CHAPTER 25

# Contour blasting

## 25.1 INTRODUCTION

The energy that is not used in the process of fragmentation and displacement of rock, sometimes more than 85 % of that developed in the blast, reduces the structural strength of the rock mass outside the theoretical radius of action of the excavation. New fractures and planes of weakness are created, and joints, diaclases and bedding planes that initially were not critical, when opened result in an overall reduction of rock mass cohesion.

This is manifested by overbreak, leaving the fractured mass in a potential state of colapse.

The negative consequences that appear are the following:

– More waste dilution of the ore in contact zones of metal mines.

– Increase in loading and haulage costs due to high volume muckpile.

– Increase in the cost of concrete for civil engineering: tunnels, hydroelectric power plants, storage chambers, foundations, walls, etc.

– The necessity to reinforce the remaining mass by use of costly support systems: bolting, meshing, scaling, metal ribbing, etc.

– The necessity to maintain and insure soundness of the remaining rock with the consequent risk for the operators.

– The inflow of water to the work area increases due to the opening and prolongation of the fractures and joints of the rock mass.

Also, in surface mining, blast control in the final slopes of the orebody could bring about the following advantages, Fig 25.1.

– Elevation of the slope angle, obtaining an increase in the recoverable ore reserves and/or a lower waste to ore ratio.

– Reduction in the risk of rockfall, lowering the necessity of wide berms and a positive repercussion on productivity and work safety.

At the same time, in underground operations, the use of contour blasts have the following advantages:

– Smaller pillar dimensions in the exploitations and, in consequence, higher recuperation of the orebody.

– Better ventilation due to less air friction along the shaft walls.

– Lower risk of damage to the advanced drilling.

Therefore, the efforts assigned to the application of contour blasting in underground and surface operations are fully justified for technical as well as for economical and safety reasons.

## 25.2 MECHANISMS RESPONSABLE FOR OVERBREAK

The mechanisms that are responsable for the phenomena of overbreak and fracturing of the rock mass are closely related to those of rock breakage that develop during the blast and were exposed in Chapter 16.

In order to control overbreak the following should be taken into consideration:

– The dynamic compressive strength of the rock that surrounds the explosive charge should not be surpassed.

– Maintain a vibration level in the rock mass that does not generate breakage from spalling.

– Use of explosives that are adequate for the type of rock in order to avoid the opening of fissures from an excessive volume of gases.

In the following, the principal mechanisms of breakage that cause overbreak will be briefly analyzed.

### 25.2.1 *Breakage by crushing and cracking*

An annular zone of crushed or permanently compressed material is often formed immediately around the blasthole wall. This occurs where the peak of the cylindrically expanding radial compressive wave exceeds the dynamic compressive breaking strain or the plastic yield of the material.

When the strain wave passes, a cylindrical shell of rock around the blasthole is subjected to intense radial compression and tangential strains develop. If these strains exceed the dynamic tensile breaking strain of the rock, a zone of dense radial fractures is formed. This zone terminates quite abruptly at that radial distance where the wave's tangential strain attenuates to a value which is incapable of generating new cracks.

### 25.2.2 *Breakage by spalling*

Where the compressive strain wave strikes an effective free face, a reflected tensile wave is created. If this reflected wave is sufficiently strong, spalling or slabbing occurs progressively from the effective free face back towards the blasthole.

The level of vibrations produced in a blast can also

Photo 25.1. Marked difference between a slope with presplitting blast (right) and conventional blasting (left).

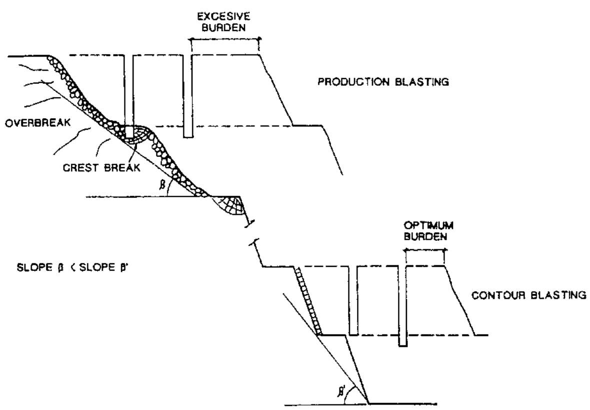



Fig. 25.1. Pit slope damage from production blasting and suggested contour blasting.

Table 25.1.

| Types of rock/joints | Peak particle velocity (mm/sec) |
|---|---|
| Soft rock, weak joints | 400 |
| Medium to hard rock and weak joints | 700-800 |
| Hard rock and strong, closed joints | 1000 |

provoke breakage by spalling if the stresses produced overcome the dynamic tensile strength of the rock:

$$\sigma = v \times \frac{E}{VC} = \rho_r \times v \times VC$$

where: $\sigma$ = Induced stress in the rock, $v$ = Peak particle velocity, $E$ = The Young's modulus of elasticity, $VC$ = Longitudinal wave or seismic velocity, $\rho_r$ = Density of the rock.

The influence of the type of filling in the joints and discontinuity planes must be taken into account, establishing the strain values of the wave that is transmitted and reflected.

$$\sigma_t = 2\frac{\sigma_i}{1 + n_z'}$$

$$\sigma_r = \sigma_i \frac{1 - n_z'}{1 + n_z'}$$

where: $n_z'$ = Ratio of rock and filling impedance, $\sigma_i$ = Tensile strength of the incident wave, $\sigma_t$ = Tensile strength of the transmitted wave, $\sigma_r$ = Tensile strength of the reflected wave.

The calculation of the peak particle velocity can be done from the equation:

$$v_{\text{crit}} = \frac{RT'}{\rho_r \times VC}$$

Orientative figures are given for peak particle velocity in different types of rock, Table 25.1.

### 25.2.3 *The opening of cracks by the gases*

The action of the gases at high pressure and temperature, opening the preexisting fractures and those created by the strain wave, can very strongly affect the control of overbreak, which means that in soft and very fractured rocks the explosives used should produce a small volume of gases.

## 25.3 THE THEORY OF CONTOUR BLASTING

A charge that completely fills a blasthole creates, during detonation and in the near vecinity, a zone in which the dynamic tensile breaking strength is amply overcome and the rock is triturated and pulverized. Outside this transition zone, the tensile force associated with the stress wave creates a pattern of radial cracks surrounding the blasthole.

When two charges are fired simultaneously, the radial cracks tend to propagate equally in all directions until the two strain wave collide in the central point between the two holes creating complementary tensile forces that are

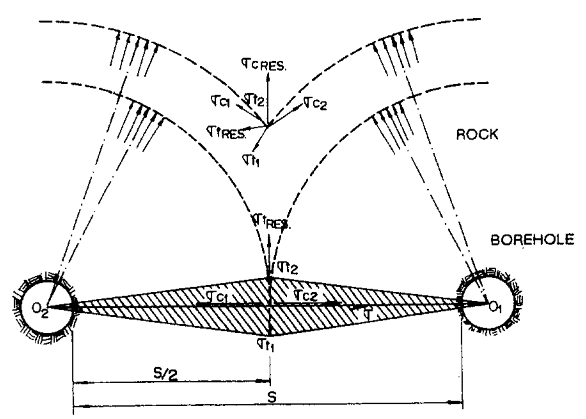



Fig. 25.2. Stresses generated by the collision of the shock waves produced by the simultaneous firing of two charges.

perpendicular to the axial plane, Fig. 25.2. The tensile stresses exceed the dynamic tensile breaking strength of the rock producing a new cracking and favoring, in the projected direction of the cut, the propagation of the radial cracks.

In the last stage, the extension of the cracks is induced by the wedging effect of the gases that invade and infiltrate them. The preferential propagation in the axial plane along with the opening effect of explosion gases allow the obtainment of a fracture plane in accord with the designed cut.

The pressure of the gases is a key element in the execution of a contour blasting, and for this reason it should be maintained until the cracks of adjacent blastholes meet, which can be achieved by adequate stemming height to avoid escape of the gases into the atmosphere.

It can then be concluded that the mechanism of a contour blast is made up of two different phenomena: one derived from the shock wave action and the other from the effect of the explosive gases, but between both a causal nexus is kept.

## 25.4 TYPES OF CONTOUR BLASTS

There are many different techniques for contour blasting that have been developped from the fifties, but nowadays the most commonly used are:

- Pre-splitting
- Cushion blasting
- Buffer blasting and
- Line-drilling.

### 25.4.1 *Pre-splitting blasts*

This consists in creating a joint or a fracture plane in the rock mass before firing the production blastings, by means of a row of blastholes, usually of small diameter, and with decoupled explosive charges.

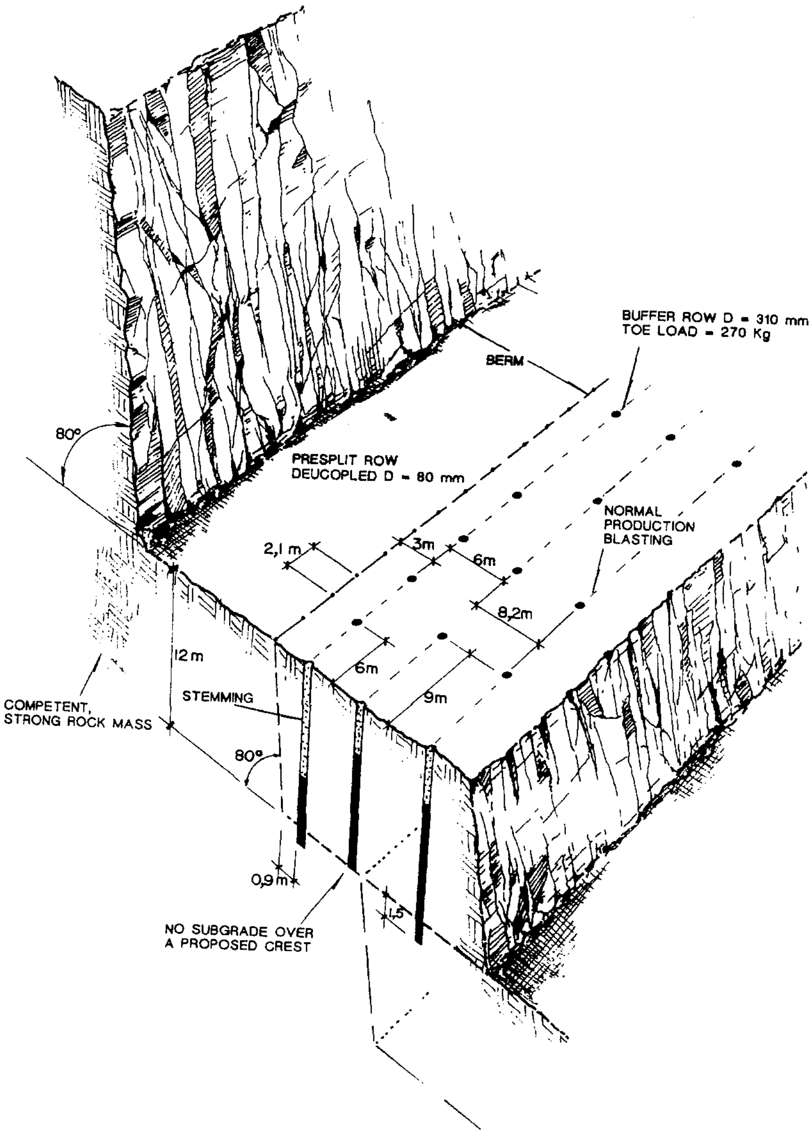



Fig. 25.3. A typical pre-split blast design.

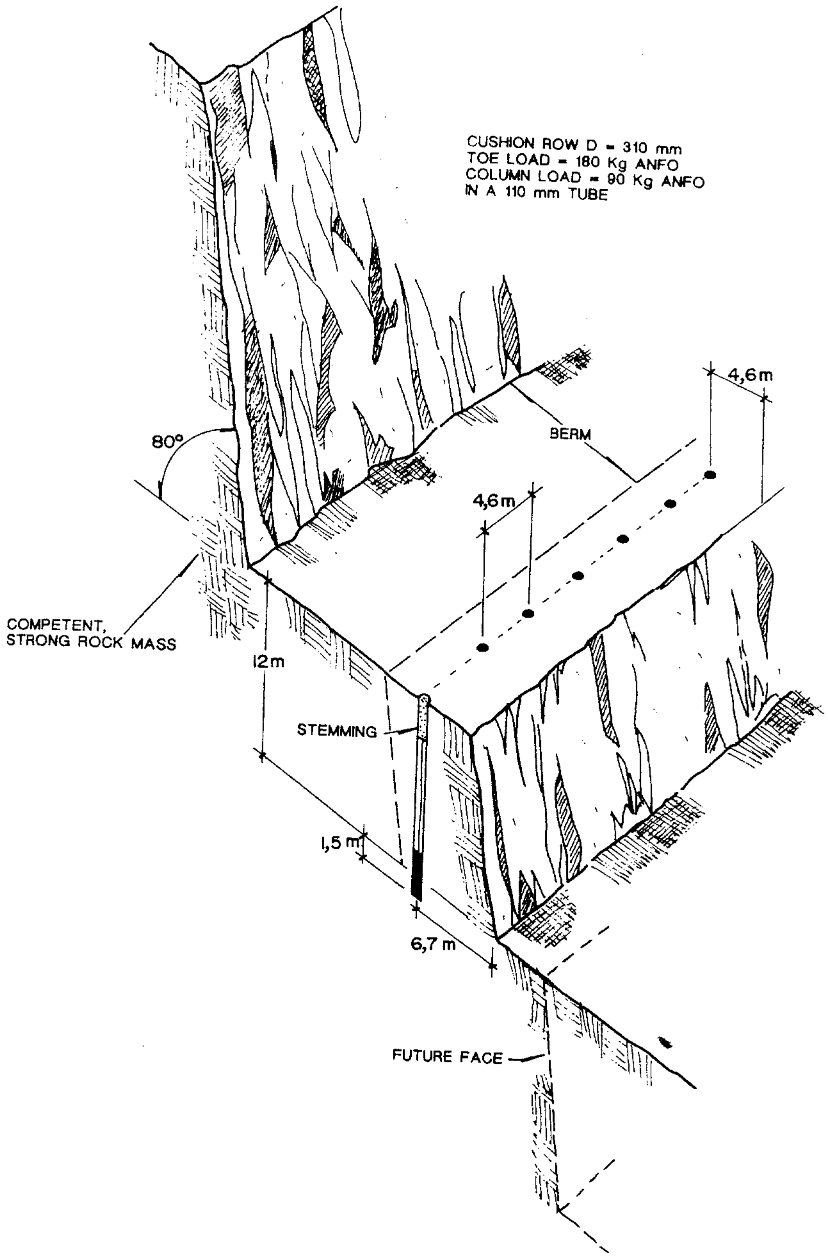



Fig. 25.4. A typical cushion blast design.

The firing of the pre-splitting holes can be done simultaneously with the stope holes, but the detonation should be from 90 to 120 ms in advance, Fig. 25.3.

### 25.4.2 *Cushion blasting*

This consists of a single-row blast with decoupled and/or decked charges.

The technique involves rock breakage towards the free face, in which the spacing between the charges is greater than in the previous method, lowering costs, Fig. 25.4.

In surface mining, when the cushion blastholes have the same diameter as those for production, the technique is called *Trim blasting.*

### 25.4.3 *Buffer blasting*

These are blasts similar to conventional blastings where the design of the last row has been modified in its geometric pattern, which is smaller, as well as in the explosive charges which are usually smaller also and decoupled, Fig. 25.5.

### 25.4.4 Line drilling

Line drilling is a breakage technique that uses empty blastholes of 35 to 75 mm in diameter which are separated from each other by 2 to 4 times the diameter. As the holes are so close to each other, they can perform in adequate geological conditions as stress concentrators or

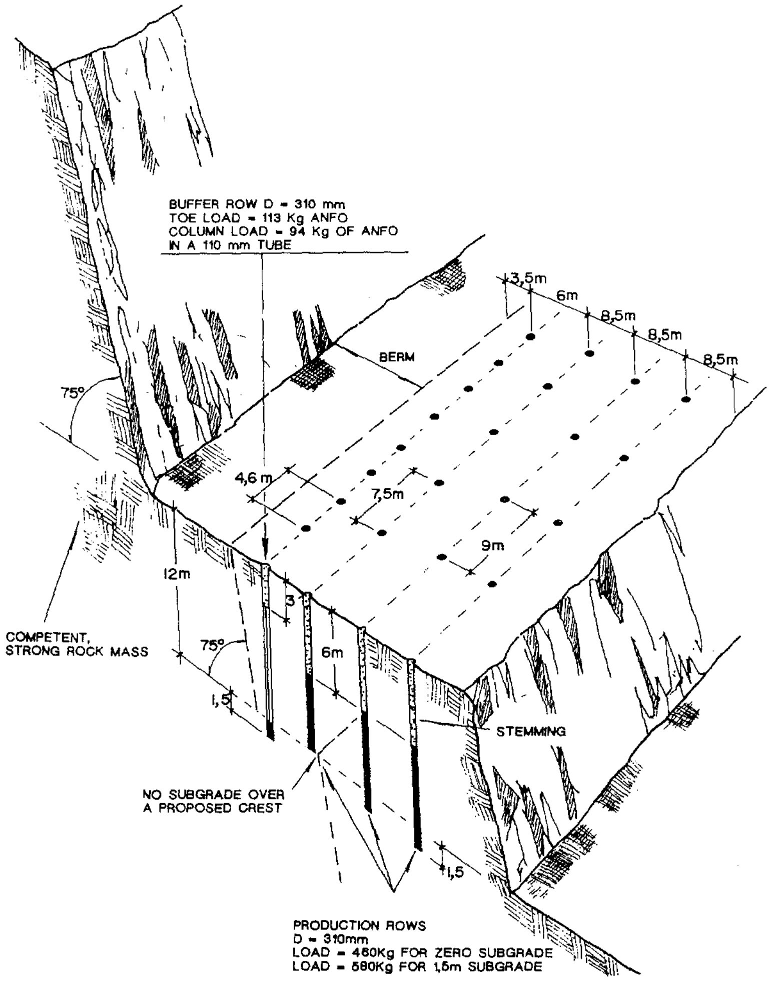



Fig. 25.5. A typical buffer blast design.

crack guides to create a plane of weakness between them.

Drilling precision is very important to obtain good results, as well as the homogeneity of the rock because, if not, the natural fissures of the rock mass tend to create plane of weakness more easily than that caused by the drilled holes.

The production blastings should be *cushion type* and the burdens and spacings in the row closest to the empty holes should be between 50 to 75 % of the conventional.

The charges should also be reduced to 50 % of the conventional.

The main advantage of this technique is its application when very small charges can cause damage behind the excavation line. On the other hand, the disadvantages are the uncertain results in heterogenous rock, the high cost, the time used in drilling and the necessity of precision.

## 25.5 THE PARAMETERS THAT INTERVENE IN A CONTOUR BLASTING

### 25.5.1 Properties of the rocks and the mass.

The rock mass properties have a marked influence on the design as well as on the results of the contour blasts. The most prominent properties are:

– The dynamic tensile and compressive strengths.

– The degree of weathering of the rock.

– Degree of fissurization, spacing between discontinuities, orientation of the fissures and their filling.

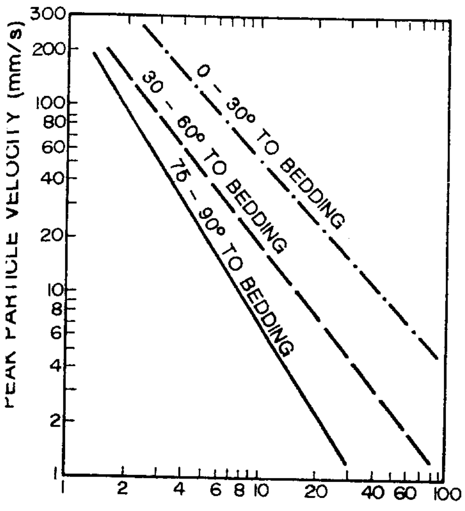
– Residual tensions of the rock mass.

Some of the practical aspects that should be taken into account are:

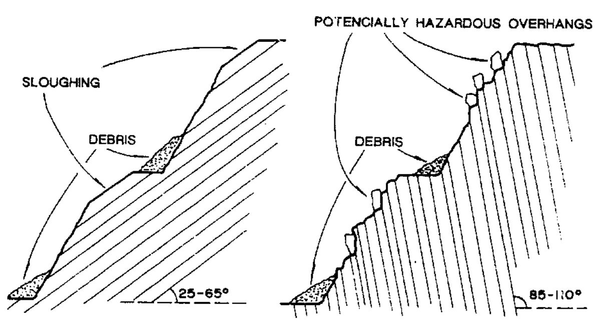
– In homogenous rock masses, the results of contour blasts can be spectacular. On the other hand, in fissured masses it has been observed that the cracking induced when the dynamic tensile strength of the rock is overcome represents a minimum percentage of overbreak, when compared to the damage produced by the wedging action of the gases.

– If the blastholes cut through any system of discontinuities and the induced strains are not sufficient to form a distribution of radial cracks, the breakage surface will be markedly influenced by the natural fissures and overbreak is more likely to occur.

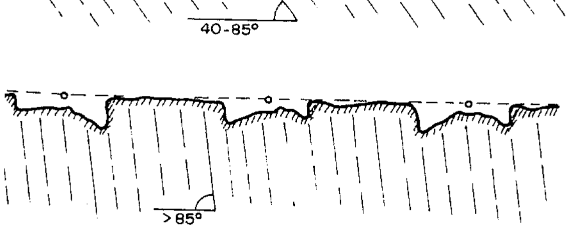
In this case, it is recommended that the charge concentration be slightly increased to generate a mass of small radial cracks and obtain, with some of these, the orientation of the plane of the cut. If the natural joints intersect




ig. 25.6. Influence of the angles formed by the stratification upon the ıws of vibration propagation.




ig. 25.7. Influence of jointing on overbreak in open pit benches Iagan, 1977).




g. 25.8. Influence of the angles formed by stratification upon the ʌws of vibration propagation.

longitudinally the walls of the blastholes, this modification will be useless.

– Tight or infilled joints in the rock mass result in lessbackbreak.

– The spacial distribution of the fissures has great influence on backbreak, especially when the mean distance between joints is less than that of the blastholes and/or stemming height. In this case it recommended that the pattern be closed in order to reduce the effect of structural control.

– Depending upon the orientation of the projected cut, with respect to the predominating structural discontinuities, the cases presented in Fig. 25.6 can be differentiated.

– In stratified formations, in which the direction of the joint planes coincide with the outline of the projected slope, overbreak can result due to plane failure if the inclination of the stratas varies between 25 and 65° and due to toppling failures if it is between 85 and 110°, Fig. 25.7. When the joints are parallel to the slope plane, a clean, smooth face is achieved with relative ease.

– The presence of water in the blastholes can reduce the efficiency of decoupling the charges as it transmits a larger strain force to the surrounding rock.

– The solution cavities of the ground that are intersected by drills provoke a loss of blasthole pressure that can affect the success of the excavation. In these circumstances, it is recommended that the holes be filled with granular material and even slightly increase the charge density.

– The angle formed by the direction of wave propagation with respect to the stratification, has influence upon the laws of vibration propagation generated in the blasts and transmitted through the rock mass, Fig. 25.8.

– The in-situ strains of the rock mass where contour blasting is planned can make presplitting unfeasible as the blasthole pressure needed to overcome such stresses would have to be very high. One solution would be to carry out a cushion blast, once part of the excavation is accomplished, which would serve to decompress and release strains in the rock mass, Fig. 25.9.

### 25.5.2 *Characteristics of the explosive*

The blasthole pressure, which is the pressure exerted by the expanding gases from the explosion, can be calculated for coupled charges from the following equation:

$$PB = 228 \times 10^{-6} \times \rho_e \times \frac{VD^2}{1 + 0.8\rho_e}$$

where: $PB$ = Borehole pressure of a fully coupled charge completely filling the blasthole (MPa), $\rho_e$ = Specific gravity or density of the explosive (g/cm$^3$), $VD$ = Velocity of detonation for a confined explosive (m/s).

In this manner, the stresses induced in the surrounding rock are proportional to $PB$. For this reason, by reducing the stress to levels that are in agreement with the rock strength, rockbreak will be diminished as well as vibration intensity.

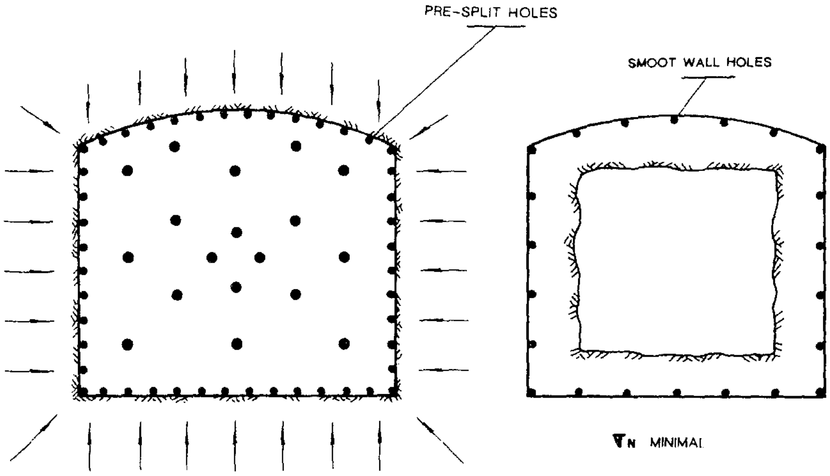



Fig. 25.9. Elimination of strains through a pilot excavation and a partial cushion blasting.

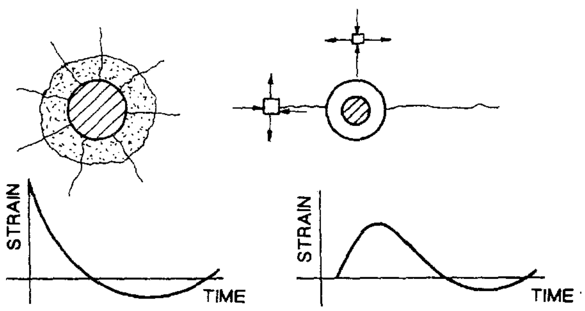



Fig. 25.10. Effect of decoupling on the stress-time curve.

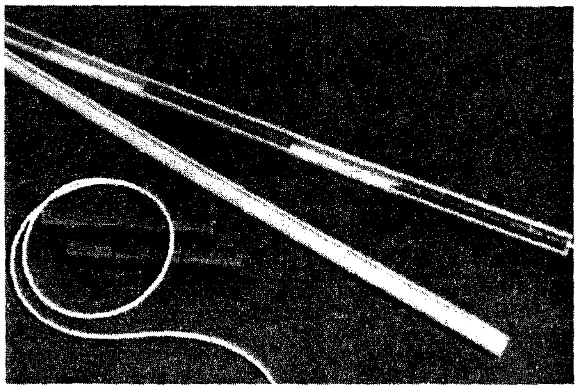


Photo 25.2. Omega tubes (ICI).

If the selection of the explosive is not sufficient to adjust to the work conditions, the engineers have many other systems on hand to voluntarily reduce blasthole pressure.

1. By incorporating inert materials to the explosive that contain air such as expanded polystyrene beads, styrofoam, woodmeal, etc. The dimishing of $\rho_e$ has a greater influence on *PB* than that indicated by the previous equation because, upon lowering the density, the detonation velocity also lowers.

2. By taking advantage of the effect of the diameter of the charge over the detonation velocity *PB* can be controlled. If the diameter of the blasthole is smaller than the peak diameter of the explosive, its detonation velocity, and consequently the blasthole pressure, can be drastically reduced.

3. By interposing a volume of air between the charge and the blasthole wall which gives a cushioning effect on *PB*. This is done by leaving an uncharged anular hole, and if this is not sufficient, the cartridges can be spaced along the length of the blasthole, Fig. 25.10.

The cushioning effect upon *PB*, when the gases expand in the air chamber, can be calculated from the quotient between the volume of explosive and the volume of blasthole elevated to a power of 1.2, which is approximately the ratio of the specific heat of the explosion gases, therefore:

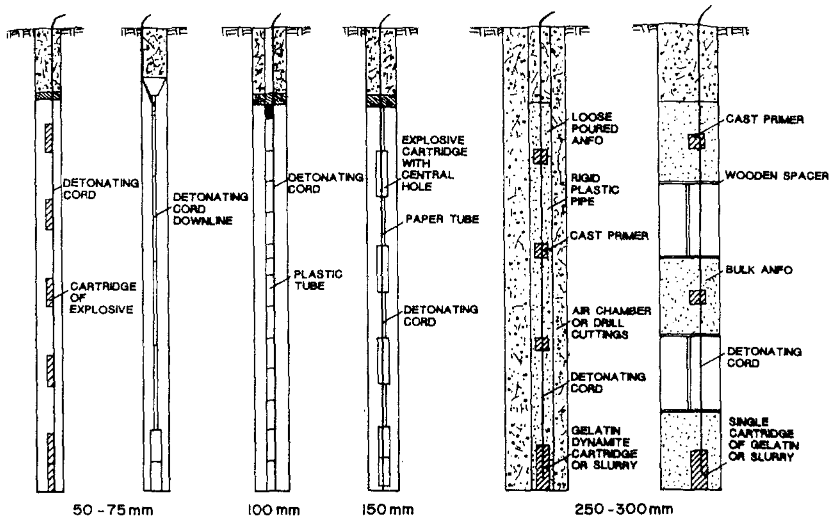



Fig. 25.11. Typical charge loads for contour blasting for blastholes of different diameters.

$$PB_e = PB \times \left[\frac{V_e}{V_b}\right]^{1.2} = PB \times \left[\sqrt{C_l}\frac{d}{D}\right]^{2.4}$$

where: $d$ = Diameter of charge, $D$ = Diameter of blasthole, $C_l$ = Percentage of explosive column that is loaded ($C_l$ = 1, for continuous charges).

### 25.5.3 *Explosives used in contour blastings*

*Conventional charges*
The first charges used for contour blastings were dynamite cartridges with a detonating cord and spaced to acheive the desired charge density. Later on, some accessories appeared on the market such as omega tubes. Photo 25.2, which facilitate the distribution of the charge.

*Special cartridges*
The manufacturers of explosives put several especially designed cartridges on the market to facilitafe and speed-up the charging of the blastholes. This way, for example, in some countries there are low density explosives placed in long tubes, reduced diameter cartridges (normally of 550 and 600 mm in length and of 11, 17 and 22 mm in diameter) that can be connected at each end, which allows the shot firer to rapidly form continuous charge columns of the desired length, Fig. 25.11. In the lower end of the column various bottom cartridges are placed and, sometimes, the whole charge load is surrounded by a detonating cord that is wrapped around in a spiral fashion.

At the moment, in Spain there are cartridged slurries in flexible plastic hoses (Riogur), with 17 and 19 mm calibers, or in rigid connecting cartridges with charge concentrations of 250 g/m and 300 g/m respectively.

The explosive hoses adapt well to surface work conditions while the rigid connecting cartridges are best adapted to underground operations.

*Detonating cords*
Recently, high core load detonating cords have appeared on the market as an alternative to the special cartridges. In Spain there are 40, 60,and 100 g of pentrite per meter: it is expected that with these a better distribution of the energy will be obtained, as they are continuous columns, and that the charging will be easier, Photo 25.3.

In the bottom of the blastholes a single cartridge of gelinite explosive or slurry should be placed.

The detonating cords have been used in demolition operations and for cutting of ornamental rocks where precise, clean cuts are a necessity, and lately with the high core load cords, in the execution of controlled excavations with drilling diameters of up to 76 and 89 mm.

*Diluted ANFO type low density mixtures*
In large diameter contour blastings, the decoupling of loose poured ANFO is achieved in a very effective way with plastic tubes or cartridges, but this is a very expensive and tedious method: on other occasions, wooden air spacers are used, Fig. 25.12. However the procedure, which is more in use each day, consists in reducing the energy generated by ANFO until the equivalent of a decoupled charge is obtained. There are three systems in practice today.

The first consists in diluting the explosive with salt up to a maximum of 20%. Salt has two functions, first as a physical diluent of the energy density and secondly as a coolant of the explosive with which the detonation velocity and the heat of the explosion are reduced. Larger percentages than indicated can provoke misfires as they increase the peak diameter and reduce initiation sensitivity. Apart from this, although the salt has no chemical reaction with ANFO, when detonating it can have an excessive cooling effect producing incomplete combustions with toxic gases. (Day and Webster, 1982).

The second, which maintains the density and is the one least used, is based upon reducing the liquid fuel to under 6 %. Therefore, while one ANFO of 94/6 develops an energy of 3780 J/g, another ANFO of 98.5/1.5 only develops 2293 J/g.

Photo 25.3. The placing of a bottom charge and the high core load cord in a contour blasthole.

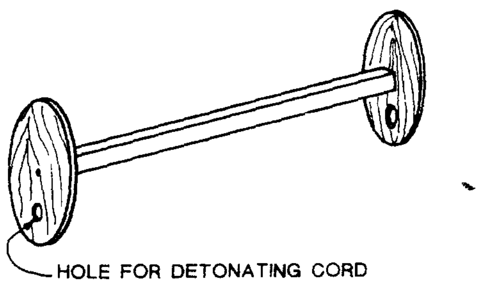



Fig. 25.12. Wooden air spacer for explosive charges in large diameter blastholes.

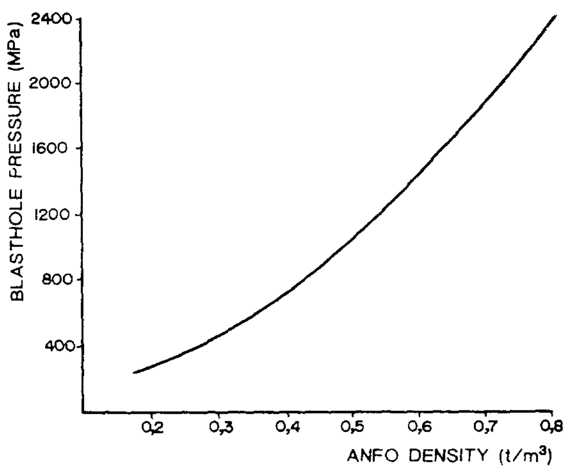



Fig. 25.13. Variation of blasthole pressure for different mixtures of low density ANFO.

The third procedure, the most popular at the moment, consists of ANFO mixed with expanded polystyrene beads from 0.5 to 3 mm, which will be called ANFOPS from now on. This dilutant, with a density of 0.03 kg/dm$^3$, has characteristics that cannot be improved with trustworthy detonation processes in large diameter blastholes, even with mixtures that contain up to 80% in polystyrene. With these explosive compounds a concentration of energy and density can be obtained per meter of up to 10% of those which correspond to pure ANFO. Therefore, for a mixture of ANFOPS with a volumetric proportion of 1:3, a density of 0.2 t/m$^3$, can be obtained, which for a blasthole of 310 mm, Fig. 25.13, gives a pressure that is twelve times lower than with ANFO. Both the lower intensity of the shock wave as well as the lower volume of gases help minimize overbreak in contour blasting.

### 25.5.4 *Drilling precision*

If drilling precision is required in any type of blast to obtain optimum results, in contour blasting this is critical, as the blastholes should be placed in the desired plane or surface and be maintained parallel at the calculated distance. The causes for blasthole deviation, their influence and the corrective measures are found below:

1. Badly marked out blastholes. This operation should be carried out by qualified personnel and not by the drillers.
2. Incorrect instalation of the drilling rig or jumbo boom, taking into account that sometimes it is necessary to level the ground or even use concrete.
3. Incorrect alignment of the drilling feed when inclined drilling is used; these errors can usually be corrected by automatic direction control systems.
4. Defective collaring of the holes.
5. Unfavorable geological conditions: schistosity, fissures, solution cavities and weathered rock.
6. As to the actual drilling techniques:

– Influence of the drilling diameter and drill steel; the smaller they are, the larger the deviation.

– Use of stabilizers, especially in fissured ground and with solution cavities.

– Control of the rotation velocity, even if the penetration velocity diminishes.

– Influence of the type of drill bit.

– Quality of drill steel used.

Once the drilling is completed, a control should be carried out to verify the blasthole deviation. In underground mines and with large diameter holes (165 mm), this can be done with Eastman type cameras.

### 25.5.5 *Geometry and initiation sequence of the blast.*

Everything explained below refers to presplitting and cushion blasting.

*Drilling diameter*

In tunnels and underground operations, the drilling diameters most used vary between 32 and 65 mm, with field trial experiences of up to 75 mm holes. In underground mining and depending upon the exploitation method, the diameters vary between 50 and 65 mm, as in the *Sublevel methods* for example, with up to 165 mm diameters in the *VCR Method* and *Longhole method.*

It has been proven that the radius of the rock cyllinder surrounding the blasthole and affected by the blast, is directly proportional to the diameter of the same, as long as a constant relationship is maintained between its length and diameter.

In Fig. 25.14, it can be seen that the stress level induced at a distance of 0.9 m from the blasthole, for the same coupling of the charge, is for a diameter of 2 inches (50 mm), three times lower than for one of 6 inches (165 mm).

In surface operations, the most frequent diameters used in the past for contour blastings were in the 35 to 75 mm range. Even today those diameters are popular in civil engineering projects and in small mines; but in large mines the diameters are increasing in size, getting up to 310 mm (12¼ inches), motivated by economic reasons and the availability of equipment, even if the technical and aesthetic results were poorer.

However, and especially for underground operations, it must be taken into account that an increase in drilling diameter immediately elevates the cost of rock support. Therefore, the combination diameter/charge load of the blasthole that gives a minimum excavation and reinforcement, as observed in Fig. 25.15, should be found.

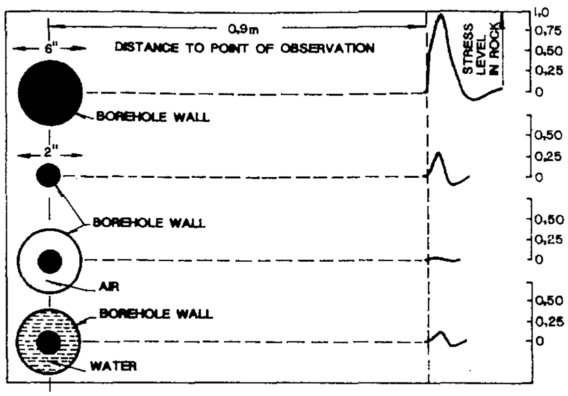



Fig. 25.14. Dynamic stresses in rock for various explosive loading conditions. (Day and Webster).

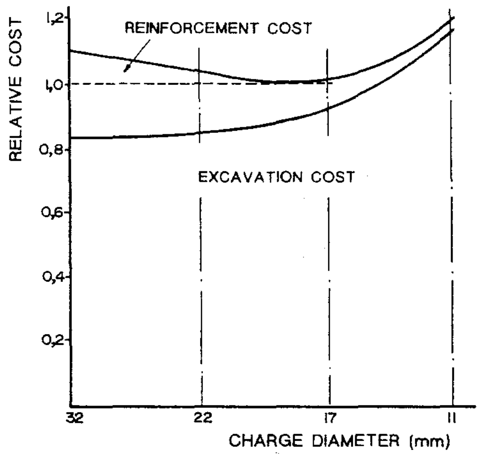



Fig. 25.15. Influence of the charge diameter upon excavation and support costs.

*Spacing and depth*

The spacing between blastholes in contour blasting depends upon the rock type and drilling diameter, and increases in the same way as this parameter.

In presplit blasting, the work is carried out with an *S/D* relationship that oscillates between 8 and 12, with a mean value of 10.

In cushion blasts the ratio *S/D* is usually around 13 to 16 with a mean value to 15, Fig. 25.16.

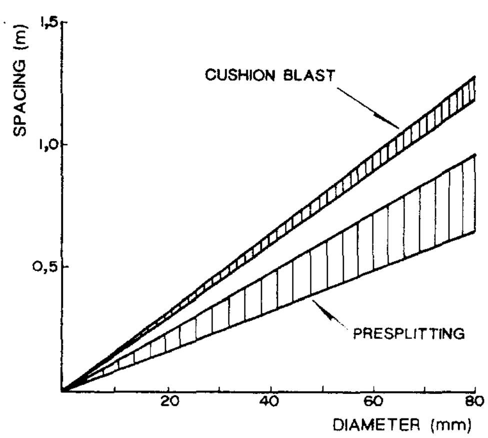



Fig. 25.16. Recommended ranges of hole spacing as a function of hole diameter for cushion blasting and presplitting.

An empirical approximation which relates the spacing dimension to the characteristics of the explosive, with or without decoupling, and to the dynamic properties of the rocks in presplit blasts, is given by Calder and Jackson (1981), in whose equation the tensile strength of the rock across the plane of the cut is equaled to the gas pressure on the walls of the blastholes, taking that these perform in an area that is equivalent to the diameter of said blastholes.

$$PB_e \times D \geq (S - D) \times RT$$

$$S \leq \frac{D \times (PB_e + RT)}{RT}$$

where: $S$ = Hole spacing, $D$ = Hole diameter, $PB_e$ = Decoupled borehole pressure of the explosive charge, $RT$ = Tensile strength of the rock.

If the in-situ tensions are high, the abovementioned equation can be modified by adding the normal stresses that act upon the presplitting plane:

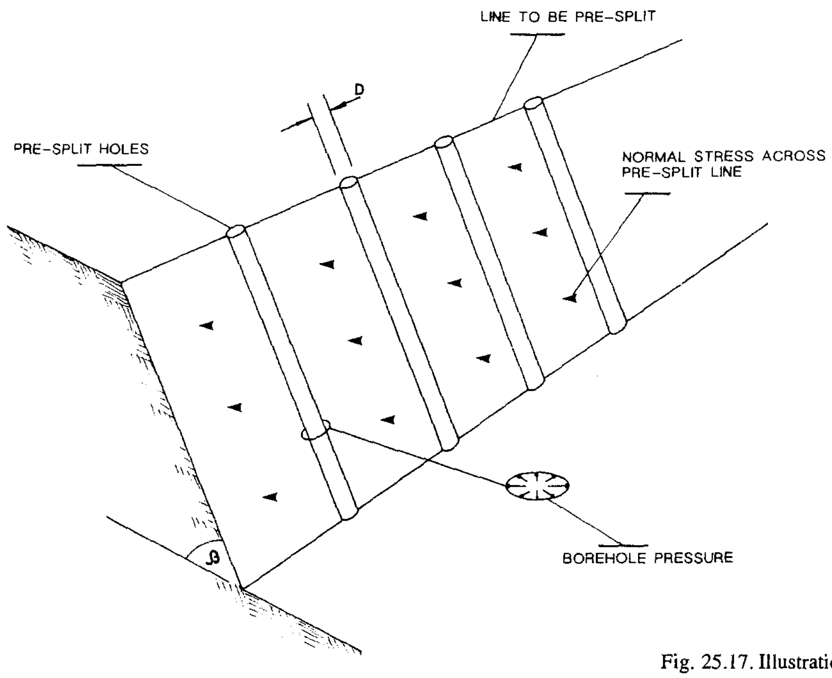



Fig. 25.17. Illustration of a pre-split mechanism.

$$S \leq \frac{D \times (PB_e + RT + \sigma_N)}{RT + \sigma_N}$$

In cushion blasts, the relationship between burden and spacing should be:

$$B = 1.25 \times S$$

As to what is referred to as the depth limit, this theoretically does not exist, but the problems derived from the lack of parallelism of the blastholes constitutes a real limitation. For example, for inclined blastholes of 32 to 65 mm, the limit is around 15 to 20 m. Minimum deviations can be obtained in large diameter boreholes with down-the-hole hammer drills.

In certain conditions, the results of contour blastings can be improved with pilot or empty holes, placed between charges holes in the same projected cut plane. In competent rock, the charging of all the holes is usually more effective than alternate charging, owing to the fact that in the latter design the spacing must be significantly reduced and, therefore, an increase in drilling per unit of created surface.

*Lineal charge density*

The calculation of the lineal charge density should be performed with the following in mind:

– The blasthole pressure must be below the dynamic compressive strength of the rock.

– The vibration level generated by the blast which induces strains in rock that is susceptible to breakage must be controlled, Fig. 25.18.

The damage will appear for a peak particle velocity level. In competent rock such as, for example, granite, if the law of propagation is not available, the following equation may be used:

$$v\,(\text{mm/s}) = 700\,Q(\text{kg})^{0.7} \times DS\,(\text{m})^{-1.5}$$

In near places, where the charge length constitutes an important parameter, the vibration intensity can be obtained by using the above equation (Persson et al., 1977), Fig. 25.19.

To solve the problem of the cut of the rock at the desired depth, the bottom charge concentration should be double the normal in a length equal to $S/2$. Larger charge concentrations would provoke cracks and overbreak in the bottom of the area.

To make a rapid and approximate calculation of the quantity of explosive necessary to design a contour blasting, the following equations can be used:

a) $q_l\,(\text{kg/m}) = 8.5 \times 10^{-5}\,D\,(\text{mm})^2$

b) $q_s\,(\text{kg/m}^2) = \dfrac{D\,(\text{mm})}{130}$

The above equations are devised as mean values for explosives with a density of 1.2 g/cm$^3$ and rocks also with mean properties, Fig. 25.20.

*Cushion blasting with detonating cord.* For some applications, when the blastholes have to be drilled very close to each other because of geological conditions, to avoid overbreak, or to demolish concrete structures, the charges can be made up of detonating cord. In these cases, the blastholes are drilled with larger diameters than those used normally, but with smaller spacings. The additional decoupling does not normally deteriorate the rock formation, but the equations for calculating the charges as a function of the diameter cannot be used, as the values

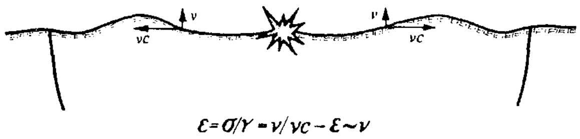



Fig. 25.18. The interrelation between strain $E$, stress $\sigma$ and vibration velocity $V$. $C$ is the wave propagation velocity and $Y$ is the Young's modulus. (Holmberg and others).

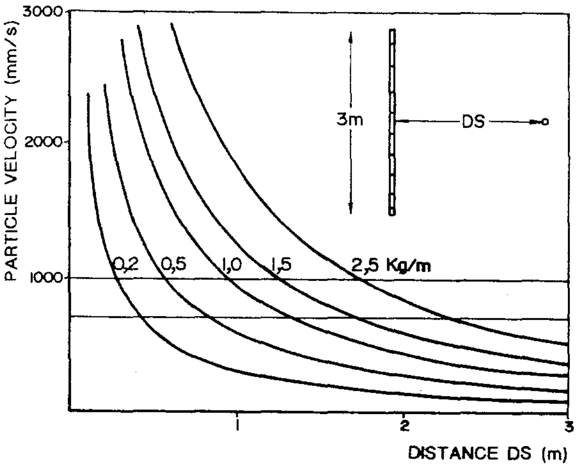



Fig. 25.19. Estimated peak vibration velocity as a function of distance for different linear charge densities, kg/m borehole length and distance ranges typical of small diameter hole tunnel blasting.

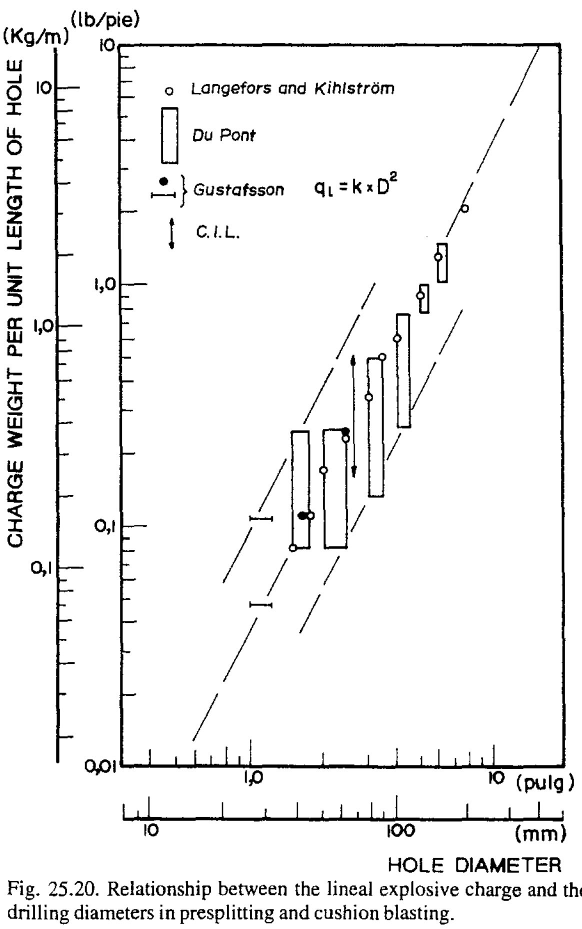



Fig. 25.20. Relationship between the lineal explosive charge and the drilling diameters in presplitting and cushion blasting.

would be too large for the spacing. The formula to calculate the charge density as a function of a prefixed spacing is:

$$q_l = 300 \cdot S^2$$

where: $q_l$ = Charge density (g/m), $S$ = Spacing (m).

*Example:*
*What charge density should a cushion blast drilled with 50 mm blastholes and spaced at 40 cm have?*

$$q_l = 300 \cdot 0.4^2 = 48 \text{ g/m} \approx 50 \text{ g/m}$$

*Stemming*

There are differences of opinion for this parameter among blasting specialists, because where some reduce stemming as the rock strength increases, others do the contrary. It seems that the latter way of doing is more logical.

In competent rock, the stemming height oscillates between 6 and 10 times the diameter and is done with the drill cuttings, aided by a paper or cotton plug in the base of the same, depending upon the diameter of the borehole. In stratified and fissured rock, it is recommended that the anular space between the charge and the wall of the blasthole be filled with fine material, in order to reduce overbreak caused by the wedging and opening effect of the explosion gases.

In very weathered rock formations, it is sometimes beneficial to reduce stemming to a minimum, or altogether, so as to allow a rapid escape of the gases and that way preserve the surface rock. With this method, the charge should be extended towards the collar of the blastholes. Because this method allows the rapid release of high-pressure explosion gases, extra precautions should be taken to avoid over pressure and damage from flyrock.

*Delay timings and intiation sequences*

As already indicated, the appearance of a crack along the length of the blasthole row is based upon the almost simultaneous effect of the respective shock waves, and for this reason the best results are achieved when all the holes are connected to the same line of detonating cord or energetized with detonators of the same number, Fig. 25.21.

When the amount of explosive detonated per unit of time has to be limited due to vibration problems, millisecond delays can be inserted between the different groups of holes, or initiate each group with a milisecond detonator of different number.

In surface operations, the spacial advance of the presplit should be twice the spacing or the burden approximately and in depth, two or more benches can be reached, depending upon the lithological and structural changes of the massive rock and the quality of drilling.

The presplitting blasts shoud be fired with a minimum of burden, which is around 12 m for blastholes of 50 mm and 20 m for those of 310 mm in diameter, because if this is not the case, the tensile and confinement conditions would not be the ideal.

When the stope blastings are fired with the presplitting, the latter should be timed in advance by at least 90 to 120 ms, so that the fracture will be totally developed before the front row of production blasts detonates.

### 25.5.6 *Stope blasting and the protection of the cushion blast*

In order to preserve the pre-split plane from damage by the stope blasts, this should be designed according to the model of buffer blasting. This type of rounds are characterized because in the row of blastholes nearest to the presplitting, the powder factor is reduced to almost half that used in a production row, and the burden and spacing are reduced to from 0.5 to 0.8 times the nominal values for the adjacent row.

The distance between the presplit plane and the buffer row cannot be very small because the shock wave would cause overbreak in the projected face, and cannot be excessively large as it would provoke a large toe volume which would require a secondary blast thereby reducing the yield of the loading equipment.

The distance between the presplit and the last row oscillates between 0.33 and 0.5 times the nominal burden of the production blast. In the boreholes of the rows that are over projected berms, the subdrilling will be reduced or eliminated in order to avoid damage to the bench top underneath. The maximum number of rows that are recommended for firing is usually three, Fig. 25.22.

As to the charge configuration in the buffer row, two tendencies are followed: the first consists in decoupling the explosive as in the pre-split row, and the second, in which the explosive is supposed to act as a spherical charge, calculating the distance from the center of gravity of the charge to the collar of the hole with Livingston's well known equation:

$$D_c = E_t \times Q^{1/3}$$

where $Q$ is the explosive charge and $E_t$ is called the *Energy Strain Factor.* For the charges in the buffer row, it is recommended that $E_t$ oscillate between 1.2 and 2 $m/kg^{1/3}$, depending if there is a weak rock mass or soft rock, respectively. The relationship length/diameter of the charge so that it will perform as a spherical charge, should be between 6 and 8.

The initiation sequence of the stope blast should be established so that the last row will have a minimum confinement and can exit easily so that the pre-split will not be damaged.

In cushion blastings, the charge density in the holes of the next rows should also be controlled.

In Fig. 25.23, for an excavation of a tunnel with cushion blasting as a finish, two stope blastings are represented. In case a), due to an overloading of the rows 1 and 2, an overbreak is produced outside the forseen profile and the blast is not effective. In case b), correct charge densities were chosen and the desired results were achieved.

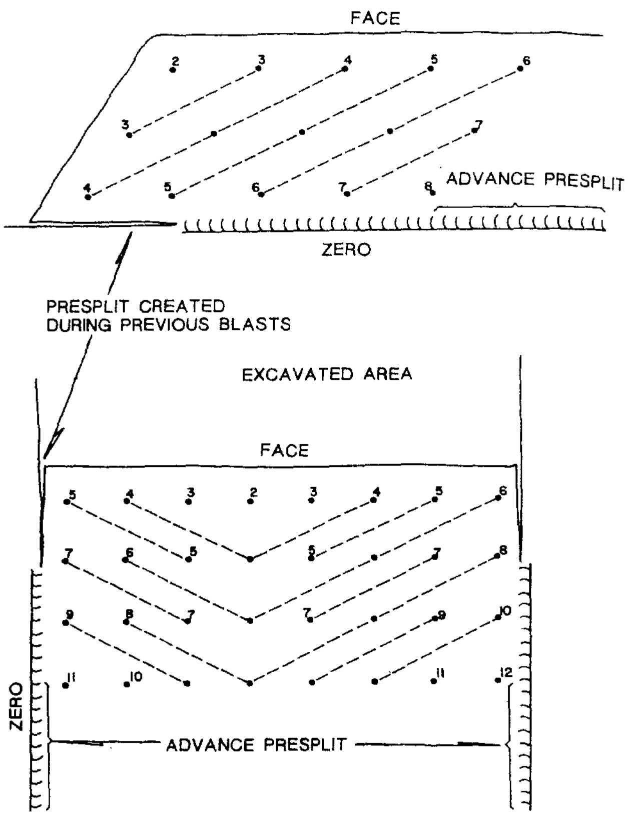



Fig. 25.21. Advance of the presplitting blasts.

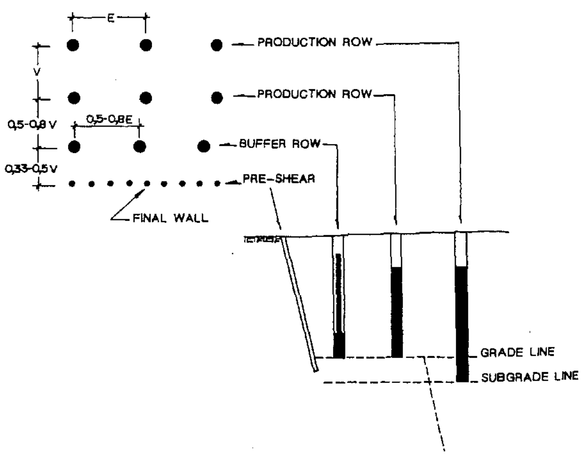



Fig. 25.22. Design for stope blasting near the pre-split line.

## 25.6 TENDENCIES IN THE FIELD OF CONTOUR BLASTING

### 25.6.1 *Presplitting with air spacers*

Since the middle of the eighties, a new technique for presplitting was put into use in the United States, called *Air Deck Presplitting – ADP.* This consists in placing a small explosive charge in the bottom of the blasthole, leaving the rest empty up to the stemming which is composed of a plug placed at a certain depth, Fig. 25.24.

Photo 25.4. Aspect of a fissured plane surface created by a presplitting blast.

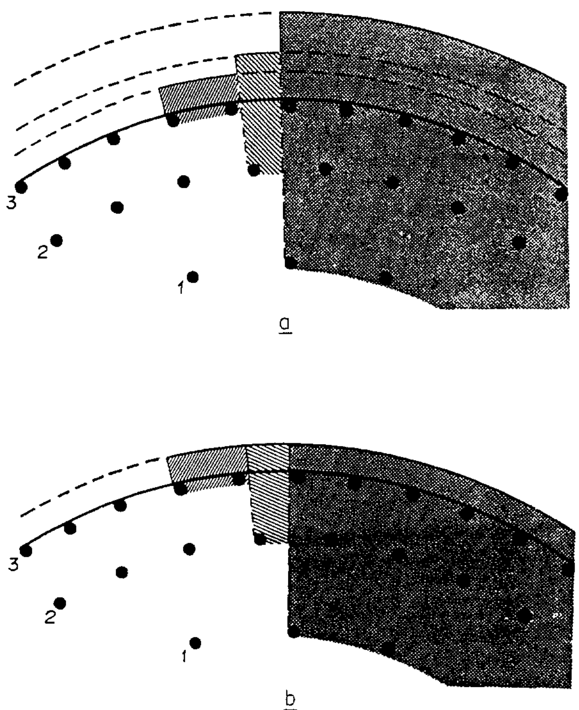




Fig. 25.23. Optimization of blast pattern to avoid damage from adjacent row to the perimeter.

Photo. 25.5. Simultaneous firing of stope and pre-split blastholes in a final slope.

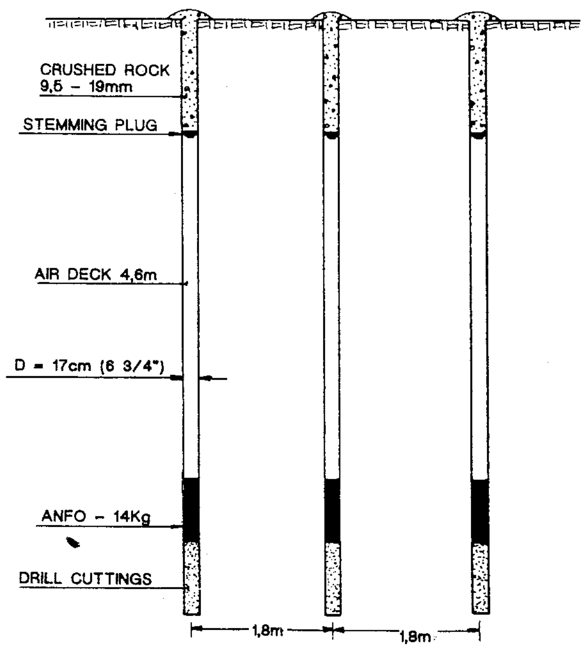




Fig. 25.24. Presplitting technique ADP with air columns.

The basic principles of this technique were introduced more than forty years ago by the Russian scientists Melhivov (1940) and Marchenko (1954) whose observations revealed a better explosive yield with improved fragmentation and displacement.

In 1948, Fourney et al of the University of Maryland, while investigating the estimulation of oil wells and carrying out small scale tests, found out that by firing charges in the bottom of air spaces, as those of blastholes, the shock waves reflected on the roof of the stemming generating longer lasting strains and intensities of 2 to 5 times those registered in the bottom where the charges are placed.

In 1982, Crosby et al used air columns for presplitting in the surface coal mine of Rietspruit, South Africa.

However, it was after 1983 when the Atlas Powder Co. started a series of tests that gave a better knowledge of the technique, extending its use to different areas from then on. A pneumatic plug was obtained that allowed a more effective stemming, with an important diminishing of airblast which was a deterring factor for use of the technique near populated areas.

The following are practical rules of design parameters which can be applied:

$$S = (16 \text{ to } 24) \times D$$
$$T = (12 \text{ to } 18) \times D$$
$$Q = (0.39 \text{ to } 1.4) \times H \times D$$
$$B = 12 \times D$$

Where: $D$ = Blasthole diameter (m), $S$ = Spacing (m), $T$ = Stemming (m), $Q$ = Explosive charge in the bottom of the hole, $B$ = Distance from the stope row (m).

There are different types of plugs: pneumatic and chemical. The first and most widely used consist in a rubber bag that can be filled with pressured air from the surface once the bag is placed at the desired depth. These plugs are sold for blastholes with diameters that go from 75 mm to 380 mm, and two of the most well known are *Power Plug* and *Hole Saver*.

The chemical plugs are made up of a cartridge with two components (isocyanate and polyol resin) which react forming a polyurethane foam in about two to five minutes depending upon blasthole temperature.

The ADP system, at present, is indicated for drilling diameters between 127 and 310 mm, lowering the costs per square meter of presplitting for the following reasons:

– Use of conventional bulk explosives that are more economical than the special presplitting charges.
– Larger spacing between blastholes.
– Possibility of using larger drilling diameters.
– Less need for specialized personnel.

Apart from the contour blasts, there are other areas of interest for the use of the ADP system such as:

– *For ore-waste separation.* The explosive placed in the orebody zone produces a fine fragmentation while the waste, where the air column is situated, is left in large pieces, allowing visual separation, Fig. 25.25a.

– *Flyrock control.* In the first row of blastholes, when the size of the burden is less than the nominal, or the geology is unfavorable, Fig. 25.25b.

– *Reduction of fines.* In some ores the production of fines gives important losses, so that by modifying the charge method, Fig. 25.25c, with air spacers, the fines can be reduced by 50 % and the powder factors between 15 and 20 % in relation to conventional use.

– *Rip rap blasting.* Fig. 25.25d. The technique is being used with success in rip rap blasting, with yields of 20 t of material per kilo of explosive, drilling blastholes of 165 mm in diameter.

– *Blasting for ornamental rock.* The technique is applied in horizontal and vertical blastholes drilled for extraction of ornamental rocks.

– *Vibration control.* Fig. 25.23e. For spacing and sequencing the elemental charges in the same blasthole and thereby diminishing the vibration level, at the same time obtaining a better slope and improved fragmentation.

### 25.6.2 *Other trends*

Since many years ago, the investigations in the field of contour blastings have aimed at designing *Fissure control charges*. The advantages of these methods are:

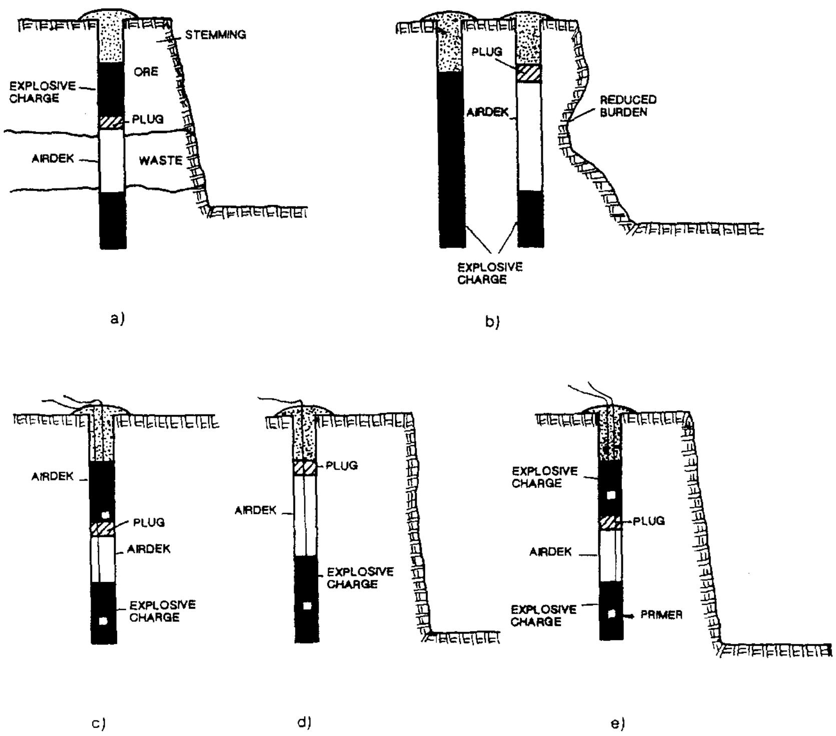



Fig. 25.25. Applications of the ADP method for different types of blasts.

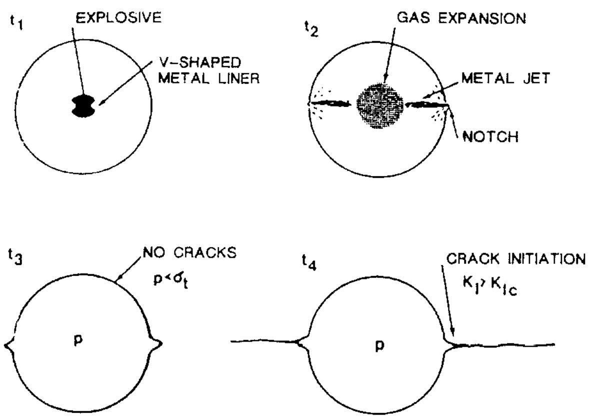



Fig. 25.26. Principle for a linear shape charge used in contour blasting (Bjarnholt and others).

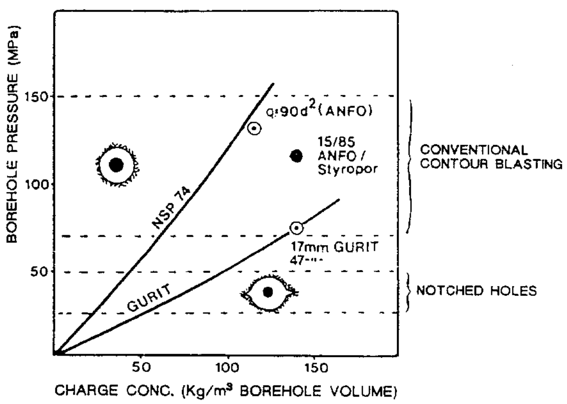



Fig. 25.27. Borehole pressure as a function of the charge concentration for some explosives.

– Conservation of the structural integrity and resistance of the remaining rock.

– Better adaptation of the excavated cavity to the dimensions of the planned profile.

– Lower powder factor per unit of blasted surface, Fig. 25.26.

– Larger spacing between holes, which means less drilling.

The techniques that are being developed at the present time are:

a) Special linear shape charges.

b) Notched blastholes.

c) A ligamented split-hole for fracture control.

The special linear shape charges work in a similar manner to hollow charges, directing the explosive energy towards two splits on opposites sides of the blasthole, Fig. 25.26.

The notched blastholes have wedge shaped cracks opened on opposite sides along the length of the walls, with the purpose of directing the fracturation produced by the pressure of the gases, taking advantage of the stress concentration off each end of the mentioned notches. With this method excellent results have been obtained, and a reduction of the powder factor between 20 and 50 % of what would normally be used, Fig. 25.27.

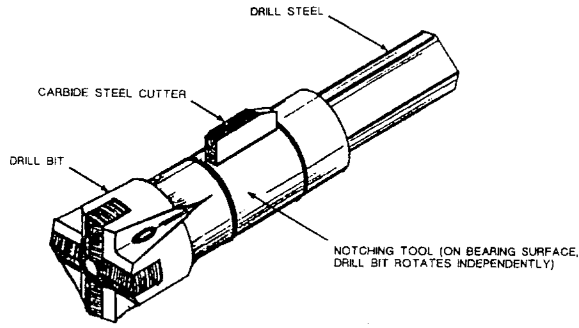



Fig. 25.28. Simultaneous drill and notch tool.

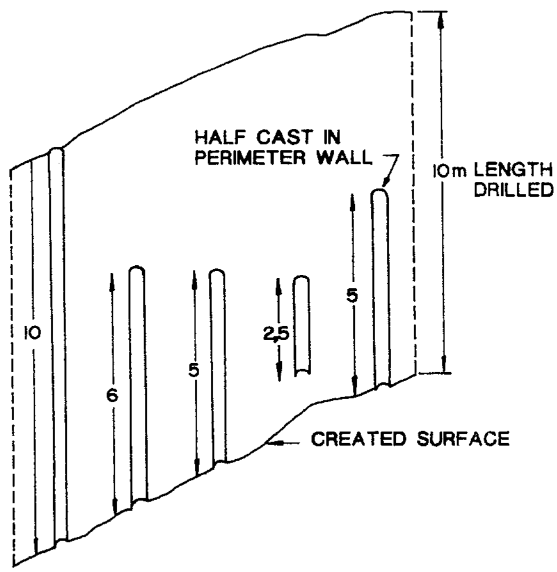



Fig. 25.29. Example of Half Cast Factor *HCF.*

There are several procedures to make the notches:

– Special drilling tools, as the one shown in Fig. 25.28.

– Water jet.

– Use of linear shape charges, Fig. 25.26.

The third technique is called *Charges with ligamented split-tubes* which consist in metal cyllinders that contain the explosive and have side openings. The metal tube has two missions: to channel the explosive energy in the splits that are opposite in the blasthole and protect the rest of the hole wall.

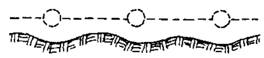
## 25.7 EVALUATION OF THE RESULTS

The evaluation of the results obtained in a contour blast can be made both quantitatively and qualitatively.

The quantitative evaluation is based on the calculation of the Half Cast Factor *HCF,* which is the quotient between the length of the half cast and the total drilling length, Fig. 25.29.

Even if the quantitative evaluation gives a value that defines the quality of the controlled blasting, it is more interesting, in order to optimize the results, to analyze the whole of the created surface, as indicated in Table 25.2,

Table 25.2.

| Type of damage | Profile of excavation | Origin of problem | Solutions |
|---|---|---|---|
| Backbreak throughout wall |  | Control blast may be overloaded. Boreholes overloaded or too close | Decrease powder load by decoupling or decking, and increase hole spacing. Move buffer row further from wall limit reduce borehole pressure of buffer charge, increase delay between buffer charges. |
| Backbreak around boreholes |  | Borehole pressure greater than in-situ dynamic compressive rock strength. | Decrease burden (for cushion blasting), use decouple or deck charges in cushion or pre-split holes. |
| Backbreak between boreholes |  | Hole spacing too close. | Increase hole spacing. |
| Blast fills to break to presplit line or very poor fragmentation |  | Spacing too great. | Reduce spacing and increase powder factor. |

for the presplitting technique. For each type of damage produced, the possible origin and solution is indicated.

## 25.8 EXAMPLE

The spacing between blastholes has to be calculated for a pre-split blast with a drilling diameter of 64 mm and continuous explosive charges of 19 mm in diameter, density of 1.1 g/m$^3$ and a detonation velocity of 4000 m/sec. The rock has an in-situ tensile and compressive strength of 17.2 and 275 MPa respectively:

1. Blasthole pressure

$$PB = 228 \times 10^{-6} \rho_e \times \frac{VD^2}{1 + 0.8\rho_e} = 2134 \text{ MPa}$$

2. Effective blasthole pressure

$$PB = 2134 \times \left[\frac{19}{64}\right]^{2.4} = 2134 \times 0.0542 = 115{,}7 \text{MPa}$$

Lower value than the compressive strength of the rock, therefore the charge configuration is valid.

3. Spacing

$$S = \frac{64 \times (115.7 + 17.2)}{17.2} = 494.5 \text{ mm} \approx 0.5 \text{ m}$$

## 25.9 EXTRACTION OF ORNAMENTAL ROCK WITH CONTOUR BLASTING

Ornamental rock is all stone that is used, in blocks or slabs, for its aesthetic characteristics such as color, texture, shine, grain, etc. and technical such as strength, facility of elaboration, polish, etc. The most common types of rock can be generically classified in three large groups: granites, marbles and marmoreal limestones.

The cutting methods consist in primary separation from the rock mass of a large block (100 to 4000 m$^3$), in parallelpiped form, which is subdivided afterwards to achieve sizes that are easily handled and within the ranges that the transformation industries require, generally lengths of 1.8 to 3.5 m, widths of 1 to 1.50 m and heights of 0.9 and 1.2 m.

The cutting technique is usually with explosives, although not exclusively, because cutting systems with helicoidial and diamond wire, with mounted rock cutters, with flame torching and with water jet kerfing are often applied.

The blasting techniques are a special type of presplitting, but with slight variations as it is of maximum importance not to damage the rock and at the same time take into account the properties: strength, homogeneity, schistocity, fissurization, etc. Although it is difficult to give general recommendations for design in this type of blasting, as there are many different rock types and exploitation conditions, the following criteria should be of use:

– *Drilling diameters.* They depend upon the phase of excavation and the type of rig used, but generally around 25 to 45 mm.

– *Spacing.* It is established as a function of the rock properties and explosive charge characteristics. The usual interval is between 4 and 8 $D$.

To be able to make an analytic calculation, the formula suggested by Berta can be applied:

$$S = \frac{2 \times PE_s \times \rho_e \times d^2}{RT \times D} + D$$

where: $PE_s$ = Specific pressure (MPa), $\rho_e$ = Density of the explosive (g/cm$^3$), $d$ = Diameter of the explosive charge (m), $D$ = Diameter of the blasthole (m), $RT$ = Tensile strength (MPa).

– *Explosives.* In the vertical benching planes detonating cords with a core of pentrite are usually used, while for the horizontal planes explosives of low detonation velocity are also used, as they generate a large volume of

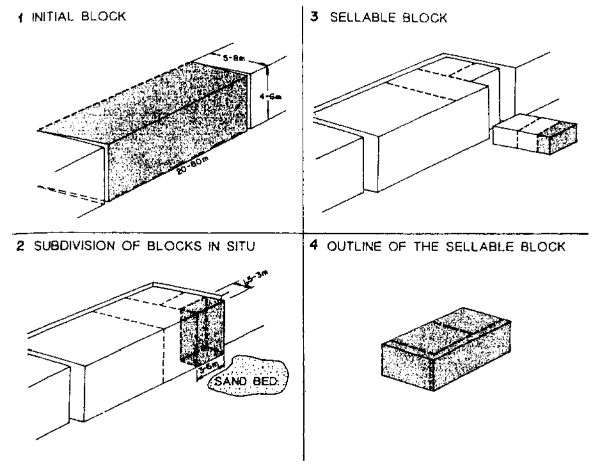



Fig. 25.30. Cutting phases in a dimensional stone quarrie.

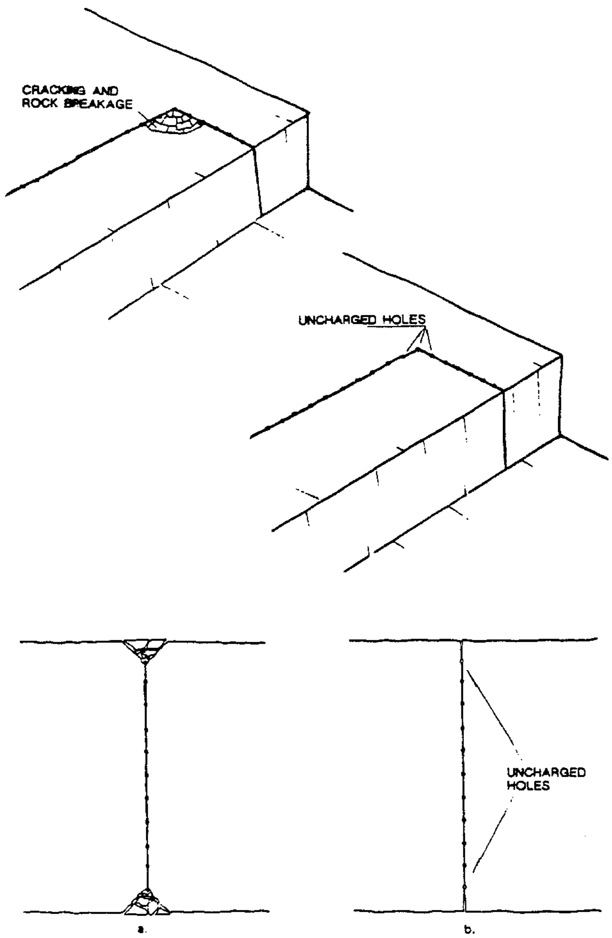



Fig. 25.31. Use of empty holes that serve as guides in the extraction of blocks.

gases. In these last planes of the cut, the structural properties of the rock mass should be used to advantage.

In some countries, there is extensive use of charges prepared in connecting plastic tubes that contain powdery

Photo 25.6. Placing a pneumatic plug to produce stemming. (Courtesy of Atlas Powder Co.)

explosives with low density and detonation velocity, made up of nitroglycerine, sodium nitrate and other ingredients.

– *Powder factors.* These vary greatly depending upon the type of rock, explosive and extraction phase. The most common values when detonating cord is used in vertical planes and for unit of cut surface are: from 80 to 150 g/m$^2$ in granites, from 40 to 80 g/m$^2$ in marbles, and from 30 to 60 g/m$^2$ in marbeal limestones.

– *Charge configuration.* The explosive columns are generally designed to be continuous and decoupled with an air chamber although, in some cases such as in hard rock, to increase the energy transmitted to the rock by the detonating cords, the blastholes are filled with water. Also, if a blackening by explosion smoke of the cut surfaces is to be avoided, the holes can be filled with sand or drilling waste.

– *Distribution of the charge in the borehole.* In order to eliminate breakings or fracturation in the corners of the blocks, it is suggested that empty holes be used at the end of the line or next to the free surfaces, Fig. 25.29. Apart from this, in vertical blastholes there is no subdrilling and they are usually drilled to a few centimeters above the horizontal plane.

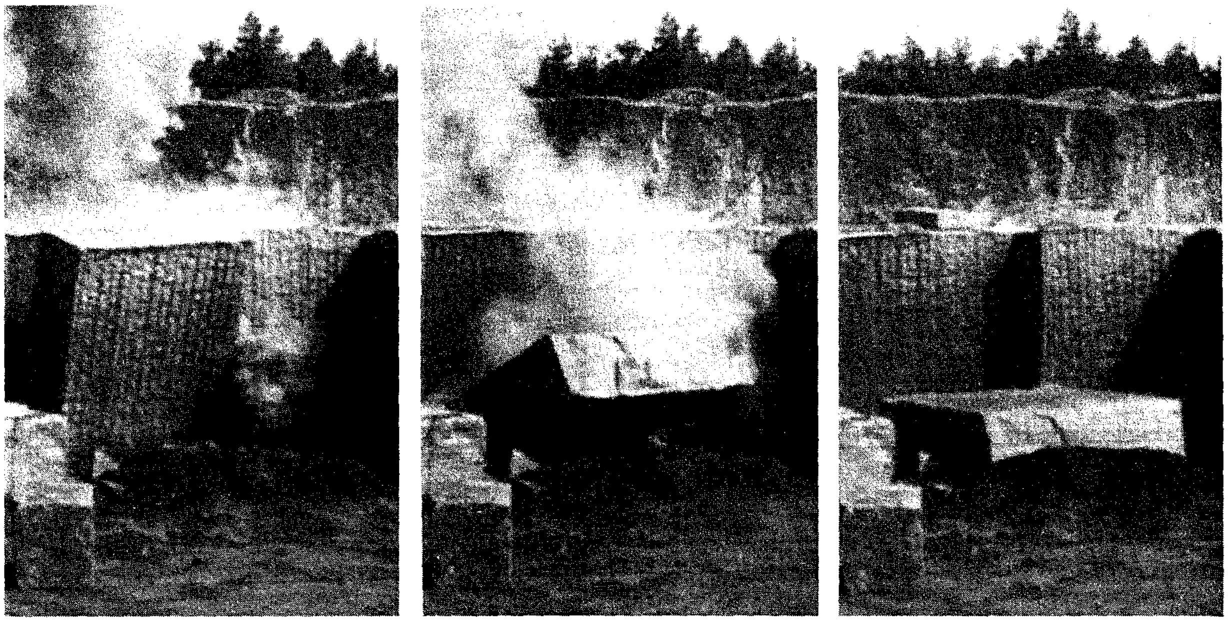


Photo 25.7. Cutting by blasting of a granite block.

– *Stemming*. They are necessary to use the maximum pushing power of the gases. As the rock characteristics become poorer, the heights are usually shortened to assure that the pressure of the gases do not act upon the rock for a long period and therefore produce damage. In general, with the detonating cords the stemmings are small, whereas with powder a larger confinement is necessary.

– *Initiation*. As in contour blasting, instantaneous initiation of all the blastholes with detonating cord is recommended.

### 25.9.1 *Example of calculation*

A block of granite is to be extracted by drilling blastholes and blasting with detonating cord. The initial data is:

- Tensile rock strength $RT = 10$ MPa.
- Drilling diameter $D = 0.032$ m.
- Diameter of the detonating cord core of pentrite $d = 0.0034$ m.
- Density of the pentrite charge $p_e = 1.3$ g/cm$^3$.
- Specific pressure $PE_s = 1200$ MPa.

What should the spacing between boreholes be?

$$S = \frac{2 + 1200 \times 1.3 \times 0.0034^2}{10 \times 0.032} + 0.032 = 0.14\,\mathrm{m}$$

The ratio $S/D$ is equal to 4.37, which is within the practical interval of 4 to 8 $D$.

If the rock were of worse quality with a tensile strength of 5 MPa, the spacing should be increased to $S = 0.26$ m.

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