedure consists of comparing the logged data to manually recorded information taken from the control panel in the drill operator's cabin. Since the operator's panel also has a depth indicator, the manually and computer recorded data can be compared for given position within a hole.

In addition, to manually recorded gauge readings, the changes in the drill cuttings can be logged against depth and the operator's 'subjective' evaluation of the rock characteristics can be obtained.

In one system the blast hole drill logging and design was built around STD-BUS and FORTH compiler was used with a Z-80 micro-processor.

Based on the algorithm, the microprocessor generates a log of each blast hole as it is drilled. A hard copy of each hole's log and 'Decked design' is available on board on the drill and the blasting supervisor uses this information to plan his blasts. The drill hole data is also retained in non-volatile memory and transferred to a central computer for further analysis, management reporting and archival storage.

Based on the above programme controlled experiments were performed in the test area. The amount of explosives and the pattern were kept constant. The decked loads based on the automated system gave markedly better fragmentation than the standard undecked designs.

5. *Computerised optimum blasting in tunnels*

An optimum design for blasting in tunnels can be reached using 'OPTES' (Optimum Tunnel Extract System) software (Flores 1984). This software functions rapidly, reliably and with great flexibility under different design parameters such as tunnel shape, geology and rock type and in addition, varies parameters making efficient decisions. The OPTES is used for the best solution, applying sensitivity analysis to important variables such as drill hole diameter, selection of type of explosive, the most adequate type of drilling equipment, etc.

According to flow chart OPTES works with an initial program of input data which enters a data base related to geomechanical characteristics of rock, properties of the explosive and information about drilling equipment, blasting and extraction of material. It provides the following.

– The proposed drilling and blasting pattern.
– The graphical layout of the blasting pattern including type of cut suitable and sequence of firing.
– Sensitivity analysis of the relevant variables of the design.
– Ranking of explosives on the market.
– Information control manual.

6. *Programme for selection of explosive and cost*

SELXPLO programme considers the mine site conditions, production requirements and cost consideration (Balara 1991; Bhandari & Balara 1992). The programme makes possible selection of explosives and gives blast design and the cost comparison between cartridge loading and bulk systems for explosives. On the interactive terminal an input screens appears in which user simply gives the mine data. For spe-

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                         BENCH BLAST DESIGN & COST

Mine Name ..................: JAY PROJECT
Production/year..[tonne]....:9.1000000.00    Hook-up.................:ROW*ROW
Hole Diameter....[mm].......:259             Type of Explosive........:SLURRY
Bench Hight......[m]........:15              Explosive Density [gm/cc]:
Rock Density...[tonne/m3]...:                1. Low Density like ANFO.........:
   1. Soft Rock.............:                2. Average e.g. ANFO & Slurry....:
   2. Medium Rock...........:                3. High like Slurry & Emulsion...:
   3. Hard Rock.............:                Powder Factor [tonne/kg].:2.9
Compressive Strength[kg/cm2]:1800            Castbooster Cost [per kg]:
Drill Dip......[deg.].......:90              Bulk Cost......[per kg]..:
Burden.........[m]..........:9               Cartridge cost..[per kg].:
Spacing........[m] .........:8               Cordex cost.....[per m]..:
Face length....[m]..........:50              Delay Relay..[per unit]..:
Toe Burden.....[m]..........:10              Primex Cost..[per kg]....:
Number of Rows..............:5               Drilling Cost..[per m]...:
Type of Pattern.............:STAGGERED

                    All Entries Correct..[Y/N]...? Y
```

Figure 17.1 Input of screen with sample data.

cial data input the simple questions are asked which are needed for the design and cost calculations. The programme computes:

– Details of blasting pattern;
– Explosive consumption;
– Cost calculations.

Input data required are hole diameter, bench height, rock density, face length, explosive density and number of rows (Fig. 17.1). There is possibility of providing rest of the input based on the standard input data and the programme calculates all other parameters.

For a user who does not know the density of the explosive he can specify the type of the explosive such as slurry, ANFO or high density NG based product. Similarly rock density if not specified then average density of soft, medium and hard rock is used. The powder factor is either specified or computed. The burden is most crucial parameter, which can either be specified or computed by using relation between hole diameter and explosive density. Based on the burden other parameters like spacing, stemming and subdrilling are calculated. It is also possible to specify spacing otherwise it is computed. Interrelations between parameters were derived by Vutukuri and Bhandari (1973) and further improved during recent years.

With these inputs, drill depth, holes per row, quantity of explosive consumed (booster and column), volume and tonnage of rock broken, powder factor and total cost of drilling and explosive per hole are then determined. The final cost per ton of rock broken is also calculated. The results are obtained in a tabulated form (Fig. 17.2).

For the cost calculation there is a possibility to feed drilling cost per meter, explosive cost and accessories cost, otherwise current market costs are assumed.

This programme for initial work is ideal for studying variation of parameters and to find optimum parameters which give minimum cost of excavation per ton of rock. After an initial design is made and adjusted to given operation, any of the inputs like hole diameter, type of explosive, powder factor or spacing to burden ratio can be


```
                               BENCH BLAST DESIGN
                                                          COST CALCULATION
      BLAST DESIGN CALCULATION
Rock Density.[tonnes/m3].:           1.90      Cast Booster cost [Rs.]:         22.17
Hole Diameter....[mm]....:          259        Bulk/Cartdg.cost..[Rs.]:       9379.12
Bench Height......[m]....:          15.00      D-Fuse cost...[Rs.]....:       2628.45
Drill Length......[m]....:          17.70      Primex Cost/hole.[Rs.].:         45.00
Face length.......[m]....:          50.00      Delay Relay cost.[Rs.].:         75.00
Burden............[m]....:           9.00      Total Explosive..[Rs.].:       9536.40
Spacing..................:           8.00      Drilling cost....[Rs.].:       4513.50
Subgrade Drilling.[m]....:           2.70      Cost per hole....[Rs.].:      14049.90
Stemming..........[m]....:           6.30      Cost per cu.m....[Rs.].:         11.02
Holes per Row............:           6         Cost per tonne...[Rs.].:          5.80
No. of Rows..............:           5
Pattern..................:STAGGERED
Volume of Rock Broken[m3]:       38232.00
Mass of Rock Broken[ton].:       72640.80
Total Explosive..[kg]....:       25048.55
Powder Factor..[T/kg]....:           2.90

              Want to Print on Printer..................[Y/N]..?
```

Figure 17.2 Blast data obtained by the use of computer programme.

varied and quick computation is obtained. Thus optimisation of blasting design is possible.

Application of artificial intelligence, expert system has also been made to optimise the results.

In a programme EXPERTIR (Cheimanoff et al. 1990), goal is to choose a drilling pattern, the initiation type, and to distribute explosive while following the local regulations. The programme is developed with several modules.

7. *BLASDRIV*

Another programme BLASDRIV provides a simple calculation to determine drilling pattern for tunnel and drive blasting using parallel hole cut pattern. The calculations use relationships developed in Chapter 13. Based on hole(s) diameter, drive geometry and explosive characteristics, calculations are made for the location of all the holes in the pattern. It is possible to optimise the computation of drilling pattern.

Computer aided blast design provides a rapid as well as reliable form, the information necessary for carrying out blasting operations. It is possible to speedily review and compare a wide range of blasting options as an aid to decision making. This capacity allows advanced problems to be solved in a couple of minutes and eliminates need for extensive trial and error experimentation.

CHAPTER 18

Ground vibrations and air blast

18.1 INTRODUCTION

On detonation of explosive charges in blastholes, apart from the effective utilisation of the explosive energy in the fragmentation and displacement, considerable part of the energy is wasted in the form of ground vibrations, air blast noise, flyrock, dust and noxious gases. This waste energy exposure creates lot of a problems, both for the workers associated in the excavation process and the local inhabitants in the nearby area. Continual noise and vibrations from blasting may be annoying to the residents. Although blasting vibrations and air blast noise are short term transient phenomena, the residents in the vicinity of operations feel that if vibrations and air blast continue then their dwelling may get damaged. Moreover, uncontrolled blasts lead to complaints and damages to buildings and structures in the vicinity of blasting operations, as well as mine workings may also get damaged. These unsavoury problems may lead to confrontation and even litigations between the mining industry and the general public. The problem continues to be of great concern to governments, explosive manufacturers, industries utilising blasting, insurance companies and scientists. With the increased awareness about environment, increased complaints and litigations about damage due to blasting have been experienced by operators. Further, it is in the interest of operators themselves that attempts be made to control harmful effects of blasting operation.

This chapter has attempted to discuss ground vibrations and air overpressure produced during blasting, factors which affect both these phenomena, human and structural response, monitoring and techniques to minimise them.

18.2 GENERAL CHARACTERISTICS OF BLAST VIBRATIONS

When an explosive charge is detonated in a blasthole, rapid release of energy takes place within a very short period causing tremendous rise of temperature and pressure. The surrounding rock melts, flows, crushes and fractures. At some distance from the explosion source the inelastic processes and elastic effects start. Only a part of the total chemical energy released is converted into elastic form. The elastic disturbances which propagate away from the explosion source are termed as seismic waves. These waves are quickly transmitted through the solid medium which comes

back to original configuration after the passage of the seismic waves. The seism observations are made in this zone only.

However, when an explosive is detonated in open air, in a very shallow hole after doing the fragmentation work the gases released to the atmosphere produce a mospheric pressure waves which are initially above ambient air pressure.

It is useful to understand the nature, and propagation of various waves i.e. bo seismic waves and air overpressure waves involved.

18.2.1 *Seismic waves*

Several types of wave forms are identified from seismic records (Fig. 18.1). Base on the characteristic features these can be divided into two basic groups, namely:

1. *Body waves*. Waves that travel through the rockmass are termed as body wave They can be further sub-divided as:

– Primary or *P*-wave. This is a compression wave which alters the volume of th body without altering its shape. In such cases, particles vibrate in the direction propagation of the wave.

– Secondary or *S*-wave. This is a shear wave resulting in change of shape onl when the medium particles oscillate perpendicular to direction of propagation.

These body waves propagate outward in a spherical manner until they intersect a boundary such as another rock layer, soil or the ground surface. At this intersectio shear surface waves are produced.

2. *Surface waves*. Waves which travel along the surface and cause ground rol They can be further sub-divided as:

– Rayleigh or *R*-wave. This is longitudinal wave and causes mainly retrograd motion.

– Love or *Q*-wave. This is a shear wave causing transverse vibration horizontall with no vertical component.

– Coupled wave. This oscillates particle in an inclined elliptical motion, havin components both in horizontal and vertical directions. The use of term 'couple implies combined *P*-wave and *S*-wave motions.

To describe the motions completely, three perpendicular components of motio must be measured. The longitudinal component, *L*, is usually oriented along a hor

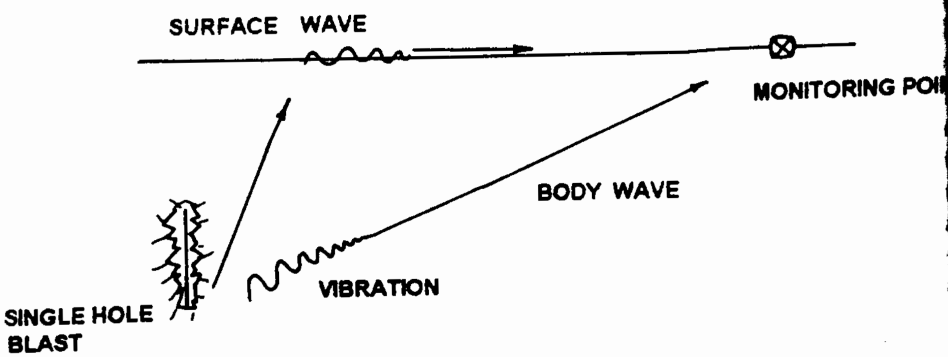

Figure 18.1 Surface and body waves.

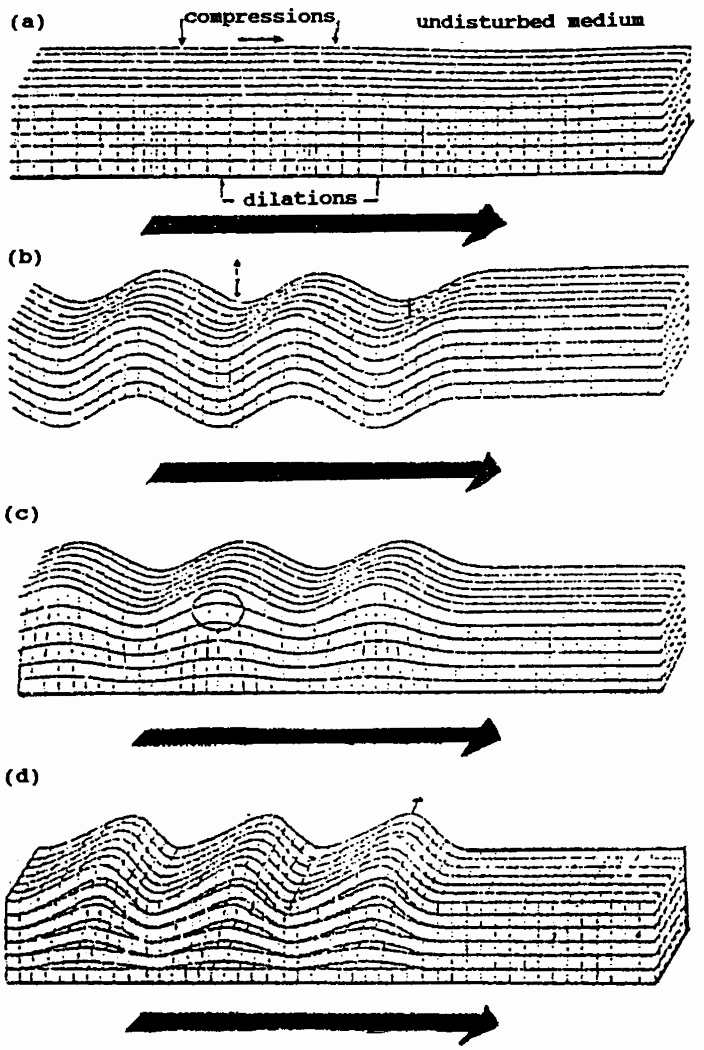

Figure 18.2 Particle motion variation with wave type a) Compressive, *P*-wave, b) Shear, *S*-wave, c) Rayleigh and d) Love wave.

zontal radius to the explosion. The other two components are vertical, *V*, and transverse, *T*, to the radial directions.

It has been noted that at small distances, all three wave types arrive together and greatly complicate wave type identification, whereas at large distances, the more slowly moving shear and surface wave begin to separate from the compressive wave and therefore, the identification becomes possible. However, most blasts are detonated as a series of smaller explosions which are delayed by milliseconds, and differences in travel paths and delay time result in overlapping arrival of wave fronts and wave types. The three wave types produce radically different patterns of motion in rock and soil particles as they pass. As a result, structures built on or in rock (or soil) will be deformed differently by each wave. The varying particle deformations of the earth (and consequently structures) for each principal wave-type are compared in Figure 18.2. In each case the wave is propagating to the right parallel to the tunnel or

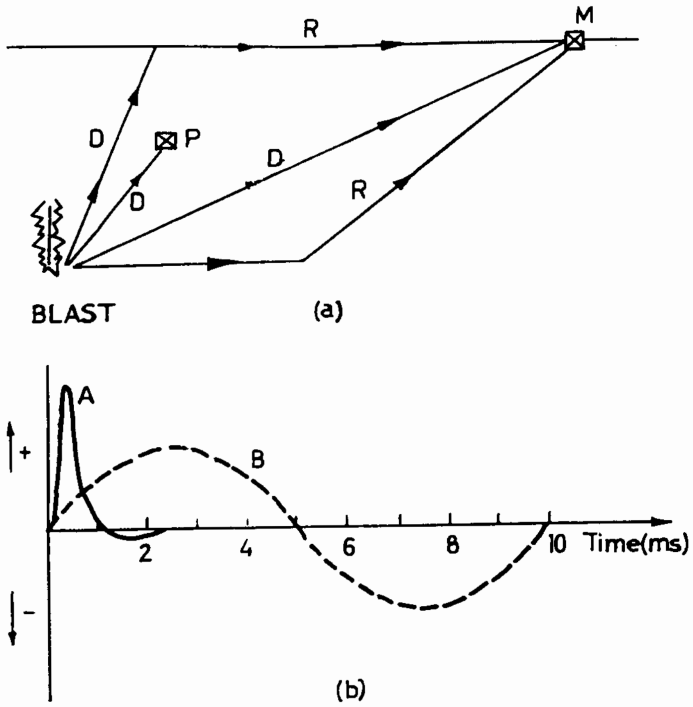

Figure 18.3 Idealised waves from blasting. a) Blast monitoring and b) Idealised waves. *D* = direct waves, *R* = reflected/refracted waves.

pipe. The longitudinal (compressive) wave produces particle motions in the same directions in which it is propagating. On the other hand, the shear wave produces motions perpendicular to its direction of propagation: either horizontal, as shown, or vertical. The Rayleigh wave produces motions both in the vertical direction and parallel to its direction of propagation.

Most blasting problems involve monitoring at position *M* as shown in Figure 18.3 and result in sinusoidal waves *B*. A close in explosion produces the single-spiked pulse, *A*, by direct transmission to position *P*. At the far field position, *M*, an explosion will produce the relatively sinusoidal pulse, *B*, through a combination of direct transmission, reflection and refraction. The idealised waves shown in Figure 18.3b are typical, wherein close-in blasting produces transient pulses which last 1000-2000 µs or 1-2 ms. On the other hand, at relatively large distances, the single sinusoidal pulses have duration of 10-100 ms. Combinations of these single pulses produce the commonly observed sinusoidal wave trains, as shown in Figure 18.3b.

Definitions

Vibrations can be defined as repeated movement about the position of rest. The basic terminology connected with vibration are (Fig. 18.4):

– Amplitude = *a*, the maximum displacement from the position of rest;
– Cycle = *c*, one complete sequence of repeated events;
– Period = *t*, the time required to complete one cycle or oscillation;
– Frequency = *f*, number of times the cycle is repeated in a unit time and expressed in cycles/second (c/s) or Hertz (Hz);

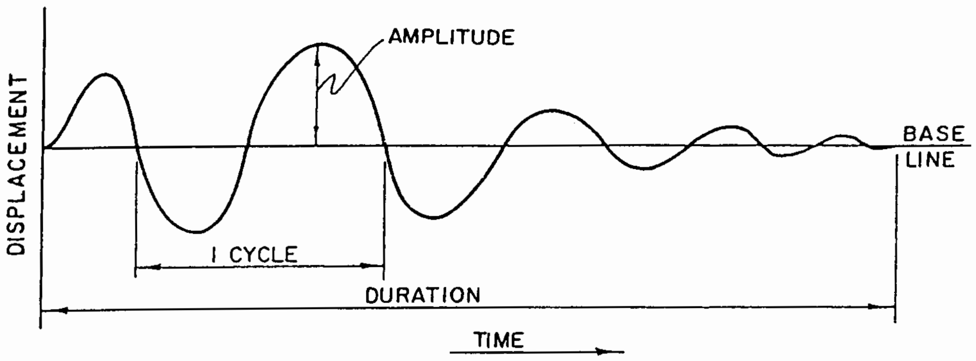

Figure 18.4 Idealised displacement versus time vibration trace.

– Particle displacement = δ, displacement of the constituent particles of the medium as they are excited by the wave energy expressed in millimetres, meters or inches;

– Particle velocity = V, the velocity at which the vibration wave travels through the medium or rate of change of displacement of constituent particles of the medium as they are excited by the wave energy expressed as $V = \delta/t$ in mm/s;

– Particle acceleration = A, acceleration or rate of change of velocity of constituent particles of the medium as they are excited by the wave energy expressed as $A = V/t$ in mm/s^2. It is also expressed in terms of acceleration due to gravity;

– Wave length = λ, it is the length of one complete cycle usually measured in mm. It can be calculated as $\lambda = V/f$.

The disturbance at a point is completely defined if the magnitudes in three mutually perpendicular directions of either displacement, velocity or acceleration are measured. All these parameters are subjected to direct measurements. Relationship between frequency, velocity and displacement is $V = f\lambda = \lambda/t$.

Seismic motions are generally aperiodic thus calculation of any parameter from another assuming simple harmonic motion will not give correct value. Integration or differentiation should be done using Fast Fourier analysis (FFT).

18.2.2 *Air overpressure*

Blast induced air overpressures (defined as pressures above normal atmospheric pressures), are the air pressure waves generated by explosions. More commonly this is termed as air blast. These are produced either by the direct action of the explosion products from an unconfined explosive in air or by the indirect action of a confining material subjected to blast loading. The wave produced by the effect of blasting interacts with the air and increases the air pressure from peak to ambient and drops to negative (i.e. below ambient pressure) slowly (Fig. 18.5). Its travel is thereafter governed by air temperature, wind direction and speed, and the presence of obstructions such as buildings, vegetation, and ground contour.

The overpressure wave near the source (blast site) contains a wide range of frequencies (20-20,000 Hz). The higher frequency portion of the pressure wave is

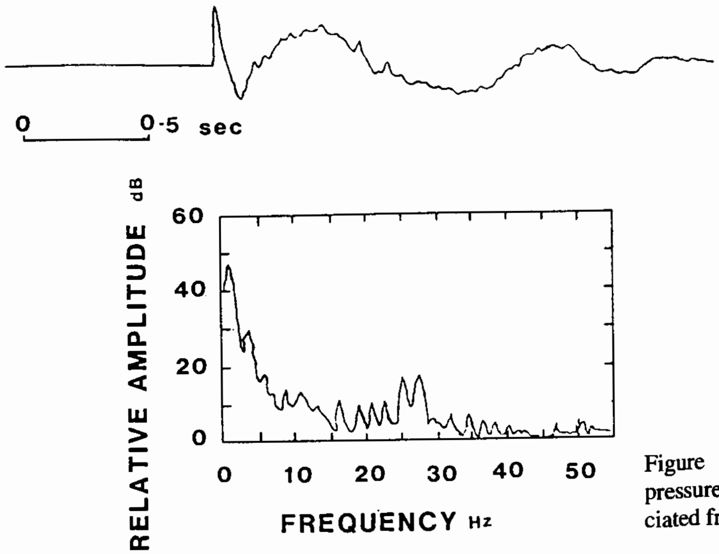

Figure 18.5 Typical air overpressure time history and associated frequency spectrum.

audible and is the sound that accompanies a blast, the lower-frequency portion is not audible, but excites structures and in turn causes a secondary and audible rattle within a structure.

However, as the distance from the site increases the higher frequencies attenuate more readily and due to this much of the energy at greater distances from the source will have frequency lying in the infra sound (below 15 Hz) region, to which the average human ear cannot respond. Unfortunately, many structures have the natural frequencies below 15 Hz, and thus readily respond to these frequencies.

Overpressure waves are of interest for three reasons. First, the audible portion produces direct noise. Second the inaudible portion by itself or in combination with ground motion can produce noise. Third, they may crack window panes; however, air blast pressure alone would have to be unusually high for such cracking.

Overpressure can be measured in two ways; either in pressure units or more commonly are reported in terms of decibels (dB), which is given as sound pressure level (SPL) by the following equation:

$$\text{SPL (dB)} = 20 \log \ [p / p_r]$$

where p is the measured overpressure in Pa and p_r is a reference pressure level. The 'threshold of hearing', representing the lowest sound audible, to the ear, is taken as reference pressure, p_r, and has a value of 2×10^{-5} Pa and represents level of zero dB.

In general, A-weighted sound level i.e. dB (A) is used. It is simple measuring technique keeping high correlation with N (noise) criteria and PNC (preferred noise criteria). It is suitable for measuring fluctuating noise and adequate for predicting general public acceptance of most community noises.

The lowest frequency which normally gives a sensation of sound for the average individual is about 25 Hz. Hz is the most common unit of sound frequency defined as the cycle per second. The ear of a normal adult human has the ability to hear a very broad spectrum of frequencies from 25-16,000 Hz.

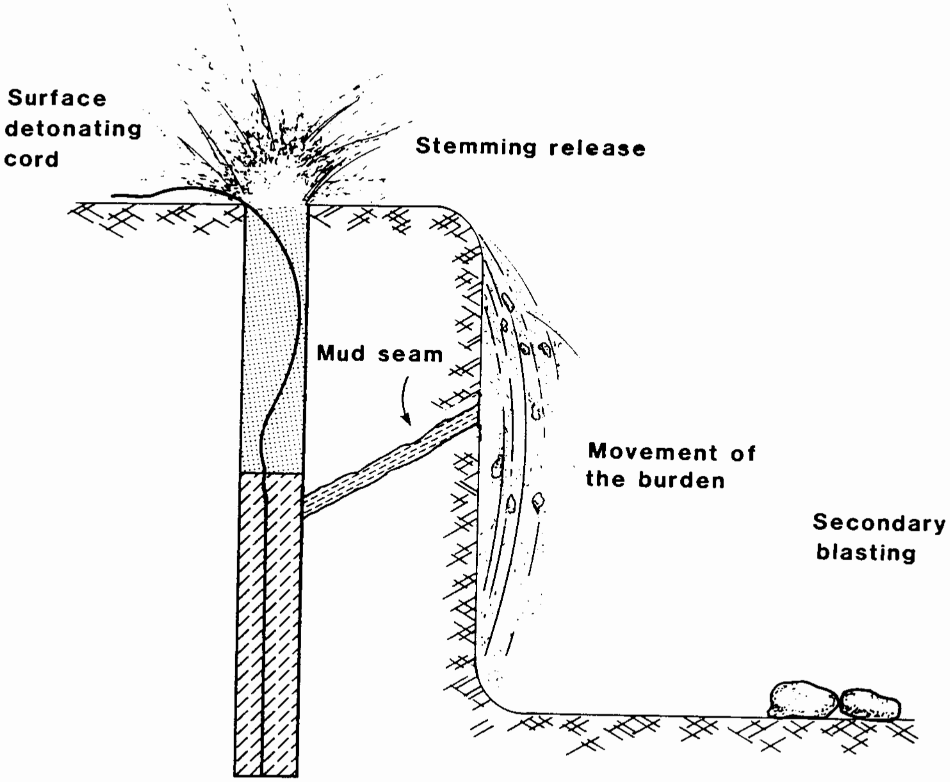

Figure 18.6 Causes of air overpressure due to blasting.

Air blasts are caused by the sudden release of gases to the atmosphere from unconfined or partially confined explosions. Such conditions arise from secondary blasting (especially plaster shooting), overcharging, poor stemming and escape of gases through fractures, mud seams or weak strata (Fig. 18.6). Uneven and over broken faces from the previous blasts result in reduced burdens which lead to sudden escape of gases to the atmosphere without performing adequate fragmentation work. Exposed detonating cord such as trunk line and down line used in blasting work are often cause of sharp overpressure pulse.

Direct rock displacement of the ground surface, impact of the falling fragments are also causes of some noise. Ground vibrations due to blasting in turn cause vibrations to propagate through the atmosphere. The vertical component of the seismic wave produces an equivalent motion in the air. This seismically induced pulse is termed as rock pressure pulse. Amplitude of this pulse is normally too low to be recorded by the sound sensors (Wiss & Linehan 1978). Sometimes the stemming ejects out from the blasthole giving rise to sudden release of gaseous energy into atmosphere compressing the air. Normally these compression waves generated in air are characterised by higher frequencies.

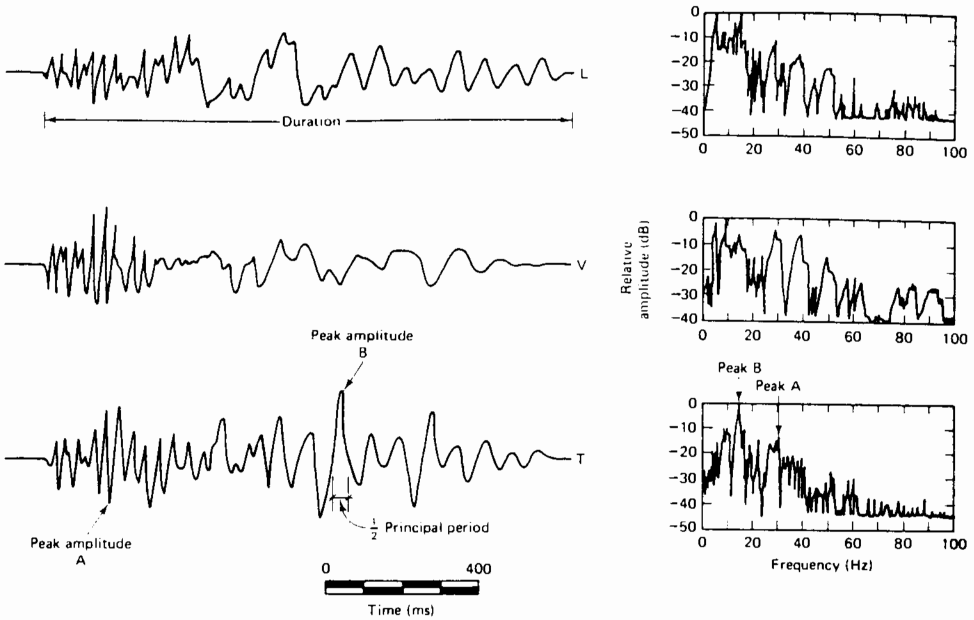

Figure 18.7 Typical coal mine blast vibration time histories (Stagg & Engler 1980).

Table 18.1 Range of typical blast ground vibration parameters (Cording et al. 1975).

Displacement	10^{-4}-10 mm
Particle velocity	10^{-4}-10^{3} mm/s
Particle acceleration	10-10^{5} mm/s^{2}
Pulse duration	0.5-2s
Wave length	30.0-1500 mm
Frequency	0.5-200 Hz
Strain	3.0-5000 μ inch/inch

18.2.3 *Typical blast vibration records*

A typical particle velocity time history at a surface coal mine is shown in Figure 18.7. The most important parameters which describe the time history are peak amplitude, principal period and duration of the vibration. All these parameters are dependent on the blast sequence and transmission medium. In normal blasting operation for tunnelling, surface mining and construction, these parameters vary substantially as may be seen in Table 18.1.

In special cases such as close-in blasting, the range of ground motions may be of very high values.

Two principal characteristics distinguish blast vibrations from either earthquake or nuclear motions. First, blast vibrations occur at higher frequencies than that of either the earthquake or the nuclear explosion. Second, the blast vibrations, by virtue of its relatively minuscule duration, carries little energy compared to nuclear or earthquake motion. A third characteristic, the range of principal frequencies, is greater for nuclear and earthquake motions.

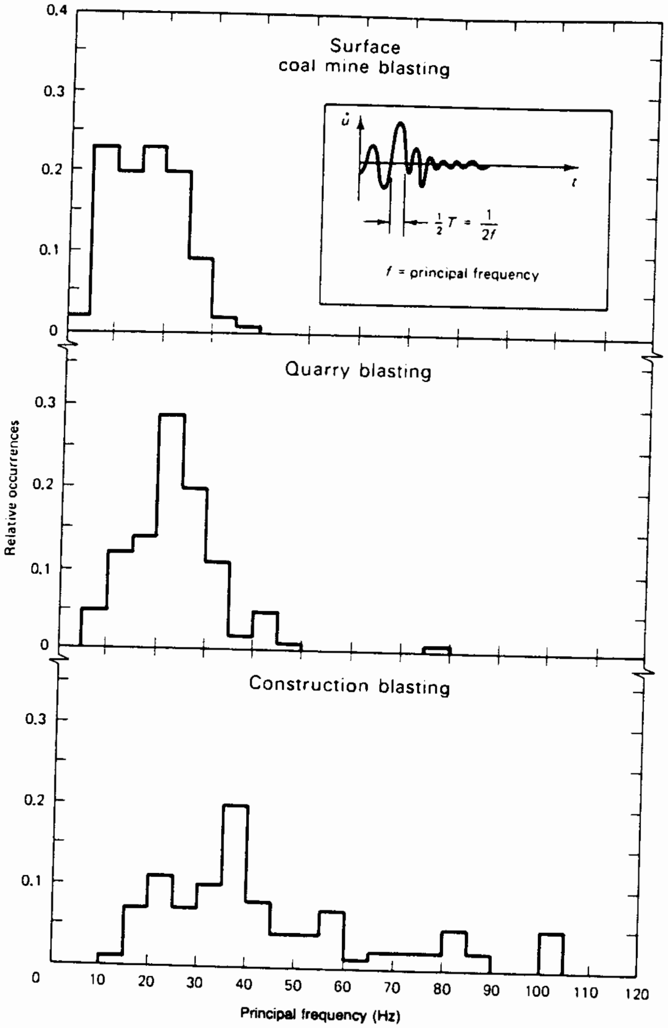

Figure 18.8 Predominant frequency histograms at structures of concern (after Siskind et al. 1980b).

The principal frequency is defined as that associated with greatest amplitude pulse and it varies by industry (Fig. 18.8). The relatively large explosions produced by surface coal mining, when measured at typically distant structures, tend to produce vibrations with lower principal frequencies than those of construction blasts. Construction blasts involve smaller explosions, but the typically small distances between a blast as well as rock to rock transmission paths tend to produce the highest frequencies.

Principal frequencies also depend on the transmission medium. High frequency motions tend to be filtered out or attenuated over shorter distances in soil than in rock.

18.3 MONITORING BLAST VIBRATIONS AND OVERPRESSURES

The objective of any measurement is to detect and record the vibratory motion of the ground or the structure and also to simultaneously measure the induced air overpressure. This motion is caused by forces which are variable in magnitude and/or direction and also the transfer or propagation characteristics of the medium. If the measurements are taken on or within some type of structures, the recorded motion will not be actual ground vibration but the response of the structures.

The parameters measured must result in a full description of the vibration, displacement, and acceleration as a function of time. The basic problem connected with the measurement of ground vibration is the establishment of a fixed point in space as reference in respect of which the vibration will be measured. During the passage of seismic energy, the entire environment is in motion rendering ordinary forms of measurement impossible. The seismograph is an instrument designed to establish a point internally that tends to remain fixed during the vibration of its housing frame.

18.3.1 *Seismograph*

A seismograph consists of basically two components – a transducer or pickup and a recorder. Magnification of the ground vibration by the seismograph is necessary for achieving the requisite accuracy, particularly at high frequencies and/or low amplitudes. A restoring force is required to be operative on the transducer to prevent its oscillation freely since it has been disturbed. This is called damping. The restoring force serves best when the seismograph is damped 60% of the critical i.e. the damping force brings the transducer to rest after the oscillation. Often blasting seismographs also include air overpressure measurement by use of microphone on an additional channel.

Modern monitoring techniques utilise full wave form recording to permit analysis of the vibration signals at any instant of time, and to use this information to understand the source and nature of the disturbance. A field portable blast monitoring system (Fig. 18.9) usually combines measurement of both ground and air blast overpressure. It consists of (1) transducers for ground vibrations and a microphone for air overpressure measurement that convert physical motion or pressure to an electrical current, which is transmitted through (2) cables to an (3) amplifying system and a magnetic tape, paper or computer (4) digital recorder that preserves the relative time variation of the original signal for eventual permanent hard-copy reproduction by a pen recorder or light beam galvanometric recorder or printer. Microprocessor based monitors are most common for data acquisition, storage and reproduction. The capabilities of high resolution video recorders and full waveform recorders have been combined to permit a synchronised view of both the blasting event and the overpressure signal. Using these techniques, it is possible to identify the precise moment at which the peak levels are generated, and therefore the causes, and then to modify blast design to reduce peak levels.

Figure 18.9 Blasting seismograph (courtesy Instantel).

18.4 FACTORS AFFECTING VIBRATIONS AND OVERPRESSURE

A number of factors influence the development, propagation and intensity of ground vibration and overpressure pulses. These can be grouped as controllable and uncontrollable factors. Controllable factors include blast geometry, type and amount of explosive used, stemming, priming and initiation. Uncontrollable factors include distance, geological conditions, initiation timing errors, meteorological conditions. Some important factors are considered in Sections 18.4.1-18.4.8.

18.4.1 *Blast geometry*

The burden is one of the important factors which affect both ground vibrations and air overpressures. A charge with proper burden will produce less vibrations than a charge with greater burden. Excessive burden increases the ground vibration because the explosive energy which is insufficient to break the burden rock will be converted into vibrations.

If the burden is less than optimum the gaseous energy is dissipated into atmosphere without doing useful work thereby causing flyrock and air blast. This early re-

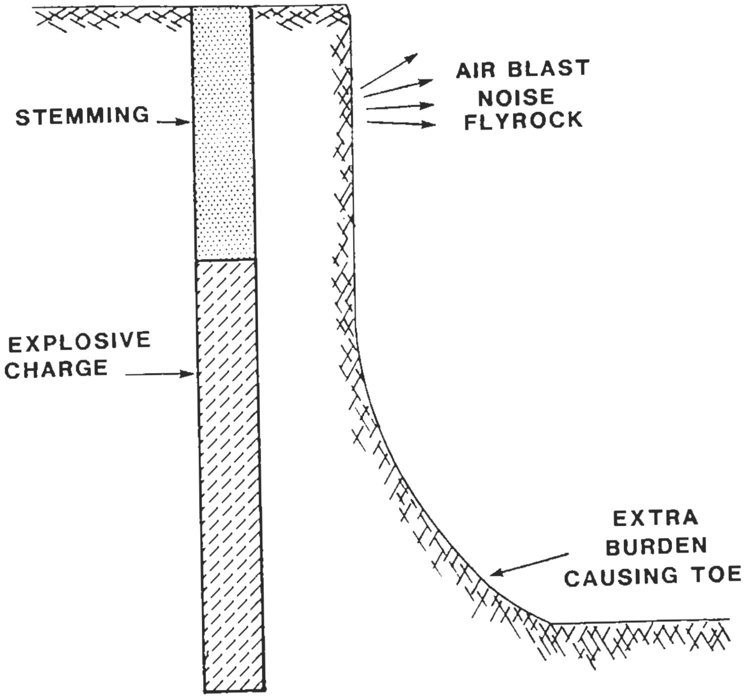

Figure 18.10 Effect of burden variation.

lease of gases generates air waves of large amplitude. Front row holes with small burden pose a problem of air blast specially in high and inclined faces (Fig. 18.10). Inclined holes, which offer uniform burden throughout are better in this situation.

Hole spacing along with delay between them is an important factor influencing air overpressure levels. It has been reported that strong air blast overpressures can be produced by the adjacent movement of a face during blasting if spacing between holes is less than the distance travelled by a sound wave during the delay time between adjacent blasthole detonations. To prevent any reinforcement of air blast from adjacent holes in a row of holes, it is recommended that the spacing between holes, divided by the time interval between the delay periods used in the holes, should be less than the sound velocity in air (330 m/s).

According to Wiss & Linehan (1978), the increased charge weight and blast confinement that result from the use of large diameter holes will produce higher amplitude ground vibrations than blasts using smaller diameter blast holes. More recent work by Redpath & Ricketts (1987) indicate that blast hole diameter has influence on vibration levels produced, suggesting that not only the weight of explosive per blast hole will determine the peak level of vibration. Large blasts of 100 holes are not expected to produce higher levels than 10-hole blasts, but merely to extend the duration of disturbance.

For minimising the ground vibrations the subdrilling length should be judiciously chosen, as the subdrilling portion creates a zone of extra explosive energy, which causes ground vibrations.

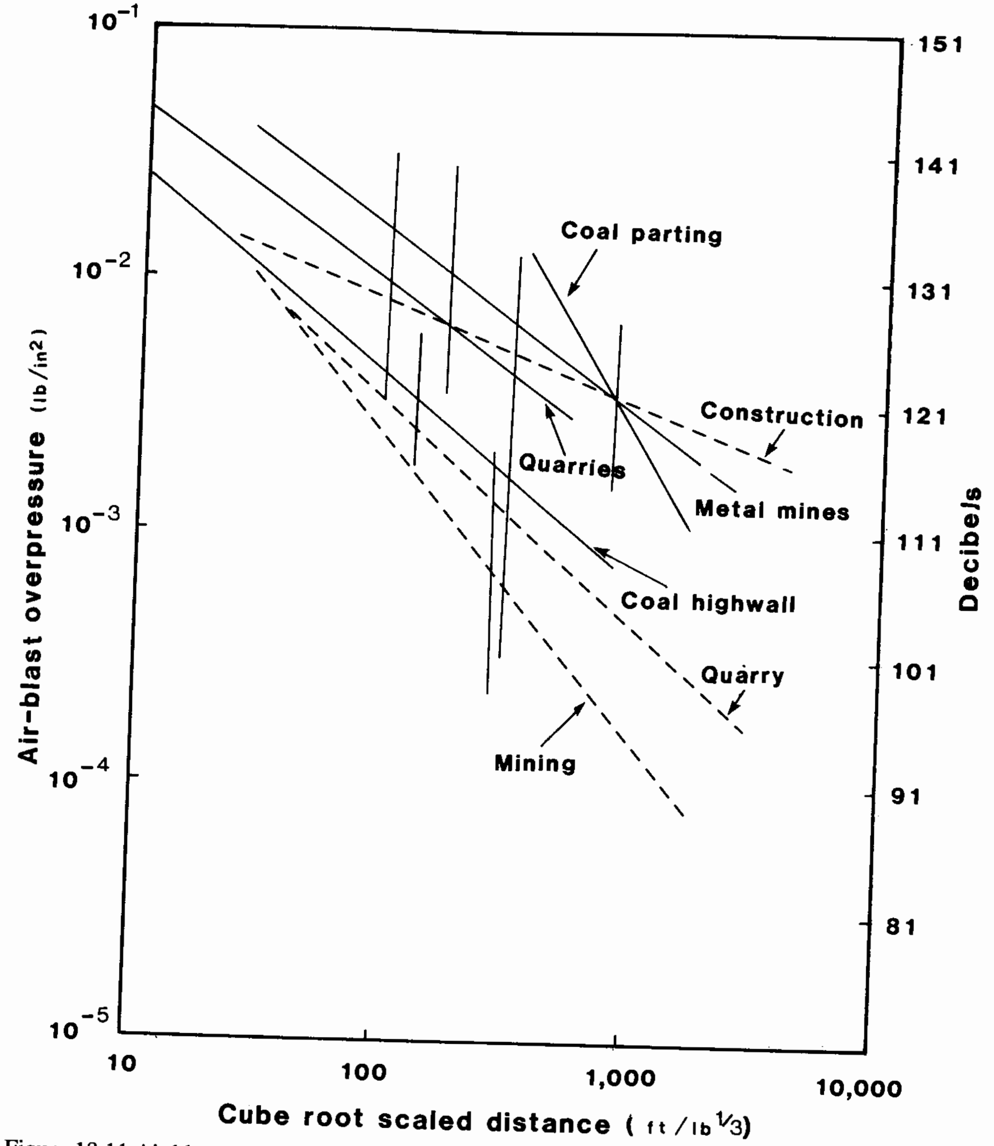

Figure 18.11 Air blast propagation from various types of mining (Siskind et al. 1980a).

18.4.2 *Type and amount of explosive*

The amplitude of pressure pulse is directly proportional to the amount of explosive used. The more the explosive the greater the energy released. The amount of explosive used per hole is also an important factor.

Charge weight per delay is most important factor which controls the intensity of ground vibrations. The larger the quantity of charge detonated per delay, the higher the vibrations.

The explosives which produce more gaseous energy than shock energy such as

ANFO, are more likely to produce ejection of gases and thereby producing higher amplitude air blast overpressures.

ANFO mixtures produce ground vibrations of lower intensity as compared to slurry and gelatinous explosives because of low density and low detonation pressure.

Siskind et al. (1980b) indicated that quarrying and construction or mining industries where finer fragmentation is needed, involving less confinement of charges, produce larger air overpressure levels (Fig. 18.11).

18.4.3 *Stemming*

There is an optimum stemming length beyond which further increments in stemming column serve no useful purpose. Ash (1968) recommended stemming lengths varying from 1/2-2/3 the burden value, depending on strata conditions. Smaller stemming columns of less than 20 times the diameter of hole may result in more ground vibrations.

The air blast levels created due to blasting are a function of the amount and efficiency of the stemming. If insufficient stemming is used, blowing out of stemming material may occur before gaseous energy is effectively utilised for fragmenting and displacing rock. Konya et al. (1981) in their studies found that the type and length of stemming controls the air blast. Their results indicated that stemming particle size of about one fourth of the blast hole diameter provides the best confinement. They suggested the stemming to burden ratio between 1-1.5. In the absence of blowing out of stemming air blast overpressure are known to drop by about 90% (Konya & Walter 1990). Air blasts could be minimised by using coarse and angular stemming material. Hagan & Kennedy (1977) recommended a stemming column of at least 25 times that of blasthole diameter (or about equal to burden), to minimise the air blast problem. The stemming length in the front row of blastholes should be adjusted on a hole to hole basis depending on the face configuration while the holes in subsequent rows can have planned stemming length.

18.4.4 *Priming and initiation*

The selection of suitable delay interval is very important in multirow large blasts. Ground vibration levels are reduced by the use of delay detonators because each delay generates its own pulses. The delay provides separate wave fronts emanating from the corresponding charges thus avoiding the superimposition of waves. The delay interval can be provided in between holes in the same row. Blasting in larger benches located in environmentally sensitive areas can be carried out by providing more than one delay within a single blasthole.

Proper burden relief has to be provided to each row, in the case of multirow blasts, for effective horizontal movement of the burden. If the delay between rows is smaller than what it should have been, the front row burden cannot move forward to sufficient distance to provide a free face to the next subsequent row. This adds to more confinement of charges in subsequent rows resulting in increased ground vibrations. Based on the burden rock movement, it was suggested that the optimum interdelay varies from 5 ms/m for short collars, high energy factors and strong massive rocks to about 10 ms/m for long stemming columns, low energy factors and weak or highly fractured strata (Hagan 1983).

Wiss & Linehan (1978) concluded from their studies that the pulses generated from two charges travel independently if the delay interval provided is more than 8 ms. Dick et al. (1983) recommended 8 or 9 ms as minimum delay that can be used between charges if they are to be considered as separate charges for vibration purposes.

Bottom initiation usually gives less noise and air blast but slightly higher ground vibration levels (Hagan & Kennedy 1977).

Whenever top priming is carried out the probability of air blast generation increases. The bottom priming on the other hand, however, produces less overpressure levels because of more confinement. In the case of top priming relatively larger stemming columns of greater than 30 times hole diameter are suggested.

The detonating cord commonly used in surface mine blasting is a major source of high frequency pressure pulses. This problem could be minimised by covering the trunkline with sand or drill cuttings. Other systems of initiation like Low Energy Detonating Cord (LEDC) and non-electric systems are preferred in environmentally conscious situations.

Use of millisecond delay detonators enable large quantities of explosives to be blasted in one shot by distributing explosives in a series of smaller charges. The distribution of charges on different delays should be carried out judiciously to avoid the additive effect from the pulses produced from each hole. Properly designed delays reduce the superimposition of air blast waves from successive charges maintaining a phase difference between them, thereby reducing the severity of air blast. If the effective delay provided between holes is proper, each pressure pulse coming from different holes behaves as a separate event, therefore, reducing the effective peak overpressure.

18.4.5 *Geomechanical characteristics of the strata*

The propagation of the ground vibration is strongly influenced by the lithology of the rock mass. Siskind et al. (1980b) reported that coal blasts produce low frequency waves compared to quarry and construction blasts, the reason being that, thick soil overburdens normally encountered in coal mining favour the development of low frequency surface waves. The strength, density, porosity of rock affect the propagation velocity of the waves significantly. Harder the rock more is the frequency. Blast vibrations are more intense in loose soil than in harder rocks (Bollinger 1971). The presence of discontinuities and the nature of filling material changes the direction of propagation.

Whenever the burden rock consists of highly fractured or jointed strata, there exists a possibility of escape of gases through them and which on reaching the atmosphere produce high air overpressure levels. If any weak band, mud seam, open joint extends from blast hole wall to the free face, the gaseous energy escapes through it creating air blast. Such weak zones should be properly loaded using decking or using weaker explosives.

In the area around a blast, the influence of rock characteristics on the rate of ground motion reduction with distance varies. Because of the inhomogeneous nature of rock it is impossible to make a reliable theoretical prediction of the rate of ground motion attenuation to be expected at a locality. The site factors which are determined by a ground motion monitoring programme are empirical allowances for the effect of local rock characteristics on ground motion.

Ground motion dissipation in rock is attributed to three mechanisms:

1. Viscous damping of ground vibrations, an effect more pronounced on higher frequencies and accompanied by a trend to lower ground vibration frequencies with increasing distance from a blast;

2. Solid friction absorption of energy in the ground motion wave, which is greater for rock with coarser grain structures and extensive porosity;

3. Scattering of the ground motion wave due to reflections at discontinuity and strata inhomogeneity in the rock, in which interactions between reflected pulses are often accompanied by a trend to selectively attenuate lower ground vibration frequencies.

Of the three above, the mechanisms which are the most unpredictable theoretically, bear the major responsibility for the reduction of peak ground motion levels.

Since rock masses are inhomogenous, ground motion waves travel through strata of different acoustical impedance. Scattering of the ground motion waves, at discontinuities, lowers the peak vibration levels. Interactions between reflected pulses alter the frequency composition of the wave train. High frequencies are selectively attenuated while some lower frequencies are added to the ground vibrations.

The presence of joints, fractures, faults, and shear zones in the path of a ground motion wave also act to scatter the peak vibrations. Some of the lateral components of ground motion are lost as the wave crosses a discontinuity. The degree of redirection and dissipation of a ground motion wave is related to the nature and frequency of structural discontinuities in rock.

The influences of rock mass characteristics on ground motion waves are too complex to be theoretically calculated. The mechanics of ground motion attenuation attributed to rock properties tend to produce a ground motion wavetrain with characteristic of the rock along the path of transmission. Thus by determining site attenuation factors in a blast, the geometric characteristics at that site can be predicted.

18.4.6 *Direction of blast*

Studies conducted by Wiss & Linehan (1978) revealed that the type of firing pattern also affects the air overpressure levels. Experimental work conducted by them showed that air blasts were at least 6 dB higher when the observation point was perpendicular to the firing pattern rather than parallel to blast face, due to blasthole reinforcement. It was also observed that if the direction of initiation was towards the affected location then increase of 10-15 dB was possible. This can be avoided by using proper delay intervals between holes. The air blast behind the face is normally weaker and less noisy because of the absence of high frequency components in this direction (Andrews 1975).

18.4.7 *Meteorological conditions*

Because air overpressure is transmitted through the atmosphere, meteorological conditions such as wind speed and direction, temperature, cloud cover and humidity will all affect the intensity of the air overpressure experienced at a distance from the blast site.

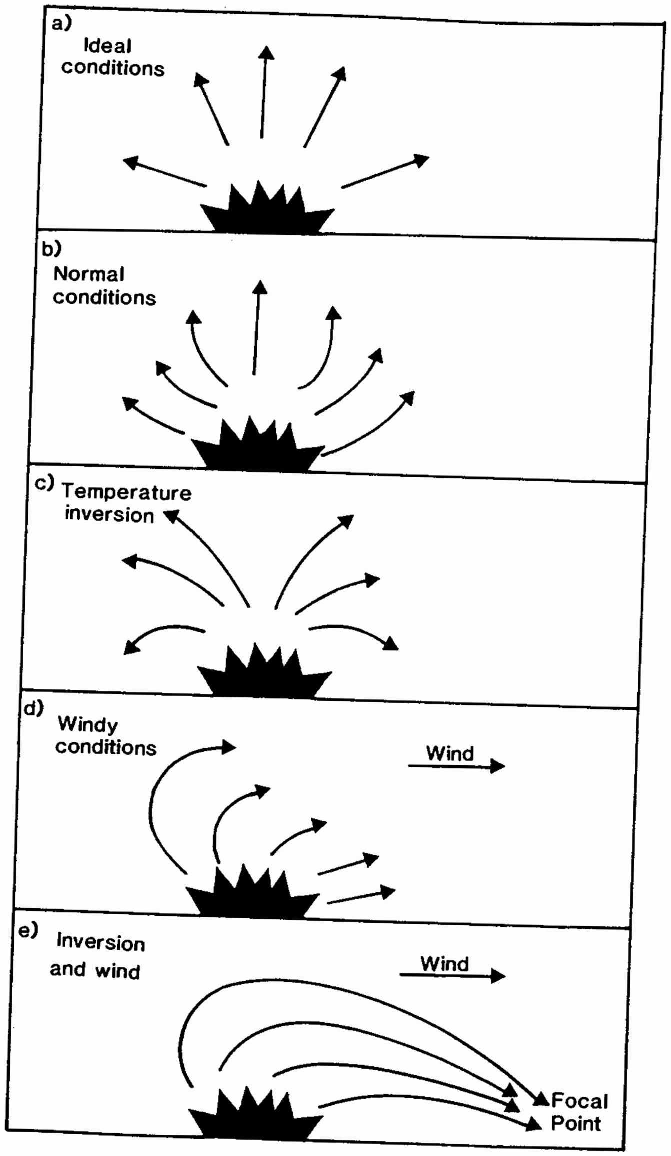

Figure 18.12 Effect of meteorological conditions on air blast propagation.

If a blast is detonated in a motionless atmosphere in which the air temperature is constant then the air overpressure intensity will solely depend on the distance of the source and will reduce by 6 dB as the distance from source doubles. Such conditions are very rare and it is more usual for temperature both to decrease and to increase with altitude in a fairly complex and changing manner. Winds are also invariably present at differing velocities and directions at differing altitudes. The overall result is that the nominal 6 dB reduction may be greater in some directions from the source and less in others.

The atmospheric conditions may affect the intensity of air blast at a particular

point. There are two regions of potential damage from air blast, 'near field' region surrounding the blast site and 'far field' region far from the blast site at unknown distance and direction. When the ground vibration is limited to 50 mm/s there is less likelihood of the any damage due to air blast in the near field region due to meteorological conditions (Walter 1983). Figure 18.12 shows the influence of wind direction and temperature inversions on air blasts.

Figure 18.12a shows the case with isothermal air and zero wind velocity conditions which would produce isovelocity conditions resulting in straight ray paths and spherical wave fronts. However, this is an ideal condition which does not exist in reality. Normally as the altitude increases, the atmospheric temperature decreases, which is known as adiabatic lapse rate, resulting in reducing the velocity of sound waves therefore refracting them upward (Fig. 18.12b). Under such conditions, there would not be much effect of air blast in far field regions, as the sound is absorbed in the atmosphere. Whenever a temperature inversion exists (where the temperature increases with altitude), the velocity of acoustic waves increases as the atmosphere becomes more dense near ground level thereby refracting the waves downward back to the ground (Fig. 18.12c). Under such conditions an increase in the overpressure levels by a factor of two to three is not uncommon. That is why blasts fired in early morning or in evening or night usually result in high noise and air blast levels. In early morning, following a clear night, with low wind speeds, the temperature at high altitude increases rapidly as compared to ground and the air immediately above it. However, as sunlight increases this temperature inversion is slowly eliminated. The presence of clouds also creates temperature inversion as they form a screen and prevent sunlight from providing ground heat to reduce the inversion.

Schomer et al. (1976) have shown that for propagation distances of 3-60 km, inversions produce zones of intensification of up to three times the average, attenuated or low air pressures at those distances, with an average increase of 1.8 times (5.1 dB). At distances less than 3 km, where high air overpressures are likely to occur, his measurements show no inversion effects.

Wind is another significant factor which influences the propagation of sound waves. Wind gradients are highly directional. The sound intensity and duration were found to be enhanced in the downward direction (Grant et al. 1967). On the downwind side, the wind adds to the velocity effect produced by the inversion, thereby increasing sound velocity and overpressure (Fig. 18.12d). It has been observed that in certain cases the overpressure created due to temperature inversion and wind (Fig. 18.12c) is as much as fifty times greater than that expected from a direct wave (Walter 1983). According to Down & Stocks (1977) in severe atmospheric conditions, the intensity of air overpressure levels may increase up to a hundred fold. Therefore, if possible, blasting should be avoided when there is atmospheric inversion and strong wind is blowing towards highly inhabited areas.

Wiss & Linehan (1978) study of air overpressures produced by surface coal mining showed that in moderate winds the typical 7.7 dB reduction for each doubling of distance is reduced by

$$7.7 - 1.6\, V_{mph} \cos \theta \text{ dB}$$

where V_{mph} is wind velocity in miles per hour and θ is the angle between the line connecting the blast and transducer and the wind direction.

18.4.8 *Distance and conditions of structures*

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

18.5 PREDICTION OF GROUND VIBRATION LEVELS

In blasting, it is extremely difficult to take into account of all the above parameters in a simple equation in predetermining the level of vibration which would be experienced at a given distance so that the quantity of explosive would not cause damage to a given structure. As a result, empirical approaches are widely used for ground vibration prediction. If the energy (E) released is taken to be directly proportional to the weight (W) of explosive detonated, an equation of the simple form $E \propto \sqrt{W}$ is observed.

Morris (1950) studied wave propagation phenomena and suggested that the amplitude (a) of particle displacement is proportional to the square root of the weight of the charge (W) and inversely proportional to the distance (D) from the blast, where K is the site constant

$$a = K.\ W^{0.5} / D$$

Habberjam & Whetton (1952) suggested a higher power for the charge weight in their formula

$$a \propto W^{0.085}$$

Other investigators (Blair & Duvall 1954; Duvall & Petkof 1959; Nicholls et al. 1971; Siskind et al. 1980b) proposed further modifications to the propagation law. Assuming cylindrical explosive geometry for long cylindrical charges, Duvall & Petkof (1959), Duvall & Fogelson (1962) of the US Bureau of Mines concluded that any linear dimension should scale with the square root of the charge weight. The corresponding relationship assumes the form

$$V = K\,(D / \sqrt{W})^{-\beta} \tag{18.1}$$

where $^{-\beta}$ is the slope of the best-fit straight line of the V versus D/W plot on a log-log scale, and K is the intercept on the particle velocity axis when $D/\sqrt{W} = 1$.

Langefors et al. (1958) and Langefors & Kihlstrom (1963) suggested the following relationship for various charging levels ($W/D^{3/2}$) to estimate peak particle velocity:

$$V = K \sqrt{(W / D^{3/2})^{\beta}} \tag{18.2}$$

For spherical geometry, the USBM investigators suggested that any linear dimension should be scaled to the cube root of charge size. Thus a scaled distance is actual distance divided by the cube root of the charge weight. Ambrasey & Hendron (1968) also obtained similar results. An inverse power law was suggested to relate amplitude of seismic waves and scaled distance to obtain the following relationship:

$$V = K\,(D / W^{1/3})^{-\beta} \tag{18.3}$$

The most common form of equation to predict the ground vibration levels at any distance from a blasthole or series of blastholes has the general form

$$V = K.\ W^{\alpha}\, D^{\beta} \tag{18.4}$$

where V is the peak particle velocity of ground vibration (measured in mm/s), W is the weight of explosive (kg) per delay period, D is the distance between blastholes and the measuring point (measured in meters) and K, α and β are site specific parameters relating to local geological conditions and explosive strength.

Ghosh & Daeman (1983) suggested that numerous inelastic effects cause energy losses during wave propagation. This inelastic effect leads to a decrease in amplitude in addition to those due to geometrical spreading. Ghosh & Daemen (1983) reformulated the propagation relations of USBM in terms of the peak particle velocity, maximum charge weight per delay (W), distance of the measuring transducer (D) and inelastic attenuation factor (p). The modified equations are:

$$V = K\,(D / \sqrt{W})^{-\beta}\, e^{pD} \tag{18.5}$$

$$V = K\,(D / W^{1/3})^{-\beta}\, e^{pD} \tag{18.6}$$

The empirical constants K, β and p may be obtained by regression analysis of two independent variables.

Pal Roy (1991) considered that when seismic waves propagate through a rock mass, they encounter an increasing volume of rock mass, causing decrease in energy density. In the process of this diminishing energy due to geometrical spreading which also cause energy loss during wave propagation and subsequently amplitude is decreased. The equation is of the form

$$V = n + K\,(D / W^{1/2})^{-1} \tag{18.7}$$

The empirical constant n is related to the category of parameters which is influenced by rock properties and geological discontinuities, whereas the empirical constant, 'K' is related to the category of parameters which is influenced by design parameters including charge weight, distance from explosion source, charge diameters, delay interval, burden, spacing, subdrilling, stemming length. As 'n' is a decay constant, its value should always be negative. Pal Roy (1991) has shown that charge per delay estimated from Langefors-Kihlstrom equation (Equation 18.2) is the maximum as compared to other equations up to a certain distance, for soft to medium hard rock. At larger distances (exceeding 150 m) and at higher vibration levels (exceeding approximately 10 mm/s), the explosive charge per delay as obtained by Equation 18.7 is the lowest. For moderate to hard rock mass including dolomite and granite, the charge per delay is the highest when estimated using Equation 18.3. Pal Roy (1991) has also shown that when explosive charge per delay is estimated using Equation 18.2, the blast results are near optimum, as compared to results obtained from other equations. Ghosh-Daemen model (Equations 18.5 and 18.6) is useful and logical for soft to medium hard rock.

Dowding (1985) states that frequency of vibration and ground strains are as critical as peak particle velocity in determining the response of the above ground structures. For below ground structures, frequency in combination with propagation velocity, controls response.

18.6 PREDICTION OF OVERPRESSURE LEVELS

For surface blasts, the combined effects of charge weight and distance from the blast source may be estimated by cube root scaling law

$$D_1 / D_2 = \sqrt[3]{W_1 / W_2} \tag{18.8}$$

where D is the distance between the blast charge and the point of observation and W is weight of the explosive charge required to produce a specified air blast overpressure.

McKenzie (1993) gives an equation to describe the decay of overpressure with distance D (in meters) as

$$\mathrm{dB} = K - c.\log (D / W^{1/3}) \tag{18.9}$$

where dB is the decibel reading with a linear of flat weighting, W is the maximum instantaneous charge weight per delay (measured in kg) and K, c are practice-specific constants. For practical purposes, the values of K and c are determined as 165 and 24 respectively. Hence Equation 18.9 becomes:

$$\mathrm{dB} = 165 - 24.\log (D / W^{1/3}) \tag{18.10}$$

For a maximum charge weight per delay of 30 kg overpressure limit needs a minimum distance of 354 m from the blast site.

Rzhevsky (1985) suggested the following equations to calculate the radii of zone of air blast action on men (r_m) and structures (r_s):

$$r_m \text{ (in meter)} = k\sqrt{W} \tag{18.11}$$

where k is the factor for blasting pattern with respect to free faces (k = 10-15), W is the total mass of simultaneously blasted charges in kg. The radius of air blast zone without damage to structures is given as

$$r_s \text{ (in meter)} = 200\sqrt[3]{W} \tag{18.12}$$

The air blast experienced in confined spaces underground, in conventional mining and tunnelling practice is liable to be higher than that experienced from the blasting above ground. Hanna & Zabetakis (1968) found that the cube-root scaling for peak overpressure in still air was only valid over a distance of about one tunnel distance. To illustrate the overpressures developed in typical room-and-pillar mining operation, Olson & Fletcher (1971) recorded air overpressure from three mine production blasts initiated with conventional half-second delay detonators. For that particular location and blasting system, the overpressure could be expressed by the equation

$$P = 4.9 \times 10^3 (D / W^{1/3})^{-2.15}$$

where P is overpressure in psi, D is distance from the blasting source (ft), W is zero-delay charge weight in lbs.

18.7 BLAST DAMAGE CRITERIA

Ground vibrations and airborne disturbances that result from blast are transient in nature. Transient means that the peak displacement is only temporary, lasts less than one-hundredth of a second, and the structure returns to its original position afterwards. Consideration is given to the effects which are associated with vibratory response of the structures in or on the rock or soil mass surrounding the blast.

Transient structural effects which occur have been grouped by Dowding (1992) as structural distortion, faulted or displaced cracks, falling objects, cosmetic cracking of wall coverings, excessive instrument and machinery response, human response, and micro disturbance. Often vibration levels are regulated so that the first four effects, those that relate to structural response, do not normally occur when vibration levels are regulated to prevent cosmetic cracking. Structural response has been classified into three categories (Edwards & Northwood 1960; Siskind et al. 1980b).

1. Threshold (cosmetic cracking) – Opening of old cracks and formation of new plaster cracks, dislodging of loose objects;
2. Minor (displaced cracks) – Superficial, not affecting the strength of the structures (e.g. broken windows, loosened or fallen plaster), hairline crack in masonry;
3. Major (permanent distortion) – Resulting in serious weakening of the structure (e.g. large cracks or shifting of foundations or bearing walls), major settlement resulting in distortion or weakening of superstructures.

The most common complaint of ground vibration due to blasting is the damage to the buildings. As the severity of blasting vibration increases, the magnitude of damage also increases. The order is as follows:

a) Dust falling from old plaster cracks;
b) Extension of old plaster cracks;
c) Formation of new plaster cracks;
d) Flaking of plaster;
e) Falling of plaster;
f) Masonry cracks form and partitions separate from exterior walls;
g) Further severe damage and ultimate collapse of the building.

Most of the complaints relate to items (a) to (d). However, plaster cracking in a building not subject to blasting is possible because of various reasons like poor workmanship, poor quality of material, settlement of ground etc. Plaster is particularly weak in tension and is brittle, and is, therefore, most vulnerable to cracking.

Normal activities like walking, door closing, jumping etc. can produce a good amount of vibration. Table 18.2 gives the range of particle velocities generated from normal activities.

Table 18.2 Range of particle velocities from normal activities.

Activity	Range of velocities observed (mm/s)	
	Same room	Adjacent room
Walking	0.01-0.10	0.025-0.063
Door closing	0.02-0.15	0.050-2.50
Jumping	0.12-12.5	0.012-1.25

Table 18.3 Risk of damage as classified by Langefors (1978).

	Sand, shingle clay (under ground-water level)	Moraine slate (soft lime-stone)	Hard limestone (quartzite sandstone, gneiss, granite, diabase)	Type of damage
Wave velocity in m/s	300-1500	2000-3000	4500-6000	
Vibration velocity noticeable in mm/s	4-28	33	70	No
	6-30	55	110	(cracking threshold value)
Insignificant	8-40	80		160
Cracking	12-60	115	230	Major crack

During the last few decades, much damage criteria has been established and implemented with varying degrees of success.

Between 1949 and 1960, different damage criteria based on displacement, velocity, or acceleration were established. In addition, complaints and lawsuits escalated, and it became apparent that the damage criteria varied considerably. At this time, another criterion of blast damage was developed based on particle velocity. It was generally agreed that the particle velocity of ground motion in the vicinity of the structure was the best damage criterion. Hence, attempts were made to convert past damage criteria in terms of particle velocity.

A few selected particle velocity damage criteria are listed as follows.

a) Langefors & Kihlstrom (1958) classified the risk of damage as shown in Table 18.3.

Blast induced effects on cracking can be scientifically observed only with visual inspection immediately before and after each blast. Because there is high probability of cracks being produced due to strains in structures due to environmental or human activity, it is important to eliminate those damages which are not due to blasting activity.

General recommendation was of keeping the vibration velocity below $V = 70$ mm/s in rock and $V = 50$ mm/s in ground with a lower propagation velocity.

They concluded that the damage was likely to occur with a particle velocity of 100-125 mm/s. A safe vibration limit of 50 mm/s was recommended.

b) Based upon the available data and USBM's own research, Nicholls et al. (1971) classified the severity of damage into three classes- no damage, minor damage, major damage.

Minor damage – formation of new fine cracks either in plaster or dry wall or opening of old cracks.

Major damage – serious cracking of plaster or dry wall and fall of material. It may indicate structural damage. They recommended the following safe vibration velocities.

No damage	<50 mm/s
Plaster cracking	50-100 mm/s
Minor damage	100-175 mm/s
Major damage	>175 mm/s

c) Bauer & Calder (1977) established the damage criteria for equipment and structures (see Table 18.4).

Table 18.4 Damage criteria suggested by Bauer & Calder 1977.

Type of structure	Type of damage	Particle velocity at which damage starts (mm/s)
Rigidly mounted mercury switches	Trip out	12.5
Houses	Plaster cracking	50
Concrete blocks in a new house	Cracks in block	200
Cased drill holes	Horizontal offset	375
Mechanical equipment pumps, compressors	Shafts misaligned	1000
Prefabricated metal building on concrete pads	Cracked pads, building twisted and distorted	1500

18.7.1 *Prediction of safe charge*

Scaling correlates ground motion levels at various distances from blasts. A scaling factor based on a dimensionless parameter for distance is used. The scaled distance is derived from effects of geometrical spreading on the outbound ground motion wave from an explosion.

The empirical scaling formula relating peak particle velocity to scaled distance has been developed from the results obtained in field using vibration monitoring equipment. Scaled distance D/W^{α} (α = 0.5 or 0.33) combines the effect of total charge weight, W, on the initial ground shock level with those of geometrical dispersion on the ground motion wave with increasing distance, D, from a blast. Square-root scaling or plotting peak particle velocity as a function of the distance, D, divided by the square root of the charge weight is more traditional than the cube-root scaling which incorporates energy consideration (Hendron 1977).

The site factors are determined from a logarithmic plot of peak particle velocity versus scaled distance as shown in Figure 18.13. The straight line best representing the data has a negative slope, β and an intercept, K, at a scaled distance of 1. In almost all cases, the scatter from data obtained is so high that in order to establish a statistically good approximation of the site factors, an extremely large number of tests would have to be set up. As this is not practical in most instances, limit lines can be established which encompass all the data obtained from many different sites.

USBM researchers monitored a large number of blast records to establish the following two safe scaled distances for proper field use

$$\frac{D}{\sqrt{W}} > 22.5\ \mathrm{m/kg^{1/2}} \tag{18.13}$$

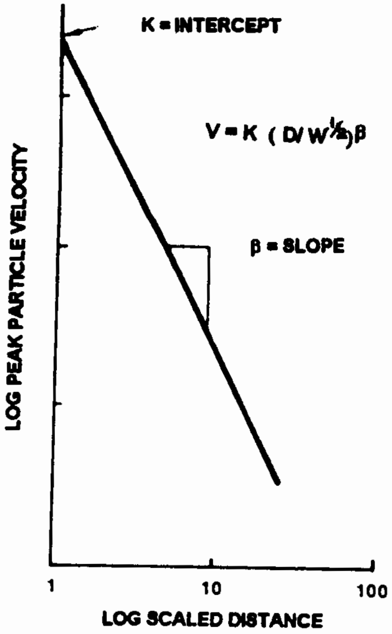

Figure 18.13 Peak particle velocity versus scaled distance showing empirical site factors.

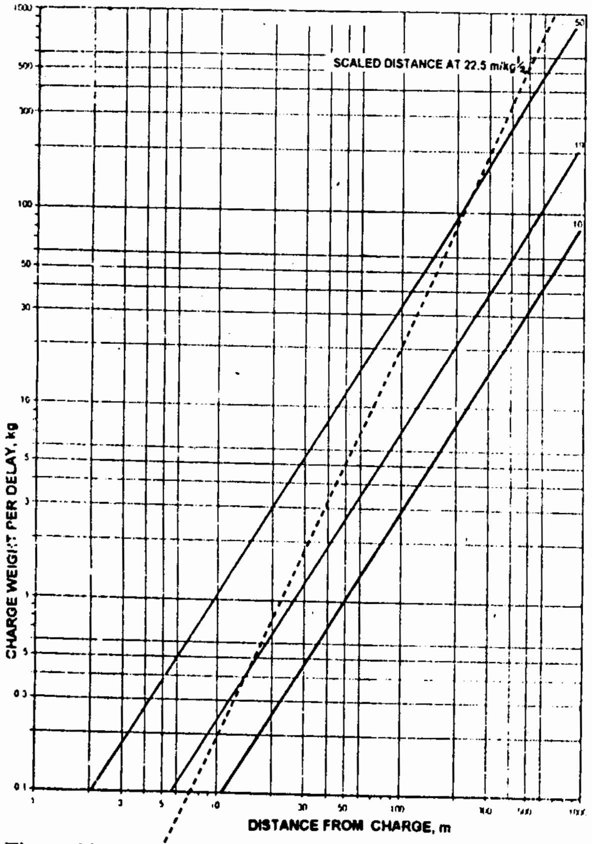

Figure 18.14 Estimating vibration levels for average rock under average conditions.

Table 18.5 Maximum charge/delay for scaled blasting distances of 22.5.

Distance (m)	Charge/delay (kg)	Distance (m)	Charge/delay (kg)
50	4.9	250	78.0
75	11.0	300	122.0
100	19.5	350	176.0
125	30.5	400	313.0
175	44.0	450	489.0
200			

$$\frac{D}{\sqrt{W}} > 9\ \mathrm{m/kg^{1/2}} \tag{18.14}$$

These equations represent the minimum scaled distance of recommendations for safe blasting. Equation 18.13 is recommended on sites where no instrument readings have been made because the equation introduces a safety factor to allow for the possibility of high seismic energy generation and propagation effect. Equation 18.14 is only recommended for sites that have actually been instrumented and if peak particle velocities of less than 50 mm/s were obtained. If readings are scattered or questionable and a mine operator wishes to use a smaller scaled distance to allow for large explosive consumption per delay, then a consultant and/or permission from the regulatory agency in the area may be required. Typical maximum charge per delay values for safe blasting distances and scaled distance of 22.5 are tabulated in Table 18.5. Figure 18.4 can be used for eliminating ground vibration levels.

These damage criterion of using a scaled distance of 22.5 implies that the probability of producing a peak particle velocity greater than 50 mm/s is very remote.

To simplify calculations, $D/\sqrt{W} = 22.5$ can be rewritten in the following two alternate forms:

$$D = D_s\sqrt{W} \quad \text{or} \quad W = (D/D_s)^2 \tag{18.15}$$

where D = Distance form blast to nearest structure, m, D_s = Scaled distance, W = Weight of explosive per delay, kg.

For example, if the closest structure to a blast is located 500 m away, and a scaled distance of 22.5 m/kg$^{1/2}$ is used, the maximum charge weight per delay is calculated as

$$W = (D/D_s)^2 = (500/22.5)^2 = 493.8\ \mathrm{kg}$$

This means that an operator could blast one hole per delay containing 494 kg of explosive. 2 holes per delay with each hole containing 247 kg of explosive, 3 holes per delay with each hole containing 164.5 kg of explosive.

From Figure 18.13 a reasonable separation between the safe and damage zones appears to be at a particle velocity of 50 mm/s. The recommended 50 mm/s is probability type criterion. If the observed particle velocity exceeds 50 mm/s there is a reasonable probability that the damage will occur to residential structures. The safe vibration criterion is not a value below which damage will not occur and above which damage will occur. Figure 18.14 represents velocity data from tests in which damage was not observed. However, the probability of damage to a residential structure increases or decreases as the vibration level increases or decreases from 50 mm/s.

In view of persistent blasting complaints, the OSM (Office of Surface Mining, USA), between 1971-1976 recommended using an increased scaled distance factor D_s = 27 in an attempt to reduce the maximum particle velocity to 25 mm/s. There was also some discussion of possibly reducing it even further to 12.5 mm/s.

However, Medearis (1976) reported that specifying a maximum ground particle velocity alone did not take into account two very significant parameters, namely the predominant frequencies of the ground motion and the structure being excited.

Although a number of authorities have adopted the criteria of maximum particle velocity, this has been ruled out in certain courts because the frequency content of the waveform and type of structure were not specified. Consequently, many authorities lowered the maximum particle velocity from 50 mm/s to as low as 5 mm/s rather than include frequency, and often without a scientific base.

The USBM began to reanalyse the blast damage problem and subsequently Siskind et al. (1980b) published the results of a comprehensive study of ground vibrations produced by blasting.

1. The amplitude, frequencies, and durations of ground vibrations change as they propagate, because of:
   - Interactions of various geological media and structural interfaces;
   - Spreading out the wave-train through dispersion; and/or
   - Absorption which is greater for the higher frequencies.
2. Close to blast, the vibration character is affected by factors of blast design, mine geometry, charge weight per delay, delay interval, direction of initiation, burden, and spacing.
3. At large distances from blast, the factors of blast design become less critical and the transmitting medium of rock and soil overburden dominate the wave characteristics.
4. Particle velocity amplitudes are approximately maintained as the seismic energy travel from one material into another (i.e. rock to soil).
5. Vibration frequency, displacement and acceleration amplitudes depend strongly on the propagating media.
6. Thick soil overburden and large distances create long-duration, low-frequency wave trains. This increases the response and damage potential of nearby structures.
7. Coal mine shots are characterised by a trailing large amplitude, low frequency wave because of larger overburden layers.
8. The combined effect of large shots, thick overburdens, good confinement and long range propagation make coal mine blast vibrations potentially more serious than quarry and construction blasts because of their lower frequencies.
9. Natural frequencies of midwalls are somewhat higher than for the structure corners.
10. Damping for midwalls was generally lower than that for structure corners. Damping controls the decay of oscillation, so that when a structure is critically damped (p = 1.0), it will return to its equilibrium position without oscillating.
11. Maximum amplification occurs for a single storey and double storey structures.

18.8 AIR BLAST EFFECTS

Many studies have been conducted to determine damage potential of explosive detonated in the air, on surface and near surface. In well contained explosive charge such as in mining, the effects of air blast on the structures are meagre compared to ground vibrations. However, the blast can be sufficiently intense to cause structural damage at about 180 dB (0.2 kg/cm^2) or greater, those produced by routine blasting under normal atmospheric conditions are not likely to do so. Many investigators indicate that 90% of routine quarry blasts produce air overpressure levels near adjacent residential areas below 120 dB level. The weakest part of a structure that will be exposed to air blast are windows, and so these are most likely to suffer damage. Poorly mounted panes might be forced out of their frames while improperly mounted panes that are prestressed will be cracked and broken more easily. Air overpressure values of 150 dB could be enough to damage a badly mounted pane, with most windows breaking at 170 dB. Structural damage may be expected at 180 dB.

US Bureau of Mines suggested an overpressure of 164 dB (0.033 kg/cm^2) as a safe limit for glass breakage. Later, Siskind & Summers (1974) summarised their results of air blast studies as in Table 18.6. These limits were further modified keeping in view damage and annoyance of a person (Siskind et al. 1980a).

Australian Environment Council recommends a safe level of 115 dB on working days and stricter limit of 95 dB on holidays and during nights (McKenzie 1993). In the UK it is more common to adhere to 120 dB limit (Wilton 1991).

Currently limits in the US are based upon wall response necessary to produce wall strains equivalent to those produced by surface coal mining-induced ground motions with peak particle velocity of 19 mm/s. These limits are presented in Table 18.7 (Dowding 1992). Because of different sound weighting scales that might be employed by monitoring instruments, the recommended levels in Table 18.7 differ by instrument system. Since structures are most sensitive to low frequency motions and the greatest air pressure occur at these subaudible frequencies, A-weighted scale cannot be employed at all. Since C-weighted scale is the least sensitive at low frequencies, its use requires the most restrictive limits.

Table 18.6 Effect of air blast on ordinary structures.

Overpressure dB	kPa	Air blast effect
177	14.00	All windows break
170	6.30	Most windows break
150	0.63	Some windows break
140	0.20	Some large plate glass windows may break, discs and loose windows rattle
136	0.13	USBM interim limit for allowable air blast
128	0.05	Complaints likely

Table 18.7 Air overpressure limits as a function of instrument frequency weighting.

Lower frequency limit of measuring system, in Hz (−3 dB)	Maximum level, in dB
0.1 Hz or lower – flat response*	134
2 Hz or lower – flat response	133
6 Hz or lower – flat response	129
C-weighted slow response	105 dBC

* Only when approved by regulatory authority.

18.9 HUMAN RESPONSE

Human reaction to blast induced vibration, especially within buildings, is a very complex phenomenon. The incoming vibration, whether ground or airborne, has characteristics that may even combine with those of the property to produce secondary noise effects which will generally compound the effect as far as any individual is concerned.

Further, the susceptibility of individuals will vary depending on their activity at the instant of the vibration arrival and also varies with factors such as age, mental attitude and previous exposure. Pal Roy (1994) concluded that often response and complaints are biased either due to service or business interest with the operator or with aim of getting compensation.

Motion and noise can be startling and can lead to a search for some physical manifestation of the startling phenomena. Many times, a previously unnoticed crack provides such confirmation of the event. Furthermore, if a person is worried and observes a crack that was not noticed before the crack's perceived significance increases over one noticed in the absence of any startling activity. These concerns are real and in the mind of observer are sincere.

In typical mining situations, significant blast-induced inaudible air overpressure and audible noise immediately follows the ground motion and intensifies human response. Both the ground and airborne disturbances excite walls, rattle dishes, and together tend to produce more noise inside a structure than outside. Thus both the audible noise as well as the wall rattle produced by subaudible air pressure can vibrate walls to produce audible noise at large distances, which are inaccurately reported by occupants as ground motions.

Most of the complaints regarding the blasting are due to air blast noise because any impulsive noise exceeding the background level by more than 10 dB is potentially disturbing and if it is repeated it may be even more complicated. Even at a safe level, neighbours feel that damage must be occurring because of high noises produced by air blasts. As explained earlier, under favourable weather conditions noise produced by the air blast propagates to further distances and in turn increases the complaints.

General human factors like age, health, tolerance limit and education all have some direct and indirect effects on human response.

18.10 CONTROLLING GROUND VIBRATION AND AIR BLAST EFFECTS

Control of blast effects near critical rock masses of constructed facilities depend upon two main considerations. First, shot designs must reduce the amount of explosives detonated at any instant and adjust the initiation sequence and reduce resulting ground and airborne disturbances. Second, the amount of explosive detonated per unit volume of rock and shot pattern must be adjusted to ensure adequate fragmentation. Therefore, at the same time, the initiation sequence must be separated in time but not in space.

Although ground vibrations and air blast are difficult to control because of their variability, much can be done to reduce their effect. The aim of any optimised blasting operation should be both to increase the efficiency of operation and to reduce the impact of the operation on structures and neighbours. This requires the operator to develop as much as understanding as possible of all the processes involved, and to make decisions based on the observed behaviour.

To reduce ground vibrations and air blasts, design of blasts need to be optimised. It needs to be ensured that the burden distance and the length of stemming column are adequate to prevent the premature release of gases. Accurate drilling of holes need to be carried out to avoid uneven burden. If excessive backbreak from previous blast is present, additional stemming should be used in the first row. Coarse stemming material will give better charge confinement than finer material.

Reduce the total charge weight by using delays. The reduction of the amount of explosive can be achieved by using smaller hole diameters, reduced bench height and/or decking. If the interval between the delay periods is at least 8 ms, the individual delays can be often treated as separate blasts and their vibration effects not cumulative. Use electric ms detonators and/or a sequential blasting machine to obtain the maximum number of delay intervals.

Eliminate hole to hole propagation between charges intended to detonate at different delay periods. Hole to hole propagation can occur when the explosive charges or boreholes are close as in benching, decked holes or in the presence of cracks.

Although distance between the blast area and residential structures is usually fixed, early planning can utilise the mining sequence such that the first blasts are strategically located farthest from the structures until a good public relations program has been adopted to deal with future decreased distances.

Excessive subdrilling and shorter stemming columns of less than 2/3 of burden should be avoided. Ensure that the burden distance and the length of the stemming column are adequate to prevent the premature release of the gases of detonation. If excessive backbreak from the previous blast is present, additional stemming should be used in the first row. Proper blast design avoiding excessive burden reduces the vibration levels. It has been observed that compared to multiple *V*-pattern, a flat *V*-pattern gives less vibrations as it gives more relief initially. It is better to reduce number of holes in the first delay of the blast. In some countries or states, regulations require blasts to be designed using a scaled distance of 27 (corresponding to maximum peak particle velocity of 25 mm/s). Where these are too restrictive, permission from the regulatory authority for a lower scaled distance may be obtained by providing a system monitoring scheme. If the program employed allows measurements of maximum particle velocity, maximum displacement and frequency of disturbance, then the US Bureau of Mines criteria illustrated in Figure 18.14 can be used with confidence.

Wherever mud seams, open joints or other weakness planes exist, in these zones deck charging with non-explosive stemming material be carried out. Use of satellite holes and pocket charges in stemming columns be avoided.

A reduction in the amount of detonating cord used together with adequate covering of the cord above the ground reduces overpressure intensity especially in the audible frequency range. Blasting with low explosive core load for trunk line is often practised and in sensitive areas even the use of low energy detonating cord (LEDC) such as Nonel is preferred. Either with LEDC or with electric blasting detonators charge initiation is preferable either at or near the bottom of the holes. Top initiation of the charge may prematurely release gases and thereby produce strong air blast. Electric delay detonators often deviate from nominal delay interval, therefore the use of sequential timer, or electronic detonators are preferable. The firing pattern should not be perpendicular to the site of interest. Superposition of air blast pulses can be minimised by controlling the rate of progression along the face. The simultaneous firing of opening holes in a pattern is to be avoided if air blast is a problem.

Secondary blasting with plaster shooting is to be avoided and preferably replaced by alternatives to blasting e.g. drop hammer.

It is better to carry out a large blast instead of a number of small blasts because the human response is found to be more dependent on the number of blasts.

If air blast complaints are continual problem it would be advisable to postpone blasting during unfavourable weather conditions if possible. Under some conditions focusing of air blast waves can take place even in remote places. A blast should not be fired when the inversion ceiling is below 30,000 m and complaints are likely to arise from those in the downward side from the blast site. Blasting should be carried out when the favourable weather conditions prevail. The favourable conditions for blasting are clear to partly cloudy sky, light winds and a steadily increasing surface temperature from day break to blast time. Blasting be conducted during the periods of high ambient noise. Australian Environment Council's air overpressure recommendations which have been adopted in New South Wales by their State Pollution Control Council indicate time and days for blasting and the limits are more stringent for weekends and for period between sunset and sunrise.

Blasting is to be avoided when surface winds are blowing towards the residential area or when the wind gradient exceeds 10 km/hour per 300 m altitude.

It is useful to have some barrier between the blast site and residential area. Example of such a barrier is growing of vegetation which works as screen and reduces an appreciable amount of noise.

Observations from full wave form overpressure signals indicate that the peak overpressure levels generated by blasting are generally produced from blast holes in the front row. Therefore it would be better to have more number of rows to have increased environment control.

Low overpressure limits are difficult to meet with in general. For example, McKenzie (1993) shows that under favourable weather conditions to meet 115 dB limit a distance of more than 350 m is required for 30 kg charge weight per delay. Under some unfavourable conditions, the charge may have to be restricted to about 1 kg. Specially problems arise when charging is partly completed and weather conditions change.

However, the unpredictable and uncontrollable effects of prevalent atmospheric conditions suggest that both the magnitude of air overpressure and the location of its maximum effect are uncertain. It is readily obscured by low level wind. Such overpressure conditions ensure that both the magnitude of air overpressure conditions imply a degree of control that does not exist. Thus as practical measure one needs to ensure that uncertain weather conditions are taken into account.

CHAPTER 19

Flyrock during blasting operations

19.1 INTRODUCTION

When blasting operations are carried out, the rock gets fragmented and the fragmented material is moved forward to make mucking of the fragmented mass easier and less costly. In addition to this desirable displacement of broken fragments in case of surface mine blasting or excavation blasts some stone pieces can get torn and travel to very large distances. Usually this unexpected projection of stone is termed as 'Flyrock' (Fig. 19.1). Flyrock is a serious environmental hazard and is often a cause of fatalities, serious injury to people, damage to equipment, buildings, property, etc. Damage due to flyrock from blasting is one of the main causes of strained relations between the management and people residing in the vicinity of blasting operations. These hazards are serious around open pit mines as the blasting rounds become bigger with larger diameter boreholes where flyrock has been known to travel a kilometre or more from the blast site (Roth 1979). This means that very large areas must be evacuated to avoid accidents. People must be protected, no matter at what cost. If necessary measures to counteract flyrock are taken, the risk of damage due to flyrock may be reduced to minimum.

This chapter describes some of the problems associated with flyrock – causes of flyrock, distance to which it can be propelled and precautions to be taken to avoid them.

19.2 CAUSES OF FLYROCK

In a survey of blasting operations it was observed that out of 34 accidents that occurred during scheduled blasting, 28 were attributed to flyrock (Britton et al. 1977). These accidents resulted in 10 fatalities and 19 injuries. A study of blasting accidents in Indian mines indicated that more than 40% fatal and 20% serious accidents resulting from blasting occurred due to flyrock (Bhandari, 1994). In the United Kingdom during 1980-85, 103 incidents were reported of which more than half of these involved flyrock projections to distance ranging from 350 m to a maximum of 900 m, with rocks weighing up to 500 kg. According Mine Safety and Health Administration, flyrock is the leading cause of blasting accidents, accounting over 60% percent of all accidents during 1987-88 (Anon. 1990). Following are some of the reasons of flyrock arising due to blasting operations.

Figure 19.1 Flyrock during blasting.

19.2.1 *Inadequate burden and spacing*

The blast design parameters such as the burden and spacing need to be calculated taking into account the hole diameter, rock characteristics and fragmentation requirements. An insufficient (or less) burden will cause breakthrough of drill hole charges, resulting in flyrock (Fig. 19.2). A burden dimension less than 25 times the charge diameter gives high specific charge, hence the excess energy results in long flyrock distances. Burden to bench height ratio less than 1.5 is also a cause of flyrock. Too large burden will cause venting of stemming material and also cratering effects giving rise to flyrock (Fig. 19.3). The use of vertical holes usually leads to considerable variation in burden at the top and bottom of the face. This is the case particularly where the face is high or is highly inclined. Front row blastholes are often collared near the crest so as to remove the heavy toe burden. Due to this explosion gases may blow-out prematurely in the upper face, causing high levels of noise, air blast and flyrock (Fig. 19.4). This effect is more pronounced for top primed charges.

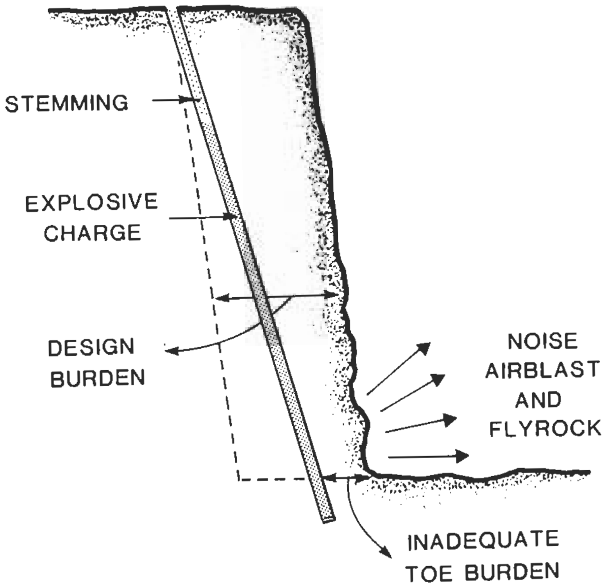

Figure 19.2 Insufficient burden caused due to deviation of hole results in flyrock.

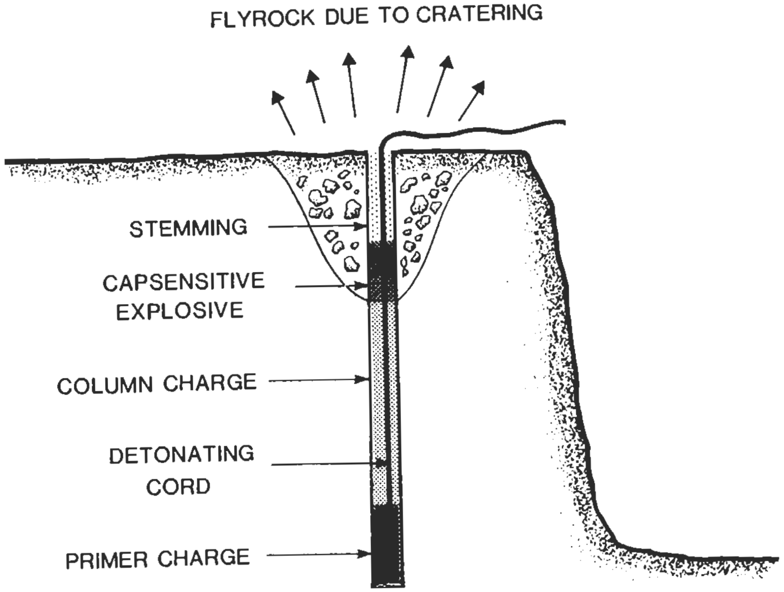

Figure 19.3 Large burden and top priming causes flyrock.

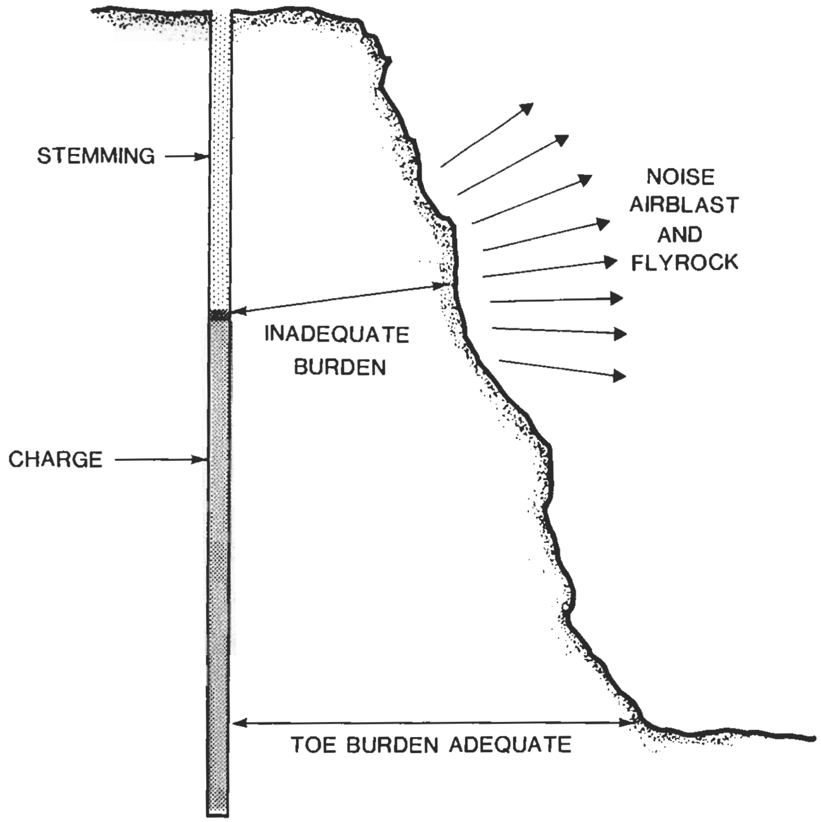

Figure 19.4 Collaring too near top of the face in high bench or highly inclined face.

Spacing appreciably less than the burden tend to cause premature splitting between blastholes and early loosening of the stemming. Both these effects encourage rapid release of gases to the atmosphere and flyrock is considerable. Too close a spacing causes crushing between holes and also cratering giving rise to flyrock. Spacing appreciably less than the burden tend to cause premature splitting between blastholes and early loosening of the stemming. Both these effects encourage rapid release of gases to the atmosphere and flyrock is considerable.

19.2.2 *Overloaded holes*

The distance to which a flyrock travels will depend on the amount of specific charge. A high specific charge throws flyrock to a longer distance than a low specific charge. This means that overloading of holes may result in excessive flyrock. When cavities are present in the strata or due to the negligence of the blasting crew if explosive charge is excessively loaded in the blastholes, then such a blast may result in excessive flyrock (Fig. 19.5).

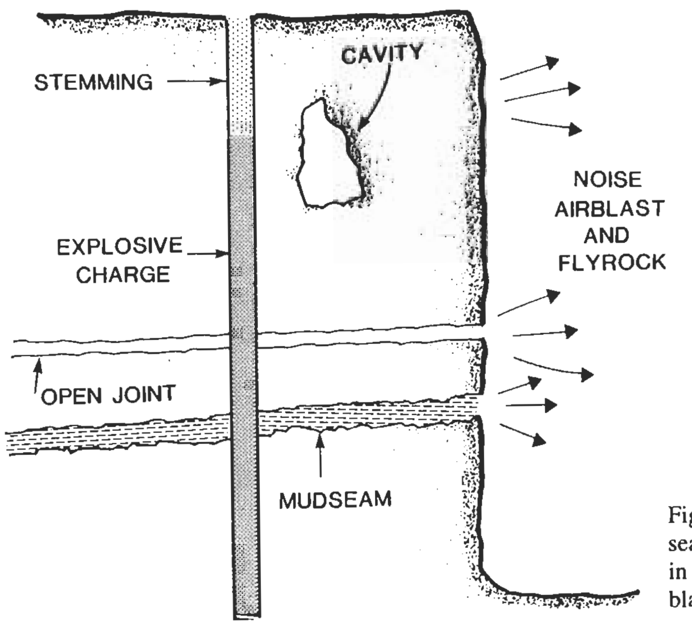

Figure 19.5 Open joints, mud seams and cavities may result in escape of gases and also blasting nuisances.

19.2.3 *Geological conditions*

Zones of weakness and voids are often causes of flyrock. Any explosive loaded in this zone will have the line of least resistance and 'blow out', causing flyrock (Fig. 19.5). Where holes loaded with explosives intersect or lie in close proximity of faults, weakness planes or joints, the high pressure gas upon initiation of the explosive jet out along these paths of low resistance, which in addition tear out rock pieces and give rise to flyrock. An abnormal lack of resistance to drill penetration usually indicates a mud seam, a zone of incompetent rock or even a void.

19.2.4 *Inaccurate drilling*

Accurate drilling provides proper burden and spacing which are essential for better results. This is achieved by proper positioning of drill to accurately collar the hole and by proper inclination of holes. However, if during drilling, the driller is not guided properly for the position and direction of the hole, the drill may deviate from its calculated position and during the time of blast the effect may be same as that of reduced burden and overloaded hole thereby resulting in flyrock (Fig. 19.2).

19.2.5 *Inadequate stemming*

Stemming improves rock breakage by confining the gases in the hole to effectively fracture and heave the rock. If stemming column is inadequate, the explosion gases are not forced to heave up the partly fractured ground but are simply allowed to 'rifle' out of the top of the blasthole at very high velocity causing considerable flyrock and air blast. Bigger size (more than ten mm size) stemming material may also fly to a longer distance as compared to fine material.

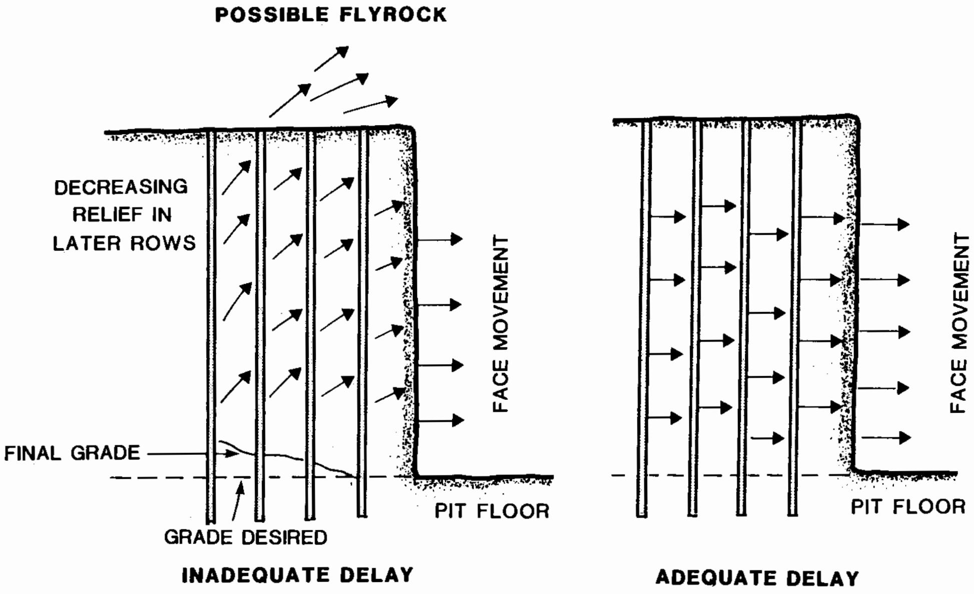

Figure 19.6 Effect of adequate and inadequate delay.

19.2.6 *Faulty delay timing and initiation sequence*

Optimum fragmentation and displacement is achieved in blasting operations by giving sufficient inter-row or inter-hole delays. The delay is needed so that the fragmented rock from the previously fired holes has enough time to move forward and accommodate the broken rock from subsequent rows. If the delay is not sufficient, movement from the back rows (in multirow blasts) will be upward rather than forward, giving rise to flyrock (Fig. 19.6). Lack of proper delay timing leads to crowding and excess burden which in turn leads to cratering thus giving rise to flyrock.

19.2.7 *Miscellaneous causes*

Secondary blasting viz. pop shooting can give rise to potential flyrock. When a hole is drilled to carefully calculated depth in stone, and if the burden is too small in any direction, the blasting results deteriorate and flyrock is propelled to a large distance.

Removal of toe by using short drill holes can cause excessive flyrock if not carefully carried out. The loose fragments at the slope of the face or at the bench may fly to a longer distance as compared to blasted rock. Larger diameter holes produce more flyrock as compared to small diameter holes due to use of heavier explosive charges.

19.3 CALCULATION OF FLYROCK DISTANCE

A safe blasting area is dependent on the knowledge of distance to which flyrock will propel. Some attempts have been made to estimate this distance.

Lundborg et al. (1975) used a semi-empirical approach to estimate flyrock throw distance. Based on conservation of momentum and the scaling laws of spherical charges a relationship between charge diameter d and rock velocity V was obtained. Once V is known then the flyrock range (L_m) is calculated from the equation of ballistic trajectories. For bench blasting Lundborg et al. (1975) proposed

$$L_m = 260\, d^{2\backslash 3}$$

where L_m is in meters and d is hole diameter in inches. In another experimental study agreement between theory and experiment has been found to be reasonable. The diameter (ϕ) of these stones are

$$\phi = 0.1\, d^{2/3}$$

where ϕ is in meters and d is in inches.

An alternative approach was attempted by Roth (1979) for obtaining the flyrock range. In this approach the critical variable in all flyrock range calculations was the estimation of V, the initial flyrock velocity. On comparison between calculated and field measured velocity data, the former values were found to be 1.6 times greater than the observed velocity.

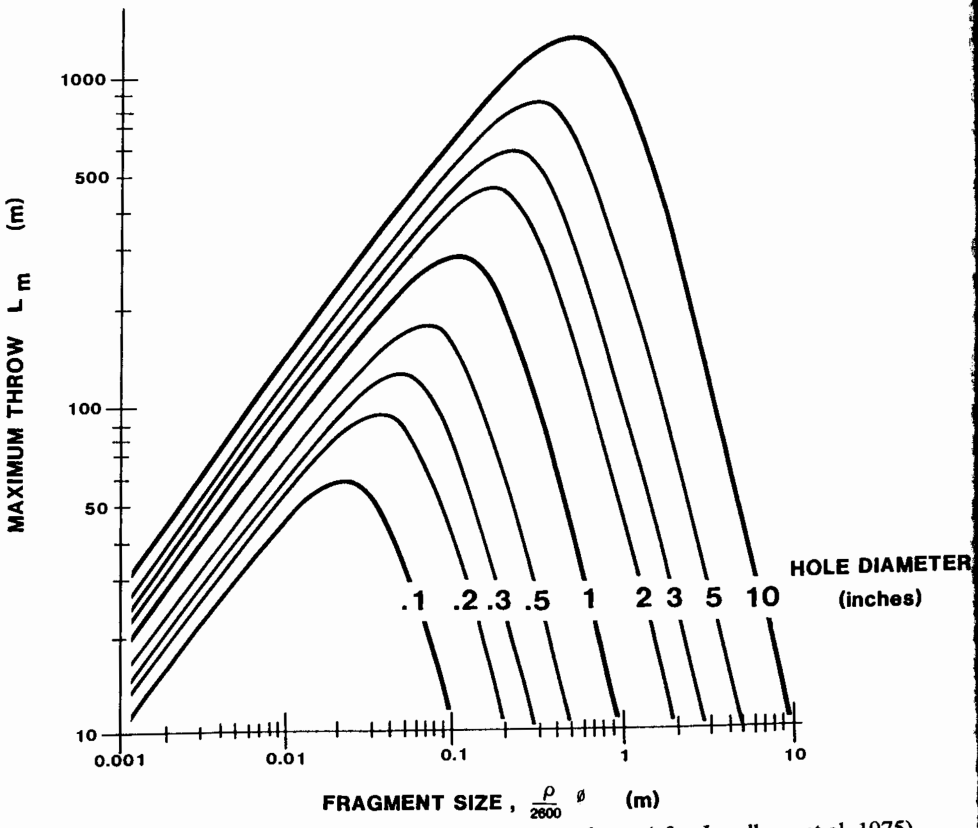

Figure 19.7 Relation between fragment size and maximum throw (after Lundburg et al. 1975).

One of the most extensive study of the distance that flyrock is thrown was conducted by Lundborg (1981). His work was based on the observations that the flyrock distance and exit velocity were proportional to the specific charge or powder factor. The result of the Lundborg's work is shown in Figure 19.7, where the throw distance has been calculated from field tests at quarries and surface mines by taking into account initial impulse energy and air resistance.

In Figure 19.7 the maximum throw distance, L is shown to be a function of hole diameter d and flyrock diameter, ϕ, in meters. The distance calculated is also affected by specific charge. Larger specific charges produce greater throw distances. Since most production blasting involves holes 75-250 mm in diameter this figure indicates that the maximum distance can be in the range of 500-1000 m. However, the probability of being struck by such a stone is extremely low. For instance Lundborg (1974) calculated that at a distance of 600 m while blasting a 76 mm diameter hole the probability of being struck by a flyrock was the same as being struck by a lightening (1 in 10,000,000).

Example

In a rock phosphate mine in India, 112, 150 and 250 mm diameter drill holes were used with a specific charge for the blasting as 0.56 kg/m^3, the flyrock range was calculated based on Lundborg's report by Rathore (1989). For a specific charge of 0.56 kg/m^3 and hole diameter 112, 150 and 250 mm shows that the maximum throw of flyrock will be 520, 700 and 1020 m respectively. For a specific charge of 0.75 kg/m^3 the maximum throw of flyrock for a hole diameter of 32 mm will be 265 m.

19.4 MEASURES TO CONTROL FLYROCK DAMAGE

It is essential to take proper measures to control flyrock, which are discussed in Sections 19.4.1-19.4.6.

19.4.1 *Proper blast design*

Proper blast design is most important not only for optimum performance but also from the point of view of safety. It includes the following important parameters:

Bench height. Benches of about 10-18 m are considered the most economical and least hazardous to work. Though the most economical bench height is often decided by drilling and loading equipment, however, deep holes deviate and increase chances of flyrock, and therefore the bench height should be restricted accordingly.

Blasthole diameter. It is governed by the properties of the ground, the degree of fragmentation required and the relative economics of different types of drilling and loading equipment. The large diameter holes increase the flyrock travel distances, and thus the drill hole diameter should be chosen according to the distances up to which flyrock can be tolerated.

Inclination of blasthole. Inclined blastholes give increased displacement and consequently muck piles are more suited for front end loaders but increased flyrock is experienced. On the other hand the use of vertical holes on high benches usually give considerable variation between the top and bottom of the face specially in the first row of holes. Front row holes are often collared near the crest to remove toe but less burden at the top may result in blasting nuisances. Potential advantage of inclined blastholes can be realised only if the drilling is carried out with a high degree of accuracy.

Burden and spacing. It is important that the burden be calculated correctly. An insufficient burden will cause break through of drill hole charges causing flyrock. Too large a burden will cause venting, throw the stemming and also cause cratering effects giving rise to flyrock. Burden to bench height ratios less than 1.5 should be avoided. The burden is generally kept 25-30 times the diameter of the blasthole. The spacing of about 1.5-3 times the burden is advantageous to reduce the flyrock. It needs to be ensured that the burden is adequate at all the points on the face. This is especially important for the burden alongside the top of the front row charges.

Charge distribution in blastholes. Distribution of the charge in blastholes is a very important consideration. The explosive energy should be distributed according to the work to be done. In strata which have two or more hard bands well separated by softer beds, greatest blasting efficiency is usually achieved by deck loading. When the number of decks exceed two or three, charging operations tend to become too complicated, labour-intensive and time consuming. Where the full thickness of rock is uniformly strong, single continuous charges should be used in preference to deck charges. Loading too high in the blasthole is not only wasteful, but may create serious flyrock hazard and, it may also entail excessive noise, air blast, overbreak and cut-off problem.

Stemming. Stemming length is generally kept about 0.7 times to equal the burden distance. The optimum stemming length, however depends very largely on rock properties, and can vary from about 16-48 times the hole diameter. Shorter columns usually cause noise, air blast, flyrock, backbreak and cut-off problems, especially where charges are top primed. Where the rock is tough and massive, the stemming column should be the shortest which prevents excessive noise, air blast, flyrock or backbreak. Where flyrock is a potential problem, the stemming length be kept equal to at least 20 times and preferably 25 times the blasthole diameter.

Initiation sequence and timing. Proper drilling and charging of the first row of holes are important. Flyrock from the first row can be avoided by the correct planning of blasting. If the first row does not give rise to flyrock then there is considerably less risk of flyrock from the drill holes deeper in the round, on the condition that the firing sequence has been made up correctly. An inter-row time interval be such that it allows each row burden to be pushed forward rather than in upward direction. Collaring of the first row is usually more difficult through the edge of the bench being uneven from earlier rounds. Each drill hole must be appraised during collaring so that there is a reasonable burden in the bottom section. Another way is to reduce the flyrock is to leave broken rock in front of the round. The broken rock barrier should have a height of at least two times the burden.

19.4.2 *Site control during blasting*

Poor site control is one of the major contributing factors in surface mine blasting accidents. About 50% accidents of flyrock are due to poor site control, where personnel for one reason or the other come too close to the blast. Site control is essentially a two element process.

– People must be prevented from entering into the shot area during and immediately after the shot. It requires the proper placement of guards and barricades on all accesses to the blast site. It also requires constant and effective communication between the blasters and the persons responsible for the site control.

– Mine personnel and equipment must be moved to a safe distance from the shot and kept there until after the shot is fired. Personnel need to be moved outside the flyrock range according to the site conditions.

19.4.3 *More experienced drilling/blasting crew*

It is to be expected that experienced drilling and blasting crew work more effectively and more safely. If separate drilling and blasting crews are used, the drillers should certainly be taught to recognise and look for potential blasting hazards such as fissures or voids.

19.4.4 *More effective communication*

It is very important that effective communication exists not only between drilling and blasting crews of the same shift and there should be proper communication with the people residing in the danger zone.

19.4.5 *Covering*

When all measures have been carried out to counteract the flyrock in connection with blasting, any residual risk can be eliminated by the use of thoroughly prepared covering. Covering can consist of various types of material that can be used over a round should satisfy some of the following demands: adequate strength, ability to join together, flexibility, relatively massive, pervious to gas, ability to cover large area, possibility to secure in position.

Generally two types of coverings are available. Heavy material and splinter protective material.

Heavy covering material. Heavy covering material is used to control flyrock but also to some extent, prevent the rock from moving forward too much. Heavy covering materials are used as blasting mats made of old rubber tyres, wire mats and iron ring mats. Rubber blasting covering mats usually made of old motor vehicle tyres, provide excellent coverage through their weight and durability. Certain rubber mats have their metallic sections insulated with plastic material- this being a great advantage. Wire mats and iron ring mats are also used in different conditions.

Splinter-protective material. Splinter protective material is primarily intended to prevent flyrock or spread of muck from the surface section of the round. Splinter-protective materials in use today are industrial felt mats, rag mats, jute matting, tarpaulins and DNW blasting mats. They are suitable, however for the blasting work where some form of hanging protection is needed. These types of coverings are good for protection from splinters. These can cover large areas and are fairly durable. Simple wire netting, preferably in several layers provide good protection with respect to small sized stones. Care must be taken to ensure, however, that no joints between lengths of electric detonator wires come into contact with the conductive material of the blasting mats. These joints should be insulated under the covering material. In some cases both the covering materials are used together, the heavy material preventing the most serious flyrock and the splinter-protective material stopping small stones. One such example is the use of sand bags placed on the top of old wire ropes covering the blast area.

Before starting mucking work, care needs to be taken to pullout the covering material since it is otherwise readily worn out. Even relatively sensitive material such as industrial felt, tarpaulin can be recovered for further use after blasting.

19.4.6 *Miscellaneous measures*

In order to avoid flyrock from the surface it is important that the surface be properly cleared of stones and other loose material before the drilling starts. The detonating cord, which is extensively used in surface mine blasting operation is a major source of high frequency pressure pulses and due to it loose stones lying close to it fly for a longer distance compared to blasted rock.

Blasting of toe holes create substantial flyrock. Therefore these must be carefully planned. It is better to blast the toe holes before the commencement of the subsequent blast.

The majority of blasting accidents in surface mines occur due to the flyrock, therefore, proper blasting precautions are needed to improve site control. Danger zone in blasting is equivalent to the area of the zone of maximum flyrock range. It is common practice to define a safe area where no risk of flyrock exists.

The safe distance may be so long that most of blasting work may become impossible. By calculating the risk, however, comparison with other risks and costs can be made and in this way decide a limit where the risks could be accepted.

CHAPTER 20

Controlled blasting

20.1 INTRODUCTION

Conventional blasting causes cracks and fractures in the rock which has been fragmented and also in the remaining rock, whereas the rock mass itself is very often part of structure which must have certain strength, be used as dimensional stone or surface/underground excavations are constructed after blasting operations. The rock strength and stability are of major importance in underground excavations to prevent rock fall that can injure people and damage equipment. Road and railway cuts are excavated through mountains where steep slopes must be maintained without failure for a long time. Open pit mines demand high steep slopes to avoid failures, costly scaling and to decrease the excavation costs for waste rock. Increased size of open pit mining operations have led to higher bench heights, larger diameter blastholes and larger sized blasts which have reduced mining costs but have also resulted in energy concentration in the blast area which can cause severe backbreak problems for the final pit walls (Fig. 20.1). The rock beyond the excavation line in underground often suffers excessive damage during conventional blasting (Fig. 20.2).

Controlled blasting techniques are adopted to reduce damage to the rock and improve the competence of the rock at the perimeter of the excavation by reducing development and growth of uncontrolled cracks (Fig 20.3). In many cases the technique is called cautious blasting, contour blasting or smooth blasting. Many a times controlled blasting techniques have included controlling damage due to vibrations, air blast and flyrock (see Chapters 18 and 19). Sometime controlled blasting refers to controlling fragmentation size distribution during blasting either to produce large sized fragments (see Chapter 16) or to produce greater proportion of fines. However, in this chapter those blasting techniques have been considered which reduce damage to the rock itself or to the remaining rock and reduce overbreak beyond the perimeter.

Controlled blasting techniques have several advantages. Besides retaining load carrying capacity of the rock as it allows desired roof curvature so that the load carrying capacity of the structure is greater. In conventional blasting overbreak tendency of rock is experienced, whereas by controlling the overbreak the smoothness and precision of rock walls are achieved and extra cost of loading the muck and concreting or providing rock reinforcement is reduced (Fig. 20.3). Finally, smooth walls result in reduced frictional resistance to air flow, and improved mine ventilation capacity.

Figure 20.1 Damage to remaining rock after a blast in an open pit mine.

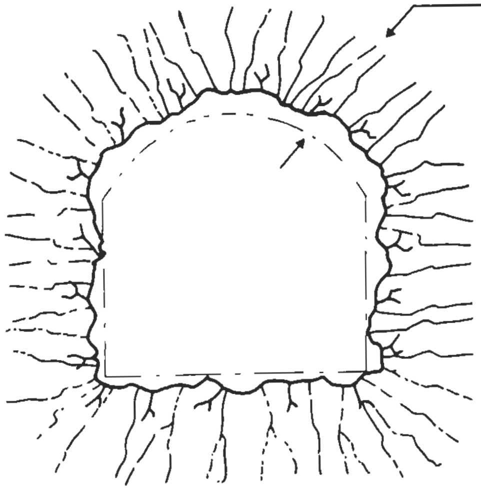

Figure 20.2 Crack zone resulting from conventional blasting.

All types of controlled blasting have one common objective to better distribute the energy delivered in the rock mass by the explosive detonation so as to reduce crushing, fracturing, overbreak of the remaining rock, and to prevent undue disturbance of the jointed mass and therefore to preserve the inherent jointed mass, and strength of the in-situ rock. A number of tools are used to control the blasting dam-

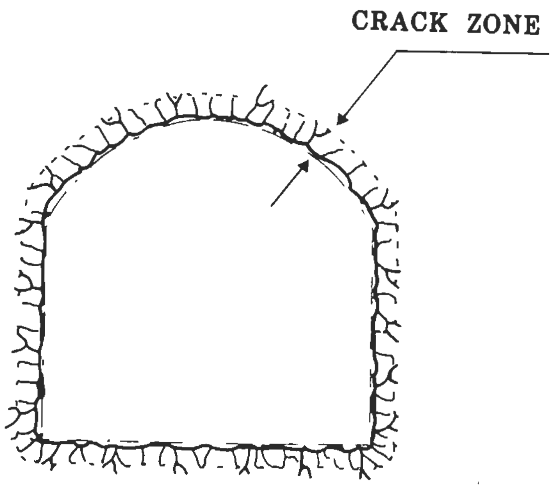

Figure 20.3 Controlled blasting results.

age which include controlling the explosive type, the loading density, blast hole diameter, burden and spacing, subgrade drilling, collar and stemming height. Good blasting design can never be carried on a piecemeal basis; the entire blasting pattern including the cut, the main part and the perimeter holes must be considered together if a satisfactory result is to be achieved.

20.2 ROCK DAMAGE

When an explosive charge detonates in a borehole the expansion of the high pressure gaseous reaction products set the borehole walls in motion outwards creating a dynamic stress field in the surrounding rock. In the region close to the charge, permanent damage occurs at a given critical level of particle velocity. Whether the damage affects the stand-up time of the rock contour or not, depends on the character of the damage, the rock structure, the ground water flow and on the orientation of the damaged plane in relation to the contour and the static load.

In addition, the high pressure of the gaseous products far exceeds the crushing strength of the rock and produces a crushed zone in the adjacent rock and further away produces a dense radial crack pattern around the hole. These radial cracks arrest quickly after producing some randomly oriented cracks which extend for significant distance from the borehole. Also the initial effect of the dynamic stress field in the nearby rock is high-intensity, short duration stress wave, which quickly decays. The continued gas expansion leads to further motion and sets up an expanding stress field in the rock mass. When the free surface is close enough to the borehole the rock breakage occurs. In the other directions the motion spreads further in the form of seismic waves. These seismic waves are complicated combination of elastic waves in which the rock reverberates in the compressive, shear and surface wave modes. Each mode or wave type (*P*-, *S*- and *R*-waves) has a characteristic propagation velocity, c, that is some fraction of the sonic velocity which is a material property of the rock mass. The particles in the rock mass experience an approximately elliptical motion, the highest velocity equal to the peak particle velocity, v, decreasing with the distance from the charge. Damage is a result of the induced strain, ε, which for an elastic medium in the sine-wave approximation, is given by the equation

$$\varepsilon = v/c$$

Since this process dissipates much of the energy in crushing the rock at the borehole and the resulting crack pattern is randomly oriented, very little control of the fracture plane is achieved.

The damage to the rock mass in a production blast with 250 mm diameter holes were studied by observing the diamond drilled core before the blast and after blasting similar core was obtained from new diamond drilled hole made parallel to the old hole. In this way increase in crack frequency could be determined (Persson et al. 1977). It was observed that for the blast described, there was 50% probability of damage at 22.5 m distance from the nearest hole and a 5% probability at 32 m distance. For a rock mass that gives incipient fracture at a vibration particle velocity within the range 700-1000 mm/s the radius of the zone of incipient fracture is about 0.25-0.35 m. Holmberg & Persson (1979) suggested that rock damage was related to the peak particle velocity induced by the blast. This peak particle velocity may be estimated by means of the following empirical equation

$$V = K\,(W^{\alpha})/D^{\beta} \qquad (20.1)$$

where V is the peak particle velocity in mm/s, W is the charge weight in kg, D is the radial distance from the point of detonation in meters, K, α and β are constants which

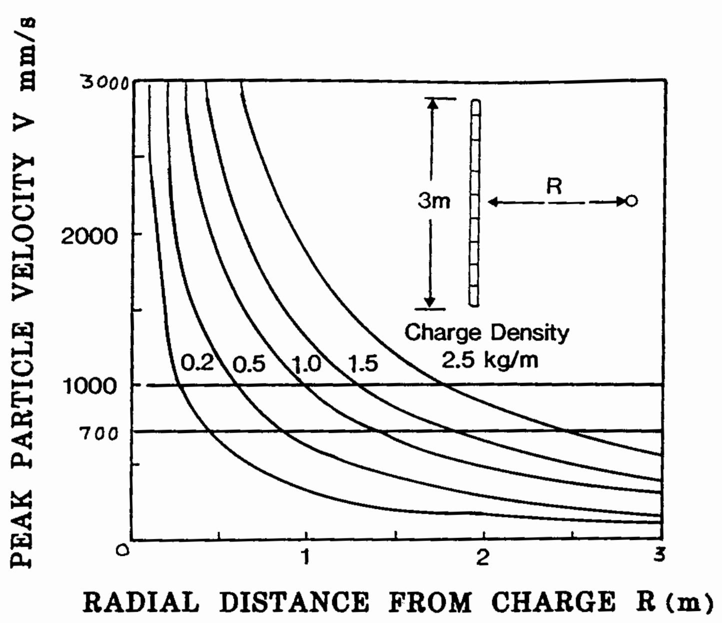

Figure 20.4 Rock damage dependent upon distance, charge density levels and resulting peak particle levels (Holmberg & Persson 1979).

depend upon the structural and elastic properties of the rock mass and which may vary from site to site.

The constants K, α and β used in Equation 20.1 depend upon the type of blast and condition of the rock mass in which the blast is carried out. Holmberg & Persson (1979) suggested values of $K = 700$, $\alpha = 0.7$ and $\beta = 1.5$ for tunnel blasting in competent Swedish rock and they used these values in calculating the results plotted in Figure 20.4. The values of these constants vary by large amounts, depending upon the conditions and assumptions made in interpreting the data. Ideally, these constants should be determined for each site by conducting a series of trial blasts and monitoring the induced particle velocity at different distances from the points of detonation.

The above equation is based on the assumption that the detonation occurs at a single point and hence it is only valid when the distance R is large compared with the length of the charge. When point under consideration is close to long charge, as is the case in tunnel blasting, the peak particle velocity must be obtained by integration over the charge length.

The zone of damage surrounding a 45 mm diameter blasthole charged with ANFO, with a linear charge density of 1.5 kg/m, is approximately 1.5 m in radius. This amount of damage is unacceptable when the blasthole is close to the final tunnel wall and steps must be taken to reduce the damage by reducing the charge. Figure 20.4 shows that the radius of the damaged zone of the rock will be reduced to about 0.3 m when the charge density is reduced to 0.2 kg/m. The use of charge density of this magnitude in the closely spaced holes is the basis of controlled blasting.

It must also be considered that the effects of the charges in the rows adjacent to the often well planned contour row of holes can also cause damage to the remaining rock. Charging the adjacent rows with a heavy charge results in cracks spreading further into the remaining rock than from the controlled blasting row (Fig. 20.5).

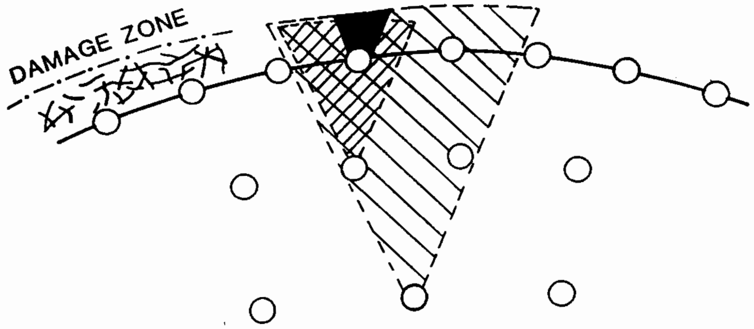

Figure 20.5 Damage resulting from holes other than contour holes.

Rustan et al. (1985), while studying damage zones in using controlled blasting in tunnel observed that the bottom charge gives a larger damage zone than the column charge.

Bauer et al. (1979) showed that lower the borehole pressure, less the amount of damage (backbreak). The rupture radius produced by the ANFO and permissible explosives is 2-4 times less than the radius of rupture from the same volume of dynamite. The borehole pressure produced by ANFO and permissible explosives is several times less than that produced by the dynamite.

Borehole pressure, and hence backbreak, can be reduced by decoupling or decking of charge. Charges are decoupled when they do not touch the borehole wall. The ratio of the charge radius to the hole radius is a measure of the coupling of a charge. When a decoupled charge is surrounded by water, its effective strength upon detonation increases. Field studies indicate that the presence of water around decoupled charges can increase the level of seismic vibrations several times. If blastholes cannot be kept dry, this becomes consideration when designing a blast.

20.3 PARAMETERS IN CONTROLLED BLASTING

The following parameters influence controlled blasting results.

1. Precision and location of drill holes:
- Shape of opening,
- Spacing, burden and their ratio,
- Hole deviation and collaring error.
2. Explosive:
- Velocity of detonation,
- Decoupling ratio,
- Density of explosive,
- Charge concentration per meter,
- Length of borehole,
- Shape of the charge.
3. Interval timing:
- Number of delays in perimeter holes,
- Delay scattering of detonators,

– Misfired holes.
4. Rock characteristics:
– Rock stress,
– Rock strength,
– Rock structure.

Various controlled blasting techniques are based on changing and adopting parameters according to the rock and blast requirement.

20.3.1 *Precision in drilling*

Drilling precision is of vital importance to the result of controlled blasting. There are several sources of faulty drilling which deviate from the theoretical final contour.

Collaring errors occur by the drill not being collared at the point where it should be. Faulty drilling which is usually measured in cm/m, depends partly on the aiming of the drill but also depends on the drill feed pressure, the characteristics of the rock, etc.

The drill holes in the final contour must have 'look-out' as shown in Figure 13.16 outside the theoretical contour in order to provide sufficient space for the drilling equipment when next round is to be drilled. This implies that a drilling rig which requires a large amount of space when collaring drill holes in the final contour also result in large 'look-out' which has negative effect on the results of blasting. Manual rock drills require less space but, on the other hand, the operator must be experienced and skilled.

The location of drill holes is important, as it is related to charge concentration, the technique adopted such as smooth blasting or pre-splitting, charge diameter, drilling costs, rock characteristics etc. Greater charge concentration can be used in larger diameter holes and also their location can be closer. It is better to optimise the charge calculations such that the damage zone from any hole in the round will not exceed the damage zone from the contour holes.

Burden and spacing have no effect on borehole pressure, but they do have influence on backbreak and face loose rock. In order to minimise face loose rock, the spatial dispersion of charges at the perimeter should be as great as possible. This means that using lesser hole spacing and loading density than that used for normal production blasting. The use of a large number of smaller charges decreases the radius of fracture around blastholes and lessens the likelihood that large volumes of explosives gases from a single charge will be channelled into a joint or fracture, causing serious backbreak. With the reduced borehole pressure explosives are used for reducing damage to the remaining rock, the burden and spacing also need to be reduced to maintain the energy factor.

20.3.2 *Explosives*

One of the primary considerations in selecting explosive charge for controlled blasting operations is its borehole pressure. This must not exceed the dynamic compressive strength of the rock. In most cases this is achieved by using explosive having less gas pressure, decoupling the charge or by decking the explosive charge. Low density explosives will produce low borehole pressure. There are several methods of lowering the density of an explosive.

1. Gassing (natural, mechanical or chemical);
2. Addition of material containing entrapped air (e.g. perlite, styrofoam, microballoons, woodmeal, hollow gas beads etc.);

Several types of explosives are in use for controlled blasting practice. Pipe charges (see Fig. 2.9) such as Gurit and Nabit of Sweden, Donorit of Austria, K-pipe of Finland and Kleen-Kut C of USA. For example, Gurit has the following characteristics

Detonation velocity:	approximately 4000 m/s
Volume of gas:	404 l/kg
Weight strength:	50 (compared with blasting gelatine)
Concentration of charge:	0.245 kg/m (17 mm pipe charge)
Concentration of charge:	0.11 kg/m (11 mm pipe charge)

The pipe charges have low volume of gas and have relatively less energy. One factor of great importance is that the explosive is packed in the form of narrow pipe charges which can be connected together by means of extension sleeves fitted with locking rings. The charges are centred in the drill holes, and this reduces the damage. These types of explosives are specially formulated. The conventional explosives when used in small diameter will be subjected to desensitisation due to channel effect (Persson & Johansson 1970).

Explosives in layflat tubing are also available. These layflat tubes contain slurry/emulsion explosives. For 89-110 mm diameter holes, 51 mm diameter cartridges are loaded into layflat tubing and suspended in the blastholes.

It is often convenient to use detonating cord (20, 40 or 80 g/m) as the thin explosive charge which provides small distributed load. Gustafsson (1975) suggested, as a guideline to use the same distance between holes in cm as the charge per meter detonating fuse. If the rock is more difficult to blast, the distance between the holes can be reduced or the charge quantity can be increased.

The costs of cartridged smooth blasting products are considerably higher than ANFO, therefore, many attempts have been made to develop ANFO with reduced strength.

Two methods to reduce the blasting strength of ANFO are to use an unbalanced mix other than 94.5% AN and 5.5% fuel oil or to use a detonating cord down line that initiates the ANFO radially. Both of them result in an incomplete reaction and considerably more noxious fumes. Moreover, the amount of explosives does not decrease.

Another method is to use decoupled ANFO. At the Pyhsalmi mine in Finland 60 mm diameter tubes with ANFO were used in vertical 90 mm diameter holes (Persson & Suakas 1988). This method reduced the explosive cost to roughly 75% of a fully charged hole but was time consuming. In horizontal holes, as in tunnel blasting or drifting, no hose is needed but the problem is to deposit an even string of ANFO that does not fill the hole.

Two methods are used in Sweden. One is called 'spoon-ANFO' because of the spoon or torpedo shaped tool which it trails the orifice of the charging hose and prevents the ejected ANFO from filling the hole completely. Another decoupling method is called 'back blowing'. In the latter a fixed amount is first deposited at the bottom of the contour hole and then dispersed along the hole, blown back, using compressed air only. The resulting charge density is roughly 0.4 kg/m in the 48 mm or a 20-25% degree of filling. Both methods require skilled personnel to achieve the desired decoupling.

ANFO diluted by mixing with inert salt, with expanded, light weight, porous polystyrene spheres (such as Styropore), sawdust or bagasse were suggested by many researchers for reducing damage to rock during blasting (Heltzen 1973; Nielsen & Heltzen 1987; Wilson & Moxon 1988; Persson 1988). In weak rock, such as manganese ore body mixture of ANFO and polystyrene spheres was successfully used in Australia. This product is used in Norway under the name 'Isonol'. The Swedish term used for this mixture of ANFO and styropore mixture is Light ANFO. Coal mining explosives which have salts and are lower in energy have been successfully used in Indian hard rock mines for reducing damage to asbestos fibres and for reducing damage to the remaining rock (Bhandari et al. 1979). A mixture of ANFO, emulsion and styropore is being marketed in a number of countries.

Linear shaped charges have been suggested for controlling growth of fractures in the desired direction (Section 20.4.6).

20.3.3 *Interval timing*

It is extremely important for the ignition pattern to be made up so that the contour holes have free breakage at the time of detonation of holes. Poor timings cause holes to malfunction. As an example, if holes fire out of sequence or without the proper time, fractures will not move in the desired direction instead cracks will move into adjacent holes or rock behind the hole thereby causing overbreak. If the timing is too fast, subsequent holes will sense a much larger than normal burden thereby resulting in cracking back into formation among other effects.

20.3.4 *Rock characteristics*

Rock characteristics contribute significantly to the degree of success achieved in a controlled blast. Homogeneous rock with high compressive and tensile strength characteristics does not crack around a charged hole to as much extent as rock of low strength and which is already fractured. In non-homogeneous jointed rocks crushing and radial cracking around the hole depends on the nature of joints. Tight or infilled joints result in less backbreak then open joints. The orientation of joints has been shown to have dominant effect on the blasting results for the final face and loose rock on the face (Bhandari & Badal 1986). It is shown that when the joints are parallel to the face, a clean smooth face results. When the joints are dipping in favour of the face, backbreak and crest fracture considerably increase. In horizontally bedded deposits there is considerable disturbance below the bench, so that when the lower bench is blasted it results in uneven breakage and crest fracture (Fig. 20.6). In horizontally bedded deposits it is advisable to reduce collar/stemming length to reduce cratering near the crest zone. In general, either the face can be reoriented in favourable direction or delay pattern can be arranged depending on the joint orientation. At the face usually there is no possibility of changing the direction but there is a choice of delay pattern.

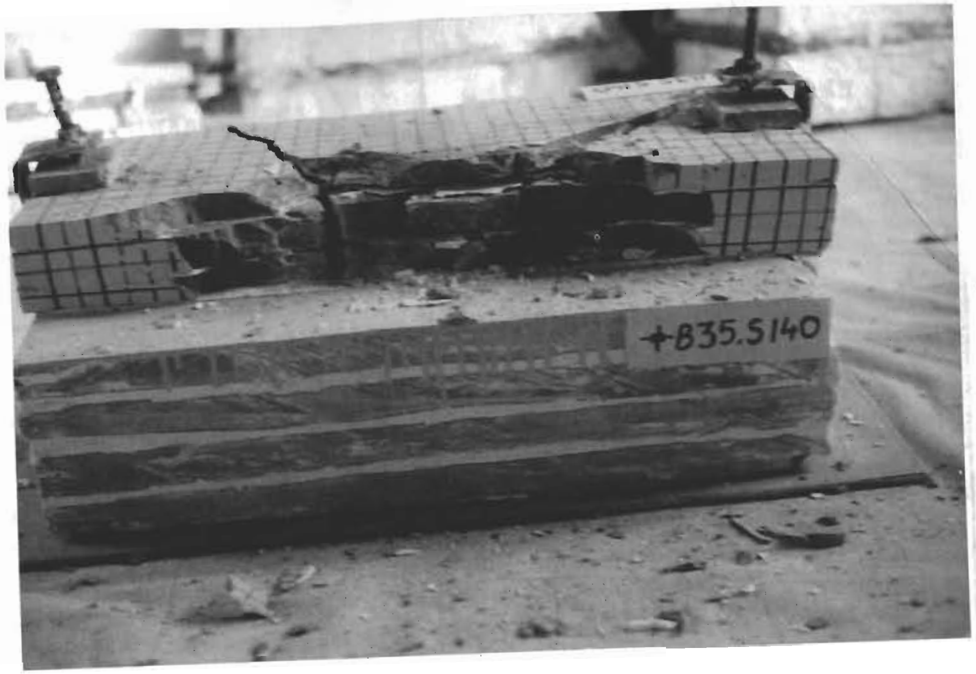

Figure 20.6 Rock damage with horizontal joints.

20.4 CONTROLLED BLASTING TECHNIQUES

There are several blasting techniques used to control blasting results. The technique selected is dependent on the nature of the project or end use of the rock face, geology, drilling equipment available and associated cost. Some common controlled blasting techniques may be classified into the following main categories:

- Line drilling,
- Pre-splitting,
- Smooth blasting,
- Cushion blasting,
- Air decking,
- Controlled fracture growth,
- Tracer blasting.

The techniques most commonly used to control damage in the final walls of excavations are smooth blasting, pre-splitting and air decking. The basic mechanics of failure for these techniques are almost identical.

When two adjacent boreholes are detonated simultaneously, the circumferential tensile stresses induced by the explosion reinforce one another and cause an increase in the tensile stress acting perpendicular to a line drawn between the two holes. This tensile stress, which is higher than that across any other radial line drawn from either borehole, tends to cause preferential crack growth along the line between the two boreholes. By carefully choosing the correct borehole spacing and charge densities a clean fracture can be caused to run from borehole to borehole around the perimeter

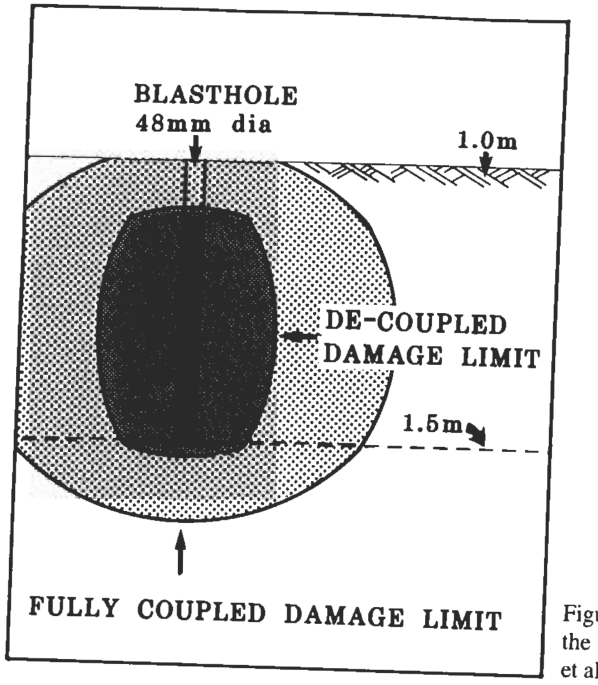

Figure 20.7 Influence of decoupling on the crushing around the hole (McKenzie et al. 1992).

In addition to the need to detonate a line of smooth blast or pre-split holes simultaneously in order to cause the stress reinforcement effect, it is also necessary to reduce the charge in each hole in order to minimise the extent of the pulverised zone and growth of radial cracks. This can be achieved by making the diameter of the charge smaller than the diameter of the borehole so that an annular air space is provided around the charge. This air space absorbs some of the initial explosive energy and reduces the magnitude of the initial high pressure impact which is responsible for the crushing of the rock immediately surrounding the borehole (Fig. 20.7).

Once the crack running between the simultaneously detonated boreholes has been initiated by the interacting stress fields, the gas pressure in the borehole plays an important role in wedging the cracks open and causing them to propagate cleanly between the holes.

When these conditions of simultaneous detonation of closely spaced blastholes with low density charges are satisfied, the amount of damage to the rock surrounding the borehole is reduced. Consequently, not only do the controlled blasting techniques give a cleanly fractured final surface but they also reduce the amount of damage inflicted upon the rock mass behind this final surface.

20.4.1 *Line drilling*

Line drilling involves the drilling of a row of closely spaced holes along the final excavation line (Fig. 20.8). It is not really a blasting technique since the line-drilled holes are not loaded with explosive. The line-drilled holes provide a plane of weakness to which the final row of blastholes can break. Line drilling is used mostly in

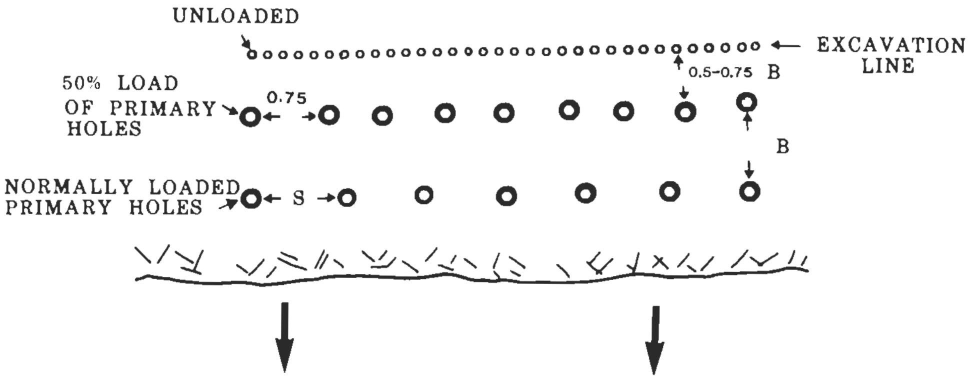

Figure 20.8 Line drilling holes along the final excavation.

small blasting jobs and involves small holes in the range of 50-75 mm diameter. Line drilling holes are spaced (centre to centre) two to four diameters apart. The maximum practical depth to which line drilling can be done is governed by how accurately the alignment of the holes can be held at depth, and is seldom more than 10 m.

To further protect the final perimeter, the blastholes in the production blast adjacent to the line drilling are often more closely spaced and are loaded more lightly than the rest of the blast, using deck charges and detonating cord downlines if necessary. Best results are obtained in a homogeneous rock with little jointing or bedding, or when the holes are aligned with a major joint plane.

The use of line drilling is limited to jobs where even a light load of explosives in the perimeter holes would cause unacceptable damage. The results of line drilling are unpredictable, the cost of drilling is high and the results are heavily dependent on the accuracy of drilling. Holes should not deviate out of the general plane of drilling by more than 150 mm. Drilling inaccuracies usually limit the depth that can be drilled. Line drilling is in general most costly and time consuming compared to the other techniques. Because there are so many variables involved, experimentation is usually necessary in order to determine proper charge weights in the row of primary blastholes nearest the lined drilled holes, and also, determine any necessary adjustments to the burden and spacing. In some cases, small amount of explosives may be loaded in line drilled holes to help fracture the web and trim the final wall. This is not a normal function of the drilling technique but rather a consequence of a change in the hardness of the formation or an increase in line-hole spacing.

20.4.2 *Pre-splitting*

In pre-splitting, sometimes called preshearing, a row of holes is drilled along the final excavation line. These holes are loaded with light charge of explosives and initiated prior to the primary blast. The actual line of pre-splitting holes can be either fired separate from the primary blast or simultaneously with primary blast but with an early delay period (Fig. 20.9). This creates a fracture zone between the holes. The fractured zone between the holes may be a single narrow crack or a thick zone of several cracks. This fracture causes a discontinuity which minimises or eliminates

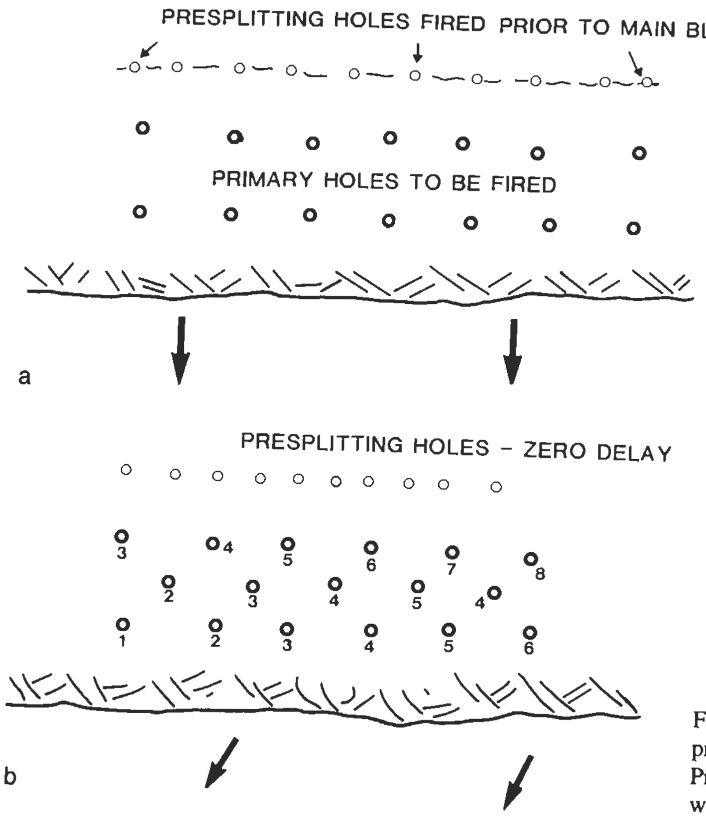

Figure 20.9 a) Pre-splitting prior to primary blast and b) Pre-splitting simultaneously with primary blast.

overbreak from the subsequent main primary blast and produces a smooth finished wall (Calder & Bauer 1983).

Pre-splitting is best carried out when the burden is composed of homogeneous consolidated rock that is resistant to shifting. In badly fractured rock, unloaded guide holes may be drilled between the loaded holes. The light explosive charge can be obtained by using specially designed pipe cartridges, partial or whole cartridges taped to the detonating cord downline, explosive cut from a continuous reel, or detonating cord having 10-20 g/m charge. A heavier bottom charge is used in the bottom part of the hole. Figure 20.10 shows three charge loading methods for pre-splitting.

In pre-splitting technique it is extremely important to find the right hole spacing and charges for the type of rock. The characteristics of the rock influence the result to a greater extent than in most other blasting methods. It is advisable to carry out a sample blasting in a short span (about 5 m) before drilling a long distance (Fig. 20.11). Optimum distance and charge values can be found by blasting with variable inter hole distances and charges.

The geologic structural orientation of the rock mass can imply that the rock splits easily along its direction of cleavage while at right angles to this direction the holes must be considerably more closely spaced with the same charge. This situation commonly occurs in quarries where large blocks of stone are taken out. In spite of homogeneous types of rock, the material behaves in completely different ways when

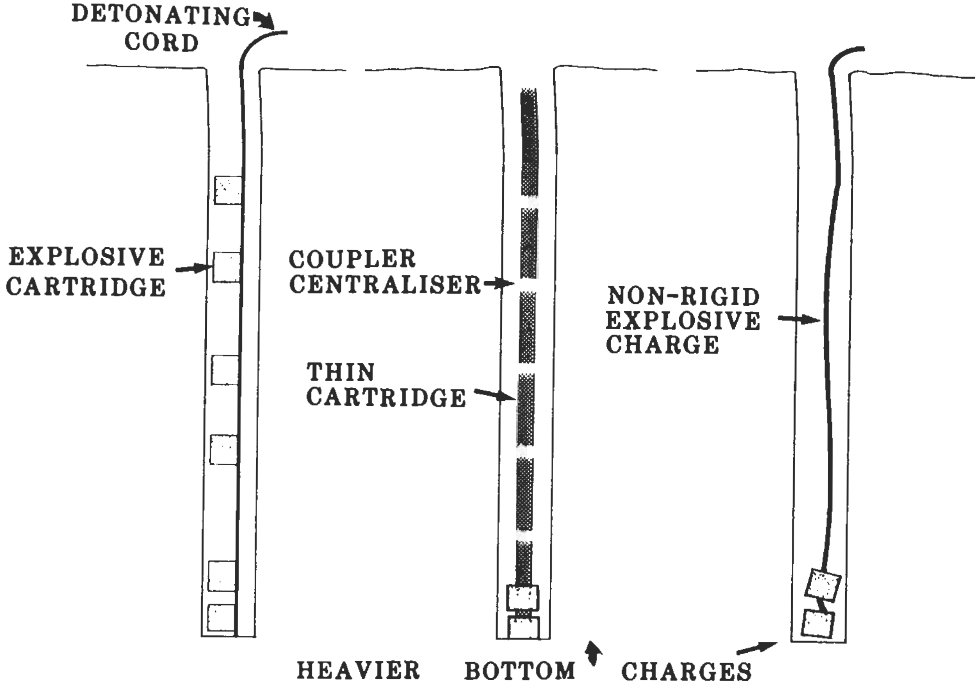

Figure 20.10 Blast hole loading systems for pre-splitting.

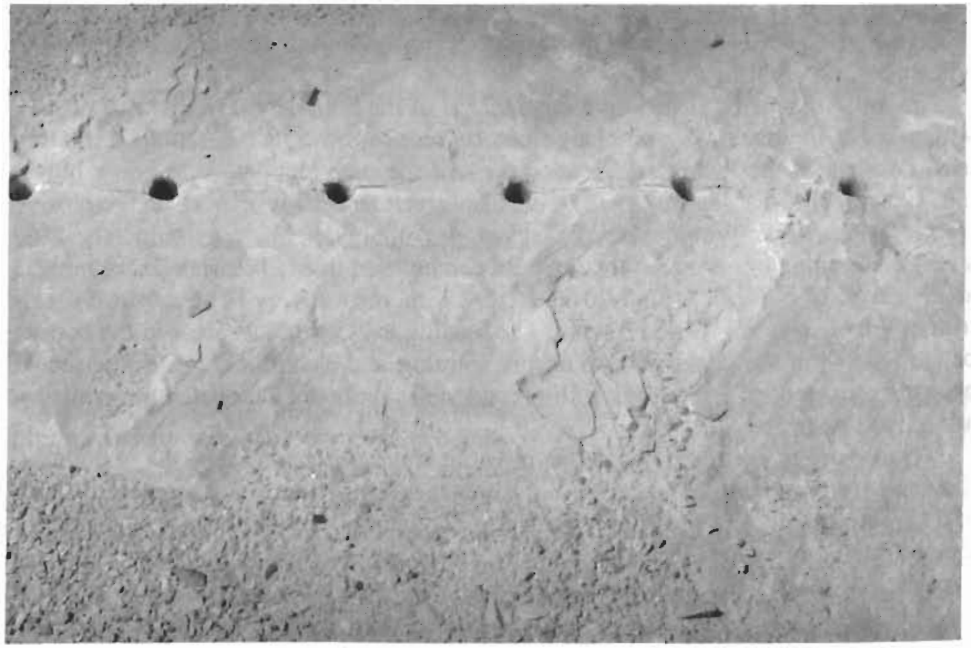

Figure 20.11 Test results for determining spacing and charge for pre-split blasting.

Table 20.1 Pre-splitting parameters (Blaster's Handbook 1977).

Hole diameter (mm)	Loading density (kg/m)	Spacing (m)
38-44	0.12-0.38	0.30-0.45
51-64	0.12-0.38	0.45-0.60
76-89	0.20-0.75	0.45-0.90
102	0.38-1.12	0.60-1.20

Table 20.2 Pre-splitting parameters (Gustafsson 1981).

Hole diameter (mm)	Charge conc. (kg/m)	Charge type	Spacing (m)
25-32	80 g	Detonating cord	0.30-0.60
25-32	0.30	Pipe charge 17 mm	0.35-0.60
40	0.30	Pipe charge 17 mm	0.35-0.50
51	0.60	Pipe charge 17 mm	0.40-0.50
64	0.46	Charge 25 mm	0.60-0.80

it is to be cracked in different directions. At the outer charges in a pre-splitting operations it can be seen that the crack formation deviates from the direction between the drilled holes and instead follows the natural cleavage direction of the rock. If the holes are drilled close enough to each other, a tendency of this type can be counteracted since crack formation is forced to follow the rows of holes.

For pre-splitting, the borehole spacing is normally 8-12 times the borehole diameter and the burden can be considered infinite. The charge diameters and charge concentration given in Table 20.1 and Table 20.2 can used for pre-splitting.

Gustafsson (1981) suggested parameters (Table 20.2) for drilling and blasting when using pre-splitting.

The drill holes are charged up to about 75% of the hole depth. In the case of rock which is highly fractured, the charge can be reduced to 55%. The only stemming used consists of the locking devices to prevent the pipe charges from being blown out. Langefors & Kihlstrom (1978) have also given tables for drilling and explosive charge parameters. When special charges are not available approximately 80% strength cartridge taped on detonating cord can be used in 87, 100 and 200 mm holes at distances of 0.7-1.0, 0.80-1.20 and 1.5-2.1 m respectively. The explosive cartridges are of sizes 25, 29 and 52 mm. The loading is 2-3 times as great in the bottom of the hole as in the upper part, to ensure splitting and clean breakage of the toe. If necessary to control vibrations millisecond delays can be detonated in sequence around the perimeter.

Because of the need to detonate the pre-split charges in advance of the main blast, the use of pre-splitting in underground excavations may involve a separate drilling and charging cycle ahead of the main blast. The inconvenience and delay caused by this additional operation tends to limit the use of pre-splitting in underground excavations.

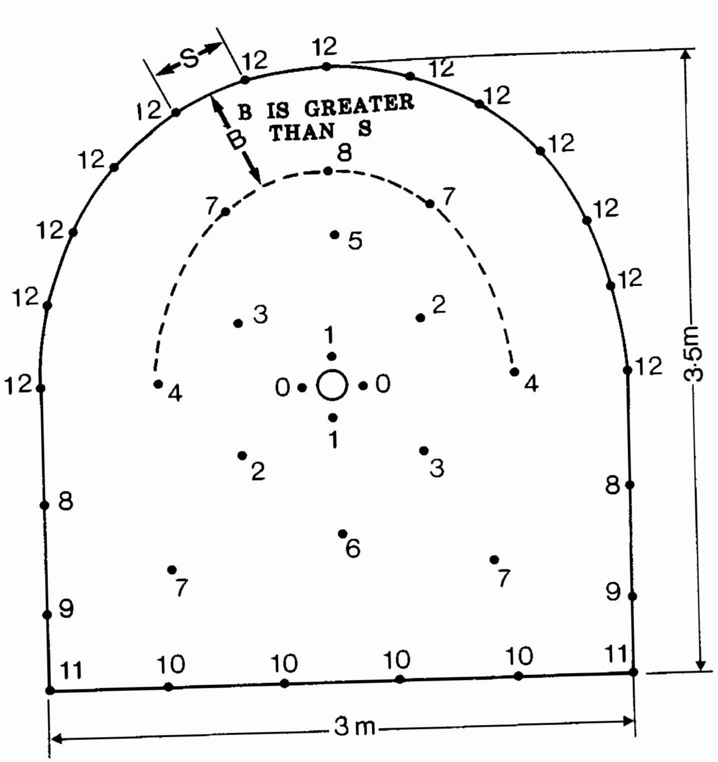

Figure 20.12 Smooth blasting in underground operations.

Table 20.3 Suggested values for smooth blasting (Gustafsson 1981).

Hole diameter (mm)	Charge concentration (kg/m)	Charge type	Burden (m)	Spacing (m)
25-32	0.08	11 mm Gurit	0.30-0.45	0.25-0.35
25-43	0.18	17 mm Gurit	0.70-0.80	0.50-0.60
43-48	0.18	17 mm Gurit	0.80-0.90*	0.60-0.70
51	0.38	22 mm Nabit	1.00	0.80
64	0.52	25 mm Nabit	1.00-1.10	0.80-0.90

* In easily blasted types of rock.

20.4.3 *Smooth blasting*

Smooth blasting also employs holes along the planned excavation limits, but the holes are detonated after the main production blast (Fig. 20.12). The aim is to slash or trim excess material from the walls and to improve their stability. A line of holes is drilled along the planned excavation limits, loaded lightly, and detonated to remove the undesirable material. Smooth blasting, also called contour blasting, perimeter blasting, trim blasting or sculpture blasting is the most commonly used method for controlling overbreak in underground openings. It involves a row of holes at the perimeter of the excavation that is more lightly loaded and more closely spaced (usually 15-16 times the hole diameter) than other holes in the blast round with usually a spacing to burden ratio of 0.8. The charges in the perimeter should be initiated simultaneously for best results in smooth blasting. Holmberg (1982) indicates that the minimum required linear charge concentration for smooth blasting and pre-splitting is a function of the hole diameter

$$w = 90\ d^2$$

where w is the linear charge concentration of ANFO-equivalent explosive in kg per meter hole and d is the hole diameter in meter.

Confinement conditions are different when smooth blasting than when pre-splitting. During pre-splitting, the production round has not yet fired and for all practical purposes, the burden is infinite. In smooth blasting, however, burdens are normally within reasonable distances since the production round has been fired. The blast design should use burden greater than the spacing to ensure that the fractures will properly link between holes rather than prematurely move towards the burden.

20.4.4 *Cushion blasting*

Cushion blasting is similar to smooth blasting and is practised in surface blasting. The term cushion blasting is used in the USA. Like other controlled blasting techniques, it involves a row of closely spaced, lightly loaded holes at the perimeter of the excavation.

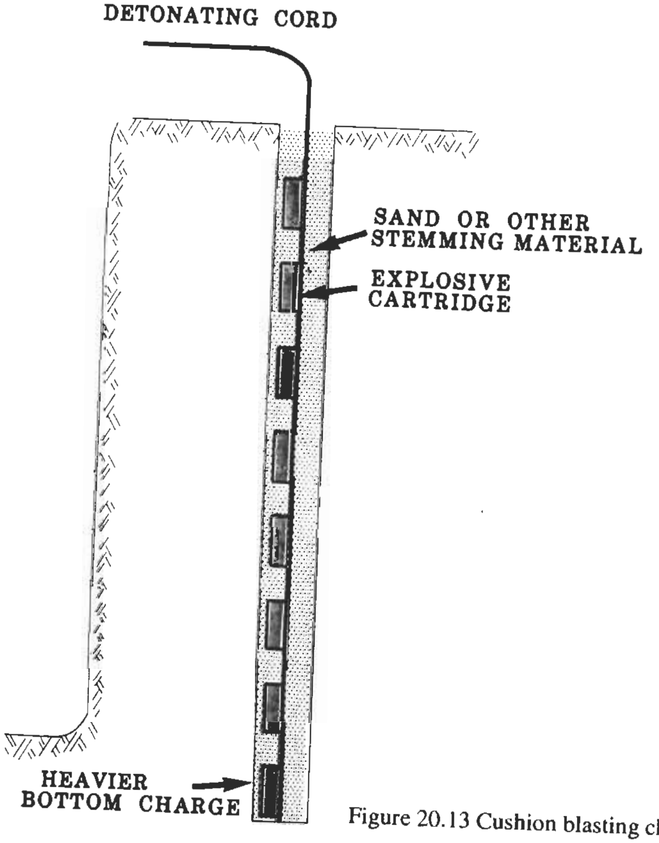

Figure 20.13 Cushion blasting charge distribution.

Table 20.4 Suggested cushion blasting parameters.

Drill hole diameter (mm)	Hole spacing (m)	Stemming (m)	Charge concentration (kg/m)
50-64	0.90	1.20	0.12-0.40
75-88	1.20	1.50	0.20-0.80
100-112	1.50	1.80	0.40-1.20
125-138	1.80	2.10	1.20-1.50
150-165	2.10	2.70	1.50-2.20

Hole sizes are commonly 100-175 mm in diameter, and holes are spaced 1.5-2.5 m apart. After the explosives are loaded, stemming material is usually placed in the void space around the charges (Fig. 20.13). The spacing between holes is normally larger than that used in the pre-splitting. The burden is designed so that it is greater than the spacing, B = 1.2-1.3S. In general subdrilling is not necessary. The holes are fired after the main excavation is removed. A minimum delay between the holes is desirable.

DuPont Blaster's Handbook (1977) gives following approximate values for various parameters (Table 20.4).

20.4.5 *Air deck technique*

The air deck technique involves loading a small charge of explosives (usually at the toe) in a blasthole that is otherwise empty (Bussey & Borg 1988; Urekar & Pankhurst 1988; Mead et al. 1993). This is similar in principle to that of air cushioning practised in Russia (Melinikov 1962; Melinikov & Marchenko 1971). Near the top, the hole is sealed with a plug (inflatable device or chemically activated plug) and then stemmed (Fig. 20.14). When explosive in the hole is fired, the peak borehole pressure is lowered due to expansion of gases in the empty space, the crushing and cracking immediately around the holes is reduced but other effects are sufficient to start the cracking process from the hole but lacks sufficient energy to cause breakage beyond the desired limits. In place of inflatable plugs, jute bags filled with sand are utilised, suspending up to the desired depth by a nylon cord (Sandhu & Pradhan 1991). Conventionally charges are decked by separating portions of the explosive column by wooden or cardboard spacers or by taping charges onto detonating cord.

Air deck has significant advantages. One of the most important is that it can be adopted to production hole diameters thus eliminating the need for different hole diameters or two different drills. Mead et al. (1993) reported that 35% reduction in explosives was achieved in trial blasts in iron ore and copper mines and by reducing the charge by 20-25% in limit blasts and up to 70% in the final row in conjunction with an air deck, equivalent levels of fragmentation were obtained while reducing back-break by typically more than 50%. Air decking can also have other applications in reduction of fines, flyrock control or in dimension stone blasting.

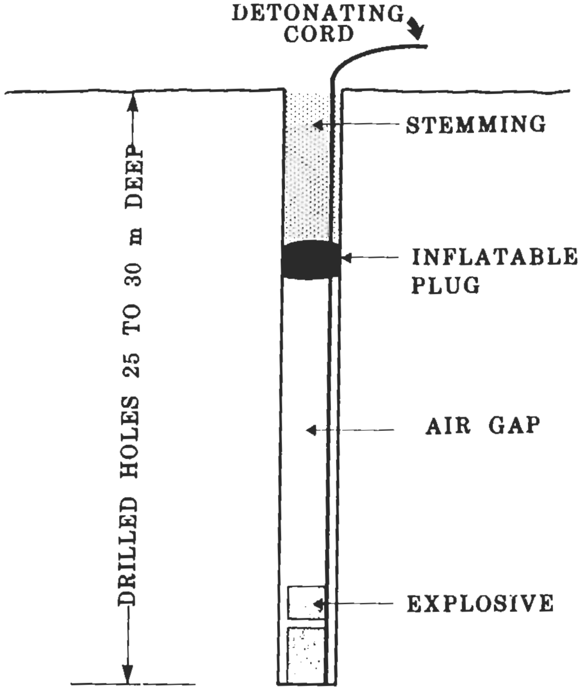

Figure 20.14 Air deck technique.

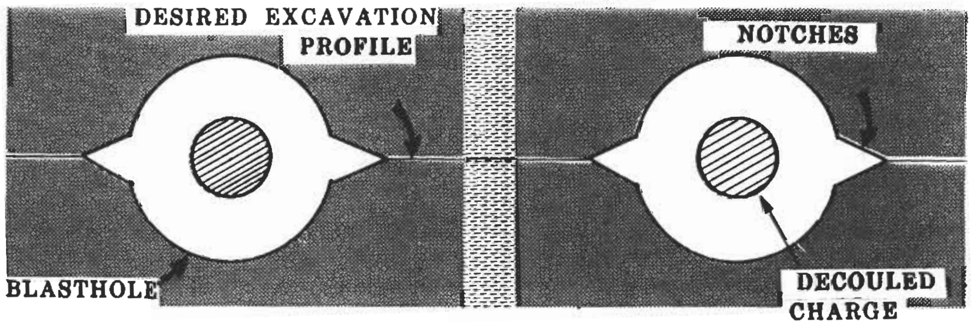

Figure 20.15 Control of excavation line by creating notch in drillhole.

20.4.6 *Controlled fracture growth*

In the situation to avoid damage in the remaining rock and to leave a strong smooth surface after the blasting operation, it would be advantageous to control the growth and development of the fractures resulting from blasting. For example in smooth blasting the ideal result is to have major cracks travelling in the directions between the perimeter holes instead of going backwards into the remaining rock. One possibility is to introduce notches into the surface of the borehole wall since this results in a very high stress concentration at the crack tip and direct the crack toward adjacent hole (Fig. 20.15).

Field & Ladegaard-Pedersen (1971) and Ladegaard-Pedersen et al. (1974) have

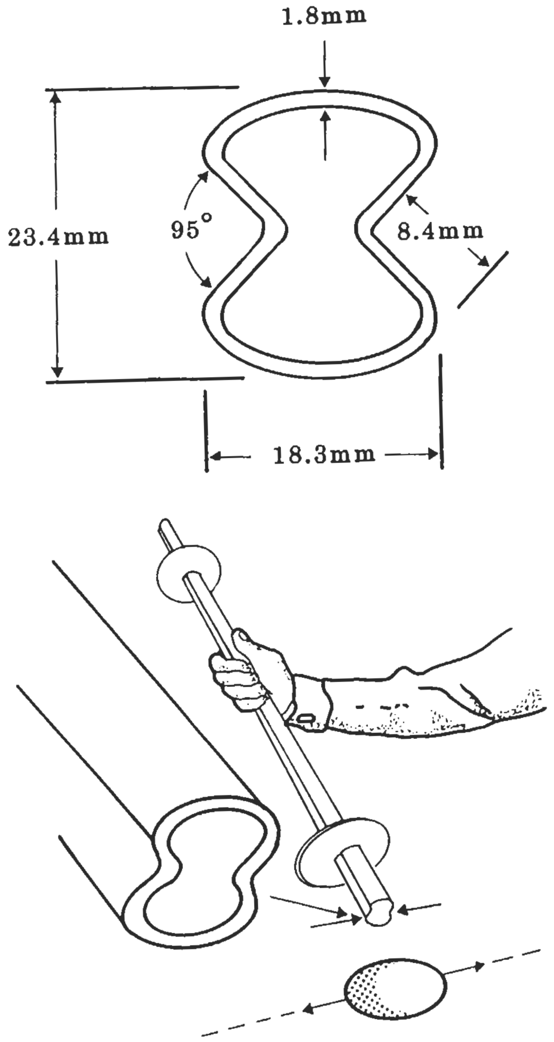

Figure 20.16 Shaped charge for creating cracks in desired direction (Ouchterlony 1992).

shown this effect by model blasting in PMMA. The crack pattern obtained by cutting two slits into a brass tube that was lowered into a bore hole and filled with explosive. The brass tubes protected the holes at all points except where fractures were required. An initial blast reduced into two short cracks at each hole and after recharging it appeared that most of the available energy went into the extension of the two cracks formed by the first shot and no unwanted radial cracks were produced.

Experiments to notch a hole have been carried out by use of a mechanical broaching tool such as suggested by Holloway et al. (1986), with a water jet (Hoshino & Shikata 1980) and with shaped charges (Bjarnholt et al. 1982; Holloway et al. 1986). Tools are available which can create notch after the hole is drilled or while drilling the hole notch is created (Holloway et al. 1986).

Linear shaped charges (Fig. 20.16) have also been used to create desired notch and induce fracture in the desired direction only (Dally & Fourney 1976; Fourney & Dally 1975, 1977). The results are reported to be 10-30% better than conventional blasting methods. However, the average cycle time was longer due to extra time required to notch the perimeter holes.

20.4.7 *Tracer blasting*

Tracer blasting is used in Canadian underground mines for overbreak control. This involves placing a detonating cord along the wall of a blasthole before charging the main column of ANFO. The detonating cord is taped to the detonator which is ideally kept in the centre, near the toe of a blasthole with the help of a spider. The procedure is simple and economic. In a traced blasthole, the initiation of a detonator near the toe is followed by the simultaneous detonation of the detonating cord and the ANFO column in the longitudinal direction. The VOD of detonating cord is much higher than the VOD of ANFO. Depending upon the location of the detonating cord shortly after detonation, it causes side initiation of ANFO column before the detonation front from the detonator arrives there. As a result of this desensitisation of ANFO, partial detonation and reduction in total energy yield occurs. The role played by these phenomena results in minimum blast damage. The technique is mainly used in Canadian development blasting This has now been used in stope blasting operations (Singh 1996).

20.5 PRACTICAL BLASTING RESULTS

The success of these techniques which are used in both underground and surface operations, depend primarily on the geology of the rock formation being blasted. In hard massive rock controlled blasting techniques are usually successful, but in loose unconsolidated formations that will not support themselves, consistently good results may not be possible. When using any of these methods, it is recommended that conservative trials be conducted to determine whether the method can be successfully applied and if so, to establish the optimum hole spacing.

As far as the practical field application is concerned, the geological variables involved in blasting must be taken into consideration. It is unrealistic to believe that the same blasting technique would be equally successful in massive igneous formations and in highly stratified sedimentary deposits.

The additional expense of controlled blasting is caused by the increased cost of drilling of smaller patterns. Methods employing small hole spacing and hole diameters are generally more expensive than if large rotary-drilled holes were used, but they produce a better rock face. Some explosives are more costly though in case of some techniques less explosives are used. The actual cost of controlled blasting is over and above the money which would have been spent in breaking the same volume of ground by using production blasting. Bauer et al. (1979) compared costs of three different controlled blasting methods. Ranked in order of decreasing cost, the methods were line drilling, pre-splitting and cushion blasting.

Terminology used for explosives and blasting

AIR BLAST An airborne shock wave resulting from the detonation of explosives. May be caused by burden movement or the release of expanding gas into the air. Air blast may or may not be audible.

AIR DECK A blasting technique wherein a charge is suspended in a borehole, and the hole tightly stemmed, with a gap between the explosive charge and stemming so as to allow gases to expand before putting pressure on the hole walls.

ALUMINISED See Metallised.

AMMONIUM NITRATE (AN) The most commonly used oxidiser in explosives and blasting agents. Its formula is NH_4NO_3.

ANFO A blasting agent consisting of ammonium nitrate prills and fuel oil.

ARCING Malfunction of an electric blasting cap caused by excessive firing current applied for too long a time.

AXIAL PRIMING A system of priming blasting agents in which a core of priming material extends through most or all of the blasting agent charge length.

BACK BREAK Rock broken beyond the limits of the last row of holes in a blast.

BACK HOLES The top holes near roof in a tunnel or drift round or holes in roof in stoping operation.

BASE CHARGE The main explosive charge in the base of a detonator.

BENCH The horizontal ledge in a quarry or mine face along which holes are drilled vertically. Benching is the process of excavating whereby terraces or ledges are worked in a stepped shape.

BINARY EXPLOSIVE An explosive based on two nonexplosive ingredients, such as nitromethane and ammonium nitrate, which are shipped and stored separately and mixed at the job site to form a high explosive.

BLACK POWDER A low explosive consisting of sodium or potassium nitrate, carbon and sulphur. Black powder is not much used today because of its low energy, poor fume quality, and extreme sensitivity to sparks.

BLAST, BLASTING The operation of breaking and displacing rock or other material, or generating seismic waves by means of explosives. Shot is also used to mean blast.

BLASTABILITY Strength of rock is related to its ease of blasting. A low strength rock is considered easily blastable. However, blasting is influenced by fractures and fissures and cavities thus a high strength rock may be having low blastability.

BLAST AREA The area near a blast within the influence of flying rock missiles, gases or concussion.

BLASTER A qualified person in charge of a blast. Also, a person (blaster-in-charge) who has passed a test, approved by appropriate authority, which certifies his or her qualifications to supervise blasting activities.

BLASTING AGENT Any material or mixture, consisting of fuel and oxidiser, intended for blasting, not otherwise defined as an explosive, provided that the finished product, as mixed for shipment or use, cannot be detonated by means of a number 8 blasting cap when unconfined.

BLASTHOLE A hole drilled into rock or other material for the placement of explosives.

BLASTING CAP A detonator that is initiated by safety fuse. See also detonator.

BLASTING CIRCUIT The electrical circuit used to fire one or more electric blasting caps.
BLASTING CREW A group of persons whose purpose is to load explosive charges.
BLASTING MACHINE Any machine built expressly for the purpose of energising electric blasting caps or other types of initiator.
BLASTING MAT See Mat.
BLASTING PATTERN See Pattern.
BLASTING POWDER A low explosive consisting of potassium or sodium nitrate, charcoal, and sulphur.
BLASTING SWITCH A switch used to connect a power source to a blasting circuit.
BLASTING VIBRATIONS The energy from a blast that manifests itself in earth borne vibrations which are transmitted away from immediate blast area.
BLOCKHOLE A hole drilled into a boulder to allow the placement of a small charge to break the boulder.
BOOSTER An explosive of special character used in small quantity to improve performance of another explosive, constituting the major portion of the charge. A booster does not contain an initiating device but is often cap sensitive. When the booster is armed with detonating device, it becomes primer.
BOOTLEG A situation in which the blast fails to cause total failure of the rock due to insufficient explosives for the amount of burden, or caused by incomplete detonation of the explosives.
BOREHOLE (blasthole) A hole, drilled into rock, to accommodate explosives for blasting.
BOREHOLE PRESSURE The pressure which the hot gases of detonation exert on the borehole wall. Borehole pressure is primarily a function of the density of the explosive and the heat of explosion.
BOTTOM PRIMING A method in which the primer is placed near the bottom of the charge or hole.
BREAKAGE A term used to describe rock fragmentation created by a blast.
BRIDGE WIRE A very fine filament wire imbedded in the ignition element of an electric blasting cap. An electric current passing through the wire causes a sudden heat rise, causing the ignition element to be ignited.
BRIDGING Where the continuity of a column of explosives in a borehole is broken, either by improper placement, as in the case of slurries or poured blasting agents, or where some foreign matter has plugged the hole.
BRISANCE A property of an explosive roughly equivalent to detonation velocity. An explosive with a high detonation velocity has high brisance.
BUBBLE ENERGY The expanding gas energy of an explosive, as measured in an underwater test.
BULK MIX A mass of explosive material prepared for use without packaging.
BULK STRENGTH Refers to the strength of a cartridge of explosive or blasting agents in relation to the same volume of straight nitro-glycerine dynamite.
BURDEN Generally considered the distance from an explosive charge to the nearest free or open face. Technically, there may be an apparent burden, the latter being measured always in the direction in which displacement of broken rock will occur following firing of an explosive charge.
BURN CUT A parallel hole cut employing several closely spaced blastholes. Not all of the holes are loaded with explosive. The cut creates a cylindrical opening by shattering the rock.
CAP See detonator.
CAPPED FUSE A length of safety fuse to which a blasting cap has been attached.
CAPPED PRIMER A package or cartridge of cap-sensitive explosive which is specifically designed to transmit detonation to other explosives and which contains a detonator.
CAP SENSITIVITY The sensitivity of an explosive to initiation, expressed in terms of detonator or a fraction thereof.
CAST BLASTING A technique of blasting in which substantial over burden is fragmented and cast to a distance. Also called explosive casting, or controlled trajectory blasting.
CARTRIDGE A rigid or semi rigid container of explosive or blasting agent of a specified length or diameter.

CARTRIDGE STRENGTH A rating that compares a given volume of explosive with an equivalent volume of straight nitro-glycerine dynamite, expressed as a percentage.
CAST PRIMER A cast unit of explosive, usually pentolite or composition, B, commonly used to initiate detonation in a blasting agent.
CAUTIOUS BLASTING See controlled blasting.
CENTRES The distance measured between two or more adjacent blastholes without reference to hole locations as to row. The term has no association with the blasthole burdens.
CHAMBERING More commonly termed 'springing'. The process of enlarging a portion of a blasthole (usually the bottom) by firing a series of small explosive charges. May also refer to the enlargement of a blasthole by jet piercing or spalling.
CHAPMAN-JOUGET (C-J) PLANE In a detonating explosive column, the plane that defines the rear boundary of the primary reaction zone.
CIRCUIT TESTER See galvanometer; multimeter.
COLLAR The mouth or opening of a borehole or shaft. To collar in drilling means the act of starting a borehole.
COLLAR DISTANCE The distance from the top of the powder column to the collar of the blasthole, usually filled with stemming.
COLUMN CHARGE A long, continuous charge of explosive or blasting agent in a borehole.
COMMERCIAL EXPLOSIVES Explosives designed and used for commercial or industrial, rather than military applications.
CONDENSER-DISCHARGE BLASTING MACHINE A blasting machine which uses batteries to energise a series of condensers, whose stored energy is released into a electric blasting cap circuit.
CONFINED DETONATION VELOCITY The detonation velocity of an explosive or blasting agent under confinement, such as in a borehole.
CONNECTING WIRE A wire used in an electric blasting circuit to extend the length of a leg wire or leading wire.
CONNECTOR Refers to a device used to initiate a delay in a detonator cord circuit, connecting one hole in the circuit with another, or one row of holes to other rows of holes.
CONTROLLED BLASTING Techniques used to control overbreak and produce a competent final excavation wall. See line drilling, pre-splitting, smooth blasting, and cushion blasting.
CONTROLLED TRAJECTORY BLASTING See cast blasting.
COROMANT CUT See parallel hole cut.
COUPLING The degree to which an explosive is in direct contact with borehole walls. Bulk loaded explosives are completely coupled. Untamped cartridges are decoupled. Also, capacitive inductive coupling from powerlines, which may be introduced into an electric blasting circuit.
COYOTE BLASTING The practice of driving tunnels horizontally into a rock face at the foot of the shot. Explosives are loaded into these tunnels. Coyote blasting is used where it is impractical to drill vertically.
CRIMP The circumferential depression at the open end of an ordinary detonator or igniter cord connector which serves to secure the fuse; or the circumferential depression in the detonator shell that secures a sealing plug or sleeve.
CRITICAL DIAMETER For any explosive, the minimum diameter for propagation of a stable detonation. Critical diameter is affected by confinement, temperature, and pressure on the explosive.
CROSSLINKING AGENT The final ingredient added to a watergel or slurry, causing it to change from a liquid to a gel.
CURRENT LEAKAGE Portion of the firing current bypassing part of the blasting circuit through unintended path.
CURRENT LIMITING DEVICE A device used to prevent arcing in electric blasting caps by limiting the amount or duration of current flow. Also used in a blasters' galvanometer or multimeter to assure a safe current output.
CUSHION BLASTING The technique of firing a single row of holes along a neat excavation line to shear the web between the closely drilled holes is fired after the main charge.
CUT An arrangement of holes used in underground mining and tunnel blasting to provide a free

face to which the remainder of the round can break. Also the opening created by the cut holes, or made mechanically such as in coal.

CUT-OFF A portion of a column of explosives that has failed to detonate owing to bridging or a shifting of the rock formation, often due to an improper delay system. Also a cessation of detonation in detonating cord.

CYCLING The ability of a material to change its crystal form with temperature (i.e. ANFO).

DEAD PRESSING Desensitisation of an explosive, caused by pressurisation. Tiny air bubbles, required for sensitivity, are literally squeezed from the mixture.

DECIBEL The unit of sound pressure commonly used to measure air blast from explosives. The decibel scale is logarithmic.

DECK (ING) Portion of a borehole loaded with explosives that is separated from other charges by inert material or air cushion in the same borehole.

DECOUPLING The use of cartridged products significantly smaller in diameter than the borehole. Decoupled charges are normally not used except in cushion blasting, smooth blasting, pre-splitting, and other situations where crushing is undesirable.

DEFLAGRATION An explosive reaction that consists of a burning action at a high rate of speed along which occur gaseous formation and pressure expansion.

DELAY A distinct pause of predetermined time between detonation or initiation impulses, to permit the firing of explosive charges separately.

DELAY BLASTING The use of delay detonators or connectors that cause separate charges to detonate at different times, rather than simultaneously.

DELAY CONNECTOR A non-electric, short-interval delay device for use in delaying blasts that are initiated by detonating cord.

DELAY DETONATOR A detonator, either electric or non-electric, with a built-in element that creates a delay between the input of energy and the explosion of the detonator.

DELAY ELEMENT That portion of a blasting cap which causes a delay between the instant of impressment of energy on the cap and the time of detonation of the base charge of the cap.

DELAY INTERVAL Time by which charges are delayed while detonating sequencing by use of delay detonation, delay connector or blasting machine.

DENSITY The mass per unit volume of a material (i.e. kg/m^{3,} g/cm^3, lb./ft^3). The density of explosive materials are often measured in terms of specific gravity.

DEMOLITION The breaking up of man-made structures (by blasting).

DETALINE SYSTEM A non-electric system for initiating blasting caps in which the energy is transmitted through the circuit by means of a low-energy detonating cord.

DETONATION A supersonic explosive reaction that propagates a shock wave through the explosive accompanied by a chemical reaction that furnishes energy to sustain the shock wave propagation in a stable manner, with gaseous formation and pressure expansion following shortly thereafter.

DETONATING CORD (DC) A flexible cord containing a centre core of high explosives which, when detonated, will have sufficient strength to detonate other cap-sensitive explosives with which it is in contact. Low Energy Detonating Cord (LEDC) containing 1 g/m to 2 g/m are unable to initiate blasting agents.

DETONATION VELOCITY The rate at which the detonation wave travels through a column of explosives.

DETONATION PRESSURE The head-on pressure created by the detonation proceeding down the explosive column. Detonation pressure is a function of the explosive's density and the square of its velocity.

DETONATING WAVE The shock wave set up when explosive is initiated.

DETONATOR Any device containing a detonating charge that is used to initiate an explosive. Includes, but is not limited to, blasting caps, electric blasting caps, and non-electric instantaneous or delay blasting caps.

DOPE Individual, dry, nonexplosive ingredients that comprise a portion of an explosive formulation.

DOWNLINE The line of detonating cord in the borehole which transmits energy from the trunkline down the hole to the primer.

DITCH BLASTING The formation of a ditch by the detonation of a series of explosive charges.

DRILLING PATTERN See pattern.

DROP BALL An iron or steel weight held on a wire rope or chain which is dropped from a height onto large boulder for the purpose of breaking them into smaller fragments.

DYNAMITE The high explosive in which the sensitiser is nitro-glycerine or a similar explosive oil.

DUPLEX WIRE Two separate insulated electric conductors enclosed in a single sleeve.

ECHELON PATTERN A delay pattern that causes the true burden, at the time of detonation, to be at an oblique angle from the original free face.

ELECTRIC BLASTING CAP A blasting detonator designed to be initiated by an electric current.

ELECTRIC STORM An atmospheric disturbance of intense electrical activity presenting a hazard in all blasting activities.

EMULSION An explosive material containing substantial amounts of oxidisers dissolved in water droplets surrounded by an immiscible fuel. Similar to a slurry in some respects.

EXPLODING BRIDGE WIRE (EBW) A wire that explodes upon application of current. It takes the place of the primary explosive in an electric detonator. An exploding bridge wire detonator is an electric detonator that employs an exploding bridge wire rather than a primary explosive. An exploding bridge wire detonator functions instantaneously.

EXPLOSION A thermochemical process in which mixtures of gases, solids, or liquids react with the almost instantaneous formation of gaseous pressures and sudden heat release.

EXPLOSION PRESSURE See borehole pressure.

EXPLOSIVE Any chemical compound, mixture, or device, the primary or common purpose of which is to function by explosion; the term includes dynamite and other high explosives, black powder, pellet powder, initiating explosives, detonators, safety fuses, squibs, detonating cord, igniter cord, and igniters.

EXPLOSIVE CHARGE The quantity of explosive charge used in a blasthole, coyote tunnel, or explosive device.

EXTRA DYNAMITE Also called ammonia dynamite, a dynamite that derives the major portion of its energy from ammonium nitrate.

EXTRANEOUS ELECTRICITY Any electrical energy that could get into an initiating device, other than the small test current from a blasting ohm meter or the firing current in an electric blasting circuit (i.e. stray current, static electricity, lightning, radio frequency energy, or high voltage power lines).

FACE The end of an excavation in a mine or quarry toward which work is progressing or that which was last done. It is any rock surface exposed to air. In quarrying and surface mining face height corresponds to bench height.

FIRE In blasting, it is the act of initiating an explosive reaction.

FIRING CURRENT Electric current purposely introduced into a blasting circuit for the purpose of initiation. Also, the amount of current required to activate an electric blasting cap.

FIRING LINE A line, often permanent, extending from the firing location to the electric blasting cap circuit. Also called lead wire.

FLASH OVER Sympathetic detonation between explosive charges or between charged blastholes.

FLOOR The bottom horizontal, or nearly so, part of an excavation upon which haulage or walking is done.

FLYROCK Rock that is propelled through the air from a blast. Excessive flyrock may be caused by poor blast design or unexpected zones of weakness in the rock.

FRACTURING The breaking of rock with or without movement of the broken pieces.

FRAGMENTATION The extent to which a rock is broken into pieces by blasting. Also the act of breaking rock.

FREE FACE A rock surface exposed to air. A face provides the rock with room to expand upon fragmentation.

FUEL In explosive calculations, it is the chemical compound used for purposes of combining with oxygen to form gaseous products and cause heat.
FUEL OIL The fuel oil, usually No. 2 diesel fuel, in ANFO.
FUMES Noxious or poisonous gases liberated from a blast. May be due to a low fume quality explosive or inefficient detonation.
FUME CHARACTERISTIC/CLASSIFICATION Quantification of the amount of fumes generated by an explosive or blasting agent.
FUSE See safety fuse.
FUSE LIGHTERS Special devices for the purpose of igniting safety fuse.
GALVANOMETER (More properly called blasters' galvanometer) A measuring instrument containing a silver chloride cell and/or a current limiting device which is used to measure resistance in an electric blasting circuit. Only a device specifically identified as a blasting galvanometer or blasting multimeter should be used for this purpose.
GAP SENSITIVITY A measure of the distance across which an explosive can propagate a detonation. The gap may be air or a defined solid material. Gap sensitivity is a measure of the likelihood of sympathetic propagation.
GAS DETONATION SYSTEM A system for initiating caps in which a flame is transmitted through the circuit by means of a gas detonation inside a hollow plastic tube.
GELATINE An explosive or blasting agent that has a gelatinous consistency. The term is usually applied to a gelatine dynamite but may also be a watergel.
GELATINED DYNAMITE A highly water-resistant dynamite with a gelatinous consistency.
GELATINOUS A property of some explosives allowing plasticity of the material.
GENERATOR BLASTING MACHINE A blasting machine operated by vigorously pushing down a rack bar or twisting a handle. Now largely replaced by condenser discharge blasting machines.
GUNPOWDER See Blasting powder.
GRAINS A system of weight measurement in which 7000 grains equal 1 lb. (1 kg = 2.2 lb).
GROUND VIBRATION A shaking of the ground caused by the elastic wave emanating from a blast. Excessive vibrations may cause damage to structures.
HANGFIRE The detonation of an explosive charge at a time after its designed firing time. A source of serious accidents.
HEADING A horizontal excavation driven in an underground mine.
HERCUDET An initiating device using gas detonation system.
HERTZ A term used to express the frequency of ground vibrations and air blast. One hertz is one cycle per second.
HIGH EXPLOSIVE (HE) Any product used in blasting which is sensitive to a No. 6 or 8 test blasting cap and reacts at a speed faster than that of sound in the explosive medium.
HIGHWALL The bench, bluff, or ledge on the edge of a surface excavation. This term is most commonly used in coal strip mining.
HOLE See Borehole.
IGNITE, TO The act of lighting; A safety fuse is ignited by a flame. A plane detonator is ignited by safety fuse.
IGNITER CORD A cord like fuse that burns progressively along its length with an external flame at the zone of burning and is used for lighting a series of safety fuses in sequence. Burns with a spitting flame at a rate of twenty-five to sixty seconds per metre.
INITIATION The act of detonating a high explosive by means of a cap, a mechanical device, or other means. Also the act of detonating the initiator.
INITIATING DEVICE A device such as a detonator which gives a powerful shock or detonation to initiate commercial explosives.
INSTANTANEOUS DETONATOR A detonator that contains no delay element.
INITIATE, TO The act of detonating a high explosive by means of detonator or by detonating cord.
JET LOADER A system for loading ANFO into small blastholes in which the ANFO is drawn from a container by the venturi principle and blown into the hole at high velocity through a semiconductive loading hose.

JOINTS Planes within a rock mass along which there is no resistance to separation and along which there has been no relative movement of the material on either side. Joints occur in sets, the planes of which may be mutually perpendicular. Joints are often called partings.
JUMBO A machine designed to contain two or more mounted drilling units that may or may not be operated independently.
KERF A slot cut in a coal or soft rock face by a hand pick or by a mechanical cutter to provide a free face for blasting.
LEAD WIRE The wire connecting the electrical power source with the leg wires or connecting wires of a blasting circuit. Also called firing line.
LEDC Low energy detonating cord, used to initiate non-electric blasting caps.
LEG WIRES Wires connected to the bridge wire of an electric blasting cap and extending from the waterproof plug. The opposite ends are used to connect the cap into a circuit.
LIFTERS The bottom holes in a tunnel or drift round.
LINE DRILLING A method of overbreak control in which a series of very closely spaced holes are drilled at the perimeter of the excavation. These holes are not loaded with explosive.
LIQUID OXYGEN EXPLOSIVE A high explosive made by soaking cartridges of carbonaceous materials in liquid oxygen. This explosive is rarely used today.
LOADING DENSITY An expression of explosive density in terms of kilogram of explosive per metre of charge of a specific diameter.
LOADING FACTOR Mass of explosive per metre of blasthole.
LOADING POLE A pole made of nonsparking material, used to push explosive cartridges into a borehole and to break and tightly pack the explosive cartridges into the hole.
LOW EXPLOSIVE An explosive in which the speed of reaction is slower than the speed of sound, such as black powder.
LOX See liquid oxygen explosive.
MAGAZINE A building, structure, or container specially constructed for storing explosives, blasting agents, detonators, or other explosive materials.
MAT A covering placed over a shot to hold down flying material; usually made of woven wire cable, rope, or scrap tires.
MAXIMUM FIRING CURRENT The highest current (amperage) recommended for the safe and effective performance of an electric blasting cap.
METALLISED Sensitised or energised with finely divided metal flakes, powders, or granules, usually aluminium.
MICROBALOONS Tiny hollow spheres of glass or plastic which are added to explosive materials to enhance sensitivity by assuring an adequate content of entrapped air.
MILLISECOND The unit of measurement of short delay intervals, equal to 1/1000 of a second.
MILLISECOND DELAY CAPS Delay detonators that have built-in time delays of various lengths. The interval between the delays at the lower end of the series is usually 25 ms. The interval between delays at the upper end of the series may be 100-300 ms.
MINIMUM FIRING CURRENT The lowest current (amperage) that will initiate an electric blasting cap within a specified short interval of time.
MISFIRE A charge, or part of a charge, which for any reason has failed to fire as planned. All misfires are considered dangerous, until the cause of misfire has been determined.
MONOMETHYL AMINE NITRATE An organic compound used to sensitise some watergels.
MS CONNECTOR A device used as a delay in a detonating cord circuit connecting one hole in the circuit with another or one row of holes to other rows of holes.
MUCKPILE A pile of broken rock or dirt that is to be loaded for removal.
MUD CAPPING Referred to also as adobe, or plaster shot. A charge of explosive fired in contact with the surface of rock, usually covered with a quantity of mud, wet earth, or similar substance. No borehole is used.
MULTIMETER (More properly called blasters' multimeter) A multipurpose test instrument used to check line voltages, firing circuits, current leakage, stray currents, and other measurements pertinent to electric blasting. Only a meter specifically designated as a blasters' multimeter or blasters' galvanometer should be used to test electric blasting circuits.

NITROCARBONITRATE (NCN) A classification once given to a blasting agent.

NITROGEN OXIDES Poisonous gases created by detonating explosive materials. Excessive nitrogen oxides may be caused by an excessive amount of oxygen in the explosive mixture (excessive oxidiser), or by inefficient detonation.

NITRO-GLYCERINE (NG) The explosive oil originally used as the sensitiser in dynamites, represented by the formula $C_3H_5(ONO_2)_3$.

NITROMETHANE A liquid compound used as a fuel in two-component (binary) explosives and as rocket fuel.

NITROPROPANE A liquid fuel that can be combined with pulverised ammonium nitrate prills to make a dense blasting mixture.

NITROSTARCH A solid explosive, similar to nitro-glycerine in function, used as the base of 'nonheadache' powders.

NONEL See shock tube system.

NON-ELECTRIC DELAY BLASTING CAP A detonator with a delay element, capable of being initiated non-electrically. See shock tube system; gas detonation system; Detaline System.

OPEN PIT A surface operation for the mining of metallic ores, clay, coal, rock, etc.

OVERBREAK Excessive breakage of rock beyond the desired excavation limit.

OVERBREAK CONTROL A method of firing perimeter holes in such a way as to avoid intensive fracturing of a wall with the aim of preserving a regular outline.

OVERBURDEN The material lying on top of the rock to be blasted; usually refers to dirt and gravel, but can mean another type of rock; e.g. shale over coal.

OVERDRIVE The act of inducing a velocity higher than the steady state velocity in an explosive column by the use of a powerful primer. Overdrive is a temporary phenomenon and the explosive quickly assumes its steady state velocity.

OVERSHOT Condition resulting from more than the necessary amount of explosive or very low burden. Usually characterised by excessive breakage or violence.

OXIDES OF NITROGEN See nitrogen oxides.

OXIDISER An ingredient in an explosive or blasting agent which supplies oxygen to combine with the fuel to form gaseous or solid products of detonation. Ammonium nitrate is the most common oxidiser used in commercial explosives.

OXYGEN BALANCE A state of equilibrium in a mixture of fuels and oxidisers at which the gaseous products of detonation are predominately carbon dioxide, water vapour (steam), and free nitrogen. A mixture containing excess oxygen has a positive oxygen balance. One with excess fuel has a negative oxygen balance.

PARALLEL CIRCUIT A circuit in which two wires, called bus wires, extend from the lead wire. One leg wire from each cap in the circuit is hooked to each of the bus wires.

PARALLEL HOLE CUT A group of parallel holes, some of which are loaded with explosives, used to establish a free face in tunnel or heading blasting. One or more of the unloaded holes may be larger than the blastholes. Examples are Coromant and burn cut.

PARALLEL SERIES CIRCUIT Similar to a parallel circuit, but involving two or more series of electric blasting caps. One end of each series of caps is connected to each of the bus wires. Sometimes called series-in-parallel circuit.

PARTICLE VELOCITY A measure of ground vibration. Describes the velocity at which a particle of ground vibrates when excited by a seismic wave.

PATTERN A plan of holes laid out on a face or bench which are to be drilled for blasting. Burden and spacing dimensions are usually expressed in metres.

PENTA ERYTHRITOL TETRANITRATE (PETN) A military explosive compound used as the core load of detonating cord and the base charge of blasting caps.

PENTOLITE A mixture of PETN and TNT which is cast, used as a cast primer.

PERMISSIBLE BLASTING Blasting according to regulations for underground coal mines or other gassy underground mines.

PERMISSIBLE EXPLOSIVES Explosives that have been approved by appropriate authority for use in underground coal mines or other gassy mines.

PLACARDS Signs placed on vehicles transporting hazardous materials, including explosives, indicating the nature of the cargo.

PLASTER SHOT See mud cap.

PNEUMATIC LOADER One of a variety of machines, powered by compressed air, used to load bulk blasting agents or cartridged watergels.

POWDER Any of various solid explosives.

POWDER FACTOR A ratio between the amount of powder loaded and the amount of rock broken, usually expressed as kilogram per tonne or kilogram per cubic metre. In some cases, the reciprocals of these terms are used but these are correctly referred as specific charge.

PREBLAST SURVEY A documentation of the existing condition of a structure. The survey is used to determine whether subsequent blasting causes damage to the structure.

PREMATURE A charge that detonates before it is intended. Premature can be hazardous.

PRESHEARING See pre-splitting.

PRE-SPLITTING A form of controlled blasting in which charges are fired in closely spaced holes at the perimeter of the excavation. A pre-split blast is fired before the main blast. Also called preshearing.

PRESSURE VESSEL A system for loading ANFO into small-diameter blastholes. The ANFO is contained in a sealed vessel, to which air pressure is applied, forcing the ANFO through a semiconductive hose and into the blasthole. Also known as pressure pot.

PRILL In blasting, a small porous sphere of ammonium nitrate capable of absorbing more than 6% by weight of fuel oil. Blasting prills have a bulk density of 0.80 to 0.85 g/cm^3.

PRIMARY BLAST The main blast executed to sustain production.

PRIMARY EXPLOSIVE An explosive or explosive mixture, sensitive to spark, flame, impact or friction, used in a detonator to initiate the explosion.

PRIMER A unit, package, or cartridge of cap-sensitive explosive used to initiate other explosives or blasting agents and which contains a detonator or other initiator.

PRIMING Use of one or more primer unit in an explosive column.

PROPAGATION The movements of a detonation wave, either in a column of explosive or from hole to hole.

PROPAGATION BLASTING The use of closely spaced, sensitive charges. The shock from the first charge propagates through the ground, setting off the adjacent charge, and so on. Only one detonator is required. Primarily used for ditching in damp ground.

PROPELLANT EXPLOSIVE An explosive that normally deflagrates and is used for propulsion.

PULL The quantity of rock or length of advance excavated by a blast round.

QUARRY An open or surface mine used for the extraction of rock such as limestone, slate, building stone, etc.

RADIO FREQUENCY ENERGY Electrical energy travelling through the air as radio or electromagnetic waves. Under ideal conditions, this energy can fire an electric blasting cap.

RADIO FREQUENCY TRANSMITTER An electric device, such as a stationary or mobile radio transmitting station, which transmits a radio frequency wave.

RDX Cyclotrimethylenetrinitramine, an explosive substance used in the manufacture of compositions B, C-3, and C-4. Composition B is useful as a cast primer.

RELIEVERS In a heading round, holes adjacent to the cut holes, used to expand the opening made by the cut holes.

RIB HOLES The holes at the sides of a tunnel or drift round, which determine the width of the opening.

RING DRILLING A method of drilling a fan-shaped vertically in a single plane from one drill rig position.

RIP RAP Coarse rocks used for river bank or dam stabilisation to reduce erosion by water flow.

ROTATIONAL FIRING A delay blasting system in which each charge successively replaces its burden into a void created by an explosive detonated on an earlier delay period.

ROUND A group or set of blastholes required to produce advance in underground headings or tunnels.

SAFETY FUSE A cord containing a core of black powder. Used to initiate blasting caps or black powder.

SCALED DISTANCE A ratio used to predict ground vibrations. As commonly used in blasting, scaled distance equals the distance from the blast to the point of concern, divided by the square or cube root of the charge weight of explosive per delay. Normally, when using the equation, the delay period must be at least 9 ms.

SEAM A stratum or bed of mineral. Also a stratification plane in a sedimentary rock deposit. The seam may also be of sand or mud and may run vertically or horizontally.

SECONDARY BLASTING Using explosives to break up larger rock masses resulting from the primary blasts. These are the rocks that are too large for easy handling.

SEISMOGRAPH An instrument that measures and supplies a permanent record of earthborne vibrations induced by earthquakes, and/or blasting (in blasting it is called a blast monitor).

SEMI CONDUCTIVE HOSE A hose, used for pneumatic loading of ANFO, which has a minimum electrical resistance of 3000 ohms/m and 10,000 ohms total resistance and a maximum total resistance of 2,000,000 ohms.

SENSITIVENESS A measure of the propagating ability of an explosives; some explosives have a tendency to propagate between boreholes at considerable distances.

SENSITIVITY A measure of how easily an explosive can be initiated by external initiation such as impact, shock flame or friction.

SENSITISER An ingredient used in explosive compounds to promote greater ease in initiation or propagation of the detonation reaction.

SEQUENTIAL BLASTING MACHINE A series of condenser discharge blasting machines in a single unit which can be activated at various accurately timed intervals following the application of electrical current.

SERIES CIRCUIT A circuit of electric blasting caps in which each leg wire of a cap is connected to a leg wire from the adjacent caps so that the electrical current follows a single path through the entire circuit.

SERIES-IN-PARALLEL CIRCUIT See parallel series circuit.

SHELF LIFE The length of time for which an explosive can be stored without losing its efficient performance characteristics.

SHOCK ENERGY The shattering force of an explosive caused by the detonation wave.

SHOCK TUBE SYSTEM A system for initiating caps in which the energy is transmitted to the cap by means of a shock wave inside a hollow plastic tube.

SHOCK WAVE A pressure pulse that propagates at supersonic velocity.

SHOT See blast.

SHOT FIRER Also referred to as the shooter. The person who actually fires a blast. A powderman, on the other hand, may charge or load blastholes with explosives but may not fire the blast.

SHOTHOLE See borehole.

SHUNT A piece of metal or metal foil which short circuits the ends of cap leg wires to prevent stray currents from causing accidental detonation of the cap.

SLOPE Used to define the ratio of the vertical rise or height to horizontal distances in describing the angle of a bench or bench face makes with the horizontal. For example, a 1½ to one slope means there would be a 1½ m rise to each metre of horizontal distance.

SITE MIXED SLURRY (SMS) Slurry mixed at the site immediately before delivery into the borehole.

SLURRY An aqueous solution of ammonium nitrate, sensitised with a fuel, thickened, and crosslinked to provide a gelatinous consistency. Sometimes called a watergel. May be a high explosive or a blasting agent depending on the sensitising material used and can be premixed at the plant or mixed at the site immediately before delivery into the borehole. See watergel, emulsion.

SMOOTH BLASTING A method of controlled blasting, used underground, in which a series of closely spaced holes are drilled at the perimeter, loaded with decoupled charges, and fired on the highest delay period of the blast round.

SNAKE HOLE A borehole drilled slightly downward from horizontal into the floor of a quarry face. Also, a hole drilled under a boulder.

SODIUM NITRATE An oxidiser used in dynamites and sometimes in blasting agents.

SPACING The distance between boreholes or charges in a row, measured perpendicular to the burden and parallel to the free face of expected rock movement.

SPECIFIC CHARGE The amount of explosive of certain strength required to blast a given volume of rock. With same hole size and pattern in a given rock, a higher specific charge will give smaller fragmentation.

SPECIFIC DRILLING The number of hole metres used per unit volume of blasted rock.

SPECIFIC GRAVITY The ratio of the weight of a given volume of any substance to the weight of an equal volume of water.

SQUARE PATTERN A pattern of blastholes in which the holes in succeeding rows are drilled directly behind the holes in the front row. In a truly square pattern the burden and spacing are equal.

SQUIB A firing device that burns with a flash. Used to ignite black powder or pellet powder.

STABILITY The ability of an explosive material to maintain its physical and chemical properties over a period of time in storage.

STAGGERED PATTERN A pattern of blastholes in which holes in each row are drilled between the holes in the preceding row.

STATIC ELECTRICITY Electrical energy stored on a person or object in a manner similar to that of a capacitor. Static electricity may be discharged into electrical initiators, thereby detonating them.

STEADY STATE VELOCITY The characteristic velocity at which a specific explosive, under specific conditions, in a given charge diameter, will detonate.

STEMMING The inert material, such as drill cuttings, used in the collar portion (or elsewhere) of a blasthole to confine the gaseous products of detonation. Also, the length of blasthole left uncharged.

STOPE The chamber or excavation from which ore is extracted underground.

STRAY CURRENT Current flowing outside its normal conductor. A result of defective insulation, it may come from electrical equipment, electrified fences, electric railways, or similar items. Flow is facilitated by conductive paths such as pipelines and wet ground or other wet materials. Galvanic action of two dissimilar metals, in contact or connected by a conductor, may cause stray current.

STRENGTH A property of an explosive described in various terms such as cartridge or weight strength, seismic strength, shock or bubble energy, crater strength, ballistic mortar strength, etc. Not a well-defined property. Used to express an explosive's capacity to do work.

STRING LOADING The procedure of loading cartridges end to end in a borehole without deforming them. Used mainly in controlled blasting and permissible blasting.

SUBDRILL To drill blastholes beyond the planned grade lines or below floor level to insure breakage to the planned grade or floor level.

SUBSONIC Slower than the speed of sound.

SUPERSONIC Faster than the speed of sound.

SWELL FACTOR The ratio of the volume of a material in its solid state to that when broken. May also be expressed as the reciprocal of this number.

SYMPATHETIC PROPAGATION (SYMPATHETIC DETONATION) Detonation of an explosive material by means of an impulse from another detonation through air, earth, or water.

TAMPING The process of compressing the stemming or explosive in a blasthole. Sometimes used synonymously with stemming.

TAMPING BAG A cylindrical bag containing stemming material, used to confine explosive charges in boreholes.

TAMPING POLE See loading pole.

TEST BLASTING CAP No. 8 STRENGTH A detonator containing 0.40-0.45 g of PETN base charge at a specific gravity of 1.4 g/cm^3, and primed with standard weights of primer, depending on the manufacturer.

TOE The burden or distance between the bottom of a borehole and the vertical free face of a bench in an excavation. Also the rock left unbroken at the foot of a quarry blast.

TOE HOLES In situations with bench heights more than 12-15 metres, horizontal holes, drilled in the bottom of the bench in order to avoid having to underdrill the vertical holes.

TRIM BLASTING See smooth blasting.

TOP PRIMING A method in which the primer is placed near the top or at the collar end of the charge.

TRANSIENT VELOCITY A velocity, different from the steady state velocity, which a primer imparts to a column of powder. The powder column quickly attains steady state velocity.

TRINITROTOLUENE (TNT) A military explosive compound used industrially as a sensitiser for slurries and as an ingredient in pentolite and composition B. Once used as a free-running palletised powder.

TRUNKLINE A detonating cord line used to connect the downlines or other detonating cord lines in a blast pattern. Usually runs along each row of blastholes.

TUNNEL A horizontal underground passage.

TWO-COMPONENT EXPLOSIVE See binary explosive.

UNCONFINED DETONATION VELOCITY The detonation velocity of an explosive material not confined by a borehole or other confining medium.

V-CUT A cut employing several pairs of angled holes, used to create free faces for the rest of the blast round.

VELOCITY OF DETONATION (VOD) The rate at which the detonation wave travels through an explosive. May be measured confined or unconfined. Manufacturer's data are sometimes measured with explosives confined in a steel pipe.

VENTURI LOADER See jet loader.

VOLUME STRENGTH See cartridge strength or bulk strength.

WATERGEL An aqueous solution of ammonium nitrate, sensitised with a fuel, thickened, and crosslinked to provide a gelatinous consistency. Also called a slurry. May be an explosive or blasting agent.

WATER RESISTANCE A qualitative measure of the ability of an explosive or blasting agent to withstand exposure to water without becoming deteriorated or desensitised.

WATER STEMMING BAGS Plastic bags containing a self-sealing device, which are filled with water.

WEIGHT STRENGTH A rating that compares the strength of a given weight of explosive with an equivalent weight of straight nitro-glycerine dynamite, or other explosive standard, expressed as a percentage.

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