CHAPTER 22

Blasting for tunnels and drifts

22.1 INTRODUCTION

The use of the underground, for mining as well as for civil engineering projects, requires the driving of drifts and tunnels in a larger number each day.

During the past few years, mechanical excavation with drifting and tunnelling rigs has advanced considerably, excavating rock with a compressive strength of up to 250 MPa. In hard rock, the tunnelling rigs have a larger field of application and offer several advantages such as: perforation without harming the surrounding rock, a smooth cut surface that reduces the necessity for support and/or a concrete lining, less personnel, etc.

However, excavation with explosives is still widely accepted as the aforementioned method has its inconveniences:

- Rigid work system as the sections must be circular.
- The ground to be drilled must not have important variations or geological upsets.
- The curves should have a radius over 300 m.
- The initial excavation is costly, and
- Personnel must be highly specialized.

Fragmentation with drilling and blasting solves most of these problems because although the sections are large, the excavations can be carried out in stages with advance drifts, lateral breakage and/or benching, plus the fact that modern jumbos can cover large sections with varying shapes. The remaining rock can be left in good condition with contour blasting using smooth blasting and presplitting techniques which are better adapted to the lithological changes of the ground, and require less machine time, which means the equipment can be sent on to other jobs.

The basic drilling cycle is composed of the following operations:

- Blasthole drilling.
- Charging.
- Blasting.
- Ventilation.
- Scaling and grouting (if necessary).
- Loading and haulage, and
- Setting up of the new round.

In the following paragraphs the present day techniques for tunnelling and drifting, as well as the calculations for drilling patterns and charges are revised.

22.2 ADVANCE SYSTEMS

The manner or pattern used to attack the tunnel and drift sections depends upon various factors:

- Drilling equipment used.
- Time available for the operation.
- Type of rock.
- Type of support, and
- Ventilation systems.

In good rock, the tunnels with sections under 100 m^2 can be excavated by the full face method. Larger tunnels which cannot be covered by the drilling unit, or those where the geomechanic properties of the rock do not permit the full face method, have to be excavated by the top heading method, Fig. 22.1.

The procedure consists of dividing the tunnel in two parts, an upper or top heading, and a lower in bench or stoping holes. The top heading is excavated as a drift and the stoping holes, which take longer than the top heading advance, are carried out by benching.

Benching can be vertical, in which case it will be necessary to have a crawler drill with a feed that is not too large in order to avoid positioning problems near the side walls. The advantage of this system is that the whole bench can be drilled and blasted without interruption and simultaneously with the top heading, Fig. 22.2.

Horizontal benching uses the same drilling equipment as for the top heading and also the same procedure for charging the explosive and mucking, but it has the problem of work interruptions.

When the rock is of poor quality, the tunnel is usually divided into several smaller sections. A common technique is to open a pilot heading in the top heading with one or two lateral stoping holes. The pilot heading, which is mainly for inspection, is done ahead of the stoping and may even be opened before starting the lateral drilling which provides better work ventilation. The excavation of the top heading is usually completed before starting rock breakage of the lower section, although in wide tunnels it may be carried out simultaneously by establishing an access between the tunnel floor and the top heading by means of a lateral ramp.

At present, one of the tunnelling procedures most widely used is known as the *New Austrian Tunneling Method*. It roughly consists of excavation in stages, such as indicated. After opening the advance drift in the upper half section, the lateral stoping holes can be carried out either simultaneously or by dephasing, using the drift cavity as a free face and placing the contour blastholes so as to achieve a permanent profile with as little damage possible to the rock, using the smooth wall blasting

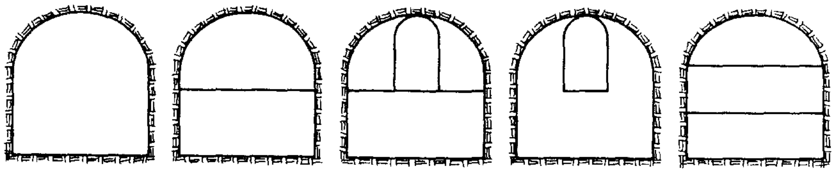

Fig. 21.1. Systems for advance in tunnel and drift excavations.

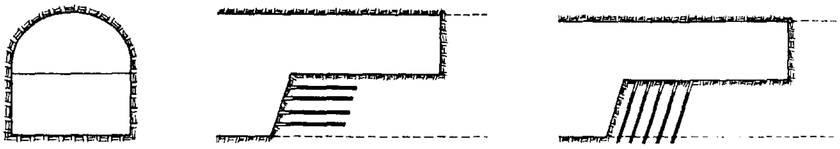

Fig. 22.2. Horizontal and vertical benching in a tunnel with the advance in two sections.

technique. Later, after mucking, shotcrete lining is built in the area to avoid decompressions and loss of compressive strength qualities of the rock.

At a certain distance from the face, which is usually the same as the advance of the rounds, the final lining should be built in by any of the existing procedures.

The excavation of the lower section is also carried out in stages, in the center part by benching and in the lateral masses with side stoping or smooth wall blasts. The drilling can be vertical or horizontal and the advance of the side stoping either simultaneous or dephased.

22.3 BLASTING PATTERNS FOR TUNNELS

The blasts in tunnels and drifts are characterized by the initial lack of an available free surface towards which breakage can occur; only the tunnel heading itself. The principle behind tunnel blasting is to create an opening by means of a cut and then stoping is carried towards the opening. The opening usually has a surface of 1 to 2 m^2, although with large drilling diameters it can reach up to 4 m^2. In fan cuts, the cut and *cut spreader* blastholes usually occupy most of the section.

Stoping can be geometrically compared to bench blasting although it requires powder factors that are 4 to 10 times higher. This is due to drilling errors, the demand made by swelling, the absence of hole inclination, the lack of co-operation between adjacent charges and, in some areas, there is a negative action of gravity as happens in lifter holes, Fig. 22.3.

Contour holes are those which establish the final shape of the tunnel and are placed with little spacing and directed towards the interior of the mass to make room for the drills in collaring and advance, Fig.22.4.

The position of the cut has influence on rock projection, fragmentation and also on the number of blastholes. Of the three positions, corner, lower center and upper center, the latter is usually chosen as it avoids the free fall of the material, the profile of the broken rock is more extended, less compact and better fragmented.

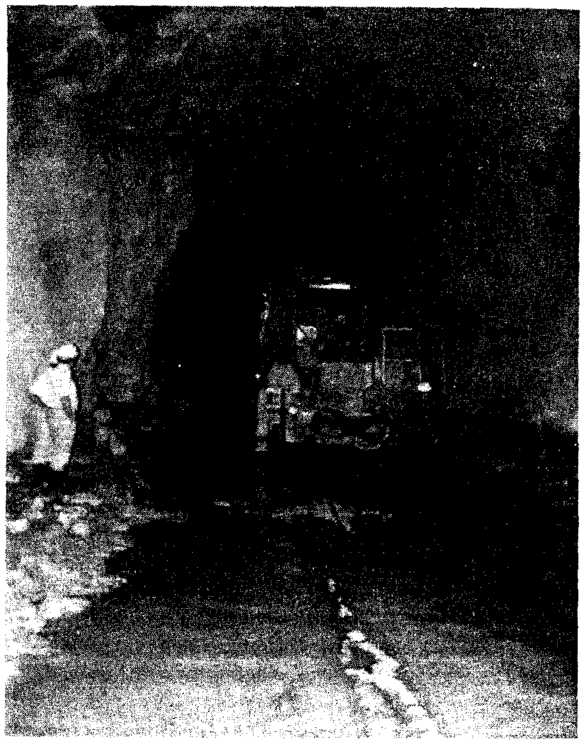

Photo 22.1. Pilot excavation and lateral stoping holes of the upper section of the pressure tunnel with a diameter of 12 m, in the Saucelle Power Station.

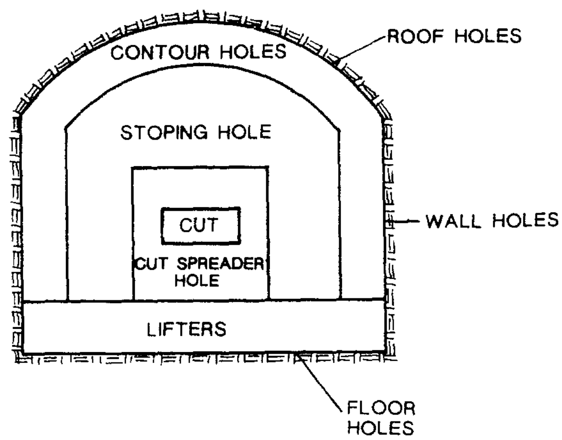

Fig. 22.3. Zones in tunnel blasting.

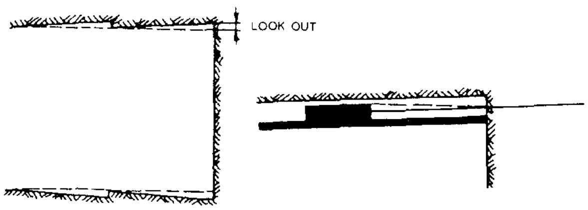

Fig. 22.4. Orientation of the contour blastholes to maintain the tunnel profile.

22.4 TYPES OF CUTS AND CALCULATION OF THE BLASTS

The blasts in tunnels and drifts are much more complex than bench blastings owing to the fact that the only free surface is the tunnel heading. The powder factors are elevated and the charges are highly confined. On the other hand, burdens are small, which requires sufficiently insensitive explosives to avoid sympathetic detonation and at the same have a high enough detonation velocity, above 3000 m/s, to prevent channel effects in the cartridge explosives placed in large diameter blastholes. This phenomena consists of the explosion gases pushing the air that exists between the column charge and the wall of the blasthole, compressing the cartridges in front of the shock wave, destroying the hot spots or excessively increasing the density of the explosive.

As to drilling, this has become more mechanized in the last decades, based upon the development of hydraulic jumbos, with one or various booms, automatized and more versatil. Because of this, the inclination has been towards parallel hole cuts as they are easier to drill, do not require a change in the feed angle and the advances are not as conditioned by the width of the tunnels, as happens with angled cuts.

Therefore, cuts can be classified in two large groups:
- Parallel hole cuts, and
- Angled hole cuts.

The first group is most used in operations with mechanized drilling, whereas those of the second have fallen in disuse due to the difficulty in drilling. They are only applied in small excavations.

In the following, the different types of cuts are explained in their order of importance, as well as calculation of the patterns and charges in the rest of the sections which are, generally speaking, independent from the type of cut applied.

22.4.1 *Cylindrical cuts*

At the moment, this type of cut is most frequently used in tunnelling and drifting, regardless of their dimensions. It is considered to be an evolution or perfection of the burn cuts which will be discussed later on. This type of cut consists of one or two uncharged or relief blastholes towards which the charged holes break at intervals. The large diameter blastholes (65 to 175 mm) are drilled with reamer bits which are adapted to the same drill steel which is used to drill the rest of the holes.

All the blastholes in the cut are placed with little spacing, in line and parallel, which explains the frequent use of jumbos which come with automatic parallelism.

The type of cylindrical cut most used is the four section, as it is the easiest one to mark out and execute. The calculation method for patterns and charges of this cut and for the rest of the tunnel zones, uses the Swedish theories recently up dated by Holmberg (1982), and simplified by Olofsson (1990), which will be studied below. Finally, other types of cylindrical cuts have been used with success and have been well experimented.

Advance per round

The advance of the rounds is limited by the diameter of the relief hole and the deviation of the charged holes. As long as the latter is maintained under 2%, the average advances X can reach 95% of the blasthole depth L

$$X = 0.95\,L$$

In the four section cuts, the depth of the blastholes can be estimated by the following equation:

$$L = 0.15 + 34.1\,D_2 - 39.4\,D_2^2$$

where: D_2 = Diameter of empty hole (m).

When cuts of NB empty holes are used instead of only one large diameter drillhole, the former equation is still valid making

$$D_2 = D'_2 \times \sqrt{NB}$$

where D'_2 is the diameter of the empty blastholes.

Cut and 'cut spreader'

The general geometric pattern of a four section cut with parallel blastholes is shown in Fig. 22.5.

The distance between the central blasthole and those of the first section should not be more than 1.7 D_2 to obtain fragmentation and a satisfactory movement of the rock (Langefors and Kilhström, 1963). The conditions of fragmentation vary greatly depending upon the type of

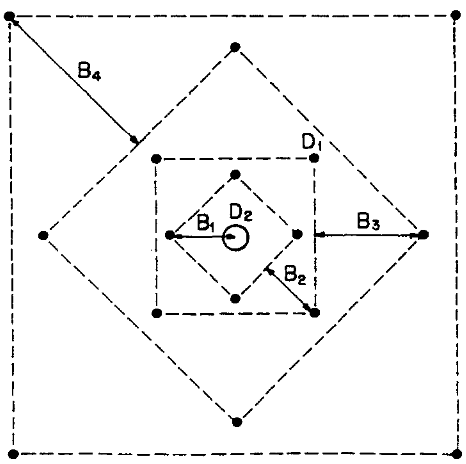

Fig. 22.5. Four section cut.

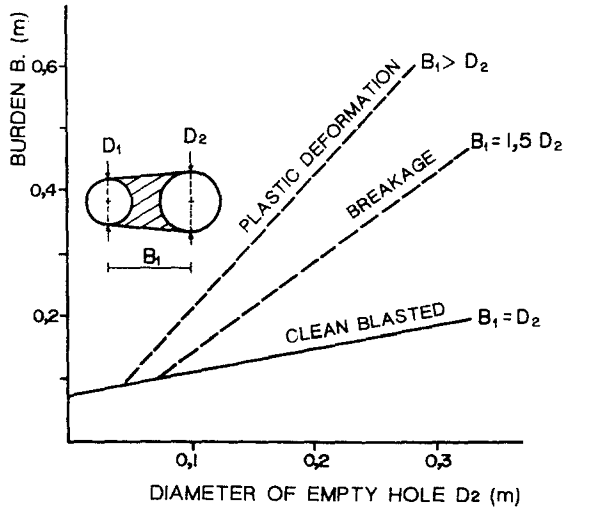

Fig. 22.6. Results of the blastings for different distances between charged and empty blastholes, and their diameters (Holmberg).

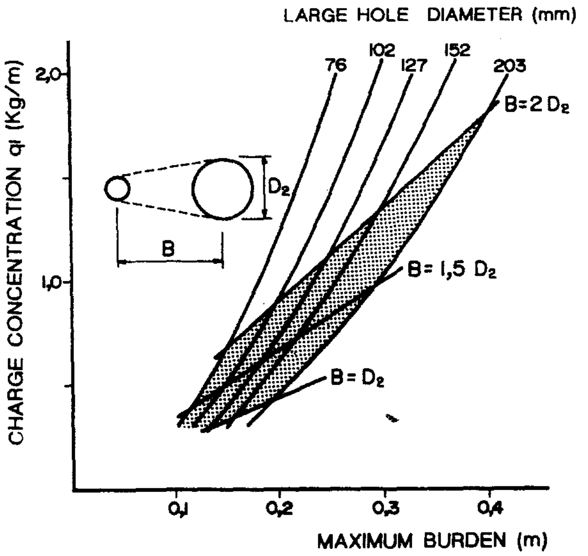

Fig. 22.7. Relationship between the lineal concentration of maximum burden and charge for the different diameters of relief blastholes (Larsson and Clark).

explosive, rock properties and the distance between the charged blasthole and the relief hole.

As reflected in Fig. 22.6, for burdens larger than 2 D_2, the break angle is too small and a plastic deformation of the rock between the two blastholes is produced. Even if the burden is under D_2, but the charge concentration is high, a sinterization of the fragmented rock and cut failure will occur. For this reason, it is recommended that the burdens be calculated from $B_1 = 1.5\, D_2$.

When drilling deviation is more than 1%, the practical burden is calculated from:

$$B_1 = 1.7\, D_2 - E_p = 1.7\, D_2 - (\alpha \times L + e')$$

where: E_p = Drilling error (m), α = Angular deviation (m/m), L = Blasthole depth (m), e' = Collaring error (m).

What usually happens is that the drilling is sufficiently good and the work is carried out with a burden value equal to one and one half times the expansion diameter. The lineal charge concentration is calculated from the following equation:

$$q_l = 55\, D_l \left[\frac{B}{D_2}\right]^{1.5} \times \left[B - \frac{D_2}{2}\right] \times \left[\frac{c}{0.4}\right] \times \frac{1}{\mathrm{PRP}_{\mathrm{ANFO}}}$$

where: q_l = Lineal charge concentration (kg/m), D_l = Drilling diameter (m), D_2 = Diameter of the relief blasthole (m), B = Maximum distance between holes and burden (m), c = Rock constant, $\mathrm{PRP}_{\mathrm{ANFO}}$ = Relative weight strength of the explosive with respect to ANFO.

Frequently, the possible values of the lineal charge concentrations are quite limited as there is not an ample variety of cartridged explosives. This means that for a pre-fixed lineal concentration, the burden size can be determined from the former equation, although the calculation is a bit more complex, Fig. 27.7.

To calculate the rest of the sections, it is considered that some rectanglar openings of A_h width already exist and that the lineal charge concentrations q_l are known. The burden value will be calculated from:

$$B = 8.8 \times 10^{-2} \sqrt{\frac{A_h \times q_1 \times \mathrm{PRP}_{\mathrm{ANFO}}}{D_1 \times c}}$$

When there is a drilling error, such as that seen in Fig. 22.9, the free surface A_h differs from the hole distance A_h in the first section, for which

$$A_h = \sqrt{2}\,(B_1 - E_p)$$

and, by substituting this value in the former equation, the following occurs:

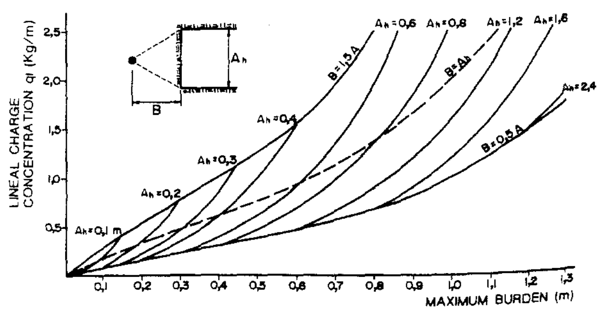

Fig. 22.8. Relationship between the lineal concentration of the charge and the maximum burden for different widths of the opening (Larsson and Clark).

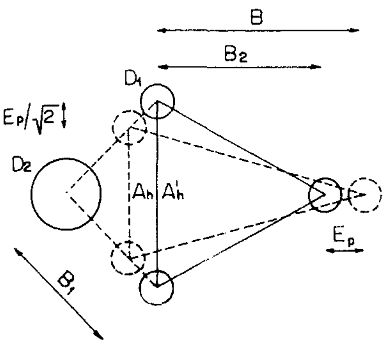

Fig. 22.9. Influence of blasthole deviation (Holmberg).

$$B = 10.5 \times 10^{-2} \sqrt{\frac{(B_1 - E_p) \times q_l \times \mathrm{PRP}_{\mathrm{ANFO}}}{D_1 \times c}}$$

This value must be reduced by the blasthole deviation to obtain the practical burden

$$B_2 = B - E_p$$

There are a few restrictions put on B_2, as it must satisfy:

$$B_2 \leq 2\,A_h$$

if plastic deformation is to be avoided. If this is not true, the lineal charge concentration should be modified using the following equation:

$$q_l = \frac{540\,D_1 \times c \times A_h}{\mathrm{PRP}_{\mathrm{ANFO}}}$$

If the restriction for plastic deformation is not satisfactory, it is usually better to choose a lower weight strength explosive in order to optimize fragmentation.

The aperture angle should also be less than 1.6 rad (90°). If not, the cut will lose its character of a four section cut. This means that:

$$B_2 > 0.5\,A_h$$

Gustafsson (1973) suggests that the burden for each section be calculated with $B_2 = 0.7\,B'$.

A rule of thumb to determine the number of sections is that the side length of the last section B should not be less than the square root of the advance. The calculation method for the rest of the sections is the same as for the second section.

The stemming lengths are estimated with:

$$T = 10\,D_1$$

Some of the problems that can arise in blastings with parallel blasthole cuts are sympathetic detonation and dynamic pressure desensitization. The first phenomenon can appear in a hole that is adjacent to the detonating hole when the explosive used has a high degree of sensitivity such as all those with nitroglycerine in their composition. On the other hand, the dynamic pressure desensitization takes place in many explosives, and especially in ANFO because the shock wave of a charge can elevate the density of the adjacent charge above the critical or death density.

Desensitization problems can be attenuated by correctly designing the initiation sequences, sufficiently delaying the successive detonation of each blasthole so that the shock wave from the last shot disappears, allowing the explosive to recuperate its normal density and degree of sensitivity.

Hagan suggests that, in order to diminish these problems, the burn-cuts be carried out by placing three relief holes in such a manner that they act as a shield between the charged holes, Fig. 22.10

Hagan has also been able to prove that fine grade rock is more subject to cut failures than coarse grade, due to the larger volume of relief opening that is needed for the expulsion of the material.

As in burn-cuts each successive detonation enlargens the available space for expansion of the blastholes which have not yet fired, the burden can get increasingly larger, therefore placing the charges in a spiral Fig. 22.11.

a) *Double spiral burn-cut*

A central blasthole is drilled with a diameter between 75 and 200 mm, surrounded by smaller blastholes that are placed and charged in a spiral, Fig. 22.12.

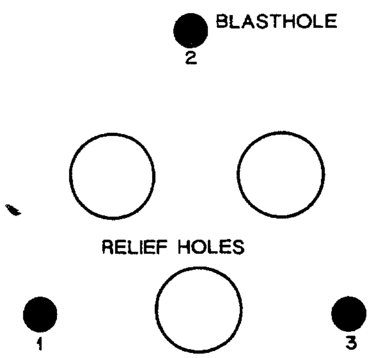

Fig. 22.10. Modified burn-cut to eliminate sympathetic detonation and dynamic pressure desensitization (Hagan).

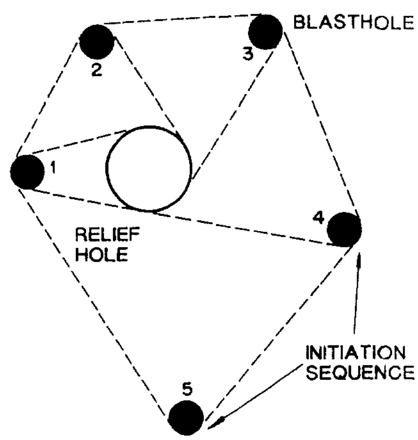

Fig. 22.11. Spiral burn-cut.

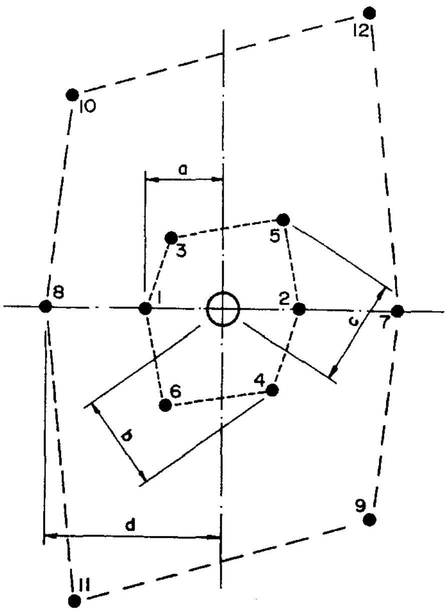

Fig. 22.12. Double spiral cut.

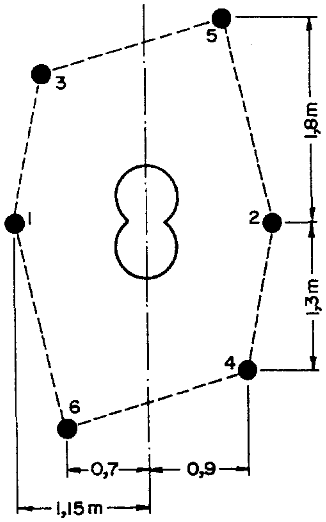

Fig. 22.13. Coromant cut.

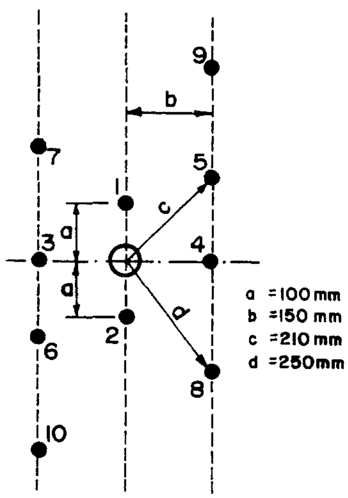

Fig. 22.14. Fagersta cut.

The blastholes 1-2, 3-4, and 5-6 are corresponding in each of their respective spirals.

b) *Coromant cut*
Consists in drilling two secant blastholes of equal diameter (57 mm), which constitute the free opening, in slit shape, for the first charges. A special drilling template is used to bore the first two holes as well as those of the rest of the cut, Fig. 22.13.

c) *Fagersta cut*
A central blasthole is drilled with a diameter of 64 or 76 mm and the rest of the charged blastholes, which are smaller, are placed according to Fig. 22.14.

This is a type of cut that is a cross between the four section cut and the double spiral, and is adequate for small drifts that use manual drilling.

Lifters
The burden for lifter holes placed in a row is calculated, basically, with the same equation that is used in bench blastings, taking into consideration that the height of the latter is equal to the advance of the round:

$$B = 0.9\sqrt{\frac{q_1 \times PRP_{ANFO}}{\bar{c} \times f(S/B)}}$$

where: f = Fixation factor. Generally 1.45 is taken to consider the gravitational effect and the delay timing between blastholes, S/B = Relationship between spacing and burden. It is usually considered equal to 1, c = Corrected rock constant.

$$\bar{c} = c + 0.05 \text{ for } B \geq 1.4\text{ m}$$
$$\bar{c} = c + 0.07/B \text{ for } B < 1.4\text{ m}$$

In lifters it is necessary to consider the lookout angle γ or inclination necessary to give a large enough space for the drilling rig to carry out the collaring for the next round. For an advance of 3 m, an angle of 3°, which has an equivalent of 6 cm/m, is enough, however it will logically depend upon the characteristics of the equipment, Fig. 22.15.

The number of blastholes will be given by

$$\text{NB} = \text{Integer of}\left[\frac{AT + 2L \times \sin\gamma}{B} + 2\right]$$

where: AT = Tunnel width (m).

The practical spacing for the corner blastholes will be:

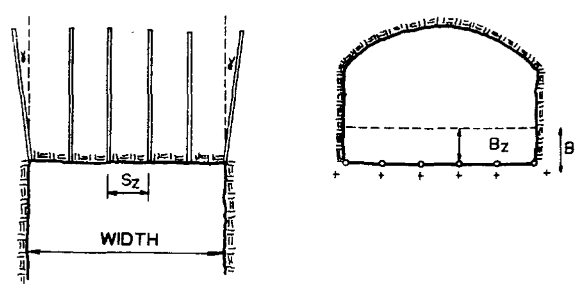

Fig. 22.15. Geometry of the lifters.

$$S'_z = S_z - L \sin\gamma$$

The practical burden B_z is obtained from

$$B_z = B - L \sin - E_p$$

The lengths of the bottom charges l_f and the column charges should be

$$l_f = 1.25\, B_z$$
$$l_c = L - l_f - 10\, D_1$$

The concentration of the column charge can be reduced to 70% of the bottom charge. However, the same concentration is usually used because of preparation time. The stemming is fixed in $T = 10\ D_1$ and the burden should comply with the following condition: $B \leq 0.6\, L$.

Stoping

The method for calculating the stoping holes is similar to the one used for lifters, only applying different Fixation Factor and Spacing/Burden relationship values, Table 22.1.

The concentration of the column charge for both types of blastholes should be equal to 50% that of the bottom charge.

Contour blasts

If the blast does not need contour or smooth blasting, the patterns are calculated as for lifters with the following values: Fixation factor, $f = 1.2$; Relationship S/B, $S/B = 1.25$; Column charge concentration, $q_c = 0.5\ q_f$, where q_f is the bottom charge concentration.

If contour blasting is to be carried out, the spacing between blastholes is calculated from:

$$S_c = KD_1$$

where K varies between 15 and 16. The ratio S/B should be 0.8.

The lineal charge concentration is determined in function with the drilling diameter. With blastholes of a caliber lower than 155 mm, the following equation is used:

$$q_{lc} = 90 \times D_1^2$$

where: D_1 is expressed in m.

Example of application

If a mine drift is to be excavated in rock ($c = 0.4$) by means of blasts with parallel blastholes and four section cut, knowing that the geometric dimensions and drilling data are:

- Tunnel width $AT = 4.5$ m,
- Abutment height 4.0 m,
- Height of arch 0.5 m,
- Relief hole diameter $D_1 = 102$ m,
- Drilling diameter $D_1 = 45$ mm,

Table 22.1.

Breaking direction of the stoping holes	Fixation factor f	S/B relationship
Upwards and horizontally	1.45	1.25
Downwards	1.20	1.25

- Lookout angle of the contour blastholes $\gamma = 3°$,
- Angular deviation $\alpha = 10$ mm/m,
- Collaring error $e' = 20$ mm.

The explosive to be used has a Relative Weight Strength with respect to ANFO of 1.09 (109%) and the available cartridges have a diameter of 25, 32 and 38 mm, which give lineal charge concentrations for a density of 1.2 g/cm^3, of 0.59, 0.97, and 1.36 kg/m respectively.

a) *Advance*

$L = 3.2$ m and $X = 3.0$ m.

b) *Cut*

First section

$B = 1.7 \times D_2 = 0.17$ m
$B_1 = 0.12$ m
$q_1 = 0.58$ kg/m $\rightarrow$ 0.59 kg/m, with $d = 15$ mm
$T = 10 \times D_1 = 0.45$ m
$A_h = \sqrt{2} \times B_1 = 0.17$ m
Charge per blasthole $Q_b = 1.59$ kg.

Second section

$A_h = \sqrt{2}\,(0.12 - 0.05) = 0.10$ m
For $d = 25$ mm $B = 0.17$ m; $d = 32$ mm $B = 0.21$ m; $d = 38$ mm $B = 0.25$ m
As $B_2 \leq 2\, A_h$, cartridges of 32 mm are chosen.
$B_2 = 0.16$ m
$T = 0.45$ m
$A_h = \sqrt{2}\,(0.16 + 0.17/2) = 0.35$ m
$Q_b = 2.62$ kg.

Third section

$A_h = \sqrt{2}\,(0.16 + 0.17/2 - 0.05) = 0.28$ m
For larger diameter cartridges $q_l = 1.36$ kg/m
$B = 0.42$ m
$B_3 = 0.37$ m
$T = 0.45$ m
$A'_h = \sqrt{2}\,(0.37 + 0.35/2) = 0.77$ m
$Q_b = 3.67$ kg

Fourth section

$A_h = \sqrt{2}\,(0.37 + 0.35/2 - 0.05) = 0.70$ m
$B = 0.67$ m
$B_4 = 0.62$ m
$T = 0.45$ m
$A'_h = \sqrt{2}(0.62 + 0.77/2) = 1.42$ m, which is comparable to the square root of the advance, which means that no more sections are needed.

$Q_b = 3.67$ kg.

c) *Lifters*

With $d = 38$ mm $q_l = 1.36$ kg/m
$B = 1.36$ m
$NB = 5$ blastholes
$S_z = 1.21$ m
$S'_z = 1.04$ m
$B_z = 1.14$ m
$l_f = 1.43$ m
$l_c = 1.32$ m

$q_c = 0.7 \times 1.36 = 0.95$ kg/m $\rightarrow$ 0.97 kg/m with $d=$ 32 mm
$Q_b = 3.20$ kg.

d) *Roof contour blastholes*
Cartridges of 25 mm with $q_l = 0.59$ kg/m are used.
$S_{ct} = 15 \times D_1 = 0.68$ m
$B_{ct} = S_{ct}/0.8 - L \times \sin 3° - 0.05 = 0.62$ m
$q_{lc} = 90 \times D_1^2 = 0.18$ kg/m, which is considerably less than 0.59 kg/m
NB = | 4.7/0.68 + 2 | = 8
$Q_{bt} = 1.77$ kg.

e) *Wall contour blastholes*
The length of contour that is left for a height of 4.0 m is:
$4.0 - B_z - B_{ct} = 4.0 - 1.14 - 0.62 = 2.24$ m, with $f = 1.2$ and $S/B = 1.25$ one has:
$B_{ch} = 1.33 - L \times \text{sen } 3° - 0.05 = 1.12$ m
$NB = |\ 2.24\,(1.33 \times 1.25) + 2\ | = 3$
$S_{ch} = 2.24/2 = 1.12$ m
$l_f = 1.40$ m
$l_c = 1.35$ m
$Q_b = 3.2$ kg.

f) *Stoping*
As the side of the fourth section is $A'_h = 1.42$ m and the practical burden of the wall contour blastholes is $B_{ch} = 1.12$ m, the available space for a tunnel width $AT = 4.5$ m is:

$$4.5 - 1.42 - 1.12 \times 2 = 0.84 \text{ m}$$
$$B = 1.21 - 0.05 = 1.16 \text{ m for } f = 1.45$$

However, $B = 0.84$ m will be used owing to the horizontal dimensions of the tunnel.
For the upper blastholes:

$$B = 1.33 - 0.05 = 1.28 \text{ m}$$

but, if $A'_h = 1.42$, $B_z = 1.14$ and $B_{ct} = 0.62$ is subtracted from the tunnel height, the following is obtained:

$$4.5 - 1.42 - 1.14 - 0.62 = 1.32 \text{ m}$$

As the difference is only 5 cm, B is made equal to 1.32 m.
The charge for stoping holes is the same as for wall holes, thus:

$$Q_b = 3.20 \text{ kg.}$$

g) *Summary*
- Cut: 16 blastholes
  (4 × 1.59) + (4 × 2.62) + (8 × 3.67) = 46.21 kg
- Lifters: 5 blastholes (5 × 3.20) = 16 kg
- Roof contour: 8 blastholes (8 × 1.77) = 14.16 kg.
- Wall contour: 6 blastholes (6 × 3.20) = 19.20 kg.
- Stoping: 5 blastholes (5 × 3.20) = 16.00 kg.

Total charge of the blast = 111.6 kg
Tunnel surface = 19.5 m²
Advance = 3 m
Volume of broken rock = 58.5 m³

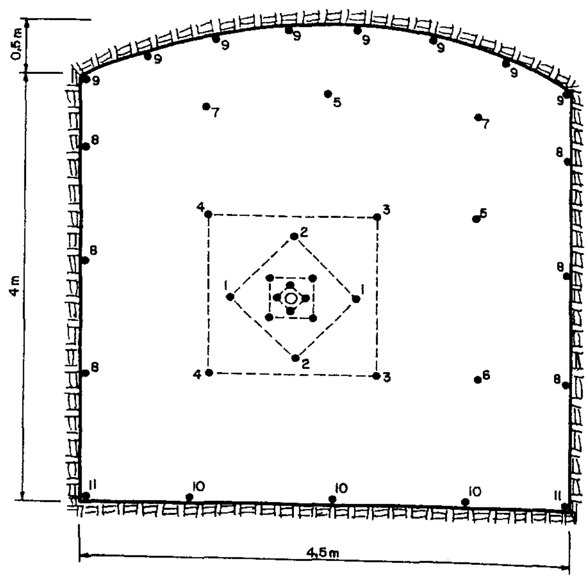

Fig. 22.16. Geometric pattern of the calculated blast.

Specific charge = 1.9 kg/m³
Total number of blastholes = 40
Total length drilled = 128 m
Specific drilling = 2.2 m/m³

Simplified calculation
In order to calculate tunnel blasts with parallel hole cuts in four sections more quickly, the equations shown in Tables 22.2 and 22.3 can be applied:

a) *The cut*

b) *Stoping*
In order to calculate the rest of the blast, start from the burden size B and the charge concentration in the bottom q_f for the explosive and diameter used. The formulas used are:

$$q_f = 7.85 \cdot 10^{-4} \cdot d^2 \cdot \rho$$
$$B = 0.88 \cdot q_f^{0.35}$$

where: d = Diameter of explosive cartridge (mm), ρ = Density of explosive (g/cm³).

Verification of the blast patterns
Once the calculation of the patterns and charges has been done and before doing the blasts, it is interesting to check and contrast the data obtained with the standard or typical results of similar operations. These verifications can be carried out with simple graphics such as those of the Figs 22.17, 22.18 and 22.19, where the powder factor is shown as a function of the tunnel section and drilling diameter, the number of blastholes per round and the specific drilling taken from the two indicated parameters.
The previous graphics refer to blasts with parallel

blastholes and only can be taken as a guide, as many parameters influence the results of the excavation: types of rock and explosives, blasthole size, types of cut, need for contour blasts, vibration limitations, etc. which can cause slight variations in the design parameters.

The final verification of the calculations will be made after the blast. The introduction of the necessary modifications after an analysis of the results in the first trials should be gradual and systematic, even to the point of not drilling the holes to their full length in the first rounds and increasing the advance little by little in each cycle.

22.4.2 *Burn cuts*

In these cuts all the blastholes are drilled parallel and with the same diameter. Some are charged with a large quan-

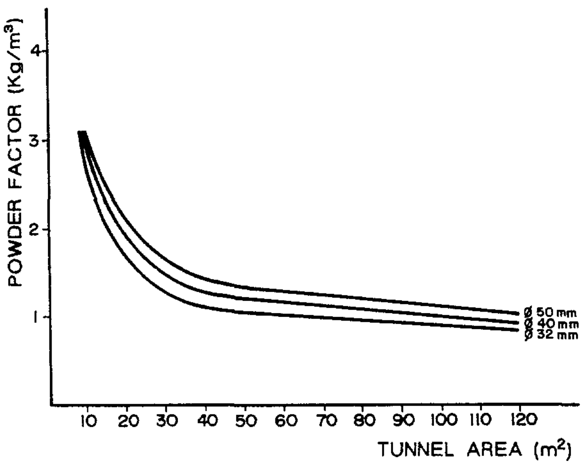

Fig. 22.17. Powder factor in function with the tunnel area (m²) and the blasthole diameters.

Table 22.2.

Section of the cut	Burden value	Side of the section
First	$B_1 = 1.5\,D_2$	$B_1\sqrt{2}$
Second	$B_2 = B_1\sqrt{2}$	$1.5\,B_2\sqrt{2}$
Third	$B_3 = 1.5\,B_2\sqrt{2}$	$1.5\,B_3\sqrt{2}$
Fourth	$B_4 = 1.5\,B_3\sqrt{2}$	$1.5\,B_4\sqrt{2}$

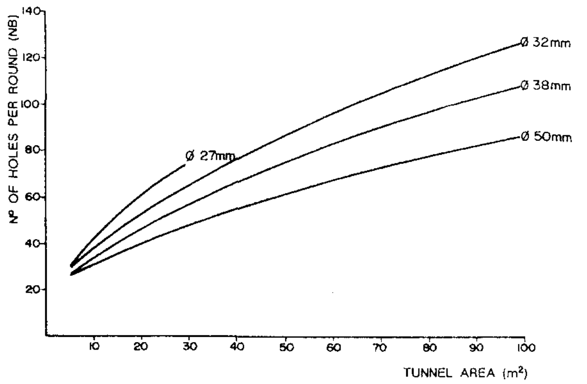

Fig. 22.18. Number of blastholes per round in function of the area.

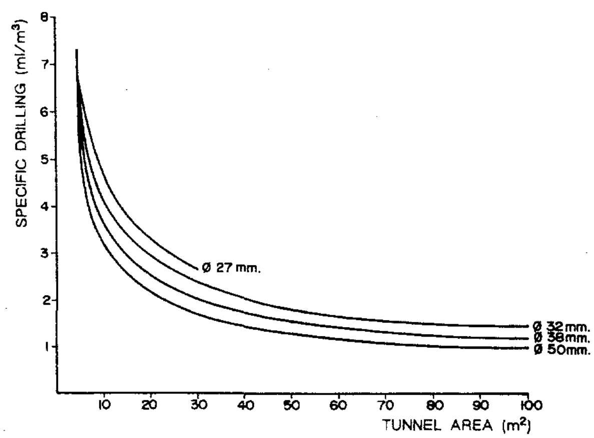

Fig. 22.19. Specific drilling in function with the tunnel area and drilling diameter.

Photo 22.2. Manual drilling in a drift face.

Table 22.3.

Part of round	Burden (m)	Spacing (m)	Length bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	B	1.1 B	L/3	q_f	q_f	0.2 B
Wall	0.9 B	1.1 B	L/6	q_f	$0.4\,q_f$	0.5 B
Roof	0.9 B	1.1 B	L/6	q_f	$0.36\,q_f$	0.5 B
Stoping						
Upwards	B	1.1 B	L/3	q_f	$0.5\,q_f$	0.5 B
Horizontal	B	1.1 B	L/3	q_f	$0.5\,q_f$	0.5 B
Downwards	B	1.2 B	L/3	q_f	$0.5\,q_f$	0.5 B

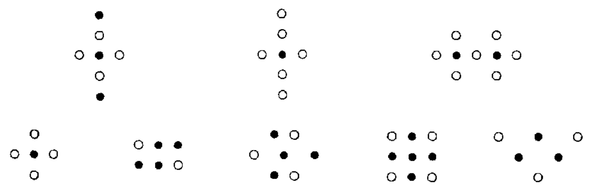

Fig. 22.20. Examples of burn-cuts.

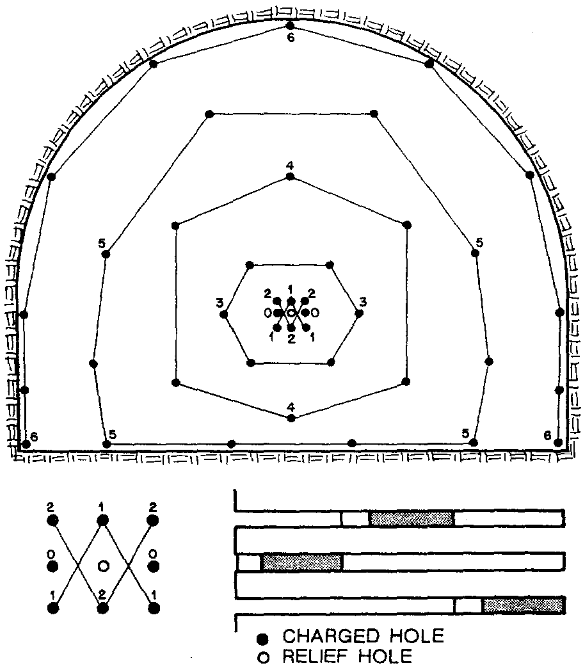

Fig. 22.21. Drift driving with the Sarrois cut.

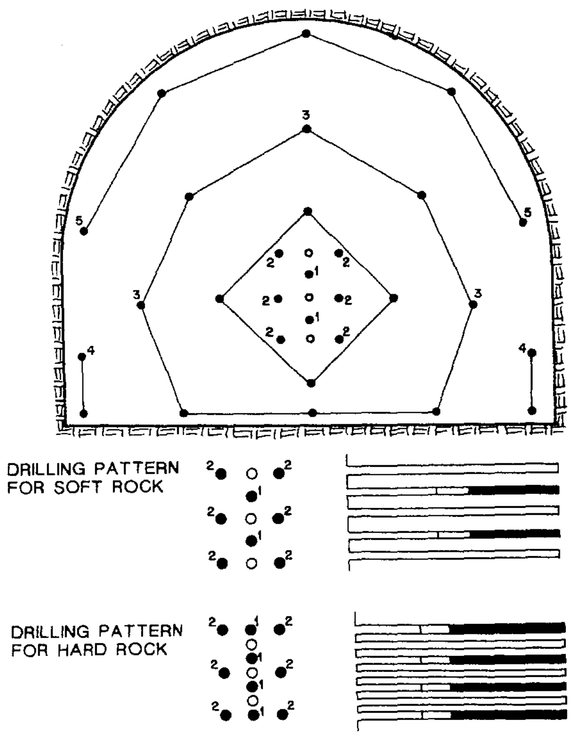

Fig. 22.22. Swedish Cut

tity of explosive while others are left empty. As the charge concentration is high, the fragmented rock is sinterized in the deep end of the cut, without presenting optimum conditions for the outcome of the round as happens in spiral cuts. The advances are reduced and do not surpass 2.5 m per round, Fig. 22.20.

One of the burn-cuts that is used in drift advances of coal mines is called *Sarrois Cut*, which is composed of eight charged blastholes and one empty one. With the drilling diameter of 38 mm, the distance between the axes of the blastholes goes from 10 cm in hard rock up to 20 cm in soft rock. This cut is used in depths of up to 2.5 m, with a high powder factor. The charges are designed as shown in Fig. 22.21, avoiding flashover in each of the blastholes with different delay timings and generally using clay plugs for stemming.

The projection of broken rock reaches a length of 5 to 6 meters from the new face and the advances oscillate between 80 and 95%.

Finally, another cut that is also used in coal mines, above all in the north of Spain, is the one called *Swedish*, where the blasthole placement, according to the type of rock, is shown in Fig. 22.22.

For a diameter of 38 mm, the distance between the vertical rows is 20 cm, the vertical separation between blastholes of the two lateral rows is 30 cm and the vertical distance between charged and empty blastholes is from 10 to 15 cm, according to the compressive strength of the rock.

The broken rock projection is better than with the Sarrois cut, although, on the contrary, the powder factor is lower. The advances oscillate between 90 and 100% of the depth and the drilling must be precise.

22.4.3 *Crater cuts*

This type of cut was originally developed by Hino in Japan, taking advantage of the cratering effect that the explosive charges concentrated in the bottom of the blastholes produced upon the nearest free surface.

This method is applied more in shaft excavations than for tunnels, although some specialists such as Hagan have recently suggested their use by placing the concentrated charges in one or various central blastholes of large diameter and distributing the stoping blastholes around the rest of the section with different charge lengths.

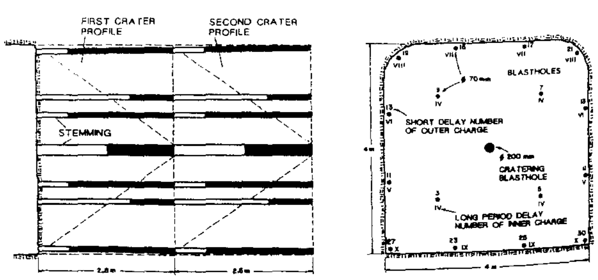

Fig. 22.23. Double crater cut using central blastholes of 200 mm (Hagan, 1981).

As the advance per round is not very large, it has been suggested that the cratering be carried out with double depth blastholes, decking and stemming the charges, Fig. 22.23.

22.4.4 *Angled cuts*

This group of cuts is used less each day because of the difficulty in drilling the holes. The advantage is a lower consumption of explosives because there is better utilization of the free face surface and the possiblity of orientation towards the visible discontinuities in the section.

The following explains the most common angled cuts.

V Cut

In these cuts in wedges or *V*, the advances are around 45 to 50% of the tunnel width. In wide tunnels, the advances are affected by the deviations of the blastholes, which is usually near 5%. Therefore, in a 5 m long blasthole, the deep end could have up to a 25 cm deviation, which might cause sympathetic detonation of the adjacent charge.

The bottom angle should not be less than 60°, because the charges would be too confined and more explosive would be necessary to achieve adequate fragmentation.

The mean parameters in the cut design, in function with the drilling diameter *D*, are the following:

- Total height of the cut $H_c = 46\,D$.
- Burden $B = 34\,D$.
- Bottom charge concentration $q_f = 990\,D^2$ (*D* in m).
- Length of bottom charge $l_f = 0.3\,L$.
- Column charge concentration $q_c = 0.5\,q_f$.
- Length of stemming $T = 12\,D$.
- Number of wedges in the vertical sense 3.

The *cut-spreader* blastholes, which are also drilled inclined with respect to the tunnel axis, Fig. 22.24, are placed according to the following equations:

- Burden $B = 24\,D$.
- Bottom charge concentration $q_l = 990\,D^2$.
- Length of bottom charge $l_f = 0.3\,L$.
- Column charge concentration $q_f = 0.4\,q_f$.
- Stemming length $T = 12\,D$.

The burden value should fulfill the following condition $B \leq 0.5\,L - 0.2$ m, which suggests that in shallow blasts the burden should be reduced.

The cut holes, or even the nearest *cut-spreader* holes should be fired with milisecond delay detonators and the rest with delay detonators.

The drilling patterns for the stoping, lifter and contour areas are calculated in the same manner as indicated for parallel blasthole cuts.

Fan cut

This type of cut was widely used several years ago, but it is not favored nowadays because of its complicated drilling.

The blasthole patterns and charges for the cut are calculated as in the 'V' cut, Fig. 22.25.

The *cut-spreader* blastholes are measured by the following equations:

- Burden (it should fulfill $B < L - 0.4$) $B = 23\,D$.
- Height of cut $H_c = 42\,D$.
- Concentration of the bottom charge (*D* and m) $Q_f = 990\,D^2$.
- Length of bottom charge $l_f = 0.3\,L$.
- Concentration of bottom charge $q_c = 0.4\,q_f$.

The initiation sequences of the cut and *cut-spreader* should be carried out with milisecond delay detonators.

The fans can be horizontal, as the former, or drilled upwards or downwards, Fig. 22.26.

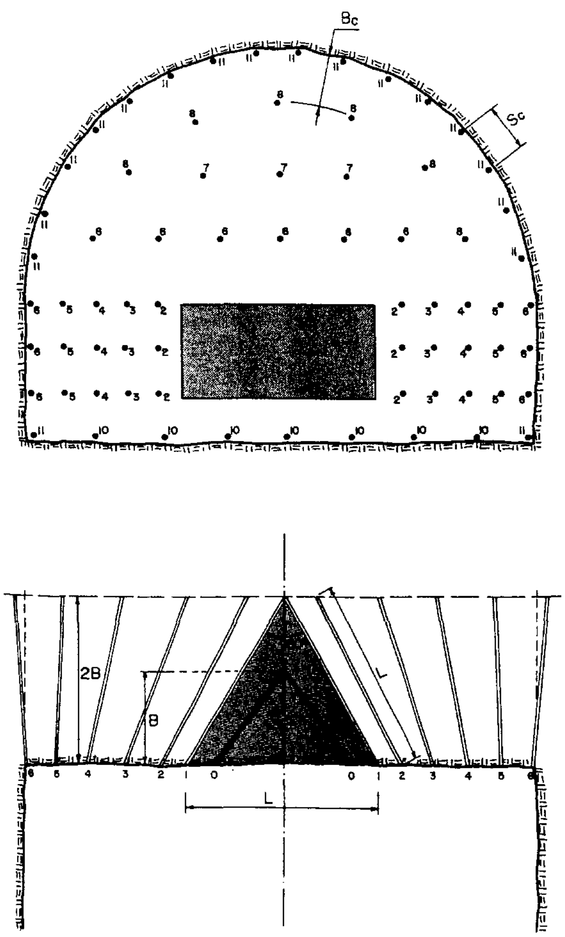

Fig. 22.24. 'V'-cut blast.

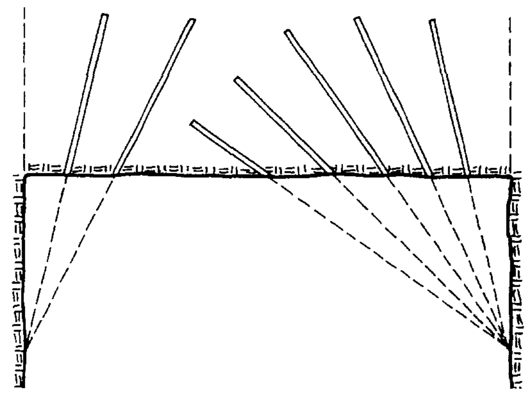

Fig. 22.25. Horizontal fan-cut.

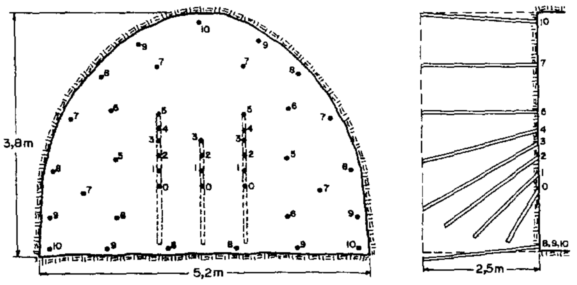

Fig. 22.26. Fan-cut floor blast.

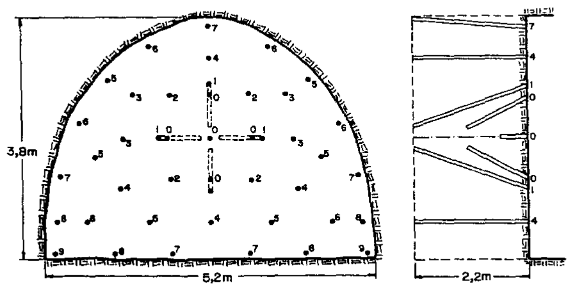

Fig. 22.27. Instantaneous pyramidal cut blast.

Instantaneous cuts

One of the variations of the *V* cut is to drill a cluster of adjacent blastholes and fire all the charges simultaneously. Advances of around 80% of the tunnel width can be obtained.

One of the inconveniences of these cuts is the great projection of the broken rock which disperses it at a considerable distance from the tunnel face.

Between the variations that exist, the pyramidal cut with one or two sections is worth mentioning, Fig. 22.27.

22.4.5 *Drifts with coal beds*

The advance blasts for drifts with coal beds on the face can be quite varied, according to the excavation sections, bed strengths, inclination, face disposition, etc., therefore only a few general considerations will be made.

The drilling patterns should be parallel to the stratification, with all the charged blastholes breaking towards the free space created in the coal bed. These cuts or cavities, can be drilled manually if the coal is soft or, as is usually done, by firing some blastholes upon the coal itself with a low delay number, Fig. 22.28. This last procedure has the inconvenience of mixing the coal with the waste, preventing its use, but it is the method which gives the most advance.

In the beds with sudden detachments of firedamp, it is recommended to leave an uncharged hole to degasify the coal. According to labor classification, because of the existance of gases, dust and other explosive or inflamable substances, the maximum explosive charges per blasthole will be determined along with the type of explosive, the length of the round and the maximum delay timings between holes.

Spanish legislation, through the ITC 10.4-10, establishes a classification of the labors in mines of second and third category, and for those labors in which the presence of gases, dust or other explosive or inflammable substances is possible, as indicated in Table 22.4.

In the mentioned Table, the type of explosive, the maximum amount per hole, the type of detonator and the maximum deviation of the blast are indicated.

22.4.6 *Drifts in salt mines*

In the sedimentary beds of soft minerals such a salts, potash, etc., the drifts to prepare the cuts can be excavated

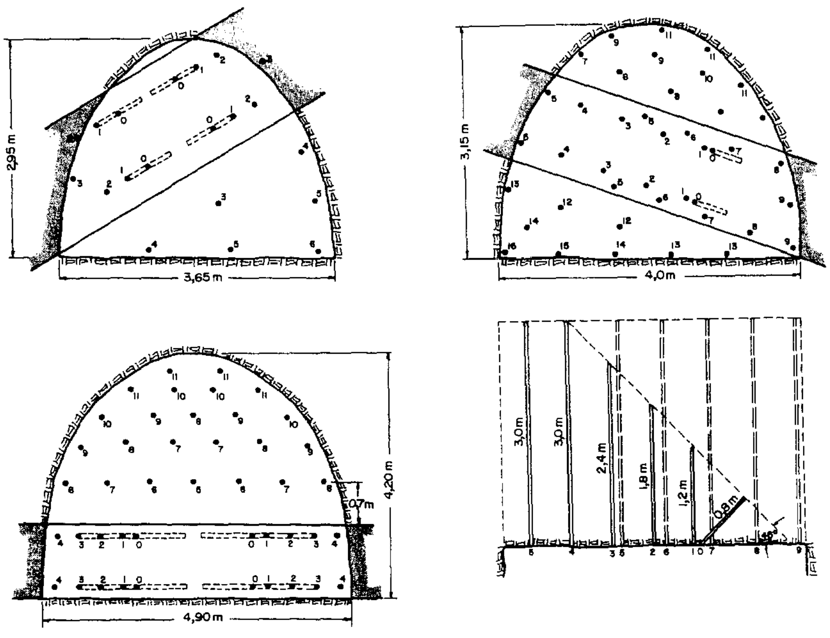

Fig. 22.28. Blasts in drifts with coal beds.

Table 22.4.

LABOR	TYPE	CONDITIONS	EXPLOSIVE	MAXIMUM AMOUNT PER HOLE IN GRAMS	DETONATOR	MAXIMUM DURATION OF BLAST	OBSERVATIONS
LABOR IN ROCK - ROCK DRIFT - CROSSCUT	1ª	- That the face does not go through coal - That the holes do not go through coal - In horizontal or upward work the fire damp concentrations at the face and last 100 m be under 0,5% - If the work is carried out with air from other labors that contains fire damp it cannot 60 over 1% - In upward labors, the maximum concentration can never de over 0,5% - In the last 30 m, there cannot be any accumulation of coal or dust, nor work faces or coal haulage drifts. - That in the last 30 m the layers of incovered coal be under 10% of the total surface of the labor in tha area and that the last layer be at least at 3 m.	UNLIMITED	NO LIMIT	DELAY OR MILLISECOND DELAY		
ROCK DRIFT	2ª	- That the face not go through coal - That the not of holes that go througt coal be under 1/5 of the total. - Mixed labors of coal and rock in which the total area of coal does not exceed 10% of total surface or those in which the number of holes which go through coal do not exceed 1/5 of total	SAFETY, 9 SAFETY, 9 bis SAFETY , 12	2.000	DELAY OR MILLISECOND DELAY	5 s	BLASTHOLES THAT HAVE CUT THROUGH COAL CANNOT BE CHARGED
ORE DRIFT	3ª	- Mixed labors of coal and rock in which the total area of un covered coal be over 10% of total surface. - That the number of holes that went througt coal be over 1/5 of total	SAFETY, 9 SAFETY, 9 bis SAFETY, 12	1.000	MILLISECOND DELAY MAXIMUM 7 nº of 20 ms 5 nº of 30 ms	125 ms	
WORK FACE	4ª	- Labors over bed/which are ventilated by the main air lines	SAFETY, 9 SAFETY, 9 bis SAFETY, 12	500	MILLISECOND DELAY 7 nº of 20 ms 5 nº of 30 ms	125 ms	
			SAFETY, 20 SR SAFETY 18 SR	2.000	MAXIMUM DELAY	5 s	WITH AIR VELOCITY V ≥ 0,5 m/S
			SAFETY 30 SR	2.500	MILLISECOND DELAY	500 ms	WITH AIR VELOCITY V < 0,5 m/S
	5ª	- Labors over bed/not ventilated by main air flow	SAFETY 20 SR SAFETY 18 SR SAFETY 30 SR	2.000 2.500	MILLISECOND DELAY MILLISECOND DELAY	↙ 500 ms ↖ 500 ms	
	6ª	- Breakage of coal masses/unventilated.	SAFETY 20 SR SAFETY 18 SR SAFETY 30 SR	1.500 2.000	MILLISECOND DELAY MAXIMUN 7 nº of 20 ms 5 nº of 30 ms	125 ms 125 ms	

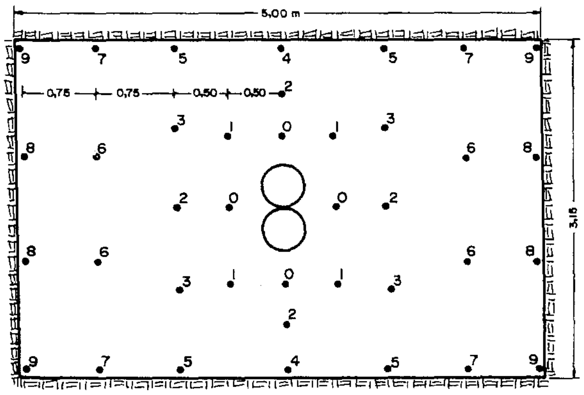

Fig. 22.29. Drilling pattern with spiral cut composed of two large diameter blastholes.

by drilling and blasting, as well as with continous miners.

The drilling is usually carried out with jumbos that are capable of opening cut holes with a diameter of up to 420 mm and a depth of 7 m, Fig. 22.29. The rest of the blastholes with diameters of 37 and 42 mm, are usually drilled parallel to the axis of the tunnel and with the same depth as those of the cut. The explosive charging should be mechanized because, otherwise, the great length of the holes makes it very difficult.

If the modus operandi is chambers and pillars, the opening of the chambers can be done with a central drift and lateral stoping holes for widening. All drilling is done horizontally, as indicated in Fig. 22.30.

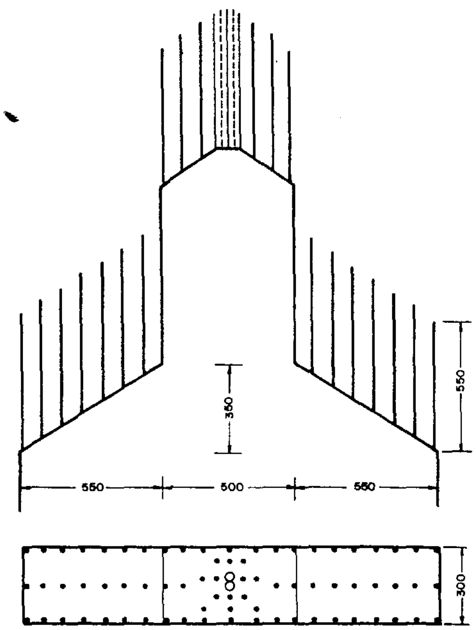

Fig. 22.30. Advance sequence for an excavation of chambers and pillars.

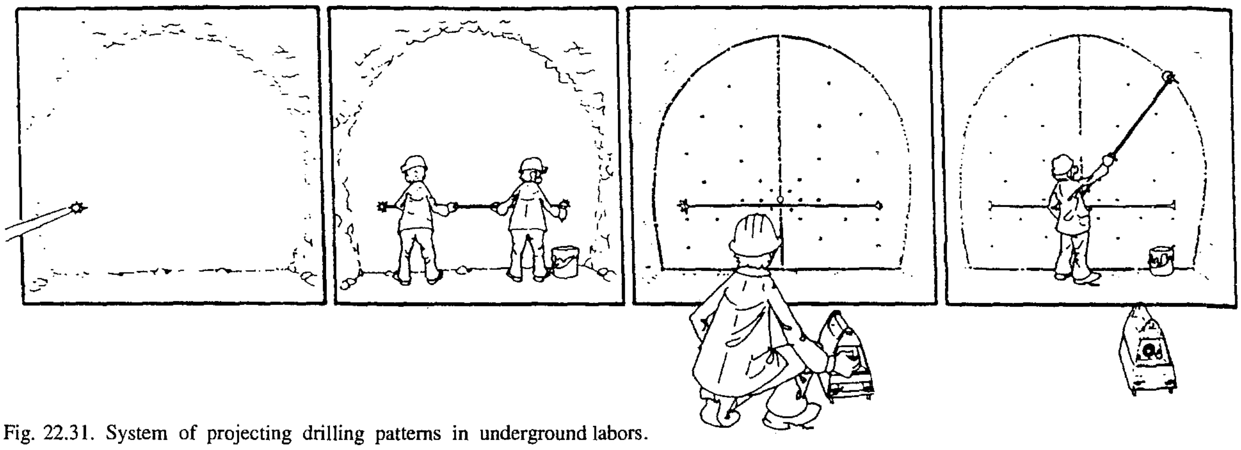

Fig. 22.31. System of projecting drilling patterns in underground labors.

22.5 EQUIPMENT FOR MARKING OUT DRILLING PATTERNS

Amongst the auxiliary equipment for marking out the collars of the blastholes in underground labors, the drilling pattern projectors are available. These units are battery run and can be placed on a tripod on the ground or upon a vehicle. Once the direction of the tunnel or shaft has been marked, two reference points are indicated on the face and, following this, the hole pattern of the round is projected on the rock. The obtained image is focused and the collaring points of the blastholes are marked with paint, Fig. 22.31.

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CHAPTER 23

Shaft sinking and raise driving

23.1 INTRODUCTION

In any project that profits from the underground such as ore recovery or the opening of large chambers, an essential part of the development work is the driving of shafts and raises. These can be either vertical or inclined, and are characterized by their lineal design and drilling difficulties.

During the last decades numerous methods have been developed, mechanizing the work by means of special techniques and drilling aparatus, increasing advances and production and improving safety conditions.

23.2 SHAFT SINKING

The methods for shaft sinking can be divided in three groups:

- The benching method (half-bottom method).
- The spiral method.
- The full-bottom method.

23.2.1 *The benching method*

This method is particularly suitable for square or rectangular sided shafts. It consists in drilling one half of the shaft cross section with a fan, leaving the other half (lower) as a sump for water or spoil, if necessary, or as a free cavity.

Blasting is similar to that in small benches with a free face, displacing the broken rock towards the space left by the former round, Fig. 23.1.

Drilling is usually done by hand or with pneumatic hammers.

23.2.2 *The spiral method*

This was first used in Sweden and consists in excavating downwards in a spiral, with a height that depends upon the diameter of the shaft and the type of ground to be fragmented. Within each cut, a section of the spiral is blasted with an angle large enough to make the time necessary to carry out a complete cut coincide with a whole unit of the available work time, Fig. 23.2.

The blastholes in each radius are drilled parallel and of the same length, as there always has to be a free face which descends with each position.

Apart from the advantages of the yield and costs of this method, other interesting aspects are that the length of the cut can be synchronized in function with the work plan, the drilling patterns and blasting systems are simple and the drilling personnel does not have to be very experienced. Lastly, as the broken rock always remains in the deepest part, the loading equipment works with a high productivity.

23.2.3 *The full-bottom method*

The full-bottom method is used frequently in shaft sinking, as it suits either rectangular or round sections shafts.

There are various techniques for blasthole placement because, as happens in drifting and tunnelling, it is necessary to create a free surface with a few blastholes unless there is a large diameter pilot hole available or expansion raises. The face is opened with plough cuts, V-cuts or cone cuts and with relief blastholes, Fig. 23.3.

V-cuts are used in rectangular section shafts. The planes of the dihedrals formed by the blastholes that are inclined between 50 and 75° should be parallel to the discontinuities, in order to use them to advantage during breakage.

Cone cuts are used most in round-section shafts due to, on one hand, the ease with which blasthole drilling can be mechanized and, on the other, the lower powder factor when compared with the plough cut. The holes are placed so as to form several inverted cone areas in the central part, as shown in Fig. 23.4.

The pull of the rounds, as well as the number of blastholes, depends upon many parameters such as: type of ground, diameter of the explosive charges, the blasting pattern, type of cut, work plan, and, above all, the section of the excavation.

Set rules cannot be given for the two design parameters mentioned because they need to be adjusted in each case. For example, the number of blastholes needed for the driving of several shafts in South Africa, with 32 mm diameter charges, can be estimated by using following equation (Espley-Jones and Wilson, 1979):

$$NB = 2\,D_p^2 + 20$$

where: NB = Number of blastholes, without including those of the perimeter, if contour blasts are carried out, D_p = Shaft diameter (m).

As to the drilling depth in each round, Wild (1984) recommends starting from the values indicated in Fig. 23.5.

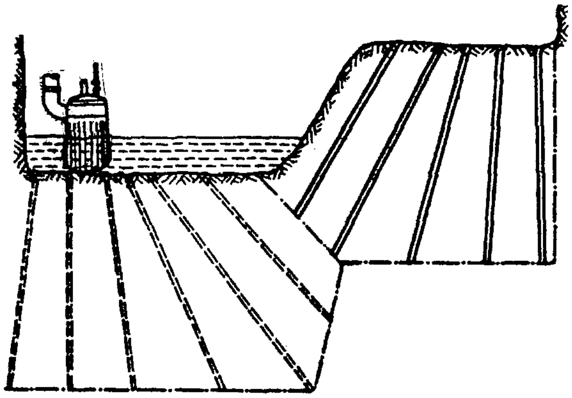

Fig. 23.1. The benching method.

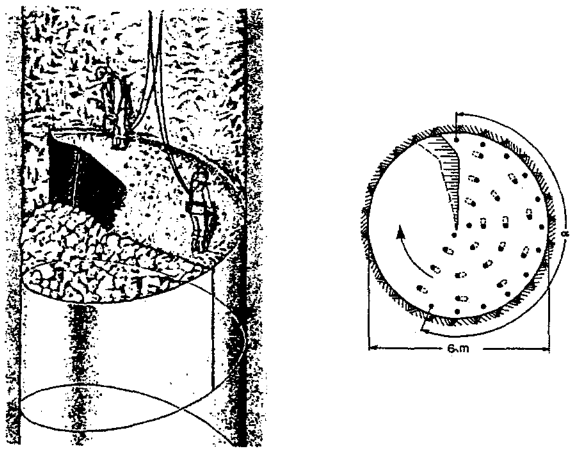

Fig. 23.2. The spiral method.

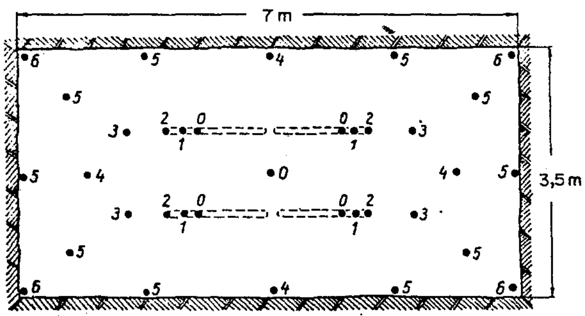

Fig. 23.3. Drilling pattern in a rectangular-section shaft.

The same author provides a graphic on the powder factor in function with the dimensions of the shafts, Fig. 23.6

The plough cut works as in tunnels and drifts, Fig. 23.7. The results obtained up to now are interesting, giving the additional advantage of easier drilling.

A variation of the former consists in blasts with a large diameter central blasthole or raise. In these cases, there is a more efective free face which favors breakage and rockthrow, as well as loading, Fig. 23.8.

If cartridged explosives are used, the relationship between the blasthole diameter and the caliber of small cartridges should be around 1.2 to 1.25, or have a space of about 10 mm for the larger ones. The use of bulk slurries is ideal for reducing the number of blastholes or for taking maximum advantage of the drilling.

The detonators are usually connected in parallel, placing the circuits in ring shape, Fig. 23.9.

23.3 RAISE DRIVING

Raises are excavations with reduced dimensions and an inclination that is over 45°. Raise driving is a typical operation in mining and the lengths can vary, up to more than 100 m. Raises are used to unite drifts on different levels closing the ventilation circuits, for the passage of ore and spoils, for the initial openings of sublevel stoping, etc. Raise driving is also frequent in civil engineering, especially in hydraulic plants and underground warehouses.

The driving of raises has been, up until recently, one of the most difficult operations in rock breakage by drilling and blasting, until the long-hole method came into use.

Raise driving is classified in two large groups according to the drilling method used, either upward or downward:

– *Upward Drilling.* Done by hand with a Jora lift or with the Alimak platform.

– *Downward Drilling.* Long-holes with pilot hole cut, with crater cut, 'VCR' cut (Vertical Crater Retreat), and the full face method.

23.3.1 *Methods of upward drilling*

These methods were the traditional and only ones in existance until the long-hole method appeared.

Classical manual method

This is applied in small operations where the amount of work to be carried out does not justify investing in special equipment and the raises are not very long.

The method consists in assembling and disassembling an inner wooden platform, simultaneously with the advance of the excavation, which not only acts as support but is also a work platform from which the blastholes are drilled with hand-held jackhammers and pushers. These structures are installed from service ramps as shown in Fig. 23.10.

The blastholes are usually placed in *V* or in fan with pulls per round of 1.5 to 2 m and initiation sequences as shown in Fig. 23.11.

This method is competetive in small mines but the work conditions are difficult and require very experienced personnel.

JORA lift method

This system was discussed earlier in the chapter on special drilling equipment. It consists in a lift that is suspended by a rope that passes through a pilot hole driven previously along the line of the raise.

The blastholes are usually drilled parallel using the

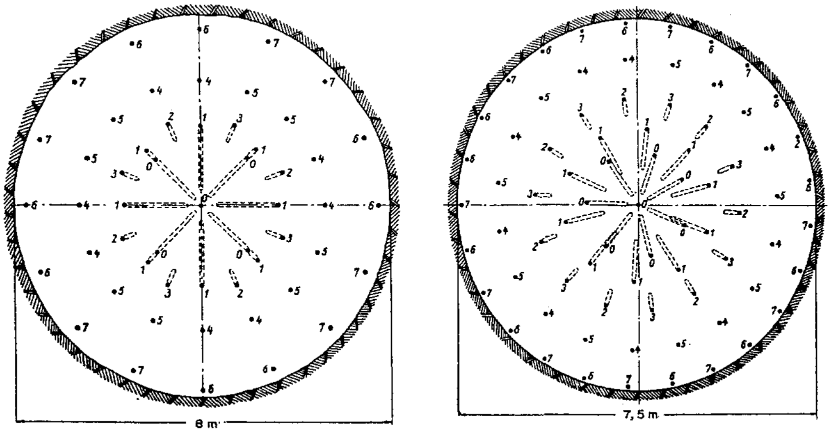

Fig. 23.4. Cone-cut drilling patterns.

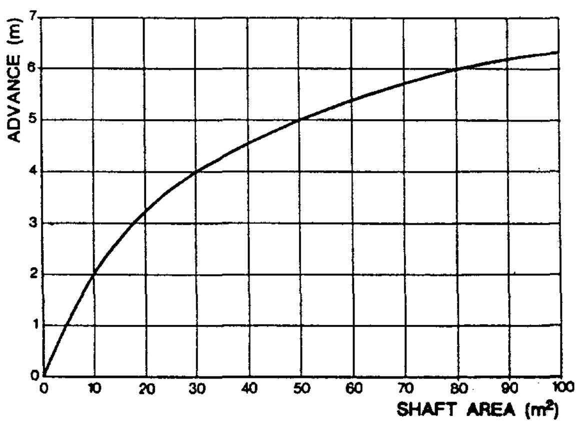

Fig. 23.5. Round depths as function of the shaft section.

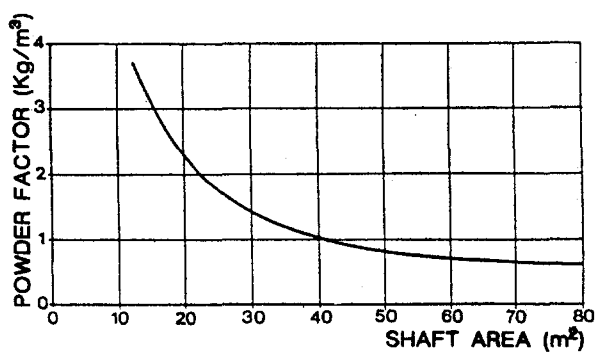

Fig. 23.6. Powder factor as function of the shaft section.

central hole with the largest diameter as cut hole, blasting 3 to 4 m per round, Fig. 23.12.

The central blasthole, apart from serving as cut hole in the blast, also provides ventilation.

Alimak platform method

This consists in a work platform that slides along a rack and pinion rail that is attached to the raise wall by expansion shell bolts.

Parallel blastholes are drilled with heavy hammers and pushers, getting advances per round of up to 3 m. In Fig. 23.13 a typical firing pattern is indicated.

Once the round is fired, the bottom of the raise is ventilated by injecting compressed air and water. Later the platform is lifted and the roof is scaled, reinitiating the work cycle again.

23.3.2 *Methods with downward drilling*

The previous methods have the following inconveniences:

– The complete cycles are very long, drilling, blasting, ventilation and scaling, which means low productivity due to lost time.

– The need for an elevated number of highly qualified personnel.

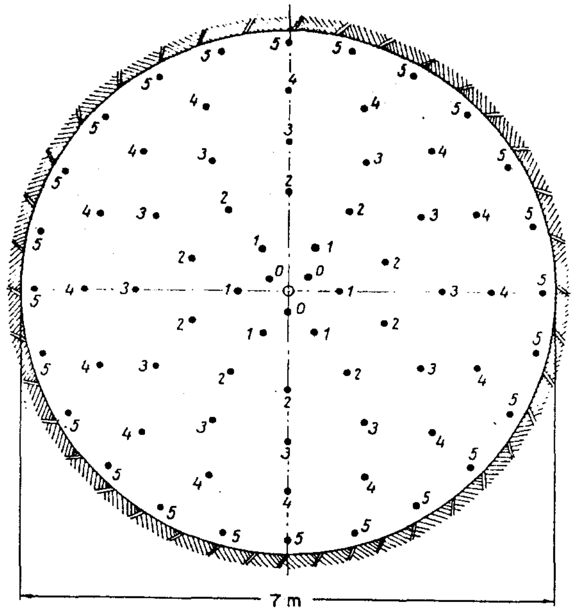

Fig. 23.7. Plough-cut holes.

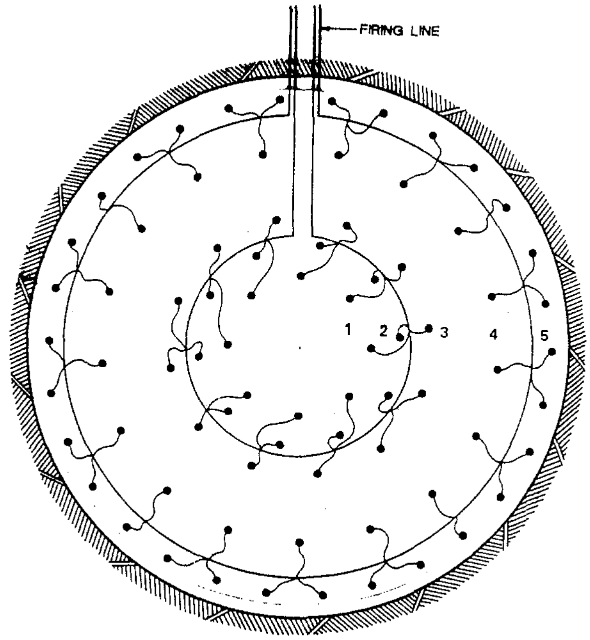

Fig. 23.9. Ring connections in a shaft blast.

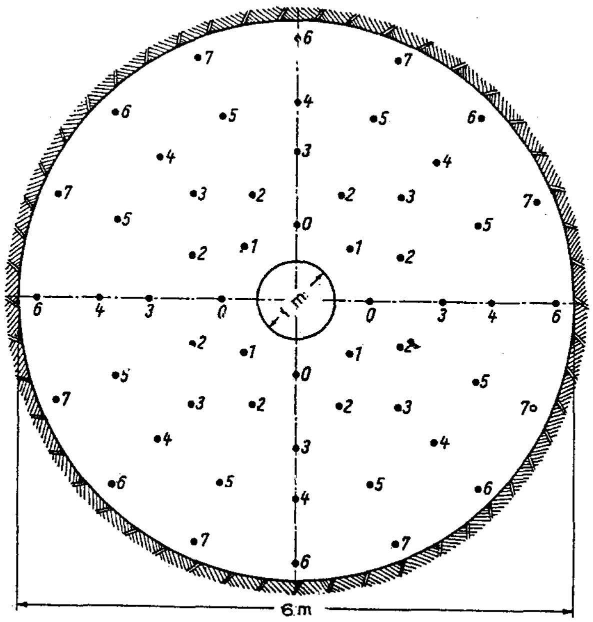

Fig. 23.8. Blast with central raise.

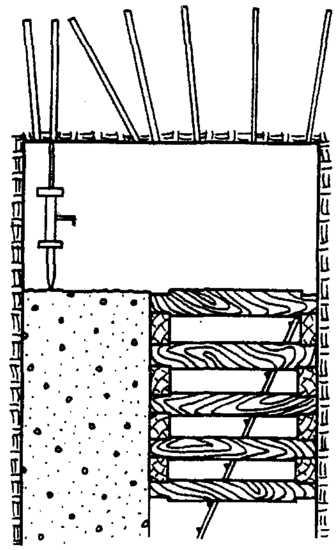

Fig. 23.10. Manual raise building.

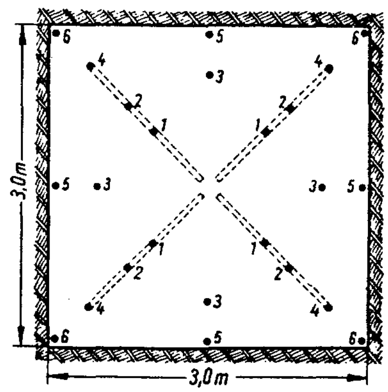

Fig. 23.11. Drilling pattern and initiation sequence in manual raise building.

– Safety and hygiene conditions are not very good.
– The cost of the operation is usually high.

To solve these problems, in the decade of the seventies various experiments were started by drilling the blastholes to the total length of the raise and later firing the blasts in stages with hanging charges.

Logically, these methods require great drilling accuracy, which motivated the manufacturers to design special equipment and accessories. Nowadays, the deviations can be maintained below 2% with top hammer drills and below 1% with down the hole drills.

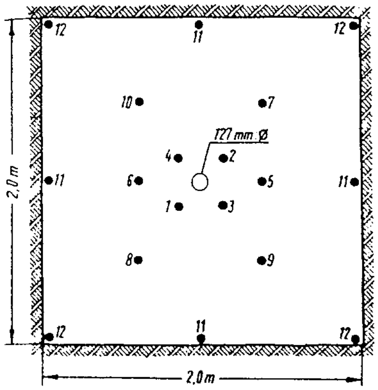

Fig. 23.12. Design of a blast with a large diameter central blasthole.

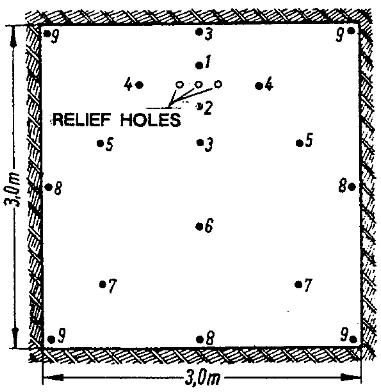

Fig. 23.13. Pattern of a shot with parallel blastholes.

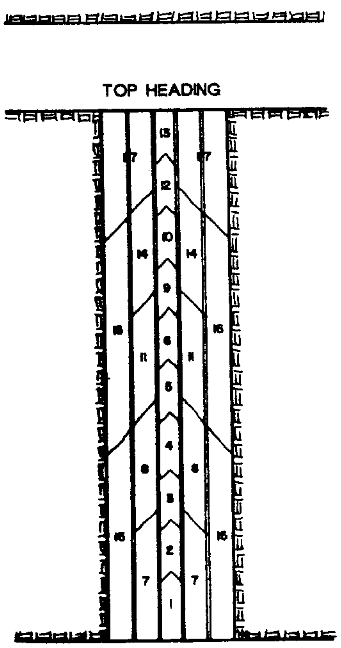

Fig. 23.14. Advance sequence in a raise.

Relief blasthole cut method

This technique, which was developed in tunnelling and drifting, was the first to be applied in raises with long-holes.

The holes are drilled with top hammer rigs with diameters between 51 and 75 mm, widening the central pilot holes up to 100 or 200 mm in diameter.

The blastholes are placed, generally, in square-sections which are fired in stages in spans of 2 to 4 m; first the cut zone and then those of the stoping zones, Fig. 23.14, although, if the engineer is experienced, it is possible to do a full-section blast by using milisecond delay detonators in the cut and delay detonators in the stoping holes.

Excessive confinement of the charges should be avoided in order to prevent sinterization of the rock. The lower ends of the blastholes are closed with any of the methods described in the Chapter on Blastings and Production in Underground Mining, and it is recommended to use water as stemming in order to eliminate stoppage.

The patterns for the blastholes of the cut can be calculated with the following equation:

$$S = D_1 + 1.25 \times D_2$$

where: S = Spacing between blastholes, D_1 = Diameter of the charged blastholes, D_2 = Diameter of the central relief hole.

And the blastholes of the stoping sections with the following equation:

$$S = 10 \times D_1 + 500$$

In each section it is suggested that the burden not be larger than the width of the central hole towards which each blasthole breaks.

The lineal charge concentrations in the cut and stoping blastholes are determined with the two following equations:

$$q_1 = 0.03 \times D - 0.85$$

$$q_1 = 0.0125 \times D_1 + 0.26$$

where: q_1 = Lineal charge concentration (kg/m), q_l = Blasthole diameter (mm).

Fig. 23.15 shows the firing pattern for the holes of the cut and the first stoping section in a raise where the central relief hole is of 150 mm.

The contour of the raises can be outlined by smooth blasting and placing the blastholes with an average spacing of 13 D.

Crater cut method

This consists in opening a cavity of approximately 1 m^2 with five blastholes of 65 to 102 mm in diameter and placing the explosive charges so that they perform as in crater blasting, Fig. 23.16.

Once the cut has been accomplished in all its length, the stoping is carried out using the patterns and charges indicated for the previous method.

The configuration and situation of the charges is determined by the Livingston theory:

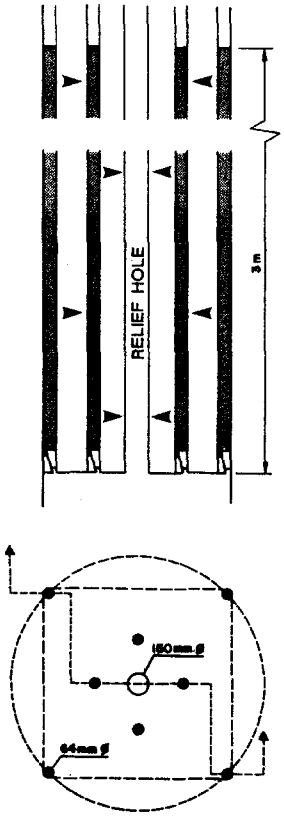

Fig. 23.15. Drilling pattern and initiation sequence in a raise with a large diameter central blasthole.

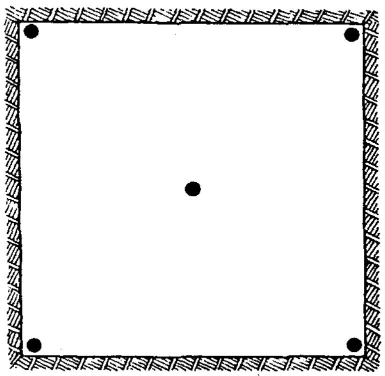

Fig. 23.16. Drilling pattern in the crater cut method.

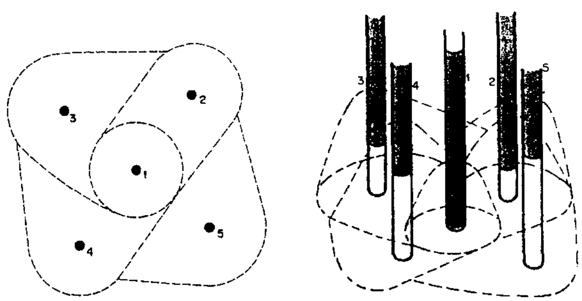

Fig. 23.17. Initiation sequence and distance from the charges to the free face.

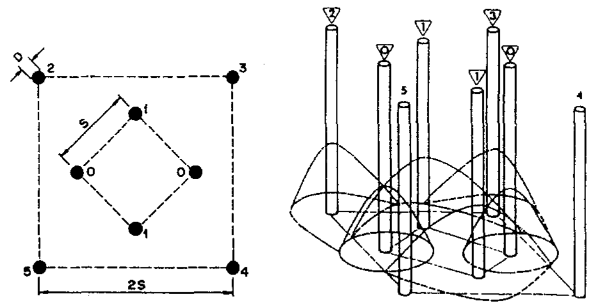

Fig. 23.18. Raise blasting pattern with the 'VCR' method.

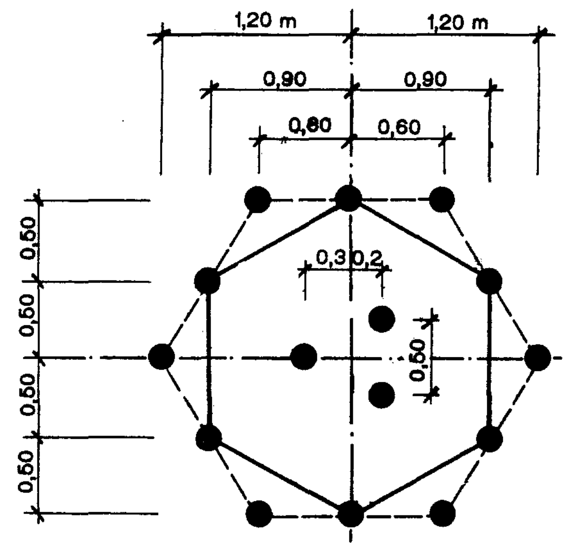

Fig. 23.19. Drilling pattern for driving a raise with a diameter of 2 m, using 165 mm blastholes.

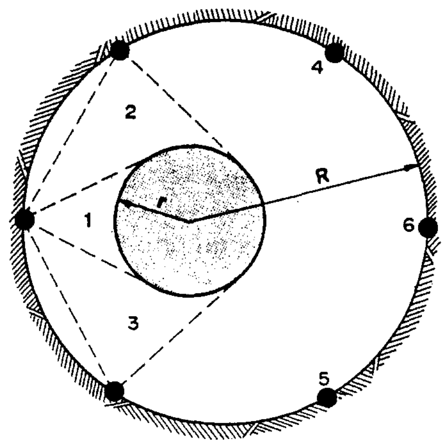

Fig. 23.20. Firing of a blast towards a large diameter pilot hole.

Table 23.1.

	Blasthole Diameter (mm)	
	114	165
Raise section	2.40 × 2.40	3.60 x 3.60
Spacing – S (m)	1.20	1.80
Watergel charge per hole – Q_b (kg)	12	30
Stemming Length – T (m)	1.5	1.8
Advance per round – X (m)	2.10	3.0

– The length of the explosive column should be under 6 D, so that they perform like spherical charges.

– The optimum charge depth is approximately 50% of the critical depth:

$$D_o = 0.5\,D_c$$

– According to the Livingston theory, the critical depth has a value of:

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

Where: E_t = The Strain-Energy factor, Q = The explosive charge (kg).

– The quantity of charge Q in the blasthole for an explosive density of ρ_e, has a value of:

$$Q = \frac{3 \times \pi \times D^3}{2} \times \rho_e$$

and, taking the average value of $\rho_e = 1.3$ g/cm^3, it is proven that the optimum depth in function with the blasthole diameter is approximately:

$$D_o = 13.7\,D$$

D_c is the distance between the free face and the center of gravity of the charge in the central blasthole. In the rest of the holes, the depth increases in intervals of some 10 to 20 cm. The blastholes should not be too close together in order to avoid rock sinterization.

The advantages of the crater cut system in comparison with the plough cut are the following:

– Lower drilling costs as there are fewer blastholes and the central hole need not be widened by reaming, and

– Drilling does not have to be as precise.

'VCR' method

At the same time that the Vertical Crater Retreat method became popular, in metal deposit operations a 'VCR' system of driving raises was developed, based on the same principles as the crater cut method, Fig. 23.18.

In this case the blastholes, with diameters similar to those used in production blasting, are placed in square sections with the all charges at the same height. Table 23.1 shows two real examples given by Lang (1981), using high density watergels.

The advantages that this method has over the preceding one are:

– Lower drilling costs and fewer blastholes.

– Easier charging of explosive.

Table 23.2.

Drilling diameter	165 mm
Distance from charge to chamber roof	1 m
Charge height	1 m
Charge height/Diameter relationship	6
Total explosive charge	21 kg
Type of explosive used	Cartridged watergel
Number of cartridges	3 (7 kg)
Advance per round	3 m

Photo 23.1. Vertical excavation of a stoping hole using the free surface of a raise drilled by a Raise-Borer.

– Raise driving all in one phase which means less drilling, and

– The possibility of using down the hole hammer drilling rigs.

Fig. 23.19 shows the drilling pattern used in the Rubiales Mine for raise driving, with 165 mm (6½″) blastholes. The pattern is composed of two hexagonals and an inner triangle. These raises are used for exploitation of the chambers, shooting the production blasts towards the face opened by them.

In Table 23.2, the most interesting data of these blasts has been compiled.

Full-face method

This consists in opening a pilot hole of 1 to 2 m in diameter with a raise borer and using it as an expansion hole.

The method is carried out in large underground civil engineering projects and in shaft sinking or for driving large section raises.

Its principal advantages are:

– Wide drilling patterns which lowers cost.

– Relatively small explosive charges which means less damage to the remaining rock.

– Possiblilty of driving the raise in one shot.

In Fig. 23.19, the pattern and initiation sequence of a blast with this method are shown. The distance of the first blasthole to the pilot hole should be small because the free face is concave and the rock is in good condition after being opened mechanically.

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CHAPTER 24

Underground production blasting in mining and civil engineering

24.1 INTRODUCTION

The principal methods for underground mining operations are defined in Fig. 24.1.

The criteria which must be kept in mind when selecting a method are those which refer, on one hand, to the morphology of the deposit and the grade distribution, and on the other hand, the geo-mechanic properties of the rock mass, taking into consideration the mineralization as well as the host rock, and also the technical and economical aspects that each one offers in the conditions of the operation in question.

In this chapter, the principal mining methods used today will be revised, placing special emphasis upon those that have been recently developed using large diameter blastholes (100 to 200 mm), enabling a high degree of mechanization to be reached along with high output and low operational costs. In Table 24.1, the mean productivity per day for each of the methods is shown.

Finally, the procedures for excavation large chambers or caves for use of underground space in non-mining applications, such as hydraulic power plants, liquid fuel tanks, toxic and radioactive residues, etc.

24.2 CRATER RETREAT METHOD

24.2.1 *Crater blasting*

The concept and development of crater blasting attributed to C.W. Livingston (1956), opened a new school of thought for better understanding of the phenomenon of blastings and the characterization of the explosives.

A few years later, Bauer (1961), Grant (1964) and Lang (1976) among others, widened the field of application of this theory, converting it into a basic tool for the study of surface as well as underground blastings.

A crater blast is that which is carried out with concentrated spherical or cubic charges and with good approximation using relatively short cyllinder charges that are detonated inside the rock mass to be fragmented.

In Fig. 24.2, the influence of the energy transmitted by the explosive to the rock, depending upon the depth of the charge and the volume of material affected by the blast. When the charge has a very shallow burial (a) most of the energy is transmitted to the atmosphere in form of airblast, up to an excessive depth (c) where all the energy is applied upon the rock, fragmenting it and producing a high intensity vibration. Between the two situations, there will be one that produces a larger crater.

In the cavities formed, three different concentric zones can be found: the aparent crater, the true crater and the mound of fragmented rock, Fig. 24.3.

The mound is subdivided into the zone of complete fragmentation and that of extreme or tensile fragmentation. In blastings with inverted faces, ths crater sizes are influenced by the effect of gravity and the structural characteristics of the rock, forming elongated, elliptic shaped cavities which correspond to the extreme rupture or stressed zones, Fig. 24.4.

The basic parameters for crater blasting are:

– The ratio Length/Diameter of the cylindrical explosive charges should not exceed 6 to 1 so that they perform as spherical charges.

– The burial depths, distance between the center of gravity and the free face, should be the optimum which is determined through practice applying the Livingston theory.

– The drilling pattern is calculated from the optimum depth and maximum volume of the craters.

Livingston determined that a relationship existed between the critical depth D_c, from which the first signs of external action in the form of cracks and fractures are noted, and the weight of the explosive Q, in agreement with the empirical equation:

$$D_c = E_t \times Q^{1/3} \text{ (Equation of Strain-Energy)}$$

where: E_t = Strain-Energy Factor, which is a characteristic constant in each Rock-Explosive combination.

The preceding equation can be written in the following manner:

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

having: D_g = Distance from the surface to the center of gravity of the charge, Δ = Relationship of depths, a dimensionless number equal to D_g/D_c.

The burial depth, where the explosive maximizes the crater volume V is known as the optimum depth D_o, therefore:

$$\Delta_o = D_o/D_C$$

where: Δ_o = Optimum depth relationship.

In order to determine the optimum burial depth, a series of tests will be carried out with attention to the following recommendations:

– The tests will be done on the same type of rock and

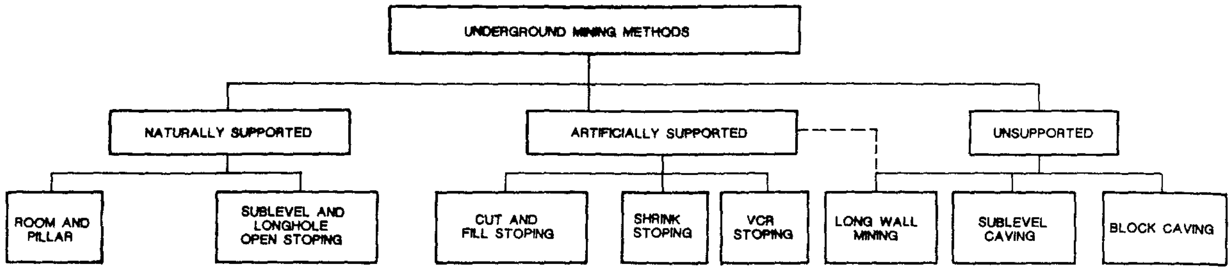
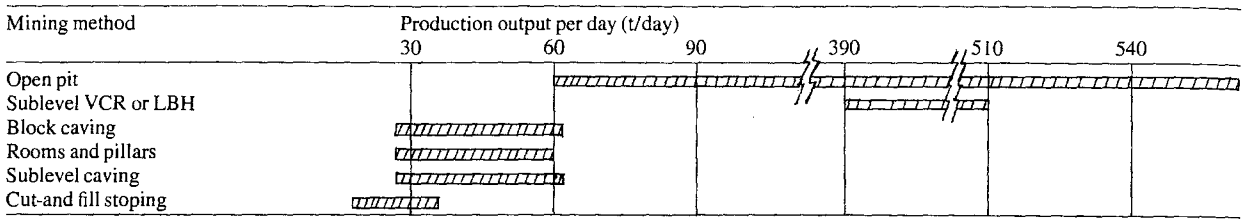


Fig. 24.1. Underground mining methods.

Table 24.1.



with the same explosive that is to be used in the production blasts.

– The blasthole diameters will be as large as possible, for example 115 mm.

– The series of pilot hole lengths will be as ample as possible to allow an ample range of burial depths, for example 15 blastholes between 0.75 and 4 m with increases of 0.25 m.

– The blastholes will be placed perpendicularly to the free face.

– The explosive charges will have a length of 6 D and will be adequately stemmed.

After each test, the volume of the crater will be measured, and afterwards, with all information in hand, the Volume-Depth curve will be established, Fig. 24.5.

In order to better describe the rock breakage procedure and the importance of charge shape, Livingston also proposed the following empirical equation:

$$V/Q = E_t^3 \times A' \times B' \times C'$$

(Equation of the Fragmentation Process)

where: A' = Coefficient of the use of explosive energy, B' = Coefficient of the behavior of the material, C' = Coefficient that takes into account the effects of the geometry of the charge.

If the charges used are spherical and the depth is optimum, the value of B' can be determined by the preceding equations because $A' = C' = 1$, $V = V_o$, and therefore:

$$B' = V/D_c^3$$

As in this type of blastings it is necessary to maximize the efective energy developed per unity of charge length, the explosives used will comply with the following characteristics: high detonation velocity, high density and the possibility of completely filling the cross section of the blasthole.

The ideal explosives for hard rock are the watergels, emulsions and gelatin dynamite, and in medium and soft rocks the lower density and velocity watergels. ANFO has a very limited range of application and is only used in soft rocks.

24.2.2 *Mining method with vertical crater retreat 'VCR' stoping*

This method consists basically in delimiting the ore stope that is to be exploited by a system of shafts directed to a different level, by drilling from the charging level the whole series of blastholes that cover the room and firing them in successive upward rounds with elongated sperical charges $L < 6\ D$, placed in the optimum depth in such a way that the excavated craters overlap to form as regular a roof as possible, Fig. 24.6.

The broken ore is hauled away through cross cuts excavated from the extraction level to the draw point of the stope. The extraction is usually done in a controlled manner, evacuating only the quantity of ore necessary to allow sufficient space between it and the stope ceiling for the next round, avoiding detachments from the side walls which would dilute the ore.

Once the deviations of the drills are under control, as well as the height of the cut in each of the craters excavated in each round, procede to charge the explosive, after plugging the lower part of the blastholes by one of the systems shown in Fig. 24.7.

Once the explosive charge has been placed at the adequate depth with its initiator and/or multiplicator, it will be stemmed to improve the confinement with a length of inert material 12 times the diameter of the blasthole, using fine sand or water to avoid the risk of obstruction, Fig. 24.8.

In this type of work a determined initiation sequence is not necessary, as in bench blasting, owing to the characteristics of the breakage mechanism in crater blasts. However, when there are charges under the mean level of

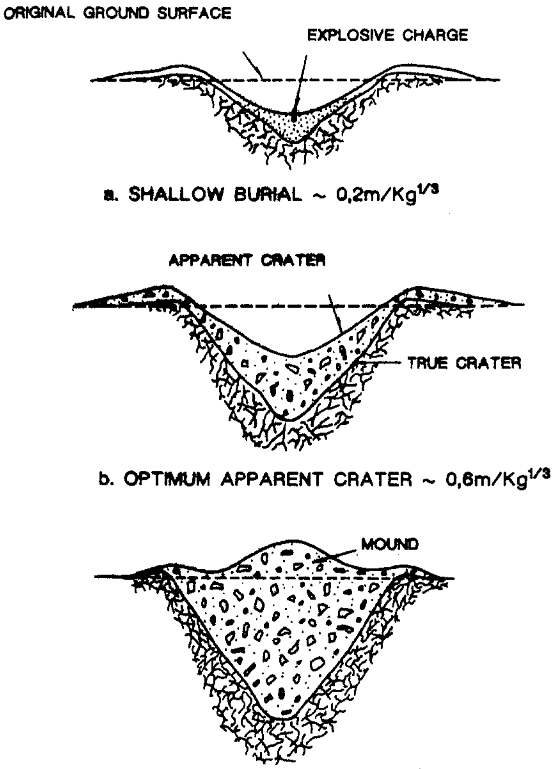


Fig. 24.2. Effects of increasing depth of burial on crater shapes.

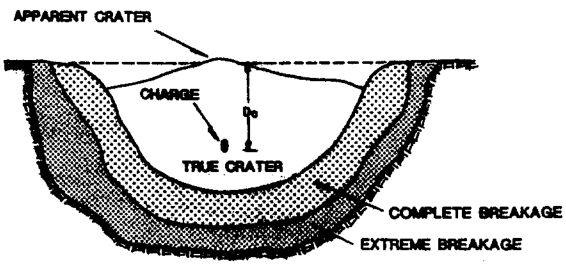


Fig. 24.3. Zones of a crater.

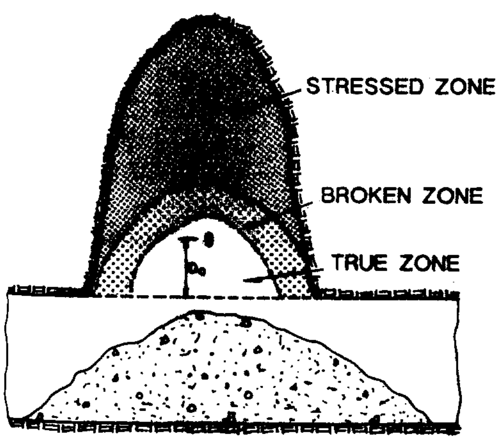


Fig. 24.4. Dimensions of the cavities created by spherical charges with inverted faces.

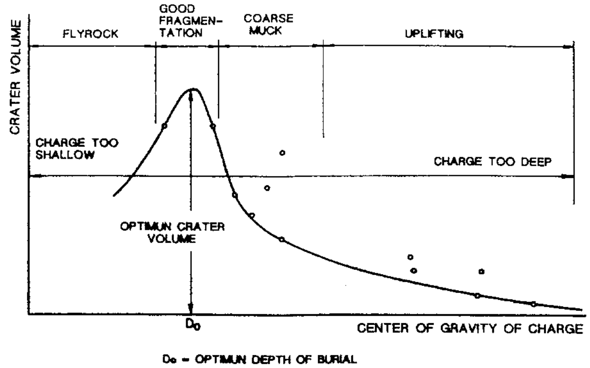


Fig. 24.5. Demonstration of the results of crater blastings.

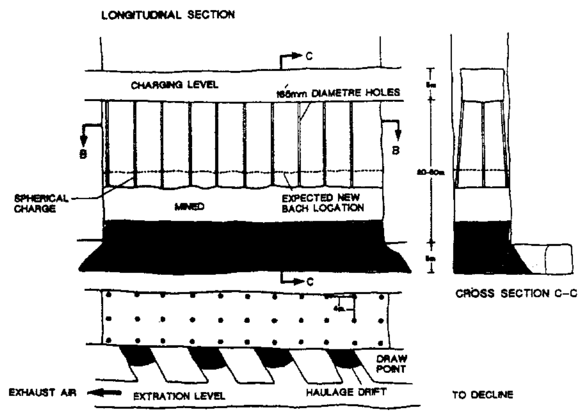


Fig. 24.6. Pattern of the 'VCR' stoping mining method.

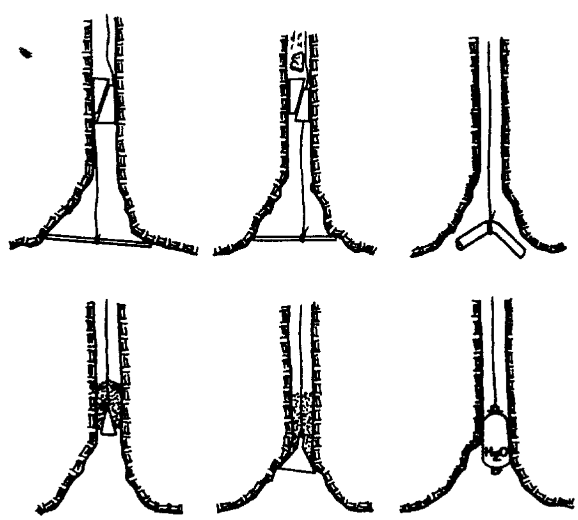


Fig. 24.7. Different ways of plugging holes (Mitchell).

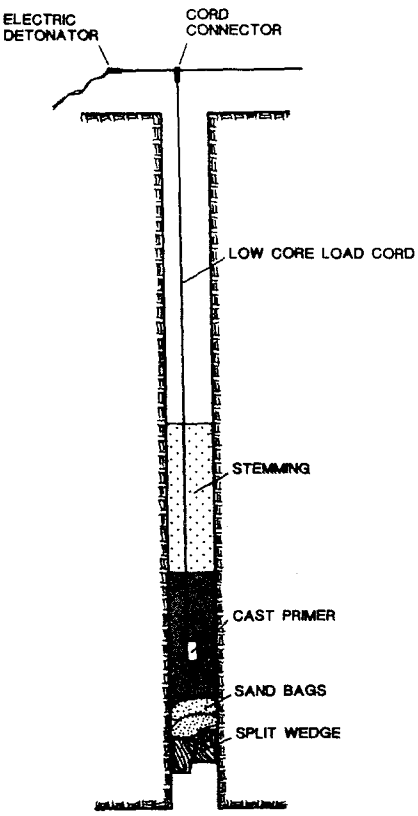


Fig. 24.8. Design of the charge in a blasthole.

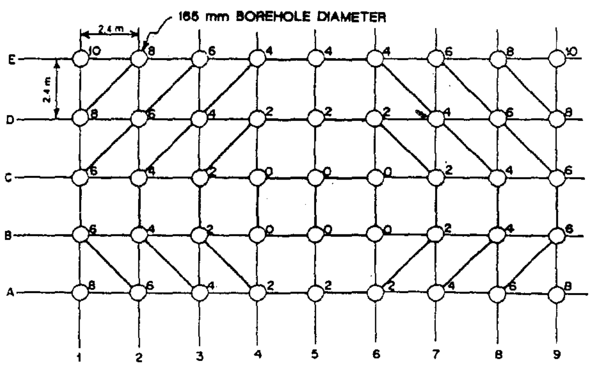


Fig. 24.9. Typical vertical crater retreat blasting delay pattern.

the stope ceiling it is recommended that they be fired first. It is also convenient, whenever possible, that each charge have two free faces, thus increasing fragmentation. In the pattern of Fig. 24.9, a typical sequence in this method is representated, so that the blastholes of the same number have two free faces, one is the stope ceiling and the other the walls of the craters excavated in earlier blasts.

The breakage of the final crown of the pillar, which is right underneath the charging level or topsill, requires the use of special blasts which can be designed knowing the mean vertical advance in each round and the dimensions of the crown. As a general guide, the criteria in Table 24.2 can be followed.

24.2.3 *Advantages and inconveniences of the 'VCR' method*

This method has the following advantages:

– Optimum safety for personnel and equipment (excluding the last blast in which the crown is broken).

– With warehouse chambers, the protection needed for the orebody side walls is reduced as the broken and swollen ore acts as support.

– As the charge weights per hole or delay are small, the vibration levels are not very high.

– The fragmentation is usually good.

– Muck loading, without remote control, can reach 70% and, if there is lateral access, it can be up to 80%.

– It is well adapted to narrow orebodies of around 3 to 10 m thick, even with inclinations that are not very elevated, and

– No need to drill raises.

Photo 24.1. Test crater blast.

Table 24.2.

Dimension of crown*	Blasting procedure
< 1.5 X	Single blast. Single symmetrically placed charge
1.6-2.0 X	Single blast. Decked charges fired simultaneously
> 2.0 X	Two separate blasts

*Function of the mean vertical advance per round X.

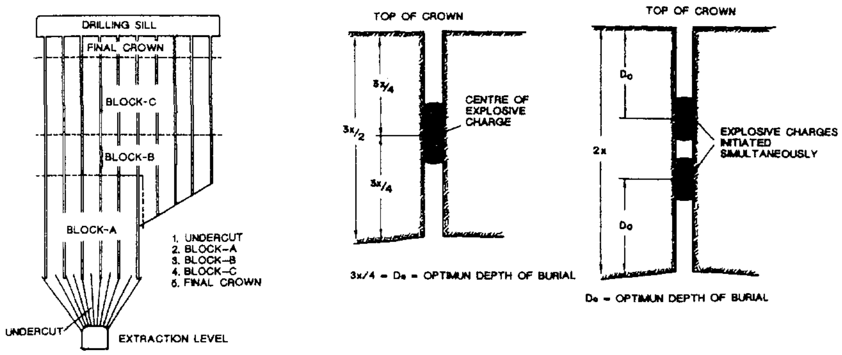


Fig. 24.10. Design of a crown blast (Hagan).

On the other hand, the problems are:

– Poor ventilation while loading the ore, requiring secondary ventilation.

– The damage to hanging walls is extensive with risk, on occasions, of caving.

– Grade control is difficult because in each round the muck piles up on that of the preceding blast and mixes during its descent, and

– After the extraction, rock can fall off the side walls and increase dilution.

24.3 LONGHOLE METHOD

24.3.1 *Long blasthole mining method 'LBH'*

The longhole method, 'LBH – Large Blasthole', is an application of the principals of open pit bench blasting to underground mining. The method affects primarily the breakage operation and, in a certain way, the preparation of stopes as, in general, the work is carried out on two sublevels, one for drilling and another for extracting. However, the operational methods are the same as in the conventional sublevel open stopping, Fig 24.11.

In the 'LBH' method, each stope is divided into three clearly differentiated sectors:

– The undercut, which carries out the mission of receiving the fragmented ore and creating a free face in the bottom of the blastholes.

– The longhole sector, where the large diameter pilot holes are drilled and represent between 85 to 90% of the chamber tonnage.

– The slot raise, which is used as the first vertical free face of the blast, for the undercut as well as for the longhole zone, Fig. 24.12.

The slot raise, or beginning of the sector, is constructed from a raise with dimensions that oscillate between 1.8 and 3.5 m, depending upon the case, and which can be excavated with a raise borer or by the 'VCR' method using the available drilling equipment.

From the raise, the undercut is created with vertical fan shaped holes, usually of 65 mm, on a pattern of 1.5 x 2 m in the bottom of the blastholes. The powder factor is around 800 g/t.

Afterwards, with the production drilling equipment, large diameter blastholes are opened (165 mm) on a triangular pattern, Fig. 24.13.

24.3.2 *Blasting in the longhole method 'LBH'*

To calculate the drilling pattern in the long blasthole zone, Langefors equation is usually applied:

$$B_{\max} = \frac{D}{33}\sqrt{\frac{\rho_e \times \mathrm{PRP}}{\bar{c} \times f \times (S/B)}}$$

where: $B_{\max}$ = Maximum burden (m), D = Blasthole diameter (mm), $\bar{c}$ = Rock constant (Taking generally: $\bar{c}$ = 0.3 + 0.75 Medium hard rocks; $\bar{c}$ = 0.4 + 0.75 Hard rocks), f = Fixation factor (Vertical blastholes; f = 1, Inclined blastholes; 3:1 f = 0.9; Inclined blastholes 2:1 f = 0.85), S/B = Relationship between spacing and burden, ρ_c = Charge density (kg/dm^3), PRP = Relative weight strength of the explosive.

The value of the practical burden is established from the maximum value, applying a correction for the deviation of the blastholes and collaring errors:

$$B = B_{\max} - 2\,D - 0.02\,\mathrm{L}$$

where L is the blasthole length.

The spacing S is determined with the equation:

$$S = 1.25\,B$$

The drilling pattern influences the sizing of the drifts or cuts of the top level of drilling.

The bench blasting in this method does not require toe breakage, therefore it is only necessary to use the column charge. The most used explosives are: ANFO for hard and medium hard rock, and ALANFO for very hard rock. If there is water present in the blastholes, the charge can be placed in a plastic covering or use watergels and low density emulsions.

The main problem in this type of blastings is the level of vibrations generated by the large quantity of explosive that can be accomodated in the blastholes.

These vibrations produce dynamic stresses that can cause damage in the underground labors or the nearby installations.

This problem is solved by decking the charges with intermediate stemming or wooden separators. After studying the vibrations, the maximum quantity of explosives needed to form an elemental charge should be determined by observing the following:

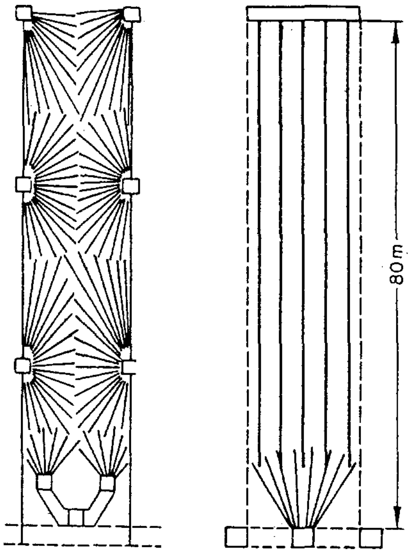


Fig. 24.11. Blasthole drilling patterns for conventional sublevel open stoping and for long blastholes.

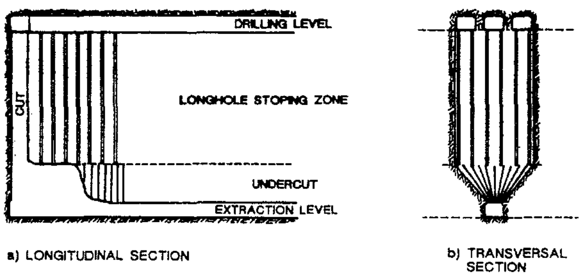


Fig. 24.12. Representative sectors of the long blasthole method.

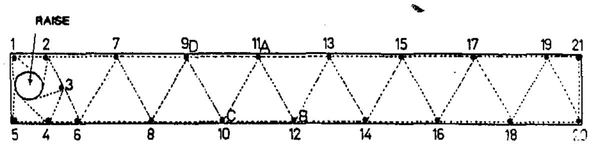


Fig. 24.13. Slot blasting to a bored raise with 165 mm blastholes (Hagan).

– The relationship Length of charge/Diameter should be maintained above 20 to obtain a good fragmentation.

– The volume of rock in front of the intermediate stemmings tends to make fragmentation worse.

– A large subdivision of the blastholes makes for a more complex charging operation and initiation system.

The length and type of intermediate stemming between deck charges should be such as to:

– Not produce sympathetic detonation or desensitivation of the adjacent charges that are initiated at different times.

– Achieve adequate fragmentation along the length of the stemming column.

– Assure that the stemming material has a size distribution close to 1/20 *D*, to inhibit secondary sulphate blasts, if necessary.

Therefore, the criteria for sizing should be, according to the type of explosive, Table 24.3.

The elemental charges by delay oscilate between 100 and 200 kg and are carried out as shown in Fig. 24.14.

The initiation of the charges can be done with electric detonators inside the blastholes or with a non-electric system, such as Nonel detanotors, low core load cord with temporized boosters, etc. In all cases boosters are necessary and it is recommended that two per charge be used.

The advantages of the non-electric initiation methods are the following:

– Lower risk of accident produced by premature firing of the detonators.

– Reduced charging time.

– Easy to handle.

– A sequential blasting machine is not necessary.

The initiation sequence of the blast is from bottom up, with the following recommended delay timings. According to Du Pont:

– Charges in the same blasthole . . . . 50 ms.

– Adjacent charges in the same row . . . . . . 10 ms.

– Charges between rows . . . . 75 ms.

Table 24.3.

Explosive	Intermediate stemming
ANFO	12 to 15 D
Watergels	16 to 18 D

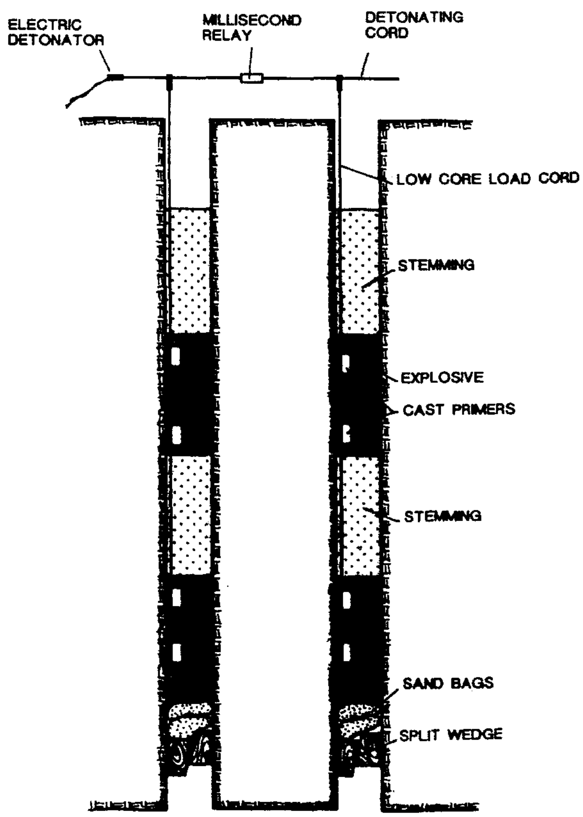


Fig. 24.14. Decked charges.

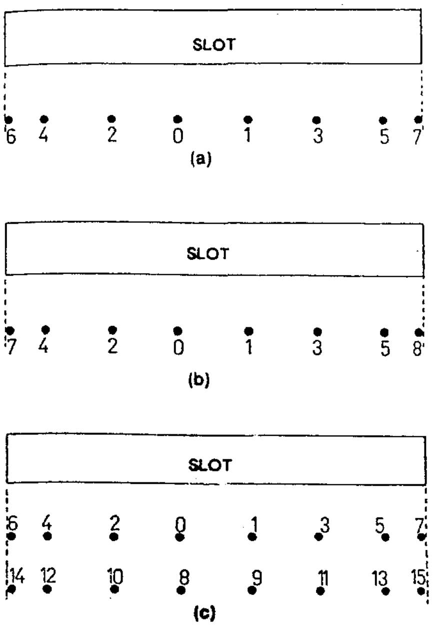


Fig. 24.15. Initiation system for blasts with continuous charges per delay: (a) standard blast, (b) extra relief for pillar-line holes and (c) double-row blast (Hagan).

When firing one row blasts with continuous column charges, the sequence will be such that initiation starts in the center of the row, Fig. 24.15a. When the probability of cut offs is minimal, due to the delay elements being inside the holes, pillar damage can be reduced by setting a long delay timing in the blastholes at each end, Fig. 24.15b.

In double-row blasts, in which each hole contains a continous column charge, the placing of only one delay per charge makes the initiation system more difficult and increases the risk of cut offs owing to the amount of delays existing between a hole in the first row and the adjacent one in the next row, Fig. 24.15c.

24.3.3 *Advantages and inconveniencies of the Longhole method 'LBH'*

The main advantages of the longhole method are:

– Extra safe working conditions and regularity in production.
– High productivity and breakage yield per lineal meter drilled.
– Very high benches, up to 70 m, which make large size blasts possible.
– Less damage to the remaining rock, as the blast has two free faces and the blastholes can be designed with decked charges.
– The possibility of loading up to 80% of the broken rock without remote control.
– Lower powder factor than with the 'VCR' method.
– Use of less expensive explosives such as ANFO instead of watergels or emulsions.
– Lower drilling and blasting costs, and
– Good grade control and low ore dilution.

The main disadvantage of this method is that it causes a compression of the muck because it falls from a great height.

24.4 SUBLEVEL STOPING WITH BLASTHOLES IN FAN PATTERN

This system is applicable to sub-vertical orebeds with side walls that have good characteristics, so that the when the ore is extracted large open stopes remain, similar to those in the 'VCR' and 'LBH' methods.

The drilling is done from drifts on the sublevels in fan shape with upward or downward blastholes, or both, whose lengths are adapted to the surrounding ore. In an effort to reduce the preparation work, which is costly, the blastholes are made as long as possible, Fig. 24.16.

The drilling rigs used are of special design with extension drill steels and bits of 51 to 64 mm. The separation between drilled sections is usually around 1.2 and 1.8 m.

The collaring, orientation and deviation of the blastholes are some of the operative conditions necessary to obtain good blasting results. This means that it is necessary to use special orientation systems and accessories, and not drill blastholes of more than 25 m.

The blasts are done with one free face, and partially removing the muck from the preceding rounds.

The yield of broken rock per lineal meter drilled is low because the planned spacing diminishes as it nears the collar, leaving part of the blastholes unused in the breakage.

The calculations of the drilling patterns are taken from the necessary powder factor, which is a function of the rock type, drilling length and width of the blast.

$$CE = CE_o + 0.03\,L + \frac{0.40}{AV}$$

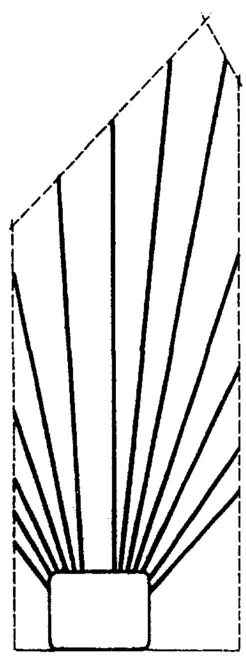


Fig. 24.16. Sublevel stoping with fan holes.

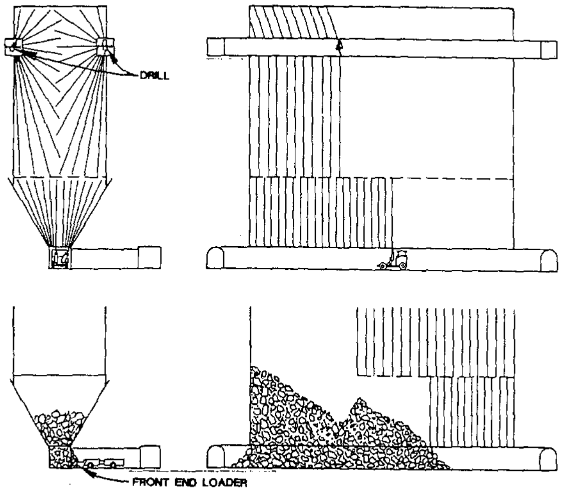


Fig. 24.17. Detail of a drilling pattern.

Table 24.4.

Type of rock	Base powder factor CE_o (kg/m^3)
Fissured and hard	0.6
With joints	0.55
Fractured	0.5
Relatively homogeneous	0.45
Homogeneous and hard	0.40
Soft and homogeneous	0.35

where: CE = Designed powder factor in the bottom of the blasthole and in one fifth the length of the same, expressed in kg/m^3 of gelatin explosive, CE_o = Base powder factor of the rock, calculated from Table 24.4, L = Blasthole length (m), AV = Width of the round (m).

The pattern in the bottom is calculated from the lineal charge concentration q_l that is to be reached, making

$$A_e = S \times B = \frac{q_l\,(\text{kg/m})}{CE\,(\text{kg/m}^3)}$$

proving the ratio:

$$\frac{S}{B} = 1.3 \text{ to } 2$$

having: S = Spacing (m), B = Burden (m).

When $S = 2B^2$ good results are usually obtained and then $A_e = 2B^2$, where the value of the burden is found in order to calculate the spacing.

The column charge is designed between 50 and 75% of the bottom charge, with sufficient length to obtain good fragmentation.

To lower drilling costs, it is necessary to get maximum results from the same based upon mechanized systems of charging.

The most frequently used explosives and blasthole filling equipment are the following:

– Cartridge gels with pneumatic chargers.

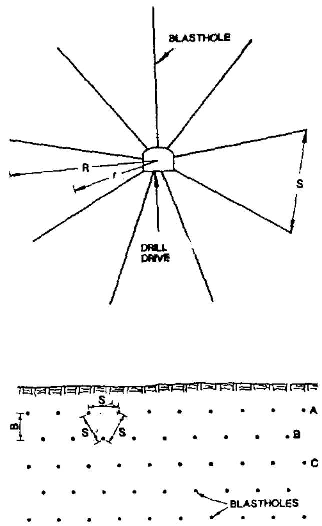


Fig. 24.18. Blastholes drilled in fan pattern in parallel planes following the standard equilateral triangle design (Hagan).

– Water gels and cartridge gels with pneumatic chargers.

– Water gels and bulk emulsions with pump units.

– Bulk ANFO with pneumatic chargers.

Recently, Hagan suggested drilling holes in a fan pattern following reorientated equilateral triangles. In Fig. 24.18, the conventional drilling pattern can be observed.

These patterns bring the following problems:

– The distribution of the explosive energy is only optimum within a cylinder of rock with r radius. As this magnitude diminishes, so does the effectiveness of the pattern.

– When the sector that has been drilled in fan pattern has an angle that is smaller than 360°, Fig. 24.19, the distribution of the energy at the farthest points of the pattern (e.g. in the volume ABC of Fig. 24.18) is unsufficient and, consequently, so is fragmentation and swell.

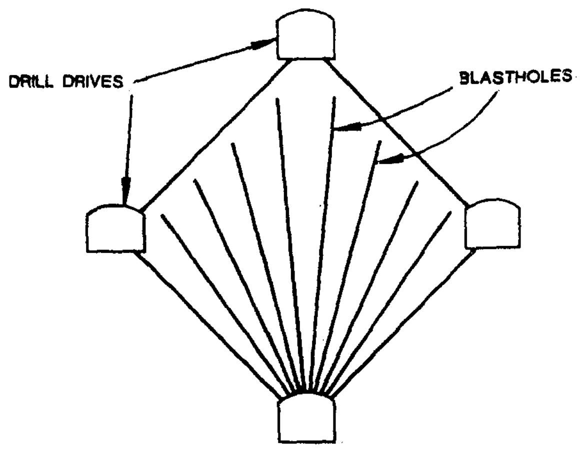


Fig. 24.19. Drilling sector with blastholes in fan pattern with a central angle that is under 360° (Hagan).

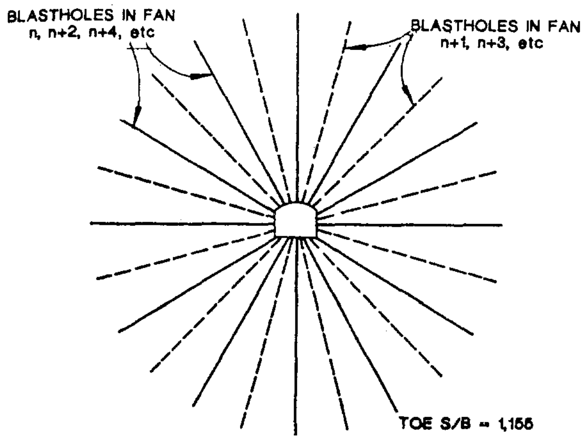


Fig. 24.20. Parallel fan holes drilled in a conventional pattern and with the *S/B* relationship value equal to 1.155 at collaring points (Hagan).

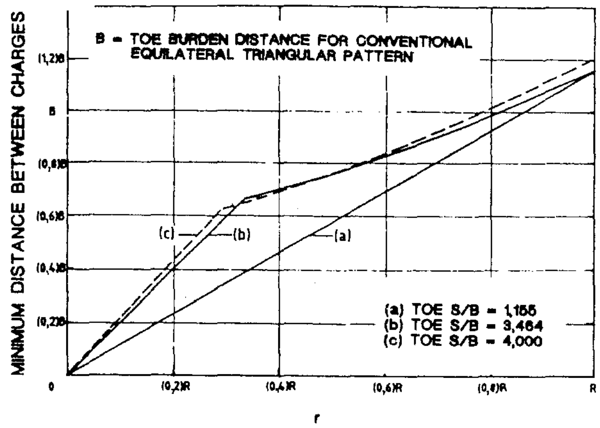


Fig. 24.23. Minimum distance between charges for different drilling patterns and values of *r* (Hagan).

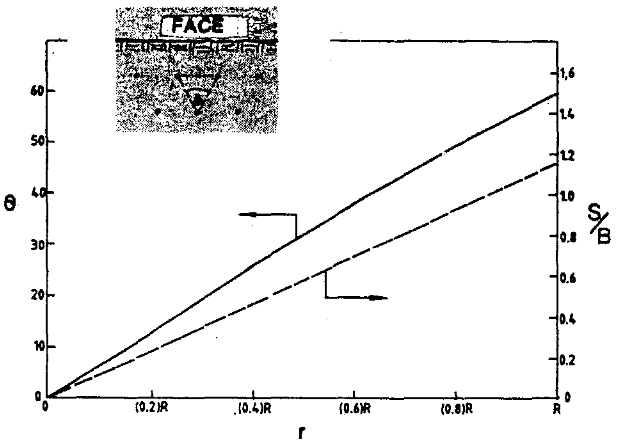


Fig. 24.21. Values of θ and *r* for different *S/B* relationships at collaring points (Hagan).

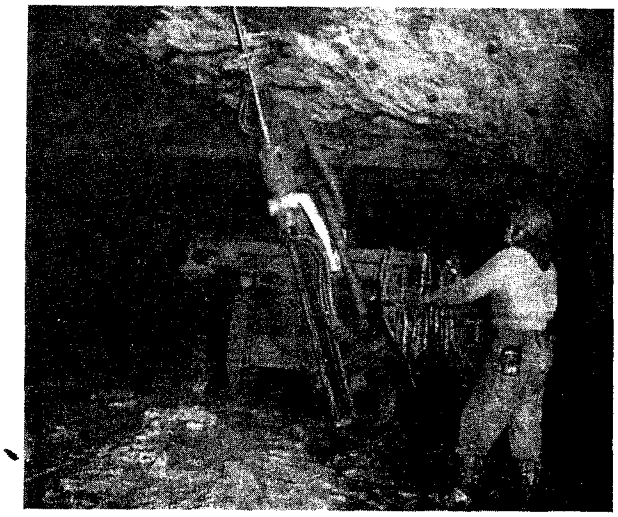


Photo 24.2. Drilling an uphole.

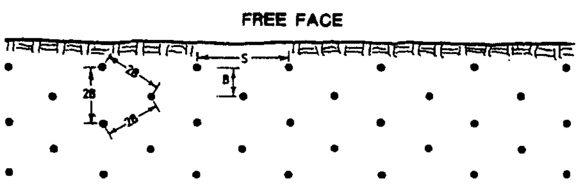


Fig. 24.22. Drilling blastholes in a fan pattern with reorientated equilateral triangle designs (Hagan).

– As the distance between charges in a fan pattern decreases, the probability of a charge initiating, desensitizing or 'robbing. the burden of other adjacent charges increases.

In Fig. 24.20, it can be seen how the parallel fans are dephased from one another with a pattern of equilateral triangles, obtaining a *S/B* relationship value that is equal to 1.155 at the collars.

In Fig. 24.21, one can observe how the values of *S/B* and θ diminish as *r* decreases from *R* to 0. This decrease signifies a smaller amount of energy freed, an increase between the predicted and actual detonation times, more energy lost in vibrations and fly rock, and a poorer distribution of the useful energy.

By rotating the equilateral triangles 30° with respect to the face line, the drilling pattern shown in Fig. 24.22 is achieved.

With these patterns, *S/B* values that are equal to 3.464 can be reached, giving the following advantages:

– For any value of *r* the minimum distance between charges is larger, for an *S/B* relationship at the collaring point of 3.464, Fig. 24.23. For this reason, the probability of the detonation of a charge initiating or harming another adjacent one with a later break time is smaller. For the reorientated patterns, the minimum distance between charges refers to between adjacent fans if $R > r > 0.3$ or within the same fan, as happens in the conventional patterns, if $r < 0.3R$. When *S/B* is under 1.0, the capacity of a charge to initiate or harm an adjacent charge rapidly increases.

– When the *S/B* relationship at the collaring point is over 2.4, part of the ore is fragmented by sections of charges near the collar that break towards balanced concave biplanar faces. As *S/B* increases from 2.4 to 4.0. the percentage of fragmented ore increases from 0% up to 64%. This only happens in reorientated patterns.

– As *r* diminishes, the blastholes deviate from their initial reorientated pattern, but they go through a conventional equilateral triangle pattern upon a cylindrical surface that has a radius of $r = 0.3\,R$.

Therefore, it can be deduced that the reorientated patterns with collar *S/B* relationships close to 3.5 allow a better use of the explosive energy, with the consequent reduction in drilling and blasting costs, avoiding the potential problems of unstability, and producing lower vibration levels.

24.5 ROOM AND PILLAR MINING

Used in horizontal or flat dip bedded deposits of salts, limestone, potash, iron, etc, with maximum inclination of 30° and rocks with stable geomechanic characteristics, ore that can be extracted by excavating large chambers and leaving pillars to support the side walls.

The pillars are placed in regular patterns, usually square, circular and sometimes rectangular. Their sizing is one of the most important aspects because it conditions the ore recovery and the stability of the operation.

In horizontal deposits, or with little inclination, the process consists in opening drifts for the extraction and haulage of the ore. This is often done by connecting these labors with the previous drifts. The dimensions of the excavations correspond with the height and width of the drifts, and the machinery used is composed of jumbos with several booms and drilling rigs which gives a high degree of mechanization.

The inclined orebodies are divided vertically into levels, from which the haulage drifts are established along the foot walls. These drifts serve as access ways to the production areas where the operation continues upwards to the next level. The drilling for the blasts in these areas is carried out with hammers installed upon pusher Mechanization is not wide spread in this method, whic means abundant manual labor and low productivity.

A variation of the room and pillar method in incline orebeds is composed of sloping drifts which serve a access to the operation zones and can be climbed by th jumbos. From these access drifts, other horizontal drif are excavated at intervals, following the ore boundary a closely as possible.

In thick deposits, the excavations cannot be carried ou with the jumbos in one phase, which means that the ore i divided vertically, and the lower part is recovered b bench blasting. The benching is carried out with conventional rigs and vertical blastholes, Fig. 24.24.

In these mining operations, roof bolting is a techniqu that is applied extensively to improve stability which apart from production drilling, should be taken into consideration as a corresponding part of the work.

24.6 CUT AND FILL MINING

This method was originally developed in Canada in the later fifties. It consists in excavating the ore by ascending horizontal sublevels, filling the holes produced by ore extraction from the deepest sublevels with waste material, which serves as support for the hanging walls and as a work platform for the drilling, charging and haulage equipment. Hydraulic filling is a usual practice because of easy transportation and the possibility of mixing it with a small percentage of cement, Fig. 24.25.

This method has the following advantages:

– High ore recovery.
– Grade and dilution control.
– Mechanization of the operations.
– Fewer problems of side wall stability and surface caving, and
– Easy and effective ventilation.

Two systems can be used for drilling and blasting:

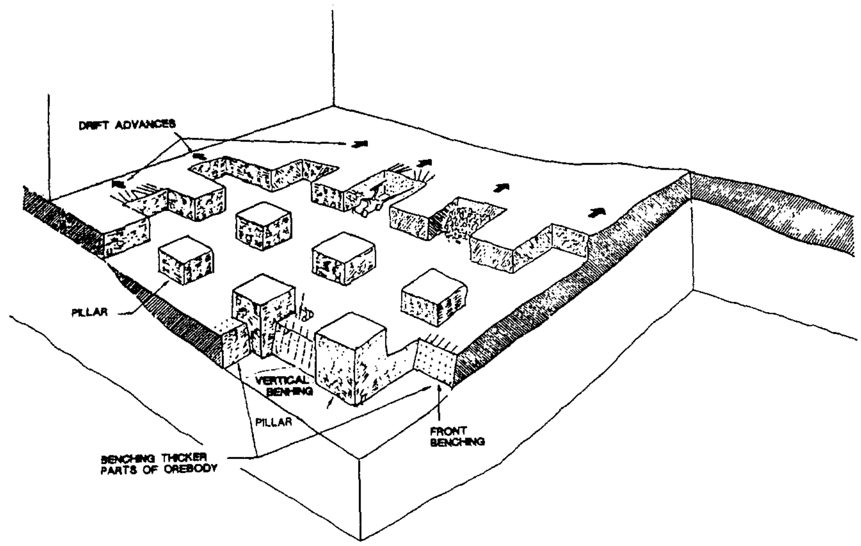


Fig. 24.24. Room and pillar mining.

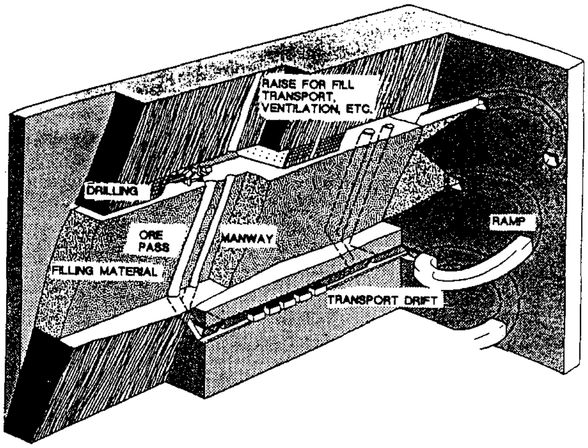


Fig. 24.25. Cut and fill mining method.

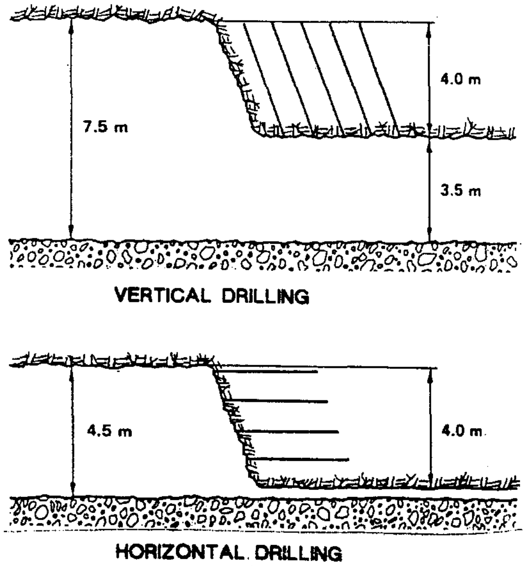


Fig. 24.26. Systems of drilling and blasting.

vertical drilling into the roof by the inverted bench method, and horizontal drilling, as in a bench that is turned 90°, Fig. 24.26.

The first system has the advantage of being able to drill a large area quite in advance of the blasting and charging operations. The quantity of ore extracted can be as large or small as desired. The drilling does not usually surpass four meters vertically, because the total height of the excavation would be excessive and the blastholes are placed with a 50 to 65° inclination. One inconvenience of the system is the irregularity and poor condition of the roof after each blasting. In some mines it is necessary to previously reinforce the ore mass with cemented cables to avoid its caving in.

The second system consists in drilling horizontal blastholes with the same length as the drill rods, also less than the four meters, and shoot towards the lower free surface. The volume of each blast is limited because it has to be drilled from the face. The advantages of this alternative system are: a more regular roof surface, lower height of the hole opened, improved selectivity and ore recovery.

Although hand held drills and pushers can be used for the drilling with blastholes of 29 to 33 mm diameters, the normal practice is to use drilling wagons and jumbos with larger diameters such as 33 to 64 mm.

The explosives used go from the conventional and the slurries to ANFO, with powder factors that vary between 200 and 260 g/t.

When drilling is vertical, the stemming of the blastholes is done with clay plugs.

24.7 UNDERGROUND CHAMBERS IN CIVIL ENGINEERING PROJECTS

During the last decades there has been an increase in the use of underground space for diverse projects:
- Hydroelectric power stations.
- Fuel storage.
- Depots for toxic and radioactive residues.
- Atomic shelters.
- Military instalations.
- Underground parking lots.

In relation with the height of the chambers, these are classified as follows:
- Small, with a total height of under 10 m.
- Large, of up to 60 m or more.

24.7.1 *Small chamber stoping*

This type of chamber is similar to large section tunnels with the excavations carried out in the same manner. Generally, the mining is divided into phases which begin in the top part with a pilot drift (1) and lateral stope holes (2), to follow by benching in the lower part with horizontal or vertical blastholes (3), Fig. 24.27.

So as to not harm the rock and diminish the thickness of the concrete lining, smooth wall blasts will be carried out in the final profiles.

The lower bench is broken once the upper section is finished, or with a certain time difference. The stoping can be done with horizontal blastholes, using the same jumbo of the other sections, or with vertical blastholes, using a surface drilling rig. The normal drilling diameters for the horizontal holes are from 32 to 45 mm and 50 mm for the vertical holes.

24.7.2 *Large chamber stoping*

In large chambers, the normal practice is to carry out the excavations from top to bottom in descending phases, with different drilling and blasting systems, Fig. 24.28.

Phases 1 and 2 are done with horizontal drilling and removal of broken rock through the pilot drift. The phases 3, 4 and 5 are normally carried out with vertical drilling and rock removal through the different access levels which, in the case of the hydroelectric power stations, can be the alternators, lower tracts of pipelines, drainage drifts, etc.

The vertical blastholes are usually limited to 4 m in

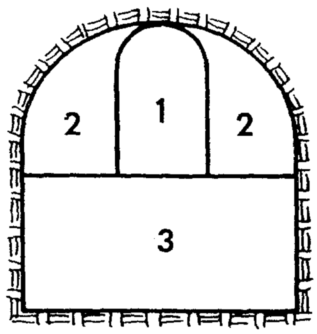


Fig. 24.27. Stoping phases in a small chamber.

Photo 24.3. Hydroelectric Power Station of Saucelle (Iberduero, S.A.).

height, because they permit the use of a *glory hole* raise for the evacuation of the broken rock.

The excavation procedures vary with the quality of the rock, but in any case it is done with the idea of assuring contour stability as it advances downwards. The excavation of the top part or arch is carried out in phases, as indicated, from a drift that is used for examination. If the ground is good, a systematic bolting of the surface with gunite reinforcement steel ribs is usually performed and, if the rock is poor, the excavation is carried out in transversal sections, leaving between them others that are not excavated until the arches already opened have been concreted.

In the hydroelectric power stations, the end of the top section depends upon the method chosen for support of the guide rails for the crane bridges. The most difficult to solve is when they rest on the actual rock. In this case, underneath the top section level a trench cut is performed to later carry out two cross cuts, in two horizontal planes, and two presplits, usually in the vertical planes. This way, the corners of the rock are formed, assuring their integrity and resistance.

Bench blasting is the easiest and therefore should be used as much as possible. The bench heights are not usually very large because the blasts are done in relatively high conditions of confinement, drifts usually enter in the central cave and it is necessary to support the side walls as the excavation procedes downwards. For these reasons, the horizontal and vertical drilling should be

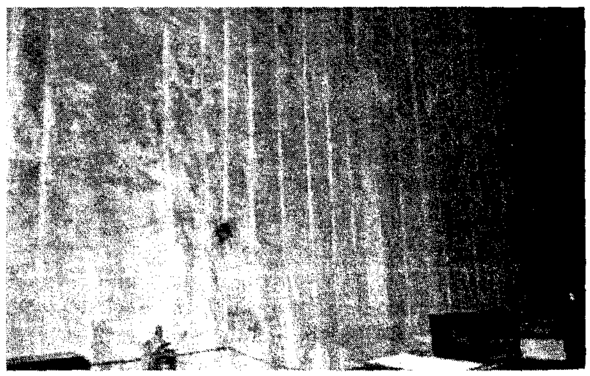


Photo 24.4. Side walls with presplitting in the Hydroelectric Power Station of Saucelle (Iberduero, S.A.).

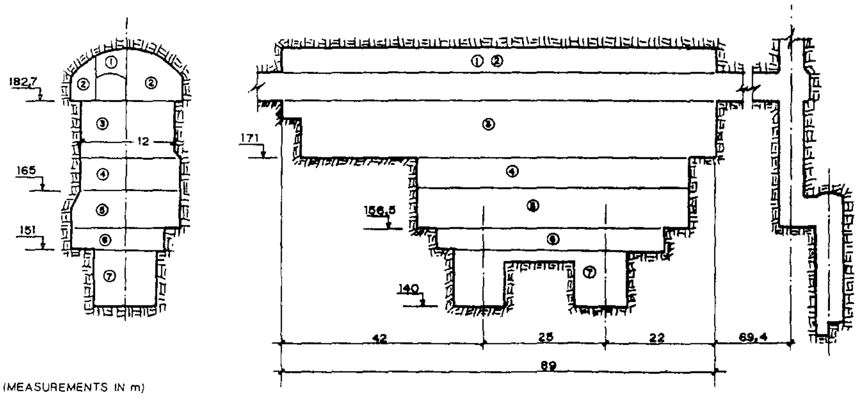


Fig. 24.28. Excavation phases in Aldeadávila II (Courtesy of Iberduero, S.A.).

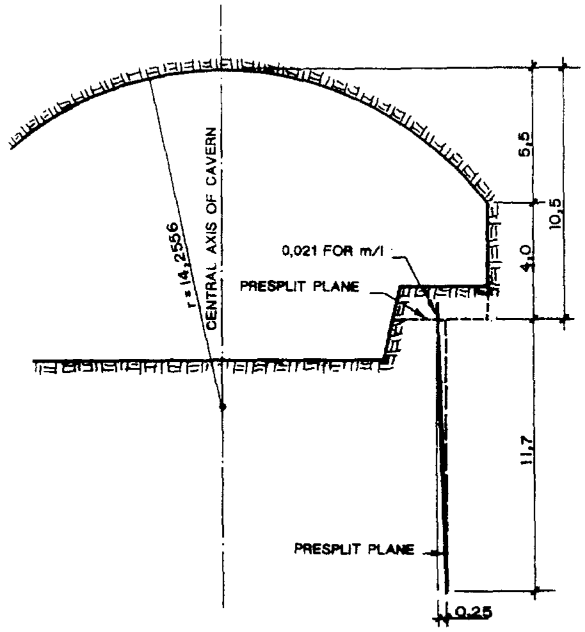


Fig. 24.29. Detail of chamber with support of guide rails.

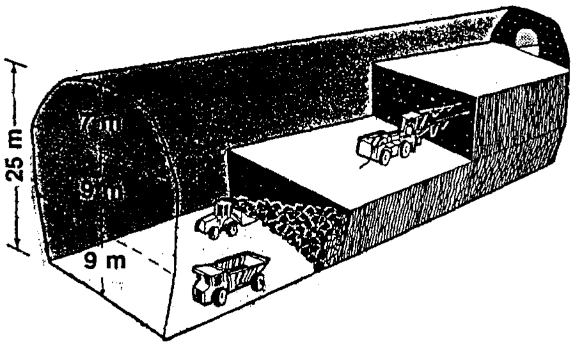


Fig. 24.30. Chamber excavation with two benches and horizontal driling with a jumbo.

simultaneous, apart from the availability of different drilling rigs.

For the removal of the broken rock from the lower parts of the hydroelectric power stations, it is recommended that the accesses be drilled beforehand, at the same time that work is being done in the top part of the caves.

As for the calibers of the blastholes, the usual ranges are 32 to 45 mm for the horizontal drilling, and from 50 to 65 mm for the vertical.

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