Chapter 18
Blasting Design for Underground Excavation

18.1 Blasting Design for Tunnel (Cavern)

18.1.1 Hole Layout and Firing Sequence

The blasts in tunnels are characterized by the initial lack of an available free surface toward 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 (reliever) is carried toward the opening. For easy illustration of the blasting procedure of tunneling, we divide the tunnel face into five separate zones, A–E (see Fig. 18.1):

A the cut section,
B the stoping holes breaking horizontally,
C the stoping holes breaking downward,
D contour holes, and
E the lifter holes.

The general firing sequence must be A > B > C > D > E.

The most important operation in the blasting procedure of tunneling is to create an opening in the face in order to develop another free face in the rock. This is the function of the cut holes. If this stage fails, the round can definitely not be considered a success.

18.1.2 Types of Cut-Hole Pattern

The cut-hole pattern can be classified in two large groups:

- angled hole cuts and
- parallel-hole cuts.

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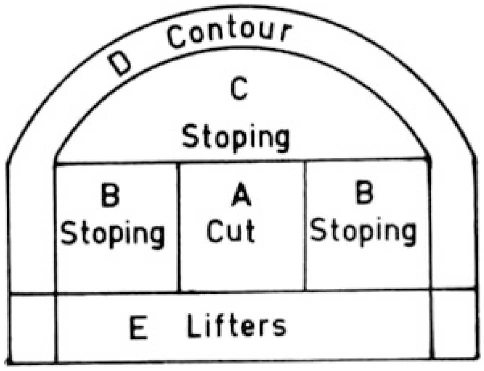


Fig. 18.1 Zones in tunnel blasting

The most common type of cuts today are the parallel-hole cuts and angled hole cuts. The angled hole cut is the old type and is still occasionally used in construction. It is quite an effective type of cut for tunnels with a fairly large cross section, and it requires fewer holes than a parallel cut.

The parallel-hole cut was introduced when the first mechanized drilling machines came to the market and made accurate parallel drilling possible.

18.1.2.1 Angled Hole Cuts

The angled cut is a traditional cut type based on the symmetrically drilled, angled holes. It has lost some of its popularity with the widespread adoption of the parallel cut and longer rounds. However, it is still commonly used in wide tunnels where the tunnel width sets no limitations on drilling. The advantage is a lower drilling length and explosive consumption than the parallel cut because there is better utilization of the free face surface and the possibility of orientation toward the visible discontinuities in the section. But its biggest disadvantage is that it ejects rock violently, the rock is thrown a considerable distance resulting in services being destroyed, e.g., ventilation, power, air, and communications, and its use is banned in some places.

The following explains the most common angled cuts.

- V-cut

V-cut is also called wedge cut. With this angular cut, a wedge is detonated out of the center of the face and after that, the remaining part of the advance length is detonated. The V-cut can be positioned vertically or horizontally (or at an angle depending on the layering of the rock mass) as a single or staged wedge. Figure 18.2 shows a horizontal V-cut.

The angle at the bottom of the cut holes should not be less than 60°. Maintaining the right angle is the main difficulty in V-cut drilling; in addition, the correct drilling angle limits the round length in narrow tunnels (Fig. 18.3). Tunnel width limits the use of the V-cut. In narrow tunnels, the advance per round can be less than one-third of the tunnel width.

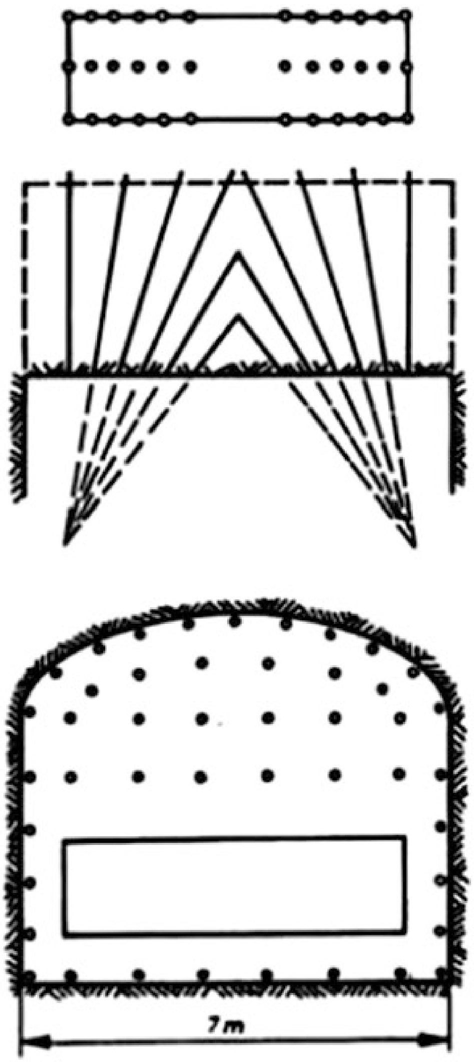


Fig. 18.2 Horizontal "V"-cut blast (reproduced from Ref. [14] with the permission from Sandvik)

- Fan Cut

The fan cut is also an angled cut. For this arrangement, several drillhole rows are placed in a fan shape. They have different lengths and angles. Figure 18.4 shows a fan cut. This type of cut was widely used before, but it is not favored nowadays because of the complicated drilling.

18.1.2.2 Parallel-Hole Cut

Parallel-hole cuts are also called cylindrical cuts. The characteristic for this cut is that the drillholes are the same length and obviously parallel to each other. At the moment, this type of cut is the most frequently used in tunneling and cavern blasting, regardless of their dimensions. This type of cut consists of one or several uncharged or relief blastholes toward which the charged holes break at intervals.

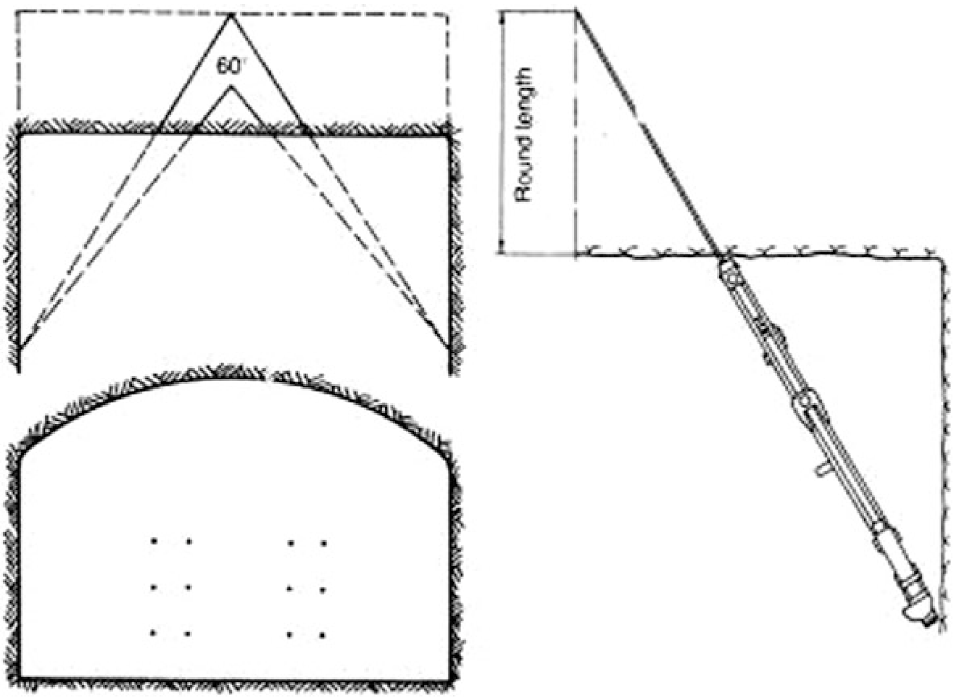
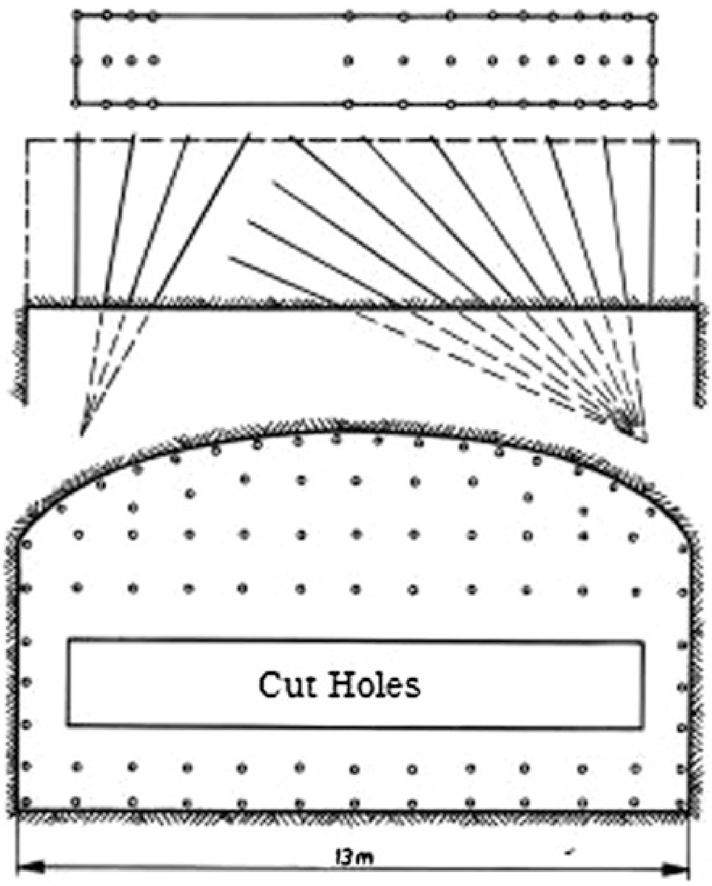


Fig. 18.3 Feed setup and drilling limitation in V-cut (reproduced from Ref. [14] with the permission from Sandvik)

Fig. 18.4 Horizontal fan cut (reproduced from Ref. [14] with the permission from Sandvik)



The principal advantages of the parallel-hole cuts are as follows:

1. The depth of the round is not dependent on the working space available for drilling holes at an angle.
2. The cut allows a deep pull even in tough rock formations.
3. It is relatively simple to drill, because all holes are parallel.
4. There is generally less throw with better fragmentation.
5. The resultant muckpile is higher, so it provides a better platform for scaling and bolting work.
6. Round length may be shortened or lengthened without any difficulties.

Principal disadvantages of the parallel-hole cuts are as follows:

1. If large relief holes are used, it requires reaming or larger drilling equipment.
2. Drilling and explosives requirements (powder factor) are higher.
3. Drilling must be accurate; otherwise, the results will be unfavorable.

There are two kinds of parallel-hole cuts: burn cuts and parallel-hole cuts with large empty hole(s). They will be discussed in separate sections later.

Burn Cuts

Burn cuts are also called Swedish cuts as they were first used in Sweden. In this cut, all the blastholes are drilled parallel and with the same diameter. Some are charged with a large quantity of explosive, while others are left empty. The empty holes provide a free face for reflection of shock waves. It is important that these holes are accurately drilled and parallel to each other in order to achieve a good blasting result. Figure 18.5 shows some hole patterns of burn cuts.

As the charge concentration is high, the fragmented rock is centralized in the far end of the cut and is difficult to breakout, so that the advance is reduced and does not surpass 2.5 m per round.

Parallel-Hole Cuts with Large Empty Hole(S): Cylinder Cuts

The difference with the burn cut is that the uncharged or relief hole(s) is (are) larger than the charged holes. The large diameter blastholes (76–175 mm) are drilled with reamer bits which are adapted to the same drill steel which is used to drill the rest of blastholes.

All the blastholes in the cut are placed with little spacing, in line and parallel. Figure 18.6 shows some parallel-hole cuts with large relief holes.

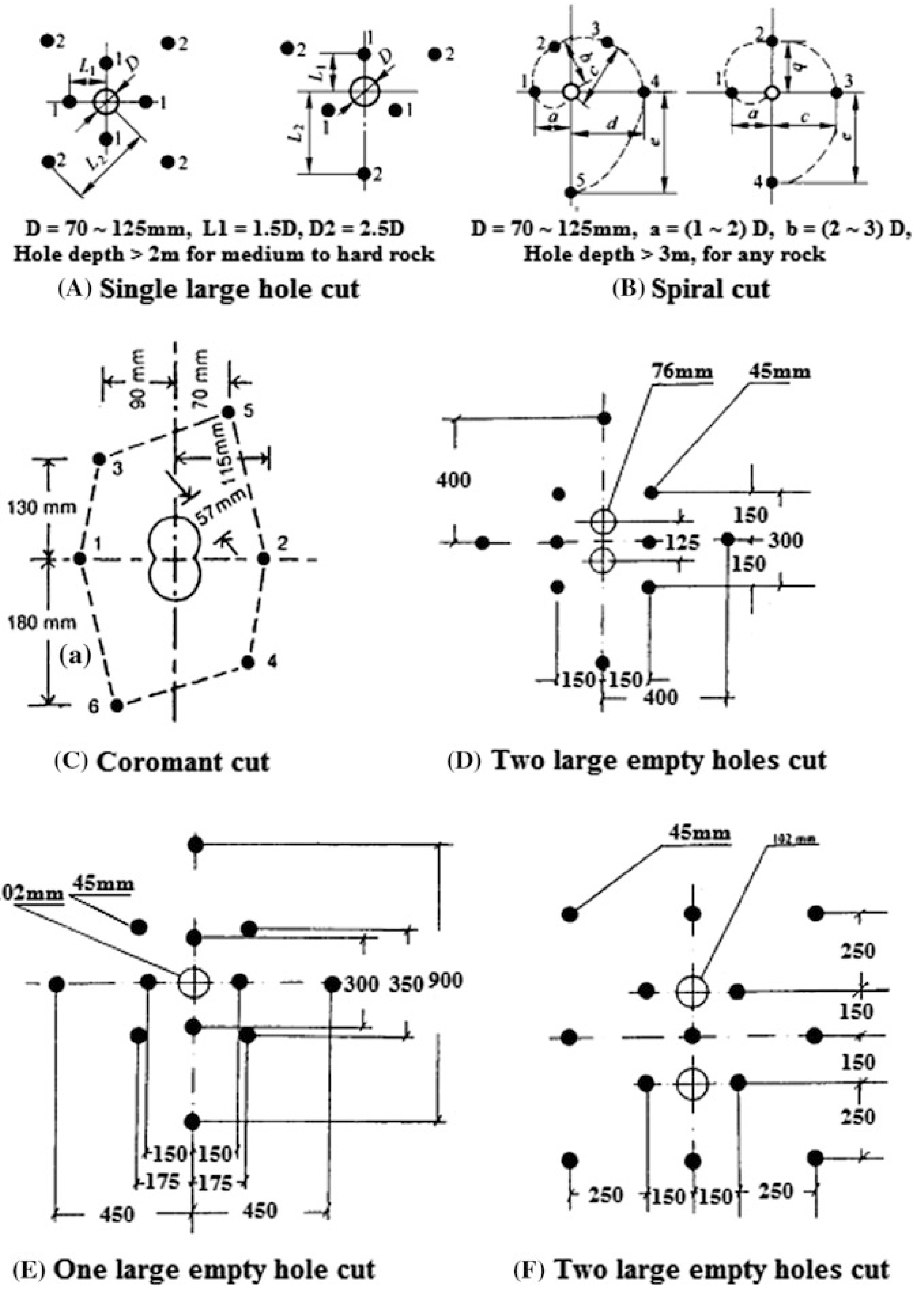


Fig. 18.5 Examples of parallel-hole cuts with large relief hole(s)

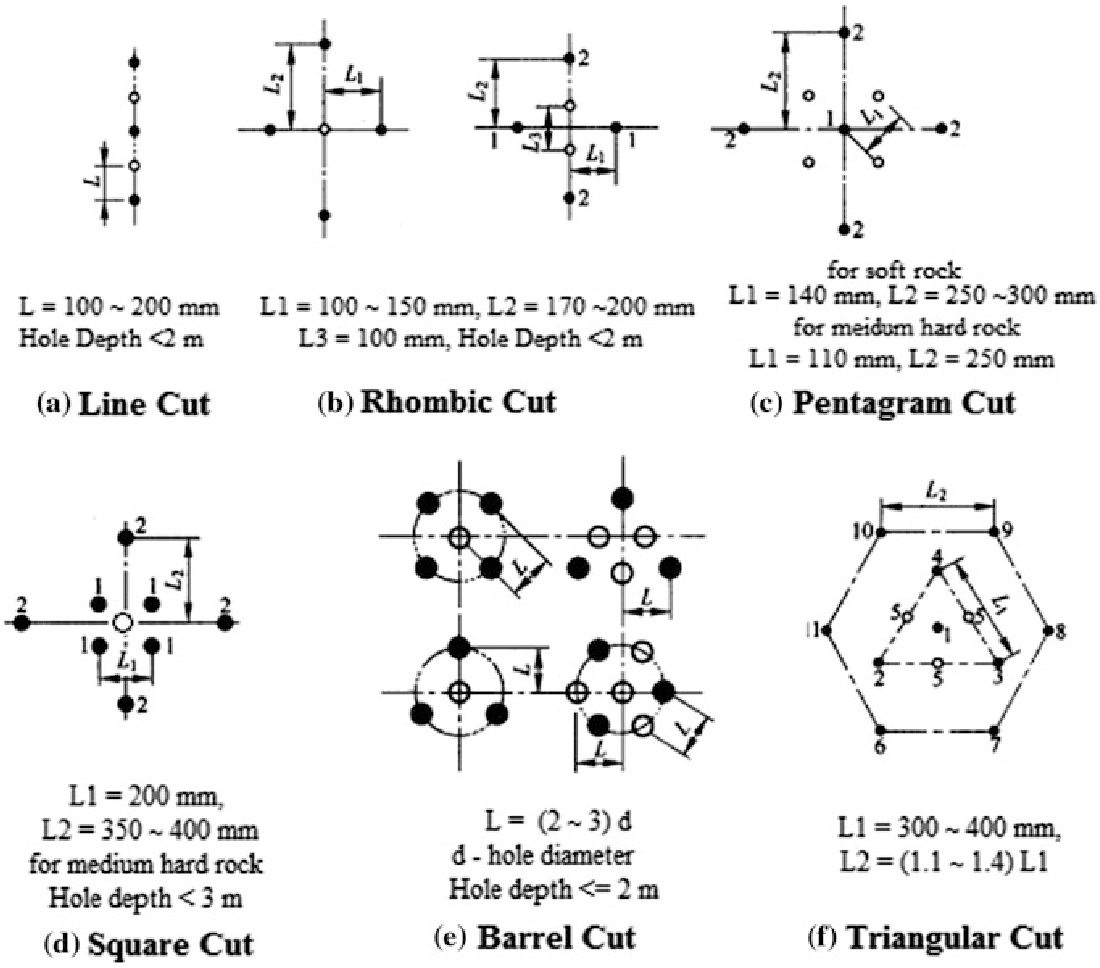


Fig. 18.6 Examples of burn cut

18.1.3 Some Important Issues on Cut Holes and Tunnel Blasting

18.1.3.1 The Concept of Cut "Freezing"

As the cut holes are tightly spaced and overloaded, some of the problems that can rise in blasting with parallel-hole cuts are sympathetic detonation and dynamic pressure desensitization. Any one of these two problems can cause the cut to "freeze" and fail. 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 those with nitroglycerine in their composition. On the other hand, the dynamic pressure desensitization takes place in many explosives, and water-based emulsion or water gel explosives are most susceptible to dead-pressing failure when they are used in closely spaced holes because the shock wave of a charge can elevate the density of the adjacent charge above the critical or death density (refer to Chap. 3, Sect. 3.7.8).

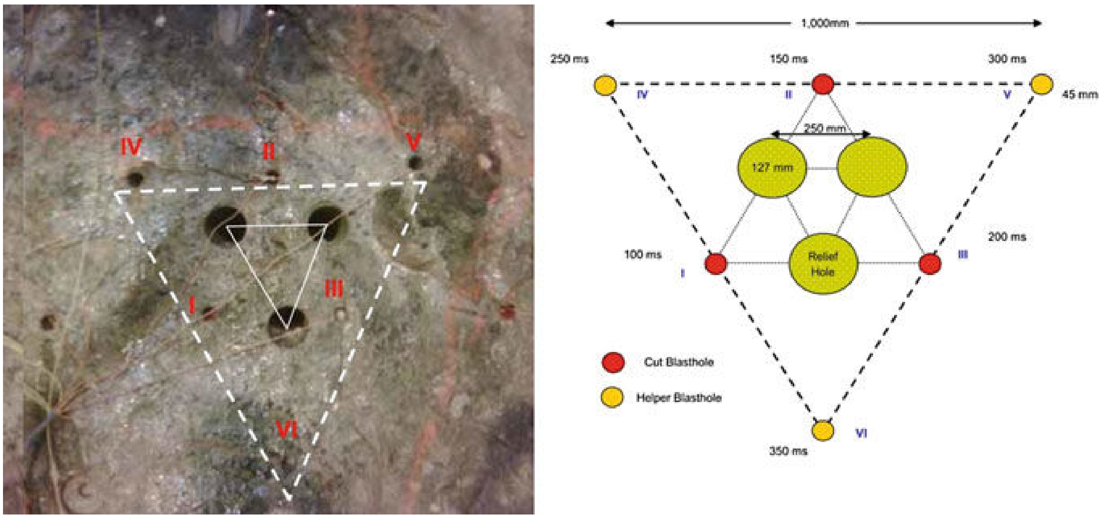


Fig. 18.7 Shielded burn-cut layout in hard rock recommended by Hagan (reproduced from Ref. [1] with the permission from Australian Institute of Mining and Metallurgy)

The following guidelines can be used to minimize cut "freezing":

1. Carefully align and drill all cut holes to ensure that they are parallel.
2. Provide more or large uncharged relief holes to accommodate the "swell" of broken rock.
3. Reduce the explosives energy per meter of blasthole in the cut (e.g., use smaller diameter packaged explosives in the cut holes).
4. Alter the geometry and spacing of the cut blastholes and relief holes, to allow for changes in ground conditions.
5. Ensure that blastholes in the cut area fire in a controlled sequence, with adequate time between successive detonations.

Figure 18.7 is suggested by Dr. T. N. Hagan [1] for eliminating sympathetic detonation and dynamic pressure desensitization.

If the cut fails due to precompression, try spreading the loaded holes farther apart. Adding more tightly spaced holes will aggravate the problem and unnecessarily increase the costs. In soft or seamy rock, adjacent loaded holes that are separately delayed should be at least 30 cm apart.

To pull rounds deeper than 2.5 m, use parallel-hole cut designs with an adequate volume of uncharged relief holes. In rounds exceeding 2.5 m, the relief-hole area should be at least 25 % of the total area in the immediate cut.

To further aid rock ejection from the burn, a small "kicker" or "booster" charge can be placed at the bottom of the normally empty void holes. These charges are delayed to fire just after the other fully loaded holes in the immediate cut have fired.

Generally, long-period delays are used to ensure that there is sufficient time for the rock from each hole, or group of holes, to break and be ejected from the cut, before subsequent holes fire. Some amount of bootleg normally occurs in the cut area. When this occurs, a similar or greater length of advance is usually lost in the rest of the face area. To minimize this lost advance, when drill steel length allows it, the cut holes should be overdrilled by 15–30 cm (6–12 in.). If this extra drilling for a few holes returns an equal length of advance for the whole face, it is well worth the investment in extra drilling.

18.1.3.2 Stemming

Stemming refers to an inert material that is placed in the borehole between the top of the explosive column and the collar of the hole.

There are two completely opposite views on the roles of blasthole stemming.

One view is that it is unnecessary to use stemming for blastholes in tunnel blasting. They said: "It has been discovered that the explosive used in tunneling the stemming does not have an improved effect for a long charged column. The inertia of the molecules within the air column in the drillhole (in relation to the very high detonation speed) is sufficient to act as the stemming." "Therefore the decision is often taken not to use stemming, in order to save cost and time" (refer to 5.6.4 Stemming on p. 169 of [2]).

But most practice has shown that it is necessary to confine the charge to localize the effect of the gases produced by the explosive reaction. When a stemmed blasthole detonates, the stemming momentarily retains the energy within the blasthole, only a few milliseconds but that is sufficient. When a blasthole is improperly stemmed or not stemmed at all, the resulting action is called "rifling." Rifling gets its name from the action of blowing the stemming material, much like a bullet from a rifle. The action of rifling can be flyrock, increased air overpressure, poor fragmentation, and boulders.

Stemming material can consist of sand, drill fines, gravel, or pea stone; sandy clay is usually used. Sometimes, wet cardboard, wet paper, or hessian bags are used as the stemming material, but practice shows they are not effective.

Practice has shown that using stemming of crushed rock granular with a blasthole-to-size ratio of 17:1 (refer to [3]) in thin plastic sausages can get the best result as the granular rock can "lock" in the blasthole due to its angular shape. Table 18.1 (refer to Table 5.5, p. 151 in [3]) gives the idea crushed rock sizes based on the blasthole diameter.

Table 18.1 Common stemming sizes based on hole diameter (reproduced from Ref. [3] with the permission from John Wiley & Sons Ltd)

Hole diameter (mm)	Hole diameter (in)	Size of stemming	Size of stemming
38	1.5	10 mm minus chips	3/8 in minus chips
50–90	2–3.5	10–13 mm chips	3/8–1/2 in chips
100–127	4–5	16 mm chips	5/8 in chips
127 and above	5 and above	19 mm minus chips	3/4 in minus chips

Source: From Atlas Powder Company (1987)

Generally, the amount of stemming material required will range from 0.7B to 1B, where B represents the burden or 10 times of the blasthole diameter. The rock characteristics affect the amount of stemming material, and more stemming amount is required in a highly fractured rock mass.

18.1.3.3 Lookout Angle

To maintain the designed cross section from one round to another, the contour holes have to be angled outside the designed cross section. This is to make sure that when drilling the next round, there is required space for the rock drill. If the contour holes were drilled parallel to the designed line of the tunnel, the tunnel face would get smaller and smaller after each round. The lookout is depending on the equipment used, but normally amounts to not less than 0.1–0.2 m (see Fig. 18.8). Modern drilling rigs have electronic or automatic lookout angle indicators that enable correct adjustment of the lookout angle relative to standard alignment. Computerized drilling jumbos make setting, adjustment, and monitoring of the lookout angle even easier.

18.1.4 Parallel-Hole Cut Design: Cylinder Cuts

The parallel-hole cut has a large number of minor variations, but the basic layout always involves the drilling of one or several uncharged large diameter holes at or very near the center of the cut. These holes give empty space for the adjacent blasted holes to swell into. As the drilling equipment has become more and more powerful, this type of cut has become more and more used. The large diameter holes (65–175 mm) are drilled with reamer bits which are adapted to the same drill steel which is used to drill the rest of blastholes.

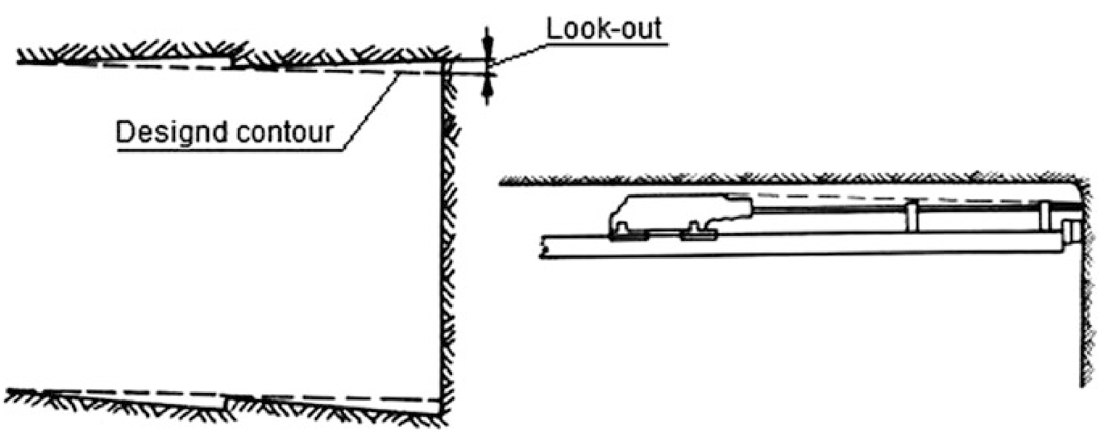


Fig. 18.8 The lookout angle (reproduced from Ref. [14] with the permission from Sandvik)

18.1.4.1 Large Hole Diameter and Number

It is proved that the diameter of the large hole should be a function of the blasthole depth used. In order to reach an acceptable advance/round 95 % of the blasthole depth, an "equivalent" hole diameter, d_f, can be calculated:

$$d_f \approx (3.2 \times l)^2 \tag{18.1}$$

where

d_f equivalent hole diameter, in mm;
l blasthole depth, in m.

Individual hole diameter now can be calculated as follows:

$$d_l = d_f / \sqrt{n} \tag{18.2}$$

where

n number of large holes.

18.1.4.2 Spacing of Cut Holes

The charged holes closest to the large hole(s) are named "cut holes."

Generally, the center-to-center distance between the empty large hole and the cut hole should be about 1.5 times the diameter of the large empty hole. That means the burden of the cut holes, v, should be as follows:

$$v = 1.5 \times d_f = 1.5 \times d_l \times \sqrt{n} \tag{18.3}$$

For two or more large holes, v is calculated for the holes marked in black in Fig. 18.9 and the remaining holes are added in order to achieve a square system.

U. Langefors and B. Kilhström indicated that v should not be more than 1.7 d_f to obtain fragmentation and a satisfactory movement of the rock [4]. The conditions of fragmentation vary greatly depending upon the type of explosive, rock properties,

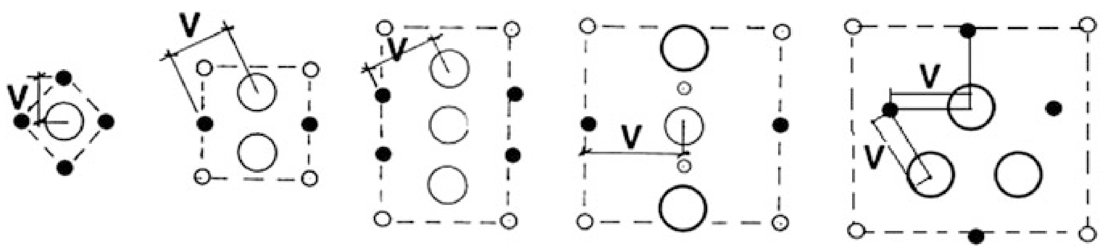


Fig. 18.9 Burden of cut holes

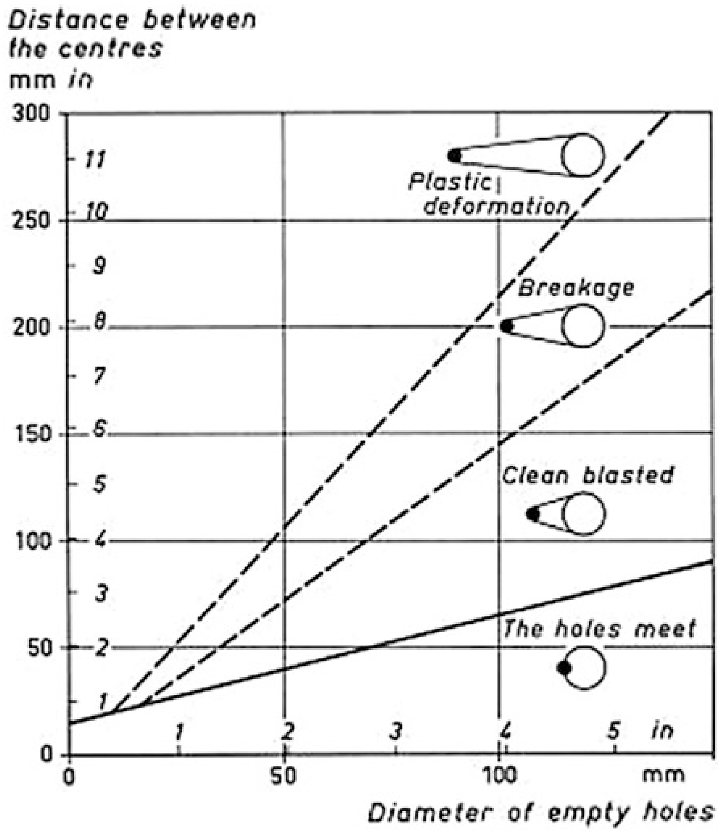


Fig. 18.10 Result when blasting toward an empty hole at different distance and dimension of the empty hole (reproduced from Ref. [4] with the permission from John Wiley & Sons Ltd

and the distance between the charged blastholes and the large empty hole. As reflected in Fig. 18.10, for burdens larger than 2 d_f, the break angle is too small and a plastic deformation of the rock between the two holes is produced.

It is quite obvious that the accuracy when drilling these holes is extremely important. When drilling deviation is more than 1 %, the practical burden is calculated from:

$$v_1 = 1.7d_f - E_p = 1.7d_f - (\alpha \times L + e^{'}) \quad (18.4)$$

where E_p is the drilling error (m), α is the angular deviation (m/m), L is the blasthole depth (m), and $e^{'}$ is the collaring error (m) (from [5]).

18.1.4.3 Four-Section Design for Parallel-Hole Cut

In order to reach an opening enough for stoping hole blasting, some cut spreader holes are needed. For the convenience of arranging these spreader holes around the cut holes, a number of quadrants are arranged (refer to Fig. 18.11).

Four-section cut is an empirical method for blasting design in underground excavations and tunnels. This method has often been used for excavating tunnels with cross-sectional area of more than 10 m^2.

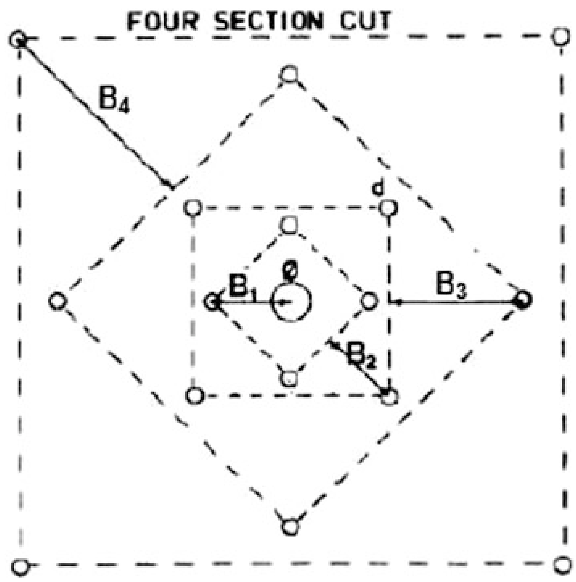


Fig. 18.11 Four-section cut method

Table 18.2 Equation for blasting pattern design of four-section cut model (refer to [15])

Section	Burden (B)	Spacing (S, X)	Stemming (S_t)
First square cut	$B_1 = 1.5\emptyset_{e2}$	$X_1 = B_1\sqrt{2}$	$S_{t1} = B_1$
Second square	$B_2 = B_1\sqrt{2}$	$X_2 = 1.5B_2\sqrt{2}$	$S_{t2} = B_1\frac{\sqrt{2}}{2}$
Third square	$B_3 = 1.5B_2\sqrt{2}$	$X_3 = 1.5B_3\sqrt{2}$	$S_{t3} = \frac{\sqrt{2}}{2}(B_1\frac{\sqrt{2}}{2} + \mathrm{B}_2)$
Fourth square	$B_4 = 1.5B_3\sqrt{2}$	$X_4 = 1.5B_4\sqrt{2}$	$S_{t4} = \frac{\sqrt{2}}{2}(\frac{\sqrt{2}}{2}\left(B_1\frac{\sqrt{2}}{2} + \mathrm{B}_2\right) + \mathrm{B}_3)$

The four-section cut is based on the parallel-hole cut. This model started with Lagnefors and Kihlström in 1963 [5] and has been further developed afterward.

The method suggests the experimental equations listed in Table 18.2. In this table, X is the length of each quadrangle side (see Fig. 18.11).

Four-section cut method includes an empty large hole in the center. If the number of empty holes is more than one, equivalent diameter is calculated by the Eq. (18.2), i.e., $d_f = d_l\sqrt{n}$. In Table 18.2, $\emptyset_{e2} = d_f$ is the diameter of the equivalent hole.

18.1.5 Blasthole Pattern for Stoping

The object of the cut is to create a free surface toward which the rest of the blasting can be carried out. The purpose of stoping holes is to attain just as large an advance with the remainder of the round as created by the cut holes, to get a satisfactory fragmentation, and to get a suitable disposal of the broken rock. At the same time, the remaining rock face should also be left undamaged.

Table 18.3 Concentration of the charge in kg/m for various burden (*V*) and extension (*B*) of the free face (preliminary table; reproduced from Ref. [4] with the permission from John Wiley & Sons Ltd)

Max. burden (V)		Extension (*B*)											
		m	0.10	0.15	0.20	0.25	0.30	0.35	0.40	0.50	0.60	0.80	1.4
		ft	0.3	0.5	0.7	0.8	1	1.2	1.3	1.7	2	2.7	4.7
m	ft	Concentration of the charge, kg/m (×2/3 for lb/ft)											
0.10	0.3		0.12	0.08	0.06								
0.15	0.5		0.30	0.18	0.13	0.11	0.09						
0.20	0.7		0.60	0.35	0.24	0.20	0.16	0.14	0.12				
0.25	0.8		1.0	0.60	0.35	0.30	0.26	0.22	0.18				
0.30	1		1.3	0.9	0.60	0.50	0.35	0.31	0.26	0.22	0.18		
0.35	1.2			1.2	0.9	0.65	0.45	0.40	0.35	0.30	0.25		
0.40	1.3			1.6	1.2	0.9	0.7	0.6	0.50	0.40	0.30	0.24	
0.50	1.7				2.0	1.6	1.3	1.0	0.7	0.60	0.50	0.36	
0.60	2					2.2	1.9	1.6	1.3	1.0	0.7	0.52	
0.70	2.3						2.5	2.2	1.8	1.3	0.9	0.7	
0.80	2.7							3.2	2.4	1.8	1.4	1.0	0.6
1.00	3.3								4.0	3.0	2.4	1.4	0.9
1.20	4									4.4	3.8	2.5	1.2
1.40	4.7										5.0	3.6	1.6
1.60	5.3											4.8	2.4
2.00	6.7												4.0

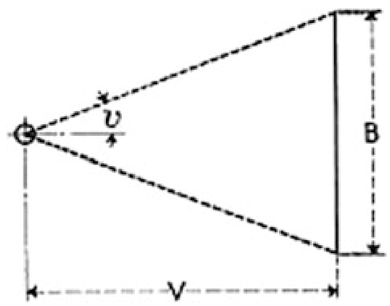


Fig. 18.12 Blasting toward a narrow opening

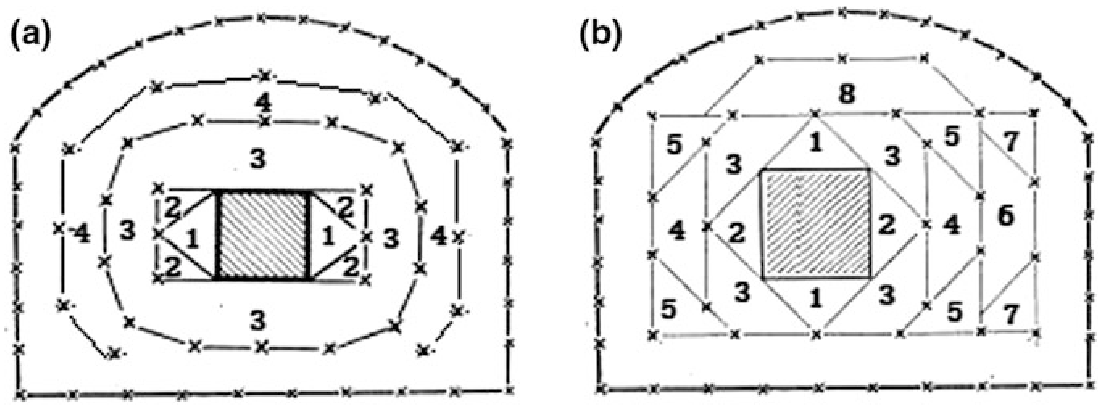


Fig. 18.13 Construction of stope pattern should be made as in the right-hand figure and not as in the left-hand one

The maximum burden of the stoping holes recommended by Langefors and Kihlström is given in Table 18.3 (reproduced from Table 7.2 in p. 187 of [4]) below and refer to Fig. 18.12.

The burden of the stoping holes should be not greater than the figure in the column of maximum burden (V). The best way to decide the place of stoping holes is using the principle of rectangularity. The right-hand stope pattern in Fig. 18.13 (from [4]) shows that in this way, the sequence of ignition to be used is clearly defined and tearing in the surrounding rock is reduced to a minimum. The left-hand stope pattern in this figure should be avoided.

18.1.6 Lifter Holes

Similar as the bench blasting, just taking the advance of the tunnel as the height of the bench height, the following equation can be used to calculate the burden of the lifter holes (from [6]):

$$B = 0.9\sqrt{\frac{q_e \times \mathrm{PRP_{ANFO}}}{\bar{c} \times f \times (S/B)}} \tag{18.5}$$

where

f	Fixation factor, generally 1.45, is taken to consider the gravitation effect and the delay timing between holes.
q_e	Explosive loading density (kg/dm^3).
$\mathrm{PRP_{ANFO}}$	Weight strength of explosive relative to ANFO (1–1.4).
S/B	Ratio of spacing/burden is usually considered equal 1.
$\bar{c}$	Rock constant calculated from c. Constant c is the quantity of explosive necessary to fragment one cubic meter of rock, normally in surface blasts and with hard rock c = 0.4 which is taken.

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

The burden B should comply with the condition of $B \leq 0.6L$, where L is the hole length.

In lifters, it is necessary to consider the lookout angle γ to give enough space for the rig drilling blastholes of the next round. For an advance of 3 m, an angle of 3°, which has an equivalent of 6 cm/m, is usually enough; however, it will depend upon the characteristics of the equipment, as shown in Fig. 18.14.

The number of lifter holes can be given by:

$$\mathrm{NB} = \text{integer of}\left[\frac{B + 2L \times \sin\gamma}{B} + 2\right] \tag{18.6}$$

The stemming is fixed in $T = 10 \times d_1$, where d_1 is the drillhole diameter.

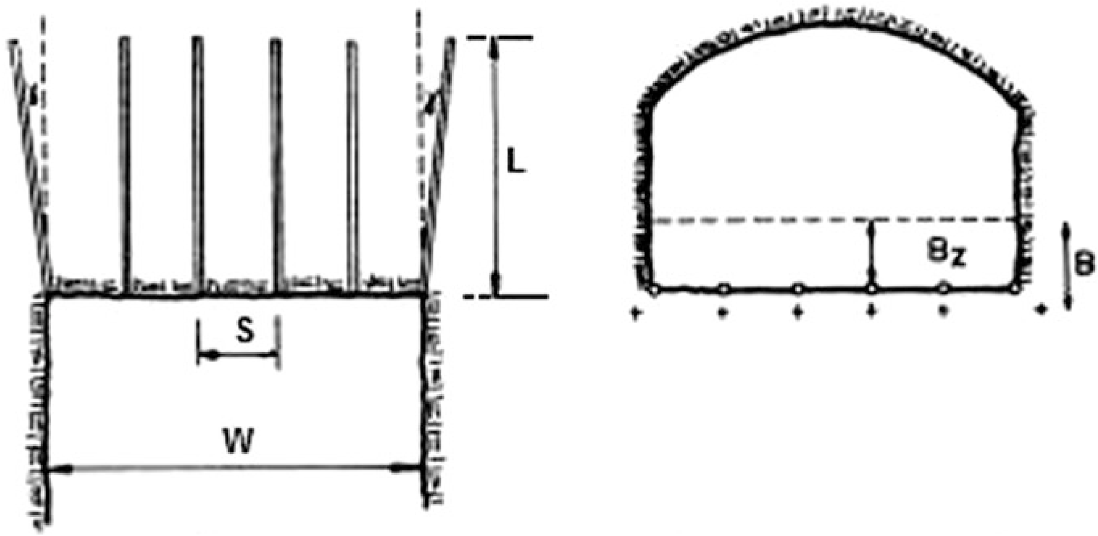


Fig. 18.14 Geometry of the lifters (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)

18.1.7 Contour Holes

The contour holes, also called perimeter holes, of tunnel blasting, especially the roof holes, are usually blasted using the smooth blasting method. The technique and parameters of smooth blasting have been discussed in Chap. 17.

If the blast does not need contour or smooth blasting, the parameters are calculated as for the lifters with the following values [6]: fixation factor, $f = 1.2$; S/B = 1.25; column charge concentration, $q_c = 0.5q_f$, where q_f is the bottom charge concentration.

18.1.8 Lineal Charge Concentration of Blasthole

18.1.8.1 Cut Holes

Langefors and Kihlström gave the guidelines of the charge concentration of the cut holes closest to the empty large holes in Table 18.4.

18.1.8.2 Stoping Holes

The recommended charge concentration of stoping holes by Langefors and Kihlström has been given in Table 18.3.

18.1.8.3 Lifter Holes

The lineal charge concentration is same as the stoping holes in practice. In fact, considering the gravitational and fixated effects, the burden and spacing are smaller than the stoping holes and the specific charge (kg/m^3 rock) is obviously higher than stoping holes.

Table 18.4 Concentration of charge (l) in kg/m for cylinder cuts and greatest distance (a) when blasting toward empty holes with diameter between $\varphi = 2 \times 57$ and 200 mm (d indicates the diameter of the loaded hole)

φ (mm)		50	2 × 57	75	83	100	2 × 75	110	125	150	200
D (mm)	32	0.2	0.3	0.3	0.35	0.4	0.45	0.45	0.5	0.6	0.8
	37	0.25	0.35	0.35	0.4	0.45	0.53	0.53	0.6	0.7	0.95
	45	0.30	0.42	0.42	0.50	0.55	0.65	0.65	0.7	0.85	1.10
a (mm)		90	150	130	145	175	200	190	220	250	330

The weight strength of the explosive is $s = 1.0$ (reproduced from Ref. [4] with the permission from John Wiley & Sons Ltd)

18.1.8.4 Contour Holes

If the tunnel blasting does not need smooth blasting, the charge concentration of the contour holes can use the following equation:

$$q_{lc} = 90 \times d_1^2 \tag{18.7}$$

where

q_{lc} lineal charge concentration of contour holes other than smooth blasting, in kg/m;
d_1 drillhole diameter, in m.

18.1.9 General Information for Tunnel Blasting Design

The following information can be used as a reference for the estimation of the tunnel blasting in the initial stage of a project and should be adjusted according to the geological condition and blasting practice along with the project progress.

18.1.9.1 Simplified Calculation for Designing Drilling and Blasting Pattern in Tunnels

The following table can be used as a quick initial design method for tunnel blasting with parallel-hole cuts (Table 18.5).

Table 18.5 Quick design of drilling and blasting pattern for tunnel blasting with parallel-hole cuts (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)

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

q_f—charge concentration in bottom of hole = $7.85 \times 10^{-4}\ xd^2\rho$; d—cartridge diameter (mm); ρ—explosive density (gm/cc); B—burden in stoping area, $B = 0.88\ q_f^{0.35}$; L—hole depth in the round
*In some cases, smooth blasting is essential and these relationships are not applicable

18.1.9.2 Relationship between Drillhole Number and Tunnel Cross-Sectional Area

- Figure of ICI's Handbook of Blasting Table [7] (Fig. 18.15)
- C. L. Jimeno et al. Figure [6] (Fig. 18.16)

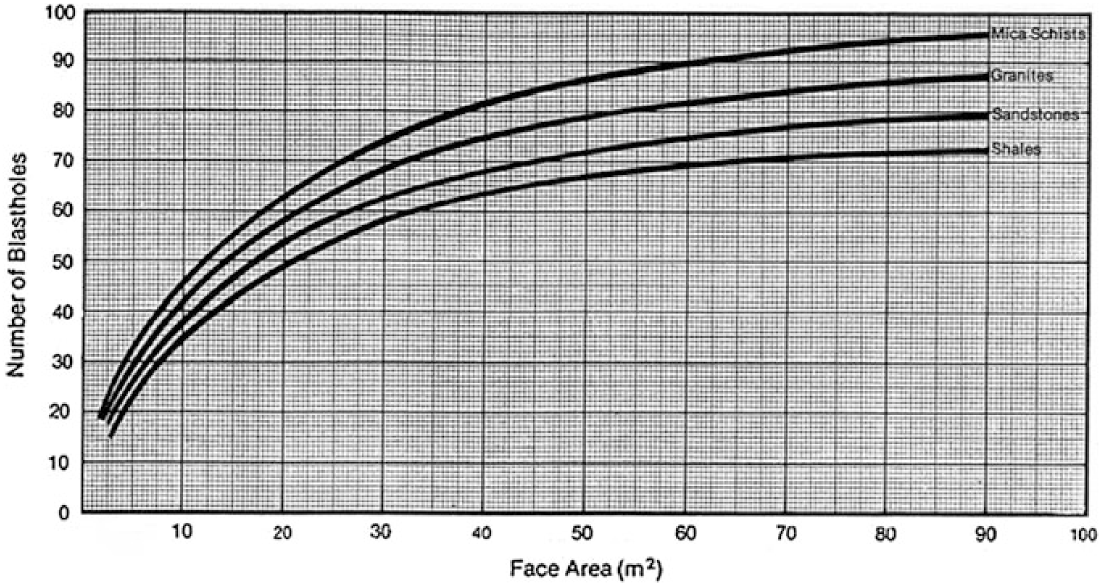


Fig. 18.15 Relationship of number of blastholes to face area (refer to [7])

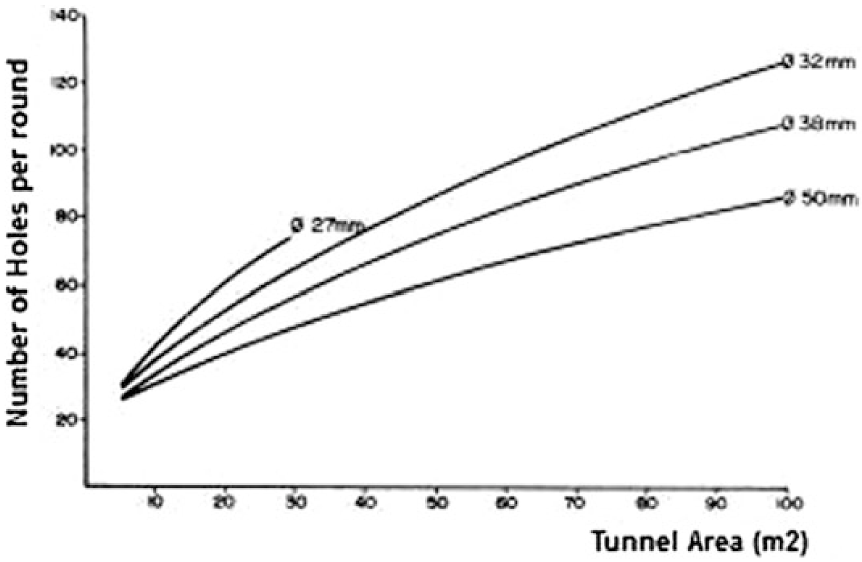


Fig. 18.16 Number of blastholes per round in function of tunnel's area (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)

18.1.9.3 Relationship between Explosives Consumption and Tunnel Cross-Sectional Area

- Figure of ICI's Handbook of Blasting Table [7] (Fig. 18.17)
- TAMROCK's Figure [8] (Fig. 18.18)

18.1.9.4 Specific Drilling

The following Fig. 18.19 is reproduced from Fig. 22.19 on p. 225 of [6]:

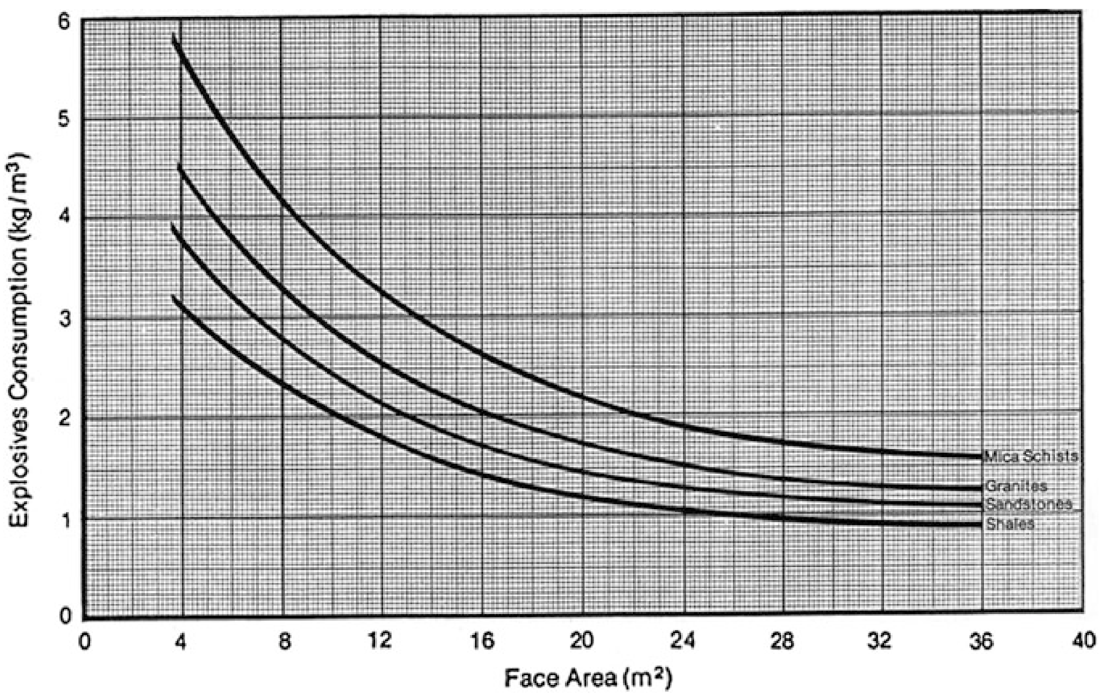
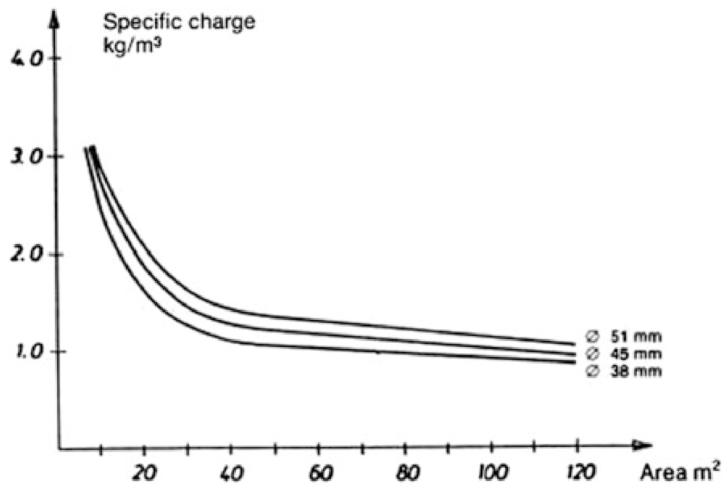


Fig. 18.17 Relationship of explosives consumption to face area (refer to [7])

Fig. 18.18 Powder factor as a function of tunnel area and blasthole diameter (reproduced from Ref. [8] with the permission from Sandvik)



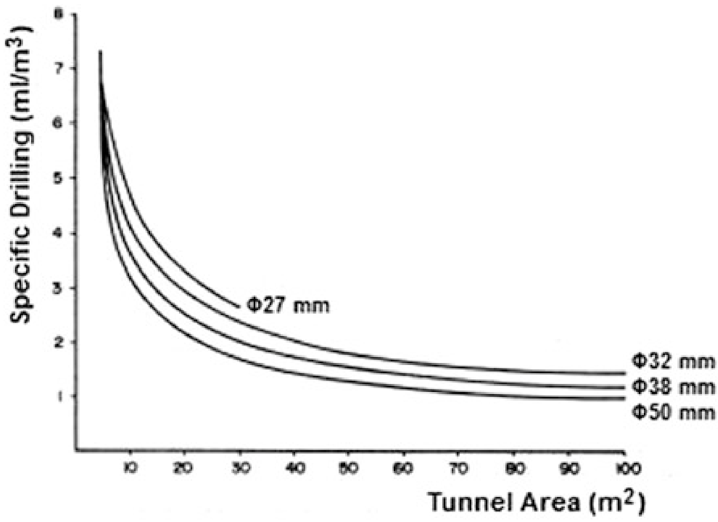


Fig. 18.19 Specific drilling as a function of the tunnel area and drillhole diameter (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)

18.2 Blasting Design for Shaft: Full Face Sinking

The full face method is used frequently in shaft sinking as it suits either rectangular- or round-section shafts. The principles described earlier for tunnel excavation may be applied with some modifications for special circumstances. As with tunnel blasting, the “cut” is critically important. There are various techniques for cut-hole design to create a free face with a few blastholes, V-cuts, or cone cuts, and parallel-hole cuts with relief holes.

18.2.1 Types of Cut-Hole Pattern

18.2.1.1 “V”-Cut

“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. Figure 18.20 is a sample of “V” cut.

18.2.1.2 Cone Cut

Cone cuts are used most in round-section shafts. It is easy to drill blastholes with a shaft jumbo. The holes are placed so as to form several inverted cone areas in the central part, as shown in Fig. 18.21.

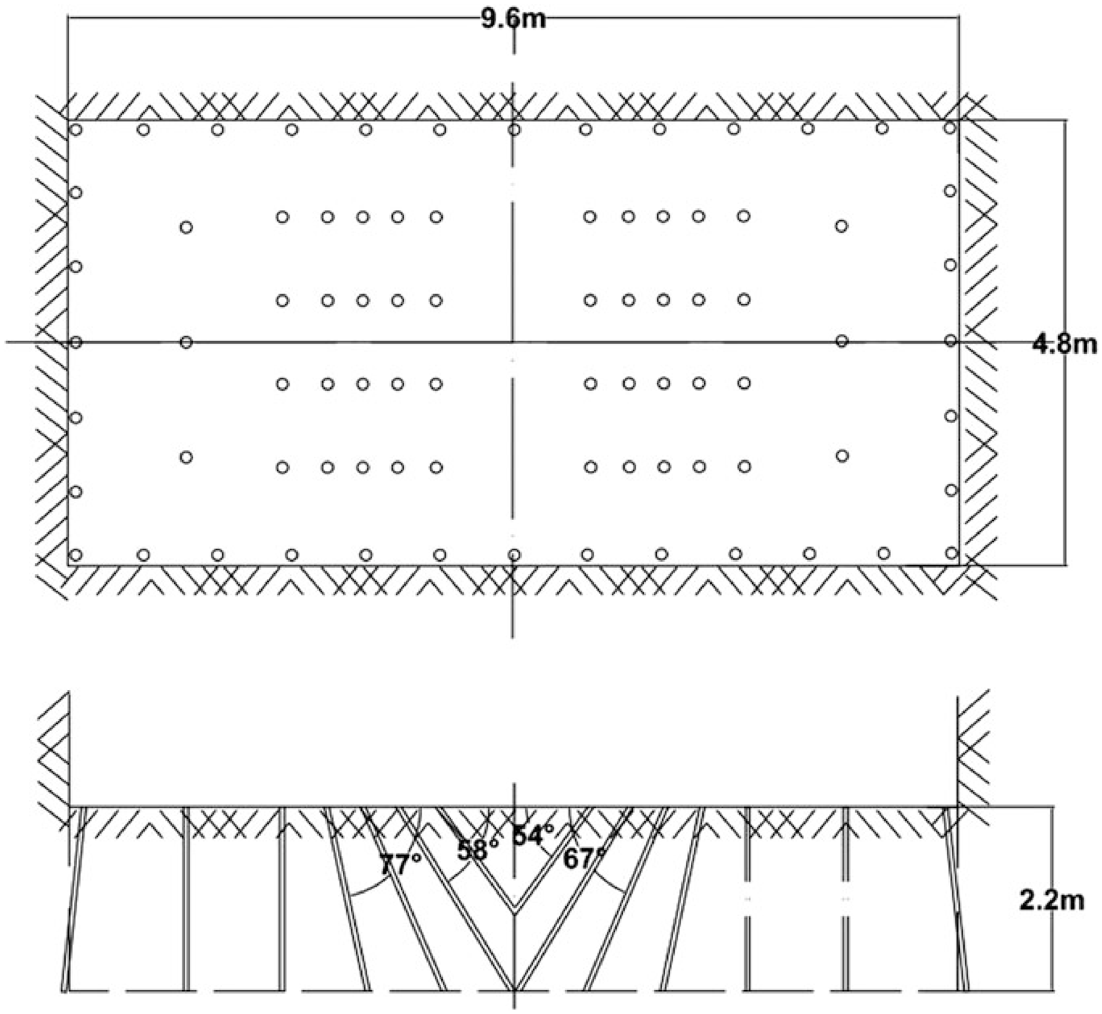


Fig. 18.20 Drilling pattern in a rectangular-section shaft

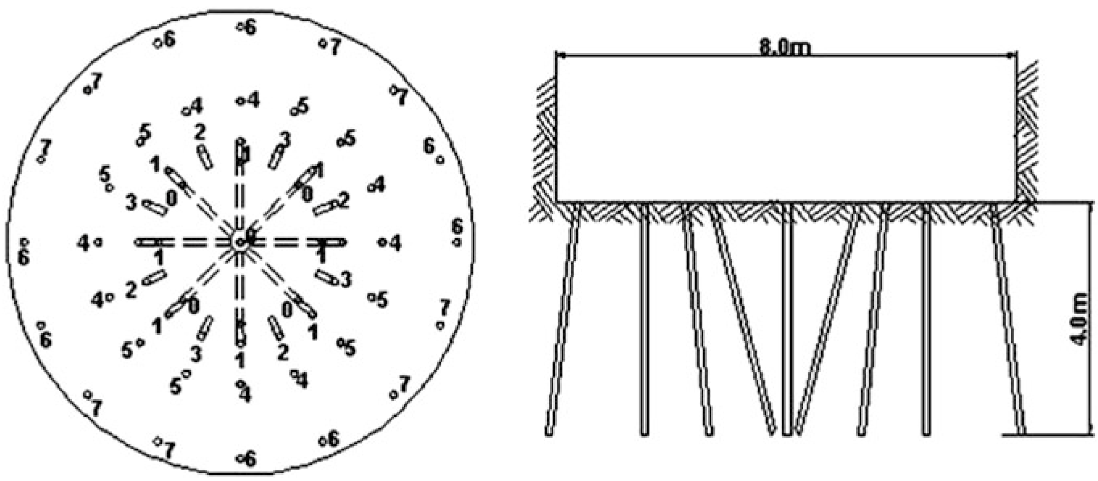


Fig. 18.21 Cone-cut drilling pattern

18.2.1.3 Parallel-Hole Cut with Relief Hole(S)

Similar to tunnel blasting, parallel-hole cut with large empty hole(s) is widely used in shaft blasting, especially for relatively small shafts. Figure 18.22 is a sample.

18.2.2 Blasting Parameters for Shaft Blasting

The pull of the rounds, as well as the number of blastholes, depends upon many factors such as the type of rock mass, the diameter of explosive charge, the blasting pattern, type of cut, the shaft size to be excavated, and the restriction of the surrounding environment (i.e., the charge weight per delay as the vibration limitation).

All blasting parameters should be adjusted time to time during the blasting practice to suit the above factors.

At the early stage, the following formula can be used to estimate the number of blastholes when 32-mm-diameter explosive charge is used:

$$NB = 2D_p^2 + 20 \tag{18.8}$$

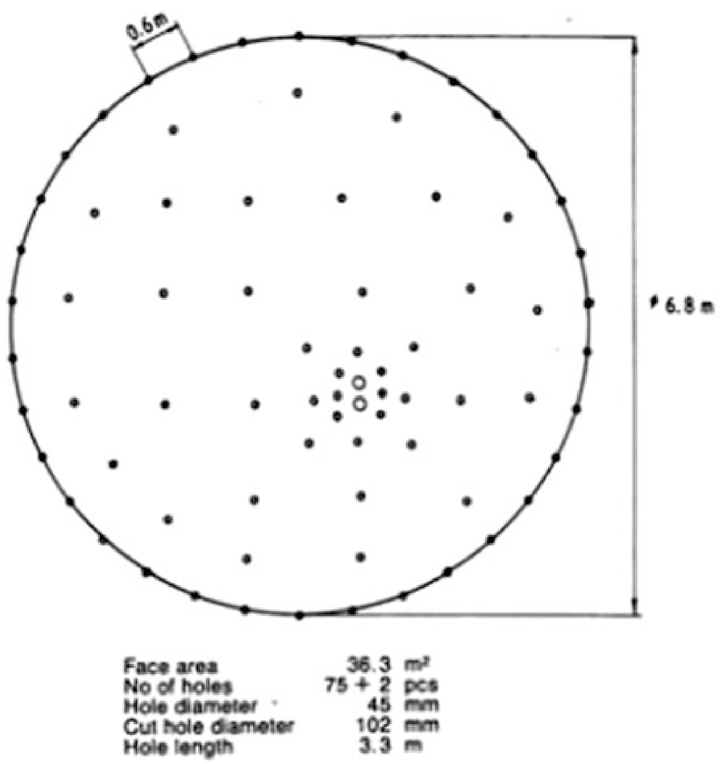


Fig. 18.22 Drillhole pattern using parallel-hole cut with an empty large hole (reproduced from Ref. [14] with the permission of Sandvik)

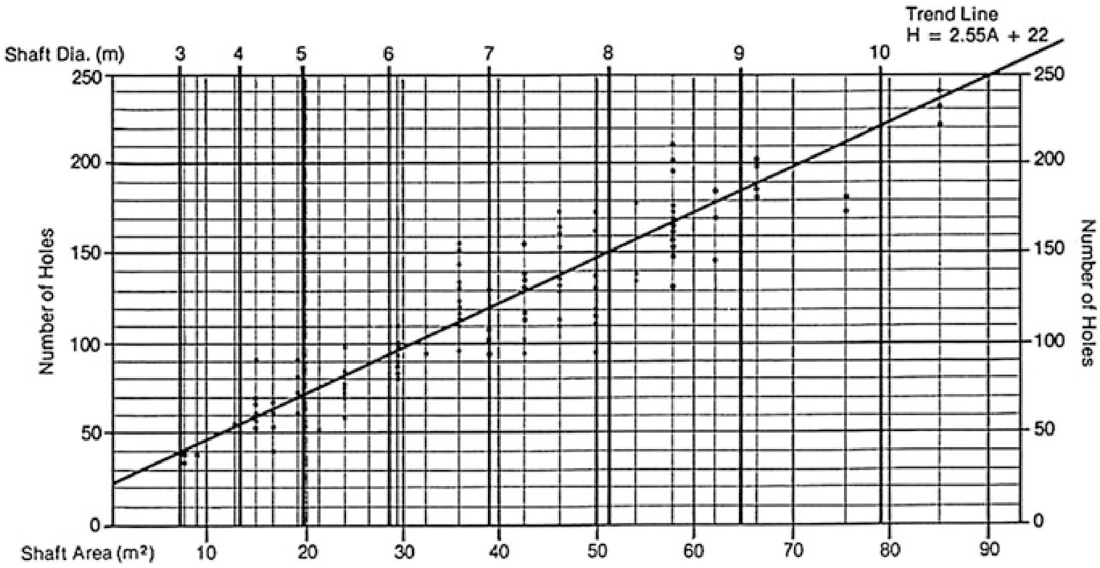
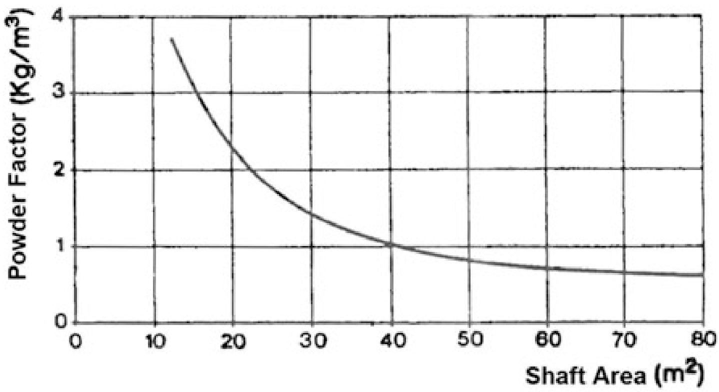


Fig. 18.23 Graph of number of holes in shaft of various size (*Source* [7])

Fig. 18.24 Round advance as function of shaft area (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)



where

NB number of blastholes excluding the perimeter holes if contour (smooth) blasting is carried out;

D_p shaft diameter (m).

The following graphs, as shown in Figs. 18.23, 18.24 and 18.25, can also be used in the early stage for estimating the blasting parameters of shafts.

18.3 Firing Sequence Design for Underground Blasting

18.3.1 Principle of Firing Sequence Design

As there is only one free face in underground blasting (excluding benching), the blastholes should be fired in a certain sequence. The firing pattern must be designed

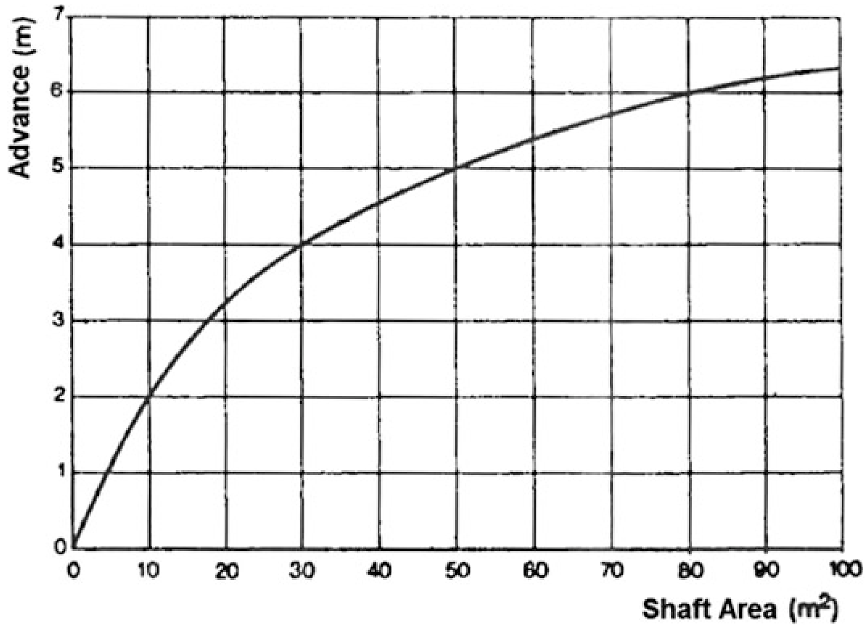


Fig. 18.25 Powder factor as function of shaft area (reproduced from Ref. [6] with the permission from Taylor & Francis Book UK)

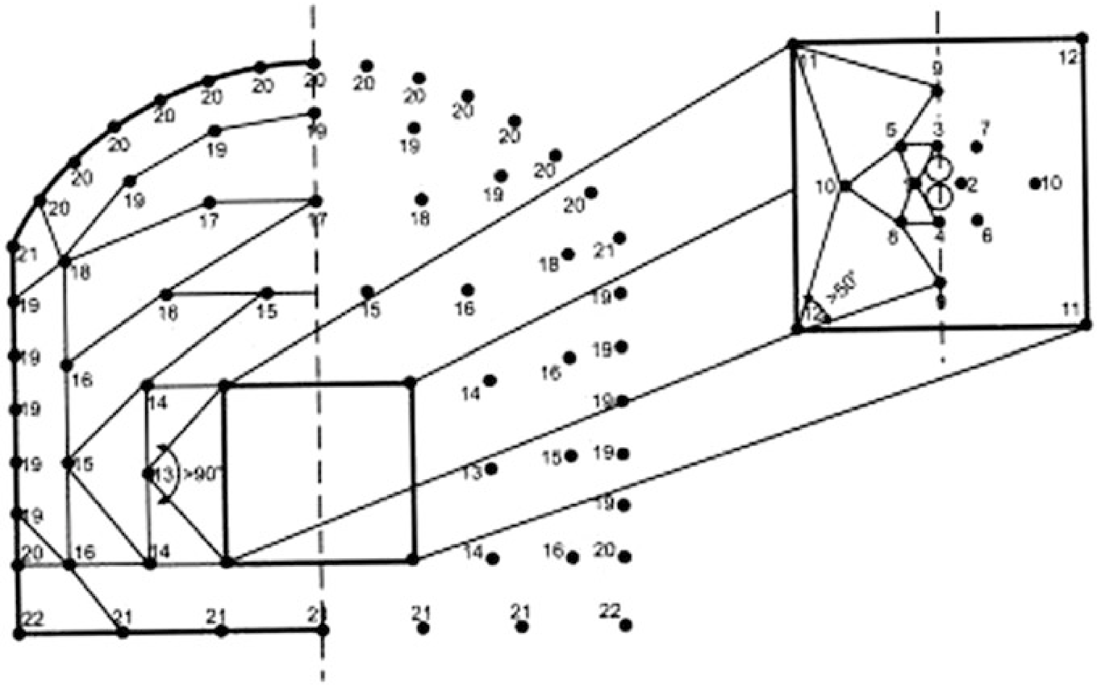


Fig. 18.26 Firing sequence for small tunnel in numerical order (reproduced from Ref. [14] with the permission from Sandvik)

so that each hole has free breakage. The angle of breakage is smallest in the cut area where it is around 50°. In the stoping area, the firing pattern should be designed so that the angle of breakage does not fall below 90° (see Fig. 18.26).

It is important in tunnel blasting to have a long enough time delay between the holes. In the cut area, it must be long enough to allow time for breakage and rock throw through the narrow empty hole. It has been proven that the rock moves with a velocity of 40–70 m/s. A cut drilled to a depth 4–5 m would therefore require a delay time of 60–100 ms to be clean blasted. Normally, delay times of 75–100 ms are used.

In the first two squares of the cut, only one detonator for each delay should be used. In the following 2 squares, two detonators may be used. In the stoping area, the delay must be long enough for the rock movement. Normally, the delay time is 100–500 ms.

For contour holes, the scatter in delay between the holes should be as small as possible to obtain a good smooth blasting effect.

The general firing sequence is as follows:

For tunnel (cavern):

Cut holes → Cut spreader holes → Stoping holes → Wall holes → Roof contour holes → Lifter holes.

For shaft:

Cut holes → Cut spreader holes → Stoping holes → Wall contour holes.

There are some important principles that must be kept in mind when designing the firing sequence:

(a) The firing sequence must start from the central cut holes and then progress outward to the tunnel contour gradually.
(b) The minimal time interval of two adjacent holes must be no less than 25 ms, especially in shaft blasting with wet holes to avoid the "water hammer effect."
(c) For the large tunnel or cavern, the blastholes on the working face can be divided into several groups. The shock tubes of all holes in each group are bunched together and connected with a surface delay connector (refer to Sect. 18.3.3 below). The delay time of the first firing in-hole detonator must longer than the longest delay time among all surface detonators; otherwise, the first fired blasthole will damage the initiation net for the tunnel blasting.
(d) If there is a restriction of ground vibration, the maximum explosive charge weight per delay of any simultaneously fired holes must be no greater than the allowable charge weight per delay.

18.3.2 Small Tunnel

If it is possible to have enough numbers of required delay intervals for a blasting round, all blastholes of the whole tunnel face can be fired in one time. Fig. 18.26 is a sample.

18.3.3 Large Tunnel

For the large tunnel or cavern, the working face can be divided into several sectors. There are two kinds of connection methods: bunch connector method and detonating cord ring connection method.

18.3.3.1 Bunch Connector Method

The shock tubes of all holes in each sector are bunched together and connected with a surface delay connector (see Fig. 18.27).

The following figure (Fig. 18.28) is a good example of using bunch connectors to fire a large cross-sectional tunnel.

18.3.3.2 Detonating Cord Ring Connection Method

In this method, all shock tubes have a “J” hook and all detonators within a delay sector are connected to a ring (circle) of two strands of 5 gm/m detonating cord (see Fig. 18.29).

- The shock tube from the detonators has “J” hooks at their end which clip onto the detonating cord (see Fig. 18.30).
- The “ring” of detonating cord should be flush with the face and should not touch any other shock tube tail hanging from an adjacent delay sector.
- Delay sectors are linked by the appropriate non-electric surface delay connector.
- The surface connectors are attached to a 0-ms delay bunch block which initiates the detonating cord.

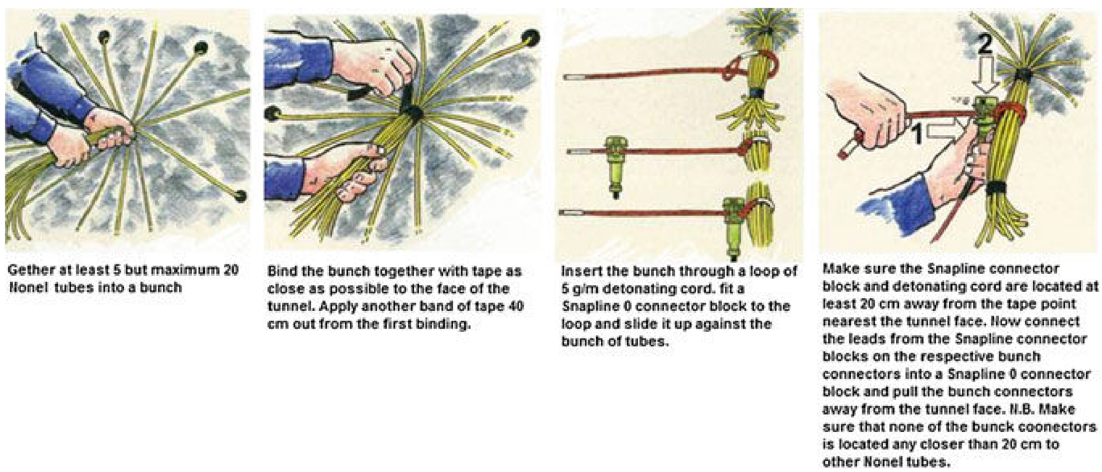


Fig. 18.27 Initiation by means of bunch connectors for a large section of tunnel (courtesy of “Nonel User’s Guide” of Dyno Nobel)

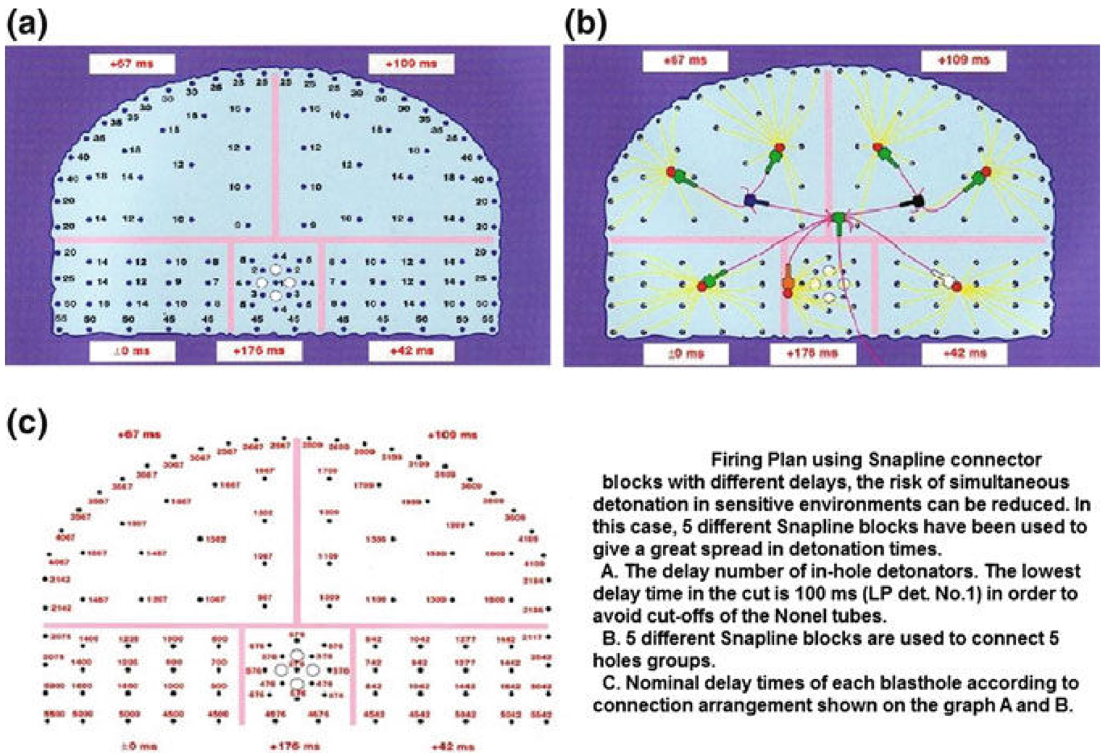
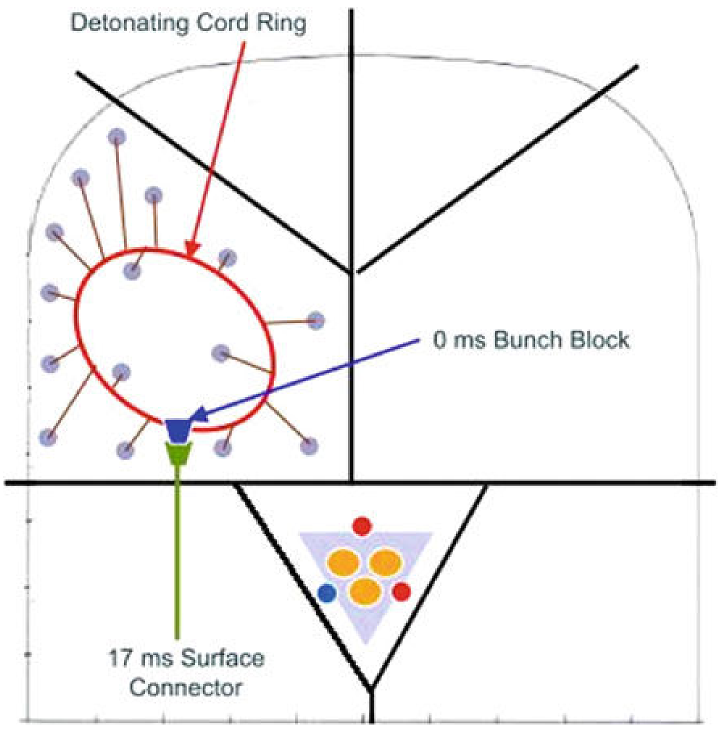


Fig. 18.28 Firing plan using snapline connector blocks with different delays (courtesy of Dyno Nobel)

Fig. 18.29 Detonating cord ring connection method (courtesy of Orica)



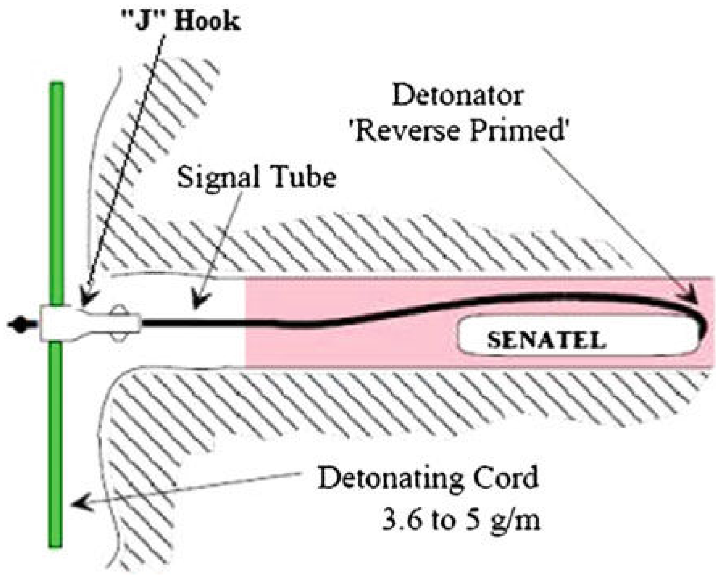


Fig. 18.30 Detonator's "J" hook clips the detonating cord (courtesy of Orica)

18.3.4 Tunnel Blasting with Electronic Detonators

Electronic detonators possess the advantages of high precision in delay time so that the blasting is more guaranteed for safety, particularly for ground vibration control. It is more and more used for construction blasting, including underground blasting, in sensitive environments, especially in urban areas. The following examples briefly show how electronic detonators are used in tunnel blasting.

18.3.4.1 Example 1: SmartShot Electronic Detonators' Tunnel Blasting Design

Referring to Fig. 18.31, all blastholes are divided into 5 subsectors.

The initiation sequence is as follows: SS3 > SS4 > SS1 > SS5 > SS2. Their igniting times are as follows: 0, 2151, 2181, 2593 and 2691 ms, respectively, with different delay intervals between holes in each sector.

18.3.4.2 Double-Deck Tunnel Blasting Using Electronic Detonators [9]

Fig. 18.33 is a tunnel blasting design using Orica's eDev™ electronic detonators. This tunnel excavation was carried out in the busy urban area of Hong Kong in 2013. Due to the complex environment of the tunnel project, the surrounding highly sensitive receivers made the allowable maximum instantaneous charge (MIC) for tunnel blasting very low, only 0.2–0.4 kg/delay in some tunnel sections.

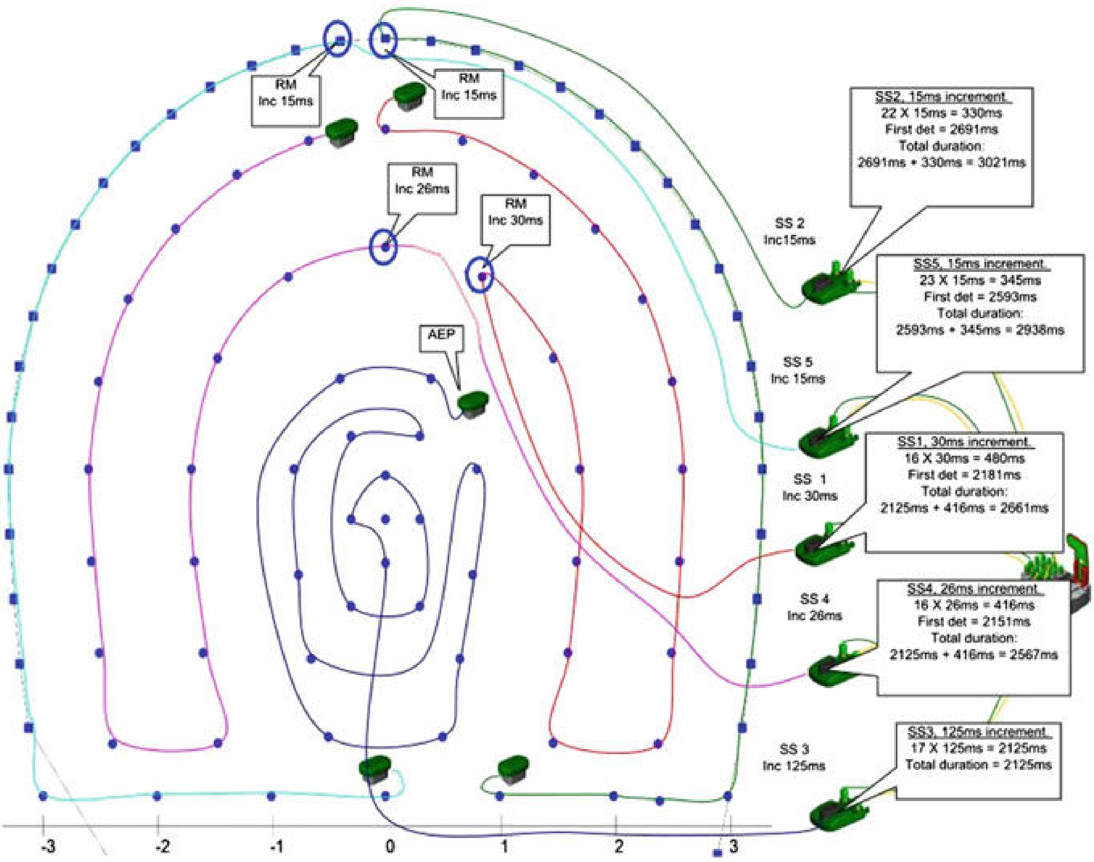
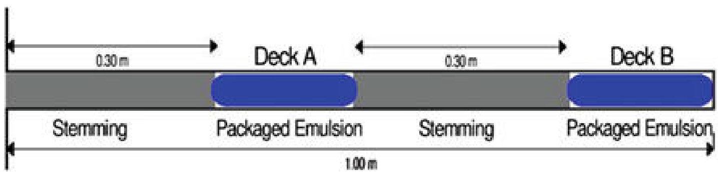


Fig. 18.31 Tunnel firing sequence design using SmartShot Electronic System (Courtesy of AMS)

Fig. 18.32 Blasthole charging structure (reproduced from Ref. [9], courtesy of ISEE)



In this design, double-deck blasting technique was applied to increase the advance of each round. The charging structure is shown in Fig. 18.32. The allowable MIC is 0.2 kg per delay. The drillhole and firing sequence pattern are shown in Fig. 18.33.

Holes surrounding the cut were moved closer, reducing burdens, and the initiation sequence was changed substantially. Commencing with hole by hole firing, the angle of initiation was gradually opened, as shown in the figure. In three dimensions, this results in expanding conical angles of initiation.

Loading and tie in time for this blast, also consisting of 358 decks, was 185 min; a small amount of experience with the previous blast offered immediate improvement in productivity. There were no notable safety issues or increase in risk. All measured ground vibration was below 1 mm/s, or monitor thresholds were not triggered. The majority of this blast pulled 100 % or more than the drilled 1.0 m length. There were very few small butts (sockets) remaining in the face (refer to [9]).

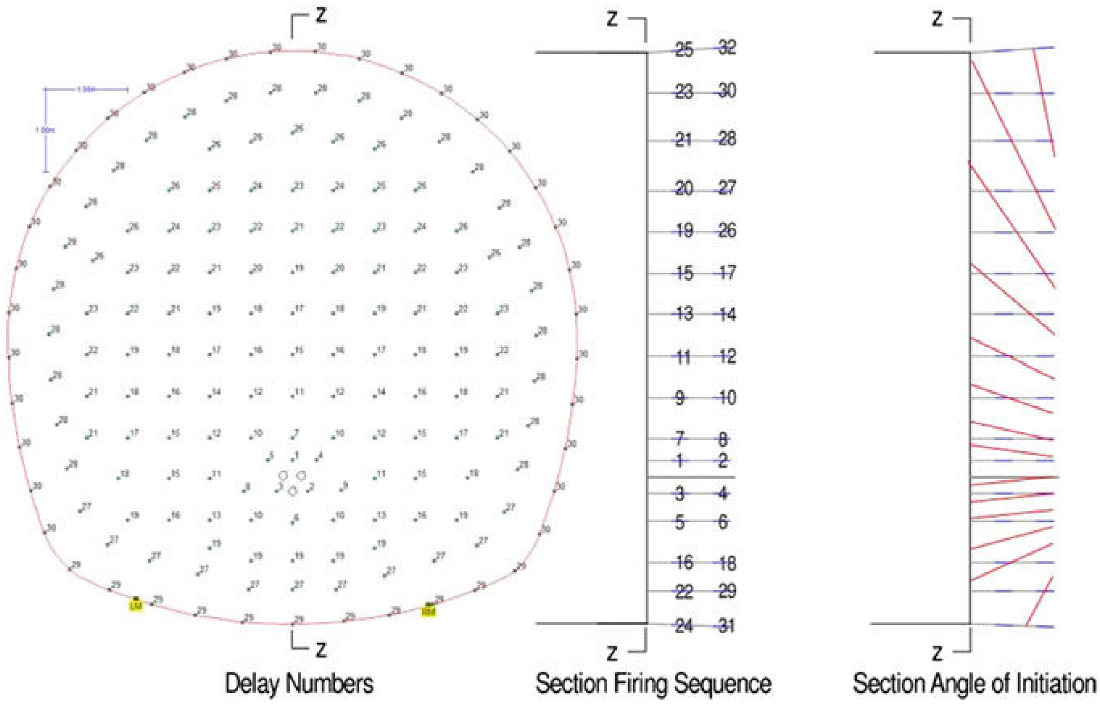


Fig. 18.33 Tunnel blasting design using double charging technique and electronic detonators (reproduced from Ref. [9], courtesy of ISEE)

18.4 Computer-Aided Tunnel Design and Management

Some drill manufacturers together with some universities have developed some software to help the engineer to do the works of tunnel blasting design and management. The manufacturers supply the software together with the drill equipment and train the engineer, drillers, and shotfirers.

Among them, Atlas Copco's "Underground Manager" and Sandvik's "iSURE®" are the most popular software. Their prominent feature is that the software is integrated with the drill operation system, and it not only guides the drilling machine to drill blastholes in accordance with the designed pattern accurately, but also collects all real-time available drilling information during the drilling process, using the MWD (Measure While Drilling) data acquisition systems, to analyze and assess the rock properties and geological features of the rock mass for guiding tunnel blasting design.

iSURE® and Underground Manager are based on Windows so the user who are familiar with windows system will not have difficult to master them.

18.4.1 Sandvik ISURE® Software: Tunnel Management Software (Reproduced from Refs. [10, 11] with the permission from Sandvik)

DTi (iSeries) is Sandvik's automated multipurpose construction drill rig. DTi works seamlessly with iSURE® (**i**ntelligent **S**andvik **U**nderground **R**ock **E**xcavation) software.

The idea behind iSURE® is to offer a tool to optimize the drill plan in a practical way and produce all the necessary information to follow up and improve the drill and blast work cycle.

iSURE® package consists of four models:

- iSURE® I Tunnel,
- iSURE® II Report (require model I),
- iSURE® III Analysis (require models I and II), and
- iSURE® IV Bolting (DTi-series only, require models I, II, and III).

Recently, Sandvik Construction has added a rig-integrated, high precision, online rock mass analysis and visualization system to its existing offering for tunneling process optimization—geoSURE. This new option is fully integrated to the iSURE® tunneling project management software, providing rock mass information and a view inside the drilled rock. Its unique features improve the overall tunneling process in terms of efficiency and quality.

- iSURE® I Tunnel: Drill and Blast Design

Tunnel is the basic module of iSURE®; it will always come with the package. It includes project files management, tunnel profiles, tunnel location, drill and blast design, and drilling and blasting patterns. This module offers one of the most revolutionary features in the iSURE®: pattern design in the end of the round, providing hole burden calculation and optimization of hole location (Fig. 18.34).

The design of the theoretical profile can be drawn manually or chosen from the standard profiles provided in iSURE®. It is also possible to import a profile in .dxf format from AutoCad. As part of the drill plan design, the tunnel module also includes detonator design into which surface delay design can be incorporated. It significantly speeds up the design work and decreases the number of errors. The charger can be supplied with the reports on the explosives and detonator demand. This allows explosives planning to start immediately.

In the drill pattern, iSURE® offers a possibility to define a range of different drilling types such as contour holes, field holes, and grouting holes.

ISURE® pullout analysis software (DTi-series only) can compare the start positions of holes to end positions of the previous round. This can be used to modify the charging or hole distances in program areas and improve the productivity.

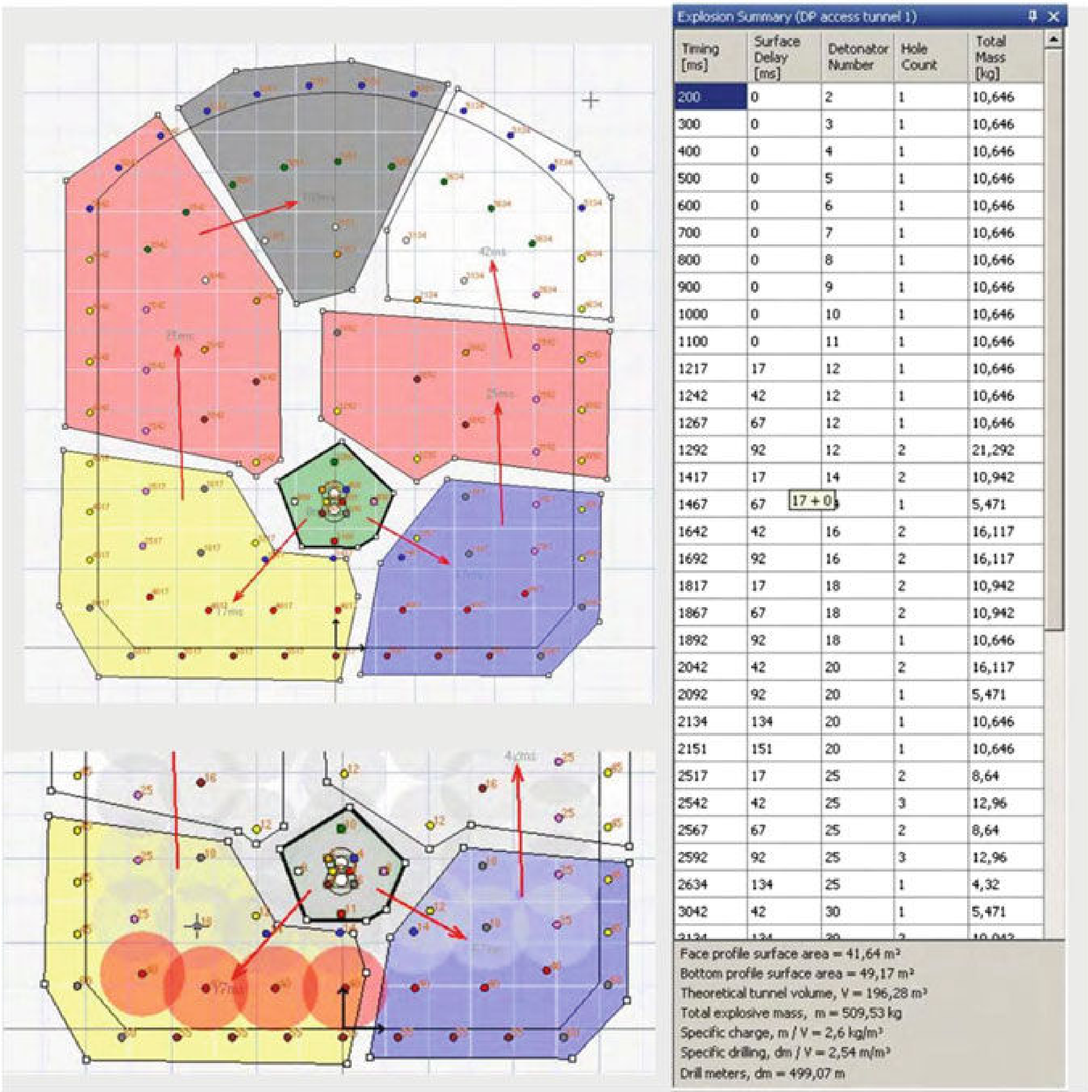


Fig. 18.34 Use iSure tunnel module to do drilling and blasting design-firing sequence design (reproduced fro Ref. [11] with the permission from SANDVIK)

- iSURE® Report: Process Control and Reporting

The report module supplies clear reports that can be used directly for reporting. Information on the actual drilling process can be received on four levels: per round, per user (instance of use), per service (maintenance interval), and per lifetime. The reports include data on the time used for different phases in the drilling process and rock tools' consumption per set intervals (Fig. 18.35).

- iSURE® Vibration Feedback (Fig. 18.36)

iSURE® introduces a practical approach to vibration control, as it knows the charges, detonation timing (momentary situation), and planned and drilled drill plan (burden). Blasting vibration data from third-party systems can be imported into

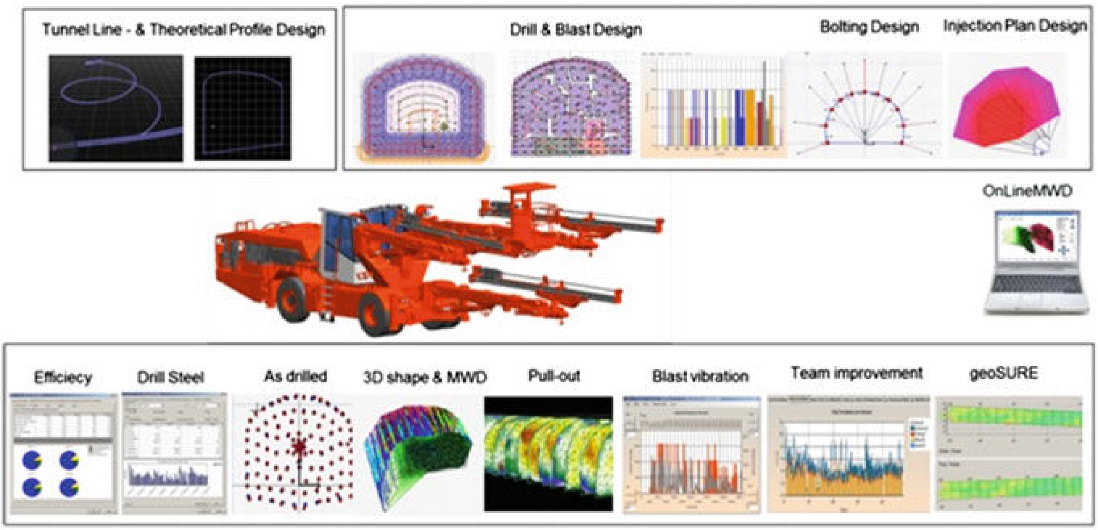


Fig. 18.35 With iSURE (Office program), the capabilities of the accurate iSeries drilling rig are fully capitalized. iSURE produces all the necessary data for drilling and blasting as well as analyzes the drill rig data. Pullout analysis and blast vibration feedback offer a new way of improving the D&B, and team improvement shows the trend of work cycles along the advance of the tunnel. geoSURE adds a rock quality reporting system that utilizes real-time data analyses onboard (reproduced with the permission from Sandvik)

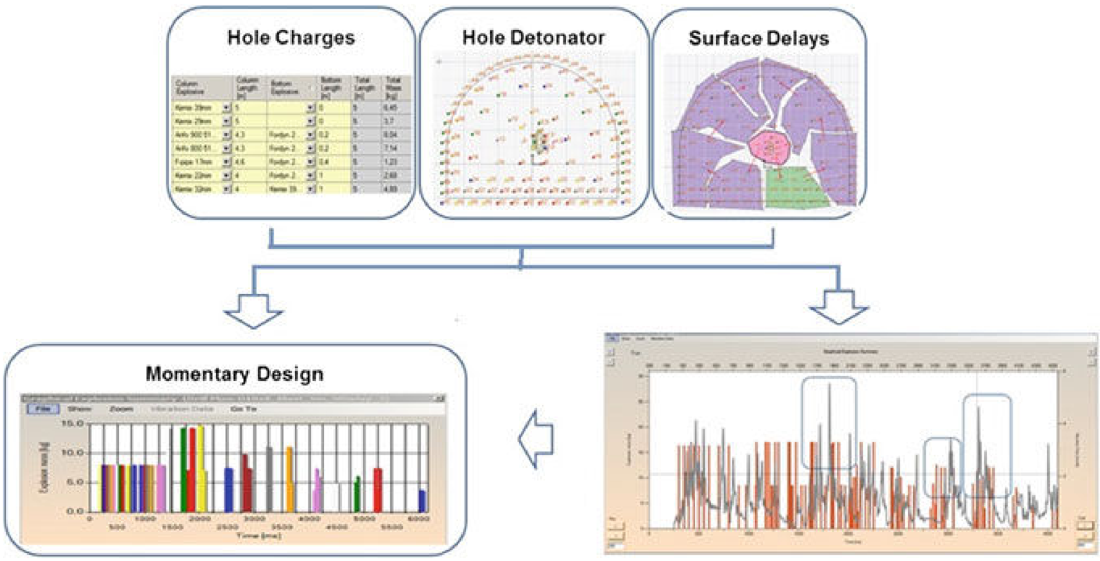
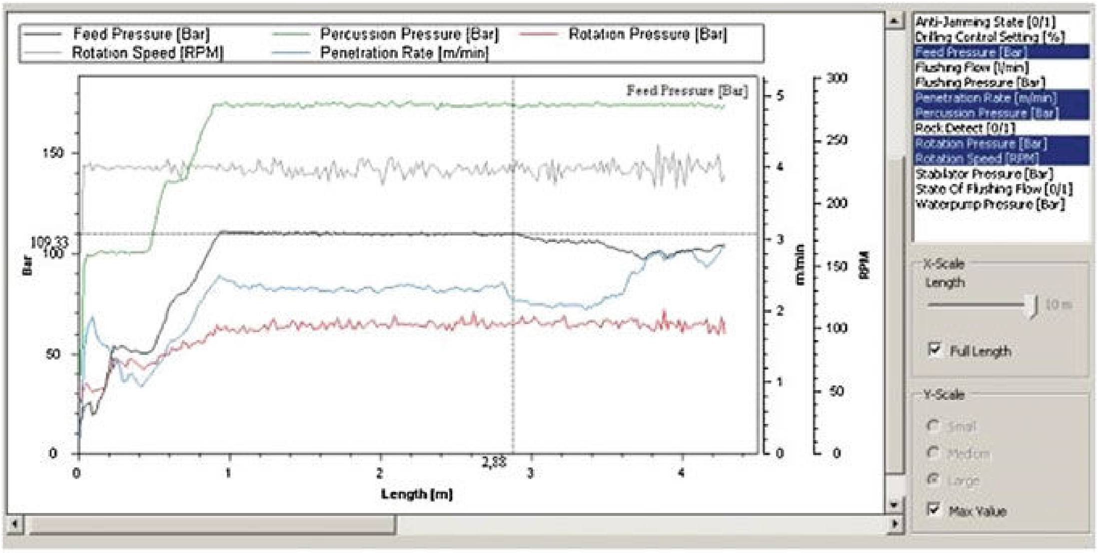
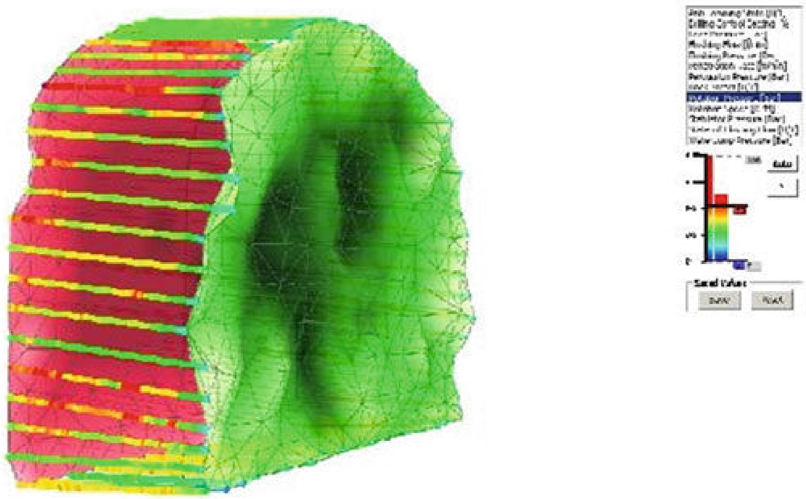


Fig. 18.36 Deviation in kg versus PPV can be pinpointed into the drill and blast plan. This connection can be used in charging work quality control, adjustment of the blast/detonation and tuning the round length while approaching a sensitive area (reproduced with the permission from Sandvik)

iSURE in order to compare blasting-generated vibration against momentary mass of explosives. This comparison of information is then used to reveal the problem areas in the drill and blast plan.



(A) Graph of parts of measured drilling parameters



(B) MWD can be visualized as colour freely rotated 3D-images of rock conditions during drilling

Fig. 18.37 iSURE use MWD to collect drilling data and analyze rock characters (reproduced with the permission from SANDVIK)

- iSURE® Analysis: MWD Data Collection and Reporting

iSURE® Analysis offers Measuring-While-Drilling (MWD) data collection and reporting for analyzing rock structure and characteristics. The module collects data on 19 parameters—more than any other software in the market—among those being parameters such as airflow, feed pressure setting, rotation speed setting, anti-jamming state, and drilling control setting. The MWD data can be studied and analyzed after drilling (Fig. 18.37).

As an option, an online MWD is available. It offers MWD data and visualization of the rock surface in real time in 3D. The data can be exported to external programs from iSURE® to do further analysis in other analysis environment.

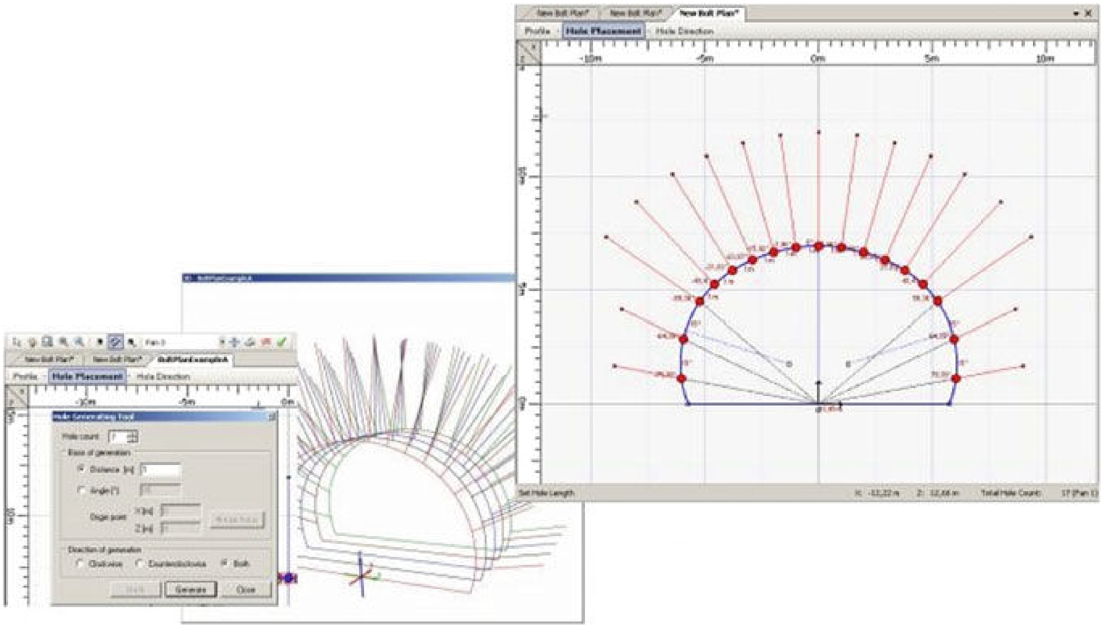


Fig. 18.38 iSURE IV Bolting module is used to design and report the bolting works (reproduced with the permission from Sandvik)

- iSURE® Bolting (DTi-series only)—Bolt Plan Design, Data Collection, and Reporting

iSURE® Bolting makes the use of one machine possible in both drilling the face holes and bolt holes simultaneously. The program enables utilization of the drill and bolt plans at the same time, with one navigation when appropriate. Reporting is possible on both the drill plan and the project coordinate systems, promoting systematic bolting and reporting of actual bolt-hole locations. Additionally, the iSURE® Analysis offers a possibility to illustrate the tunnel profile based on the actual bolt locations.

The bolting module allows the design of up to 5 bolting fans in the same plan. The plan includes hole placement and direction, hole generating and fan management tools, and 3D visualization (Fig. 18.38).

- geoSURE

Based on the MWD, Sandvik Construction recently launched geoSURE. The new system provides accurate geological information for companies or individuals involved in tunneling or underground operations, through a completely new, rig-integrated onboard rock mass analysis and visualization system. Designed to be used with Sandvik underground construction drill rigs, it delivers real-time onboard analysis of the rock mass which includes features such as fracture, rock strength, and water indication. The extended analysis of the data allows the evaluation of the rock strength class, rock quality class, and rock quality index. These features can then be further visualized using the iSURE® tunneling project management software. The 2D planar view provides an overall outlook of the tunnel section,

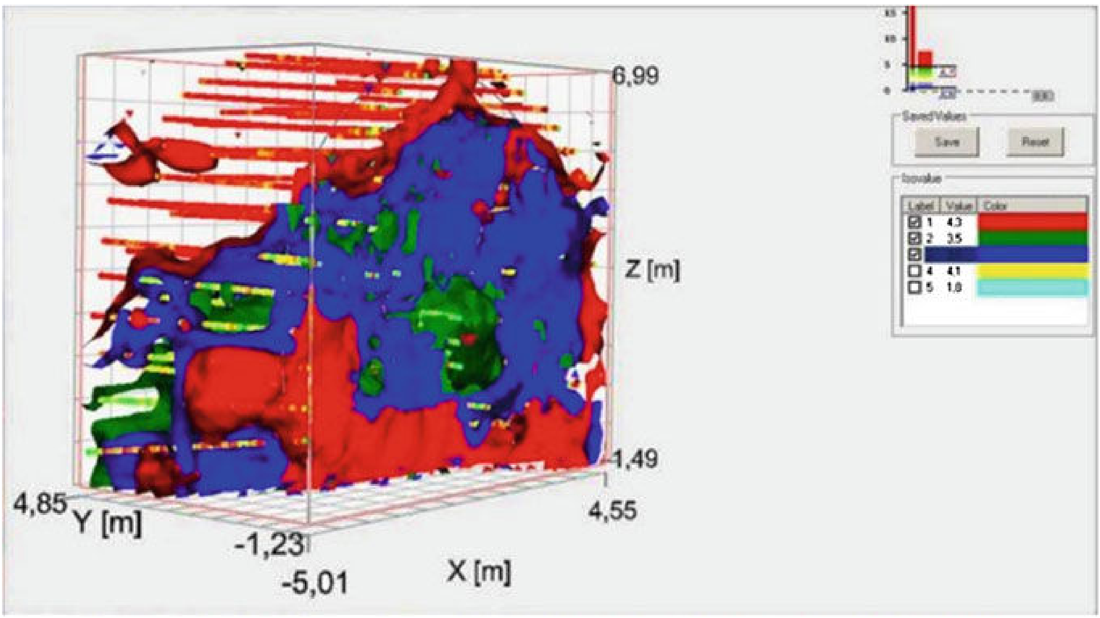


Fig. 18.39 geoSURE—rock mass analysis and visualization system (reproduced with the permission from Sandvik)

including 2D interpolation, side view, top view, and unrolled view. For more detailed inspection, the 3D structural view of the tunnel section can be used. This features 3D interpolations, plane intersections, and iso-surfaces, and curves. The iSURE® system also provides the former 3D view and one hole MWD diagram with the new geoSURE variables.

Not only is it an easy way to fulfill the most advanced reporting requirements in the industry, but it also acts as an important tool for the assessment of rock reinforcement or injection requirements. Additionally, it serves as an assisting tool for charging and blasting control as well as a complementary tool for geological mapping (Fig. 18.39).

18.4.2 Atlas Copco: Underground Manager MWD (Reproduced from Refs. [12, 13] with the permission from Atlas Copco)

Underground Manager (Tunnel Manager) is a support software for planning, administration, and evaluation of the drilling operation in mining and tunneling projects.

The Windows-based program Underground Manager (UM) is suited for Atlas Copco Boomer rigs equipped with Rig Control System (RCS) and the Advanced Boom Control function (ABC Regular or ABC Total). The upgraded Tunnel Manager software is available in three different packages: Tunnel Manager, Tunnel Manager Pro, and Tunnel Manager MWD.

The most advanced package, Underground Manager MWD, offers a completely new functionality built on several years of research at Luleå University of Technology in Sweden.

The first package, Underground Manager, offers basic functions, such as generating drill plans and following up the result. The second package, Underground Manager Pro, contains the upgraded Measure-While-Drilling function, whereas the most advanced package, Underground Manager MWD, also contains the possibility to analyze the collected data. This means that the user can swiftly translate rock drilling data into relevant rock mass characteristics such as rock hardness and fracturing.

UM is based on an SQL Server Compact Edition database, where all tunnel data (tunnel lines, laser lines, drill plans, contours, section logs, MWD logs, etc.) are kept in a defined structure.

The procedure starts with a 3D image of the tunnel that is imported into the Underground Manager tool. This includes tunnel lines, contours, fix points, and laser points.

- Drill Plan Generator and Charging and Firing Pattern

The drill plan designing is a key function of UM where all lines, shapes, and holes (length, lookout, type, and diameter) are allocated. Each defined section of a tunnel can also be given an individual contour design for generating drill plans as shown in Fig. 18.43. Any given number of holes can be allocated that may vary between the defined or interpolated contours. Once the drill plan is complete, the UM can be linked up with Atlas Copco's Rig Remote Access System (Fig. 18.40).

Following proper analysis of the geometry and rock conditions, a lot of preparation goes into selecting the right explosives and firing sequences in order to achieve good contours and also make sure that vibrations do not exceed stipulated limits and regulations for the project. All these can be designed and simulated using the UM, which offers dynamic drawing tools for both charging and blasting. By selecting designated sections and using a drawing tool, the UM enables various blasting scenarios to be tried and tested. Sections of blastholes can be given individual delay times according to a chosen sequence, including block and surface delays, as shown in Fig. 18.41. This is very useful in ensuring that the right ignition sequence is employed. Any changes to the drill plan will be synchronized and shown in the firing and charging pattern.

- MWD and reporting

The technique is to extract rock mass properties while drilling is called MWD, which stands for "Measurement While Drilling."

The parameters recorded are as follows: penetration rate, feed force, percussive pressure, rotation pressure, rotation speed, damp pressure, water pressure, and water flow.

The MWD technology consists of two separate processes: registration of data and presentation/evaluation/interpretation of data.

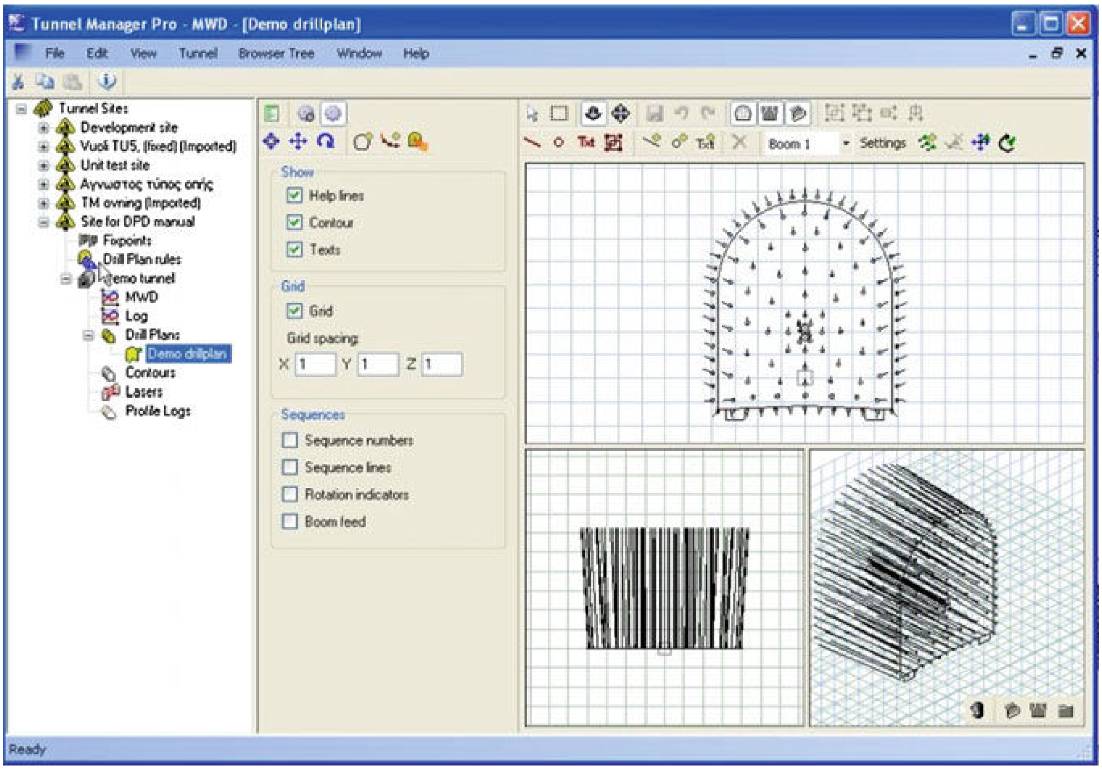


Fig. 18.40 Edit a drill plan using tunnel manager MWD (reproduced with the permission from Atlas Copco)

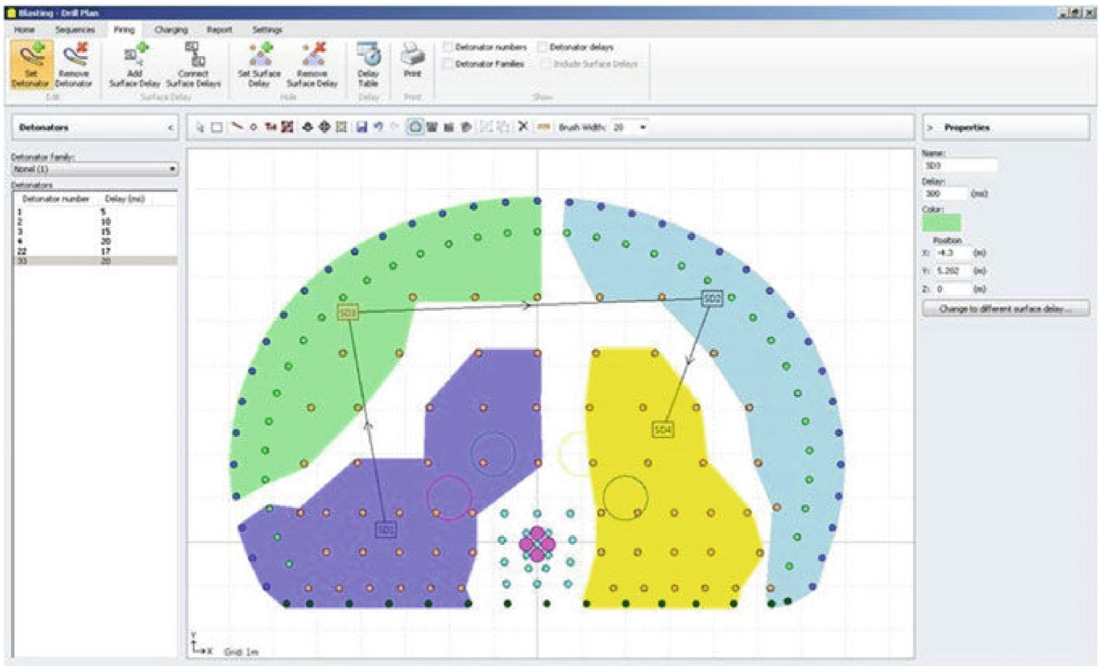


Fig. 18.41 Firing sequence design (reproduced with the permission from Atlas Copco)

The registration part has developed as the drill rigs get more and more computer capacity, but is still a critical process.

The evaluation/interpretation took a great step forward when Håkan Schunnesson published his doctor's thesis "Drill process monitoring in percussive drilling for location of structural features" at Luleå University of Technology in 1997.

MWD technology is one tool that can generate considerably better information on the real characteristics of the rock mass. MWD is often employed in projects developed in sensitive host rock due to either nearby structures or installations, such as in urban infrastructure projects, or because of poor ground conditions. In these cases, rock conditions can be visualized through MWD technology and be displayed as maps over the tunnel perimeter or some specific holes.

Optional reporting features in MWD also include geological indices based on the drilling process and estimates of hardness and fracturing. Rock hardness and rock fracturing indices are calculated by the program. This evaluation model incorporates data such as penetration rate, drill speed, feed pressure, and other parameters that, when combined, provide an index for rock variations (see Fig. 18.42). This index is often matched with real observation.

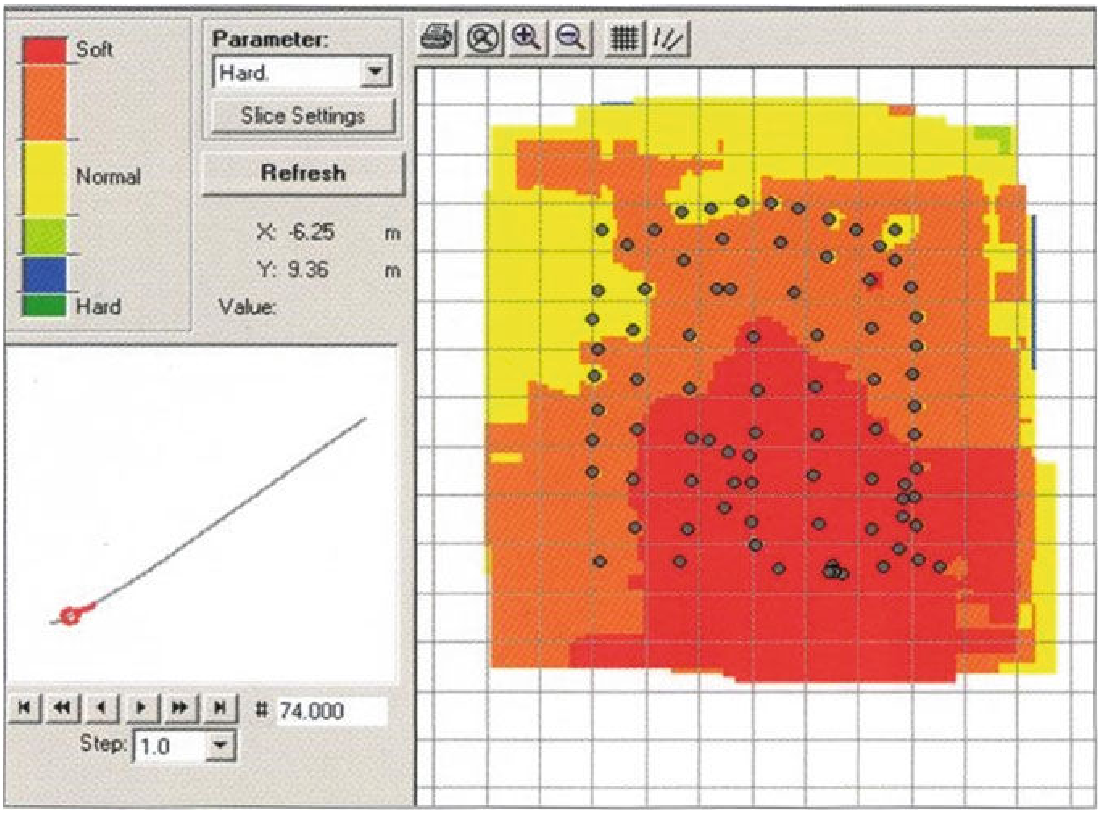


Fig. 18.42 Using MWD output the index of rock hardness distribution on the tunnel face (reproduced from Ref. [13] with the permission from Atlas Copco)

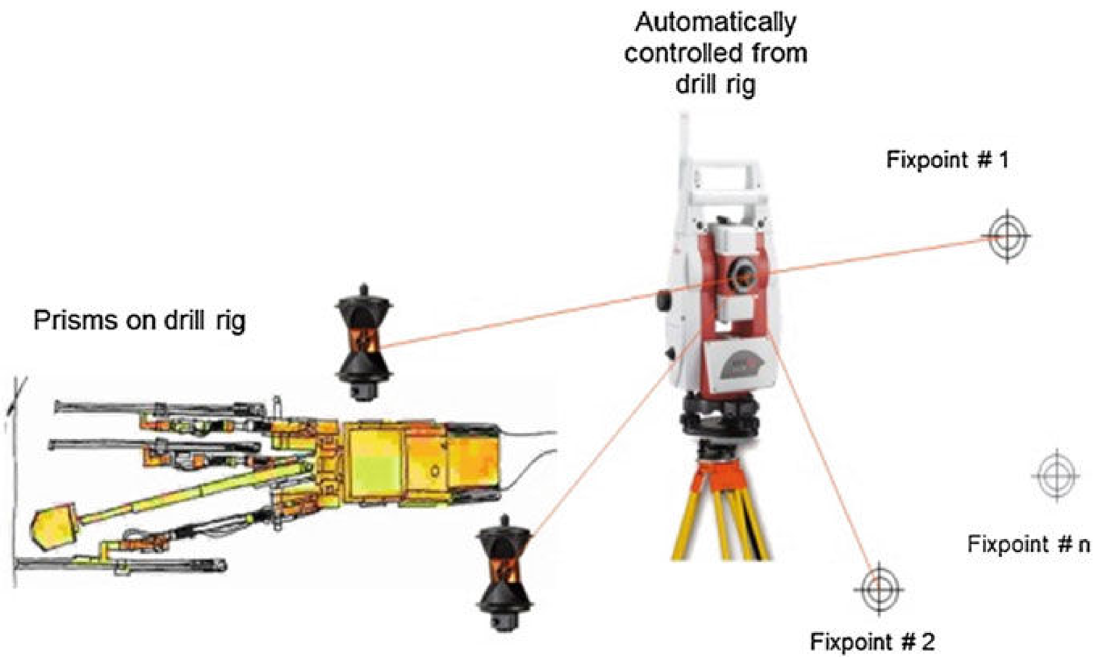


Fig. 18.43 Total navigation system with a tripod-mounted (reproduced with the permission from Atlas Copco)

- Total navigation system

That is an input method of surveying with a total station which starts with a station setup where the instrument is placed on a tripod and either centered over a known point, or set up as a free station.

As shown in Fig. 18.43, the instrument is centered over an existing point with known coordinates and the orientation of the instrument is obtained by reference direction measured against another known point. The second case is free station which means that the instrument is set up in any location, preferably next to the item to be measured in order to obtain a higher accuracy. Determination of the total station's locations is carried out by measuring against other known objects. Known point is the most common method of station establishment, but the method of free station is preferred when measuring is made within a small geographic area.

In recent years, the total station has evolved to become increasingly automated, thereby simplifying the surveying work. An example of this is the so-called automatic prism lock, or automatic target recognition (ATR), which enables the instrument to be directed toward the prism automatically using a CCD camera (Jodahl and Larsson 2007). The technology requires only a rough orientation of the instrument, while the fine tuning is handled automatically. ATR technology along with the development of the automatic prism called LOCK (or automatic target tracking) has also enabled more effective surveying.

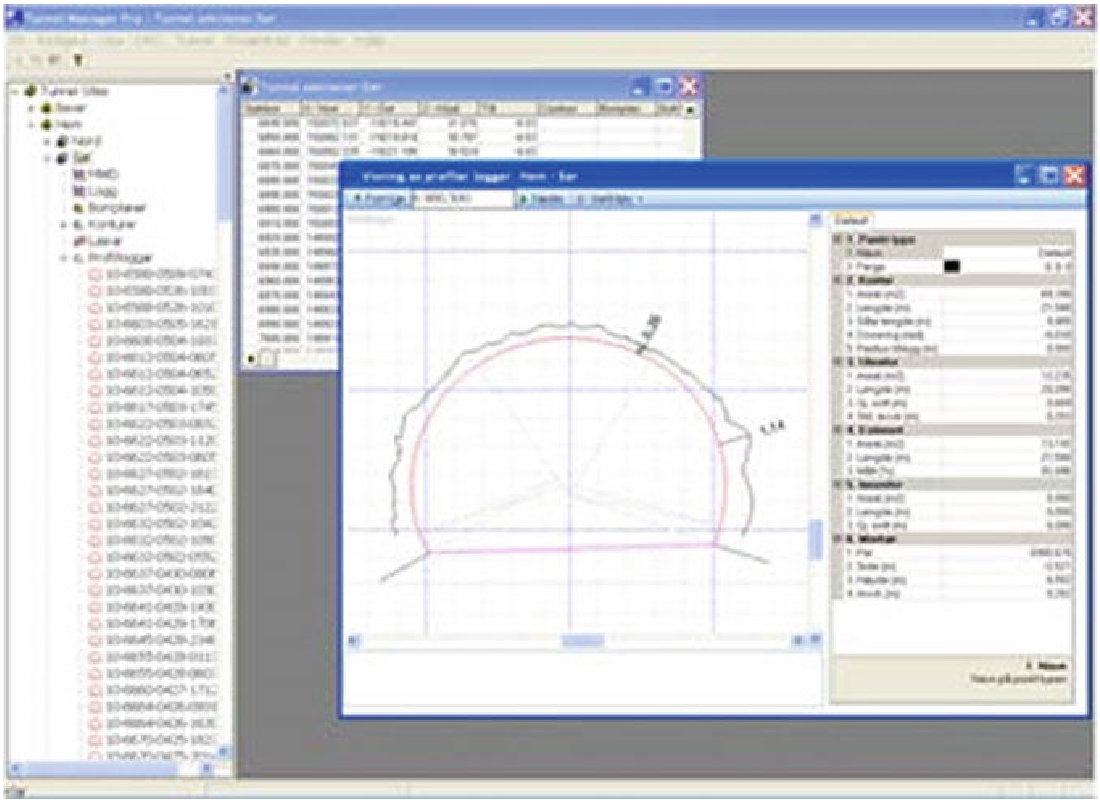


Fig. 18.44 Using tunnel profiler to measure overbreak (reproduced with the permission from Atlas Copco)

By attaching the total station control panel directly on the prism pole and through radio connection to communicate with the instrument, the meter could move from the instrument directly to the object to be measured. This has completely changed the method of data entry. Today, in many cases, only one person is required to carry out field work. Another technique that is still evolving toward a better accuracy is reflectorless measurement, where neither prism nor reflectors are used. Instead, the distance is measured directly at the object via the reflection of the laser beam. The technology allows inaccessible objects to be measured, however, with a slightly reduced accuracy than when the prism is used.

- Tunnel profiler (Fig. 18.44)

Tunnel profiler is a fully integrated system for measuring the excavation profile and is reliant on network communications. The accuracy of the scanned surface is 3–5 cm, by saving the overbreak scanning of the profile and making adjustments on the drilling pattern may save some 5 cm on overbreak. By limiting the overbreak, it can reduce the cost of concrete for secondary lining, reduce the mucking cost of extra rock material, and shorten the construction time.

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