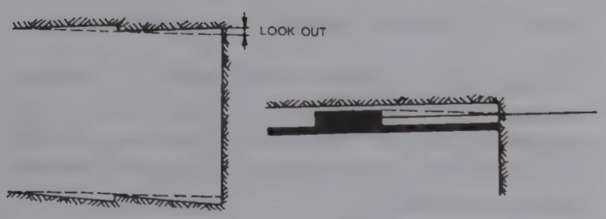

Fig. 22.4. Orientation of the contour blastholes to maintain the tunnel profile.

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

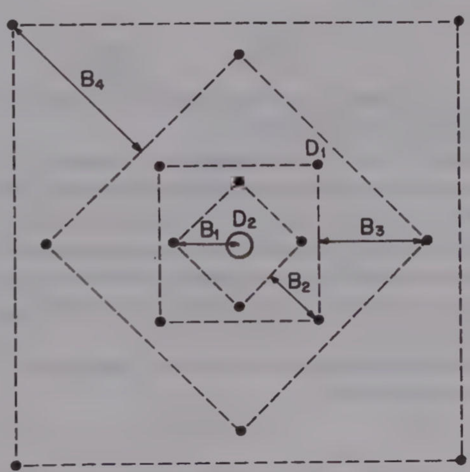

Fig. 22.5. Four section cut.

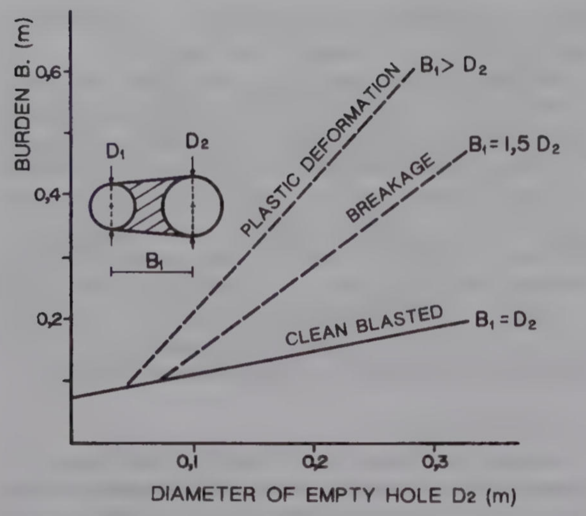

Fig. 22.6. Results of the blastings for different distances between charged and empty blastholes, and their diameters (Holmberg).

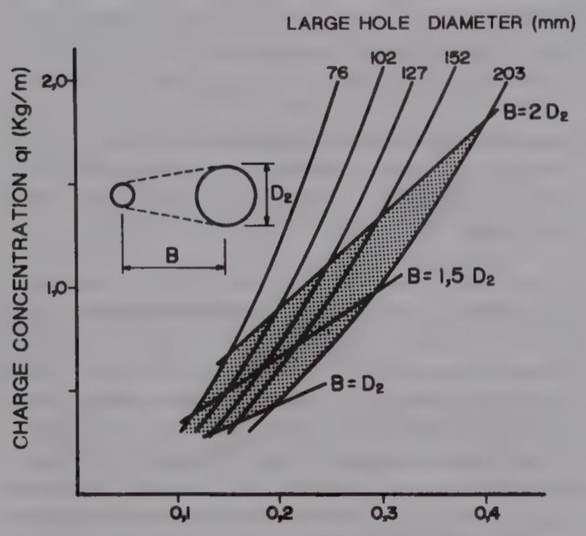

Fig. 22.7. Relationship between the lineal concentration of maximum burden and charge for the different diameters of relief blastholes (Larsson and Clark).

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:

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

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, PRP_{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:

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

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:

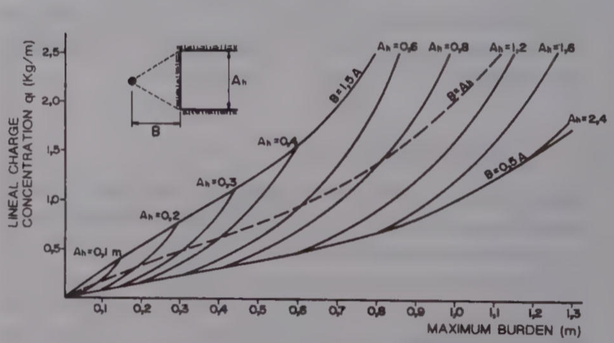

Fig. 22.8. Relationship between the lineal concentration of the charge and the maximum burden for different widths of the opening (Larsson and Clark).

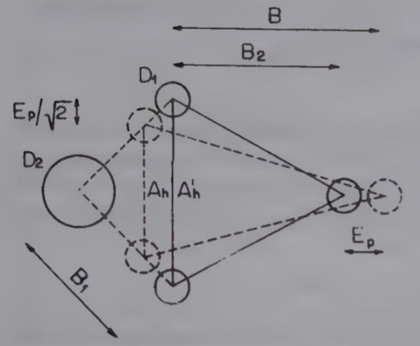

Fig. 22.9. Influence of blasthole deviation (Holmberg).

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

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.

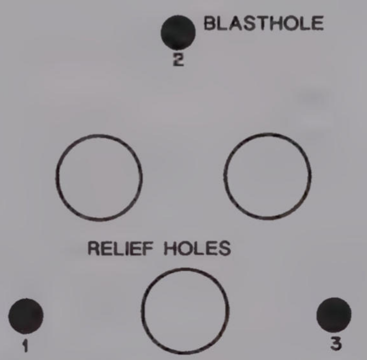

Fig. 22.10. Modified burn-cut to eliminate sympathetic detonation and dynamic pressure desensitization (Hagan).

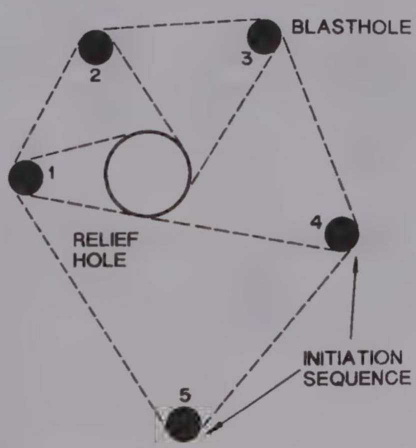

Fig. 22.11. Spiral burn-cut.

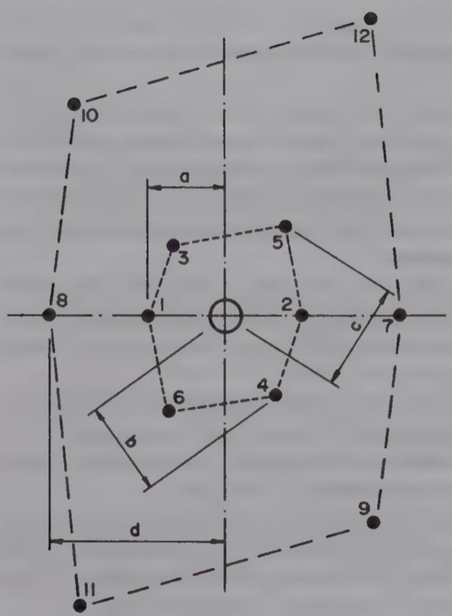

Fig. 22.12. Double spiral cut.

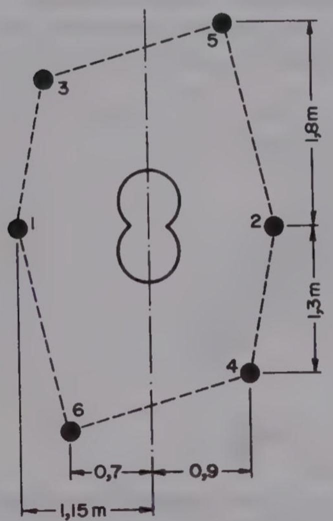

Fig. 22.13. Coromant cut.

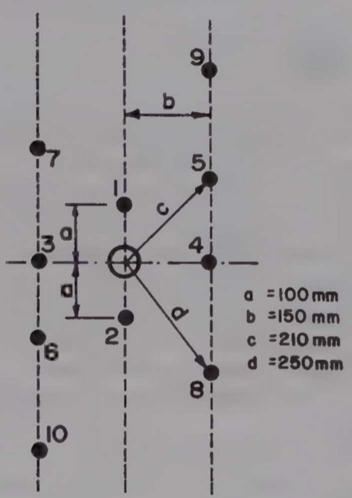

Fig. 22.14. Fagersta cut.

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:

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

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

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

where: AT = Tunnel width (m).

The practical spacing for the corner blastholes will be:

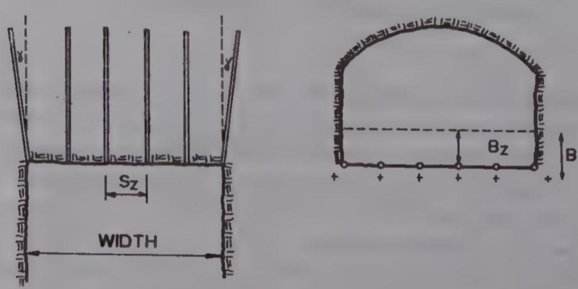

Fig. 22.15. Geometry of the lifters.

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

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
  $(4 \times 1.59) + (4 \times 2.62) + (8 \times 3.67) = 46.21$ kg
- Lifters: 5 blastholes $(5 \times 3.20) = 16$ kg
- Roof contour: 8 blastholes $(8 \times 1.77) = 14.16$ kg.
- Wall contour: 6 blastholes $(6 \times 3.20) = 19.20$ kg.
- Stoping: 5 blastholes $(5 \times 3.20) = 16.00$ kg.

Total charge of the blast = 111.6 kg
Tunnel surface = 19.5 m^2
Advance = 3 m
Volume of broken rock = 58.5 m^3

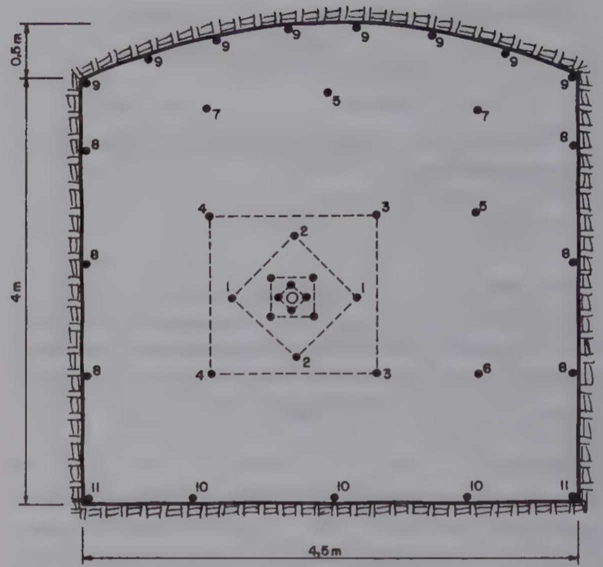

Fig. 22.16. Geometric pattern of the calculated blast.

Specific charge = 1.9 kg/m^3
Total number of blastholes = 40
Total length drilled = 128 m
Specific drilling = 2.2 m/m^3

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/cm^3).

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-

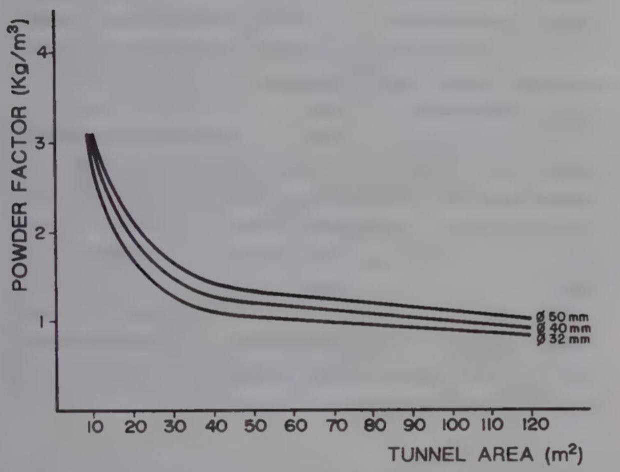

Fig. 22.17. Powder factor in function with the tunnel area (m^2) and the blasthole diameters.

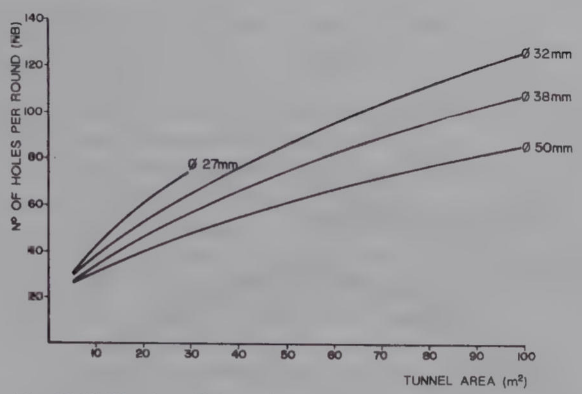

Fig. 22.18. Number of blastholes per round in function of the area.

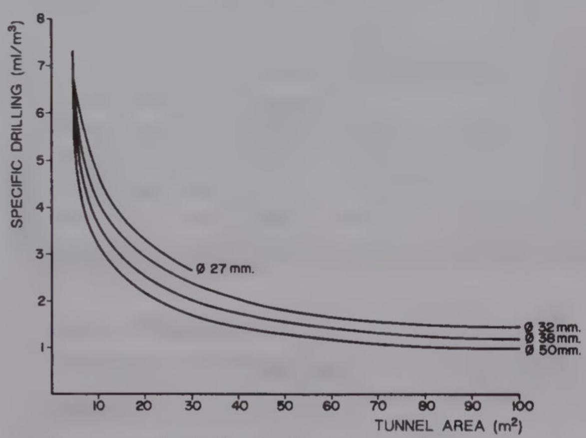

Fig. 22.19. Specific drilling in function with the tunnel area and drilling diameter.

Photo 22.2. Manual drilling in a drift face.

Table 22.2.

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	$B_4 = 1.5\,B_3\sqrt{2}$	$1.5\,B_4\sqrt{2}$

Table 22.3.

Part of round	Burden (m)	Spacing (m)	Length bottom charge (m)	Charge concentration		Stemming (m)
				Bottom (kg/m)	Column (kg/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

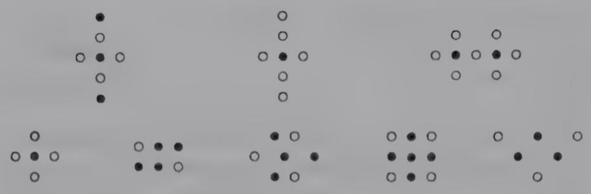

Fig. 22.20. Examples of burn-cuts.

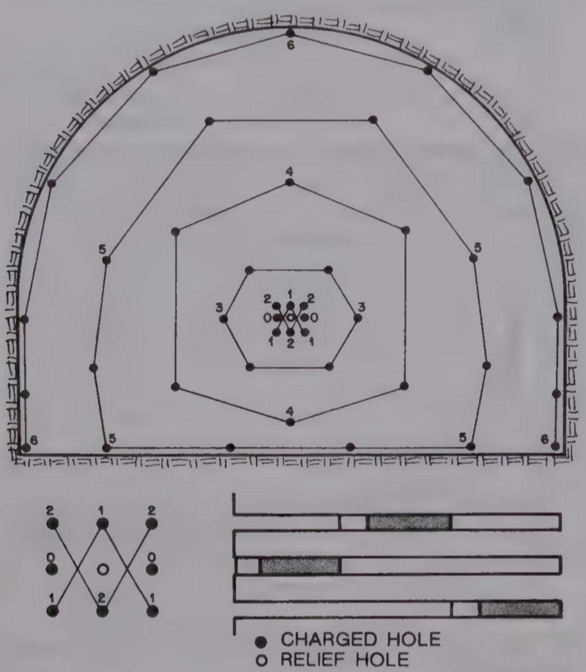

Fig. 22.21. Drift driving with the Sarrois cut.

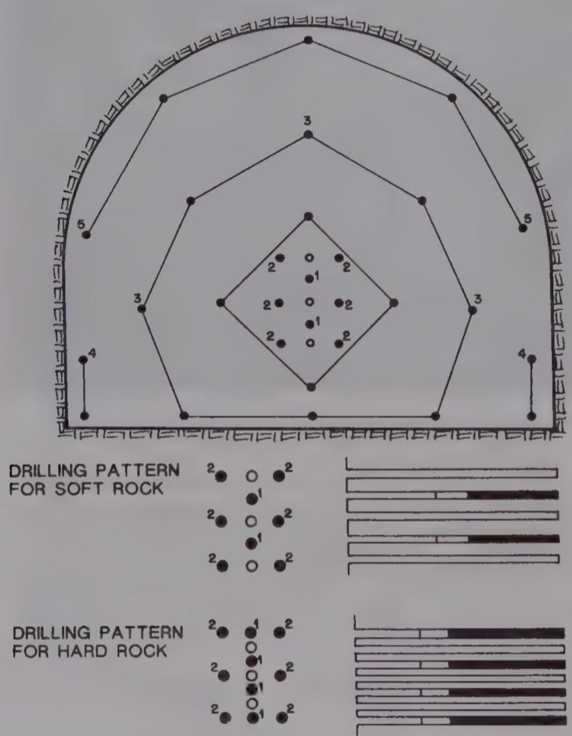

Fig. 22.22. Swedish Cut

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.

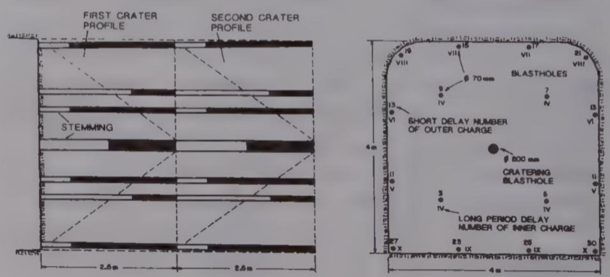

Fig. 22.23. Double crater cut using central blastholes of 200 mm (Hagan, 1981).