TUNNEL BLASTING – EMULSION EXPLOSIVES AND PROPER BLAST DESIGN ARE THE PREREQUISITE FOR BETTER EFFICIENCY

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ABSTRACT

Construction of Subway systems for underground highways, Railways & mass transport, hydroelectric installations, water diversion system etc are some of the most important Infrastructure projects for any country. The blasting engineers ultimate challenge is to provide desired fragmentation with sufficient advance / progress and at the same time keeping ground vibration level within limit as per prevailing law.

In this article the author tried to discuss the best possible explosives available for undertaking such a important job (i.e., Emulsion Explosives and its superiority over other form of explosives available) and various blast designs, sequence of excavation, types of strata encountered and steps required for its control, sequence of delays to be used in blasting etc.

Regarding controlled blasting for carrying out ideal blast in tunnel, i.e., to blast in such a way that minimum of damage to the rock that remains and a minimum of over break is made. This has many advantages like, less rock damage to achieve greater stability and requirement of less ground support, safer tunneling operation as less scaling is required, less over break makes a smoother hydraulic surface for an unlined tunnel and less concrete required to fill excess void created by over break.

At the same time, the author discussed the theory of blast induced ground vibration during tunneling and to control such adverse effect.

INTRODUCTION: For centuries mankind has excavated caverns and tunnels in the earth for a myriad of uses. In India excavations at Ajanta and Ellora are the best-known examples. In modern days, underground hard rock excavation and Tunneling is highly specialized type of work requiring very special type of explosives and drilling & blasting techniques. The size and scope of some of the projects can be tremendous and important for nation building such as construction of Subway systems for underground highways, Railways & mass transport, hydroelectric installations, water diversion system etc. As these projects are very special in nature, careful consideration is required in selection of explosives, selection of drills and their dia., designing of each round of blast to meet the specific site conditions, project constraints and economy.

The blasting engineers ultimate challenge is to provide desired fragmentation with sufficient advance / progress and at the same time keeping ground vibration level within limit as per prevailing law. The controllable factors are explosive type, charge weight & geometry, distribution within a given rock mass. The uncontrollable factors are geologic condition of rock mass, environment condition and existing law of the land. The blasting engineers' main challenge is to encounter those uncontrollable factors effectively by varying and proper utilizing controllable factors with desired economy.

The development of tunneling techniques in recent years have been impressive and with the **introduction of high power emulsion explosives** (which have been replaced Nitroglycerine based explosives effectively) and high capacity drilling equipments the rate of advance/ progress have been improved drastically and efficiency in tunneling have been improved both in terms of time and money. It is important that blast holes are **drilled at the right locations and with right inclination**. The marking of holes to be drilled on the face as well as collaring and drilling must be done with great accuracy. Drilling errors has a significant effect upon the overall results of the underground tunneling operations. In this paper we will discuss about explosives and blasting techniques for effective tunneling operation.

The prime objective in Tunnel blasting is to obtain maximum advance/pull per round of blast and to keep cost within reasonable limit. Therefore, very cautiously the type of explosives, drilling pattern (Wedge Cut or Parallel Holes with Dia. of empty holes), spacing & burden, number of holes to be drilled per round, Delay sequence etc., are to be selected. Cycle time is to be kept minimum as far as possible.
Cycle of operation include Drilling, Charging, Blasting, Ventilation, Scaling, Support work, Grouting, Loading and Transport, and Setting out for the next blast. The factors on which a great deal of tunneling operations depends are:

1. Type of explosives used for tunneling blasting operations.
2. Blast design and selection of dia. & location of holes in compatible with the geology of strata, designed area of opening, Environment, existing laws etc.

TYPE OF EXPLOSIVES (*EMULSION EXPLOSIVES)*: Nitroglycerine based explosives have been the workhorse of hard rock blasting for more than 100 years. ANFO and Slurries, though easier to store, handle and use, fall far behind in performance as compared to Nitroglycerine based explosives. Years of research have led to development of emulsion explosives and now for tunneling operation, all over the world, Emulsion explosives have taken the lead in regard to its use.

Brief description of Emulsion explosives: Emulsion explosives are the intimate and homogenous mixer of oxidiser and fuels. It consists of micro droplets of Super – Saturated oxidiser solution in oil matrix. These are in the form of water – in – oil emulsion. The internal phase is composed of solution of oxidiser salts e.g. Ammonium Nitrate etc. dispersed as microscopically fine droplets, which are surrounded by a continuous fuel phase. The emulsion, thus formed, is stabilized against liquid separation by an emulsifying agent.

A bulking/gassing agent – for density control, is then dispersed thorough out the basic emulsion matrix. The gassing agent can either be ultra fine air bubbles or artificial bubbles from glass, resin or plastic. The bulking agents determine and control the sensitivity of emulsion products – whether emulsion is cap sensitive or booster sensitive.

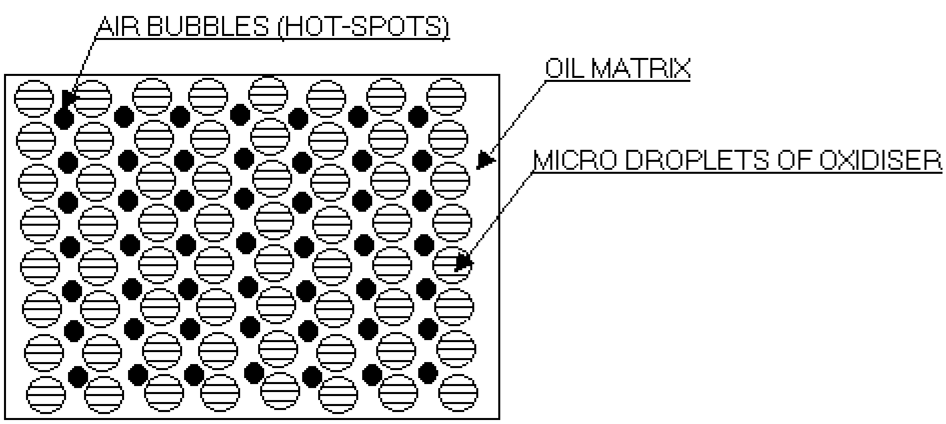
Since its micro cell is coated with an oily exterior, the emulsion has excellent water resistance property.

The output energy of emulsion is very near to the calculated energy.

Viscosity of emulsion can be varied by changing type of fuel oil used. Thus, emulsion explosive can be made from putty type consistency to pump-able type consistency. Therefore emulsion explosives can be made in cartridge form of various sizes or can be used as bulk explosive in opencast as well as in underground workings.

The ultra fine air bubbles or artificial **air bubbles used as gassing agent** acts as sensitiser. When initiating shock wave applies to emulsion explosives the ultra fine air bubbles gets heated first and act as **'Hot-spots'** in emulsion having very high temperature (about 1500 -1800 °C). At this high temperature the explosive reaction takes place (Fig-1).

The resulting emulsion can serve as detonable matrices to carry solid fuels such as Aluminium powder, prilled AN etc.



EMULSION EXPLOSIVES

Fig- 1

<u>Properties of Emulsion Explosives:</u>

1. Emulsion explosives are much **better water resistant** than water gel slurry or ANFO; as oil phase is at outside i.e., water phase is enveloped within oil phase (Water – in – Oil emulsion).

2. Emulsion explosives are much **safer to handle, use and store** as it is relatively insensitivity to detonation by friction, impact or fire. Therefore, it is much safer than NG based explosives and enhances safety standard of the workings.

3. High VOD can be obtained. VOD depend upon the oxidixer droplet size (0.2 to 10 micron). Therefore, **toughest rock conditions can be tackled** very effectively and efficiently without compromising safety standard.

4. Critical diameter of emulsion explosives again depends upon droplet size and sensitizer used. Now a days, emulsion explosives having 25mm dia cartridge are also being used for underground / tunneling blasting operations very effectively.

5. Because of the intimate mixture between oxidizer and fuel, emulsion explosives have **higher energy than water gel slurries or ANFO** and it matches with energy level of Nitroglycerine based explosives. Thus, emulsion explosives have suitably replaced hazardous Nitroglycerine based explosives all over the world without compromising performance.

6. Since Emulsion explosives are **well oxygen-balanced**, generates a **minimum of noxious fumes and far less smoke**. Post blast fume characteristics is much more favourable than NG based explosives. Hence, use of Emulsion explosives in tunneling operations have **shortened the ventilation time**.

PARAMETER DESIGN OF BLASTS: The parameter of blast design for tunneling mainly depend on the factors like **Geology of rock condition** of the area and the **Technique of the blasting** adopted which again depend on Area of cross section of tunnel, Drilling & type of cuts, Delay sequence etc.

Geology, Rock condition & Site Characterization: Tunnel is one of the most hazardous projects in Engineering and construction. It is also one of the most expansive. Therefore, extensive planning, surveying, testing of ground samples etc., are done in pre-excavation stage of the project. There exist a number of uncertainties and unknowns about the site/ area and these become the major challenges to the designers of tunnel. A list of some of the most concern areas is:

a) Tackling uncertainties when dealing with any underground project
b) Geology of the area determines the feasibility and the cost of the project

c) Engineering properties of rock may change drastically with wide range of conditions, rate and direction of loading etc.
d) Ground water is the most difficult parameter to predict and most troublesome during construction
e) The most common method to determine underground rock condition is by core drilling, which only recover about 0.0005 % of the excavated volume of the tunnel with most exhaustive survey works, which leaves a great deal of room for uncertainties.

Probably, the first activity in actual design phase is characterization of the site of the excavation and rock mass into which tunnel is to be driven. This include a) Topography of the area & the climate; b) Location of underground structure with respect to the ground surface and rock formation boundaries; c) Structural stability of rock body, presence of faults and stress concentrations; d) Hydrology of the area i.e., permeability of the ground and groundwater flow rate; e) Rock type, mass, their genesis and their homogeneity; f) Degree of weathering & weatherability of rock; g) Geologic discontinuities and other defects; h) Deformability characteristic under short and long term loading; i) In-situ stress and hydraulic / dynamic loads; j) Geometric and mechanical properties of systematic and extensive discontinuities.

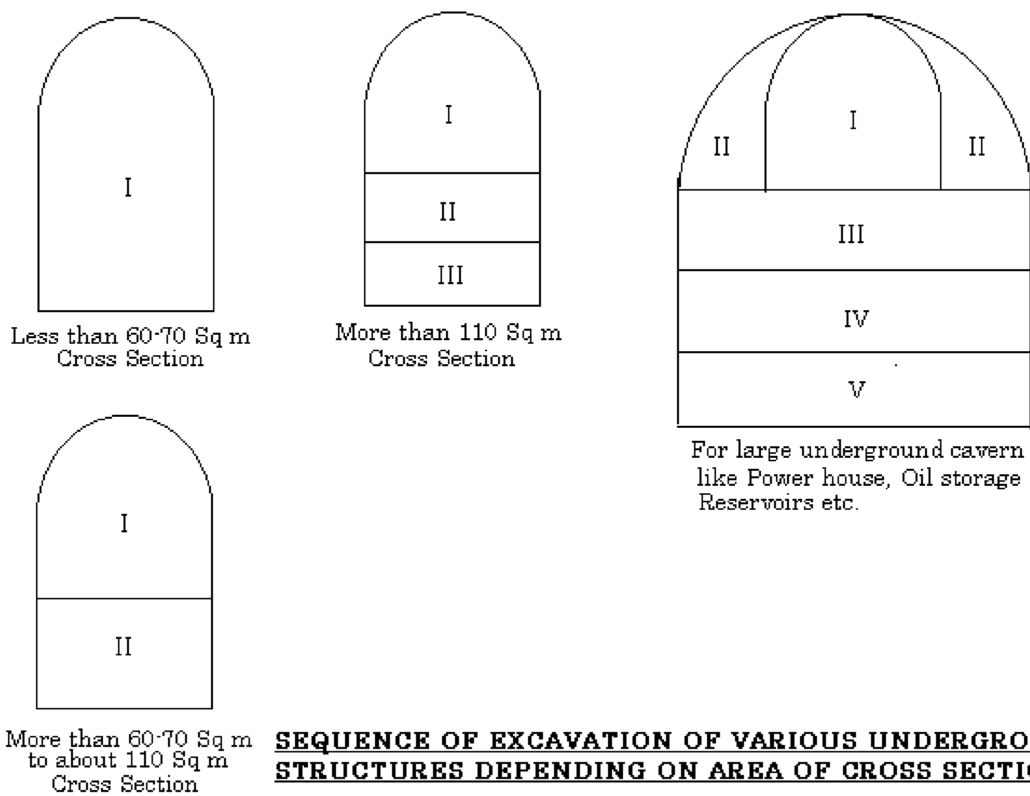
The most important is the rock must fulfill its ability to remain stable, when excavated. Therefore, it is important to consider the joints and cracks in a rock as these discontinuities can serve as a point of failure in a rock mass under stress. Tunnel engineers generally classify rocks on the basis of resistance to deformation (Strength), amount of weathering and general resistance to weathering. It has been seen that, Igneous and metamorphic rocks are more resistant to deformation and weathering than sedimentary rocks.

Technique of the blasting: Tunneling in rocks is currently performed mainly by blasting, as this method only is capable of providing sufficiently high effectiveness and economics in the construction of tunnel in tough rocks. Tunneling by 'tunnel borers' is considered to be less effective especially as regards the construction of tunnels of large cross sectional areas.

Unlike bench blasting, tunnel blasting has only one free face and holes are drilled normal to the free face surface. In such a situation, the explosives charge will blow out a narrow funnel- shaped crater. But if the hole is drilled at a certain angle to the free face, the result will be better, as the major part of the gasses will break out the rock in the direction of free face. Alternatively, if large diameter dummy holes parallel to the blast holes are drilled, the breakage performance is better as the large diameter dummy holes provide additional free face. The initial opening/cut created either by angled holes or by holes drilled parallel to large diameter dummy holes are widen subsequently by the holes fired after cut holes using proper delays. In other words, the main difference between tunnel blasting and bench blasting is that tunnel blasting is done towards one free surface, while bench blasting is done towards two or more free surfaces. The rock is thus more constricted in the case of tunneling, and a second free face has to be created towards which the rock can break and be thrown away from the surface. This second face is produced by a cut in the tunnel face, which can be a parallel hole cut, a V-cut, or a fan-cut. After the cut opening is made, the stopping towards the cut begins. The final shape of the cross section is given by trimmers or contour holes with closer spacing and comparatively smaller charge.

Sequence of excavation of various underground structures depending on area of cross section of the excavation is given in Fig –2. Partial-face-blasting is sometimes more practical or may be required by ground condition or equipment limitations. Thus, tunnel-driving methods can be divided as:

a) Full-face tunneling method,
b) Top heading and bench method,
c) Benching with horizontal blast hole,
d) Benching with down holes.



SEQUENCE OF EXCAVATION OF VARIOUS UNDERGROUND STRUCTURES DEPENDING ON AREA OF CROSS SECTION

Fig-2

For tunnels generally smaller than 60 – 70 m² cross-section, full-face excavation gives maximum economy and efficiency. Full-face excavation is applicable for tough rock having little jointing. Blast holes are drilled into tunnel face either at right angles to the face (Burn Cut) or at an angle to the face (Wedge Cut). For good roof condition and with availability of better support system tunnel having larger cross-section than 60 – 70 m² also can be effectively excavated by full-face method.

For medium tunnel more than 60 – 70 m² area of cross section, normally top heading and benching sequence are adopted. In this method, driving an upper heading across the full width, one third to half of the final tunnel section. The lower section is removed later by benching. Top heading is generally driven throughout the full length of the tunnel before benching begins. In some operations the bench blasting is carried out simultaneously, but at another location within the tunnel.

For excavation of very large underground cavern etc., more number of stages is employed. There are many other variations such as, a centre crown drift, followed by two crown side drifts, then bench in one, two or three stages.

Tunnel Driving Methods by blasting in Rocks: The shot holes in a tunnel are arranged in a particular form or pattern. The drifting pattern, holes are generally divided into three groups, e.g. Cut holes, Easer and Trimmers.

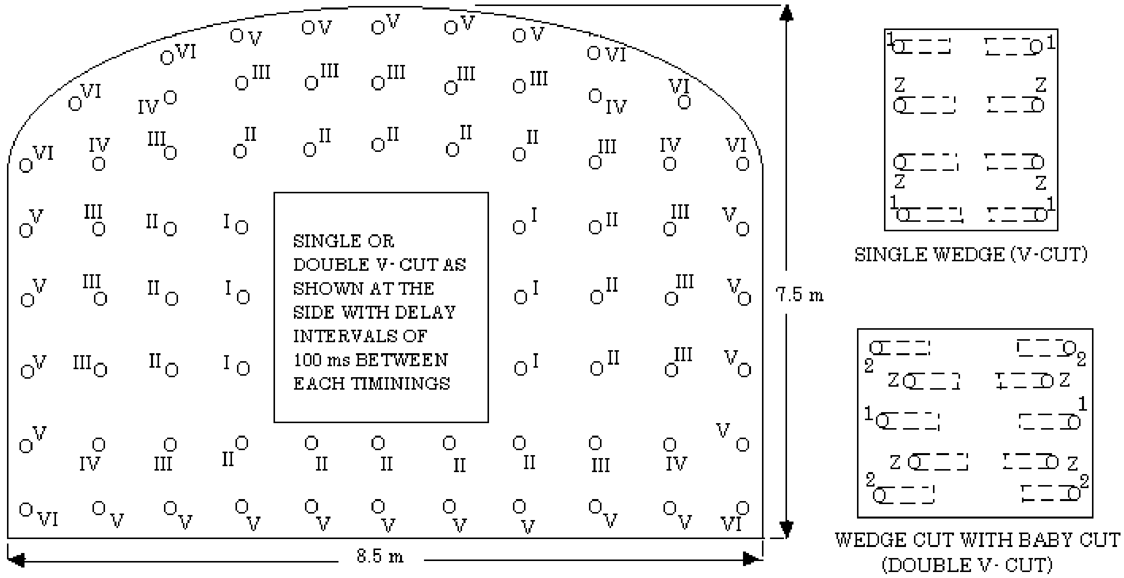
Cut holes: These holes are fired first to created free face for the shots of easers. Since these holes first make the opening in the face, they are prime responsible for the depth of pull.

Easers**:** The shot holes of this group are placed in the tunnel around the cut holes in two or more rings depending on the cross - sectional area. These holes ease the burden between the succeeding shot holes to enlarge the excavation area of the tunnel. Sometime Easer holes are also called stopping holes or enlarger holes.

Trimmers: The shot holes of this group are place around the easer, which are fired at the last to make the final shape of the tunnel. They are also called Contour holes.

The following type of cuts commonly used in Tunneling:

Wedge Cut or V- cut: Horizontal cut holes are driven in inclined at an angle 45 to 60 degree to the face towards the centre. Maximum concentration of charge at the apex of the cut holes, which are fired first to create a free face for the rest of the shot, which are fired next with the help of delays. If cross-section of tunnel allows, for deep pulls, burden should be reduced by giving another shallow wedge cut known as 'Baby Cut' (Double V-Cut). These relieving holes (Baby Cut) should be fired prior to the main wedge cut pattern (Fig –3).



DRILL ROUND USING SINGLE OR DOUBLE WEDGE (V-CUT) WITH DELAY SEQUENCE (FOR CUT HOLES 100MS DELAYS & 500MS DELAYS FOR EASER & TRIMMER)

Fig- 3

Cone / Pyramid / Diamond Cut: Four or Six cut holes are driven at the middle of the face, which converges at the end to form either, a Cone or a Pyramid or diamond shape. Maximum concentration of charge is at the apex of these cut holes, which are fired first to create a free face for the rest of the shot, which are fired next with the help of delays.

Drag Cut / Draw Cut: This type of cut is most suitable for the laminated rocks for "controlled blasting " in drivage of smaller cross-sectional area to break the rock along the cleavage planes.

Parallel holes Cut (*Burn Cut*) and its hole diameter & placement: Burn cut makes it possible to increase the depth of the round much more than is possible with angled cuts such as wedge/ cone/ pyramid cuts. Burn cut usually need more holes per round and a somewhat higher charge factor; but by increasing advance per round, economies result from ability to take advantage of the optimum depth of round to be fitted in the most economical cycle for drilling, blasting and mucking.

Clusters of parallel shot holes are drilled at perpendicular to the face to blast out a cavity in the centre of the heading. Some of the holes are heavily charged with explosives while others are kept empty to provided free face for reflection of shock waves. There is specific geometrical relationship between the diameter of empty hole and spacing between the centres of empty hole and charged holes in a given rock, which gives the essential condition of free breakage. It is most important that Burn Cut holes be drilled parallel as possible and at the design distance from each other. Deviations can result poor blasting of the cut. The burn cut should be drilled at least 150 mm longer than the blast holes in the remainder of the round. All types of rock expand on breaking, the amount of expansion tending to increase with degree of fragmentation. Some time, the drilling pattern and explosives usage may cause a swell of broken rock that is sufficiently high to solidly freeze the cut. Burn cut having sufficient number of uncharged holes (Relief holes) reduce chances of such freezing. A suitable selection of Delays in the round also prevent freezing.

Burn cuts can be drilled anywhere in the face. Fundamentally, all the variations of burn cut utilize the same principle. Unlike the angled cuts which are designed to break out a wedge or cone of material, burn cuts are intended to shatter/ pulverize the rock, breaking into small fragments which are expelled by blast to leave a roughly cylindrical opening.

It is a good practice to incorporate one or more large diameter uncharged relief holes together with smaller diameter charged blast holes in Burn Cut. Larger the diameter and more the number of uncharged relief holes in the cut, less serious is the consequence of drilling deviations, less chances of freezing and better advance per round of blast. When it is impossible to drill large diameter relief holes, it is a common practice to leave some of the holes uncharged, which acts as relieving holes for Burn Cut. Burn cuts drilled out of large diameter uncharged relief holes and smaller diameter uncharged relief holes are shown in Fig- 4. It is very important that proper delay sequence is used in blasting. Each hole should be fired with delays of different timings. For cut holes, a delay period of about 100 ms between successive holes is recommended. The objective is to provide ample free face before firing commences in a particular hole. Therefore, the delay timing should be selected in such a way, to allow the fragmented rock to come out before a hole being fired. For Easer and Trimmer holes, it is a good practice to use longer delays, preferably half-second delays.

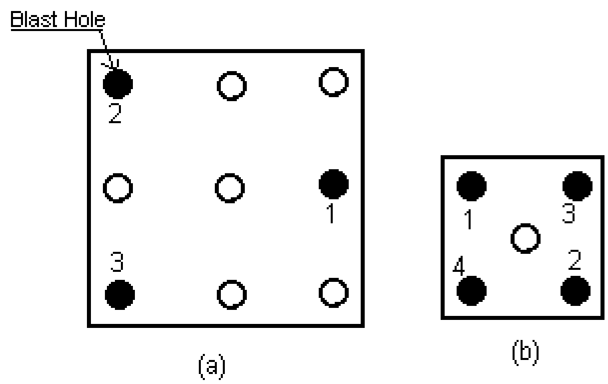
In Burn cut, the advance per round does not depend upon the working space available for drilling blast holes at acute angles to the face. With a wedge cut, the width the tunnel restricts the angle of cut holes and hence, the advance per round.

The cut may be placed alternately on the right or left side, in relatively undisturbed rock. To obtain good forward movement and centering of the muck pile, the cut may be placed approximately in the middle of the cross section and quite low down. The cut holes occupy an area of approximately 2 sq m. small tunnel faces may need only cut holes and contour holes. When designing the cut, the parameters such as, Diameter of the large empty hole; Burden; and Charge concentration are the most important for achieving good result.

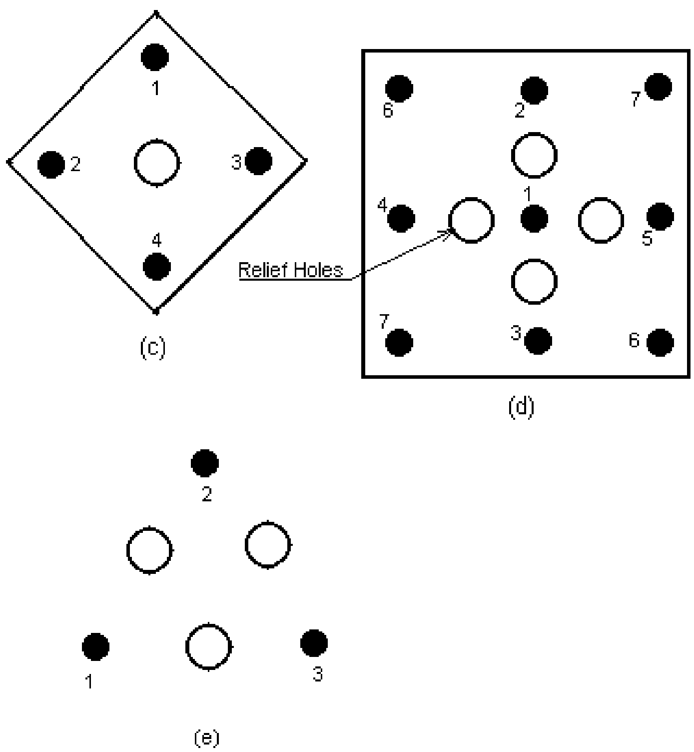
One of the parameter for good advance of the blasted round is the diameter of the large empty hole. Larger the diameter, the deeper the cut may be drilled and a greater advance can be expected. One of the most common causes of short advance is too small an empty hole in relation to the hole depth. An advance of approximate 90% can be expected for a hole depth of 4m (45mm dia.) and one empty hole of 102 to 120 mm diameter. If several empty holes are used, a fictitious diameter has to be calculated in accordance with the formula $D = d \sqrt{n}$, where D= fictitious empty hole large diameter, d=diameter of empty large holes and n=number of holes. In order to calculate burden in the first square, the diameter of large hole is used in the case of one large hole and fictitious diameter in the case of several large diameter holes. The distance between the blast hole and large empty hole should not be greater than 1.5 d for opening to be clean blasted. If the distance is longer, there is merely breakage and when the distance is shorter there is great risk that the blast hole and empty hole will meet. Therefore, $a = 1.5\ D$, or $a = 1.5\ d$. Holes closest to the empty hole s must be charged carefully. Too low a charge concentration in the hole may not break the rock, while too high a charge concentration may throw the rock against opposite wall of the large hole with such a velocity that the burden rock will be re-compacted there and not blown out through the large hole. Full advance is restricted then.

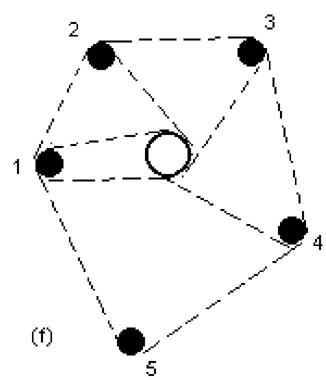
In parallel cuts, three relief holes provide a larger expansion volume for blast holes. Three parallel relief holes provide better results in medium to soft rock formation also and prevent freezing than single relief hole system. Spiral cut configuration also useful as it enlarges the expansion volume available for later firing holes. The burden increased for each blast. Configuration of relief holes in parallel cut have been shown in Fig-4 and Fig-5.

Perimeter holes are generally drilled with a lookout, diverging from theoretical wall line by upto about 100mm, since it is not possible to drill holes right at the edge of the line of excavated opening. Successive blasts result in zigzag shape in the tunnel wall. Therefore over break is generally is unavoidable.



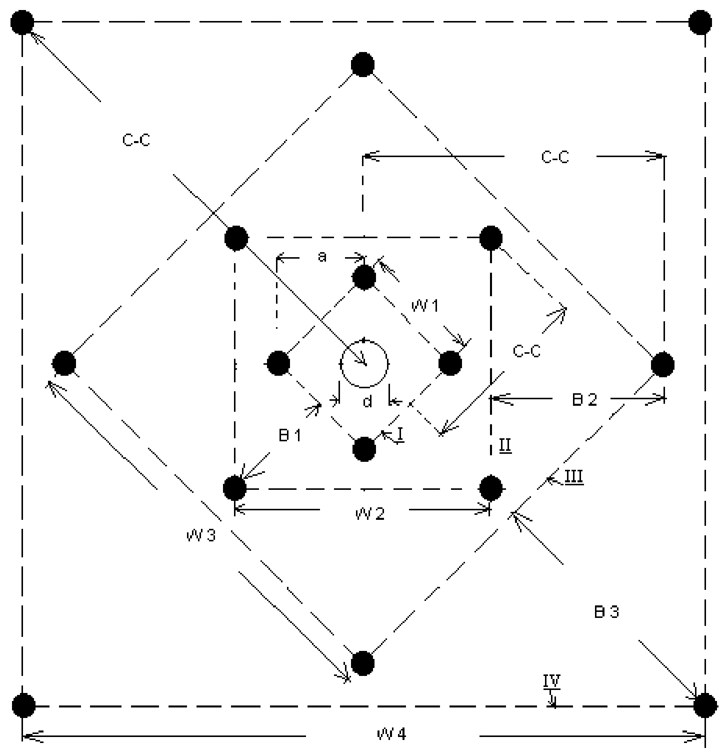
Burn Cuts with small Dia.Relief Holes





Burn Cuts with Large Dia. Relief Holes

Fig-4



BURN CUT BLASTING PATTERN

Fig-5

Number of Blast holes: There are number of Thumb Rules available for deciding number of blast holes to be drilled for techno-economy purpose. One of them is given below:

Swedish Rule: N=37.6 + 1.36 S (In tough rocks)
N = 30.9 + 1.0 S (In Medium tough rocks
Where S=Cross-section of tunnel.

However, requirement of number of hole, specific charge etc. are site specific and vary from site to site, which depend on Ground strata, Cross section of opening and many more factors. The optimum number of holes, specific charge etc., required are to be established by conducting number of trials at the site. Some examples of Specific charge requirement and number of blast holes required have been tried in Fig-6 and Fig-7.

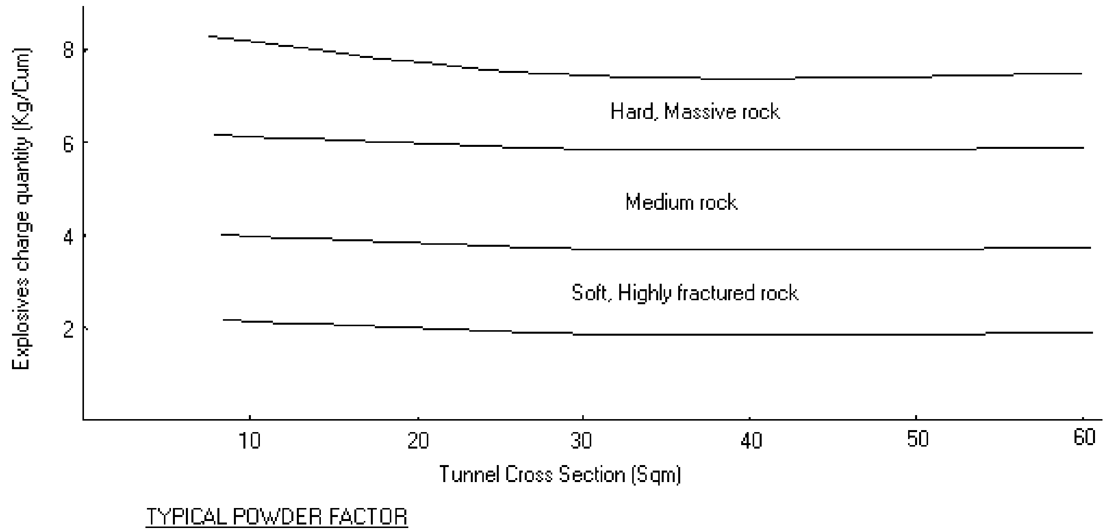


Fig-6

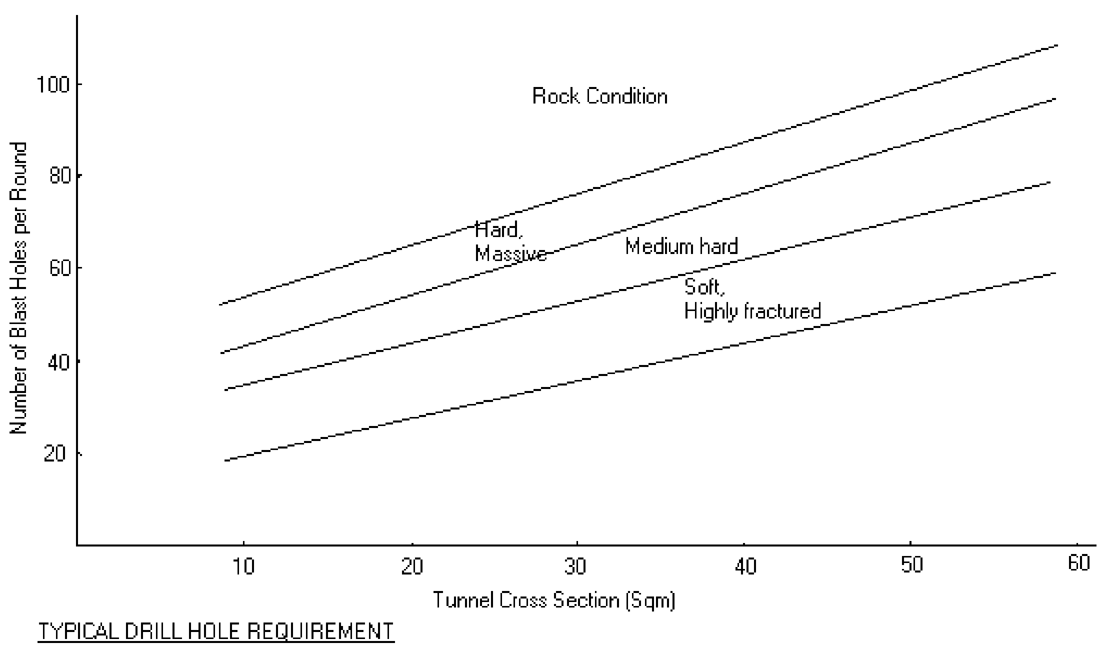


Fig - 7

Explosives consumption (Specific charge and Drill factor): Two parameters often calculated for blast design: 1) Specific charge i.e., consumption of explosives in kg per cubic metre of blasted rock, 2) Drill factor i.e., total length of drill hole in metre per cubic metre of blasted rock.

The consumption of explosives in tunnel blasting is higher than in bench blasting. The specific charge is 3 to 10 times higher than that for bench blasting depending on strata condition, cross-section of opening etc. The reason behind the requirement of higher specific charge are large drilling scatter, the confinement of the round, heave of lower rock upwards to ensure swell, lack of co-operation between adjacent blast-holes in the fragmentation work etc. The consumption of explosives is greatest in the cut area of the blast. One square metre area around the empty hole/s in a parallel cut generally consume approximately 7.0 kg/ m^3, and the specific charge decrease with the distance from the cut until it reaches a minimum value of about 0.9 kg/ m^3. Similarly, drill factor also depend on strata condition, cross-section of opening, diameter of drill used etc. Typical value of drill factor may vary from 0.8 to 6.0 m/m^3.

Delay Sequence and use of NONEL: As mentioned earlier, blasting condition of tunnel is more difficult than bench blasting. The selection of delays and its timings should be such that, a free face is effectively created by moving out the broken rock mass so much that the rock volume after swell from subsequent blasts must be accommodated. To understand above the fracturing/ breaking time and time required to spreading out cracks in rocks to be studied. The time required to spreading out cracks and formation of fracturing/ breaking depend on the rock condition. The speed of developing cracks/ fractures is faster for hard and brittle rock than softer rocks. This rock breaking speed varies from 1 to 3 ms time per metre. Whereas, ejection speed of rock after blast may be about 20 to 30 metre per second, i.e., 2 to 3 cm/ms. Thus, for a 4 metre long hole broken rock takes about 300 to 400 ms for complete removal from the cut. For this reason, long period delays/ half-second delays are preferable for tunnel blasting. Some time, combination of short and long delays are used. The short delays mostly used in cut holes, as it requires shattering effect initially for rock breaking to create free face for easer holes.

Controlled Blasting: The ideal blast result in a minimum of damage to the rock that remains and a minimum of over break. This is achieved by controlled blasting, which has many advantages:

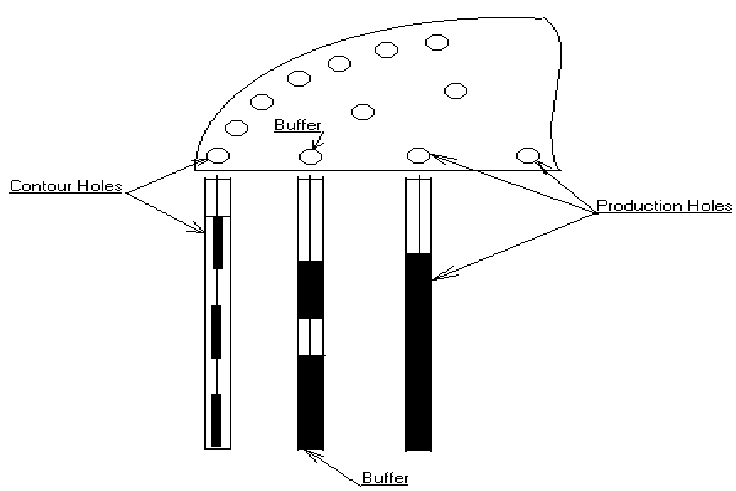
a) Less rock damage means greater stability and less ground support required.
b) The tunneling operation is safer as less scaling is required.
c) Less over break makes a smoother hydraulic surface for an unlined tunnel.
d) For a lined tunnel, less concrete required to fill excess void created by over break.

Controlled blasting involves a closer spacing of Contour holes or perimeter holes or Trim holes, which are loaded lighter than the Buffer and production holes. Generally, as a thumb rule 10 to 12 times hole diameter in medium to tough rock and 5 to 6 times hole diameter in poor, fragmented rock are kept as spacing. In fact, burden and spacing of Buffer and Contour row should be about 75% and 40% that of production rows respectively. Since, controlled usually require more blast holes, it takes longer to execute and uses more drill steel. For these reasons, contractors are often reluctant to incorporate principles of controlled blasting.

Loading and charging of contour holes are done with explosives of low VOD packed in small diameter cartridges in relation to drill diameter used. Unlike production drill hole blast where higher charge concentration is required, contour drill holes require low charge concentration and explosives should be lightly distributed all along the length of the bore hole. Some time use of high grammage Detonating Fuse (about 40 gm/m core wt. To 60 gm/m core wt.) for contour blasting can give effective result in tunneling and civil construction works. This result in an air cushion effect, which prevent over break and reduces in-situ rock damage.

More than just design of proper perimeter blast, other aspects of controlled blasting also to be looked at. Blast damage may occur long before the trim holes are detonated. Controlled blasting requires careful design and selection of all aspects of hole diameter, hole charge, hole spacing & burden, delays as well as careful execution of work. One of the keys to successful controlled blasting is precise drilling – deviation of holes from their design location lead to alter spacing & burden, causing blast damage and irregular surfaces.

Other parameter to be looked at is charging of holes of Buffer row. In the buffer row the charge factor should not be more than 0.6 Kg/m^3 to terminate the back break along the line of buffer row. At the same time, charge factor in the perimeter row should not be more than 0.4 Kg/m^3 and it should be well distributed throughout the blast holes. Hollow bamboo spacers of 150 mm long may also be used to achieve better distribution of smaller cartridges in perimeter holes. A gradual reduction of charge factor from Cut holes at the center towards the perimeter of the tunnel will have efficient control on over break. Fig-8 shows charging of contour holes, buffer and production holes.



CHARGING OF CONTOUR, BUFFER AND PRODUCTION HOLES

Fig-8

Observation of the blasted surfaces after the blast can give better clues to the accuracy of drilling and blasting and effectiveness of the technique. A measure of success is the half-cast factor, is the ratio of half-casts of the blast holes visible on the blasted surface to the total length of trim holes. Depending on the rock quality a half-cast factor of 50 to 80 percent can be said to be quite satisfactory. There are other means to verify the extent of rock damage because of blast behind the wall. This may be done using Seismic refraction techniques and borescope or permeability measurements in cored bore-holes. The extent of disturbed zone may vary from as little as 0.1 – 0.2 m with excellent controlled blasting to more than 2 m with un-controlled blasting.

Blast induced Ground vibration: When an explosives charge is detonated in a blast hole, the rock immediately surrounding the charge is fractured, split apart and is displaced. At a certain distance from the blast holes, the explosives energy decreases to a level, which causes no further shattering or displacement and continuous to travel through the rock as an elastic ground vibration. The ground motion is literally a wave motion spreading outwards from the blast, much as ripples spread outwards from the impact of a stone dropped into a pool of water.

The ground / rock through which the wave travels is considered to be elastic medium, composed of innumerable individual particles. As a disturbance, of these particles are set into a random oscillatory motion about their position, a ground motion wave being generated. Each particle transmits energy successively to the next.

The total energy of ground motion wave generated in the rock around a blast varies directly with the charged detonating. As the ground motion wave propagates outwards from a blast, the volume of the rock subject to the compression wave increases. Since the energy in a ground shot is distributed over successively greater volume of rock, the ground motion must decrease. Thus energy losses occur with each successive transmission, so that as the ground wave spread outwards, it diminishes in intensity and the particle gradually return to their rest position.

In a well-designed blast, most of the explosion energy is spent in breaking of ground and throw of the blasted rock. A small amount of energy is converted into ground vibration. When blast holes are under or over charged and absence of proper free face a great deal of liberated energy is wasted and converted into ground vibration, as explosion energy is not utilized in fragmenting / breaking of rock and throw.

Effect of Ground Vibration: Ground vibration can cause physical damages to mine plant structures and to neighbouring residences. The most type of damage associated with ground vibration is the aggravation of existing minor cracks in the structure. The damages to the structure depend on the intensity of the vibration; which mostly depend on the distance of the structure from the blast site, Explosives charge weight per Delay, Frequency of vibration, Blast geometry and confinement. Therefore, the following primary variables are responsible for damages:

- Distance of structure from blast site (Peak particle velocity reduces as the distance increases)
- Explosive charge weight per delay
- Frequency of vibration (Low frequency wave below 10 Hz cause more damage to structure)

Vibration propagation Equation: The most accepted index (USBM's PREDICTOR EQUATION) of Ground vibration generated by blasting is the Peak Particle Velocity (PPV). It has been well established that, PPV depends on maximum charge per delay, distance from blast-site to measurement point and Ground geology, and the relationship is as follows:

$$\mathbf{V = K\,(D/\sqrt{Q})^{-B}}$$

Where, V is the **Peak Particle Velocity (PPV)**,

D is the **Distance** of the measuring transducer

Q is the **Maximum charge weight per delay**.

K and B are **Site constants**, to be determined by regression analysis.

$D/\sqrt{Q}$ is called as **Scaled Distance**.

Taking logarithm of both sides of the Predictor equation, we get

$\text{Log } V = \text{Log } K - B \text{ Log } (D/\sqrt{Q})$

If, $Y = \text{Log } V$; $X = \text{Log } (D/\sqrt{Q})$ and $C = \text{Log } K$; then the above equation represents a straight line of the form $Y = C - BX$ (Straight Line plotted on Log V as Y axis and Log $(D/\sqrt{Q})$ as X axis below); where B is the negative slope of the straight line and C is the point of interception on Y axis. Relationship of Scaled Distance with Peak Particle Velocity (PPV) on Log scale shown in Fig.-9.

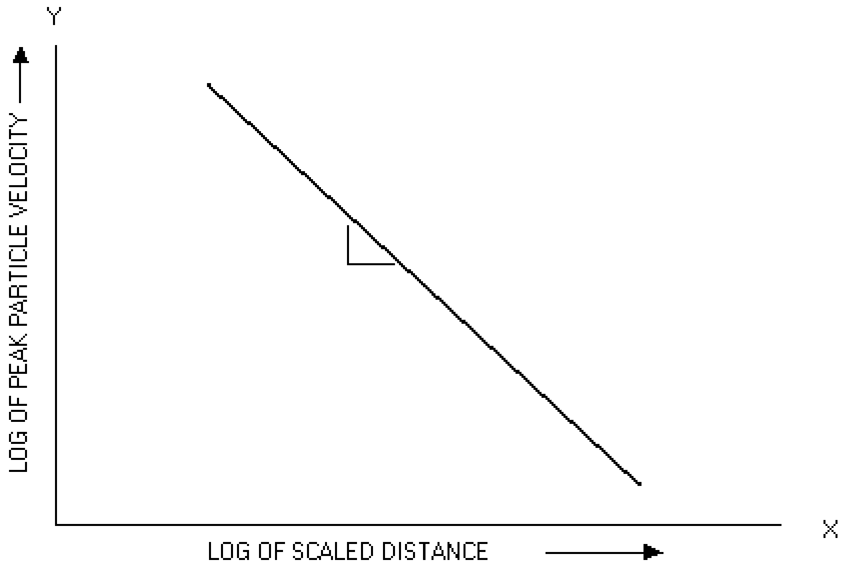


Fig-9

Damage to structures caused by blasting is related to PPV. It is generally recognized that a PPV upto 50mm/s is less dangerous for residential structure/ buildings. In fact, most well built structures can withstand PPV greater than 50 mm/s. Human perception is far more sensitive to blast vibration than structures. Vibrations are clearly noticeable at PPV as low as 5 mm/s and disturbing at 20 mm/s. Perception of vibration is, to a degree, a function of frequency of the vibration; low-frequency vibration (< 8 Hz) are more readily felt than high frequency vibrations. Furthermore, vibrations are more objectionable during night hours. Setting acceptable vibration limits in an urban area requires adherence to the locally established code and practices. For mining in India, DGMS has made recommendations on threshold limit of PPV.

Conclusion: Underground hard rock excavation and tunnels are highly specialized type of job, which requires special type of explosives, drilling and blasting techniques. In modern days because of special emphasize given on infrastructure development these projects have become great national importance. Therefore execution of such projects within stipulated time and money has become all the more essential. Moreover, every possible steps should be taken to minimize adverse effect of such excavations such as, controlling over break of strata, ground vibration to prevent damages of nearby structures etc. To cope up the challenge faced by mining engineers for such important projects extensive knowledge on explosives, blast design, perception about change in geology of the area, local environmental rules etc. are the prerequisite.

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