Chapter 26

Design of a surface blast

26.1 INTRODUCTION

Earlier chapters dealt with different aspects of explosives and rock fragmentation through a blast.

Blasthole drilling is a relatively straightforward subject without much danger, but blasting is rather intricate and highly dangerous.

To avoid any fatalities and tragic circumstances, very careful planning is necessary in choosing various parameters related to blastholes, selecting explosives and the other accessories used for engineering the blast.

This chapter is devoted to giving the basic knowledge for the design of a very efficient blast in a surface mine without causing any recognizable hazards to the surroundings.

26.2 BLASTHOLES IN A MINE BENCH

A blasthole drill stands on a mine bench top and drills blastholes in it. Blastholes drilled in the rock mass appear as shown in Figure 26.1.

In the early days of blasthole drilling, when churn drills were used for the purpose, the blastholes could only be drilled in a vertical direction. This trend continued during the dawn of rotary blasthole drilling as well. Now, inclined blastholes as shown in the right hand drawing in Figure 26.1, are also drilled in many mines. As will be seen, inclined blastholes often prove more appropriate from the viewpoint of reduction in cost. Ideally the angle of inclination of blastholes is 45°, but practical limitations of rotary drills limit the angle of inclination to 30°.

On certain occasions, in addition to the main blastholes, some additional vertical or inclined blastholes, like those shown in the left hand drawing in Figure 26.2, are also required for the appropriate fragmentation of rock mass. These blastholes are of half or even quarter depth depending upon the characteristics of the rock mass. Such blastholes are referred to as satellite blastholes.

Similarly, situations also arise when additional horizontal blastholes like those shown in the right hand drawing in Figure 26.2 become necessary. They are termed snake holes.

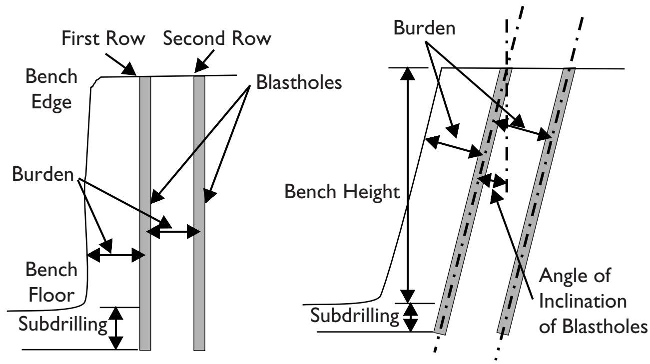


Figure 26.1 Vertical and inclined blastholes in a bench.

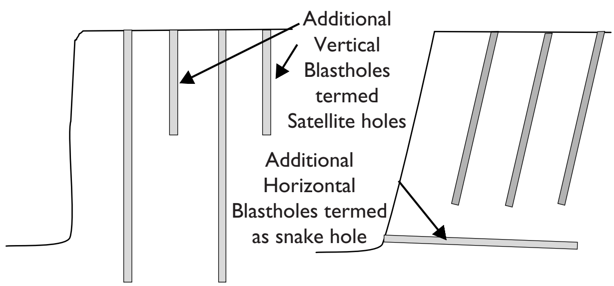


Figure 26.2 Additional vertical blastholes near bench top or horizontal blastholes at the bench floor.

26.3 TYPES OF BLASTING IN SURFACE MINES

No doubt, blasting activities in large mines are carried out for breaking the rock mass, but the associated objectives are different for each one of them. This gives rise to different types of bench blasting techniques, described briefly in the following subsections.

26.3.1 Conventional bench blasting

Conventional bench blasting is also called production blasting. In this type of blasting the purpose of the blast is to break a large portion of rock mass to yield such fragment sizes that can be easily loaded by using rope shovels or hydraulic shovels into the dumpers. These dumpers move the overburden to a waste site and the ore to a crusher site or store yard.

In surface coal mines when the coal layer is very thick, even the coal is blasted and loaded into dumpers by using front end loaders.

Loading by this equipment is shown in Figure 26.3.

Figure 26.3 Loading of a dumper by using rope shovel, hydraulic shovel and front end loader.

26.3.2 Secondary blasting

The loading and moving equipment used in mines has limitations. Pieces of rocks beyond a certain size can neither be lifted nor carried. Even the crusher cannot accept rock pieces beyond a certain size for further crushing.

Economical reasons always force the blast designer to design the blast in such a way that the largest fragments yielded through the blast are of a size acceptable to the lifting or moving equipment.

Due to so many uncontrollable factors in blasting, in some cases large rock pieces are left even by a well designed blast. Such large pieces can be moved only after their further blasting. Such blasting of large rock pieces is called secondary blasting.

In early days these large boulders were blasted after drilling one or more blast-holes and charging them. Nowadays hydraulic excavators equipped with a hydraulic hammer attachment are more often used for this purpose.

26.3.3 Cast blasting

In many places coal is found in horizontal layers that underlie a large bed of overburden. In such cases the most economical method to reach the coal is by blasting the top overburden layer in the form of a long strip and moving it to one side by using a giant walking dragline as shown in Figure 26.4. In such mines the technique of cast blasting can be adopted with advantage.

The advantage of cast blasting can be appreciated by means of the sketch in Figure 26.5. If the bench is blasted in the normal way, the profile of the top of the muck heap reaches points ABCD. Here the blasted overburden in the area ABF has already moved to the area of heap of shifted overburden without use of the dragline. Similarly, if the bench is blasted by using the technique of cast blasting the top of the muck heap reaches points LMNP. In this case all the blasted overburden in the area LMF has already moved to the area of the heap of shifted overburden without use of the dragline.

Since area LMF is much larger than area ABF cast blasting proves greatly advantageous by reducing the workload of the dragline. (See also article 26.7.3 below).

26.3.4 Presplitting

In simple words presplitting means creating a fully cracked surface between the area to be blasted and the area to be kept intact. Such a surface is created by

Figure 26.4 Overburden removal by a walking dragline.

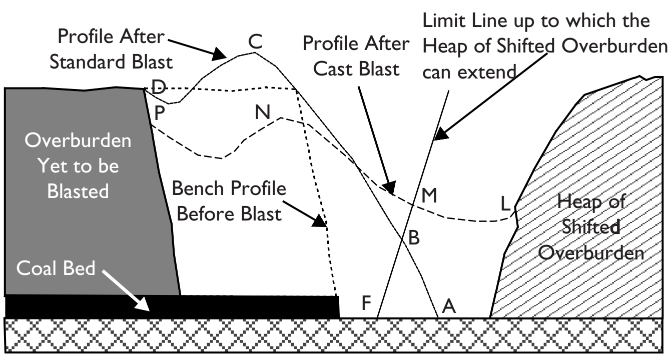


Figure 26.5 Advantage of cast blasting.

charging closely spaced, small diameter blastholes drilled in a line and blasting them instantaneously.

Diameters of blastholes for the presplitting operation is of the order of 100 to 125 mm. The most common method of drilling such blastholes is by rotary percussion drilling through hydraulic drifters because it gives the fastest penetration rates.

Usually the presplit surface is at an angle of 15° with the vertical as shown in Figure 26.6. The small diameter of the blastholes and the close spacing ensure that a well-trimmed smooth surface is formed without any damage on the either side of the blasthole plane. The presplit surface does not allow significant transmission of shock waves from the main blast into the rock mass meant to be kept intact. To meet this objective, often a row of buffer blastholes is also drilled in front of the presplit blasthole line. Blastholes in this row are charged to evolve limited energy.

Ideally a single fracture connects adjacent blastholes, and half of the hole remains at each presplit hole. If excessive crushing and radial cracking is observed at the plane it indicates excessive use of explosive.

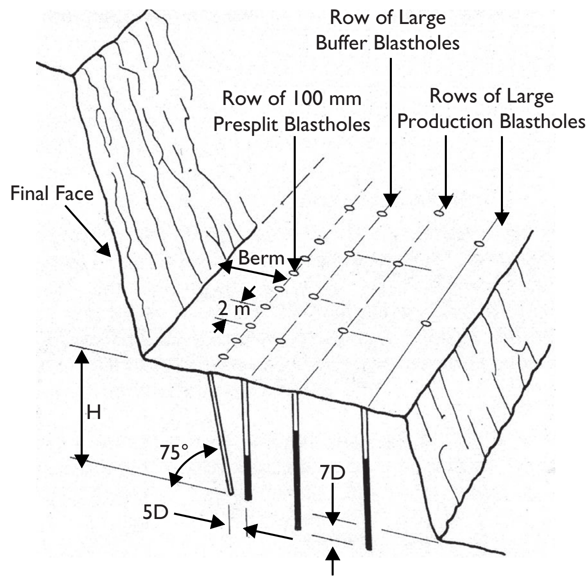


Figure 26.6 Presplitting behind the main bench.

Closer spacing with an appropriate quantity of explosive gives a better surface. However, because of the high cost of drilling, the optimum spacing is the largest at which radial cracks will join and form a continuous undamaged surface.

Smooth blasting is also a type of presplitting where the blastholes are of much larger diameter, often drilled with the same drill that is used for drilling production blastholes in the main bench.

Rotary blasthole drills like the Bucyrus 39R are very suitable for drilling blastholes for smooth blasting because they can drill blastholes at –15° angle, and therefore can carry out drilling operations by standing on the main bench. This enables leaving much smaller berm as illustrated in Figure 26.7.

26.3.5 Snake hole blasting

Snake holes are the horizontal blastholes drilled at the bottom of the bench as shown in Figure 26.8. When these blastholes are drilled and blasted prior to blasting the main vertical or inclined blastholes, they create a plane of separation in a manner very similar to the presplitting blastholes. The advantage of snake hole blasting techniques are:

1 Less fracturing of rock mass below the newly created bench floor. Eventually when this floor becomes the top of bench, the hazard of flyrock is at a reduced level.
2 Improved floor of the bench where the movement of mining equipment is made smoother.
3 Need for less concentration of explosive at the bottom of the main vertical or inclined blastholes in the bench.

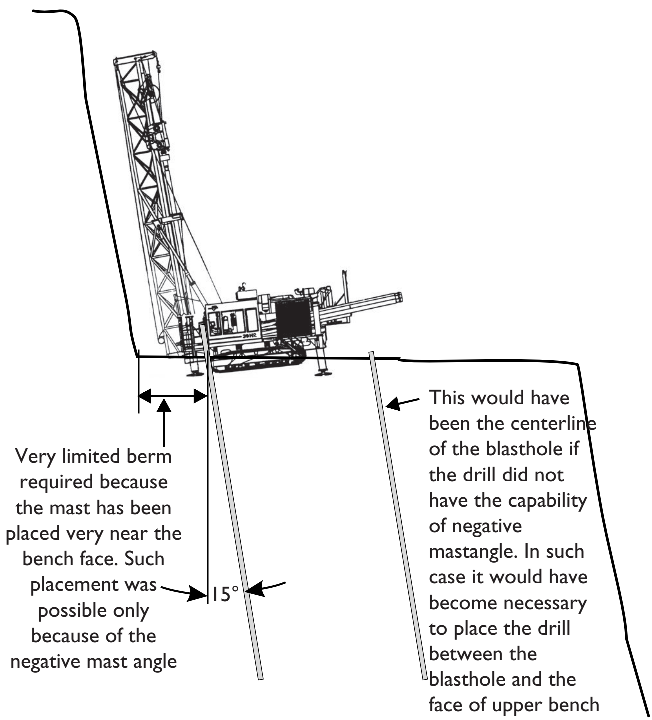


Figure 26.7 Usefulness of –15° drilling angle of Bucyrus 39R.

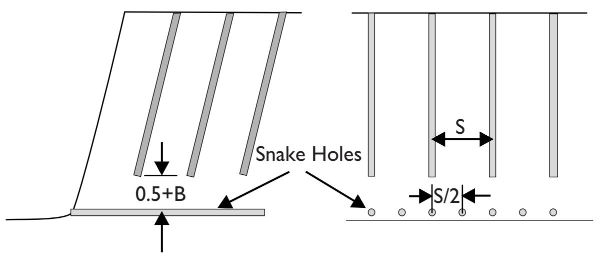


Figure 26.8 Snake hole blasting.

The use of snake hole blasting is often avoided because it needs an additional drill specially equipped to drill horizontal blastholes. Such a drill is usually good for drilling small diameter blastholes and cannot be replaced by the large rotary drills used for production blastholes.

In some European countries snake hole blasting is common because in those countries even in very large mines production blastholes are of small diameter and the drills used for production hole blasting can also be used for snake hole blasting.

In a few large US mines snake hole blasting has become unavoidable because geological conditions do not allow a smooth bench floor by the normally adopted technique of optimum subdrilling and subsequent blasting with heavy concentration of high explosives at the bottom of the blastholes.

26.3.6 Rip rap blasting

When huge rock-fill dams are to be built on rivers, or long breakwaters are to be constructed for harbors, an enormous quantity of large pieces of rock are required. Depending upon design parameters their weight may reach even 3000 kg. A special blasting technique that yields such large rock pieces is called rip rap blasting (see article 26.7.1 below).

26.4 WHAT IS INVOLVED IN DESIGN OF A BLAST

The outcome of a blast is gaged through the following parameters.

1 Fragmentation of the rock mass
2 Displacement of the muck pile
3 Profile of the muck pile
4 Misfires
5 Ground vibrations
6 Flyrock
7 Airblast

In designing a blast a designer has to give due consideration to all the controllable and uncontrollable variables that are likely to have an effect on the outcome parameters, and choose apt magnitudes of controllable variables in such a way that the magnitudes of the outcome parameters is within the desired range.

The uncontrollable variables are properties of rock specimen, bedding, dip, strike, faults, joints, discontinuities, ground water and weather.

Controllable variables are listed in Table 26.1.

26.4.1 Powder factor

The first step in designing a blast is to chose the appropriate explosive and ascertain the powder factor.

In surface mining practice the most commonly used explosive is ANFO. As reasons for this choice have been explained earlier, they are not repeated here.

Table 26.1 Controllable variables involved in blast design.

Parameter names	*Parameter names*
Explosive and Powder Factor	Blasting Direction
Number of Free Faces	Size of the Blast
Blasthole Diameter	Blasthole Depth
Blasthole Inclination	Burden
Blasthole Spacing	Subdrilling
Stemming Height	Blasthole Drilling Pattern
Blasthole Charging Pattern	Firing Sequence

As ANFO is mixed on site and poured into the blastholes, it is possible to change the properties of the ANFO that goes into the blasthole at a certain depth. This is done by adding a sensitizer like Al or adding some more powerful explosive to the mixture. Actually by doing so we are changing the basic powder factor of ANFO.

The basic powder factor can be chosen by using some formulae presented earlier, but Table 26.2 makes the task easier by presenting appropriate powder factors for ANFO under different conditions of surface mining.

In the days of manual calculations, the blasts were designed without considering any variation of powder factor but with modern computerized blast design programs, a blaster gets guidance on changing the explosive density in the same blasthole depending upon the variation in properties of the intact rock and rock mass.

26.4.2 Blasting direction

As has been stated in the previous chapter, when the rock mass is in form of beds and the beds are dipping, the direction of blastholes must be such that shooting is achieved either with the dip of the beds or against the dip of the bed.

The worst situation is when shots are along the strike. To remedy the situation the solution is in changing the direction of blast.

26.4.3 Number of free faces

A greater number of free faces available for a blast means more energy in reflected tension waves. In such situations a larger number of blastholes can be fired together. However, in practice no more than two free faces are possible for the same blast.

When mucking operations are carried out by shovel and dumper or loader and dumper combination, blasting is carried out with only one free surface.

When the mucking operation is to be carried out by dragline usually two free faces, as shown in Figure 26.4, are practiced. The dragline carries out scraping excavation on one surface and then by slewing into about 90° it empties the bucket in front of the other surface. This second surface, which is right angles to the first surface, becomes

Table 26.2 Powder factors of ANFO in surface mining.

		Powder factor	
Type of mine	*Method of excavation*	*in lb/yd^3*	*in kg/m^3*
Surface Metal Mining		0.6–1.0	0.356–0.593
Surface Coal Mining with			
	60 yd^3 (45.87 m^3) Dragline	0.5–0.7	0.297–0.415
	30 yd^3 (22.94 m^3) Shovel	0.6–1.1	0.356–0.653
	17 yd^3 (13 m^3) Front End Loader	0.6–1.6	0.356–0.950
Surface Coal Mining with Cast Blasting		0.9–1.5	0.534–0.890
Quarrying		0.6–1.5	0.356–0.890
Construction Blasting			
	Open Excavation	0.25–0.8	0.149–0.475
	Trenching	2.0–3.0	1.187–1.780

necessary to ensure that cast blasting can be carried out on the side of this surface. A careful look at the Figure 26.4 will give a lucid idea of the advantage of two free surfaces in such situation.

26.4.4 Blasthole diameter

Several factors are required to be considered while choosing the appropriate diameter for blastholes.

The desired diameter of the blastholes in large surface mining operations is decided mainly by considering the following factors.

1 Desired rate of production
2 Desired fragmentation
3 Properties of rock specimen
4 Properties of rock mass

The following throws more light on other factors.

26.4.4.1 Desired rate of production

Since mining is a commercial proposition, the rate of production is fixed on an economical basis even before starting the mining project. It has the highest influence on the desired diameter of the blasthole.

If consideration is given to the unconfined compressive strength of rocks, the approximate relation between production rates and blasthole diameter is as shown in Table 26.3.

In surface mining practice the rate of drilling and rate of removal of the blasted rock must match. Thus, the diameter of blastholes is loosely related to the capacity of the shovel bucket as matched in Table 26.4.

When a dragline is used for overburden removal, the diameters of blastholes can be even larger because the bucket capacity of the dragline chosen for the operation is much larger. Therefore, even if very large pieces of rocks are formed in blasting they can be moved by the dragline.

The usual range of diameters of blastholes while using a dragline is 250 mm to 381 mm.

Table 26.3 Average production rate in formations of different hardness from blastholes of different diameters.

	Likely production rate in m^3/h for one m blasthole length in rocks with compressive strength as under		
Blasthole diameter	*Soft rock UCS < 70 MPa*	*Medium rock UCS 70 to 180 MPa*	*Hard rock UCS > 180 MPa*
200	600	150	50
250	1200	300	125
311	2050	625	270

Table 26.4 Blasthole diameters based on shovel bucket capacities.

Bucket capacity of the shovel (m^3)	*Hole diameter range in mm*
4.5	76–127
7.5	127–215
9.17	171–250
11.5	200–270
15.3	229–311
20	250–349
35	270–381
50	311–445

Blasthole diameter is also influenced by the bench height. This will be elaborated later in the appropriate section.

26.4.4.2 Desired fragmentation

Closely spaced small diameter blastholes certainly give smaller fragment size but large diameter blastholes do not give proportionally larger fragment size.

What the distribution of particle size will be after a blast can be predicted with reasonable accuracy by many mathematical models developed for the purpose. One such model is the Kuz-Ram model. Table 26.5 presents the prediction of fragment size distribution of blasts for 203 and 311 mm diameter blastholes, where all other factors remain the same except those mentioned in the table. The data in the table prove both the points mentioned in the previous paragraph.

Actually such mathematical models – even though they indicate a relationship between fragment size distribution and blasthole diameter – are not useful for determining blasthole diameters, but can be used for verification that the fragment size will be good enough for the loading and hauling equipment when a particular diameter is selected.

26.4.4.3 Properties of rock specimen

In the absence of a more rigorous approach as explained in a previous chapter, the powder factors to be chosen for some of the commonly occurring rocks are given in Table 26.6.

If blastholes are charged with more than the optimum amount of explosive, the rock mass gets fragmented to small size. With this, the need for crushing decreases but the overall cost increases.

26.4.4.4 Presence of geological structures

Properties of the rock mass also need to be taken into account while fixing the diameter of blastholes. The number and placement of joints in the rock mass may

Table 26.5 Fragment size distribution as per Kuz-Ram model.

Fragment size (m)	Percent passing for blasthole with following details	
	Hole dia. = 203 mm Burden = 5.075 m Spacing = 6.343 m	Hole dia. = 311 mm Burden = 7.775 m Spacing = 9.718 m
0.00	0.0%	0.0%
0.05	5.2%	4.0%
0.10	17.6%	13.7%
0.15	33.6%	26.9%
0.20	50.2%	41.4%
0.25	65.1%	55.5%
0.30	77.1%	67.9%
0.35	85.9%	78.0%
0.40	91.9%	85.7%
0.45	95.6%	91.1%
0.50	97.7%	94.7%
0.55	98.9%	97.0%
0.60	99.5%	98.4%
0.65	99.8%	99.2%
0.70	99.9%	99.6%
0.75	100.0%	99.8%
0.80	100.0%	99.9%
0.85	100.0%	100.0%
0.90	100.0%	100.0%
0.95	100.0%	100.0%
1.00	100.0%	100.0%
1.05	100.0%	100.0%
1.10	100.0%	100.0%

Table 26.6 Powder factors for different rocks.

Rock category	*UCS range MPa*	*Rock name*	*Powder factor in kg/m³*
Very Soft	about 50	Coal, Potash, Soft Shale, Marl	0.15–0.25
Soft	50–100	Sandstone, Hard Shale, Limestone, Slate, Conglomerate	0.25–0.40
Medium	100–200	Schist, Hornfels, Hard Limestone, Marble, Serpentinite, Dolomite	0.4–0.60
Hard	200–300	Andesite, Dolerite, Hard Iron Ore, Basalt, Granite,	0.60–0.70
Very Hard and Hard plus Tough	More than 300	Taconite, Skern, Quartzite, Hard Basalt	0.70–1.00

necessitate reduction of blasthole diameter, fixed on the basis of desired production rate.

A rock mass in a mining bench is rarely monolithic. It is usually inter-spread with joints. A monolithic block of the rock exists only within the boundaries of such joints.

The process of formation of cracks and subsequent fragmentation of rock mass starts from the blasthole and continues undisturbed till it encounters some discontinuity in the form of joint, fault, fold, unconformity etc. Beyond such a discontinuity the formation of cracks and subsequent fragmentation is reduced considerably.

Suppose the structure and spacing of joints in a bench is like the one shown in Figure 26.9 A, and if large diameter blastholes with larger spacing as shown in the figure are drilled and blasted, then two of the blocks do not have any blasthole within them. This will result in poor fragmentation within those two blocks. However, if smaller blastholes with smaller spacing, as shown in Figure 26.9B, are chosen then each of the blocks has at least one blasthole within it and will thus result in good fragmentation in all the blocks.

Reduction of diameter of blastholes beyond a certain point can also be counter-productive. This happens because as the diameter of the blastholes is reduced, the number of blastholes required to be drilled increases disproportionately. This increases the cost of drilling. Further, the cost of explosives, cost of accessories, and cost of charging the blastholes increase for a larger number of blastholes. In very small diameter blastholes it is not possible to use inexpensive explosives like ANFO.

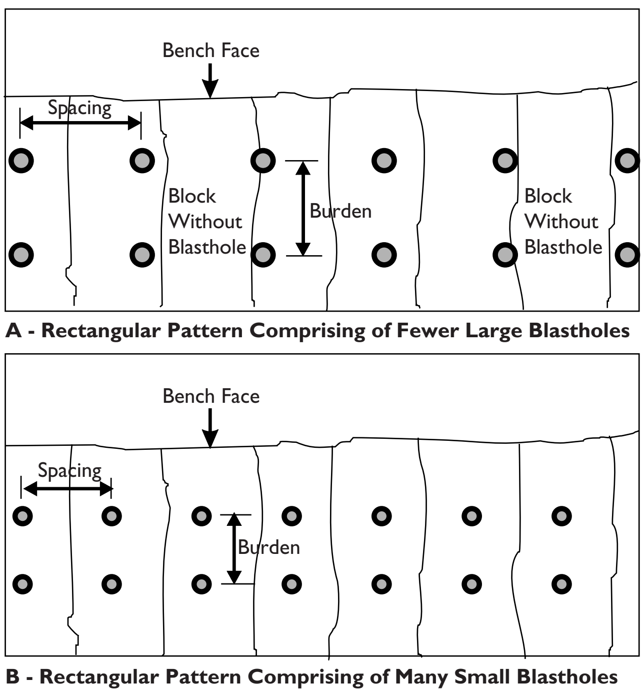


Figure 26.9 Dependency of blasthole diameter on joint structure.

26.4.5 Blasthole depth

Really, this should be called length of blasthole. However, in surface mines, as most of the blastholes are in a vertical direction, the term is colloquially called depth.

The length of the blasthole includes subdrilling. It depends upon height of bench, subdrilling and angle of inclination of the blasthole.

For the purpose of material removal after blasting, one of the three alternatives are used.

1 Combination of Shovel and Dumper
2 Walking Dragline.
3 Combination of Wheel Loader and Dumper

The manner in which these combinations work differs, the most appropriate method of drilling and blasting also differs for each case.

26.4.5.1 Combination of shovel and dumper

Shovel-dumper combination is chosen in most of the metal mines, where the ore body is irregularly placed as shown in Figure 1.3 or on coal mines where the ore body is not in a horizontal or nearly horizontal layer.

As shown in Figure 26.10, a shovel starts scraping the blasted rock into its bucket at the level of the ground on which it stands and moves the bucket forward while raising it till the maximum attainable height H is reached, or the bucket is completely filled with blasted rock. It then slews into an angle of about 120° to 135°, with the bucket filled with the broken rock and held at a height of H_1, which is higher than the height of the dumper body. When the bucket comes into position above the dumper body, the contents of the bucket are unloaded in dumper body by opening the bottom plate of the bucket.

After dumping the broken rock, the shovel turns back again and starts scraping the broken rock face. A complete cycle of excavation and dumping of blasted rock takes average time of about 22 to 26 seconds.

The maximum attainable height of scraping, called cutting height, for a shovel depends upon the machine model. For most of the shovels used in large surface mines the cutting height lies between 10 m to 18 m.

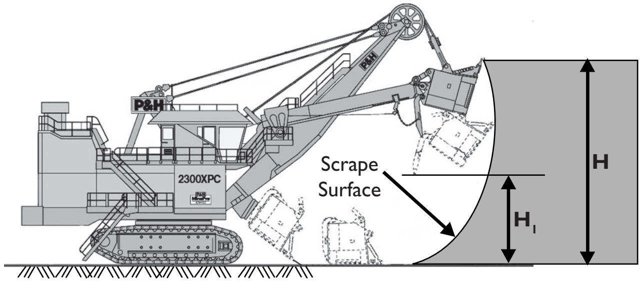


Figure 26.10 Working of an electric shovel.

The height of the bench and so the depth of the blasthole is decided on the basis of maximum cutting height of the shovel, after giving due consideration to the subdrilling needed and the inclination of the blasthole. A formula, as given below, was proposed for fixing bench heights in metal mining that uses shovel-dumper combination.

$$H = 10 + 0.57 * (C_c - 6)$$

where
H = Bench height in m
C_c = Capacity of shovel bucket in m^3

However, this formula should be used with great caution because it holds good only for a few small and medium size shovels.

Table 26.7 gives some details of the bucket capacities and heights of cut for some well known rope shovels.

26.4.5.2 Walking dragline

Walking draglines are used in coal mines. They do not need any hauling equipment because with their long booms they can scrape blasted overburden and throw it a long distance away. The working of a walking dragline is shown in Figure 26.11.

While removing the overburden lying above a horizontal or near-horizontal coal bed, a dragline stands on the blasted overburden and throws its bucket at a long distance on the top of the exposed coal bed. It then starts pulling the bucket towards itself. In this action the bucket scrapes the overburden and gets filled. The filled bucket is then lifted well above the ground and the dragline turns into an angle of about 90° and dumps the material into a heap on the side.

As the draglines are far bigger than shovels, the depth to which they can excavate is much greater. Boom length, bucket capacity, depth of excavation for some well known walking draglines is given in Table 26.8.

In some coal mines the coal beds are at a shallow depth and the thickness of the overburden layer is small. In such cases the bench height and depth of blastholes is decided on the basis of the thickness of the layer and inclination of the blastholes rather than the depth of excavation of the dragline.

Table 26.7 Heights of cut for some electric mining shovels.

Make	*Model*	*Bu. capacity m^3*	*Bu. capacity MT*	*Height of cut (m)*
Bucyrus Intl.	295HD	21.3	38	
Bucyrus Intl.	295HR	25.5	45	
Bucyrus Intl.	395	35.7	63.5	
Bucyrus Intl.	495	61.2	109	18.4
P&H Mining	2300XPC	25.5	45.4	13.5
P&H Mining	2800XPC	35.7	63.5	16.6
P&H Mining	4100XPC	61.2	108.9	16.8

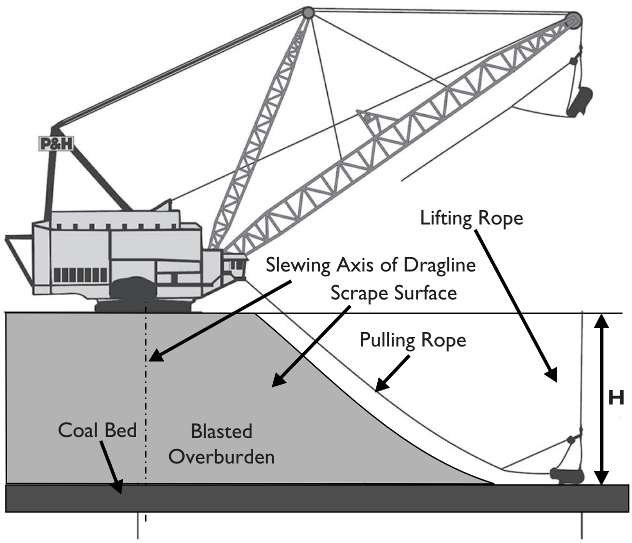


Figure 26.11 Working of a walking dragline.

Table 26.8 Depth of excavation for some walking draglines.

Make	*Model*	*Boom length (m)*	*Bucket capacity (m^3)*	*Excavation depth (m)*
Bucyrus Intl.	680 W	58 (min)	24	
		90 (max)	12	
Bucyrus Intl.	W2000	75 (min)	34	50.5
		101 (max)	24	74.5
Bucyrus Intl.	8050	99 (min)	61	
		99 (max)	45	
Bucyrus Intl.	8200	84 (min)	84	
		122 (max)	51	
Bucyrus Intl.	8750	109 (min)	116	
		132 (max)	76	
P&H Mining	9010C	100 (min)	57	56.4
		107.7 (max)	38	68.3
P&H Mining	9020C	88.4 (min)	89	54.8
		123.4 (max)	54	85.7
P&H Mining	9030C	99.1 (min)	117	56.5
		129.5 (max)	90	76.7

26.4.5.3 Combination of wheel loader and dumper

In coal mines the layers of coal are soft and distinctly separate from the overburden. Therefore, wheel loaders are sometimes used as equipment for loading coal into dumpers in coal mines.

A wheel loader works differently than a shovel or a dragline. With its bucket in the lowermost horizontal position, as shown on the right side in Figure 26.12, it moves forward. With this movement the coal or other loose material gets filled into its bucket. When sufficient volume of loose material is filled into the bucket it turns the bucket in upright position and simultaneously starts lifting the bucket in a backward movement on a curve. By the time it goes sufficiently on the back side, the bucket is in its highest position. The wheel loader again starts moving on a curve to approach the dumper placed on its side. When it has moved sufficiently towards the dumper and its bucket is in position just above the body of the dumper, the wheel loader tilts its bucket so as to unload the material in the dumper body. This completes one cycle and the dumper is again ready to start a second cycle.

Unlike the shovel, a wheel loader does not scrape material. It has relatively low height and its bucket raises only to sufficiently high level to load a dumper of matching size. A wheel loader finds it a little more difficult to load material from a steeply sloping heap but is at ease while loading material from a low height, well spread heap. It is, therefore, well suited for handling coal.

Bench height and, therefore blasthole depth, is limited in the case of coal beds where a wheel loader has to work.

The bench heights selected for blastholes of different diameters varies considerably. Usually followed trends in this regard are as expressed through Table 26.9.

If blastholes in a bench are vertical with no subdrilling, their blasting gives rise to a large stump as shown in left hand drawing in Figure 26.13. That is the reason why subdrilling to some depth below the intended ground surface created after blasting becomes essential.

Subdrilling value i.e. length of subdrilling, is usually kept to 8 to 12 times the blasthole diameter.

26.4.6 Blasthole inclination

Often blastholes are drilled at some angle with the vertical rather than the usual vertically downward direction. Such inclined blastholes offer many advantages but many disadvantages as well.

Various parameters associated with vertical and inclined blastholes are shown in Figure 26.13.

Advantages of a blast engineered with inclined blastholes are:

1 Length of explosive column in the blasthole increases and at the same time true burden, shown by B_1 in the figure, decreases. With this either better

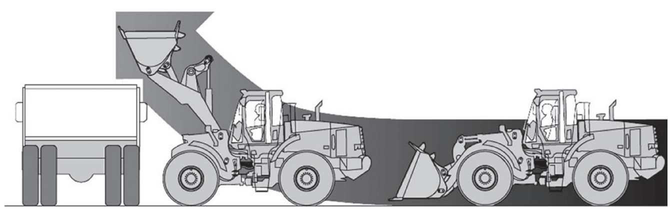


Figure 26.12 Working of a wheel loader.

Table 26.9 Bench heights chosen for blastholes of different diameters.

		Bench heights in m (ft)																
Blasthole dia. mm	*ins*	*9.14 (30)*	*10.06 (33)*	*10.97 (36)*	*11.89 (39)*	*13.11 (43)*	*14.02 (46)*	*14.94 (49)*	*15.85 (52)*	*17.07 (56)*	*17.98 (59)*	*18.90 (62)*	*20.12 (66)*	*21.03 (69)*	*21.95 (72)*	*22.86 (75)*	*24.08 (79)*	*24.99 (82)*
152.40	6																	
158.75	6.25																	
171.45	6.75																	
187.33	7.375																	
200.03	7.875											Preferred						
228.60	9																	
250.83	9.875																	
259.88	10.625																	
279.40	11																	
311.15	12.25																	
349.25	13.75		Not Preferred															
381.00	15																	
406.40	16																	

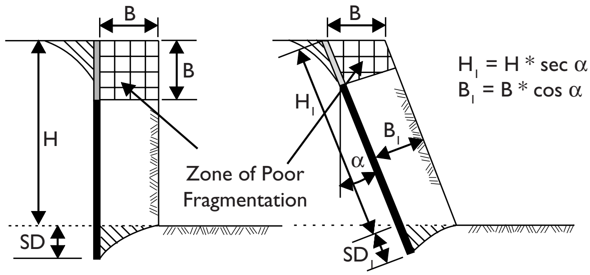


Figure 26.13 Parameters associated with vertical and inclined blastholes.

fragmentation results or the burden B can be increased to get the same level of fragmentation.

2 Volume of zone of poor fragmentation near the bench top, shown by square hatching in the figure, decreases. This means better overall fragmentation.
3 Length of subdrilling can be decreased because formation of hump at the newly formed bench floor is decreased. In certain cases subdrilling can be totally eliminated.
4 Yield of blast, i.e. the volume of blasted rock per meter length of the blasthole, is higher in case of inclined blastholes as compared to vertical blastholes. For this reason cost of drilling and blasting is decreased.
5 Possibility of backbreaks is much less in a blast with inclined blastholes. Backbreaks caused by vertical blastholes like those shown in Figure 26.14 are very dangerous. In many cases they make it difficult to place the blasthole drill for drilling holes in the first row.
6 New bench face gets a profile in such a way that the burden for the inclined blastholes in the next round is more or less uniform. The slope of the bench face is also more stable. This is elucidated in Figure 26.15.
7 Probability of misfires is reduced.
8 Large quantum of explosive energy, which always radiates in a plane perpendicular to the blasthole alignment, is directed towards the free surface. In addition to better fragmentation this also results in less energy for creating ground vibrations.
9 Reduced over-crushing of the rock mass, hence less wastage in the case of minerals like coal.

Disadvantages linked with inclined blastholes are:

1 Aligning the blasthole drill for inclined blastholes is rather difficult and often results in alignment error by incorrectly choosing the spot for starting the drilling operation.
2 Collaring i.e. start of drilling an inclined blasthole, is somewhat more difficult.
3 Chances of hole deviation while drilling inclined blastholes are far higher. Drilling operations have to be done very carefully.

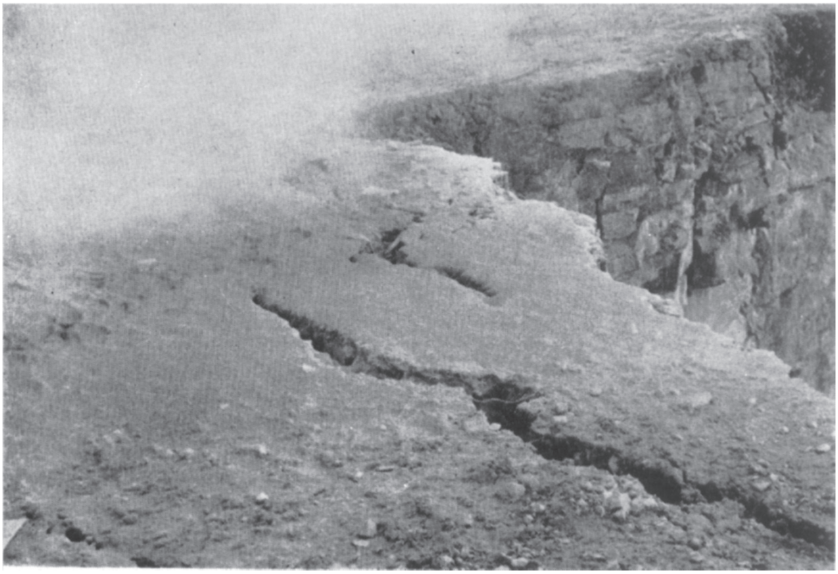


Figure 26.14 Dangerous backbreak.

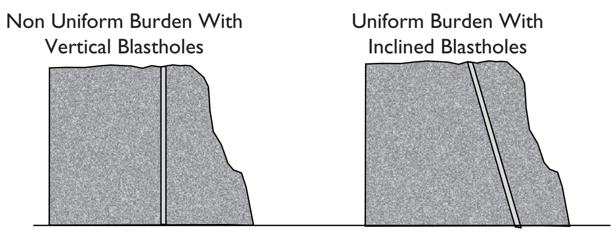


Figure 26.15 Uniform burden with inclined blastholes.

4 More wear and tear is experienced by all the drill string components. This is particularly true in the case of large diameter rotary drills that have to exert very high feed force and torque on the drill string.
5 Since the blasthole drill has to resist heavy horizontal force, the fatigue life of the drill is also reduced.
6 Throw and flyrock hazards of blasting inclined holes is somewhat higher than that of vertical holes.
7 Lesser penetration rate due to reduced efficiency of flushing.
9 Larger quantum of rock mass is thrown at a greater distance. Due to this the muck pile is of lesser height.

All the above disadvantages become more and more severe as the blasthole inclination increases.

In mines equipped with wheel loaders and dumper combination, inclined blastholes are practiced in many cases. The angle of inclination is usually of the order of

15° to 20° but in some cases it is as high as 30°. Properties of the rock mass, particularly the orientation of joints and other types of discontinuities, play an important role in the choice of angle of inclination.

The muck pile obtained from a blast of vertical blastholes is more suitable for shovel and dumper combination. Almost all metal mines practice blasts of vertical blastholes.

Long inclined blastholes tend to deviate more than vertical blastholes. Therefore, blastholes in dragline mining are usually vertical.

26.4.7 Burden

If burden is excessively large, then only cracks will be developed in the rock mass. There will be no separation and hence no fragmentation will occur. The energy released by the detonation of explosive will be utilized in causing heavy vibrations in the rock mass. As against this, if the burden is very small, the gases will escape towards the bench face with very high velocity and the fragments of the rock near the face will be thrown in the air violently to cause flyrock problems that can prove disastrous or fatal in many instances.

Several formulae have been proposed by mining technologists for the calculation of burden. They all relate burden value in terms of some other parameters. It can be stated that evaluation of burden by using these formulae gives a value lying somewhere between 25 * D to 40 * D, where D is the diameter of the blasthole.

The most appropriate value of burden certainly depends upon the hardness of the rock mass. Figure 26.16 shows the variation of burden for different diameters of blastholes and different rocks.

Density of rock as well as type of explosive used for blasting have significant influence on the value of burden, because in the case of heavier rocks a higher part of

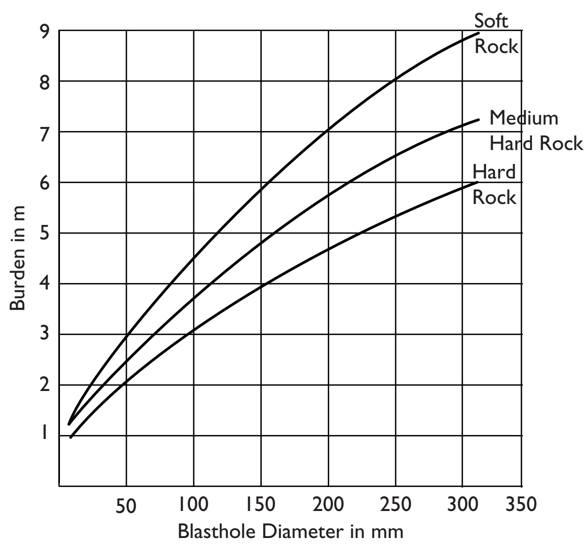


Figure 26.16 Burden as a function of blasthole diameter.

the explosive energy is spent in lifting and throwing the rock mass. In order to avoid this, burden is reduced while blasting with low energy explosives or when the bench has high density rock mass. Table 26.10 gives guidelines for calculating the value of burden for rocks of different densities in case of ANFO and Slurry Dynamite which are low and high energy explosives respectively.

Many researchers have tried to establish a correlation between burden spacing and other blasthole parameters. Details are given in a later subsection.

26.4.8 Blasthole spacing

Spacing is usually calculated from the value of burden. When blastholes are drilled for a mine bench, spacing of value 1.1 * B to 1.5 * B, where B is the value of burden, is chosen. Even within this range the value of 1.1 * B has been found to be more appropriate for large diameter blastholes and 1.5 * B is more suitable for small diameter blastholes. Figure 26.17 gives guidelines for appropriate choice of spacing in terms of burden.

Spacing value of less than 1.0 * B should never be used except when using controlled blasting techniques such as smooth blasting or cushion blasting.

When blastholes in a row are very closely spaced, a crack from one blasthole progresses to the other blasthole very rapidly. With this, the tendency of forming a huge block of rock mass between the first row and bench face increases. Large craters are also formed at the bottom of the blasthole. This gives rise to stumps at the toe of the previous bench face. If the blastholes have large spacing in the same row, the chances of formation of humps in the newly formed bench face increase. Similarly the likelihood of formation of stumps at the toe of the previous bench increases.

Table 26.10 Dependence of burden on rock density and type of explosive.

	Values of burden in terms of blasthole diameter D for rocks of different densities		
Type of explosive	*Low 2200 kg/m³*	*Medium 2700 kg/m³*	*High 3200 kg/m³*
ANFO	28 * D	25 * D	23 * D
Slurry Dynamite	33 * D	30 * D	27 * D

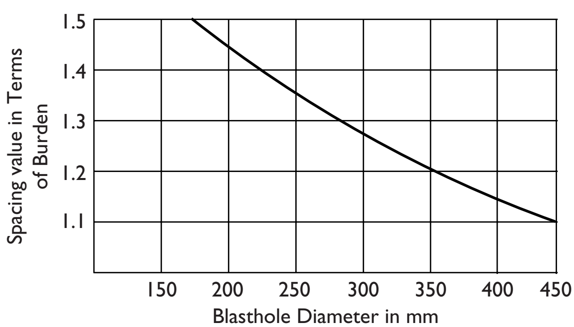


Figure 26.17 Spacing as a function of burden.

Both these phenomena are illustrated in Figure 26.18.

Determining the spacing-to-burden ratio is greatly affected by the geological conditions of the rock mass. Conditions when holes in the same row have to be blasted at different instances do arise. Often value of spacing is arrived at after onsite experience.

26.4.9 Subdrilling

Subdrilling means the length of blasthole drilled below the level of surface of the bench floor. This is well illustrated in Figure 26.1.

In bench blasting it is always desirable that after the bench blast and subsequent removal of the fragments, the new floor formed will be at the same level as that of the original bench face, and there will be no stumps i.e. protrusions of unfragmented rock out of the newly formed surface. The need for subdrilling is to ensure these objectives are easily achieved.

A blasthole is always initiated from its bottom. Therefore, the detonation zone travels from bottom to top. For this reason the compression waves do not travel in a plane perpendicular to the blasthole but at an upward angle. If a blasthole drilled without subdrilling is detonated from the bottom, there is always a zone as shown in the left hand sketch in Figure 26.19 where compression waves do not reach and tension waves are not reflected. Thus, the rock mass in this zone remains unfragmented, giving rise to stumps.

Whereas if a blasthole drilled with subdrilling is detonated from the bottom, there is no zone of unfragmented rock as shown in right hand sketch in Figure 26.19. Thus, no stumps remain when the fragmented material is removed

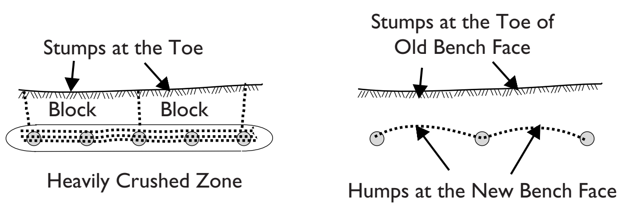


Figure 26.18 Effects of spacing as a function of burden.

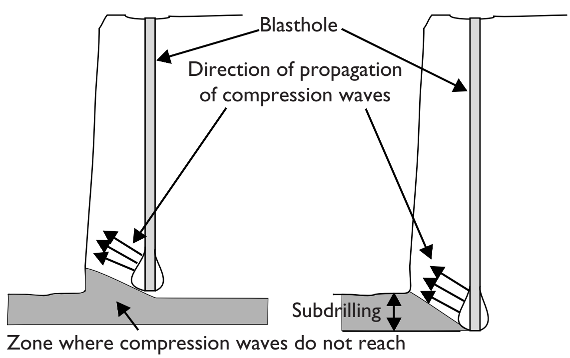


Figure 26.19 Need for subdrilling.

Subdrilling length has to be optimum. If the subdrilling is too large it will result in:

1 Increase in drilling and blasting costs
2 An increased vibration level
3 Excessive fragmentation of the bench, affecting slope stability
4 Increased risk of cutoffs and overbreak

Optimum length of subdrilling depends upon the strength of the rock. Table 26.11 presents the subdrilling length to be used for blasting in intact rocks of different strength. In cases where the rock is highly fractured, subdrilling length can be reduced by about 25% as shown.

In some coal mines there is a horizontal plane of weakness between the overburden layer and coal layer. In such cases, to ensure that only the overburden layer is fragmented and removed without fragmenting the coal layer, no subdrilling is practiced. In fact, in such circumstances it may be worthwhile to have the bottom of the blasthole at some distance above the top of the coal bed.

If the blastholes are inclined, the subdrilling length can be reduced as shown in Figure 26.20.

Table 26.11 Recommended subdrilling length in terms of burden.

Strength classes of the rock	*Subdrilling in intact rock condition*	*Subdrilling in highly fractured conditions*
Soft Rock with Easy Toe	0.1 B to 0.2 B	0.07 B to 0.15 B
Medium Rock with Normal Toe	0.3 B	0.25 B
Hard Rock with Difficult Toe	0.4 B to 0.5 B	0.3 B to 0.4 B

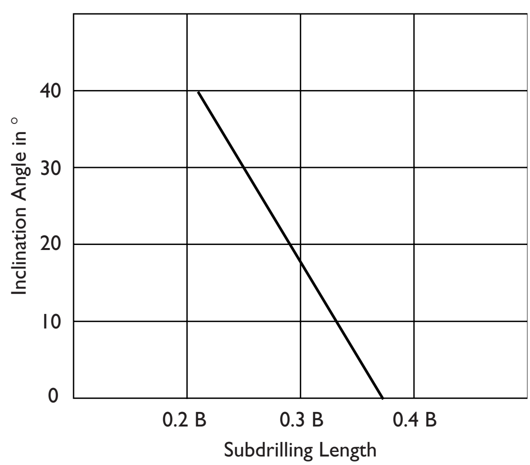


Figure 26.20 Reduction of subdrilling with blasthole inclination.

26.4.10 Stemming height

While charging the blasthole, some part of the blasthole is intentionally filled with inert material like rock chips formed in drilling. This is called stemming. Almost invariably some length of a blasthole at its top is used for stemming. The purpose of stemming is to confine gases generated in detonation of the explosive and utilize the energy generated by the blast in the most effective manner.

If stemming length is very small it leads to a premature escape of the gases leading to an airblast and a danger of flyrock, the hurling of rock fragments in a blast. On the other hand, if stemming length is excessively long, the portion around a blasthole near bench top is not subjected to sufficient cracking by the blast and large boulders are formed. It also leads to reduced loosening of the rock and subsequent difficulty in its removal.

Apart from stemming length, the other parameters associated with stemming column are the type and size of material to be used for stemming. Studies have shown that coarse angular material, such as crushed rock, is the most effective stemming product. The optimal stemming length varies between 18D to 30D, where D is the diameter of the blast hole. Crushed rock of size between 0.04 D and 0.06D is found to be most suitable as stemming material. It has been found to effectively lower the stemming length by about 41%. Use of stemming plugs in the blasthole gives lower mean fragment size.

For some specific objectives, stemming is also used in between two or more zones of explosive columns.

26.4.11 Size of the blast

The shape of the blast refers to the geometrical form of the area on the bench top from which all the blastholes are detonated together with delays in their detonation timings. In surface mines the shape of the area is almost invariably rectangular.

Such area has length and width. The length is measured along the bench crest and the width is measured perpendicular to the bench face.

As the first, second, third rows, and so on, are blasted with delay timings, the fragmented rock mass from the burden of the first row is thrown on the bench floor at a long distance. The fragmented rock mass from the burden of the second row is also thrown horizontally but a larger portion of this sits on the top of the rock mass of the first burden but near to the bench. For the blast of the third row the same phenomenon is repeated and the height of the heap increases as shown in Figure 26.21.

The advantage of a large blast area is that it allows more effective utilization of loading and hauling equipment.

Disadvantages of a large blast area are:

1 The intensity and duration of vibrations created in the ground by the blast increases.
2 The face of the bench formed after removing the blasted material from bench floor becomes crooked like the one shown in Figure 26.15.
3 Fly rock hazard increases as it becomes difficult to introduce appropriate delays in the blasting of rear rows.

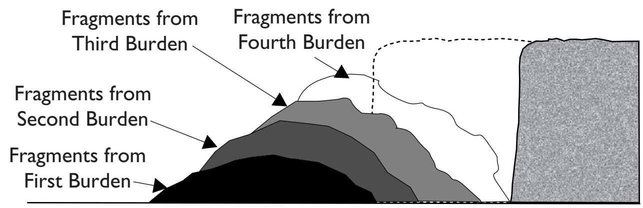


Figure 26.21 Formation of a heap of fragments after the blast.

If the equipment used for material removal can handle tall heaps of blasted fragments, then even 7 or 8 rows can be taken for a blast. In the case of coal beds the number of rows reduces to 3.

26.5 CALCULATION OF BURDEN

The most important parameter in blast design is the burden. This is because all the rockmass contained in the burden needs to be fragmented by the energy evolved in the blast.

Due to the impossibility of mathematical derivation, many researchers have tried to arrive at empirical relations between many parameters of the blasthole and the environment and the burden. The following elaboration is based upon this.

Some of these relations are explained here below.

Fraenkel (1952)

$$B = (R_v * L^{0.3} * l^{0.3} * D^{0.8})/50$$

where

B = Burden in m
L = Length of blasthole in m
l = Length of charge in m
D = Diameter of blasthole in mm
R_v = Resistance factor. It varies between 1 and 6. For rocks with high UCS it is 1.5 and for rocks with low UCS it is 5

Pearse (1955)

$$B = K_v * 10^{-3} * D * (PD/RT)^{0.5}$$

where

B = Maximum burden in m
K_v = Rock property constant that varies between 0.7 to 1.0
D = Diameter of blasthole in mm

PD = Detonation pressure in kg/cm^2
RT = Tensile strength of rock in kg/cm^2

Hino (1959)

This method of calculating the burden, proposed by Hino, takes into consideration many variables but somewhat intricately.

The first equation is:

$$B = (D/4) * (PD/RT_d)^{1/n}$$

where
B = Burden in m
D = Diameter of blasthole in cm
PD = Detonation pressure kg/cm^2
RT_d = Dynamic tensile strength in kg/cm^2

The value of the characteristic constant n is to be determined from the data obtained in the crater-forming test for the explosive as under.

$$n = \log(PD/RT_d)/\log(2D_o/(d/2))$$

where D_o is the optimum depth of the center of gravity of the explosive charge in cm. It is to be determined graphically by using the equation

$$D_p = \Delta\Sigma V_1^{0.333}$$

where
d = Diameter of the charge
D_p = Depth of the CG of charge
Δ = Relationship ratio of depth D_p/D_c
D_c = Critical depth of CG of charge
Σ = Volumetric constant of the charge
V_1 = Volume of charge used in the test

CG means center of gravity.

Langefors and Kihlstrom (1963)

Equation proposed by Langefors and Kihlstrom for calculating burden is as under.

$$B_{max} = (D/33) * ((\rho_e * PRP)/(C_o * f * (S/B)))^{0.5}$$

where
B_{max} = Maximum burden for good fragmentation in m
D = Diameter at bottom of blasthole in m
ρ_e = Density of explosive in the blasthole in kg/L

PRP = Relative Weight strength of the explosive
f = Degree of fixation of the blasthole.

For vertical blastholes f = 1, for blastholes inclined at 3:1 f = 0.9 and for blastholes inclined at 2:1 f = 0.85

S/B = Spacing to burden ratio

The value of C_o is to be determined as under.

Let the quantity of explosive needed to fragment 1 m^3 of rock mass be called C. For hard rocks C is usually taken as 0.4. This value is to be modified as

For B = 1.4 to 15 m $C_o = C + 0.75$
For B = <1.4 m $C_o = C + 0.07/B$

From the B_{max} determined by using above Formulae, the actual B to be used is to be calculated as

$$B = B_{max} - e_c - d_b * H$$

where
e_c = Collaring error in m/m
d_b = Blasthole deviation in m
H = Bench height in m

Lopez Jemino (1980)

Lopez proposed the following equation for calculating burden

$$B = 0.76 * D * F$$

where
B = Burden in m
D = Diameter of blasthole in inches
F = Correction factor

The correction factor is to be evaluated by using following equations.

$$F = f_r * f_e$$
$$f_r = ((2.7 * 3500)/(\rho_r * VC))^{0.333}$$
$$f_e = ((\rho_e * VD)/(1.3 * 3660^2))^{0.333}$$

where
ρ_r = Specific gravity of rock in g/cc
VC = Velocity of seismic propagation in m/s
ρ_e = Specific gravity of explosive in g/cc
VD = Velocity of detonation for explosive in m/s

These formulae are valid for blastholes of diameters from 165 to 250 mm. When the diameter of blasthole is larger the burden should be reduced by a factor 0.9.

Konya and Walters (1972 and 1983)

In the year 1972 Konya and Walters proposed an equation for calculation of burden as under,

$$B = 3.15 * D * (\rho_e/\rho_r)^{0.333}$$

where

B = Burden in ft
D = Diameter of explosive in inches
ρ_e = Specific gravity of explosive in g/cc
ρ_r = Specific gravity of rock in g/cc

On the basis of burden distance arrived at by using the above equation, the spacing distance can also be calculated as per following equations suggested by Konya and Walters. For instantaneous blast of single row blastholes

$S = (H + 2B)/3$ when $H < 4B$
$S = 2B$ when $H > 4B$

For sequenced blast of single row blastholes

$S = (H + 7B)/8$ when $H < 4B$
$S = 1.4B$ when $H > 4B$

In all the above equations

B = Burden calculated as above in ft,
S = Spacing in ft
H = Bench height in ft

Stemming distance to be kept as under,

For massive rock $T = B$
For Stratified rock $T = 0.7 * B$

In the year 1983, Konya and Walters refined their approach and came up with the following equations.

$$B = ((2\rho_e/\rho_r) + 1.5) * D$$

Stemming distance $T = 0.7B$
Subdrilling distance $J = 0.3B$

26.6 RELATIONSHIP OF BLASTHOLE PARAMETERS

For every blast there is such a relationship between the geometrical parameters of the blasthole, that proves to be the best for the blast from the viewpoint of fragmentation as well as safety.

Ash adapted the simplest approach to establish the relationship. The following elaborations are largely based on his paper published in 1963.

All the parameters used for establishing the relationship are shown in Figure 26.22.

Spacing to burden relationship

As has been seen earlier in this chapter, spacing is considered to be proportional to the burden and can be expressed as,

$$S = K_s * B$$

where K_s is the proportionality constant.

Burden to blasthole diameter relationship

Let us presume that for a certain burden and spacing, one unit of blasthole depth gives satisfactory blasting. Therefore, a volume V of rock mass fragmented by the energy output of the explosive in one unit depth of blasthole works out as:

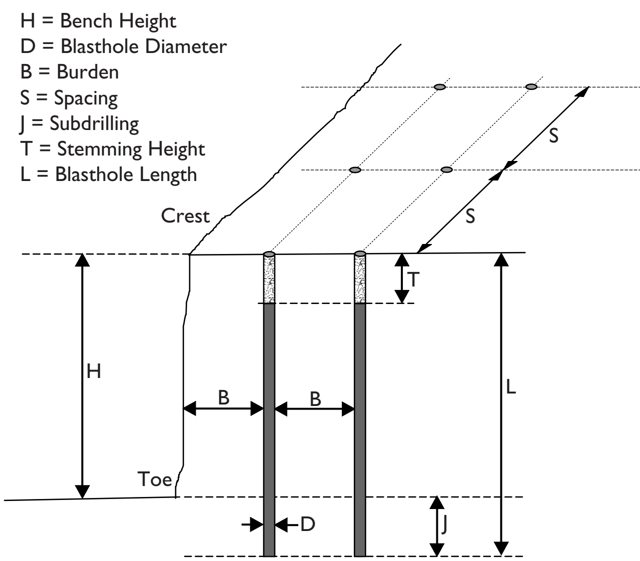


Figure 26.22 Symbolic nomenclature of a blasthole.

$$V = B * S * 1 = K_s * B^2$$

We can equate the explosive energy output from one unit length of blasthole as

$$E_o = (\pi/4) * D_2 * E_u * \rho_u$$

Since in a blast the explosive is same, and the charging density of the explosive is constant, we have

$$E_o \propto D^2$$

Similarly,

$$E_o \propto V \propto D^2 \text{ i.e. } B^2 \propto D^2 \text{ or } B \propto D$$

This gives:

$$B = K_b * D$$

Naturally, as the diameter of the blasthole increases the burden also increases, in the burden-to-blasthole diameter ratio K_b.

Subdrilling to burden relationship

As has been seen earlier, for all the explosives and particularly ANFO, the velocity of detonation at the point of detonation is somewhat low but increases as the detonation progresses. Further, the energy output is also proportional to the velocity of detonation.

The purpose of keeping some subdrilling length is to ensure that the velocity of detonation in the explosive reaches its maximum when the detonation reaches the floor level of the bench.

In case of ANFO it has been found that the length required for the velocity of detonation to reach its maximum is 6 times the diameter of the blasthole.

It is also true that the detonator and primer together are never at the bottom of the blasthole but at a height of about 2D. Therefore, $J = 8D$ and since $D \propto B$, we have $J \propto B$.

In other words,

$$J = K_j * B$$

where K_j is the subdrilling-to-burden ratio.

Stemming to burden relationship

In bench blasting practice, the stemming length is also considered to be related to the burden by a stemming-to-burden ratio K_t. In other words,

$$T = K_t * B$$

Based upon some tangible logic involved in the blasting process, Langefors and Kihlstrom have concluded that $K_t = 0.7$

Bench height to burden relationship

Bench height is always more than burden. The ratio of bench height to burden is symbolized by K_h and the equation relating bench height and burden is

$$H = K_h * B$$

On the basis of a study of the outcome of many blasts, Ash established that K_h should be more than 1.6.

Length of blasthole

When the blastholes are vertical the length of blasthole is the sum of bench height H and subdrilling length J. The equation below can be established in this regard:

$$L = H + J$$

When blastholes are inclined with the vertical at an angle β, the relationship takes the form:

$$L = H/\cos\beta + (1 - \beta/100) * J$$

26.7 DESIGN OF CONVENTIONAL SURFACE BLAST

Various parameters of a conventional bench blast with blastholes are shown in Figure 26.25.

A method to affix the values of these parameters is as under. This method is based on the crater theory put forth by Livingstone. One of the basic assumptions for the blasthole is that Length/Diameter is less than 50.

1. Ascertain the diameter of blasthole D, on the basis of hourly production requirements by using Table 26.3.
2. On the basis of the diameter of the blasthole and the type of explosive used for blasting, find the burden B, and spacing distances S, by using Table 26.12.
3. By using Table 26.13 determine the subdrilling length J, in terms of blasthole diameter.
4. Arrive at the value for stemming length T, on the basis of blasthole diameter from Table 26.14.
5. Bench height H, can be determined on the basis of Table 26.15. As shown by Table 26.9, great variation can be allowed in bench height.
6. Length of blasthole $L = H + J$
7. Specific volume in m^3/m $V_b = \pi * (D_2/4) * 10^{-6}$

Table 26.12 Burden and spacing for different explosives in different types of rock mass.

Type of explosive	*Parameter*	*Type of rock mass and compressive strength in MPa* Soft UCS < 70	Medium 70 < UCS < 180	Hard UCS > 180 MPa
ANFO	Burden B	28D	23D	21D
	Stemming S	33D	27D	24D
Watergels/ Emulsions	Burden B	38D	32D	30D
	Stemming S	45D	37D	34D

Table 26.13 Subdrilling length for blastholes of different diameter.

Fragment weight in kg	*Blasthole diameter* 180–250 mm	250–445 mm
Subdrilling-J	7 D to 8D	5 D to 6D

Table 26.14 Stemming length for different types of rock masses.

Parameter	*Type of rock mass and compressive strength in MPa* Soft UCS < 70	Medium 70 < UCS < 180	Hard UCS > 180 MPa
Stemming-T	40D	32D	25D

Table 26.15 Bench height for different types of rock masses.

Parameter	*Type of rock mass and compressive strength in MPa* Soft UCS < 70	Medium 70 < UCS < 180	Hard UCS > 180 MPa
Bench Height-H	52D	44D	37D

8 Breakage volume is $V_r = B * S * H$
9 Yield of breakage is $Y_b = V_r/L$
10 Bottom charge length $L_b = 8D$ to 10D.
11 Column charge length is $L_c = L - T - L_b$

From the above basic parameters and properties of explosives like density of bottom charge ρ_b and density of column charge ρ_c, other parameters related to the blast can be determined as below.

1 Concentration of column charge $C_c = \rho_c * V_b$
2 Concentration of bottom charge $C_b = \rho_b * V_b$
3 Total bottom charge $Q_b = L_b * C_b$
4 Total column charge $Q_c = L_c * C_c$
5 Total quantity of explosive $Q_t = Q_b + Q_c$
6 Powder Factor $V_r = Q_r/V_r$

Example 1

In a large copper mine, blastholes of diameter 311 mm are being drilled to cope with the desired production rate. The formation has UCS of 150 MPa. ANFO with a charge density of 0.85 kg/L is used as the explosive in the column charge, and emulsion with charge density of 1.35 kg/L is used as explosive in the bottom charge.

Determine various parameters of the blast.

Solution 1

Since D = 311 mm and formation UCS is 150 MPa,
Burden B = 23D = 23 * 0.311 = 7.153 m
Spacing S = 27D = 27 * 0.311 = 8.397 m
Subdrilling J = 6D = 6 * 0.311 = 1.866 m
Stemming length T = 32 * D = 32 * 0.311 = 9.952 m
Bench height H = 44 * D = 44 * 0.311 = 13.684 m
Length of blasthole L = H + J = 15.55 m
Specific volume $V_b = \pi * 311^2/4 * 10^{-6} = 0.07597\ m^3/m$
Breakage volume V_r = 7.153 * 8.397 * 13.684 = 821.91 m^3
Yield of breakage Y_b = 821.91/15.55 = 52.856 m^3/m
Bottom charge length L_b = 8 * 0.311 = 2.488 m
Column charge length L_c = 15.55–9.952–2.488 = 3.11 m
Density of bottom charge is 1.35 kg/L = 1350 kg/m^3
Density of column charge is 0.85 kg/L = 850 kg/m^3

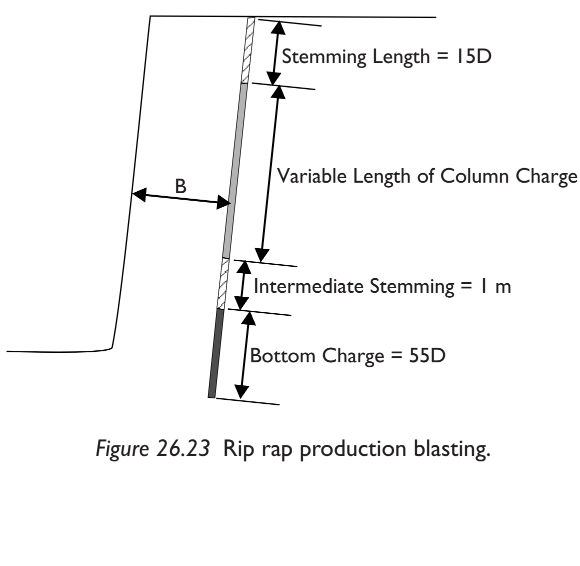


Figure 26.23 Rip rap production blasting.

Concentration of bottom charge $C_b = 0.07597 * 1350 = 102.56$ kg/m
Concentration of column charge $C_c = 0.07597 * 850 = 64.57$ kg/m
Total bottom charge $Q_b = 2.488 * 102.56 = 255.17$ kg
Total column charge $Q_c = 3.11 * 64.57 = 200.81$ kg
Total quantity of explosive $Q_t = 255.17 + 200.81 = 455.98$ kg
Powder factor = 455.98/821.91 = 0.555 kg/m^3

26.8 DESIGN OF OTHER TYPES OF BLASTS

Apart from conventional surface blasts, described in the previous section, there are other types of blasts in large mines. These are designed on somewhat different criteria.

This section deals with such blasts.

26.8.1 Rip rap production blasting

As said earlier, the objective of rip rap blasting is to get large pieces of rock. The following should be noted in the design of a blast in large quarries meant for the purpose. Relevant blasthole alignment is shown in Figure 26.23.

1 The rock mass should be as monolithic as possible. A heavily fractured rock mass is almost useless for the purpose as it gives smaller rock pieces.
2 The rock should be as dense as possible.
3 The diameter of blastholes is around 100 mm.
4 Bench height should be between 15 to 20 m.
5 Blastholes should be inclined at an angle 5° or 10° with the vertical.
6 Burden should be about 35 * D to 40 * D.
7 Spacing should be 1.5 * B to 2.0 * B.
8 Subdrilling should be 10 * D
9 Powder factor to be used in case of bottom charge should be:
   For rocks with UCS > 100 MPa PF > 650 g/m^3
   For rocks with UCS < 100 MPa PF < 500 g/m^3
10 Charge density in the plane of cut should be:
   For rocks with UCS > 100 MPa CD > 500 g/m^2
   For rocks with UCS < 100 MPa CD < 250 g/m^2
11 The whole row of the blastholes should have instantaneous charge.

Fragment distribution obtained in homogeneous rocks by using above blast design parameters has been found to be as given in Table 26.16.

26.8.2 Snake hole blasting

Initially snake hole blasting was thought of as an excellent means of achieving very good fragmentation. However, soon it was noticed that the disadvantages experienced were far too great to be offset by the advantages.

Table 26.16 Fragment size distribution achieved in rip rap blasting.

Fragment weight in kg	*Percentage in the fragmented rock*	
	Rock with UCS < 100 MPa	*Rock with UCS > 100 MPa*
>3000 kg	30	50
1000–3000 kg	20	25
50–1000 kg	25	25
Finer Than 50 kg	25	10

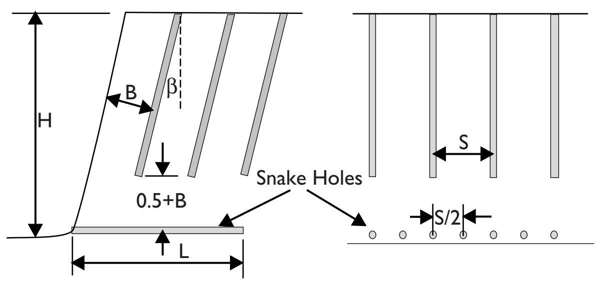


Figure 26.24 Snake hole blasting.

The main disadvantages were as under:

1 Very specialized drills are required for drilling horizontal blastholes.
2 Drilling horizontal blastholes without significant deviation is rather difficult.
3 Penetration rate of drilling horizontal blastholes is very low.
4 Horizontal drilling activity hampers the movement of other pieces of equipment to a large degree.
5 Working at two levels (on bench top and bench floor) is quite cumbersome.
6 Flyrock havoc caused by blast in the horizontal blastholes is excessively high and cannot be easily controlled.

Today, snake hole blasting is practiced in very few instances. Naturally, there has neither been much accumulated experience nor significant research to arrive at design criteria.

Snake blastholes are shown in Figure 26.24.

So far the practice followed in the field has been:

1 The horizontal blastholes are of diameter 89 to 110 mm, so drilling by small size top hammer or DTH hammer drill is possible.
2 Vertical blastholes are usually at an angle of 5° to 15°.

3 Vertical blastholes are drilled to such a depth that the distance between the bottom of the vertical blastholes and the horizontal blastholes, which in other words is the burden B_h of the horizontal holes, is (0.5 + B) m, where B is the burden for vertical blastholes.
4 Spacing of the vertical blastholes is as per the normal criteria of $S = K_s * B$, where K_s equals 1.0 to 1.5.
5 The spacing S_h of horizontal blastholes is 0.5 * S, where S is the spacing of vertical blastholes.
6 Horizontal blastholes have length L (i.e. depth) equal to $n * B/\cos\beta$, where β is the angle of inclination of vertical blastholes.

For the following aspects, the field practice is not known but author's recommendations are:

a Stemming length for vertical blastholes should be 0.8 * B to 1.0 * B.
b Stemming length of horizontal blastholes should be $1.0 * B_h$.
c Explosive used in vertical blastholes can be ANFO but in the horizontal blastholes explosive with higher strength- such as ALANFO- should be used.

26.8.3 Cast blasting

In the early 1980s an idea of using explosive energy for throwing the fragmented rockmass to the adjacent waste heap simultaneously through the same blast was born. Over the last decades the techniques of such use have been refined greatly.

For a successful cast blast the following parameters of the bench, pit, wall etc. should be practiced. The parameters are shown in Figure 26.25.

Bench height should be more than 12 m.

Pit width should be 1.25 times the height of the bench face.

Presplitting is essential because it enables the burden for the subsequent blast round to be the same for all the blastholes in the first row. Presplitting blastholes should be 251 mm dia., with spacing of 3 m in soft or tough rocks, and 5 to 6 m in hard and brittle rocks.

The type of formation matters in choosing the quantity of explosive. Since soft rocks fracture quickly and the gases formed in the explosion escape early, the throw in the soft rock masses is low. To compensate for this the quantity of explosive

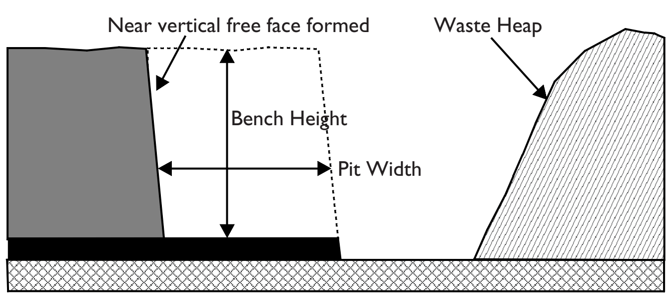


Figure 26.25 Parameters in cast blasting.

required in blasting soft rock is rather high. Hard rock masses require less quantity of explosive.

Burden also greatly depends upon geological factors of the rock mass. It is usually determined from a nomograph on the basis of total charge weight in a blasthole and rock blastability.

The explosive selected must be able to give sufficient ejection velocity so the broken fragments are thrown in the appropriate direction to the appropriate distance. The ejection velocity can be calculated from the following empirical equation:

$$V_e = 1.14 * (B/(0.07853 * d^2 * \rho_e * PAP))^{-1.17}$$

where

V_e = Ejection velocity in m/s
B = Burden in m
d = Charge diameter in cm
ρ_e = Density of explosive g/cm^2
PAP = Absolute weight strength in cal/g

Minimum ejection velocity is about 15 m/s.

Spacing should be about 1.3 to 1.6 times burden.

The blasthole layout pattern should be rectangular staggered.

Stemming should be 20 to 25 times blasthole diameter.

Subdrilling should be low i.e. 4 to 6 times blasthole diameter. This is essential so as to ensure that the coal layer is not broken, mixed and thrown with the overburden.

Powder factor varies between 0.3 to 0.8 kg/m.

All the blastholes in a row should be fired simultaneously without any delay so long as only one face is involved. If two faces are involved then the firing sequence should be as shown in Figure 26.26.

In order to ensure that material fragmented and thrown from the first row blast does not infringe with the material fragmented and thrown by the second row blast,

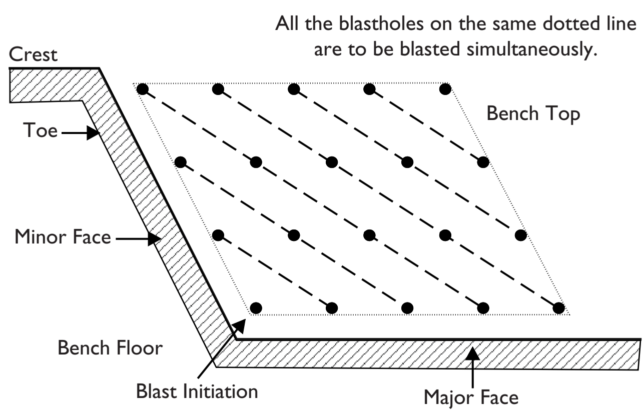


Figure 26.26 Firing sequence for cast blasting with two faces.

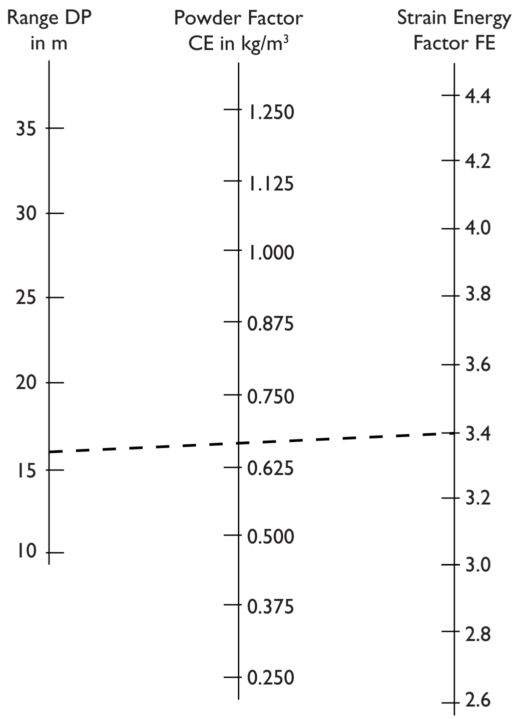


Figure 26.27 Nomograph 1.

the delay timing between rows should be worked out on the basis of 20 to 25 ms/m of burden.

A company, named D'Appolonia Consulting Engineers, has developed nomographs to evaluate some of the variables used in cast blasting. The nomographs are shown in Figure 26.27 to Figure 26.30 and their use is explained through Example 2.

Example 2

Cast blasting method is to be adopted in an existing coal mine. The following are the details of the existing method.

Blasthole diameter –250 mm
Strain energy factor FE –3.4
Bench height –13.5 m
Throw distance DP –16 m
Density of explosive –0.85 kg/L

Determine other parameters of the cast blast.

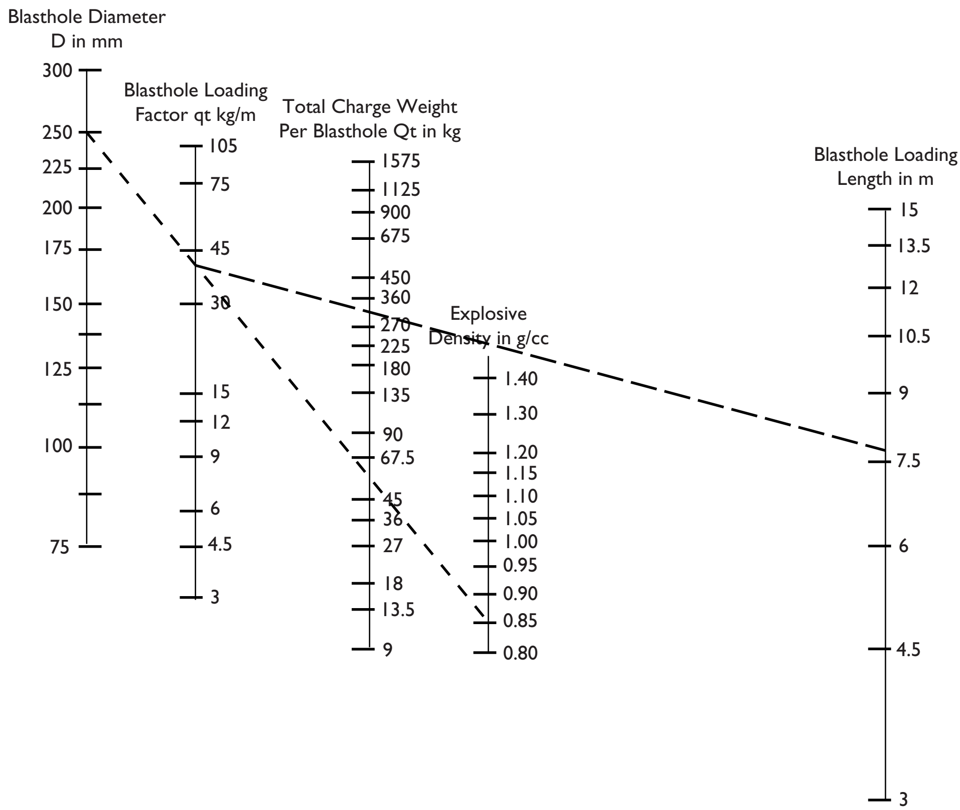


Figure 26.28 Nomograph 2.

Solution 2

The stepwise solution by using D'Appolonia Method is as under.

Step 1 – In nomograph 1, draw a straight line from FE = 3.4 to DP 16 m. This line intersects the powder factor (CE) line to give a powder factor value of 0.67 kg/m^3.

Step 2 – Using nomograph 2, draw a straight line from D = 250 to ρ_c = 0.85 kg/m^3. This line intersects the blasthole loading factor q_t as 40 kg/m.

Step 3 – Calculate C_1 and C_2 by assuming K_1 and K_2 to be equal to 1. This actually means that burden and spacing are both equal to each other.

$$C_1 = (10.66 * q_1)/CE * K_2 = 10.66 * 40/(0.67 * 1) = 636.42$$

$$C_2 = 0.3 * K_1 * 636.42/H = 0.3 * 1 * 636.42/13.5 = 14.14$$

Step 4 – By using the above values of C_1 and C_2 find out the value of C_3 from nomograph 3.

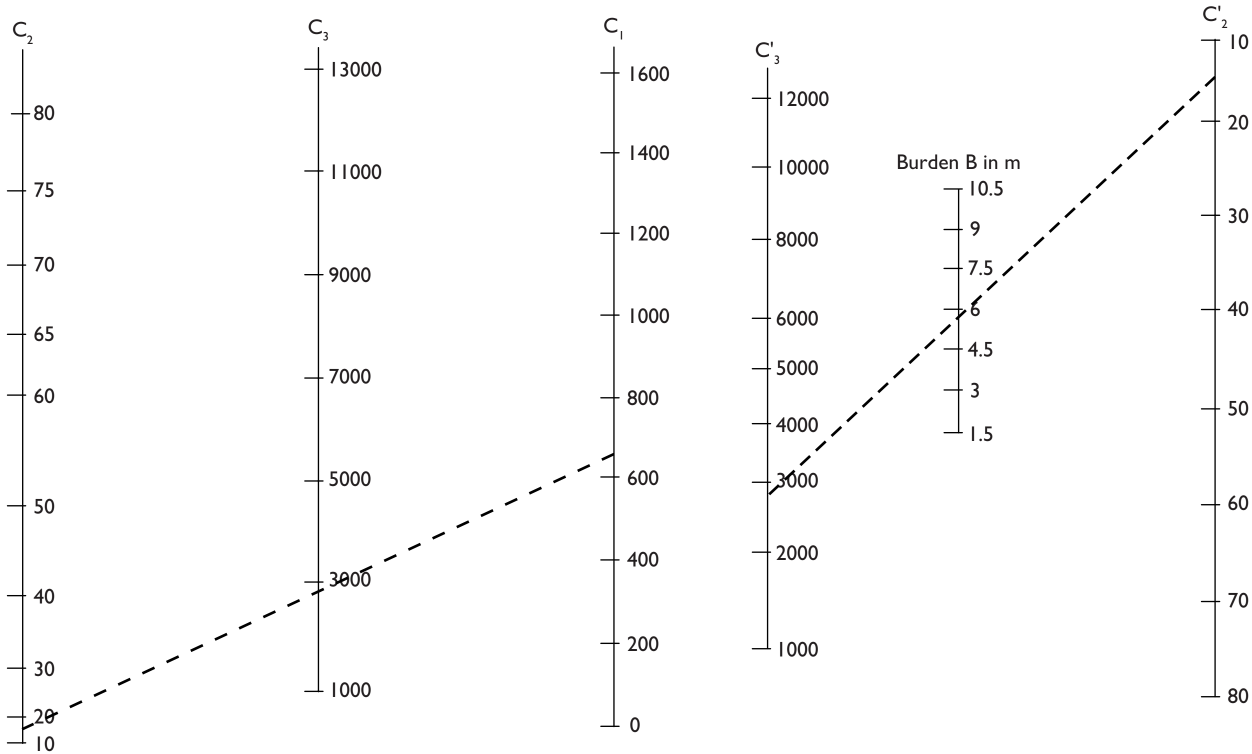
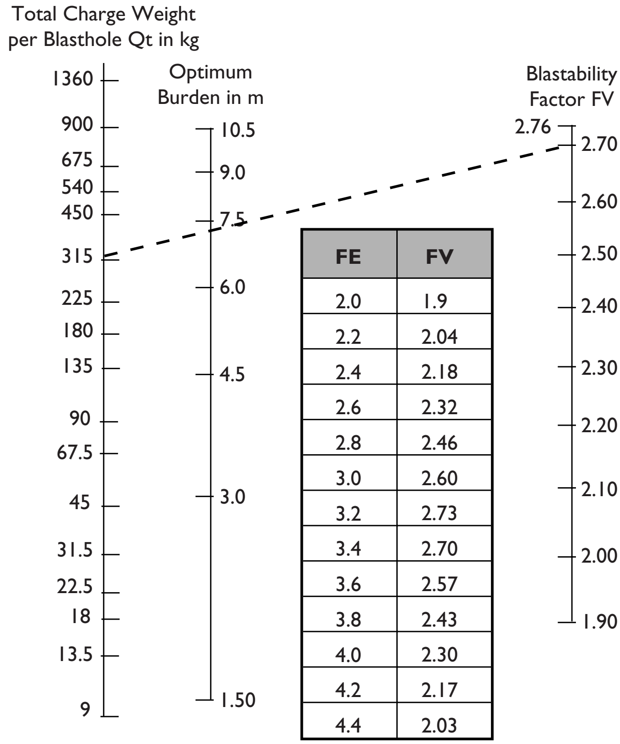


Figure 26.29 Nomograph 3.



FE	FV
2.0	1.9
2.2	2.04
2.4	2.18
2.6	2.32
2.8	2.46
3.0	2.60
3.2	2.73
3.4	2.70
3.6	2.57
3.8	2.43
4.0	2.30
4.2	2.17
4.4	2.03

Figure 26.30 Nomograph 4.

Step 5 – Let $C_3 = C'_3$ and $C_2 = C'_3$. Use these values to find out the burden B from nomograph 3. With this we get value of burden B as 5.8 m.

Step 6 – The length of charge in the blasthole works out to $l = H - K_1 * B = 13.5^{-1} * 5.8 = 7.7$ m

Step 7 – Using nomograph 2 again, determine Q_t from the values of q_t and l. In this case since $q_t = 40$ kg/m and $l = 7.7$ m. Total charge weight Q_t works out to 320 kg.

Step 8 – Since FE = 3.4, from the table in nomograph 4 the value of FV can be found out as 2.7. Now, using nomograph 4 with values of Q_t and FV to be 320 kg and 2.7 respectively the optimum burden B_o can be found out to be 7.2 m.

Step 9 – If the values of B and B_o are nearly equal the use of nomographs is over and stemming length T as well as spacing S can be determined by the equations $S = K_2 * B$ and $T = K_1 * B$.

Step 10 – In this case B and B_o are not nearly equal. For such cases the calculations are required to be repeated from step 3 by using different values of K_1 and K_2 and recalculating C_1 and C_2.

To avoid blind guesswork about K_1 and K_2, a rule of thumb has been suggested by D'Appolonia. The rule is to use the relation $K_2 = K_1^3$ for estimation.

It can be seen that with a value of $K_1 = 0.86$ and $K_2 = 0.636$, C_1 and C_2 work out to 1043.6 and 19.95 respectively. For these values B and Q_t works out to 7.3 m and 301.3 kg. With this value B_o also works out to 7.3 m.

Thus, Spacing S = 0.636 * 7.3 = 4.6428 m.

Similarly T = 0.86 * 7.3 = 6.28.

Since the procedure described above is based on nomographs backed by sound mathematics, the whole procedure can be converted into a computer program. With such a program it is possible to get quick and more precise results.