Drilling and blasting as a tunnel excavation method

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DRILLING & BLASTING

AS A TUNNEL EXCAVATION METHOD

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1. INTRODUCTION

By far the most common technique of rock excavation is that of drilling and blasting. From the earliest days of blasting with black powder, there have been steady developments in explosives, detonating and delaying techniques and in our understanding of the mechanics of rock breakage by explosives.

The drill & blast method is still the most typical method for medium to hard rock conditions. It can be applied to a wide range of rock conditions. Some of its features include versatile equipment, fast start-up and relatively low capital cost tied to the equipment. On the other hand, the cyclic nature of the drill & blast method requires good work site organization. Blast vibrations and noise also restrict the use of drill & blast in urban areas.

At fault in this system are owners and managers who are more concerned with cost than with safety and design or planning engineers who see both sides but are not prepared to get involved because they view blasting as a black art with the added threat of severe legal penalties for errors.

While TBM's are used in many tunneling projects, most underground excavation in rock is still performed using blasting techniques. The design team should specify or approve the proposed method of excavation. In modern tunnel and underground cavern excavation, it is possible to select from many different methods. The following factors should be taken into consideration when selecting the method:

- Tunnel dimensions
- Tunnel geometry
- Length of tunnel, total volume to be excavated
- Geological and rock mechanical conditions
- Ground water level and expected water inflow
- Vibration restrictions
- Allowed ground settlements

The methods can be divided into drill & blast, and mechanical excavation. Mechanical methods can be split further to partial face (e.g. roadheaders, hammers, excavators) or full face (TBM, shield, pipe jacking, micro tunneling).

2. EXPLOSIVES

The development of explosives has moved in the direction of safer products with better fumes characteristics. Modern explosives like Emulite and Dynamex M are well oxygen-balanced with a minimum of noxious fumes.

Technical properties; An explosive has three basic characteristics:

- It is a chemical compound or mixture ignited by heat, impact, friction or a combination of all three

- Upon ignition, it decomposes very rapidly in a detonation (as opposed to a deflagration, which is slower and occurs for instance with gunpowder),

Upon detonation, heat (4500 °C) and large quantities of high-pressure gases (250.000 bar) are rapidly released. The gases expand rapidly at high force to overcome the confining forces of the surrounding rock formation. In commercial blasting, energy released by a detonation results in four basic effects; rock fragmentation, rock displacement, ground vibration and air blast. The technical properties of explosives used in surface rock excavations are;

- Efficiency and stability
- Easy detonation and good explosive properties
- Safe handling
- Good film characteristics
- Non-toxic
- Water resistance and good storage properties
- Environmental properties
- Resistance to freezing
- Oxygen balance
- Shelf life

2.1. Types of Explosives

An extensive range of different types and grades of explosives is made to suit all blasting requirements. A breakdown of blasting explosives is presented in Table 1.

Table 1 *Breakdown of blasting explosives.*

Explosive base	*Explosive type*	*Features*
Nitro-glycerin	Dynamite Gelatin	A highly adaptable cartridge explosive currently widely used because of its excellent performance in smaller diameter holes.
Ammonium nitrate	ANFO	A low-cost, high-power, extremely safe, liquid explosive made from porous prilled ammonium nitrate and fuel oil. Poor water resistance.
Water Oil	Slurry Emulsion	Essentially, ANFO which becomes water resistant by adding water and forming either a gel (water gel), or creating astable oil/water emulsion and ANFO called heavy ANFO. Available in package or liquid form.

2.1.1. Dynamite and gelatins

Over the years, ammonium nitrate has become more important in dynamite, replacing a large portion of nitroglycerin. There are three basic types of dynamite: granular, semi-gelatin and gelatin. Gelatin and semi-gelatin dynamite contain nitrocotton, a cellulose nitrate that combines with nitroglycerin to form a cohesive gel. The viscosity of this product depends on the percentage of nitrocotton. Granular dynamite, on the other hand, does not contain nitrocotton and has a grainy texture. Usually, the higher the nitroglycerin percentage, the more water resistant the explosive becomes. Dynamite issued in bottom charges. Low-percentage dynamite is often used in column charges. These explosives are often used in small diameter blastholes at construction blasting sites. Paraffin paper covers are used for small-cartridge explosives (d=40 mm) and plastic bags for bigger cartridges. In certain smooth blasting projects, pre-splitting and generally for tasks where a very light charge is necessary, special dynamite can be used as a ready-made tube charge.

2.1.2.Aniitti

Aniitti is a non-nitroglycerin, ammonium-nitrate explosive, which among other things contains trotyl (TNT) and aluminum. Aniitti in cartridges is used especially in drifting. Aniitti contained in plastic bags is mostly used in large holes as a column charge and in clearing.

2.1.3. Original ANFO

ANFO is a mixture of ammonium nitrate and fuel oil (5.7%), in which ammonium nitrate acts as the oxidizer and the fuel oil acts as the fuel. ANFO offers great economy and safety in modern blasting applications. It is generally one-third to one-half cheaper than nitroglycerin explosives and it is considerably safer to handle because it is non-sensitive. In many types of blasting, ANFO produces better fragmentation due to its high gas producing properties. ANFO is among the best explosive for blasting dry holes in excess of 51 mm (2") in hole diameter, which are conductive to breakage by gaseous expansion. However, it is not so good in small-diameter blastholes and conditions that require very high detonating velocities. The main disadvantage of ANFO is that it is not water resistant: if it gets wet, it no longer detonates. Some ANFO products can be charged in wet holes, if the water is removed before charging. Leaving ANFO in a loaded hole for an extended period of time should therefore be avoided. Additionally, ANFO can not be detonated by a normal detonator. Detonation velocity changes with the diameter of the blasthole and reaches the highest velocity of 4400 m/s in a 250 mm blasthole. Likewise, detonation velocity decreases with the diameter of the blasthole. When the diameter is less than 25 mm, the detonation will not be stable. ANFO is most suitable in middle and large-diameter blastholes (75 to 250 mm) under dry conditions. Initiation of ANFO should not be made with detonating cord in small and medium-sized blastholes (25 to 100 mm). The detonating cord will initiate the ANFO diametrically (axial priming) and as the ANFO will not reach a stable velocity of detonation (2000 to 4400 m/s), the chemical reaction will be incomplete.

2.1.4. Reduced ANFO

Different methods of mixing ANFO with inert material has been tried, but today the most commonly used material for reduction is expanded polystyrene spheres. Due to varying density, ANFO and the polystyrene spheres tend to separate. However, a new charging technique developed by Dyno Industries Norway solved this problem. The technique is similar to that used for concrete grouting. ANFO and the reduction material are stored in separate containers but then mixed together in a charging hose through which the mixture is blown into the blasthole. Two containers are needed for charging by this method. However, it is most practical and least bulky when one container is placed inside the other.

2.1.5. Heavy ANFO

Heavy ANFO is a mixture of ANFO and emulsion explosive. It is becoming more popular because it is often as effective as pure emulsions and considerably cheaper. Prilled ANFO consists of spheres or prills of sensitized ammonium nitrate infused with fuel oil. Maximum density occurs when the prills are in mutual contact, which leaves voids that, in heavy ANFO, are filled with a base emulsion, cold pumpable explosive with sufficient viscosity and stability. The sensitizing mechanism for emulsion explosives is provided by voids. In heavy ANFO, the prills act as voids or density adjusters and the emulsion fills the gaps. The proportions of the constituents can be varied to alter sensitivity, energy and water resistance. With the normal range of compositions, density increases with emulsion content up to a maximum of 1.3 kg/dm^3. The charge's sensitivity is opposite to both the density and emulsion content. Both energy and sensitivity peak at a density of approximately 1,3. Adding microballoons to the base emulsion enhances sensitivity, energy and water resistance, however it also increases the cost. Water resistance is also dependent upon emulsion content and quality, sensitivity, degree of blending and particularly storage time during and after loading.

2.1.6. Slurries

Slurries are specially designed for large-hole blasting and wet conditions. Slurries are not normally cap-insensitive and therefore must be initiated with a primer. Slurries are water resistant and are either pumped straight into the hole or applied in plastic bags. Slurry contains ammonium nitrate, and often aluminum, water and substances to keep the explosive homo-geneous. The properties of any individual composition depend on the type and proportions of various solid ingredients. Because it is denser than water (1,4 – 1,6 kg/dm^3), the slurry sinks to the bottom of a wet blasthole. The detonation velocity of slurries ranges from 3,400 m/s to 5,500 m/s.

2.1.7. Emulsion explosives

Emulsions are so-called two-phase products. The dispersed phase is dissimulated throughout a continuous phase. Explosive emulsions consist of a mixture of fuel and oxidant components. The oxidizers are primarily nitrates and the fuels mostly mineral or organic hydrocarbon derivations. The oxidant/fuel ratio is approximately 10/1. When inspected with an electron microscopy, the structure of the emulsion exhibits a polyhedral shape with each droplet covered by a film of the fuel phase. The detonation reaction occurs at the boundaries between the two phases. Emulsions provide increased explosive efficiency because both phases are liquid and the dispersed nitrate solution droplets are smaller than other conventional explosives; 0.001 mm as opposed to 0.2 mm. They are tightly packed within the fuel phase and provide increased surface contact, effectively enhancing the reaction. The strength of the reaction can therefore be altered by changing the degree of fuel and oxidant. The water content of the nitrate is also reduced by using super-saturated solutions. Increased efficiency is reflected in the detonation velocity, 5000-6000 m/s for emulsions compared with approximately 3200 m/s for ANFO and 3300 m/s for slurry. This high-velocity detonation is one of the major advantages of emulsions as it provides high-shock energy, which is a significant factor in hard rocks. Unlike other explosives, emulsions do not use chemical sensitizers. Instead, voids in the emulsion fulfills this requirement. The number of voids determines the density of the mixture. The viscosity and density of any emulsion is determined largely by the physical characteristics of the organic fuel phase, which can vary from liquid fuel oil to viscous waxes. Unlike slurries, emulsions can not be gelled or cross-linked, their structure being characteristic of the nitrate and continuous fuel phase. Bulk emulsion explosives are especially suited for open-pit mining because of their water resistant properties and chemical stability. Emulsions are structurally distinct from slurries because they do not contain thickeners or gelling compounds, and require mixing at approximately 80°C. Also they do not contain structural additives, the phases must cool by 30° - 50°C before the fuel phase becomes semirigid. By using different percentages of microballoons and aluminum, a wide range of emulsion explosives can be manufactured. The cap sensitive range is intended for small and medium-diameter blastholes and is delivered in paper shells and plastic bags. The non-cap sensitive range is intended for medium and large-diameter blastholes in bench blasting, and is delivered in plastic hoses. Pumpable bulk emulsions, which are economical alternatives to ANFO in quarrying, mining and tunneling are also very interesting. Pumpable bulk emulsifying and mixing process can take place either at the bulk emulsion station or on the mixing/loading truck. When the explosive is prepared at the station, the explosive truck acts as a simple transport and loading unit for the explosives. Alternatively, the truck can be used as a mixing and loading unit. In the latter case, explosive preparation will take place at the worksite, allowing continuous blasthole loading with precise amounts of various mixtures. This system enables explosives to be produced economically. Loading is accurate, and bulk emulsion can easily be pumped at speeds of up to 200 kg/min. A further advantage is that it can even be pumped into holes containing water. The substance can additionally remain unchanged for months even under severe conditions.

3. BENCH & TUNNEL BLASTING

Work on the strength of jointed rock masses suggests that this strength is influenced by the degree of interlocking between individual rock blocks separated by discontinuities such as bedding planes and joints. For all practical purposes, the tensile strength of these discontinuities can be taken as zero, and a small amount of opening or shear displacement will result in a dramatic drop in the interlocking of the individual blocks. It is easy to visualise how the high pressure gases expanding outwards from an explosion will jet into these discontinuities and cause a breakdown of this important block interlocking. Obviously, the amount of damage or strength reduction will vary with distance from the explosive charge, and also with the in situ stresses which have to be overcome by the high pressure gases before loosening of the rock can take place.

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. This restricts round length, and the volume of rock that can be blasted at one time. 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 and can be either a parallel hole cut, a V-cut, a fan-cut or other ways of opening up the tunnel face. After the cut opening is made, the stoping towards the cut will begin. The stoping can be compared with bench blasting, but it requires a higher specific charge due to higher drilling deviation, desire for good fragmentation, and absence of hole inclination. In addition, overcharge of a tunnel blast does not have the same disastrous effect as in an open air blast, where high precision in calculation is a must.

4. DRILLING & BLASTING CYCLE

The typical cycle of excavation by blasting is performed in the following steps (Figure 1):

- Drilling blast holes and loading them with explosives.
- Detonating the blast, followed by ventilation to remove blast fumes.
- Removal of the blasted rock (mucking).
- Scaling crown and walls to remove loosened pieces of rock.
- Installing initial ground support.
- Advancing rail, ventilation, and utilities.

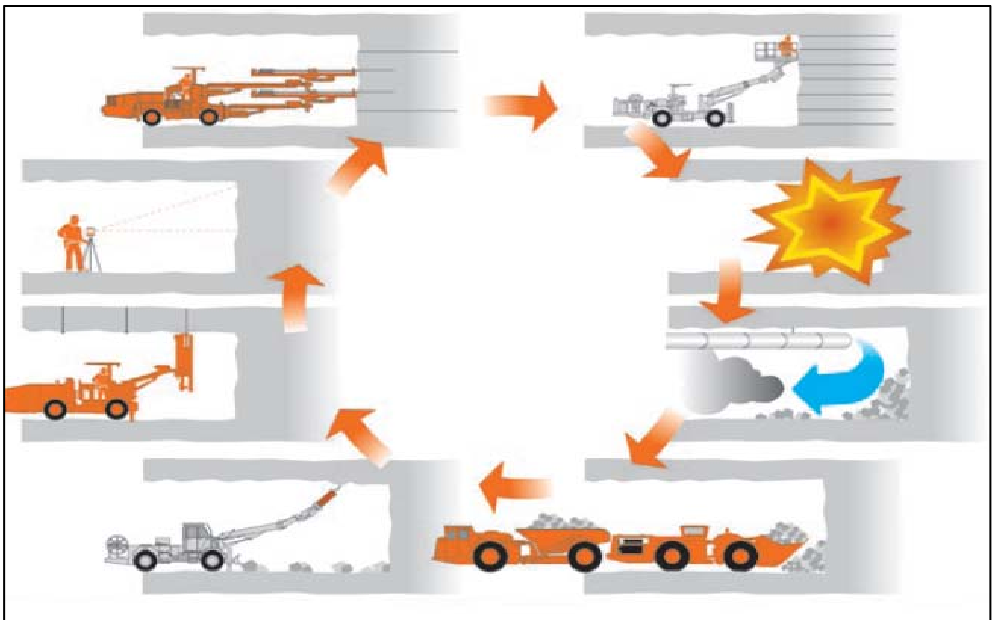

Figure 1 *Drill and blast cycle.*

5. DRILLING PATTERN

The drilling pattern ensures the distribution of the explosive in the rock and desired blasting result. Several factors must be taken into account when designing the drilling pattern: rock drillability and blastability, the type of explosives, blast vibration restrictions and accuracy requirements of the blasted wall etc. Many mines and excavation sites still plan their drilling patterns manually, but advanced computer programs are available and widely used. Computer programs make it easier to modify the patterns and fairly accurately predict the effects of changes in drilling, charging, loading and production. Computer programs are based on the same design information used in preparing patterns manually. Drilling pattern design in tunneling and drifting is based on the following factors:

- Tunnel dimensions
- Tunnel geometry
- Hole size
- Final quality requirements
- Geological and rock mechanical conditions

- Explosives availability and means of detonation
- Expected water leaks
- Vibration restrictions
- Drilling equipment

Depending on site conditions, all or some of the above factors are considered important enough to determine the tunnel drilling pattern. Construction sites typically have several variations of drilling patterns to take into account the changing conditions in each tunnel. Drifting in mines is carried out with 5 to 10 drilling patterns for different tunnel sizes (production drifters, haulage drifters, drawpoints, ramps etc.) The pattern is finalized at the drilling site.

Hole sizes under 38mm in diameter are often considered small, holes between 41mm – 64 mm intermediate, and those over 64 mm large. Most tunneling operations today are based on hole sizes between 38 -51mm in diameter. Only cut holes are larger than 51mm. Rock drills and mechanized drilling equipment used in tunneling and drifting are designed to give optimum performance in this hole range. Drifting rods are designed to match hole sizes and needs of horizontal drilling. Typical applications use tunneling rods and 1 1/4" and 1 1/2" drill steel sizes. Drill steels between 1" and 1 1/8" are used for hole sizes less than 38mm. The number of holes needed per tunnel face area decreases as hole size increases. The difference is not much in small tunnels, but becomes more significant in large tunnel face areas. Small hole sizes require smaller steels, but these bend more easily, giving rise to inaccurate holes and poor blasting. When designing a drilling pattern in tunneling, the main goal is to ensure the optimum number of correctly placed and accurately drilled holes. This helps to ensure successful charging and blasting, as well as produce accurate and smooth tunnel walls, roof and floor. A drilling pattern optimized in this way is also the most economical and efficient for the given conditions. The tunnel of drift face can be roughly divided into four sections (Figure 2).

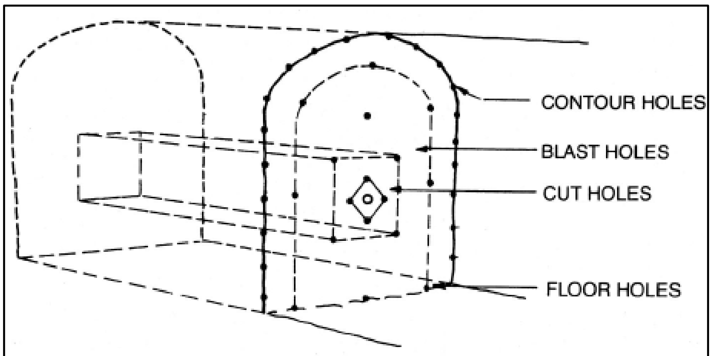

Figure 2 *Types of holes in a tunnel face.*

6. DEFINITIONS

6.1. Stoping

The holes surrounding the cut are called stopeholes. The diameter of a stopehole is typically between 41 - 51mm. Holes smaller than 41mm may require drilling an excessive number of holes to ensure successful blasting. Holes bigger than 51mm can result in excessive charging and an uncontrolled blast. Holes are placed around the cut section in an evenly distributed pattern using a space/burden ration of 1:1.1. If hole size is between 45 - 51mm, typical spacing and burden are both between 1.0m - 1.3m. Actual rock conditions and ability to drill in the required positions are factors that can reduce or add to the number of holes needed. The design of the drilling pattern can now be carried out and the cut located in the cross section in a suitable way.

6.2. Contour holes

Floor holes have approximately the same spacing as stope holes, but the burden is somewhat smaller; from 0,7m to 1,1m. Inaccurate or incorrect drilling and charging of the floor holes can leave unblasted bumps, which are difficult to remove later. The contour holes lie in the perimeter of the drilling pattern. In smooth blasting, contour holes are drilled closer to each other and are specially charged for smooth blasting purposes. Spacing is typically from 0.5m to 0.7m and burden varies between 1 and 1.25 times the space. This type of layout makes it possible to use special smooth blasting explosives, which limits the width and depth of the fracture zone in the walls and roof caused by blasting. In special circumstances, two or more smooth blasting rows can be used. In tunneling, however, contour holes are blasted with stopeholes, but timed to detonate last. The result in smooth contour excavation mostly depends on drilling accuracy.

6.3. Look-out angle

The drilling pattern also includes information on the look-out angle needed at different points on the tunnel face. The look-out angle is the angle between the practical (drilled) and the theoretical tunnel profile (Figure 3). If the contour holes are drilled parallel to the theoretical line of the tunnel, the tunnel face gets smaller and smaller after each round. To ensure that the correct tunnel profile is maintained, each contour hole is drilled at slight angle into the tunnel wall, the look-out angle, which of course can not be smaller than that permitted by the profile of the rock drill. Perimeter holes are usually drilled with a lookout, diverging from the theoretical wall line by up to about 100 mm (4 in.)

since it is not possible to drill right at the edge of the excavated opening. The size of the drill equipment requires a setback at an angle to cover the volume to be excavated. Successive blasts result in a tunnel wall surface shaped in a zigzag. Therefore, overbreak is generally unavoidable.

Adjusting the look-out angle by eye requires an experienced and skillful operator. Modern drilling rigs have electronic or automatic look-out angle indicators that enable correct adjustment of the look-out angle relative to standards alignment. Computerized drilling jumbos make setting, adjustment and monitoring of the look-out angle even easier. An incorrect look-out angle produces over or underbreak, both of which give uneconomical results. Other aspects such as curve and tunnel inclination also need to be considered when the drilling pattern is designed. Any excavation later on is both costly and time-consuming.

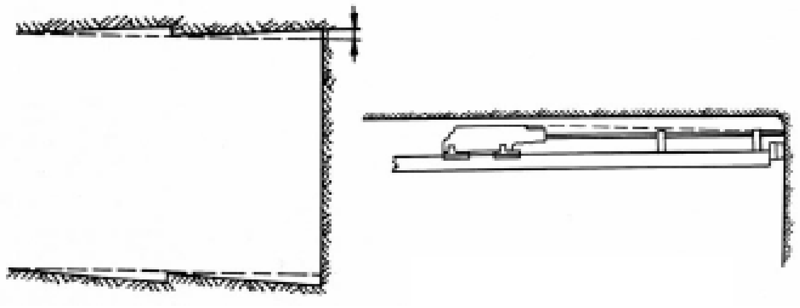

Figure 4 . The look-out angle.

6.4. Powder factor & Specific Charge

Two parameters are often calculated from a blast design: the powder factor or specific charge (kilograms of explosives per cubic meter of blasted rock) and the drill factor (total length of drill holes per volume of blasted rock (meter/cubic meter)). These are indicators of the overall economy of blasting and permit easy comparison among blast patterns. The powder factor varies greatly with the conditions. It is greater when the confinement is greater, the tunnel smaller, or when the rock is harder and more resilient. Rocks with voids sometimes require large powder factors. For most typical tunnel blasting, the powder factor varies between 0.6 and 5 kg/m^3. The powder factor can vary from 1 kg/m^3 in a tunnel with an opening size greater than 30 m^2 to more than 3 kg/m^3 for a size less than 10 m^2, in the same type of ground. Typical drill factors vary between 0.8 and 6 m/m^3.

7. CHARGING OF BLASTHOLES IN TUNNELS

The development of tunnel driving techniques has been tremendous during the last few years. The drilling techniques have developed from pneumatic drilling machines to electro-hydraulic drilling jumbos with very high capacity. The charging of the blastholes can be carried out quickly either manually with plastic pipe charges or mechanically with pneumatic charging equipment.

7.1. Charging With Tamping Rod

Tamping rod is used to tamp explosives cartridges in holes of small to medium diameters. The tamping rod should be made of wood or plastic. Any metallic fitting or pike should be of copper or brass. The diameter of the rod should be approx. 10 mm smaller than that of the blasthole thus giving space for legwires, NONEL tube, safety fuse or detonating cord.

7.2. Charging With Pneumatic Machines

Principally two types of pneumatic charging machines are available:

- Semi-automatic charging machines for cartridge explosives
- Pressure-ejector vessels for ANFO.

Semi-automatic charging machines are useful for upward holes, underwater blasting and fissured rocks where cartridges tend to jam but where a semi-ridged plastic hose could be introduced to the bottom of the hole. Pressure-ejector vessels for ANFO are mostly used in tunneling. Free flowing ANFO is normally poured into blastholes which are vertical or close to vertical. For horizontal and upward blastholes, the principal method of charging is via pneumatic charging devices. Such devices are also used for the charging downward blastholes where higher charging density is required. The principle of the charging machine is that the ANFO is transported from the container through a plastic hose, into the blasthole by pneumatic pressure.

Two main types of pneumatic charging machines for the charging of ANFO are available:

- Pressure vessel machines which use high pressure in the container. The ANFO is pumped through the hose into the blasthole.
- Ejector units where the ANFO is sucked from the container and blown through the hose into the blasthole.

Combined pressure/ejector machines are also available. ANOL is a pressure vessel device for charging ANFO in all kinds of applications. Prilled ANFO can be charged in upward blastholes with an inclination of up to 35° without running out. The flow of ANFO is remotely controlled via a charger. As ANFO is highly corrosive, all machine parts that come in contact with ANFO are made of stainless steel. ANOL is manufactured in sizes of 100, 150, 300 and 500 liters. The charging machine is a combined pressure/ejector unit for the charging of prilled ANFO in upward blastholes with diameters between 32-51 mm and a depth of up to 45 m. The ANFO is transported by the ejector at such a high velocity into the blasthole that the prills are crushed and stay in the blasthole. The flow of ANFO as well as the velocity of the ANFO through the hose are remotely controlled by the charger. The charging hose is anti-static as the ANFO is transported through the hose at high velocity causing a risk of static electricity accumulation. Due to this risk, all ANFO charging units must be grounded during charging operations.

7.3. Charging With Pump Trucks

In tunnel blasting operations, the explosive or blasting agent may be charged into the hole by a pump truck. An explosive or blasting agent, such as emulsion, can be manufactured at an on-site plant and pumped directly from the plant into the pump truck. Care must be taken when charging holes containing water. The charging hose must be introduced below water level to the bottom and lifted at the same pace as the hole is filled to avoid separation of the explosive column by water pockets.

8. BLASTING METHODS & CUT TYPES IN TUNNEL EXCAVATIONS

In tunnel excavation, blasting works outward from the first hole around the uncharged holes in the cut. Each blast provides more space for the following ring of blast holes. Successful blasting of the cut section is critical to the success of the whole round. Because the cut holes initially have only one direction in which to expand, the specific charge in the cut is considerable higher than in the rest of the tunnel and can even exceed 10kg/m^3. Most stoping holes (especially in large tunnels) have a large expansion area. These holes are considered close to surface blasting holes for charging calculations. The same explosive, normally ANFO, is used for stope hole charging as in the cut area. Development of explosives has moved in the direction of products with better fumes such as emulsion explosives. Lightened explosives or special smooth blasting explosives are used for smooth blasting. Initiating systems like NONEL decrease charging time and add further safety to the blasting operation because it is insensitive to electrical hazards. Contour holes should be blasted almost last with detonating cord or with the same detonating number. It is important to blast each smooth blasting section (walls or roof) simultaneously to achieve a smooth and even surface. Electronic detonators will perhaps become the detonators of future in tunneling, too, due to increased timing precision. Bottom holes are blasted last right before the bottom corner holes. This lifts the loosened rock pile a little, which makes mucking easier. The specific charge and specific drilling can become quite high in small tunnels due to the restricted free space available.

The blasting sequence in a tunnel or drift always starts from the "cut", a pattern of holes at or close to the center of the face, designed to provide the ideal line of deformation. The placement, arrangement and drilling accuracy of the cut is crucial for successful blasting in tunneling. A wide variety of cut types have been used in mining and construction, but basically they fall into two categories: cuts based on parallel holes, and cuts that use holes drilled at certain angles. The most common types of cut today is the Parallel and V-cut (Figure 5). The V-cut is the older of the two and is still widely used in construction. It is an effective type of cut for tunnels with a fairly large cross-section and requires fewer holes than a parallel cut. The parallel cut was introduced when the first mechanized drilling machines came on the market making accurate parallel drilling possible.

All cut holes in the large hole cut are drilled parallel to each other and the blasting is carried out towards an empty large drill hole which acts as an opening. The parallel hole cut is a development of the burn cut, where all the holes are parallel and normally of the same diameter. One hole in the middle is given a heavy charge and the four holes around it are left uncharged, in other cases the middle hole is left uncharged and the four holes are charged. However, the burn cuts generally result in less advance than the large hole cuts. The burn cut will therefore be disregarded and only the large hole cuts will be dealt with. The cut may be placed at any location on the tunnel face, but the location of the cut influences the throw, the explosives, consumption and generally the number of holes in the round.

If the cut is placed close to wall, there is a probability of better exploitation of the drilling pattern with less holes in the round. Furthermore, the cut may be placed alternatively on the right or left side thus placing the cut in relatively undisturbed rock. To obtain good forward movement and centering of the muckpile, the cut may be placed approximately in the middle of the cross section and quite low down. This position will give less throw and less explosives' consumption because of more stoping downwards. A high position of the cut gives and easily loaded muckpile, but higher explosives' consumption and normally more drilling due to upwards stoping.

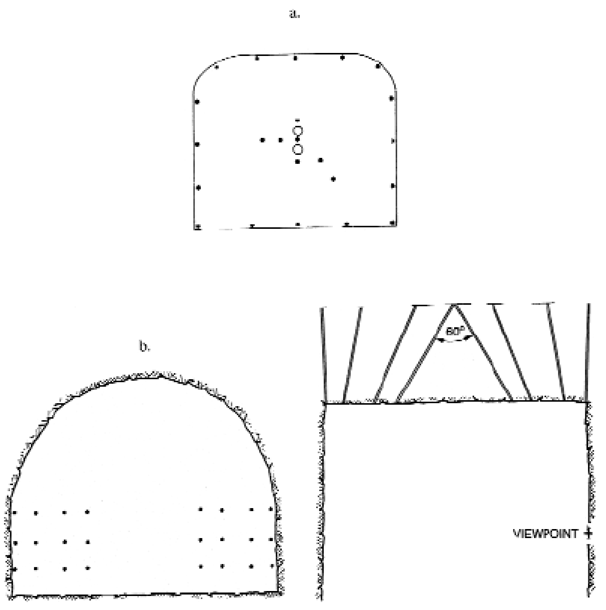

Figure 5 *The parallel cut (a) and V cut (b).*

8.1. Parallel Cut

The parallel cut has a large number of minor variations, however the basic layout always involves one or several uncharged holes drilled at or very near the center of the cut, providing empty space for the adjacent blasted holes to swell into. Uncharged cut holes are typically large, 76-127 mm in diameter. A less common alternative is to use "small hole" openings (several small holes instead of one or two large holes). Small hole opening make it possible to use the same bit size throughout the whole drilling pattern. Experience proves that big hole openings give more reliable results than small hole openings. To successfully blast a full round, the cut must be drilled, charged and blasted correctly. Cut holes are drilled very near to each other, as parallel as possible, as shown in figure 6. Specific drilling in the cut section may rise above 10drm/m^3. Apart from the large cut holes, other holes in the cut are the same size as the stope (face) holes. Large cut holes are normally drilled by reaming. First, a smaller, for example, 45 mm diameter hole is drilled then reamed to the final size using a pilot adapter and a reaming bit. Drilling holes several meters long as close together as possible demands great accuracy, but the advanced boom design and automated functions of modern drill jumbos make this quite easy. The parallel cut is especially suitable for modern mechanized tunneling equipment. This cut type has also made long rounds common in small tunnels. An earlier version of the parallel cut is the "burn cut" which does not use uncharged holes, relying instead on a very strong charge to burn the rock. Today, the parallel cut has replaced the burn cut.

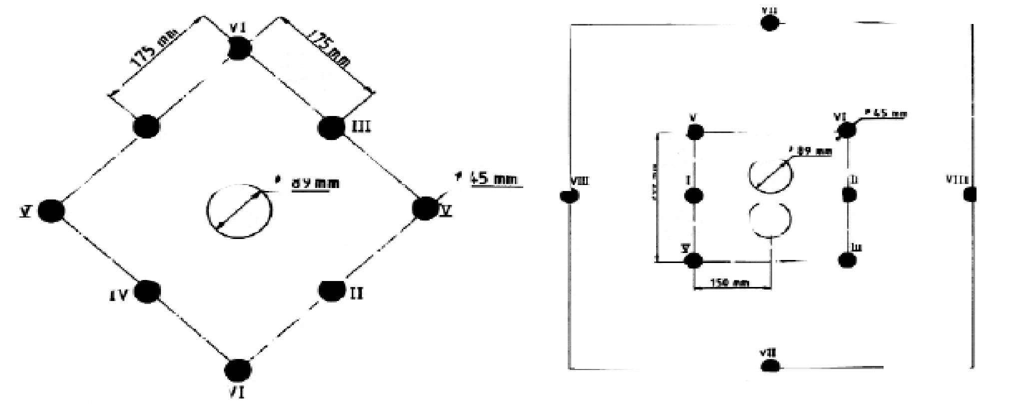

Figure 6 Typical 2m x 2m cut hole arrangement

8.1.1. Purpose of cut holes

In the parallel cut, the cut holes provide enough expansion space for the remaining blasted rock around it. The face area of a typical parallel cut varies from 1.6m x 1.6m to 2.5m x 2.5m. The right size is determined according to area of the tunnel face. Big, uncharged cut holes (76-127mm dia.) provide an opening for the blasted, expanding rock from the surrounding cut holes. All holes are drilled very close to each other and detonated each with its own detonation number (figure 6). The main idea is for each hole to loosen the rock in the front of it, allowing it to expand and fill the available open space. Cut holes are quite heavily charged and the blasted cut becomes a square opening. Basically, only drilling errors limit the gained advance per hole length.

When designing the cut, the following parameters are important for a good result:

- Diameter of large hole
- Burden
- Charge conditions

Additionally, drilling precision is of utmost importance, especially for the blast holes closest to the large hole (holes). The slightest deviation can cause the blasthole to meet the large hole or make the burden too big. An exceedingly big burden causes breakage or plastic deformation in the cut, resulting in a short advance. A parameter for good advance of the blasted round is the diameter of the large empty hole. The larger the diameter, the deeper the round can be drilled and a greater advance expected. One of the most common causes of short advance is an overlay small hole in relation to the hole depth. An advance of approx. 90% can be expected for a hole depth of 4 m and one empty hole 102 mm in diameter. If several empty holes are used, a fictitious diameter must be calculated. The fictitious diameter of the opening may be calculated by the following formula:

$$D = d\sqrt{n}$$

where D = Fictitious empty large hole diameter
d = Diameter of empty large holes
n = Number of holes

In order to calculate the burden in the first square, the diameter of the large hole is used in one large hole and fictitious diameter in several large holes.

8.1.1.1. The firing pattern

The firing pattern must be designed so that each hole has free breakage. The angle of breakage is smallest in the cut area where it is around 50°. In the stoping area the firing pattern should be designed so that the angle of breakage does not fall below 90° (Figure 7). It is important in tunnel blasting to have a long enough time delay between the holes. In the cut area, it must be long enough to allow time for breakage and rock throw through the narrow empty hole. It has been proven that the rock moves with a velocity of 40 - 70 meters per second. A cut drilled to a depth of 4 - 5 m would therefore require a delay time of 60 -100 ms to be clean blasted. Normally delay times of 75 - 100 ms are used. In the first two squares of the cut, only one detonator for each delay should be used. In the following 2 squares, two detonators may be used. In the stoping area, the delay must be long enough for the rock movement.

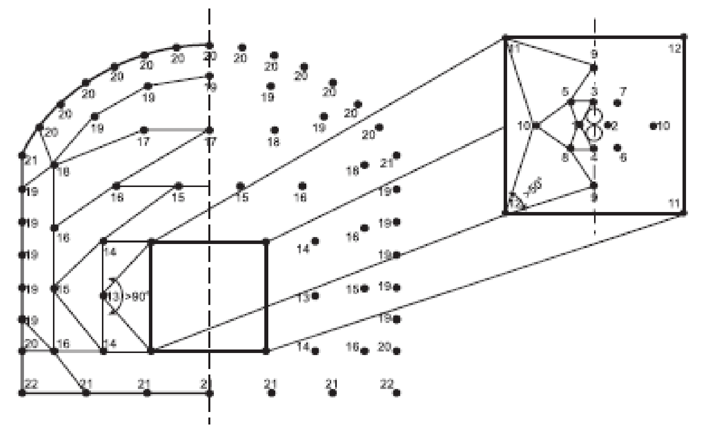

Figure 7 . Firing sequence for tunnel in numerical order.

Normally, the delay time is 100 - 500 milliseconds. For contour holes, the scatter in delay between the holes should be as little as possible to obtain a good smooth blasting effect. Therefore, the roof should be blasted with same interval number, normally the second highest of the series. The walls are also blasted with the same period number but with one delay lower than that of the roof. Detonators for tunneling can be electric or non-electric. Contour holes should be initiating with detonating cord or with electronic detonators to obtain the best smooth blasting effect.

8.1.2. Calculation of the 1st square

The distance between the blasthole and the large empty hole should not be greater than 1.5ø for the opening to be clean blasted. If longer, there is merely breakage and if shorter, there is a great risk that the blasthole and empty hole will meet. Therefore, the position of the blastholes in the 1st square is expressed as:

a = 1.5ø

Where a = C - C distance between large hole and blasthole
ø = Diameter of large hole

In the cases of several large holes, the relation is expressed as:

a = 1.5 D

where a = C - C distance between the center point of the large holes and the blasthole
D = Fictitious diameter

8.1.3. Charging of the holes in the 1st square

The holes closest to the empty hole(s) must be charged carefully. An insufficient charge concentration in the hole may not break the rock, while an excess charge concentration may throw the rock against the opposite wall of the large hole with such high velocity that the broken rock will be re-compacted and not blown out through the large hole. In this case, full advance is not obtained. The required charge concentration for different C-C distances between the large hole and nearest blasthole(s) can be found in figure 8. The normal relation for the distance is a = 1.5 ø. An increase in the C-C distance between holes causes subsequent increment of the charge concentration. The cut is often somewhat overcharged to compensate for drilling errors which may cause insufficient breakage angle. However, excess charge concentration causes re-compaction in the cut.

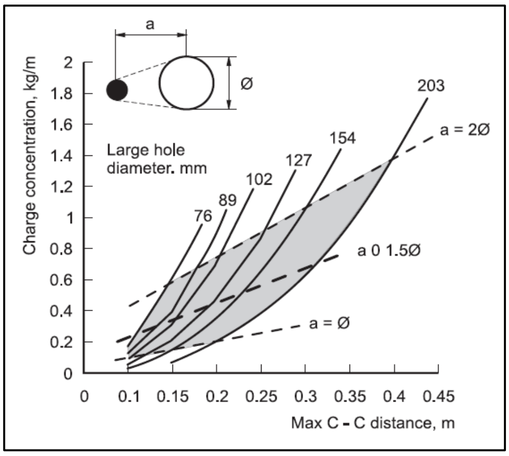

Figure 8 *Minimum required charge concentration (kg/m) and maximum C - C distance (m) for different large hole diameters.*

8.1.4. Calculating the remaining squares of the cut

The calculation method for the remaining squares of the cut is essentially the same as for the 1st square, but differs in that breakage is towards a rectangular opening instead of a circular opening. As is the case of the 1st square, the breakage angle must not be too acute as small angles of breakage can only be compensated to a certain extent with higher charge concentration. Normally, the burden (B) for the remaining squares of the cut is equal to the width (W) of the opening. B = W. The charge concentration obtained in figure 9. is that of the column of the hole. In order to break the constricted bottom part, a bottom charge with twice the charge concentration and a height of 1.5 • B should be used. The stemming part of the hole has a length of 0.5 • B.

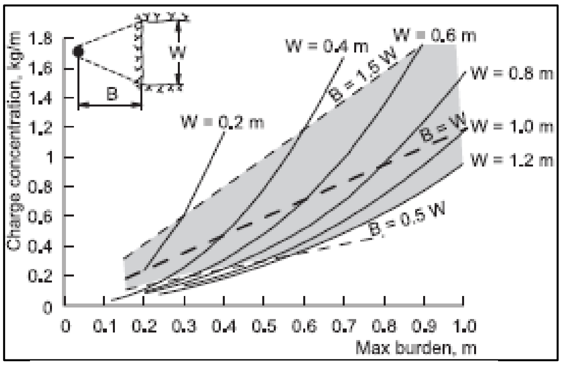

Figure 9 *The required minimum charge concentration (kg/m) and maximum burden (m) for different widths of the opening.*

8.2. V-cut

The V cut is a traditional cut based on symmetrically drilled, angled holes. It has lost some of its popularity with the widespread adoption of the parallel cut and longer rounds. However, it is still commonly used in wide tunnels where tunnel width sets no limitations on drilling. The working principle of the V cut is similar to surface excavation applications. The V cut requires slightly fewer hole meters than the parallel cut, which gives it an advantage in large tunnels. The V cut is based on surface blasting principles in which the angle for rock expansion equals or exceeds 90 degrees. The angle at the bottom of the cut holes should not be less than 60 degrees. Maintaining the right angle is the main difficulty in V-cut drilling; and, the correct drilling angle limits round length in narrow tunnels (figure 10a.). Tunnel width limits the use of the V cut. In narrow tunnels, the advance per round can be less the one third of the tunnel width, which increases the number of rounds and the amount of drilled meters when excavating small tunnels. V-cuts are easily drilled with mechanized rigs in large tunnels where tunnel width sets no limitations. The cut normally consists of two Vs but in deeper rounds the cut may consists triple or quadruple Vs. Each V in the cut should be fired with the same interval number by MS detonators to ensure coordination between the blastholes in regard to breakage. As each V is blasted as an entity one after the other, the delay between the different Vs should be in the order of 50 ms to allow time for displacement and swelling.

8.3. The fan cut

The fan cut (figure 10b) is another example of angled cuts. Like the V-cut, a certain tunnel width is required to accommodate the drilling equipment to attain acceptable advance per round. The principle of the fan is to make a trench-like opening across the tunnel and the charge calculations are similar to those in opening the bench. Due to the geometrical design of the cut, the hole construction is not large, making the cut easy to blast. Hole drilling and charging is similar to that of cut holes in the V-cut.

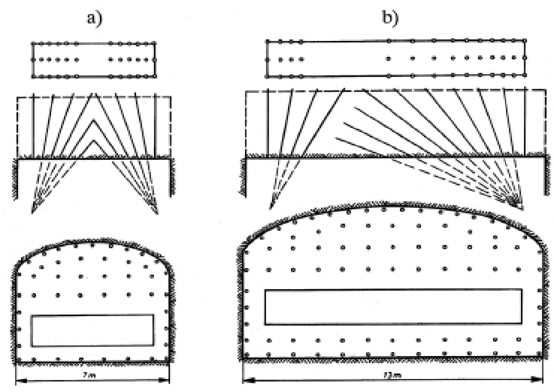

Figure 10 V cut (a) and Fan cut (b).

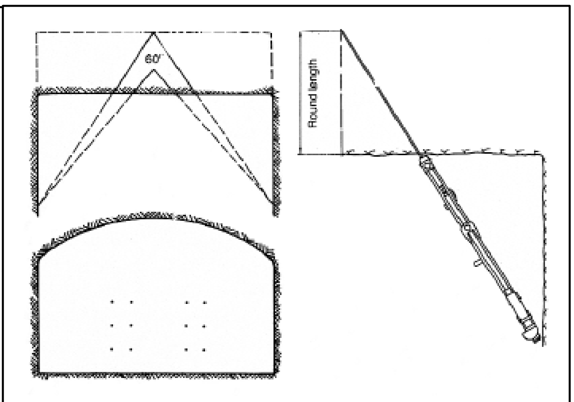

Figure 10 Feed set-up and drilling limitations in V cut.

9. TUNNEL EXCAVATION METHOD

Most tunnels are advanced using full-face excavation. The entire tunnel face is drilled and blasted in one round. Blastholes are usually drilled to a depth somewhat shorter than the dimension of the opening, and the blast "pulls" a round a little shorter (about 90 percent with good blasting practice) than the length of the blastholes. The depth pulled by typical rounds are 2 to 4 m (7-13 ft) in depth. Partial-face blasting is sometimes more practical or may be required by ground conditions or equipment limitations. The most common method of partial-face blasting is the heading-and-bench method, where the top part of the tunnel is blasted first, at full width, followed by blasting of the remaining bench. The bench can be excavated using horizontal holes or using vertical holes similar to quarry blasting. There are many other variations of partial-face blasting, such as a center crown drift, followed by two crown side drifts, then by the bench in one, two or three stages. Reasons for choosing partial as opposed to full-face blasting include the following:

- The cross section is too large for one drill jumbo for example: Underground openings of the sizes usually required for powerhouses, valve chambers, and two or three-lane highway tunnels are usually excavated using partial-face blasting excavation.
- The size of blast in terms of weight of explosives must be limited for vibration control. The ground is so poor that the full width of excavation may not be stable long enough to permit installation of initial ground support.

The design of the drilling pattern for tunnels should correlate with tunnel shape and size. The cut is normally placed vertically in the middle or side section, and horizontally on or slightly under the center line of the tunnel. The exact place is often left or right of the tunnel's center point and varies with each round (Figure 11). Sometimes the tunnel is excavated in several sections, such as a top heading, followed by benching with lifters. The top is excavated as described above, but benching with lifters only requires stope holes since the excavated top heading acts as the "cut". It is also possible that the opening for the blast or the cut section has been produced earlier by other means such as the full profile method (tunnel boring). In such cases, cuts are not required and the remaining excavation holes are drilled as stope holes. It is recommended that ditches and drains be excavated at the same time as the tunnel face but sometimes their design is more complicated and they must be excavated separately.

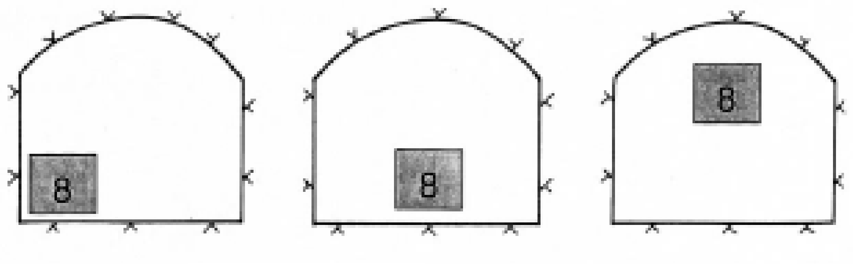

Figure 11 Various cut locations

10. TUNNELLING ACCURACY

Accurate tunneling, and accurate drilling and charging go hand-in-hand. The following topics need to be planned in advance to ensure accurate tunnel profile:

- Known geological and rock mechanical conditions
- Planned drilling pattern/patterns, correct hole size and hole length for the planned excavation (Figure 12)
- Smooth blasting (contour blasting) procedure
- Correct rig set-up
- Correct hole alignment and look-out angle, with special consideration for the walls, roof and floor
- Cut placement; inclined and curved tunnels are especially prone to under and

overbreak in the walls and roof and "bumps" in the tunnel floor
- Accurate charging, the correct detonators and drilling pattern
- Continuos follow-up procedures

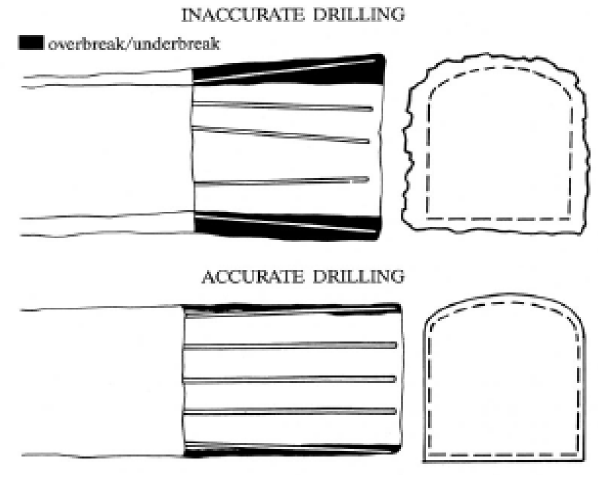

Figure 12 Cumulative drilling errors in drifting and tunneling.

10.1. Controlled Blasting

The ideal blast results in a minimum of damage to the rock that remains and a minimum of overbreak. This is achieved by controlled blasting. The required amount of shotcreting and concrete casting can be significantly reduced by using smooth blasting, particularly in poor rock conditions. Smooth blasting increases the number of holes needed for the drilling pattern by roughly 10 - 15%. Rock hardness is occasionally incorrectly considered to be the only dominant factor when optimizing the drilling pattern. The change from very hard rock to soft rock therefore causes a change in the drilling pattern. Rocks that are hard but abrasive are fairly easily blasted, where as the blastability of rocks such as some limestone, although relatively soft, is poor.

Control of rock damage and overbreak is advantageous for many reasons:

- Less rock damage means greater stability and less ground support required.
- The tunneling operations will also be safer since less scaling is required.
- Less overbreak makes a smoother hydraulic surface for an unlined tunnel.
- For a lined tunnel, less overbreak means less concrete to fill the excess voids.

Controlled blasting involves a closer spacing of the contour or trim holes, which are loaded lighter than the remainder of the holes. A rule often used is to space contour holes about 12-15 times hole diameter in competent rock, and 6-8 times hole diameter in poor, fractured rock. Because controlled blasting generally requires more blast holes than otherwise might be required, it takes longer to execute and uses more drill steel. For these reasons, contractors are often reluctant to employ the principles of controlled blasting. But controlled blasting requires more than just the design of proper perimeter blasting. Blast damage can occur long before the trim holes are detonated. Controlled blasting requires careful design and selection of all aspects of the round-geometry, hole diameter, hole charges, hole spacings and burdens, and delays as well as careful execution of the work. One of the keys to successful controlled blasting is precise drilling of blast holes. Deviations of blastholes from their design locations quickly lead to altered spacings and burdens, causing blast damage and irregular surfaces. Modern hydraulic drills are not only quick but also permit better precision than was the norm. The highest precision is obtained with the use of computer-controlled drill jumbos in homogeneous rock. Inspection of the blasted surfaces after the blast can give good clues to the accuracy of drilling and the effectiveness with which blasting control is achieved. A measure of success is the half-cast factor. This is the ratio of half casts of blast holes visible on the blasted surface to the total length of trim holes. Depending on the quality of the rock and the inclination of bedding or jointing, a half-cast factor of 50 to 80 percent can usually be achieved. Irregularities in the surface caused by imprecise drilling are also readily visible and measurable. The regularity and appropriateness of the lookout should also be verified. Other means to verify the quality of blasting include methods to assess the depth of blast damage behind the wall. This may be done using seismic refraction techniques and borescope or permeability measurements in cored boreholes. The depth of the disturbed zone can vary from as little as 0.1-0.2 m (4-8 in.) with excellent controlled blasting to more than 2 m (7 ft) with uncontrolled blasting.

10.2. Damage Control

A common misconception is that the only step required to control blasting damage is to introduce pre-splitting or smooth blasting techniques. These blasting methods, which involve the simultaneous detonation of a row of closely spaced, lightly charged holes, are designed to create a clean separation surface between the rock to be blasted and the rock which is to remain. When correctly performed, these blasts can produce very clean faces with a minimum of overbreak and disturbance. However, controlling blasting damage starts long before the introduction of pre-splitting or smooth blasting. As pointed out earlier, a poorly designed blast can induce cracks several metres behind the last row of blastholes. Clearly, if such damage has already been inflicted on the rock, it is far too late to attempt to remedy the situation by using smooth blasting to trim the last few metres of excavation. On the other hand, if the entire blast has been correctly designed and executed, smooth blasting can be very beneficial in trimming the final excavation face.

The correct design of a blast starts with the very first hole to be detonated. In the case of a tunnel blast, the first requirement is to create a void into which rock broken by the blast can expand. This is generally achieved by a wedge or burn cut which is designed to create a clean void and to eject the rock originally contained in this void clear of the tunnel face. In today's drill and blast tunnelling in which multi-boom drilling machines are used, *the most convenient method for creating the initial void is the burn cut.* This involves drilling a pattern of carefully spaced parallel holes which are then charged with powerful explosive and detonated sequentially using millisecond delays. Once a void has been created for the full length of the intended blast depth or 'pull', the next step is to break the rock progressively into this void. This is generally achieved by sequentially detonating carefully spaced parallel holes, using one-half second delays. The purpose of using such long delays is to ensure that the rock broken by each successive blasthole has sufficient time to detach from the surrounding rock and to be ejected into the tunnel, leaving the necessary void into which the next blast will break. A final step is to use a smooth blast in which lightly charged perimeter holes are detonated simultaneously in order to peel off the remaining half to one metre of rock, leaving a clean excavation surface. Since a pre-split blast carried out under these circumstances has to take place in almost completely undisturbed rock which may also be subjected to relatively high induced stresses, the chances of creating a clean break line are not very good. The cracks, which should run cleanly from one hole to the next, will frequently veer off in the direction of some pre-existing weakness such as foliation. For these reasons, smooth blasting is preferred to pre-split blasting for tunnelling operations.

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