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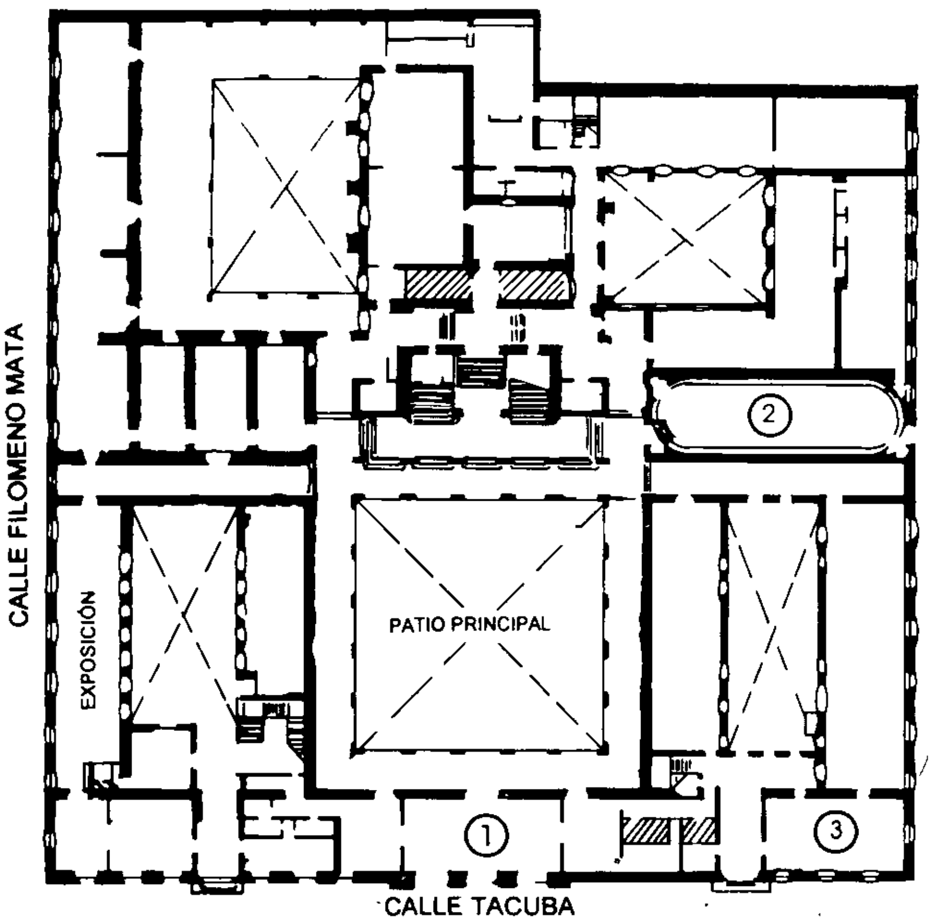

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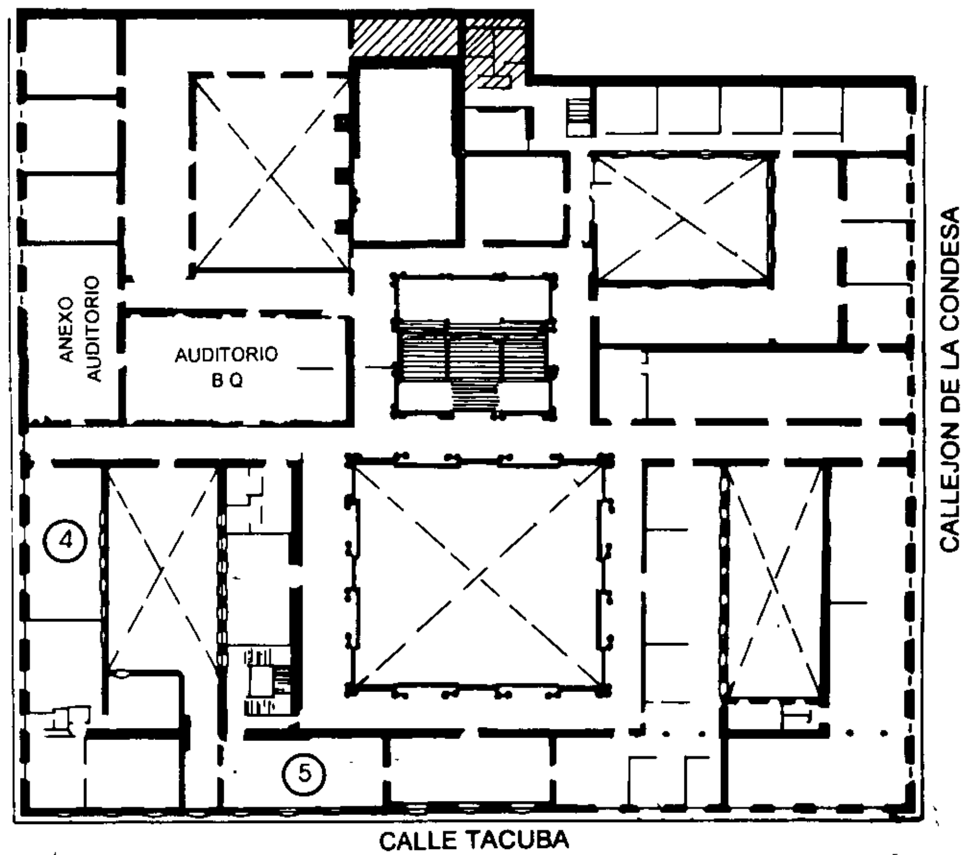

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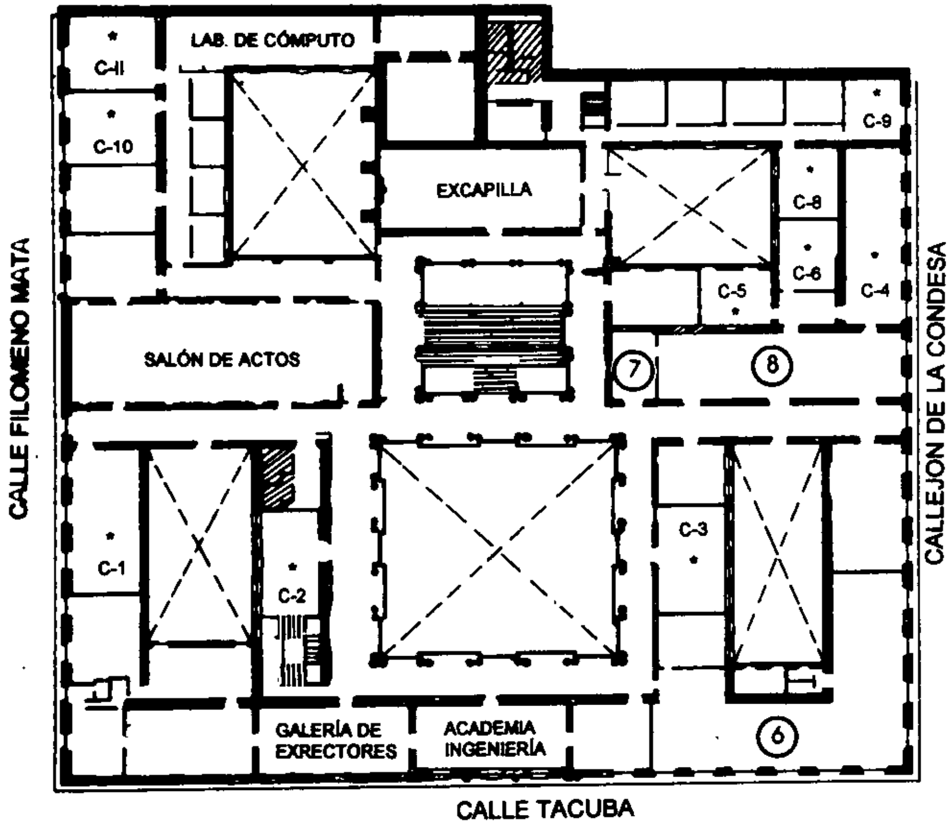

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TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

NOTES ON DETONATION PHISICS

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NOTES ON DETONATION PHYSICS

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CHAPTER 1

THE DETONATION PROCESS

1.1 Introduction

According to Persson[1] steady state detonation along a cylindrical charge can be regarded as a self propagating process in which the axial compressive effect of the shock front discontinuity changes the state of the explosive so that exothermic reaction sets in with the requisite velocity.

This reaction in homogeneous liquid explosives such as nitroglycerin is completed in a time interval of the order of 10^{-12}seconds[1]. In high explosives, such as RDX and PETN it is completed in about 1μsec . In composite explosives containing AN the reaction times are considerably longer. The significance of this will be demonstrated later.

1.2 Shock waves

Compressional waves of small intensity are propagated in gases at the velocity of the sound. Let us suppose that a column of gas is set in motion by a piston which is accelerated into it. Let us also consider that the velocity of the piston is a staircase function of time. Each step transmits a small compressional wave which advances through the gas already set in forward motion and heated by the previous waves. Since the velocity of the wave is larger at elevated temperatures, the new wave overtakes the previous[2]. Therefore the velocity, pressure and temperature gradients in the front of the wave grow steeper with time. If there is no dissipative mechanism (e.g. heat diffusion) the gradients become infinite[2].

This type of wave, in which a discontinuity has developed is known as a shock wave. The area of pressure rise is called the shock front. The front advances with a speed higher than the sound speed. The shock velocity depends on the conditions behind. If the pistons continues accelerating so does the front. If the piston maintains a constant velocity, the front maintains a constant velocity as well. If the piston decelerates a wave of rarefaction is formed ahead of it. Finally this wave overtakes and weakens the shock front.

It follows that the velocity of the front is determined by the conditions behind the front. The wave does not maintain itself. Rather it depends on the support provided by the piston.

1.3 Detonation waves

However from our experience we know that steady detonation waves exist. In this case the role of the piston is played by the reaction taking place in the detonation wave.

Let us consider a plane detonation wave which has been established in an explosive (Figure 1). The wave front advances into the unconsumed explosive with a constant velocity D and it is followed by the reaction zone. If an observer is moving with the velocity D of such a front, the wave will appear to him/her as in Figure 1. Undetonated explosive flows into the shock front AA' with constant velocity $U_0 = -D$. Its pressure, temperature and density and internal energy per unit mass are P_1, T_1, ρ_1, E_1 at all points to the right of AA'. The wave front is considered to be a discontinuity in comparison to the changes occurring behind it. Therefore at AA' these values change to values P_2, T_2, ρ_2, E_2. These values change at some later stage.

The apparent velocity of the mass leaving the front is $-(D-U_p)$ where U_p is the particle velocity (mass velocity) in the zone between AA', BB', relative to the fixed coordinates.

If we consider a region of flow surrounded by a tube of unit sectional area and two planes, one just before the detonation front and one right after it, the mass flowing in must equal the mass flowing out (conservation of mass). The mass flowing in per unit time is $\rho_1 D\,dt$. The mass flowing out is $\rho_2(D-U_p)dt$. Therefore :

$$\rho_1 D = \rho_2(D-U_p) \qquad (1)$$

Furthermore the difference in momentum should be equal to the impulse of the net force. Thus:

$$\rho_1 Ddt\, D - \rho_1 Ddt(D-U_p) = (P_2-P_1)dt$$

or $$P_2-P_1 = \rho_1 DU_p \qquad (2)$$

P_1 is very small compared to the detonation pressure. Therefore it can be ignored and equation (2) can be written as :

$$P_2 = \rho_1 DU_p \qquad (3)$$

From equation (1), one can obtain:

$$U_p = (1-\rho_1/\rho_2)D \qquad (4)$$

According to Cook[3] U_p/D and ρ_1/ρ_2 are slowly variable functions of the original density. Thus:

$$U_p = f(\rho_1)D \qquad (5)$$

where $f(\rho_1) = 1 - \dfrac{\rho_1}{\rho_2}$

Therefore equation (3) can be written as:

$$P_2 = \rho_1 f(\rho_1)D^2 \qquad (6)$$

For most cases (explosives having a density between 0.9 -

1.4g/cc) it is sufficiently accurate to assume $f(\rho_1) = 4.0$. Under this approximation, the detonation pressure in atmospheres when the velocity of detonation is given in meters per second, is given by the following equation[8]:

$$P_2 = 0.00987\,\rho\,D^2/4 \qquad (7)$$

This is a relationship of great practical value. It allows the estimation of the detonation pressure when only the detonation velocity and the initial density are known. It is worth mentioning that the detonation velocity can be measured accurately in the laboratory.

Apart from equations (1) and (2) other equations are used in the theory of detonation. Many of these fall outside the area of interest of these notes. They are mentioned in the following to assist the reader in further studies.

The conservation of energy is expressed by the following equation:

$$E_2 - E_1 = \frac{1}{2}(P_2+P_1)(V_2-V_1) \qquad (8)$$

This is known as the Rankine-Hugoniot equation.

A fourth equation is the equation of state of the reaction products of the explosive.

The above four basic equations are not enough to calculate the five unknown quantities behind the detonation front (energy, density, detonation velocity, pressure and particle velocity). A fifth condition is necessary. This is the Chapman- Jouguet hypothesis stating that the detonation velocity equals the local sound speed plus the particle velocity at the detonation state. Therefore:

$$D = C + U_p \qquad (9)$$

Equations (1),(2),(8),(9) and the equation of state of the detonation products are essential for the calculation of the detonation parameters in the thermohydrodynamic codes.

1.4 The Detonation Head Model(3,4)

Practical explosives are used normally in the form of cylindrical charges. Cook's detonation head model illustrates the sequence of events taking place. Figure 2 shows the detonation head formation in a cylindrical unconfined charge. With strong priming a detonation wave travels out from the primer and along the charge. This is responsible for the promotion of the necessary exothermic detonation reactions within the explosive charge. At the back of the primer the high pressure gases expand into the surrounding air. As this expansion takes place it permits a release wave or a rarefaction wave to travel down the charge behind the detonation front. This always lags the detonation front for reasons which were explained earlier. In a similar manner at the sides of the charge immediately after the detonation wave the gases expand into the atmosphere. Again two release waves are travelling into the charge. The detonation front, rear release wave and side release waves define a region called the detonation head. The detonation head is a region associated with high pressure and high density. The shape of the detonation head depends on the geometry of the charge and changes as it travels out from the initiation source. This is due to the approximately constant relationship between the release wave velocity and the detonation velocity. Initially the shape is that of a section of a truncated cone with curved front and rear surfaces. Further away from the initiation the length of the detonation head grows so that it is controlled from the side release waves which meet on the axis of the charge forming a cone. It has been found (X ray radiography) that the length of the cone when the detonation is fully developed is approximately equal to the diameter of the charge. The density inside the detonation head is constant and approximately equal to 4/3 ρ_1 where ρ_1 is the initial density of the explosive. The distance from the initiator to the point where the full head is formed is approximately equal to 3 1/2 charge diameters for unconfined charges. As the explosive enters the detonation head it reacts. If it is in a granular form (e.g ANFO prills) the reaction starts at the surface and proceeds radially towards the centre of the prill. As it was mentioned in the previous the energy liberated supports the detonation. If the reaction is not completed inside the head the energy liberated is less than the maximum available and the detonation velocity is less than the maximum. This is what is normally known as non-ideal detonation. It is worth mentioning that non ideal detonations can be stable; indeed a great number of commercial explosives used by the mining industry today detonate at non ideal velocities at the diameters at which they are used.

The detonation velocity is the most important parameter of the detonating explosive. It is well known that the velocity of detonation is a constant characteristic of a particular explosive when the other parameters are kept constant. It was explained that the knowledge of the detonation velocity can lead to fairly accurate estimates of the detonation pressure which is of particular importance and cannot be measured directly. In the next chapter the parameters influencing the detonation velocity will be discussed.

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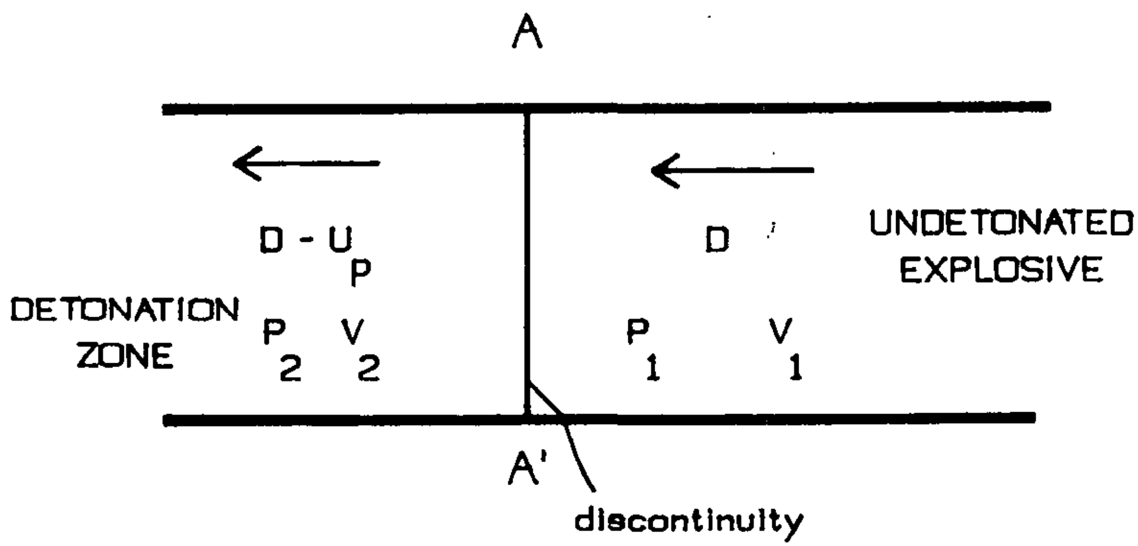
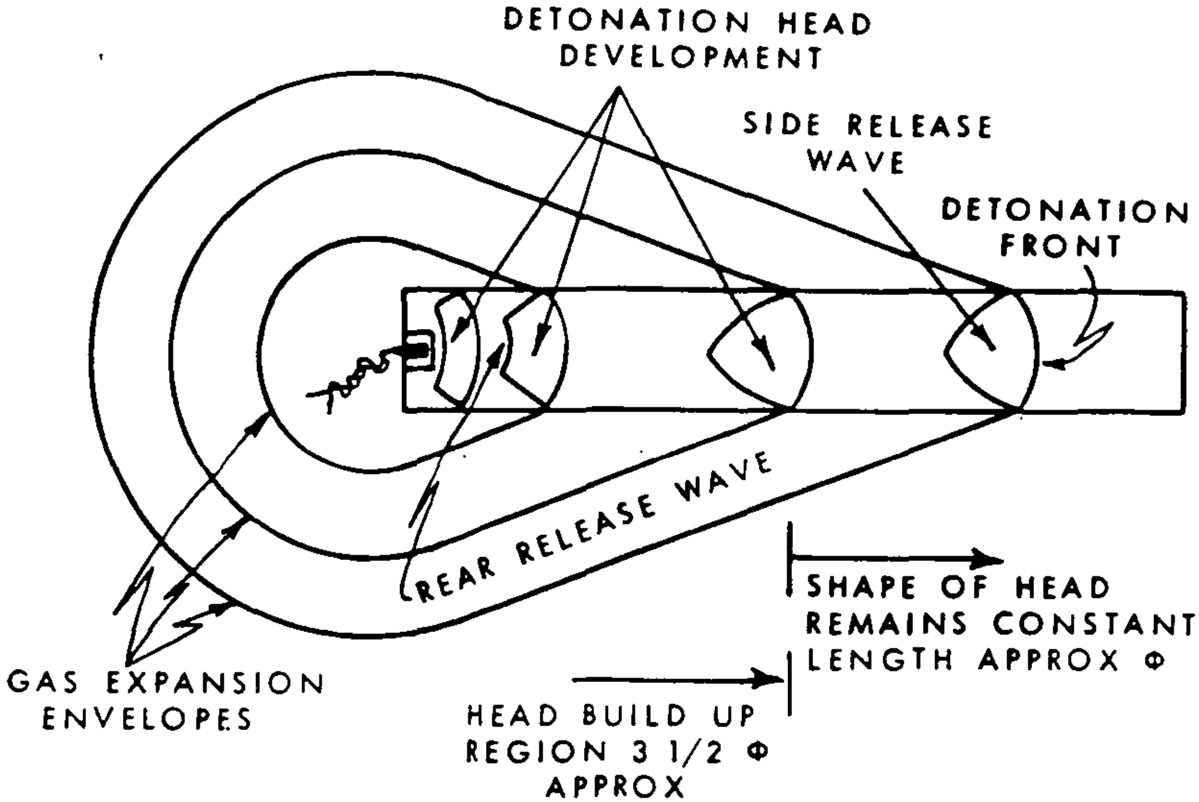


FIGURE 1: SECTIONAL DIAGRAM OF A DETONATION WAVE

Observer moves to right at wave velocity D.

The discontinuity is at rest



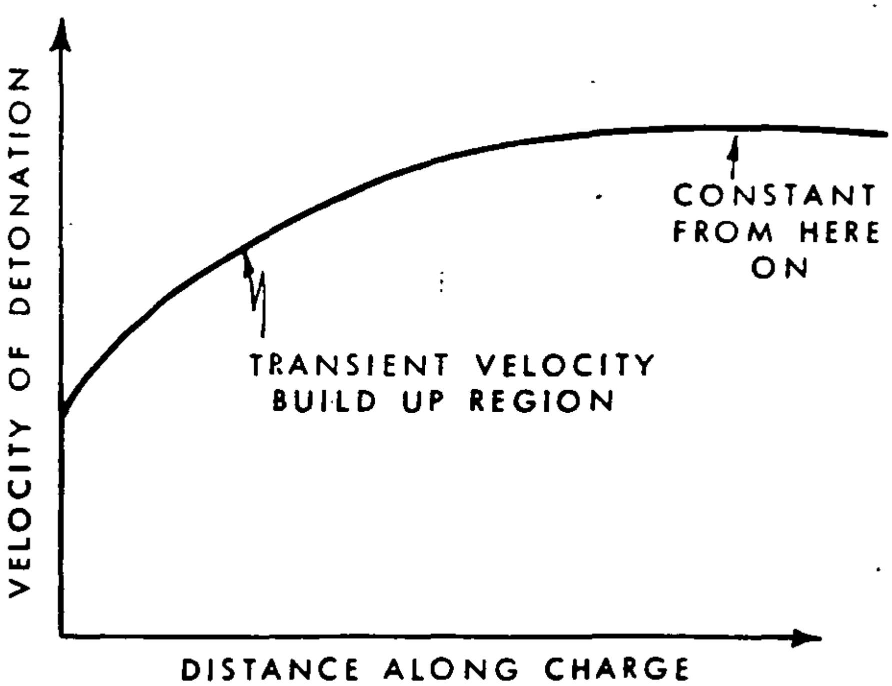


FIGURE 2: DETONATION HEAD FORMATION

(AFTER COOK AND BAUER)

EQUATIONS OF STATE

An equation of state is normally a pressure - volume - temperature relationship. Ideal gases have an equation of state expressed as:

$$PV = nRT$$

where P is the pressure
T is the temperature
n is the number of moles of gas
R is the universal gas constant and
V is the volume.

However real gases do not always behave according to the previous equation. It is obvious that a real gas cannot be cooled to zero volume. Under certain conditions gases turn into liquids or solids.

The origin of the deviations from ideality is the interaction between particles. Molecules excercise attractive forces when they are separated by some distance and repulsive forces when they are very close together.

Repulsive forces are short term interactions while attractive forces have a relatively long range. Figure 1 provides a plot of the compression factor $Z = PV/RT$ against pressure applied on the gas. One can obtain an indication of the imperfection at different pressures. For a perfect gas $Z = 1$ under all conditions. For a real gas the case is somewhat different. At very low pressures all gases behave almost ideally ($Z = 1$). At high pressures the repulsive forces dominate and $Z > 1$, while at moderate pressures $Z < 1$ due to the attractive forces. Obviously an equation of state for the detonation products has to reproduce this behaviour of real gases.

EQUATIONS OF STATE FOR DETONATION PRODUCTS.

The equations of state used for detonation calculations are of two types: those which do not treat chemistry explicitly and those which do. The latter contain individual equations of state for the component molecules and a mixture rule for combining them to give an equation of state for any composition. The composition of the detonation products is calculated by assuming chemical equilibrium.

At this point it is worth mentioning that much of the work involving the development of an equation of state has been employed in an inverted form. Experimental values are used to calibrate an assumed form of an equation of state. Attempts to develop a general, completely theoretical equation of state have failed to produce a good result.

The most common equations of state for detonation products are:

1. <u>The Abel Equation of State</u>.

The Abel equation of state is a form of the Van der Waal's equation of state. It can be expressed as:

$$P(V-\alpha) = nRT$$

where α is a constant.

It was found that this form did not produce acceptable results for many cases of condensed explosives. Cook[1] provided a modification expressing α as a function of the volume of the detonation products without considering their chemical composition. He showed that the empirical values of the covolume fall in a common $\alpha(V)$ curve.

2. The Becker - Kistiakowsky - Wilson Equation of State.

The most popular equation of state is the BKW equation. The equation has the following form:

$$\frac{PV}{RT} = 1 + xe^{\beta x}$$

where $$x = \frac{K}{V(T+\theta)^{\alpha}}$$

and $K = \kappa \Sigma k_i x_i$

with $\alpha, \beta, \kappa, \theta$ and k_i empirical constants. The constants k_i of each molecular species are the covolumes. For the mixture each k_i is multiplied by x_i, the mole fraction of species i, and summed to find the effective covolume.

According to a parameter study performed by the Los Alamos Laboratory, one may adjust the BKW parameters α, β, κ and θ and the covolumes of the detonation products. Cowan and Fickett[2] have shown that for a given α and β one may adjust κ to obtain the experimental velocity of detonation. The slope of the curve relating detonation velocity and density can be changed by changing β.

By using one explosive as a standard it was possible to obtain a set of parameters which can be used for a variety of explosives. BKW has been calibrated for RDX and TNT. The most common parameters used today are shown in Table 1[(3,4)]. It has been found that the RDX parameters result in realistic values of the detonation parameters (pressure and velocity of detonation). The parameters which have been developed based on TNT as the standard produce reliable results for very oxygen deficient systems which produce large amounts of carbon in the detonation products.

The best fit for RDX parameters should not be used in predictions of the detonation state parameters. This set was developed in order to have $(dP/dT)_V > 0$ at pressures of the order of 0.5 Mbar. It has been found that this set of parameters results in poorer predictions than the RDX set.

3. Other Equations of State

Other equations of state have been developed by Fickett and by Jacobs, Cowperthwaite and Zwisler[(4)].

These equations are similar and they are based on statistical mechanics. They use the Lennard-Jones potentials to describe the interactions between the molecules. The general form of the intermolecular potential energy is shown in Figure 2. When the molecules are squeezed together, the nuclear and electronic repulsions dominate the attractive forces. The repulsions increase steeply with decreasing separations. One approximation is the the hard sphere potential where it is assumed that the potential energy rises abruptly to infinity as soon as the particles come within some separation distance σ (collision diameter).

Normally the intermolecular potential is written as:

$$V = C_n/R^n - C_6/R^6$$

This is the Lennard-Jones (n,6) potential. Often the (12,6) potential is written in the form:

$$V = 4\varepsilon[(\sigma/R)^{12} - (\sigma/R)^6]$$

where ε is the depth of the potential well and
σ is the separation distance at which V=0.

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FIGURE 1: COMPRESSION FACTOR VS PRESSURE

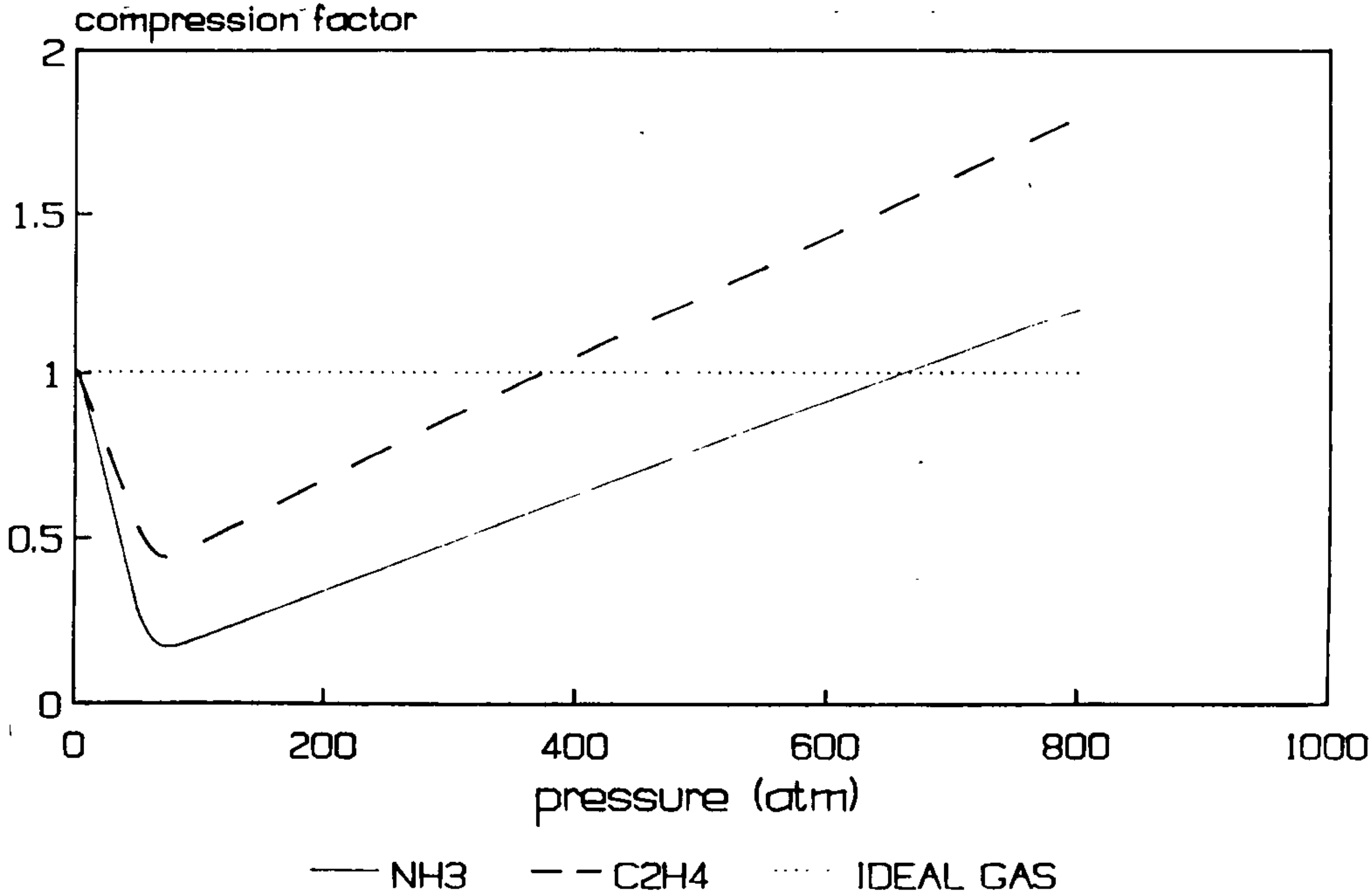

FIGURE 2: POTENTIAL ENERGY BETWEEN MOLECULES

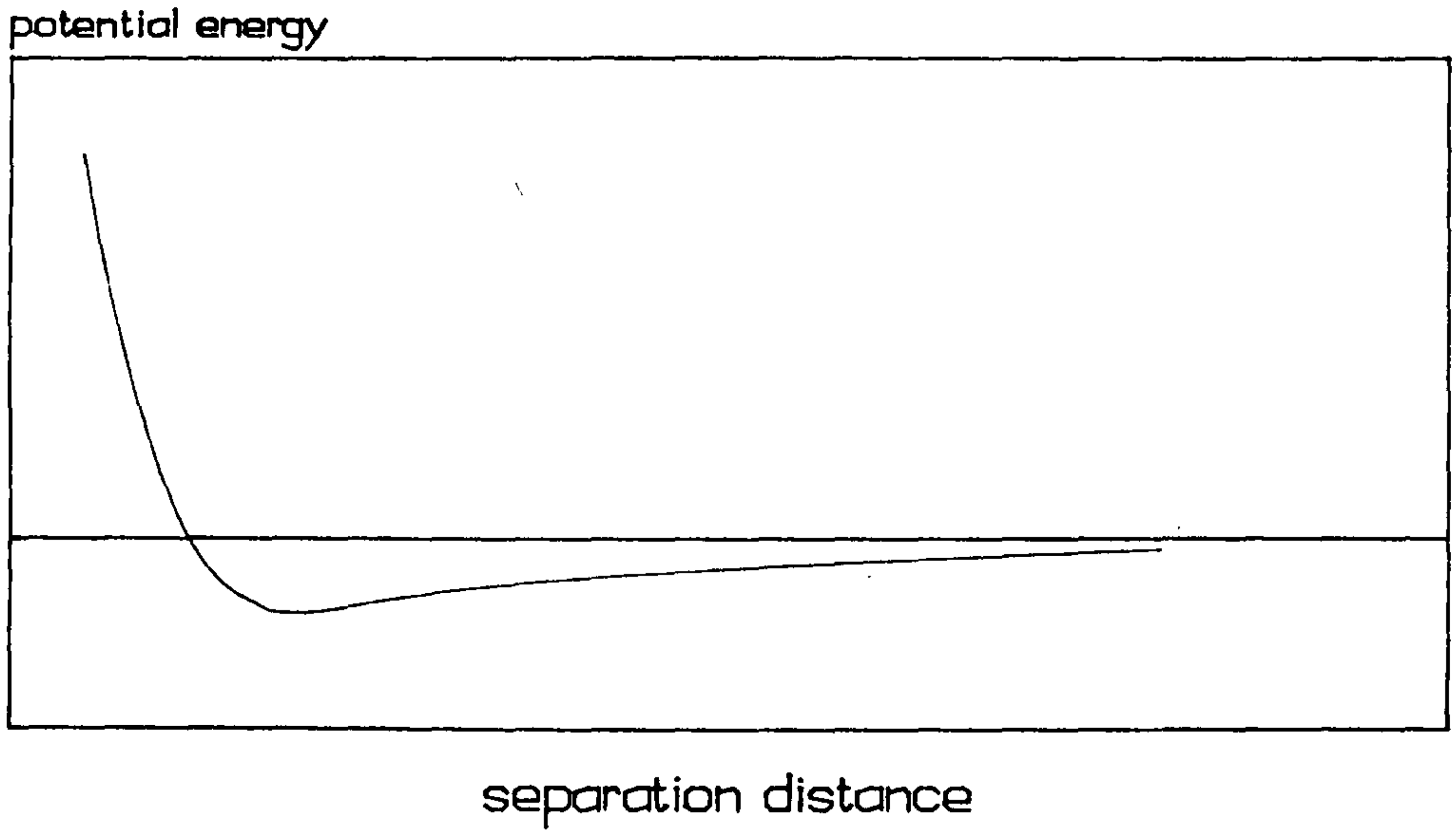

TABLE 1

COMMONLY USED BKW PARAMETERS FOR HIGH DENSITY EXPLOSIVES

NO.	PARAMETER SET	β	κ	α	θ
1	Fitting RDX	0.181	14.15	0.54	400
2	Fitting TNT	0.09585	12.685	0.50	400
3	Best fit for RDX with $(\partial P/\partial T)_V > 0$	0.16	10.91	0.50	400
4	Default parameters	0.10	11.85	0.50	400

CHAPTER 3

EXPLOSIVE PROPERTIES

3.1 Introduction

A variety of factors influence the explosives selection process. This chapter discusses the most important of them and the parameters which influence them.

3.2 Velocity of Detonation

The velocity of detonation is the velocity at which the detonation wave travels through an explosive charge. The detonation wave travels at speeds above the normal sound speed of the unreacted material. Typical detonation velocities for commercial explosives range from 2500 to 7000 m/sec. The detonation velocity is the most important property of the explosive. It can be measured easily and accurately and it can be used for the calculation of the detonation and borehole pressures which are of importance in explosive applications. The velocity of detonation of a particular explosive depends on factors such as charge diameter, confinement, density and particle size.

3.1.1 *The effect of charge Diameter*

Let us consider a typical velocity of detonation - diameter curve as shown in Figure 1[2]. If the diameter is too small the explosive fails to detonate. At some minimum diameter stable detonation occurs. This minimum diameter is called the critical diameter of the explosive.

As the charge diameter is increased the detonation velocity is increased as well. However when a certain diameter is reached, further increase in diameter does not result in an increase of the detonation velocity. At this point a maximum detonation velocity of the explosive is reached. This velocity is called the ideal detonation velocity of the explosive and is the value predicted by thermohydrodynamic codes.

The detonation head model as developed by Cook[1] can be useful in explaining the shape of the observed detonation velocity - diameter curves. Figure 1 illustrates the length of the established detonation heads in charges of various diameters and indicates what happens when a solid particle of explosive enters the detonation head. For the small diameters, the degree of reaction is small and the energy liberated is not enough to support a detonation. As the diameter is increased the detonation head length is increased and for the same size of particle the degree of reaction increases. At the critical diameter the degree of reaction is sufficient to support stable detonation. If the diameter is increased further a larger amount of explosive reacts in the detonation head. When the ideal detonation occurs, the full amount of explosive reacts in the detonation head.

3.2.2 *Effect of Confinement*

The effect of confinement is to lower the rate of expansion of the gases off the side of the charge[2]. This in turn slows down the rate at which the lateral rarefaction travels into the reaction region. As a result it takes longer for the side release waves to meet on the charge axis. The length of the detonation head is thus increased. This is shown in Figure 2[1], where the development of the detonation head is outlined for both the confined and the unconfined cases. Therefore, if the explosive was not reacting fully at a particular charge diameter, the effect of confinement would be to increase the degree of reaction and consequently the detonation velocity at this diameter. Similarly, confinement will reduce the critical charge diameter (Figure 3)[2].

However confinement cannot be quantified. Steel, glass, various kinds of rock and soil will produce a different effect. For this reason most of the tests are done with the explosive charge unconfined.

3.2.3 *Effect of Particle Size*

If the size of the explosive particles is reduced at a given charge diameter in the non ideal velocity region, the degree of reaction is enhanced because of the increase of the surface area. Furthermore since the grains are smaller, they are consumed faster in the detonation head. As a result the critical diameter is decreased and the explosive reaches ideal detonation at a smaller diameter (Figure 4)[2].

3.2.4 *Effect of Density*

If the density is increased, the specific energy is increased; as a result the ideal detonation velocity is increased. It has been found that the detonation velocity and the density are related linearly. Figure 5[3] shows the detonation velocity density relationship for various explosives.

However if the density is increased beyond a critical point, steady state detonation is not possible. The phenomenon is called dead packing and a qualitative explanation can be given by the fact that the volume of the entrapped air is insufficient to provide enough hot spots for the reaction to proceed[2].

The relationship between critical diameter and density is shown in Figure 6[5]. It is obvious that apart from the density in which the material is dead packed there is a critical density below which the explosive will not shoot.

3.2.5 *Effect of Temperature*

The initial temperature of the explosive has a small influence on the velocity of detonation at diameters well above the critical. However the critical diameter is dependant on the initial temperature. Figure 7 shows the effect of the temperature on the critical diameter powdered TNT[4].

In the case of commercial liquid explosives the effect is more pronounced. Figure 8 shows the effect of low temperatures on the critical diameter of typical slurry explosives[5]. The effect on solid explosives is almost negligible.

3.2.6 *Effect of Water*

Generally dynamites are not affected by the presence of water inside boreholes. Ammonium nitrate mixed with fuel oil has no water resistance. The product absorbs water and soon becomes desensitized. Generally performance drops drastically as the weight of water in the composition is increased.

3.3 Detonation Pressure

The detonation pressure is a very important parameter. It is an indicator of the ability of the explosive to produce the desired fragmentation in the rock. However, due to its high magnitude the detonation pressure cannot be measured directly. For this reason the experimental determination is difficult.

The detonation pressure is related to the square of the detonation velocity. Parameters which influence the detonation velocity have a very significant effect on the detonation pressure.

3.4 Detonation Temperature

The detonation temperature is the parameter about which the least amount of information is available[6]. The detonation temperature is measured from the brightness of the detonation front as it is observed by a sensor. However it is not known how much radiation is absorbed from the partially decomposed material between the sensor and the front. Furthermore, any gas bubbles in the material will flash brightly when they are impacted by the detonation wave. This, obviously, will affect the measurement.

3.5 Fumes

It must be assumed that in all cases explosive fumes are to some degree toxic. Excess oxygen causes the formation of nitrogen oxides while oxygen deficiency causes the formation of carbon monoxide.

In the United States the fumes of any explosive are classified after detonating the explosive in a Bichel bomb and analyzing its fumes. The following classes exist[7]:

A. Permitted explosives (USBM)

Fume class	Toxic Gas ft^3/lb	Toxic Gas l/kg
A	< 1.25	< 78
B	1.25 - 2.50	78 - 156
C	2.50 - 3.75	156 - 234

B. Rock blasting explosives

Fume class	Toxic Gas ft^3/lb	Toxic Gas l/kg
1	< 0.16	10
2	0.16 - 0.33	10 - 21
3	0.33 - 0.67	21 - 42

Canada uses the same standards. However explosives of class 2 or 3 cannot be used in underground mines unless special application has been made to and permission is received from the authorities (EMR).

It is worth mentioning here that the relative toxicity of the fumes is important and this is not shown in the above tables. NO_2 is much more toxic than CO (about 6 times as much)[8].

It has been found that the fumes depend on[2]:

1. The oxygen balance
2. Marginal priming
3. Water attack
4. Critical diameter
5. Gaps in loading
6. Deflagrations.

3.6 Energy of Explosives

Explosives are substances that rapidly liberate their chemical energy as heat to form gaseous and solid decomposition products at high temperature and pressure. The hot and dense detonation products produce shock waves in the surrounding medium and upon expansion impart kinetic energy to the surrounding medium. The energy released in the detonation process is given by the following formula:

$$Q = \Delta H_f(\text{products}) - \Delta H_f(\text{reactants})$$

where ΔH_f is the heat of formation.

The energy per unit weight is called the weight strength of the explosive.

The energy per unit volume is called the bulk strength of the explosive.

Sometimes it is useful to express the weight and the bulk strengths as relative values obtained by dividing the strength (weight or bulk) to the corresponding strength of a standard explosive. The commercial industry normally uses AN/FO as the standard explosive.

3.7 Shelf Life

The shelf life of an explosive determines the maximum time period the explosive can be in storage. Various explosives age and their use is unsafe or they cannot be detonated reliably.

3.8 Pressure Desensitization

Commercial explosives can be susceptible to hydrostatic heads. Hydrostatic heads can compress the explosive to high densities and "dead packing" can result.

3.9 Measurement of the Detonation Properties

3.9.1 *Detonation Velocity*

There are various methods of measuring detonation velocities. These are outlined in the following:

i The continuous probe method.

The system consists of the explosive charge, along the central axis of which a uniform resistance probe is inserted, a constant current source, a triggering source and an oscilloscope.

The resistance probe consists of a resistance wire inserted into a small diameter brass tube. The resistance wire is a nichrome wire having an accurately known linear resistance.

The oscilloscope is connected in parallel to both the current source and the probe (Figure 9)[5]. At detonation the wire resistance probe is consumed. However the circuit remains closed due to the fact that the detonation wave is sufficiently ionized. The circuit follows Ohm's law. Therefore, since current is constant, the voltage change with time shown on the oscilloscope, is proportional to the resistance. Knowing the full voltage drop across the probe and the length of the probe, the voltage drop can be converted to distance along the charge. Therefore the velocity of detonation can be calculated by interpreting the voltage drop - time record provided by the oscilloscope.

ii. Start-stop method

Two probes are placed at a known distance apart in the explosive. Each probe consists of two wires placed in close proximity. When the detonation wave contacts each probe it shortens the circuit by bringing the two wires in contact. By measuring the signals obtained by either a counter or an oscilloscope one can measure the detonation velocity.

iii. Streak camera method

The method is shown in Figure 10[9]. The streak camera uses a mirror which rotates at the centre of the drum. The film is placed on the drum. The field of view of the camera lens is masked except for a narrow slit. The charge is aligned so that its axis is parallel to the slit of the camera. The light generated by the detonation front enters through the slit and after being reflected on the rotating mirror, leaves a mark on the film. Thus the streak camera trace is essentially a time distance record. The slope of the trace made by the luminous wave provides the velocity of detonation. A typical streak camera record is shown in Figure 11[10].

iv. D'Autriche Method

This is the least sophisticated method. It is outlined in Figure 12[9]. The method uses a detonating cord both ends of which are inserted in the explosive at a known distance apart. A metal witness plate is placed close to the middle of the detonating cord. The detonation wave in the charge initiates the detonating cord at both ends. When the detonation waves travelling in opposite directions in the detonating cord collide, they leave a dent in the witness plate. This helps to find the position in the detonating cord at which the collision took place. Thus, the distance, and therefore the time, each wave travelled in the detonating cord can be found. The difference in the times the two waves travelled in the cord provides the time it took the detonation wave in the test charge to travel the distance l.

3.9.2 *Detonation Pressure*

The measurement of the detonation pressure is normally based on photographic techniques. These techniques require a streak camera and accurate experiments (aquarium technique). In the aquarium technique, a transparent liquid serves as a pressure gauge for measuring transient pressures. The transparent liquid has to be selected in such a way that the reflected wave at the gauge-liquid interface is either a weak shock or a very weak rarefaction. The technique, as described by Cook[8] consists of the following two stages:

i. Initially the Hugoniot of the liquid which serves as a gauge is determined. The experimental set up is shown in Figure 13. The method consists of the simultaneous measurement of the shock velocity at the free surface and the free surface velocity as the shock emerges from the transparent medium. Observations of the shock velocity and the free surface velocity are made by using a streak camera. By changing the height (h) of the liquid inside the container, one changes the shock velocity and the free surface velocity. By assuming that the particle velocity of the liquid at the interface is half of the free surface velocity the relationship between shock velocity and the particle velocity in the liquid (Hugoniot) is obtained.

ii. The experimental set up for the second part of the technique is shown in Figure 14. In this experiment, the velocity of detonation in the explosive charge and the initial transmitted shock velocity in the liquid are measured. From the transmitted shock velocity in the liquid and the known Hugoniot of the liquid, the initial pressure in the liquid can be calculated. The corresponding pressure in the detonation head is calculated by using the following relationship:

$$P_d = P_{il}[(\rho U_s)_{il} + \rho_{1e} U_{se}]/(2(\rho U_s)_{il})$$

where

P_d is the detonation velocity

ρ_{ie} is the initial density of the explosive

U_{se} is the detonation velocity

$(\rho U_s)_{il}$ is the initial impedance of the liquid and

P_{il} is the initial pressure in the liquid.

The initial pressure in the liquid is calculated by the well known relationship

$$P_{il} = \rho_1 U_{s1} U_{p1}$$

where P_{il} is the pressure in the liquid

U_{s1} is the shock velocity

U_{p1} is the particle velocity and

ρ_1 is the initial density of the liquid.

Because of the difficulty in measuring detonation pressures it is often necessary to calculate the detonation pressure from the detonation velocity by using the approximate formula:

$$P = \frac{\rho D^2}{4}$$

where P is the detonation pressure

ρ is the initial density of the explosive and

D is the measured detonation velocity.

3.10 References

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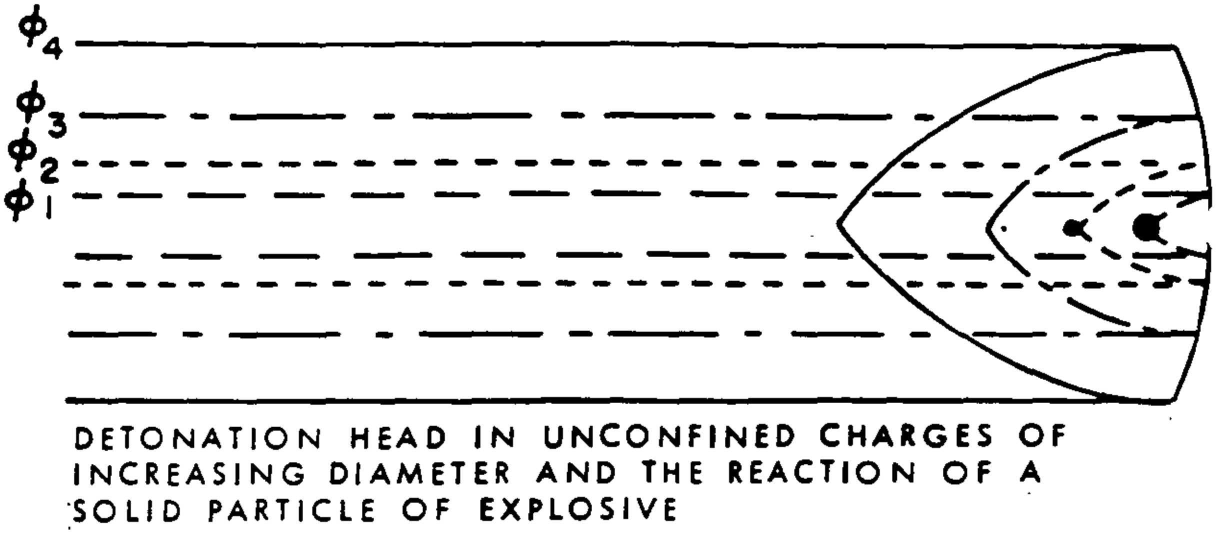


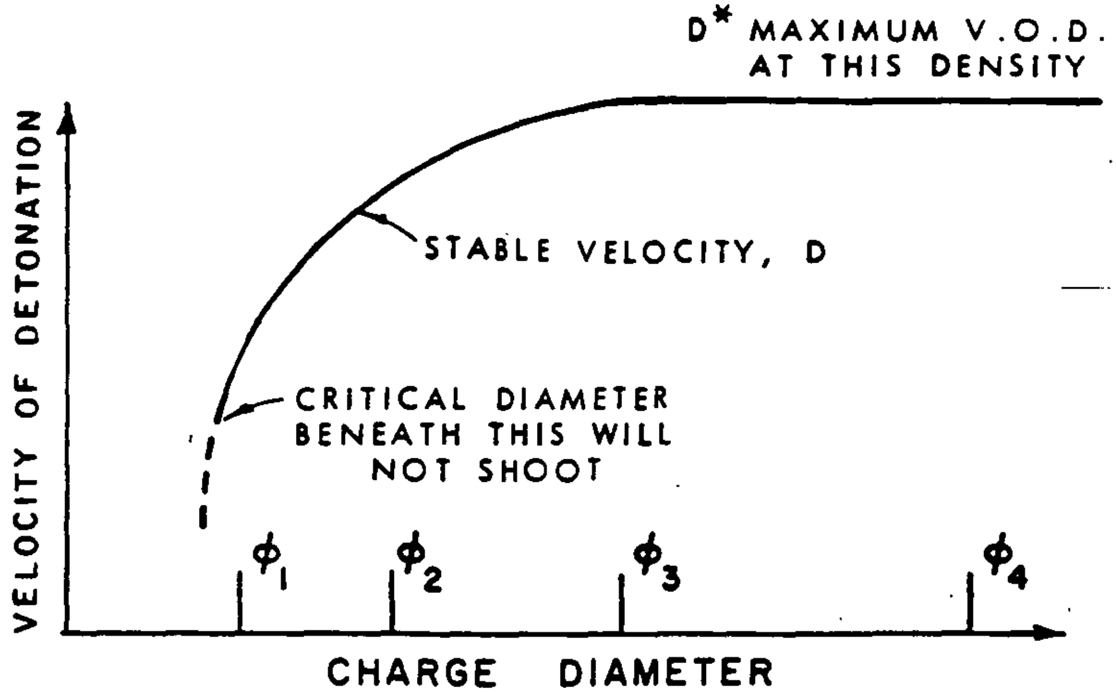


FIGURE 1: TYPICAL VELOCITY OF DETONATION CHARGE DIAMETER CURVE FOR A GRANULAR EXPLOSIVE (AFTER BAUER)

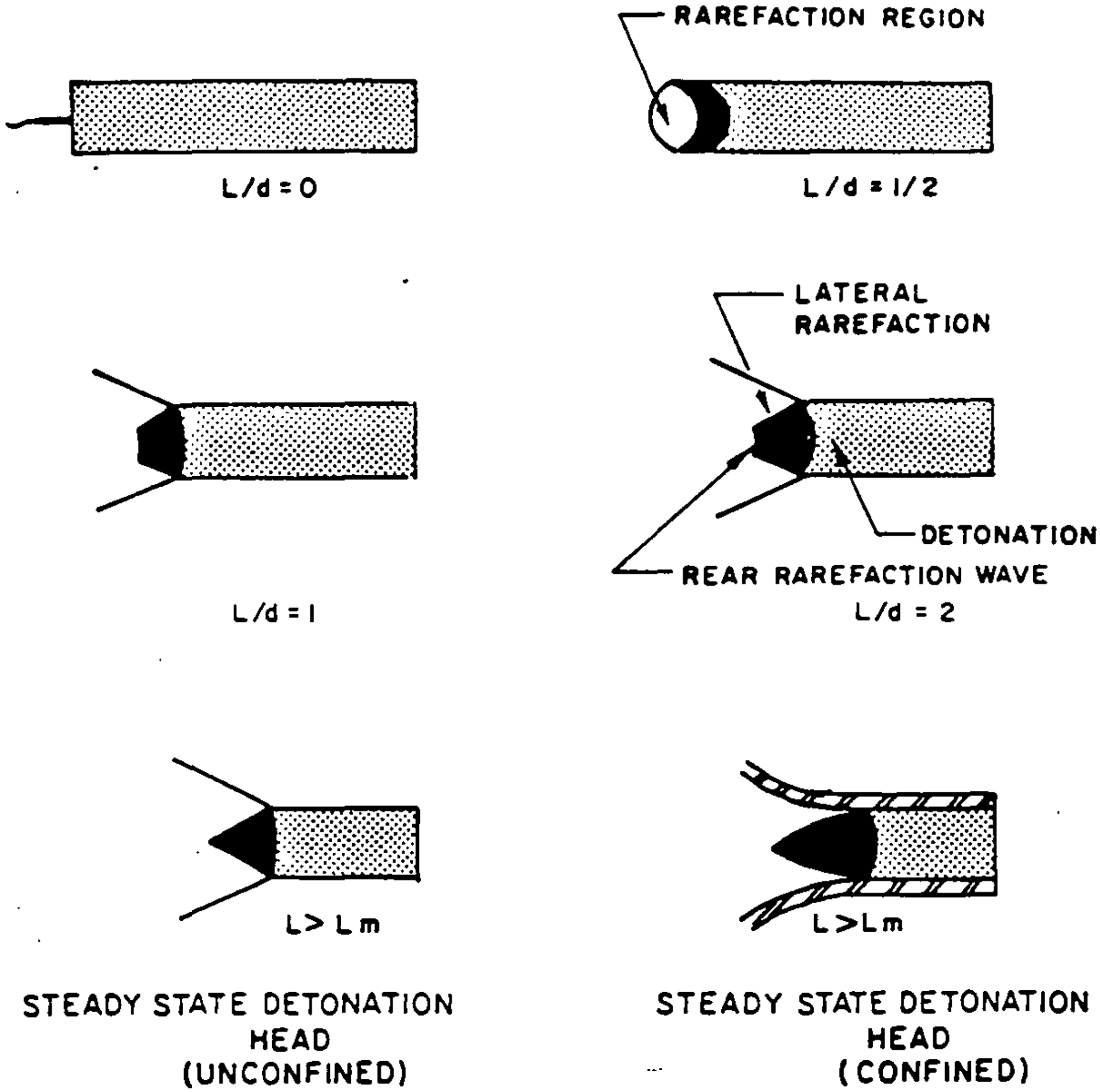
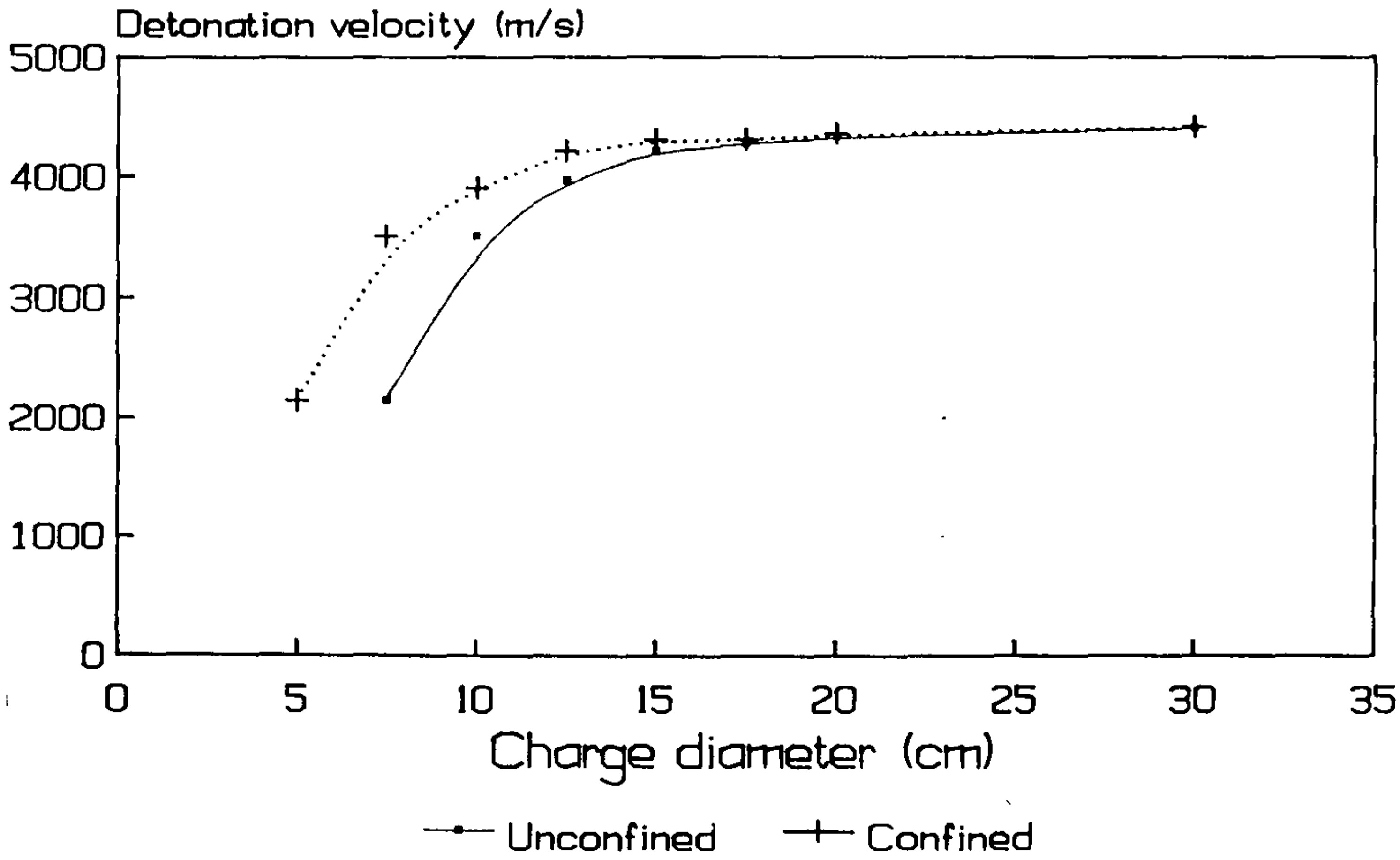


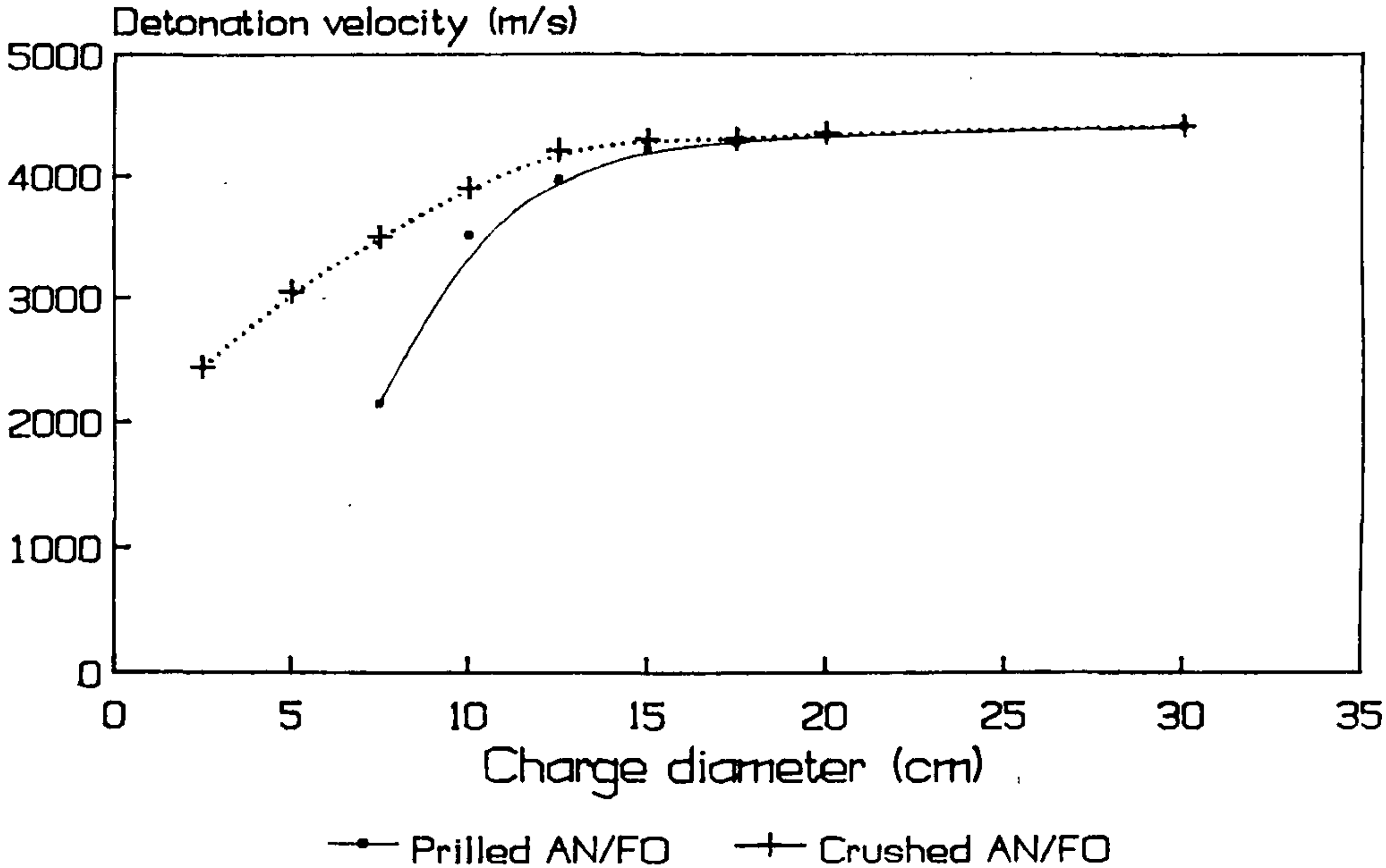
FIGURE 2: DEVELOPMENT OF THE DETONATION HEAD
(AFTER COOK, 1958)

FIGURE 3: VOD - CHARGE DIAMETER CURVES FOR CONFINED AND UNCONFINED ANFO



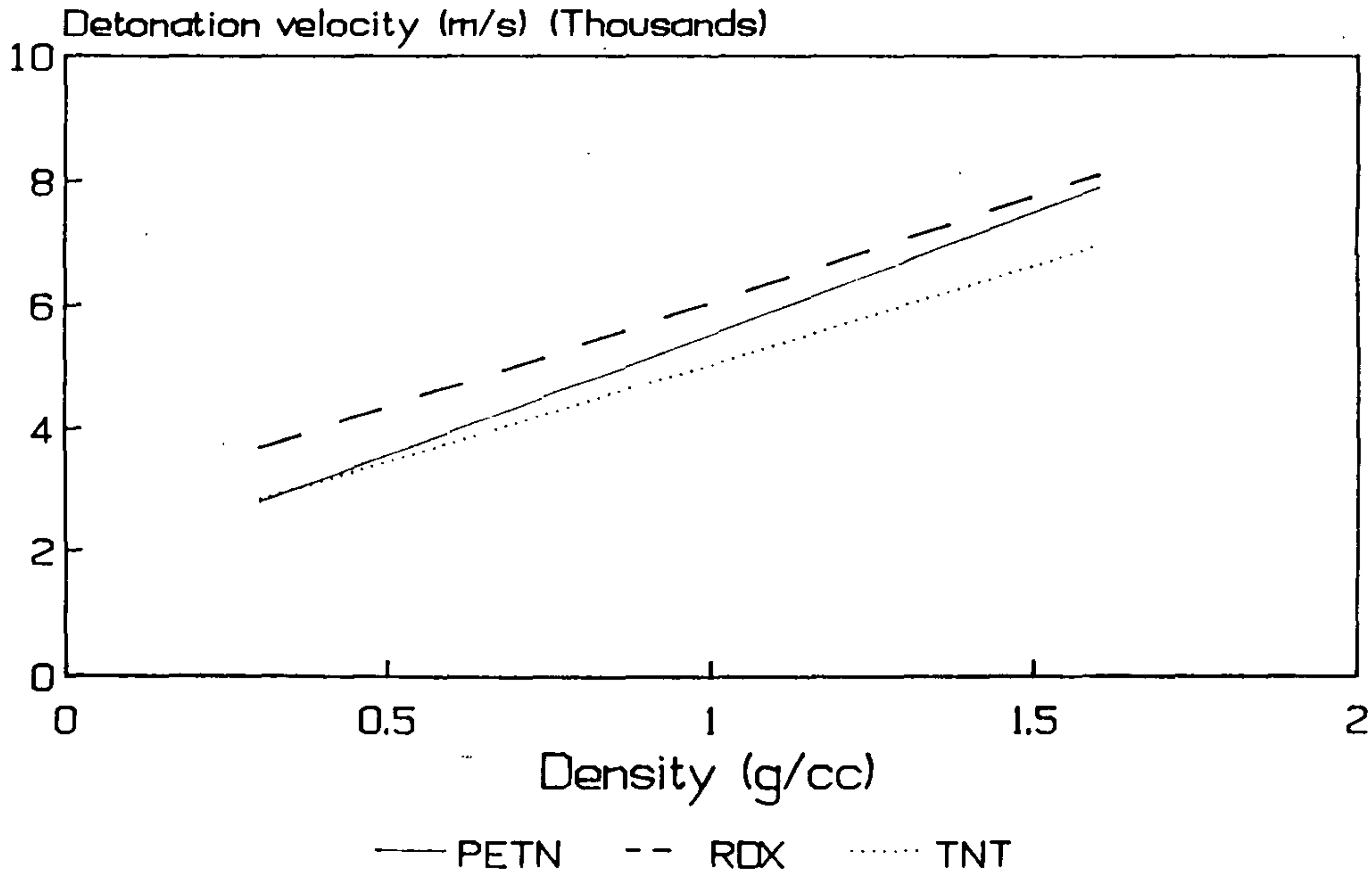
density = 0.85g/cc

FIGURE 4: EFFECT OF THE PARTICLE SIZE ON THE VELOCITY - DIAMETER CURVE OF AN/FO



density = 0.85g/cc

FIGURE 5: DETONATION VELOCITY - DENSITY RELATIONSHIPS



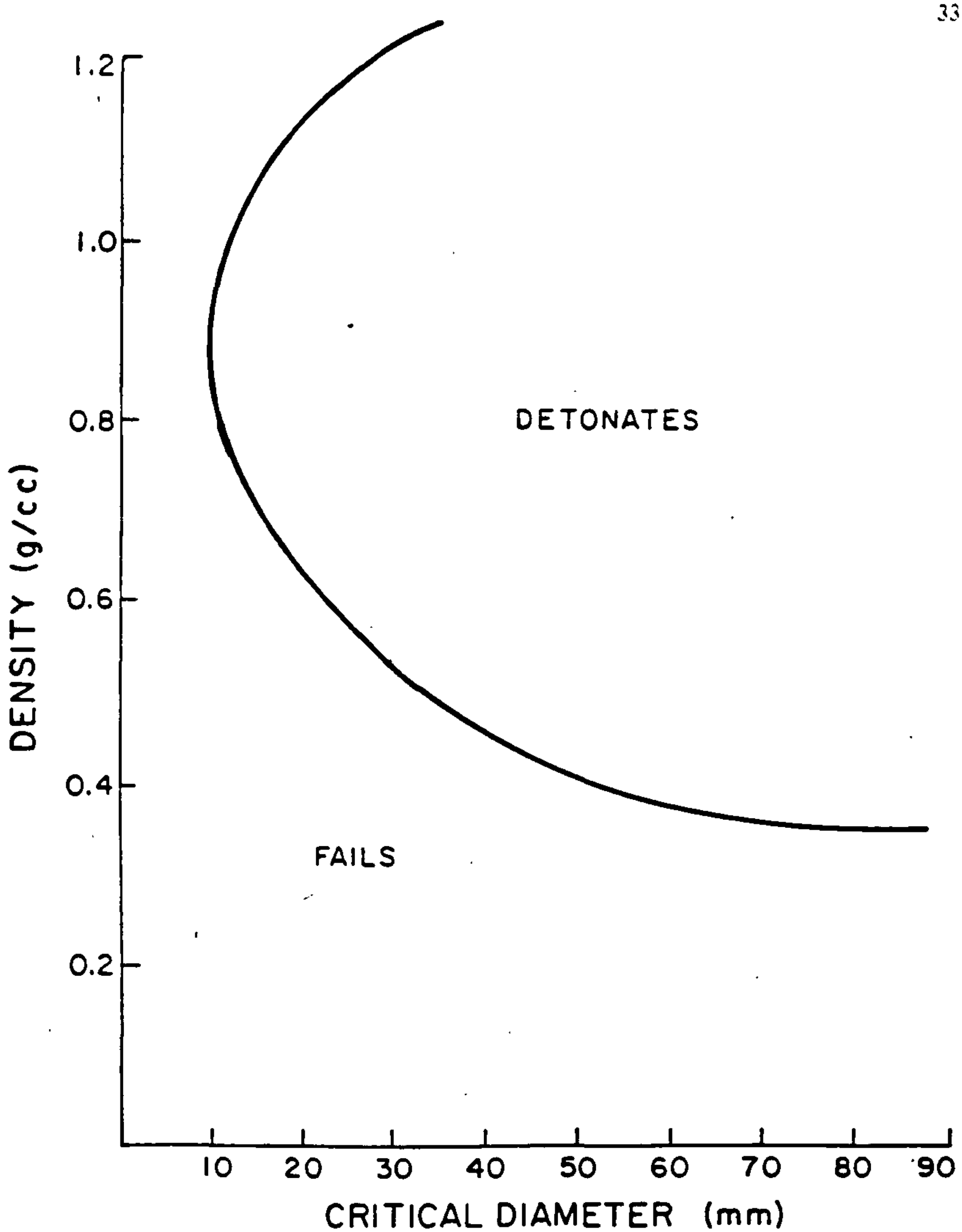


FIGURE 6: EFFECT OF THE DENSITY OF A TYPICAL EMULSION ON THE UNCONFINED CRITICAL DIAMETER

FIGURE 7: EFFECT OF TEMPERATURE ON THE CRITICAL DIAMETER OF TNT

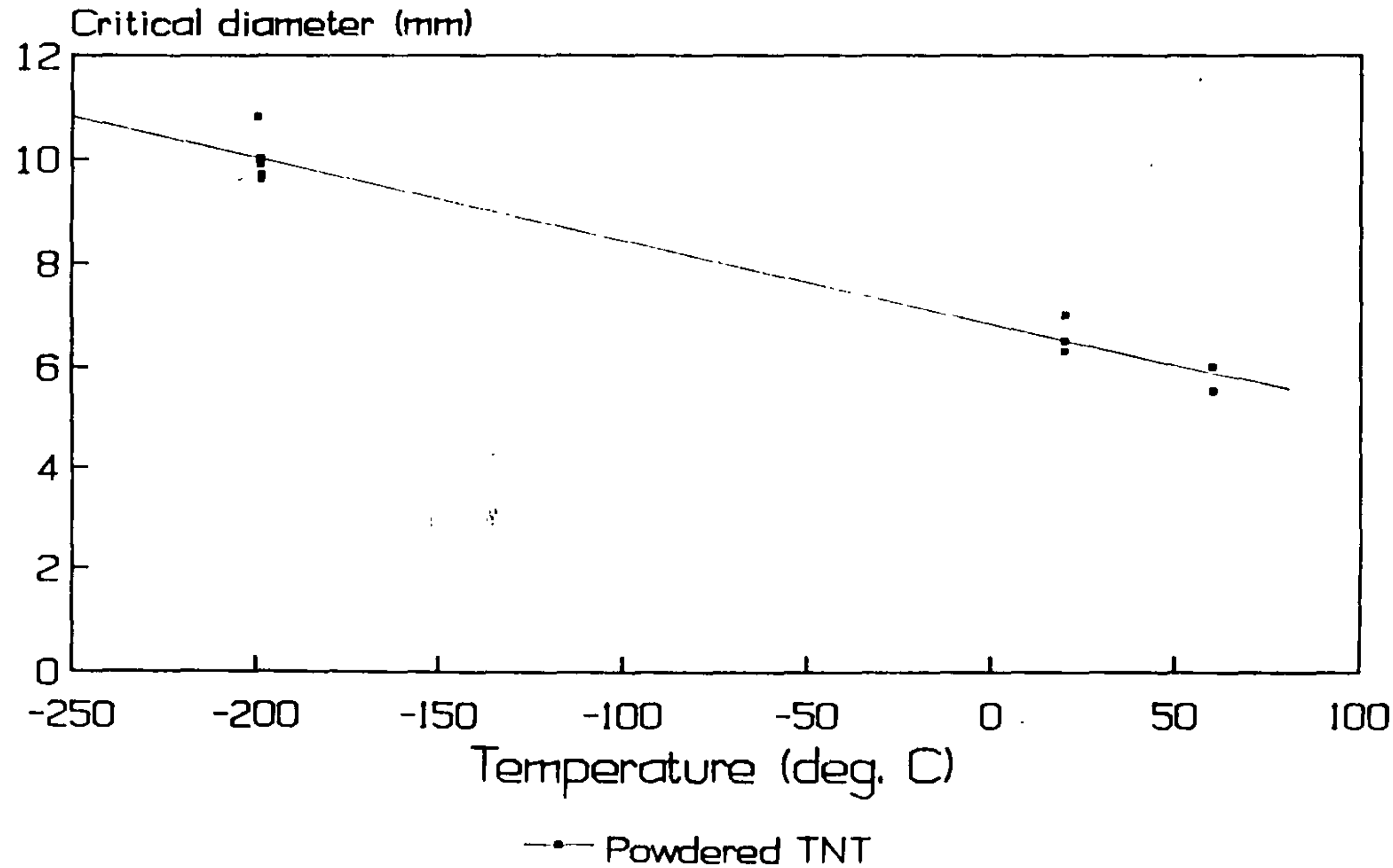
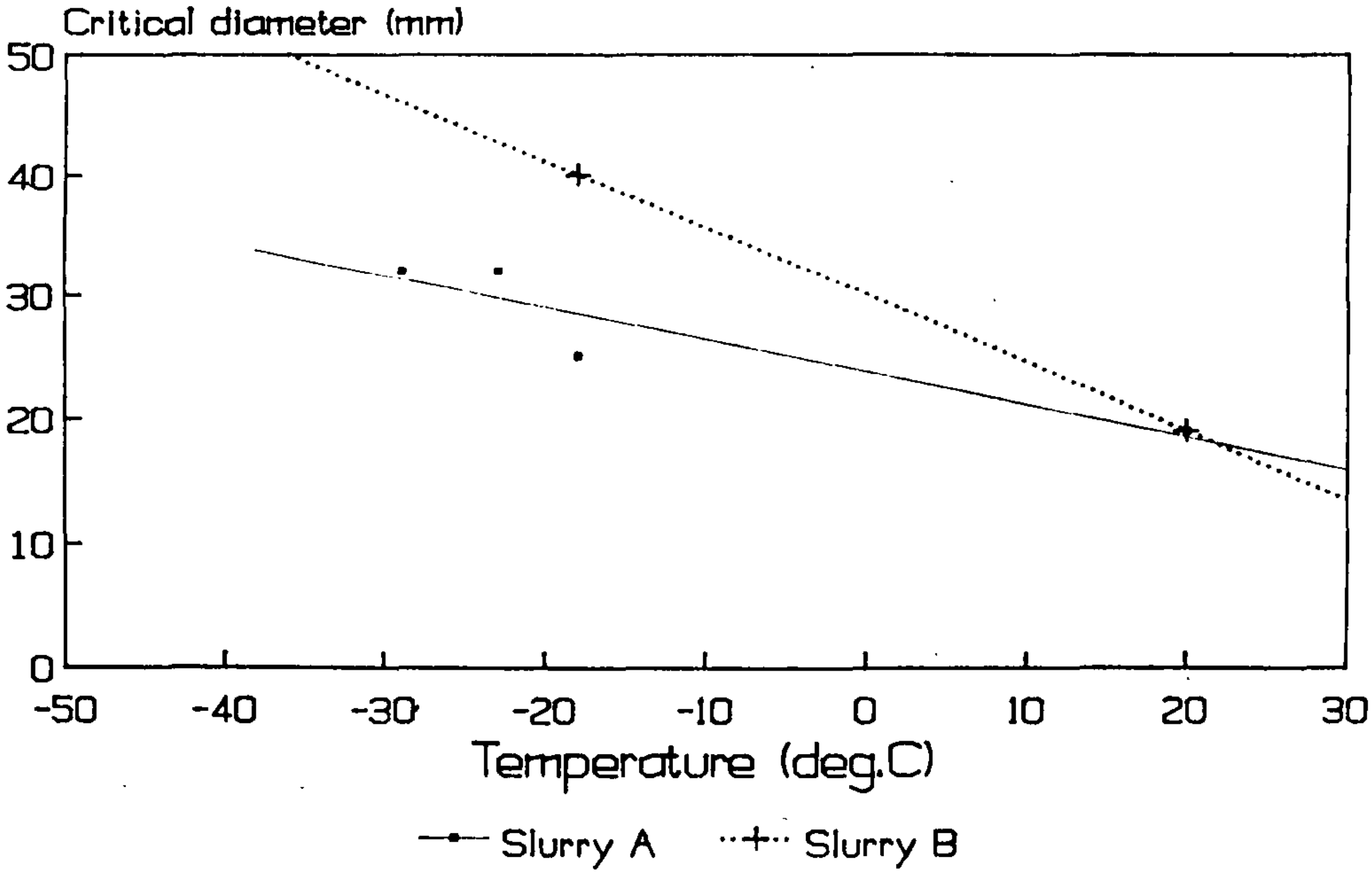


FIGURE 8: EFFECT OF TEMPERATURE ON THE CRITICAL DIAMETER OF SLURRIES



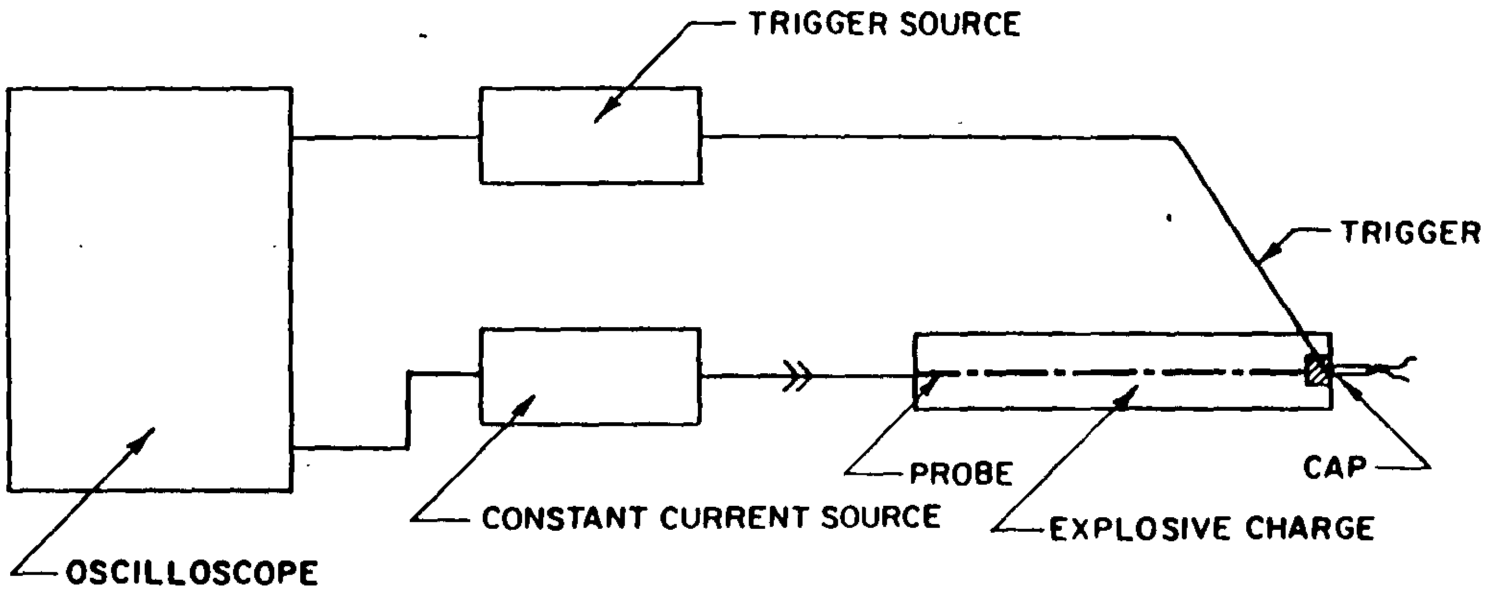


FIGURE 9: SCHEMATIC REPRESENTATION OF THE CONTINUOUS VELOCITY SYSTEM FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION

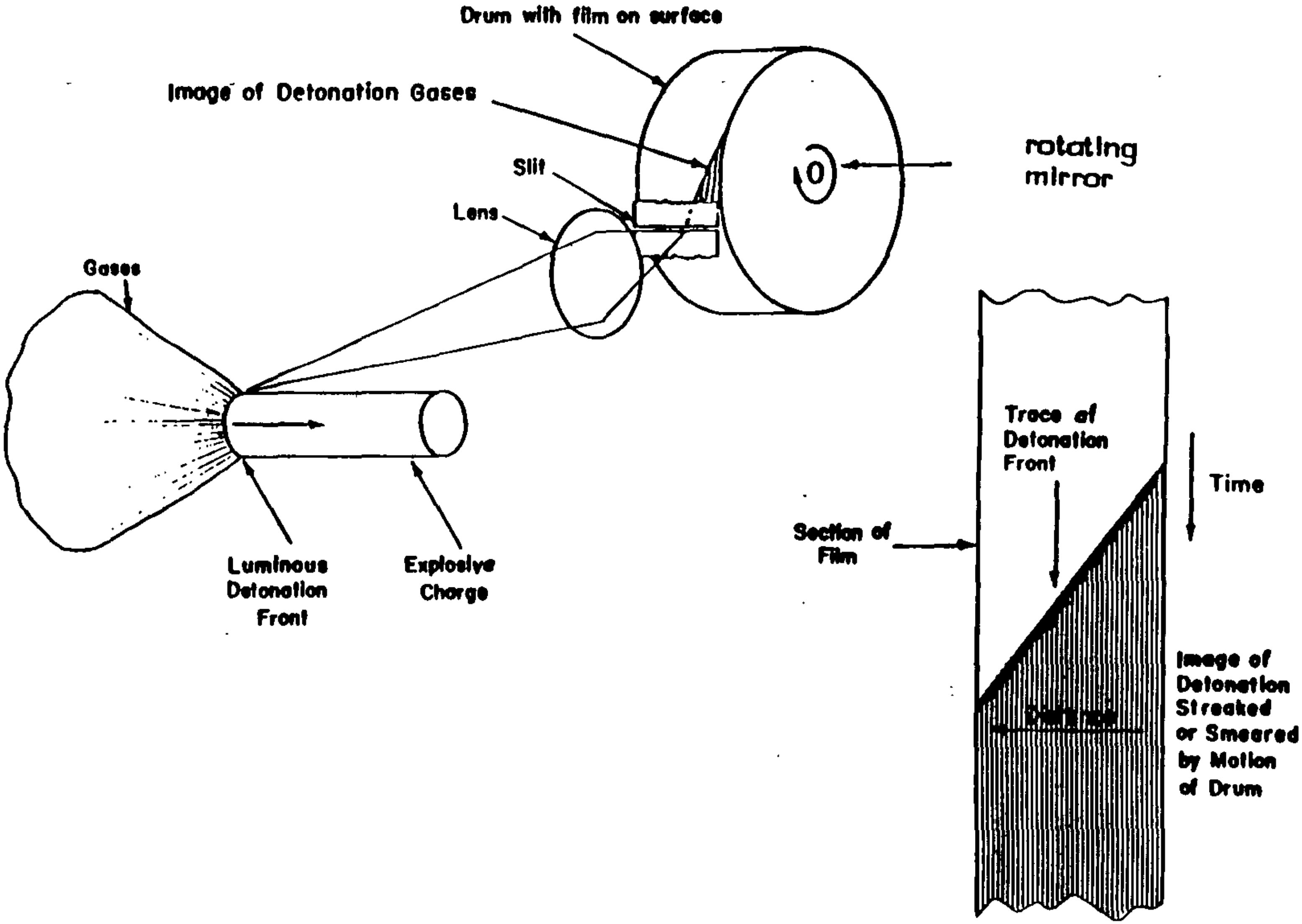


FIGURE 10: MEASUREMENT OF THE VELOCITY OF DETONATION
BY USING A STREAK CAMERA (ref. 9)

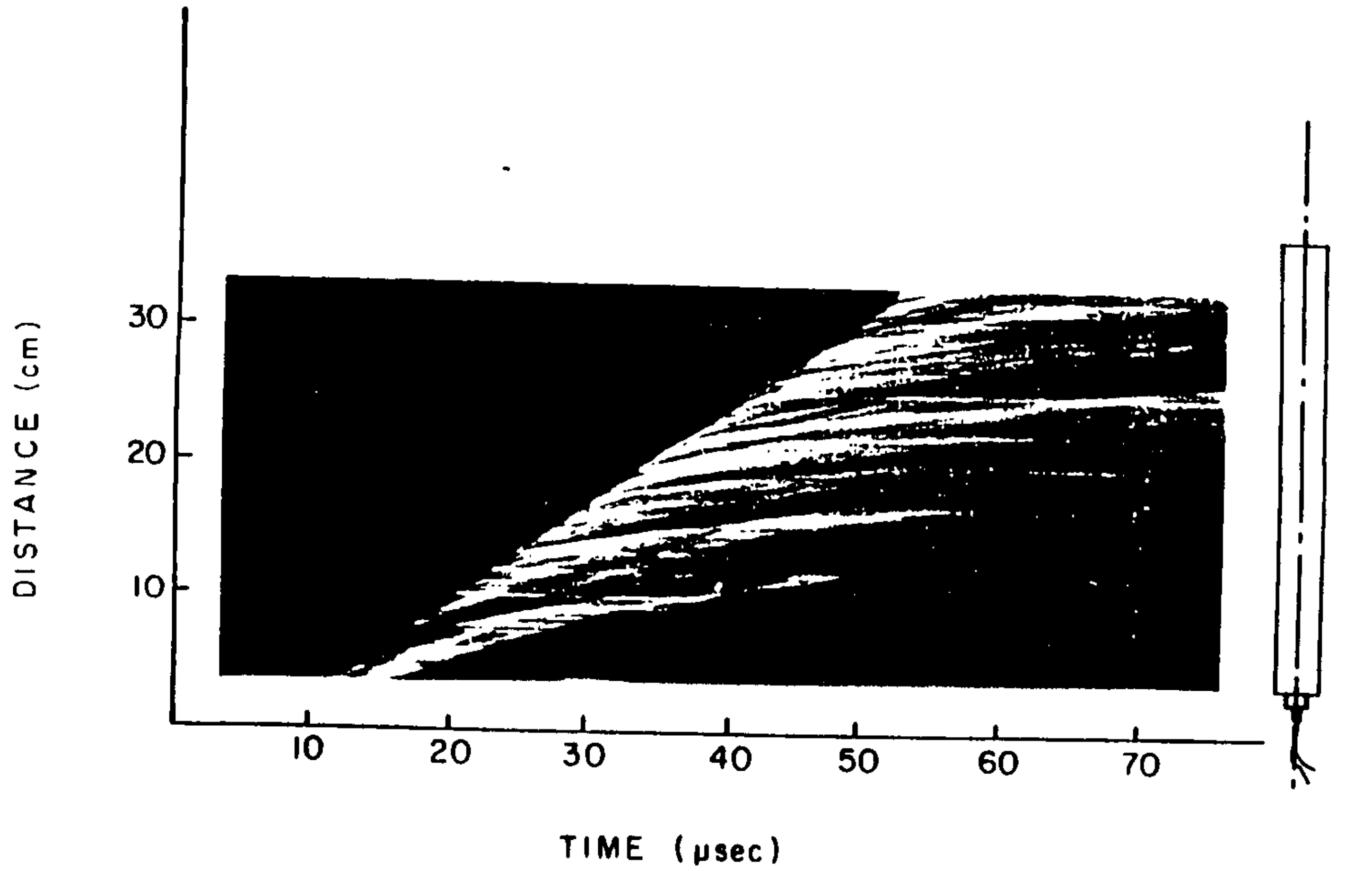


FIGURE 11: TYPICAL STREAK CAMERA RECORD FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION OF PENTOLITE

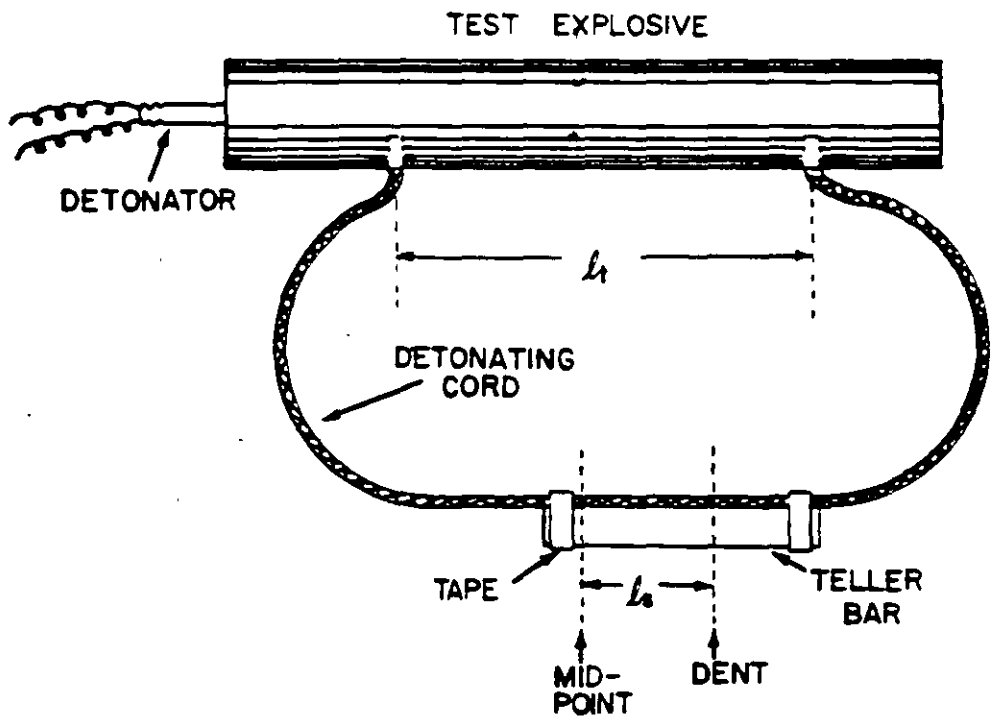


FIGURE 12: D'AUTRICHE METHOD FOR THE MEASUREMENT OF THE VELOCITY OF DETONATION

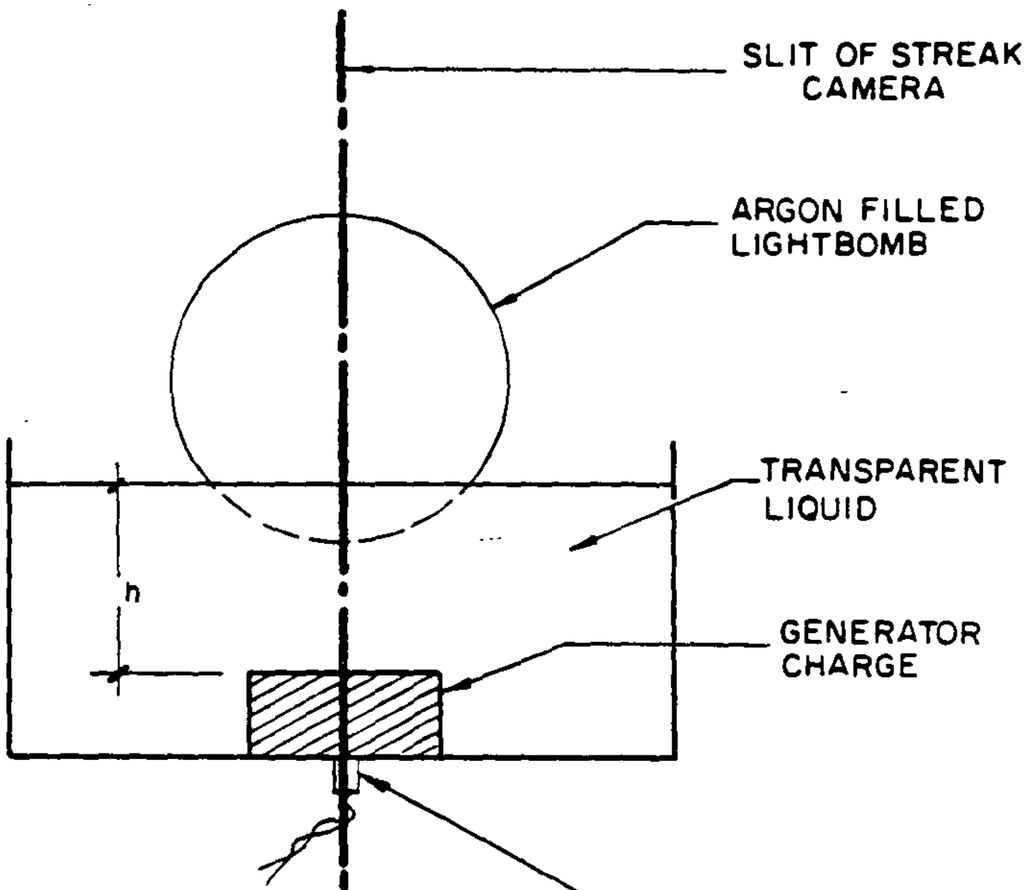


FIGURE 13: EXPERIMENTAL SET UP FOR DETERMINING THE HUGONIOT OF THE TRANSPARENT LIQUID

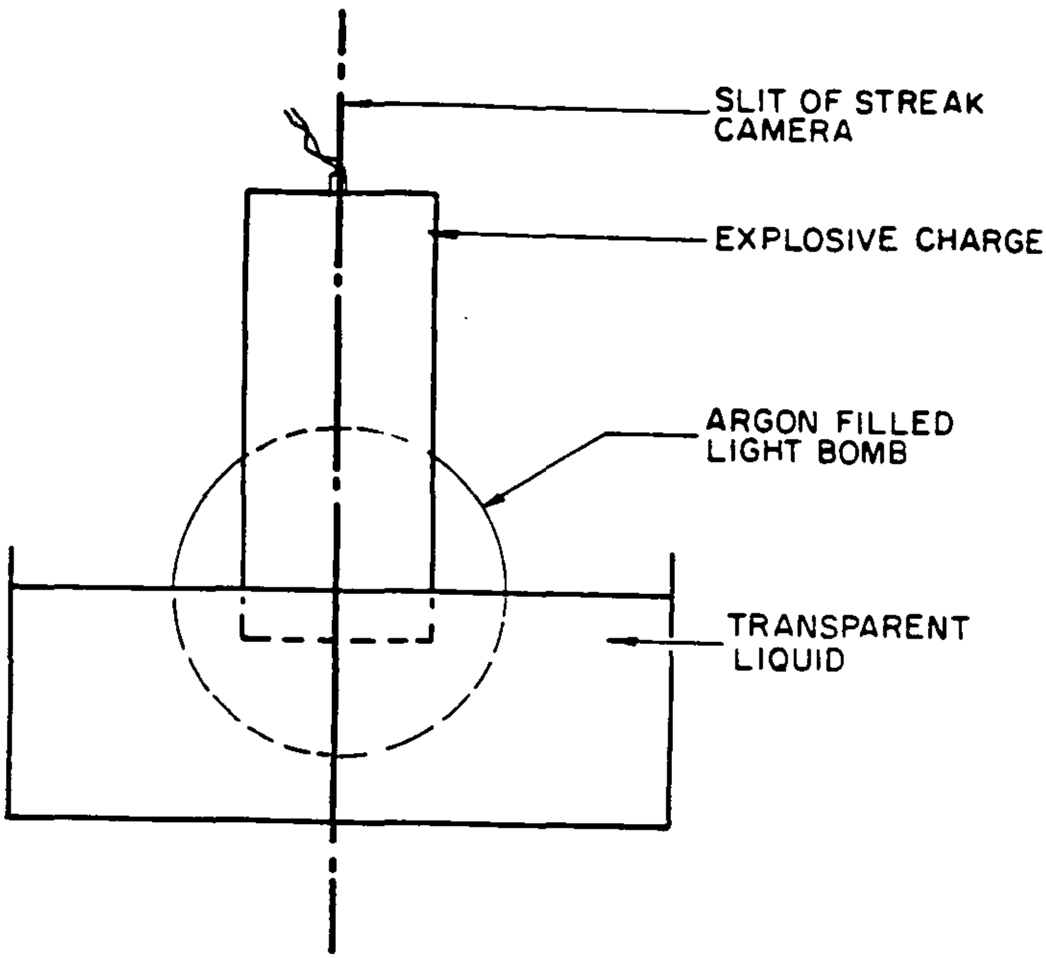


FIGURE 14: EXPERIMENTAL SET UP FOR THE MEASUREMENT OF THE DETONATION VELOCITY AND THE INITIAL SHOCK VELOCITY IN THE TRANSPARENT LIQUID

CHAPTER 4

GAP AND FRICTION SENSITIVITY OF EXPLOSIVES

4.1 Introduction

The gap sensitivity of explosive represents its ability to propagate through barriers. The gap sensitivity of an explosive is an important property to be considered in blasting operations. If the sensitivity is low, the detonation in the borehole can be interrupted because of obstacles (rocks) or air gaps. On the contrary, an explosive which is very sensitive can be dangerous to handle and can detonate sympathetically in the boreholes. Cross propagation of adjacent holes is very undesirable since this eliminates the effects of delays and results in excessive vibrations and poor fragmentation.

However one has to differentiate between solid gap and air gap sensitivity because the phenomena involved in each case are considerably different.

The friction sensitivity determines the safe handling of explosive charges. Charges can be subjected to friction forces when loaded in blastholes. These can be of a significant magnitude especially where pneumatic loaders are used.

4.2 Underdriven and Overdriven Detonations

The detonation state (C-J state) represents a dynamic stable condition. If the detonation wave encounters a small gap in the explosive charge, it will weaken temporarily and will come back to the original stable condition once the perturbation is passed.

The same will happen if the detonation wave encounters a part of the explosive which has greater energy. Temporarily it will strengthen but later it will reach the stable condition.

Consider the situation shown in Figure 1 a. A detonation is transmitted from a donor explosive to an acceptor explosive. In this case there are three possibilities; the shock wave transmitted in the acceptor can be stronger than the detonation wave in the acceptor, the shock wave can be of equal magnitude to the detonation wave in the acceptor or the shock wave can be of a smaller magnitude than the detonation wave in the acceptor.. The first case is called overdriven and the last case underdriven detonation. It has been found that in the case of an overdriven wave the strength always decays until the C-J condition is reached. In the case of the underdriven wave the detonation builds up to the C-J value. However, there is a limiting strength (below which the wave decays and detonation does not propagate. This limiting strength is of importance since it determines the conditions required for safe handling and reliable initiation of explosive materials.

4.3 The Gap Test

Experimentally a simple way to determine the sensitivity of an explosive to initiation is represented in the gap test. The gap test is shown in Figure 1 b. The experiment consists of a donor charge, an attenuator and an acceptor charge. By varying the attenuator thickness, different underdriven waves are transmitted to the acceptor. The thickness of the attenuator at which 50% of the times the acceptor detonates is called critical gap thickness. At that thickness the shock wave in the acceptor has a limiting value above which the acceptor has a high probability of detonation. The gap material is normally a standard solid material. Air gaps are not desirable because hot decomposition products of the donor explosive will impinge directly on the acceptor.

The result of the gap test depends on the geometry of the donor and acceptor charges as well as the attenuator material and the donor explosive. For this purpose various laboratories standardize gap tests by using the same donor and the same attenuator material. Thus the results of the tests are indicative of the explosives shock sensitivity.

Typical gap tests are shown in Figures 2 and 3.

The following factors affect the result of a standard gap test:

1. Density. The effect of density is shown in Figure 4[2] where the critical gap pressure is plotted against the percent of the theoretical maximum density. It is obvious that the explosive becomes less sensitive as the theoretical maximum density is approached. This is a general trend obtained in a variety of explosive compositions[2].

2. Temperature. The effect of temperature is shown in Figure 5. This is a general trend for any material in which the reaction rate increases with temperature[2].

3. Composition. It is obvious that the result of the gap test is composition dependant. It has been found that if wax is added to RDX or TNT, the shock sensitivity is decreased. However if wax is added to ammonium nitrate, the sensitivity is drastically increased. This happens because of the combination of an oxidizer

with a fuel and the dominant factor is the oxidation-reduction reaction. Figure 6 is typical of this phenomenon[2].

4. Acceptor diameter. Initiation is controlled not only from the magnitude of the impacting shock wave but from its duration as well. The reduction of the diameter of the acceptor has changed the duration of the shock wave. It is recommended that the charges are tested at a diameter above the minimum diameter for ideal detonation, where this is possible. According to Price the critical initiating pressure - diameter relationship should follow a curve as in Figure 7[5]. Experimental results by Moulard indicate the same trend for Composition B[6].

5. Confinement. Price has found that confinement of the acceptor in the test prevents the lateral rarefaction from producing a large disturbance. The confinement gives a result which is comparable to that which would be obtained for a very much larger diameter unconfined charge. The result may approach that which would be obtained in the one dimensional flow[2]. In Figure 8 the critical gap pressures for confined charges are compared to the critical cap pressures of unconfined charges. It is obvious that confinement increases the sensitivity of explosives.

4.4 Air Gap Sensitivity

This term denotes the initiation of an explosive charge without a priming device by the detonation of another charge in the neighbourhood. The transmission mechanism is complex. The important parameters are the shock wave, the hot reaction products of the donor and the flying parts from the casing of the donor charge. Various tests are conducted to determine the air gap sensitivity of explosives. In Europe the smallest diameter of manufacture is used in the test charges which are tested unconfined[3]. This will provide the largest gap below which detonation will always be observed. Confinement however affects the result. For this purpose coal mining explosives are tested in pipes which simulate boreholes. It is recommended that gap tests simulating the conditions of application are performed to determine the gap sensitivity of a particular product.

4.5 Initiation by Friction

The mechanism of heating by friction has been investigated by Bowden and co-workers. When solid bodies are pressed against each other contact will occur only at the summits of the surface irregularities. The total area of contact is a small fraction of the total surface area[4]. When the bodies are sliding against each other heat is developed at the regions of contact. Hot spots are created at the points of contact and their temperature depends on the pressure, sliding velocity and heat conductivity of the sliding material. The contact material with the lowest melting point determines the hot spot temperature. When melting occurs its supporting capacity is taken over by other points[4]. According to Bowden if the melting point of the slider is below the critical hot spot temperature for the explosive, detonation does not occur.

Several friction tests have been developed. The Swedish[4] developed a friction test in which the explosive is subjected to stresses similar to those when the explosive is charged in boreholes. The test consists of a block of granite which has a semi-cylindrical groove. A thin layer of explosive is placed in the groove and a slider moves on top. Various loads are put on the slider. The slider moves at a constant speed and the result is recorded as a function of the load.

In Germany a sample is placed on a roughened porcelain plate(3). The sample is put on top of it and a porcelain cylinder is placed on top with various loads. The plate moves at a certain speed and the result is recorded as a function of the load.

Similar tests have been developed in other countries.

4.6 References

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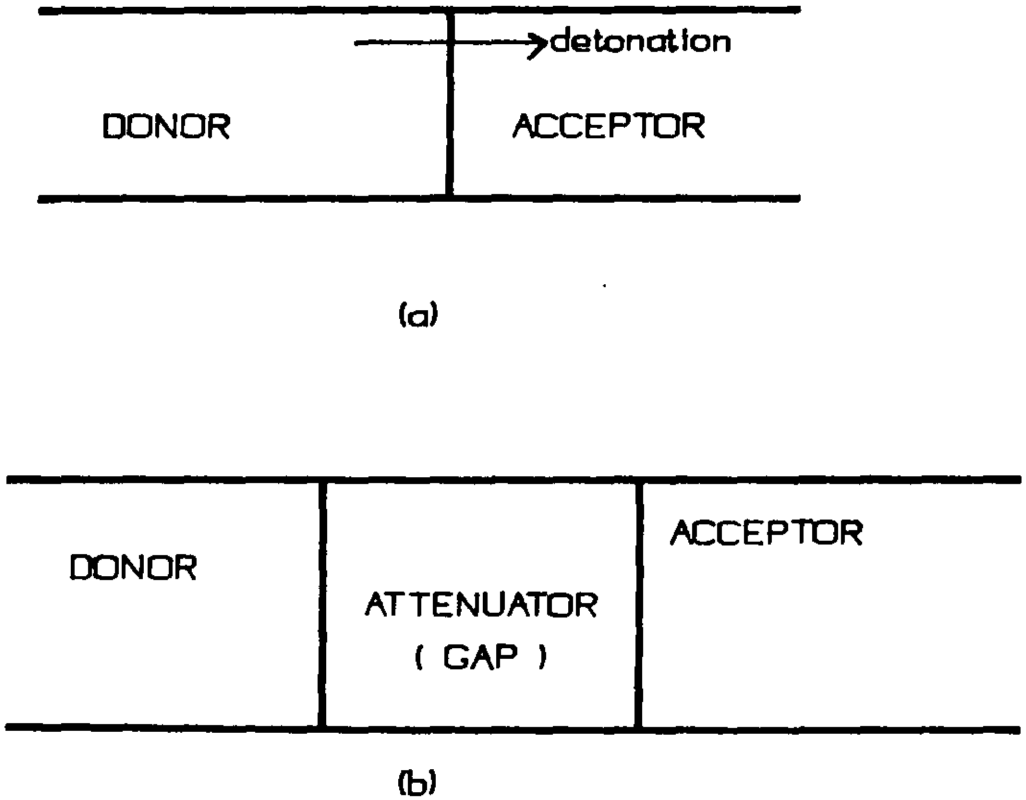


FIGURE 1: TYPICAL GAP TEST CONFIGURATION

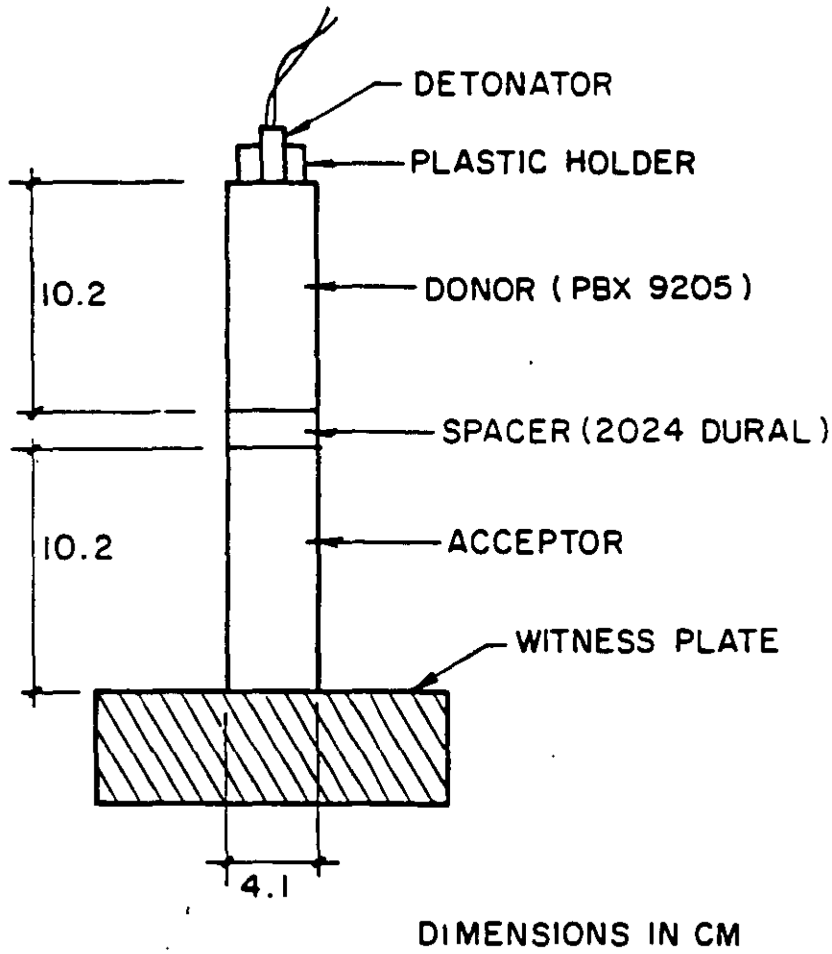


FIGURE 2: THE LOS ALAMOS LARGE GAP TEST

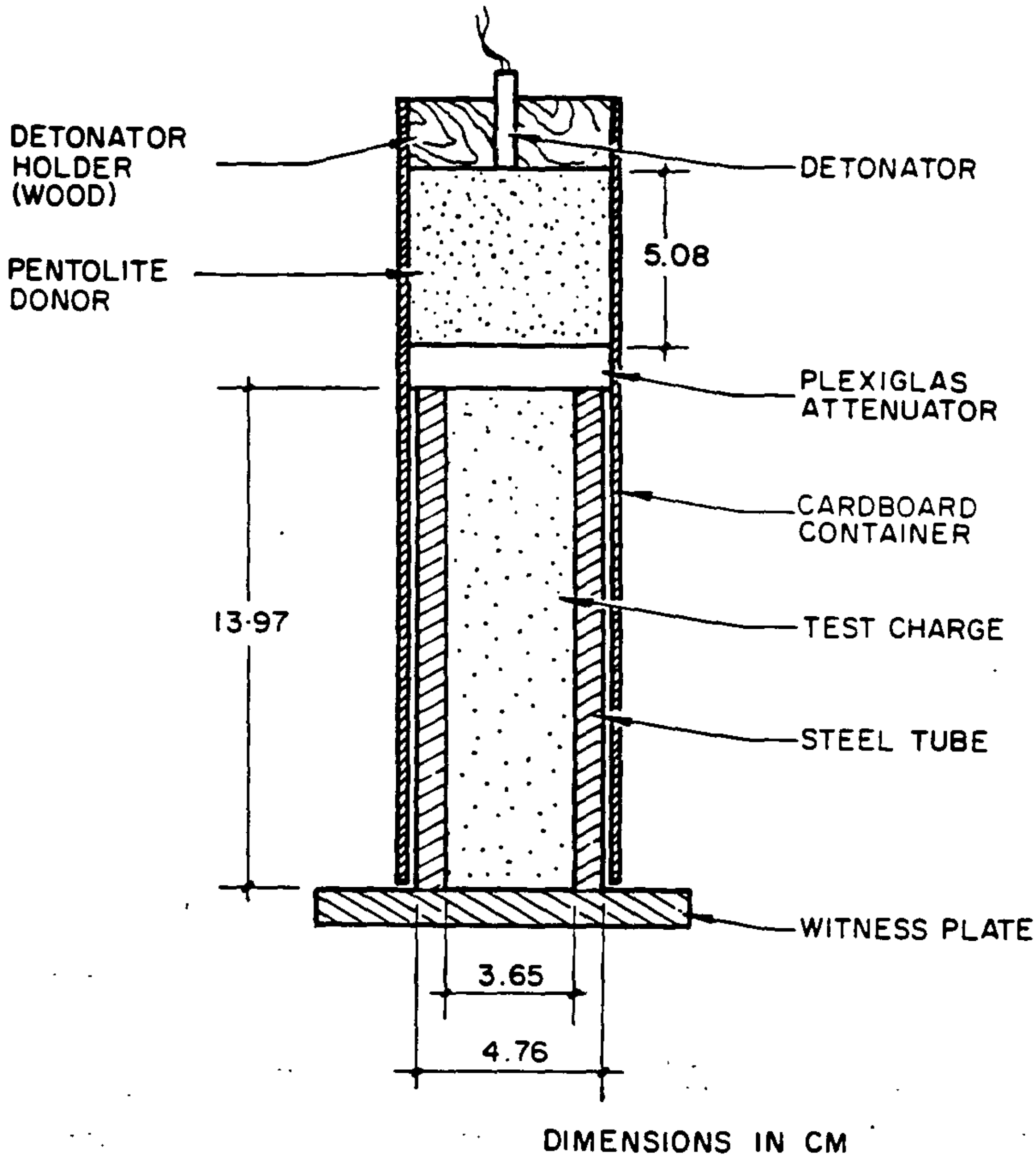


FIGURE 3: THE NOL GAP TEST

FIGURE 4: EFFECT OF DENSITY ON CRITICAL GAP PRESSURE

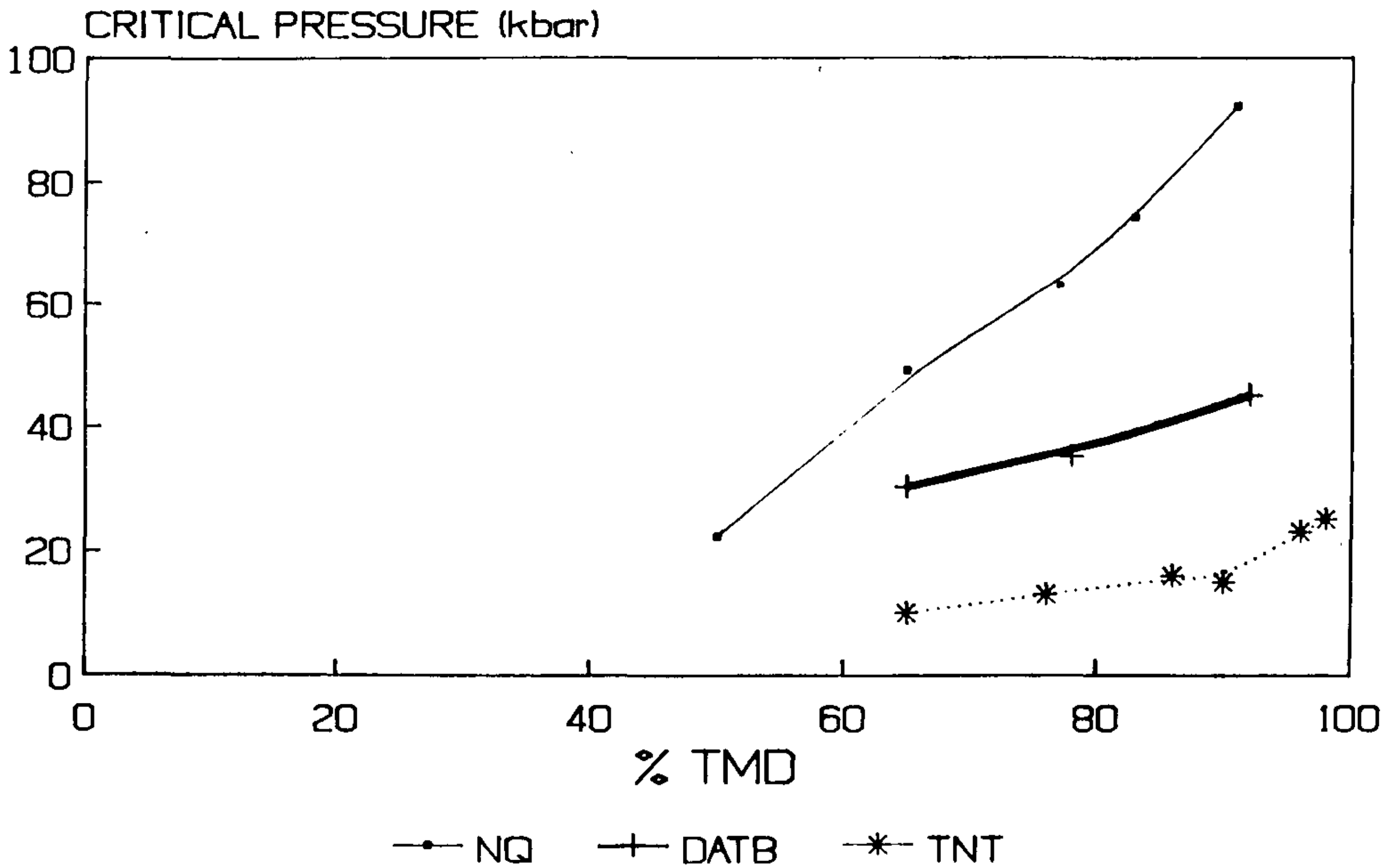


FIGURE 5: EFFECT OF THE COMPOSITION ON CRITICAL GAP PRESSURE

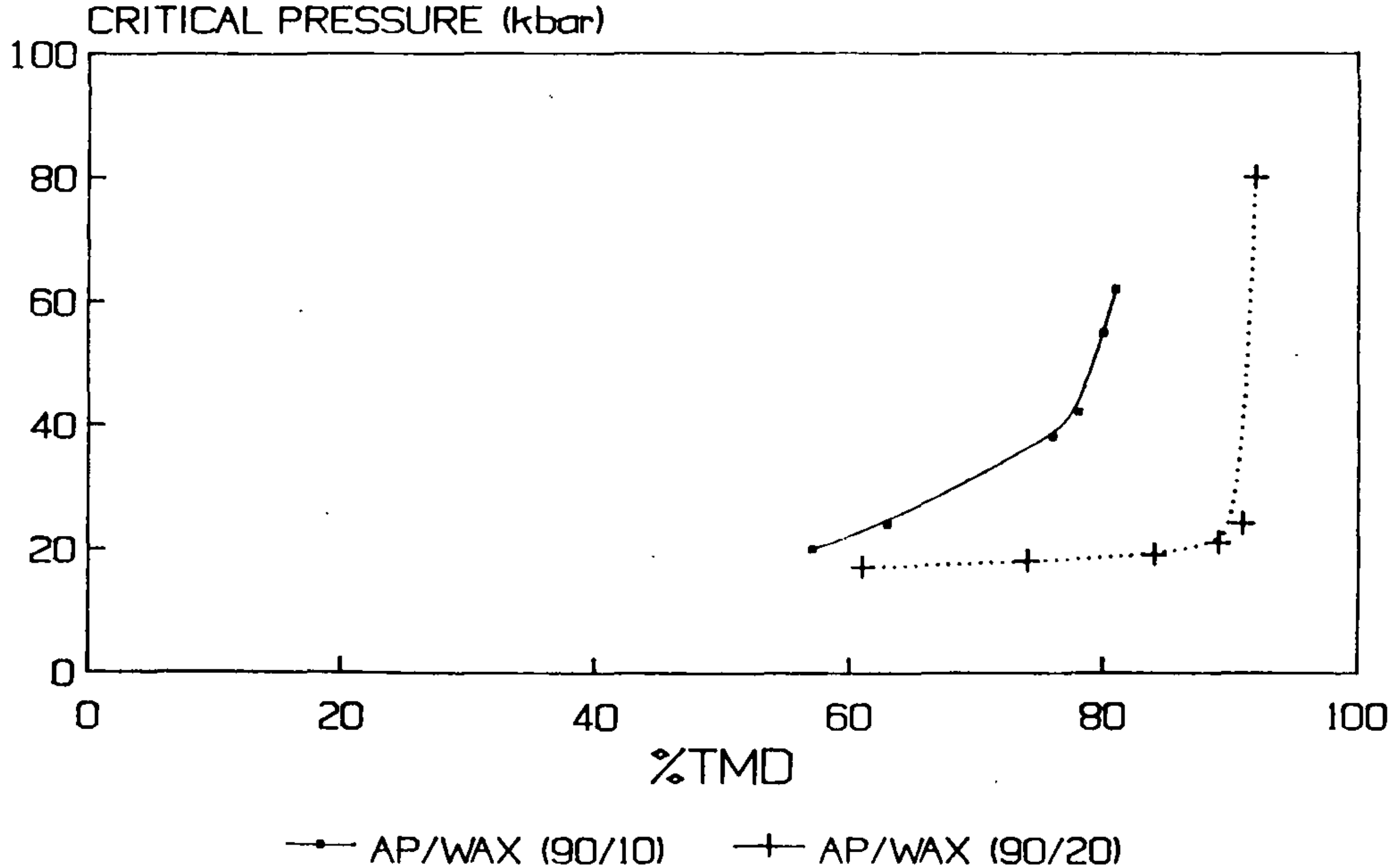
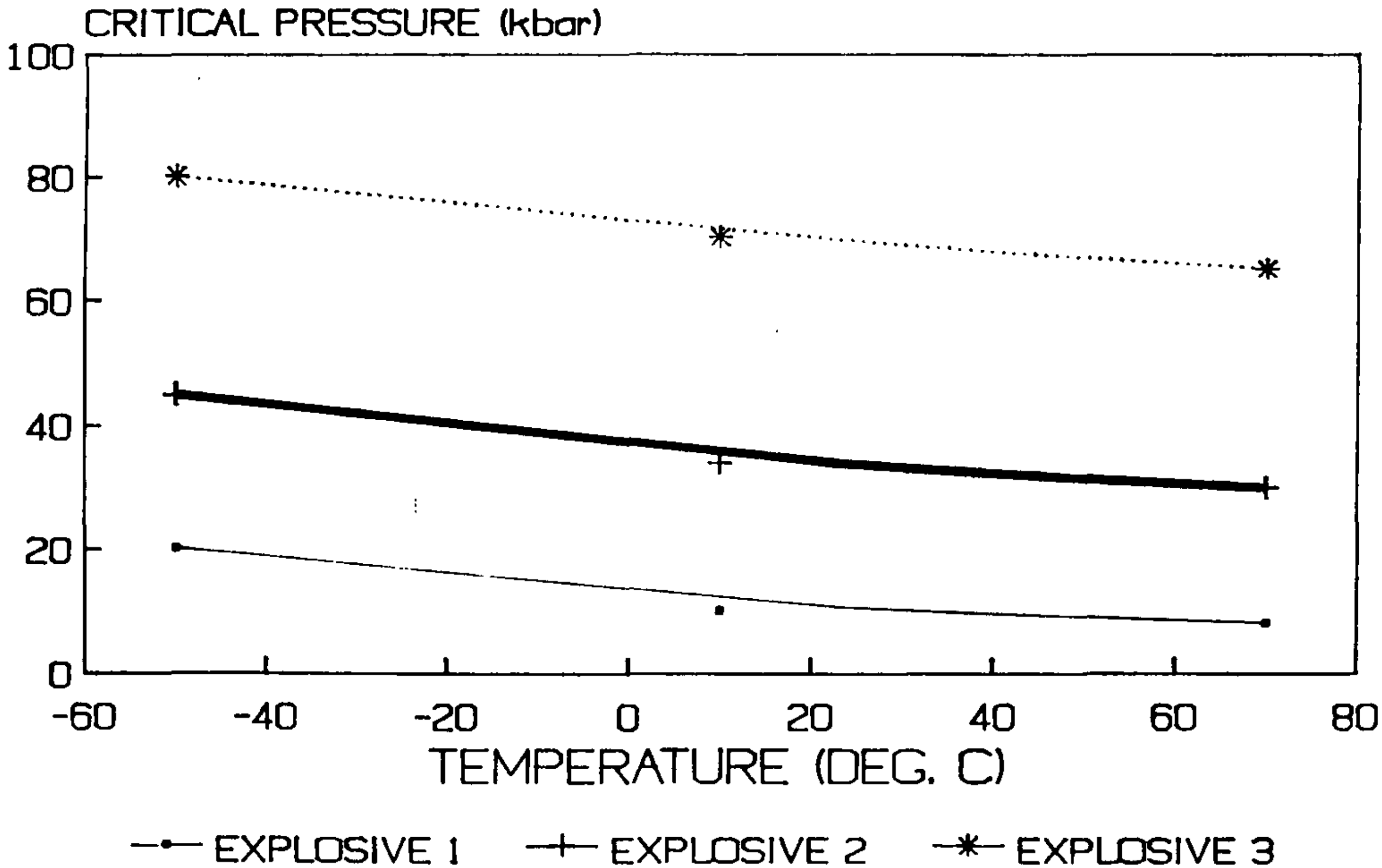


FIGURE 6: EFFECT OF TEMPERATURE ON THE CRITICAL GAP PRESSURE



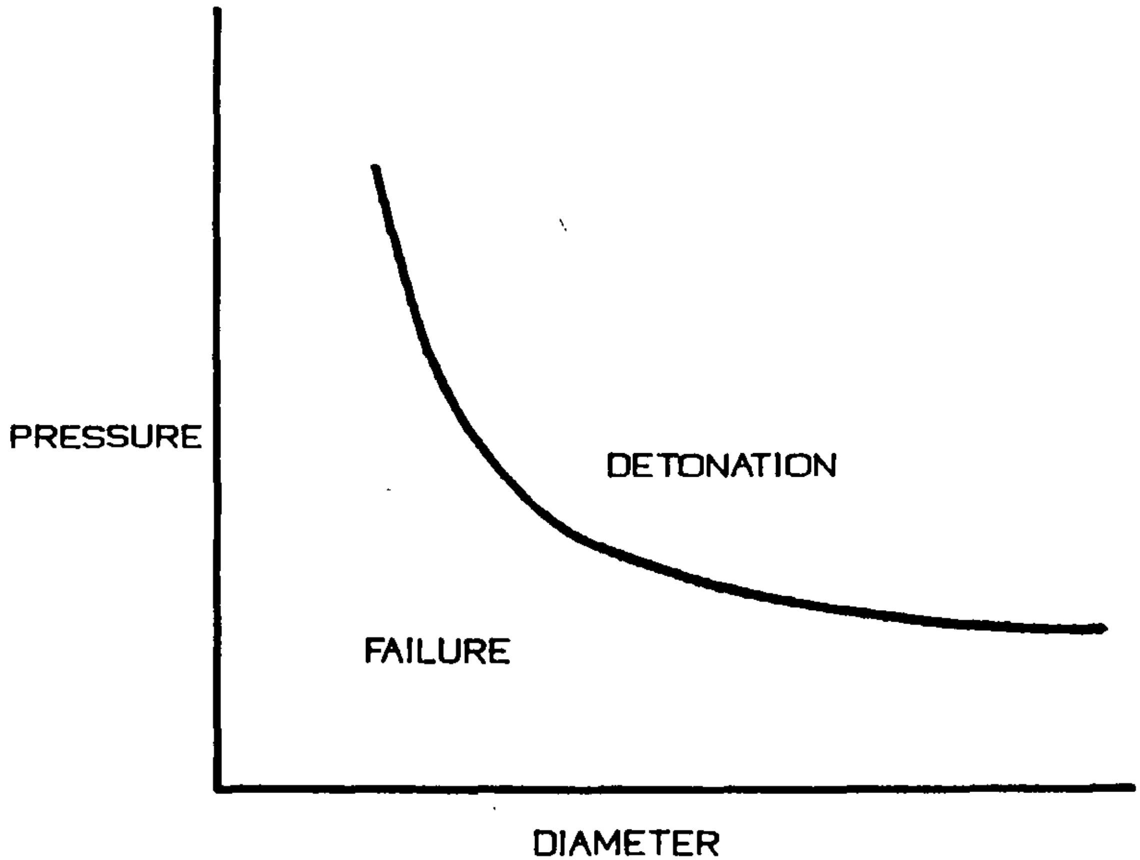
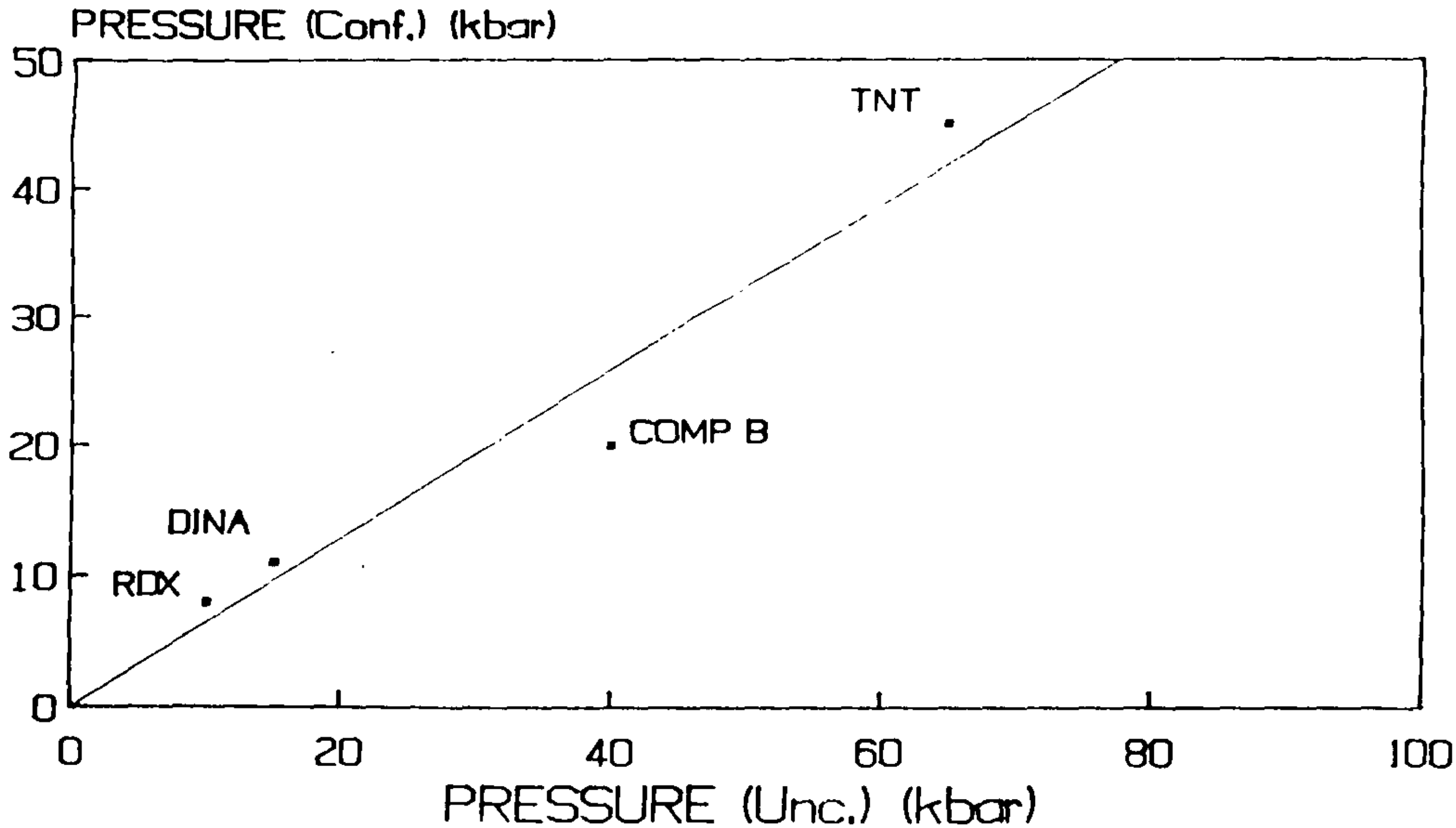


FIGURE 7: VARIATION OF CRITICAL PRESSURE WITH DIAMETER

FIGURE 8: EFFECT OF CONFINEMENT ON CRITICAL GAP PRESSURE



FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

CHAPTER 11
BLASTING THEORY

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

CHAPTER 11
BLASTING THEORY

by R. Frank Chiappetta

1. INTRODUCTION

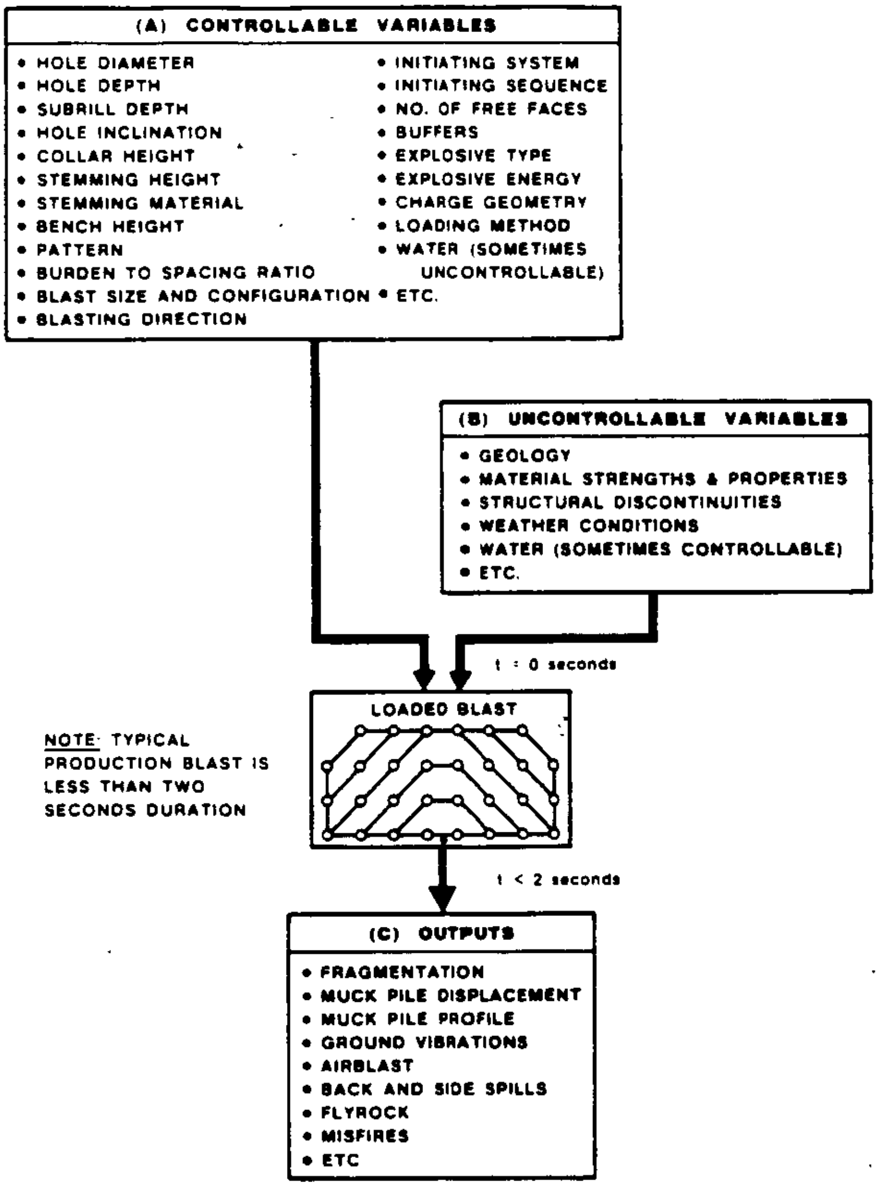
Blasting theory is perhaps one of the most interesting, thought provoking, challenging and controversial areas of our industry. It encompasses many areas in the science of chemistry, physics, thermodynamics, shock wave interactions, and rock mechanics. In broad terms, rock breakage by explosives involves the action of an explosive and the response of the surrounding rock mass within the realms of energy, time and mass. Past, current and new blasting theories are presented along with the factors affecting fragmentation and general blast design criteria. The chapter content has been carefully selected to emphasize the concepts associated with each blasting theory rather than a rigorous mathematical, physical, or chemical treatment through formulae. Where formulae are introduced, they are merely to enhance the concepts presented.

In spite of the tremendous amount of research conducted in the last few decades, no single blasting theory has been developed and accepted that adequately explains the mechanisms of rock breakage in all blasting conditions and material types. Given specific test environments, conditions and assumptions, individual researchers have contributed valuable information and insight as inputs into blasting theories, although a simple "plug-in" type formula for predicting "optimum fragmentation" is still largely unresolved There is as yet no consistent and widely applicable theory of blasting, but only a number of limited and disconnected theories, many of which are empirical in nature and based on ideal blasting conditions. Blasting theories have been formulated and based on pure speculation, years of blasting experience on a trial and error approach, laboratory testing, field investigations, and mathematical and physical models adapted from other disciplines of science.

Primary breakage mechanisms have been based upon:

- Compressional and tensile strain wave energy
- Shock wave reflections at a free face
- Gas pressurization on the surrounding rock mass
- Flexural rupture
- Shear waves
- Release-of-load
- Nucleation of cracks at flaws and discontinuties
- In-flight collisions

Since so many schools of thought surround blasting theory, one must be prepared to investigate not only the theories, but the overall field input variables that are inherent in any blast design to have any practical meaning. Given the diverse nature of field conditions encountered and the overwhelming number of blast design variables to select from, blast results may not always be easily predicted as is outlined in Figure 11-1. Where one theory is successful in one specific environment or application, it may not be as predictive in another.



FIELD MODEL ILLUSTRATING BLAST DESIGN INPUTS AND OUTPUTS

FIGURE 11.1

Often more than one theory is needed to clarify or explain certain results. Parallel this approach to the physicist trying to explain light with only one theory, that is, the wave theory. With the passage of time it became apparent that everything associated with light could not always be adequately explained with this theory alone and hence, another theory, the particle or "packets of energy" theory was developed to explain the phenomena of light in which the first theory failed. With both theories, the physicist could now explain many of the mysteries surrounding light which eventually led to new developments such as the laser. Similarly, in trying to define the mechanisms of rock breakage by explosives, more than one theory or explanation is often needed. In any case, a blasting theory should not only attempt to explain and predict the breaking process, but more importantly, it should suggest and allow new methods and techniques to improve on current blasting practices.

2. TIME EVENTS FOR THE BREAKING PROCESS

There are basically four time frames designated as T1 to T4 in which breakage and displacement of material occur during and after complete detonation of a confined charge.

The time frames are defined as follows:

T1 — Detonation
T2 — Shock or Stress Wave Propagation
T3 — Gas Pressure Expansion
T4 — Mass Movement

Each time frame is first discussed separately, and then discussed in conjunction with blasting theories for an overall, more detailed explanation and meshing of events. Although these are treated as discrete events, it should be emphasized that in a typical shot hole or production blast, one event phase can occur simultaneously with another at specific time intervals.

a. T1 — DETONATION

Detonation is the beginning phase of the fragmentation process. The ingredients of an explosive consisting of a fuel and oxidizer combination; upon detonation, are immediately converted to high pressure, high temperature gases. Pressures just behind the detonation front are in the order of 9 Kbars to 275 Kbars, while temperatures range from approximately 3000° to 7000°F.[2]

Detonation pressure is generally expressed as a function of the velocity of detonation and density of the explosives as,

$$P_d = (2.325 \times 10^{-7}) \times \rho \times VOD^2$$

Where P_d = detonation pressure in Kbars
ρ = density in g/cc
VOD = velocity of detonation in ft/sec.

To change detonation pressure from Kbars to lb/in², multiply Kbars by 14,700. Generally, explosives yielding higher detonation pressures are required to fracture materials which are massive, fine grained, hard, tightly bonded and strongly consolidated with heavy burdens. Typical values of detonation pressure for selected explosives are presented in Table 11-1.

TABLE 11.1
DETONATION PRESSURES FOR SELECTED EXPLOSIVES

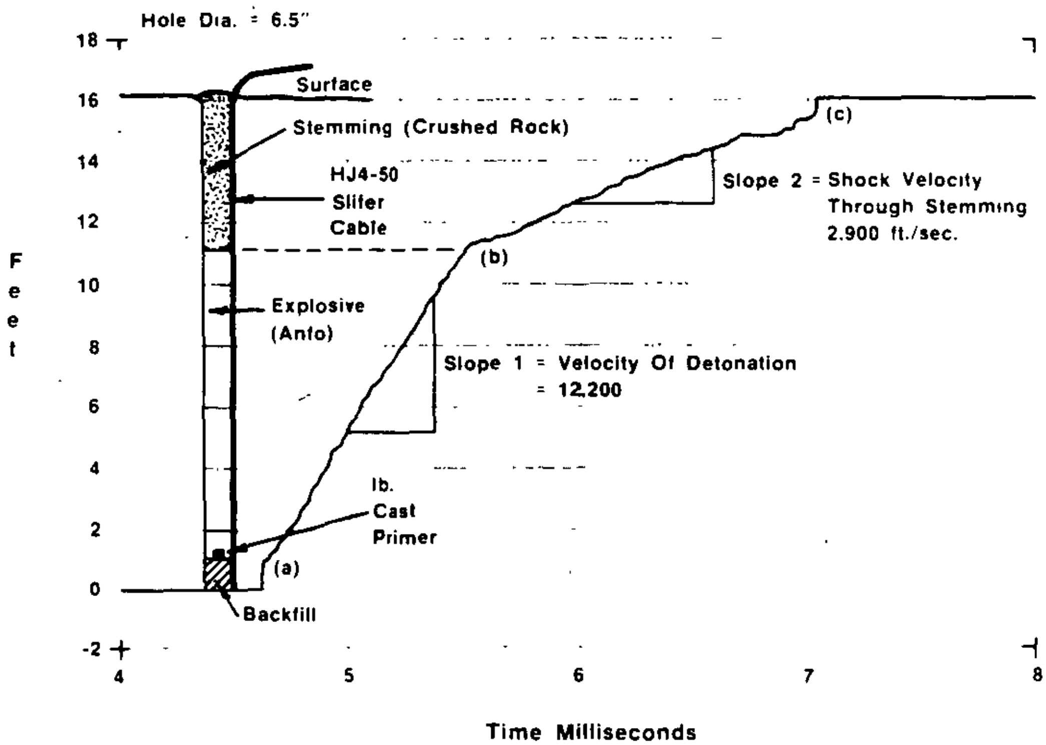
Explosive	Density (g/cc)	VOD (ft/sec)	Detonation Pressure (Kbars*)	Pressure (psi)
ANFO	0.81	12,000	27.00	396,900
POWERMAX 420	1.19	19,000	100.00	1,470,000
HI-PRIME	1.40	20,000	130.00	1,911,000
"G" BOOSTER	1.60	26,000	251.00	3,689,700

*1 Kbar = 14,700 PSI

The detonation wave starts at the point of primer initiation in the explosive column and travels at supersonic speeds. Supersonic refers to velocities which are faster than the speed of sound in the explosive. Typical velocities of detonation for commercial explosives range from 8,000 to 26,000 ft/sec. This velocity, sometimes referred to as the steady-state velocity, remains fairly constant for a given explosive, but varies from one explosive to another, depending primarily on the composition, particle size and density of the explosive. To a lesser extent, the steady state velocity is also affected by the degree of confinement and explosive diameter.

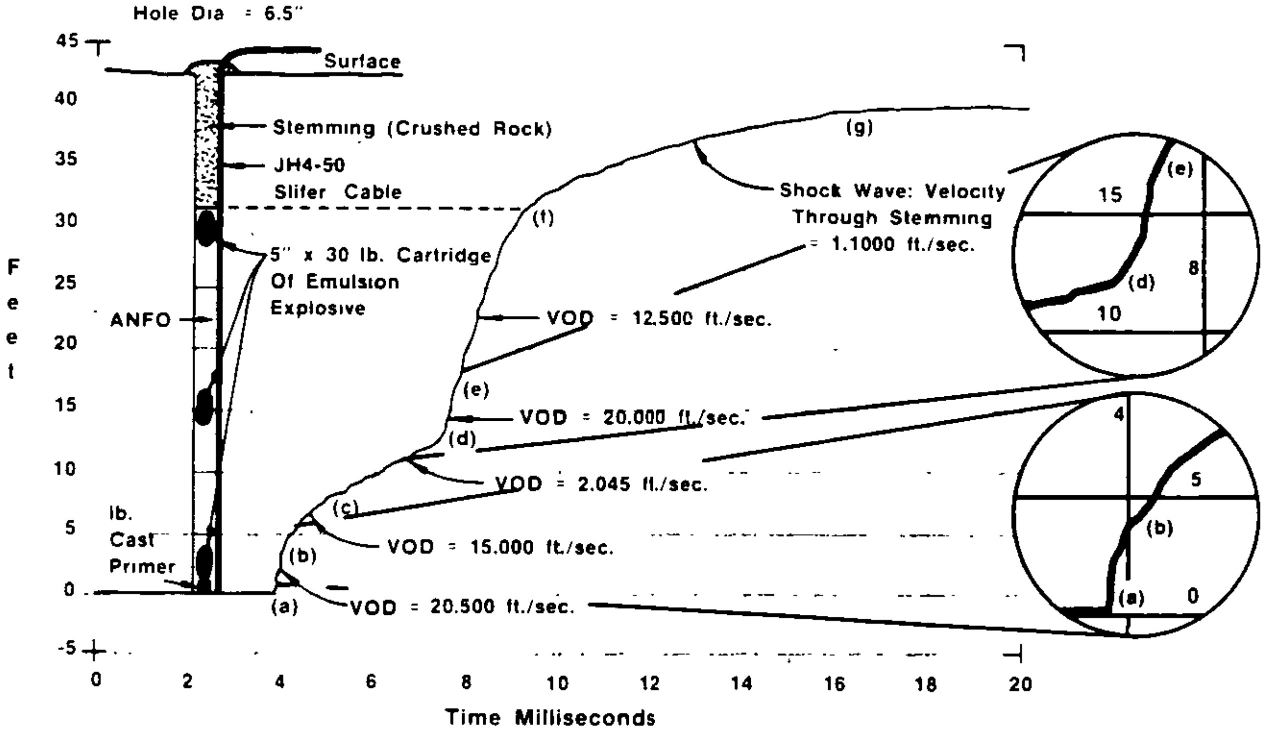
Since the velocity of detonation is greater than the velocity of sound in the explosive, the explosive material directly in front of the detonation head is totally unaffected until the detonation head passes through it. In a typical 30 foot explosive column loaded with an explosive having a characteristic velocity of detonation of 10,000 ft/sec. complete detonation and energy release within the entire column would occur in about 3 milliseconds. For an explosive with a velocity of detonation of 20,000 ft/sec. detonation and energy release would be complete in 1.5 milliseconds. Detonations of this kind are self-sustaining due to the inertia of the explosive itself that provides confinement necessary to maintain conditions for fast chemical reaction rates.

Figure 11-2 and 11-3 illustrate two typical hole load configurations. Velocity of detonation within the explosive column was measured with the SLIFER System developed at SANDIA NATIONAL LABORATORIES For a continuous 11 foot column of cartridged ANFO. the velocity of detonation was measured to be 12,200 ft/sec as indicated by the slope of the straight line segment between point (a) and (b) in Figure 11-2. The straight line is indicative of a consistent explosive composition. constant density and a stable velocity of detonation. As detonation progresses along the column. not only is a



VELOCITY OF DETONATION MEASUREMENT USING THE SLIFER SYSTEM DEVELOPED AT SANDIA NATIONAL LABORATORIES
FIGURE 11.2

shock wave imparted into the surrounding medium adjacent to the borehole wall, but is also imparted into the stemming as indicated by the slope of the straight line segment between points (b) and (c). In this case, the shock wave velocity through the stemming was measured to be 2,900 ft/sec. or approximately ¼ that of the velocity of detonation.



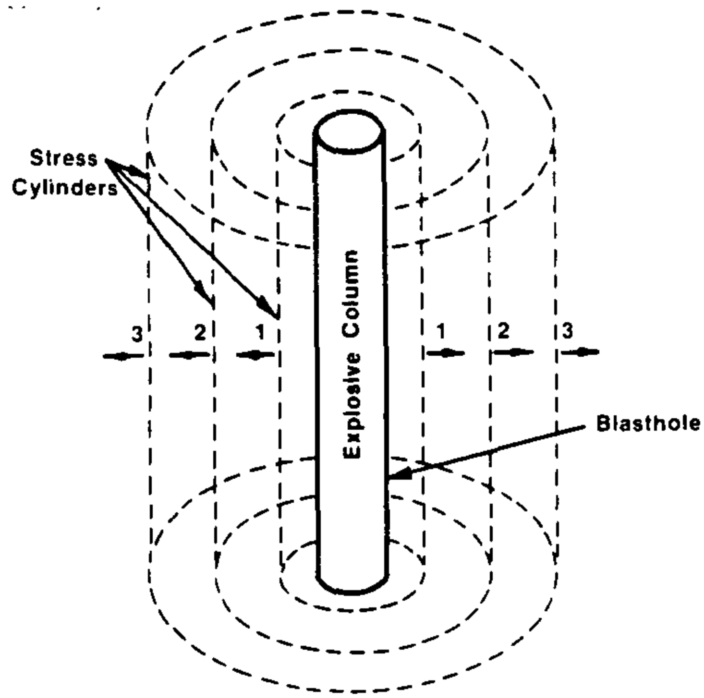
VELOCITY OF DETONATION MEASUREMENT USING THE SLIFER SYSTEM DEVELOPED AT SANDIA NATIONAL LABORATORIES.

FIGURE 11.3

In Figure 11-3, results are shown using ALTERNATE VELOCITY techniques with a hole loaded with ANFO as the main charge, with cartridges of APEX 260 emulsion spaced 11-12 feet along the column. Without direct measurements of the continuous velocity of detonation, much of the information would not have been discernable in the field by direct observation. Many important points are noteworthy in the results. Between points (a) and (b), the velocity of detonation for the 3 foot length of emulsion cartridge is 20,500 ft/sec. Between (b) and (c) the velocity of detonation is reduced from 20,500 ft/sec to 2,045 ft/sec within the ANFO and the detonation is sustained at the lower velocity until point (d) is reached. At point (d) the detonation head encounters another emulsion cartridge, which when detonated, at 20,000 ft/sec between points (d) and (e), brings ANFO back up to its normal velocity of detonation of 12,500 ft/sec. Thus, even a low order ANFO detonation can act as a very effective primer for the emulsion cartridge. The decrease in velocity between points (b) and (c) is attributed to water trickling into the bottom part of the hole from the surrounding rock mass. Although ANFO can tolerate up to a 10% water saturation level, it does so at the cost of blasting efficiency. If the center emulsion cartridge was not present, one of two things would have occured It may have sustained a low order ANFO detonation with a velocity of 2,045 ft/sec throughout the remaining explosive column, or it would have soon failed. It has been demonstrated in field trials that where an explosive of higher velocity of detonation is embedded sparingly within the column of a main explosive with a lower velocity of detonation, that better results are generally achieved. The greater the difference in detonation velocities and the harder the material to be blasted, the more pronounced are the results

b. T2 — SHOCK AND STRAIN WAVE PROPAGATION

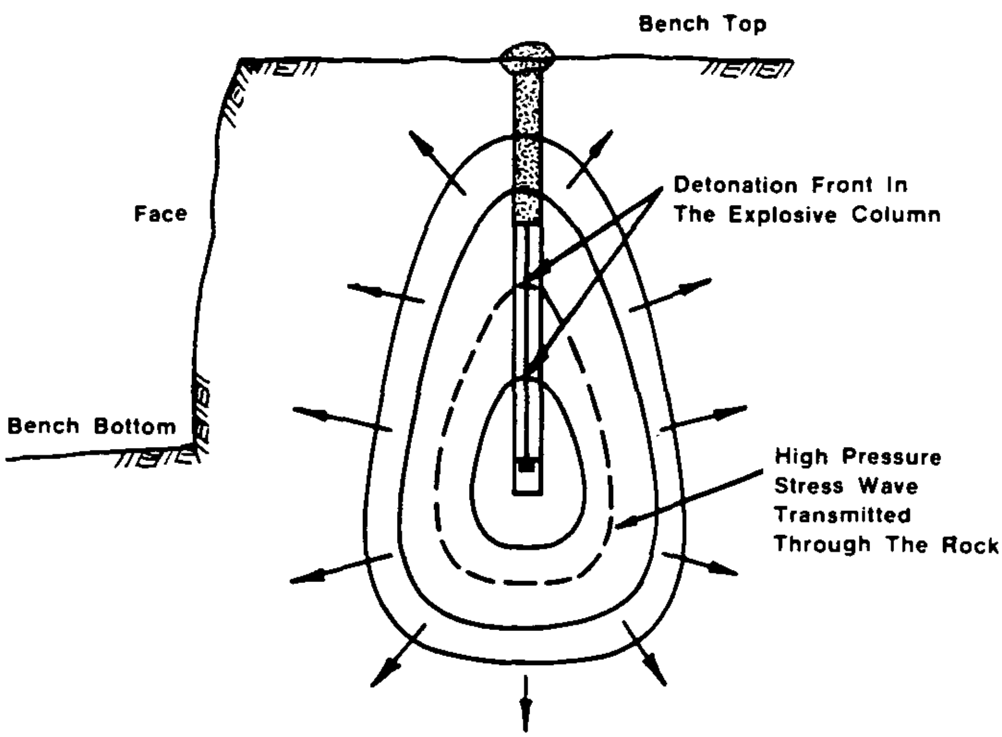
The second phase, immediately following detonation or in conjunction with the detonation phase of T1, is the shock and strain wave propagations throughout the rock mass. This disturbance or emitted



1,2,3 Successive Positions Of Stress Wave

THEORETICAL POSITIONS OF THE OUTBOUND DISTURBANCE FROM A COLUMN CHARGE
FIGURE 11.4

pressure wave(s) emitted into the rock mass results, in part, from the rapidly expanding high-pressure gas impacting the borehole wall. The geometry of dispersion depends primarily on the shape of the charge. If the charge is shot, with a length to diameter ratio of less than or equal to 6:1, then the disturbance is propagated in the form of an expanding sphere. If the charge is long, with a length of diameter ratio of greater than 6:1, then the disturbance is propagated in the form of an expanding cylinder. (Figure 11-4). However, in a typical, bottom primed, cylindrical shot hole normally encountered in bench blasting, the strain waves originally formed near the point of initiation are already in progress and propagating into the surrounding medium, while the detonation is still progressing within the explosive column. Thus, close to the shot hole, strain wave propagation is neither perfectly spherical nor cylindrical but more like that shown in Figure 11-5.



SECTION THROUGH THE FACE DURING DETONATION SHOWING EXPANDING STRESS WAVE FRONT

FIGURE 11.5

The pressure next to the borehole wall will rise instantaneously to its peak and then rapidly decay exponentially. The quick decay is due to cavity expansion of the borehole and increased gas cooling. Cavity expansion around the borehole can occur through crushing, pulverization, and/or displacement of material and can range anywhere from about one to three hole diameters depending on the medium and explosive used. Generally, extensive compressive, shear and tensile failure occur as a region of pulverized material since the wave energy is at its maximum near the borehole wall.

As the strain wave front proceeds outward, it has a tendency to compress the material at the wave front through a volume change. At right angles to this compressive front, there exists another component referred to as the tangential or "hoop" stress. The tangential stress, if large enough, can cause tensile failures at right angles to the direction of propagation. The largest tensile failures are expected to occur close to the borehole where the tangential stress is high enough for failure to occur. Both the compressive and tensile components of the wave front decay with distance from the borehole.

When the compressive wave front encounters a discontinuity or interface, some of the energy is transferred across the discontinuity and some reflected back to its point of origin.[4] For the most part, the partitioning of energy depends on the ratio of the acoustic impedance of the materials on either side of the interface, as illustrated in Figure 11.6. Acoustic impedance, Z, for any material is defined as

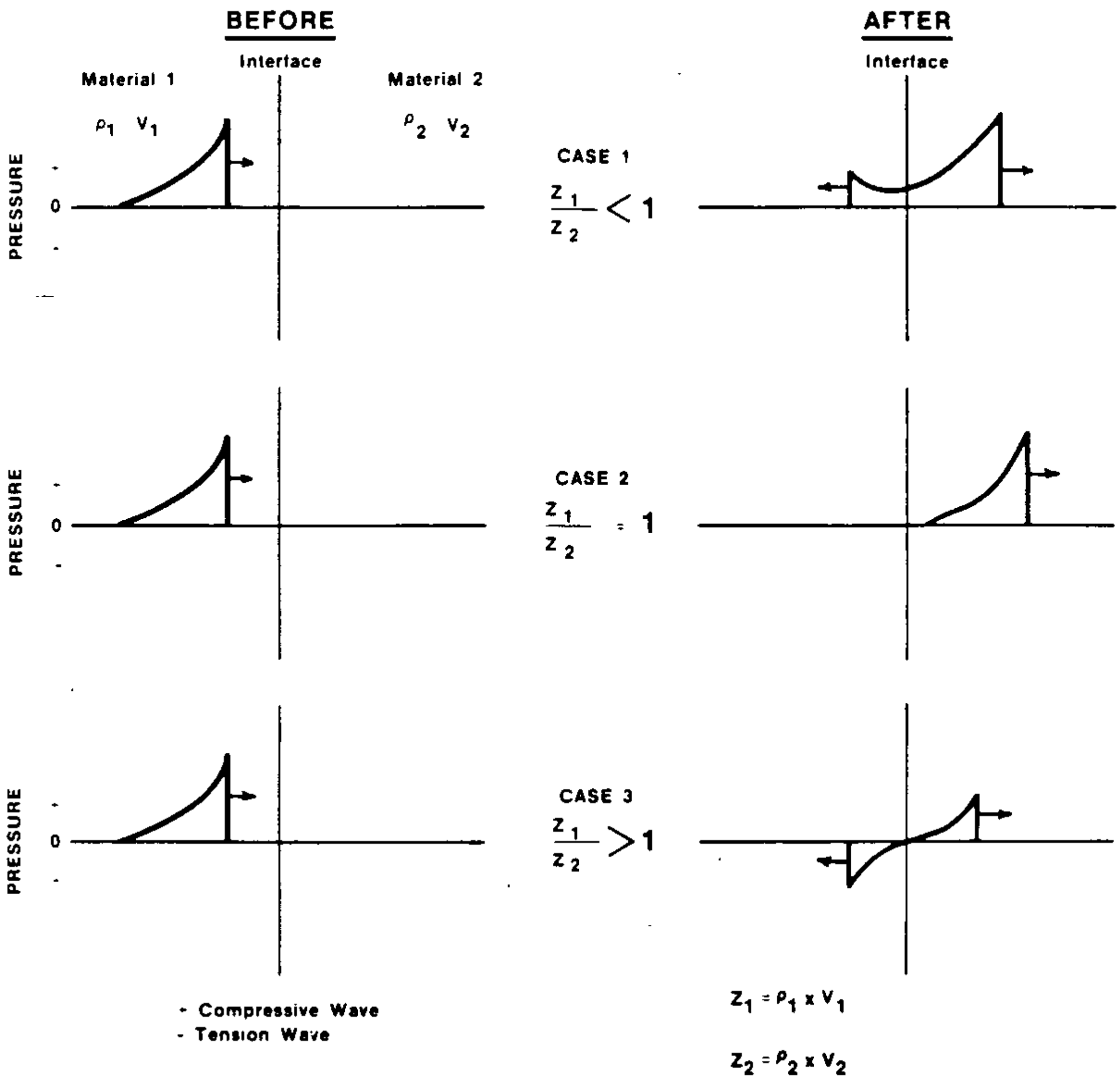
$$Z = \rho \times V_p$$

where:

Z	=	acoustic impedance
ρ	=	density of material
V_p	=	sonic velocity of material

In reference to Figure 11-6, where the ratio of the acoustic impedance of material 1 to material 2 is less than one, some of the wave energy is transferred into material 2 and some reflected back, but both waves remain compressional. When the acoustic impedance ratio is 1, all of the energy is transferred into material 2 and no reflected wave occurs. When the impedance ratio is greater than 1, then some of the energy gets transferred into material 2 as a compressive wave and the remaining energy gets reflected at the interface as a tensile wave. When a compressive wave travelling through rock encounters an interface such as a free face, nearly all of the energy will be reflected back as a tensile wave. If the burden distance between the free face and explosive column is relatively small in contrast to normal burdens for a chosen explosive, then most of the energy is consumed in spalling at the free face.

The interaction of stress waves in the outgoing compressive and reflected tensile modes around discontinuities and flaws within the rock mass is an area of intense research and is considered to be quite important in some of the newer blasting theories.



INTERACTION OF STRESS WAVES AT AN INTERFACE
FIGURE 11.6

c. T3 — GAS PRESSURE

During and/or after strain wave propagation, the high pressure, high temperature gases impart a stress field around the blasthole that can expand the original borehole, extend radial cracks and jet into any discontinuity. It is during this phase where some controversy exists as to the main mechanism of fragmentation. Some believe that the fracture network throughout the rock mass is completed while others believe that the major fracturing process is just beginning. In any case, it is the gases that have jetted into discontinuities and the fracture network that is either fully developed or being developed, which are responsible for the displacement of broken material.

It is not clear as to the exact travel paths that gases take within the rock mass, although it is agreed that they will always take the path of least resistance. This means that gases will first migrate into existing cracks, joints, faults, and discontinuities, in addition to seams of material which exhibit low cohesion or bonding at interfaces. If a discontinuity or seam between the borehole and free face is sufficiently large, the high pressure gases will immediately vent to the atmosphere, rapidly reducing the total confinement pressures, and results in reduced displacement of broken and fragmented material.

The confinement time of gas pressures within a rock mass vary significantly depending on the amount and type of explosive, material type and structure, fracture network, amount and type of stemming, and burden. ATLAS studies, with the use of high-speed photography in full scale bench blasts, have shown that gas confinement times before the onset of movement can vary from a few milliseconds to tens of milliseconds [3]. To date, confinement times have been measured to range from 5 to 110 milliseconds for a variety of materials, explosives and burdens. Generally, but not always, confinement times can be decreased by employing higher energy explosives, decreasing the burden, or a combination of both. This applies equally to material at the bench face or at the bench top, as in the case of stemming blowouts or cratering. Refer to Figures 12.35 and 12.36 Vibration/Airblast for specific examples of gas confinement times for stemming blowouts. It is evident that only suitably burdened and well stemmed charges can deliver their full potential of additional gas extension fracturing and mass movement.

d. T4 — MASS MOVEMENT

Mass movement of material is the last stage in the breaking process. The majority of fragmentation has already been completed through compressional and tensile stress waves, gas pressurization or a combination of both. However, some degree of fragmentation, although slight, occurs through in-flight collisions and also when the material impacts the ground. Generally, the higher the bench height, the greater is this type of breakage owing to increased impact velocities of individual fragments when falling onto the bench floor. Similarly, material ejected from opposite rows of a "V-shot" design upon head-on collisions can result in increased fragmentation. This phenomenon was evidenced and documented with the use of high-speed photography of bench blasts.

Mass burden movement of fragmented material is shown in Figure 11-7 for a number of typical face conditions encountered in bench blasting operations. Face profiles and velocities are based on the results of high-speed photographic analysis performed at the ATLAS POWDER COMPANY. Where no subdrilling is utilized, (a and b), two types of face movement may be encountered. In 11-7a the entire length of face burden, directly in front of the explosive column, moves out similar to a plane wave and the face velocity at any point is constant. This behavior is usually encountered where material is very competent, quite brittle, and structured with well defined, largely spaced joints, much greater than the spacings or burdens employed in blast designs. When the material is soft, highly fissured, and/or closely jointed as might be found in coal and some sedimentary deposits, face profiles resembling that of flexural rupture is more likely. In this case, the greatest displacement and velocity occur adjacent to the center of the explosive column with the least amount of movement occuring at the toe and crest. When identical conditions in 11-7b are assumed and when subdrilling is employed, face movement results in much the same way except that the toe burden is displaced upwards faster and at a greater angle to the horizontal.

The first three cases assumed a relatively straight face between the crest and toe, however, in many bench blasting operations, the condition is more like that illustrated in Figure 11-7d, where toe burden is considerably greater than the crest burden. The toe burden is too great for the explosive selected, hence, very little movement occurs at the toe while the greatest displacement results in the upper half of the bench.

Three options are available to increase toe movement:

- Employ angle drilling in an attempt to maintain constant burdens from the crest to the toe
- Use a higher energy bottom charge in the current vertical drill holes
- Decrease the burden with the current vertical drill holes

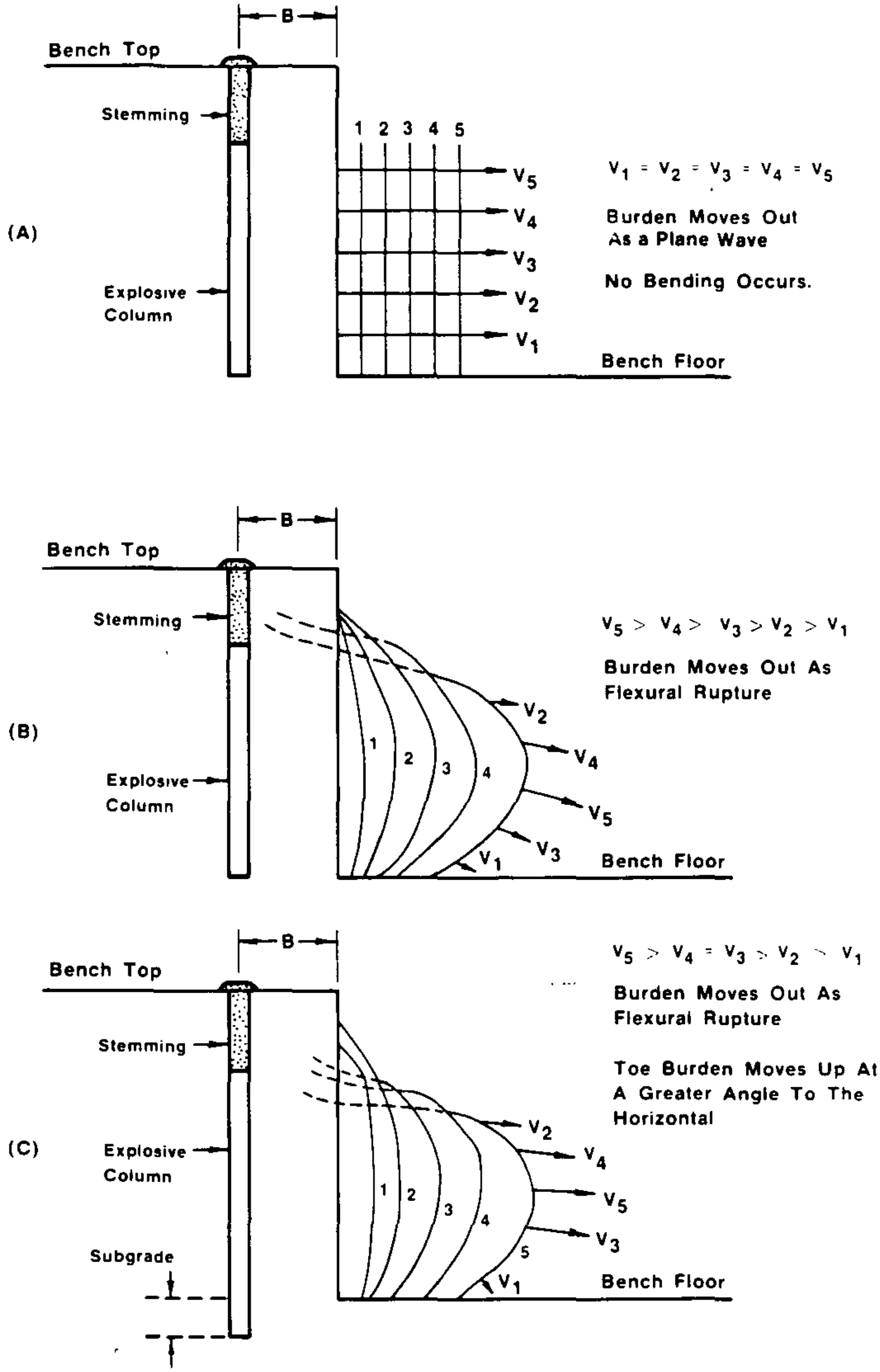
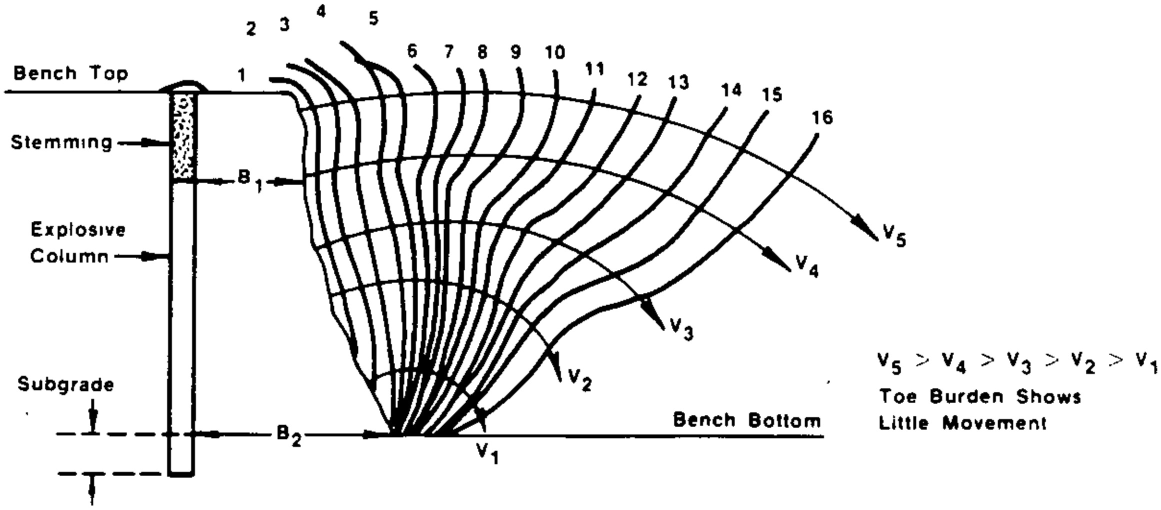

(A)
B
Bench Top
Stemming
Explosive Column
1 2 3 4 5
V5
V4
V3
V2
V1
V1 = V2 = V3 = V4 = V5
Burden Moves Out As a Plane Wave
No Bending Occurs.
Bench Floor
(B)
B
Bench Top
Stemming
Explosive Column
V5 > V4 > V3 > V2 > V1
Burden Moves Out As Flexural Rupture
1 2 3 4
V2
V4
V5
V3
V1
Bench Floor
(C)
B
Bench Top
Stemming
Explosive Column
Subgrade
V5 > V4 = V3 > V2 > V1
Burden Moves Out As Flexural Rupture
Toe Burden Moves Up At A Greater Angle To The Horizontal
1 2 3 4 5
V2
V4
V5
V3
V1
Bench Floor

FIGURE 11.7

(D)



(E)

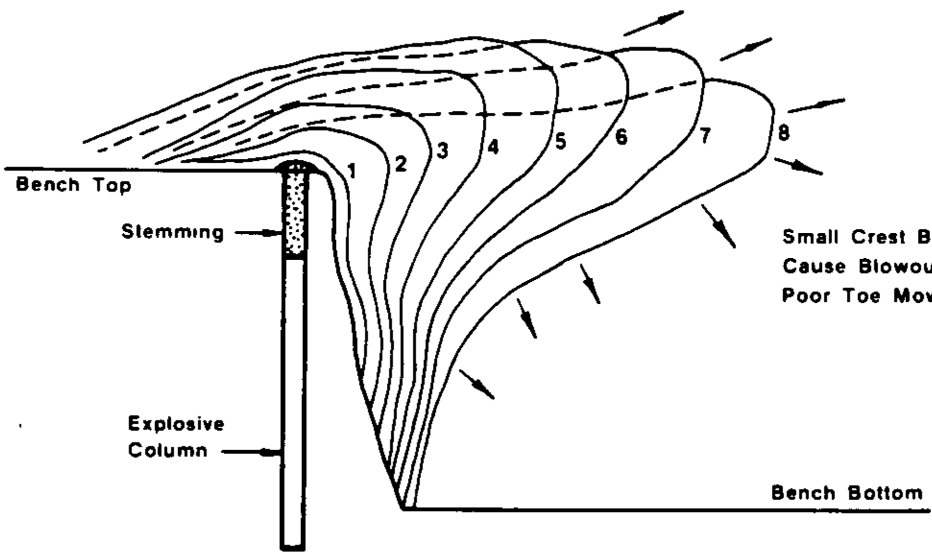


FIGURE 11.7 (Cont'd)

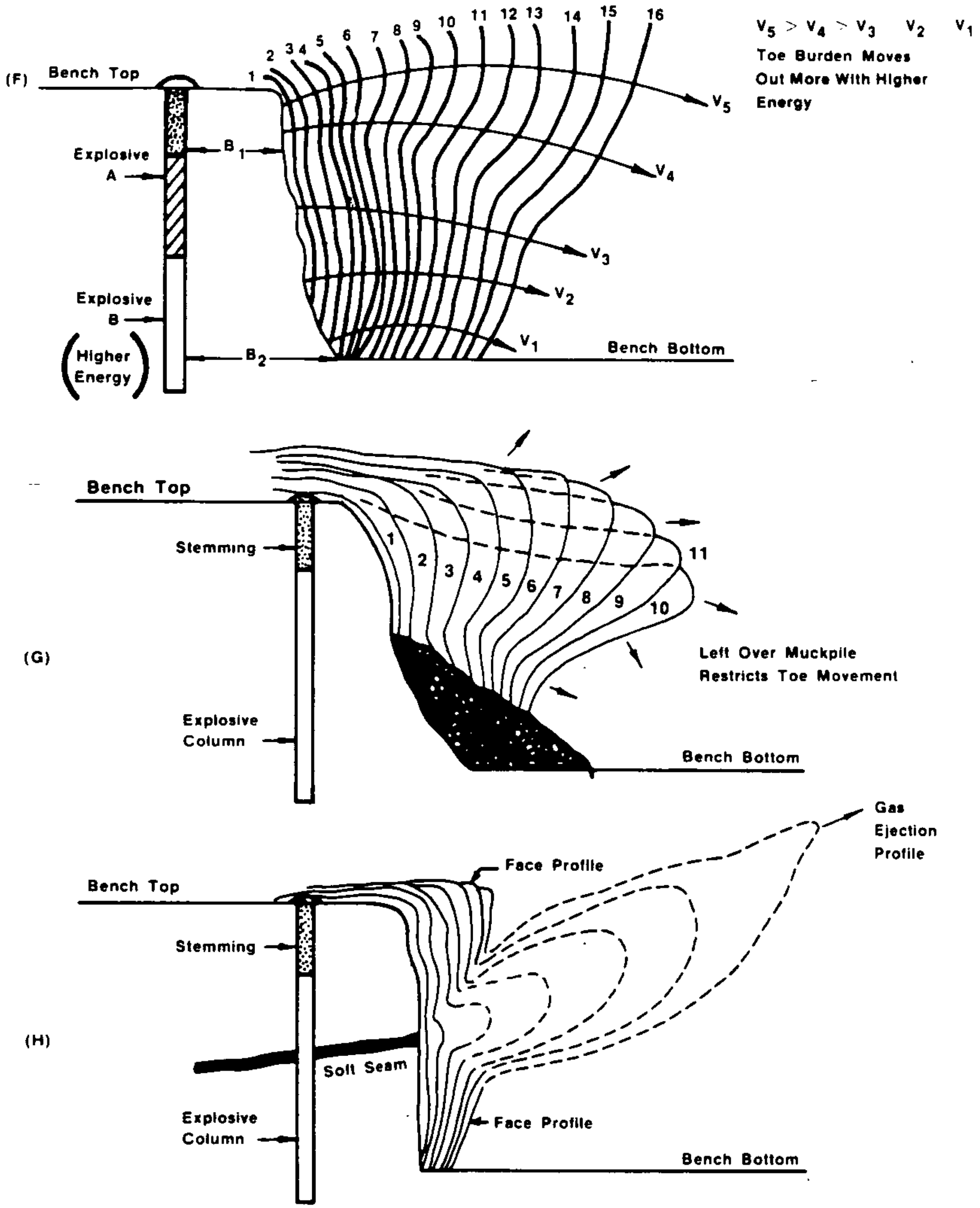


FIGURE 11.7 (Cont'd)

In selecting the latter, care should be exercised so as not to decrease the burden to the point of obtaining the condition shown in Figure 11-7e The toe burden is now correct for the explosive selected, but the crest burden is substantially reduced This may bring about many adverse conditions near the crest burden such as flyrock, blowouts, and increased airblast complaints Because confinement pressures are released near the crest (in this case, a path of least resistance relative to the toe burden), restricted toe movement will result It is better to use the same burden, but with a higher energy bottom charge near the toe This load configuration as shown in Figure 11-7f tends to pressurize more of the burden mass for longer periods without adverse effects, and adequate toe movement generally results

Where large leftover muckpiles are left against the face, Figure 11-7g, toe movement will be restricted and increased ground vibration levels are likely. Unless the situation requires a buffer, such as when blasting in the vicinity of mining equipment or to avoid dilution of an ore blast adjacent to a waste muckpile, it should be avoided

Where seams are encountered in a blast, Figure 11-7h, tremendous gas ejections with velocities up to 600 ft/sec can occur. When such gas venting occurs, it will adversely affect other parts of the burden to displace adequately and inevitably leads to poor overall blasting results. A stemming deck immediately adjacent to the seam will give better results.

e. TIME EVENTS T1-T4 COMBINED

Up to this point, time events T1 to T4 have been discussed more or less as separate isolated events. However, in a real blasting environment, more than one event can occur at the same time.

Consider a single vertical hole in a quarry face with the primer located near the bottom of the hole as is illustrated in Figure 11-8 Assume the explosive used is 40 feet of ANFO with a velocity of detonation equal to 13,000 ft/sec, the material blasted is limestone with a sonic wave velocity of 15,000 ft/sec and a density of 2.3 g/cc Upon initiation of the primer, it takes only a few microseconds and a distance of 2 to 6 hole diameters up the column to form a full detonation head. When a full detonation head is formed, it travels up the explosive column with a velocity characteristic of the steady-state velocity, (in this case 13,000 ft/sec). It takes approximately 3.0 ms for the 40 foot column of ANFO to be completely detonated.

Within this 3.0 ms, many other things have occurred. Starting at the bottom of the hole and progressing up the column, borehole

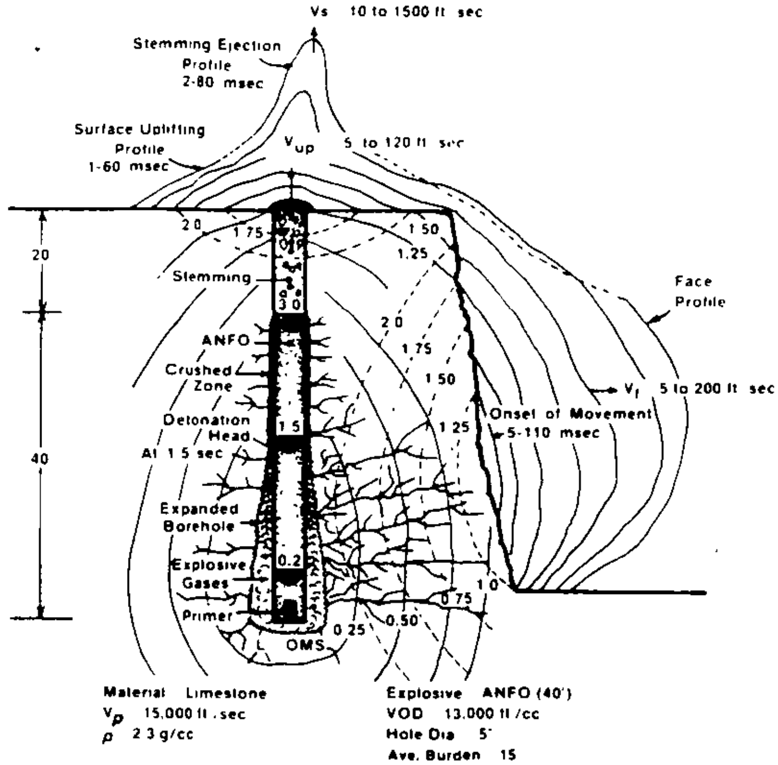


ILLUSTRATION SHOWING THE INTERACTION OF TIME EVENTS T1 TO T4 IN A TYPICAL QUARRY BENCH

FIGURE 11.8

expansion through crushing of the borehole walls has taken place. This produces compressive stress waves with tangential components emanating from the borehole walls and progressing outward in every direction with a velocity characteristic of the sonic wave velocity of limestone. It takes approximately 1.0 msec for the compressive strain wave to transverse 15 feet of burden to the free face. Behind the strain wave propagation some radial cracks start to develop in the crushed zone region of the borehole with a velocity ranging from 25 to 50% of the P-wave velocity for limestone. If the intensity of the compressive strain pulse is high enough, new cracks and/or extensions of pre-existing cracks and flaws can be initiated anywhere between the crushed zone next to the borehole and the free face. The greatest number of cracks are generally found closest to the borehole.

When the compressive wave strikes a free face, it is immediately converted to a tensile strain wave which starts at the free face and travels back through the rock mass towards the borehole. Owing to the new fractures created from the outgoing compressive strain wave. the tensile strain wave will take somewhat longer to travel the same burden distance of 15 feet If the burden is small enough and the intensity of the reflected strain wave is large enough, then some spalling at the free face or bench top is expected, although no significant mass movement will occur

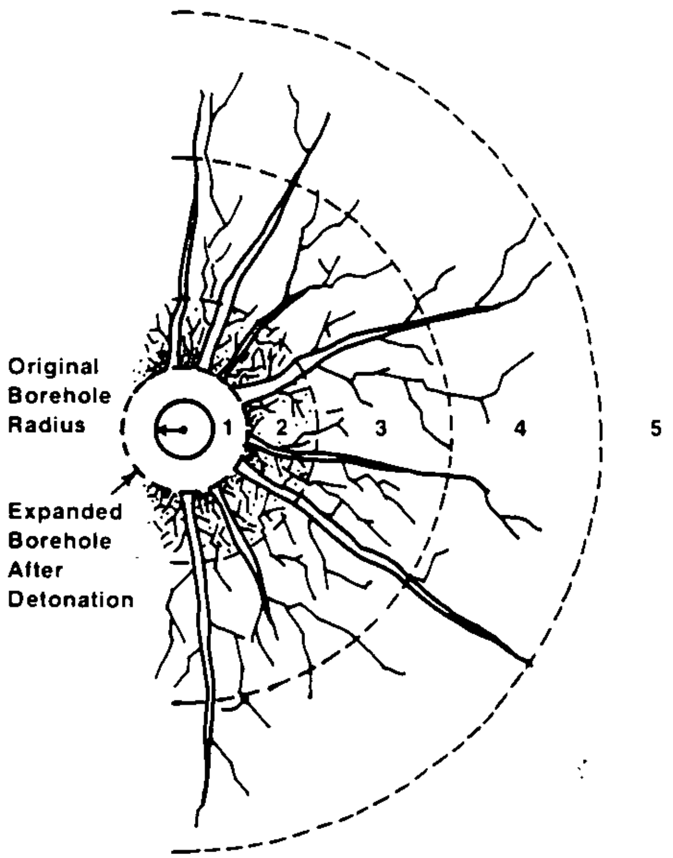
At 3 ms after detonation and complete reaction of ANFO, the original high temperature, high pressure gases have reached a new equilibrium due to borehole expansion. Both temperature and pressure have dropped significantly resulting in an energy reduction ranging from 25 to 60% of the theoretical energy originally available This remaining energy acts on the surrounding "preconditioned" rock mass to displace it in the direction of least resistance Further fragmentation can occur at this stage from gases entering and extending preexisting cracks or discontinuities It is at this stage where some blasting theories are contradictory Some believe that the major fracture network is completed within about 3 ms due to the interaction of stress waves on the surrounding material, while others believe that the major fracture network is just beginning.

Regardless of which time frame is responsible for the development of a fracture network, mass movement and displacement of material at the bench top or face occurs much later in time due to the confinement of gas pressure within the rock mass The onset of mass movement depends on the material response in conjunction with the strain and gas pressure stimulus generated from the explosive For typical stemming and burdens encountered in the field, bench top swelling occurs between 1 to 60 ms, stemming ejections between 2 to 80 ms and bench burdens between 5 to 110 ms. Surface uplifting velocities around the collar region of a hole occur between 5 and 120 ft/sec, stemming ejections between 10 to 1500 ft/sec and burden velocities between 5 to 200 ft/sec. Gas ejection velocities at discontinuities have been recorded as high as 700 ft/sec and often occur in less than 5 ms.

3. RUPTURE RADIUS

The degree of damage and fracturing around a borehole can be characterized by four zones as illustrated in Figure 11-9. In the crushed zone immediately around the borehole, the explosive induced pressures and stresses exceed the dynamic compressive strength of the rock by factors ranging from 40 to 400. These high pressures acting against the borehole wall will crush, pulverize and shatter the surrounding rock mass, causing intense damage This zone is also referred to as the hydrodynamic zone in which the elastic rigidity of the rock becomes insignificant. (6)

Next to the crushed zone is a region defined by a severely fractured zone referred to as the non-linear zone. Here fracturing can range from severe crushing through partial fracturing, to plastic deformation Extension



Legend

1 Crushed Zone

2 Severely Fractured Zone

3 Moderately Fractured Zone

4 Least Fractured Zone

5 Rock Undamaged

ZONES OF RUPTURE RADIUS
FIGURE 11.9

of cracks can occur from previously formed cracks by the tangential component (hoop stress) of the shock wave, infiltration of gas pressure and at flaw sites.

In zones 3 and 4 (elastic zones) tensile failures and crack extensions occur in a less intense mode because the stress wave amplitude has attenuated significantly. Much of the original energy from the detonation has been consumed in the form of heat, friction, and fracturing in zones 1 and 2. The peak amplitude of the compressive stress is now much smaller than the compressive strength of the rock so no new fractures are likely in this wave type. However, the tangential stress component of the wave is still substan tially larger than the tensile strength of the rock. Since the tensile strength of rock is about 1/15 to 1/10 of the compressive strength, the tangential stress of the wave is large enough to cause radial fractures These new fractures are formed from the extension of cracks in the non-linear zone (zone 2) or from cracks initiated from microfractures and flaws inherent in a typical rock mass.

Once the tangential stress has attenuated below the critical tensile strength of the rock, no further breakage occurs beyond this point as illustrated in zone 5 (Figure 11-9) Once the wave or disturbance passes into and through this zone, the individual particles of the medium will oscillate and vibrate about their rest positions within the elastic limits of the rock and so no permanent damage results It is this region where seismic waves are carried considerable distances and are responsible for ground vibrations.

Table 11-2 gives an idea of the degree of maximum damage found around the crushed and fractured zones in terms of charge radii for a number of conditions Results are based on the works of many researchers, conducted in a number of different materials with varying explosives For a given explosive, the rupture radius is greater in soft rock than hard rock Given the same rock, the rupture radius is greater for higher strength explosives than lower strength ones. Thus, the degree of radial rupture is influenced by the explosive, material properties and structure

TABLE 2
DEGREE OF DAMAGE AROUND A BOREHOLE IN TERMS OF CHARGE RADII

SOURCE	EXPLOSIVE	EXPLOSIVE AMOUNT	CHARGE SHAPE	MATERIAL OK ROCK TYPE	CRUSHED ZONE IN CHARGE RADII (MAX)	RADIUS OF DAMAGE IN CHARGE RADII (MAX)	COMMENTS
Olsen (7)	C4	0 25 kg	S	Granite	—	18	
		2 00 kg	S	Granite	—	20	
Siskind (8)	60% Dynamite	—	C	Shale	—	45-55	
	ANFO	—	C	Shale	—	15-22	
Cattermole (9)	60% Dynamite	—	C	Tuffaceous Pyroclastic	3 0	20-30	
Colorado (10)	—		-	Soft Rock		26-29	
School of Mines			-	Hard Rock	-	20 23	
Derlich (11)	Nuclear (TNT) -		—	Granite	1 9	4 9	
Atchison (12)	—	3 6 kg (max)	C	Granite	3-4 5	—	
D'Andrea (13)	C4	0 00 216 kg to 0 467 kg	S	Granite	2 3	—	
Siskind (14)	ANFO	—	C	Granite	—	14	
Kutter et al (6)	Underwater Spark Discharge	—	S	Plexiglass & Rock	—	6	Theoretically Calculated
			C	Plexiglass & Rock	—	9	Theoretically Calculated
Vovk et al (15)	-	—	—	Granite Limestone & Concrete	8-12	30-50	
Borg (16)	Nuclear			Competent	2 7-3 5	-	Horizontal Fracturing
					2 0	-	Below Shot Point

4. BLASTING THEORIES (Past & Present)

In this section, blasting theories of the past and present are discussed in concept form. Table 11-3 is a list of some of the more common thoughts regarding breakage mechanisms and the researchers responsible for their introduction. This list is by no means complete but it does illustrate how certain thoughts on blasting theory started with the simple reflection theory after World War II and progressed to the more complex nuclei or stress-wave flaw theory of the present.

Since each theory has inherent strengths and weaknesses, the main concepts of each theory are best explained with a brief description. Blasting theories discussed are:

a) Reflection Theory (Reflected Stress Waves)
b) Gas Expansion Theory
c) Flexural Rupture
d) Stress Waves & Gas Expansion Theory
e) Stress Waves, Gas Expansion & Stress-Wave/Flaw Theory
f) Nuclei or Stress-Wave/Flaw Theory
g) Torque Theory
h) Cratering Theory
i) Cratering Mechanisms

a. REFLECTION THEORY (Reflected Stress Waves) (17, 18, 19, 20)

One of the first attempts to explain, analytically, how rock breaks when a concentrated explosive charge is detonated in a borehole near a free surface was with the reflection theory. The concept was simple, straight forward, and based strictly on the well known fact that rock is always less resistant in tension than in compression. A compressive strain pulse is generated by the detonation of an explosive charge, moves through the rock in all directions with a decaying amplitude, and is reflected only at a free surface. At the free surface, the compressive strain pulse is converted into a tensile strain pulse that progresses back to its point of origin. (See Figure 11-10) Since rock is weakest in tension, it is easily pulled apart by the reflected tensile strain pulse and damage at the face appears in the form of spalling. The high pressure, expanding gases, are not deemed directly responsible for the major degree of fracturing that occurs.

A more detailed explanation follows: Detonation of an explosive charge in rock generates a large quantity of high temperature, high pressure gas in a very short time. Typically, this occurs in a few microseconds for small cylindrical charges and in a few milliseconds

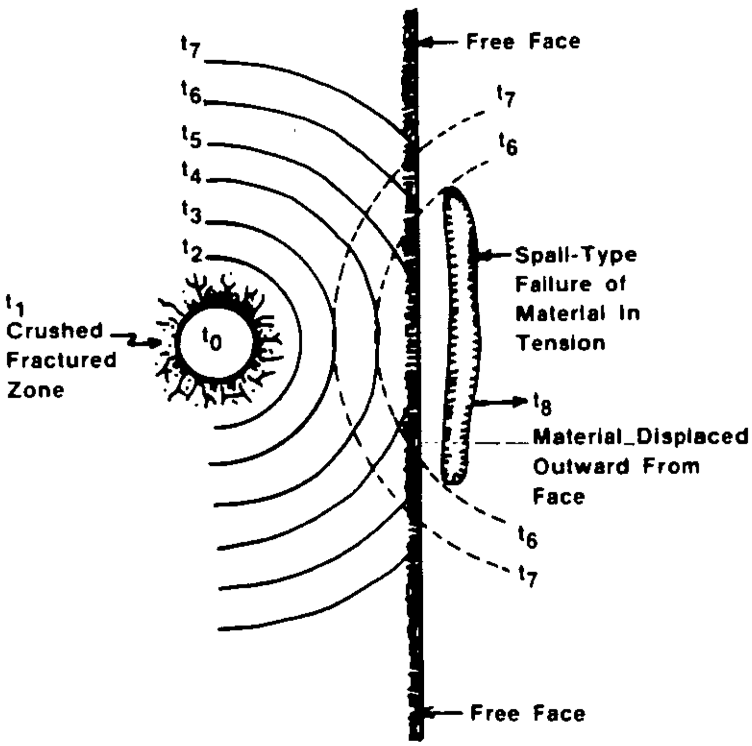
TABLE 11.3
BLASTING THEORIES AND
THEIR BREAKAGE MECHANISM

DATE	RESEARCHER(S)	BREAKAGE MECHANISMS				
		TENSILE REFLECTED WAVES	COMPRES-SIONAL STRAIN WAVES	GAS PRESSURE	FLEXURAL RUPTURE	NUCLEI STRESS-FLAW
1949	Obert, Duvall (17) (18)	1				
1956	Hind (19)	1				
1957	Duvall, Atchison (20)	1				
1958	Rinehart (21)	1				
1963	Langfors, Kihlstrom (22)		2	1		
1966	Starfield (23)	1				
1970	Porter, Fairhurst (24)		2	1		
1970	Persson, Lunborg, Johansson (25)		1			
1971	Kutter, Fairhurst (6)		1	1		
1971	Field, Ladegarrd - Pederson (26)		1	1		
1972	Johansson, Persson (27)	2		1		
1972	Lang, Faureau (28)	4	2	1		3
1973	Ash (29)			1	1	
1974	Hagan (30) (31)		1			
1978	Barker, Fourney, Dally (32) (33) (34)					1
1983	Winzer, Anderson, Ritter (35)					1
1983	Adams, Margolin (36) (37)					1
1983	McHugh (38)					1

for long cylindrical charges found in normal bench blasting. This gas pressure acting against the borehole wall generates a compressive strain or stress pulse of high amplitude which will crush and/or fracture rock next to the borehole. This stress pulse travels radially outward in all directions from the shot point at speeds equal to or greater than the velocity of sound in the medium. Due to wave divergence and energy absorption by the rock, the pulse amplitude decreases very rapidly. Thus, the extent of the crushed zone immediately next to the borehole is relatively small.

When a longitudinal compressive stress strikes a free surface, two reflected pulses are generated, a tensile and shear pulse The amount of energy imparted to each depends on the angle of incidence of the compressional stress pulse. Of the two reflected pulses, the tensile one predominates in breaking rock as it moves back into the rock.

The effective transfer of detonation pressure to stress in the rock depends on the impedance match of the explosive to rock. A smaller explosive to rock impedance ratio was shown to provide a more effective transfer of this pressure to stress. The concept of reflection breakage is illustrated in Figure 11-10. The time order of key events are:



t_0 — detonation generation of high pressure, high temperature gases

t_1 — borehole walls are crushed and slightly fractured due to high gas pressure, and borehole expands

t_2-t_4 compressional strain pulse propagates outward in all directions

t_5 - part of compressional strain pulse impinges on free surface

t_5-t_6 — part of pulse continues to travel outward and part of it is reflected at the free surface as a tensile strain pulse

— slab of rock begins to detach from free face and moves forward

t_7+ - other compressive stress pulses arrive at the newly formed face and repeats breaking process

RELECTION THEORY
TENSILE FRACTURE BY RELECTION
OF A COMPRESSIVE STRAIN
PULSE AT A FREE SURFACE

FIGURE 11.10

Slabs broken off closer to the hole are displaced with lower velocities.

b. GAS EXPANSION THEORY (25) (39)

The pressure acting on the walls of an explosive filled hole. upon detonation. will be approximately one-half of the detonation pressure due to expansion of the borehole This pressure will propagate out from the borehole into the rock as a shock wave The material between the borehole and the shock front is compressed and flows elastically or plastically. depending on the pressure and strength of the rock Some radial cracks form next to the borehole wall starting at about two hole radii out and then propagate radially inwards as well as outward. The greatest frequency of radial cracks are next to the borehole. but a few extend farther out When no free face exists. a small number of these radial cracks become very much larger than the others.

By the time the shock wave reaches the free surface. radial crack lengths formed are less than one quarter of this distance. At this stage the longest of cracks have extended inwards and reached the borehole wall. Gas pressure is now capable of entering these cracks and if the pressure is high enough can reach out towards the crack tips. thus further elongating the cracks. This has the effect of aiding cracks that interact with the returning tensile wave and cause them to reach the free surface Up to this point. acceleration of the rock mass between the hole and free face has been negligible. Only after the cracks have reached the free surface is the rock accelerated by the remaining gas pressure

The key point of the gas expansion theory are:

- Radial cracks are initiated not immediately next to the borehole but about two hole radii out and extend inwards toward the hole as well as outwards towards a free face.

- Rock displacement does not occur until pressurized radial cracks extend to the free surface.

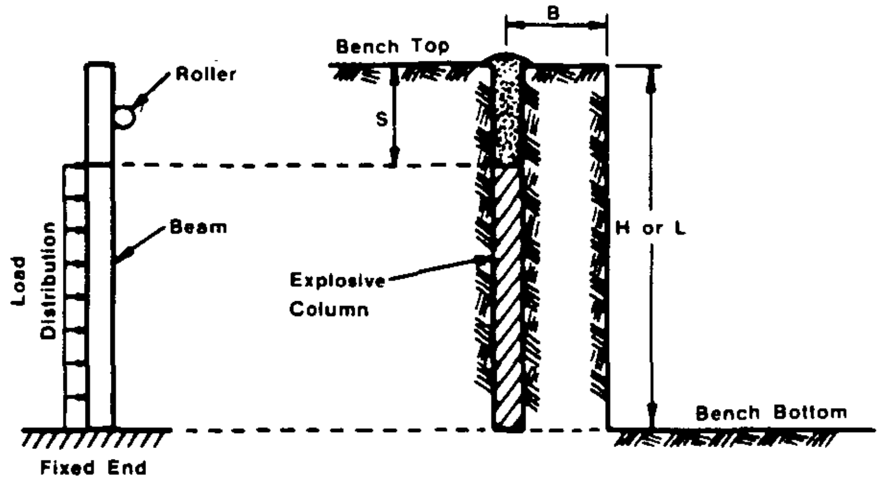
c. FLEXURAL RUPTURE (A Gas Expansion Theory) (29)

During detonation of an explosive confined in a borehole. two distinct pressures are formed: one from the detonation itself and the other from the highly heated gases acting on the borehole walls In this theory, ninety percent of the total energy to break rock is in the latter Detonation pressure acts only momentarily against any one part of the borehole's internal surface area, while gas pressure is sustained considerably longer until some form of cavity volume change occurs. Gas pressure, then, is the major component responsible for fragmentation and flexural rupture.

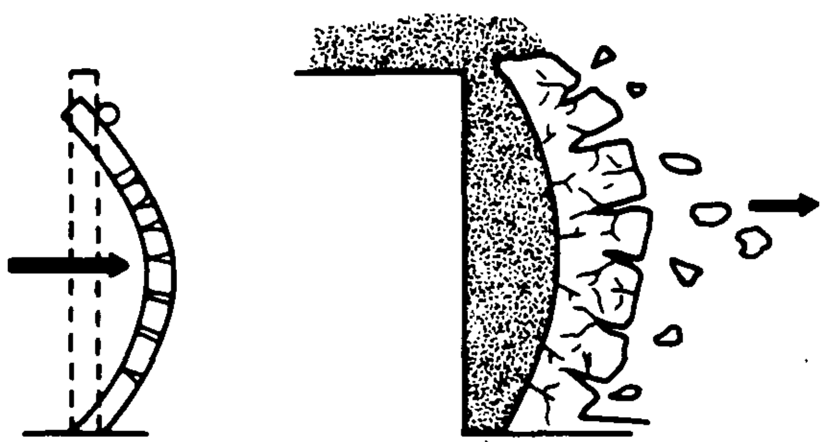
Radial cracks form only in planes parallel with the borehole axis. No cracks develop where the explosive is not in immediate contact, thus most cracks form adjacent to the borehole wall where tangential stresses are produced within the borehole's wall as the cavity is pressurized. Providing strain energies at crack tips are adequate, extension of fractures continue. Breakage by reflection of strain energy at a free face is considered negligible. Gas pressure drives the radially produced cracks through the burden to the free face and displaces rock through bending and in the direction of least resistance generally following naturally occuring planes of weakness It is during this final stage where the major breakup of intact material takes place.

Breaking of rock by flexural rupture is analogous to bending and breaking a beam as illustrated in Figures 11-11 and 11-12. A rectangular beam is used to represent the field configuration of bench height, H, and burden, B, in the form of a modified cantilever beam model. The fixed end of the beam represents toe conditions while a roller, placed directly opposite the center of the stemming column, represents the stemming function. The roller allows the collar region to rotate and move longitudinally but does not allow deflection normal to the borehole axis. Although not shown for clarity of concept, the beam thickness in Figures 11-11 and 11-12 is actually equal to the burden. Borehole pressure is represented as a load distributed along the length of blasthole containing the explosive. Rock weight of the bench segment is considered negligible relative to the load resulting from the borehole gas pressure. Maximum contribution of total rock load acting at floor level is only at a ratio of about 1:100,000 or more compared to gas pressure.

The degree of fragmentation is controlled by the stiffness property of the burden-rock mass. This stiffness depends on existing restraints to movement, rock (Young's modulus), radially-cracked block's geometric shape as defined by its average thickness, width, and length. In terms of blast configuration, burden, spacing, and bench height are the controlling factors for any given rock.



BEAM BENDING MODEL BEFORE DETONATION
FIGURE 11.11



BEAM BENDING MODEL AFTER DETONATION
FIGURE 11.12

To achieve adequate flexural rupture, the burden to length (B/L) ratio becomes critical because stiffness varies with the third power of this ratio For a given explosive diameter and reflective B value, decreasing the bench height L has the effect of,

i) stiffening the burden rock

ii) reduces fragmentation

iii) inhibits the necessary lateral and upward displacements needed to break collar material and remove toes

Reducing burden for a given bench height has the opposite effect Doubling the bench height for a given burden, or reducing the burden by one-half for a constant bench height has the effect of reducing the stiffness theoretically some eight times, although in practice a B/L ratio of 1/3 is often adequate.

d. STRESS WAVE AND GAS EXPANSION THEORY (6)

In 1971, Kutter and Fairhurst combined the concepts of strain wave induced fracturing and gas pressure as the main mechanisms to fragment rock Their experiments were performed with homogenous plexiglass and rock models.

After detonation, an intense pressure wave is emitted into the rock from the impact of the rapidly expanding high pressure gas. This pressure rises immediately to its peak and can be assumed to be one-half to one-quarter of the detonation pressure. Due to cavity expansion around the borehole and the cooling of the gases, the pressure decays exponentially. In spite of the decay, the pressure is sufficient to exert a quasi-static pressure on the rock boundary for a relatively long time.

The amount of energy in the shock wave is calculated to be only a fraction of the total energy released by the explosive. In granite this was measured to range between 10 to 18 percent while in salt it was only 2 to 4% of the total energy released. The remaining energy is contained in the gas pressure. However, the compressional wave energy is sufficiently high next to the borehole to cause extensive breakage.

A radially fractured zone is the first fracture pattern to develop around the new expanded cavity. Next to develop is a ring of wider spaced radial cracks.

This width of radially fractured zone depends on:

- the tensile strength of the rock
- wave velocity of the rock
- input pressure of the explosive
- detonation velocity of explosive
- extent of energy absorption in the rock mass

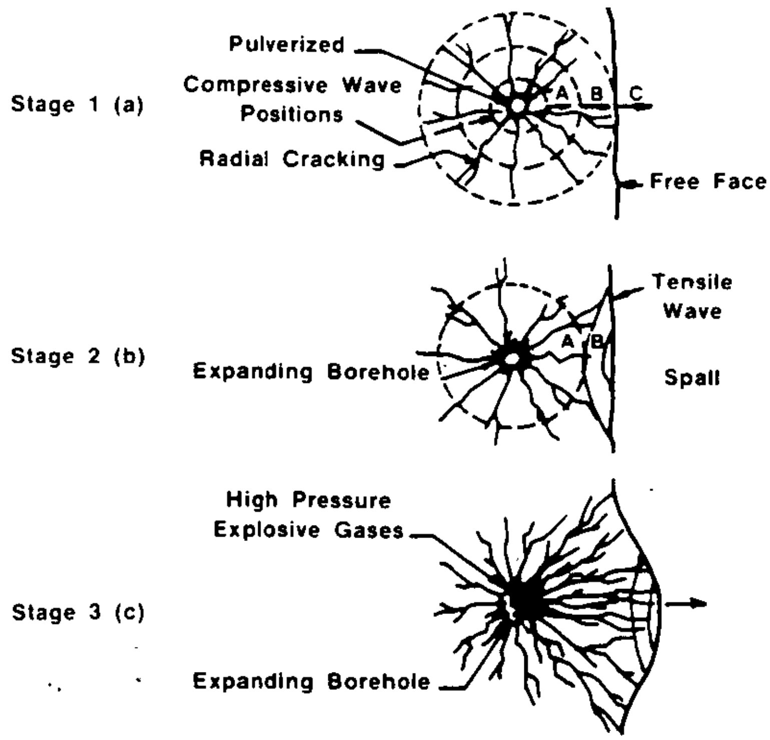
The diameter of the fractured zone was theoretically calculated to be around six hole diameters for a spherical charge and nine hole diameters for a cylindrical charge. It is in this expanded or equivalent cavity that the gas pressure becomes active and not in the original borehole. Thus cracks are pressurized and free to extend toward a free face. The original stress wave functions only to precondition the rock by initiating (in tension) radial cracks at the borehole wall

The main points of interest of the stress wave and gas expansion theory are:

- Both stress waves and high pressure gases play an important role in fragmenting material. Neither the strain wave or gas pressure alone is responsible for rock fragmentation in blasting.
- Radial cracks originate at the borehole wall.
- Pre-existing cracks would reinitiate under stress, but no new cracks would form in the area occupied by an old crack.
- Presence of a free surface favors extension of gas pressurized radial cracks in that direction.
- In-situ stresses affect the direction in which radial cracks travel.
- For a given borehole size, an increase of explosive charge beyond an optimum amount does not increase the fractured zone, but results only in additional crushing around the cavity.

e. **GAS EXPANSION, STRESS WAVES, STRESS-WAVE FLAW, AND REFLECTION** — (Combined Theory) (28)

Stage 1—On detonation of the explosive the high pressure shatters the rock in an area adjacent to the drill hole. The outgoing shock wave traveling at 9,000 to 17,000 feet per second sets up tangential stresses that create radial cracks which move out from the region of the hole. The first radial cracks develop in one to two milliseconds. (Figure 11-13a)



FRACTURES OPENED UP AND PROPAGATED BY GAS EXPANSION PRODUCING AN ISOLATED FRAGMENTED ROCK MASS OR CRATER
FIGURE 11.13

Stage 2—The pressure associated with the outgoing shock wave of the first stage is positive If the shock wave reaches a free face it will reflect, but in so doing the pressure falls rapidly to negative values and a tension wave is created. This tension wave travels back into the rock and since this material is less resistant to tension than to compression, primary failure cracks will develop due to the tensile strength of this reflected wave. If these tensile stresses are sufficiently intense they may cause scabbing or spalling at the free face. (Figure 11-13b)

In rock breaking this spalling effect appears to be of secondary importance It has been calculated that the explosive load must be in the order of 8 times the normal load to cause failure of the rock by reflected shock wave alone.

In the first and second stages, the function of the shock wave energy is to condition the rock by inducing numerous small fractures In most explosives the shock wave energy theoretically amounts to only 5 to 15% of the total energy of the explosive. This strongly suggests that the shock wave is not directly responsible for any significant amount of rock breakage. but it does provide the basic conditioning for the last stage of the breakage process.

Stage 3—In this last stage the actual breakage of rock is a slower action. Under the influence of the exceedingly high pressure of the explosion gases. the primary radial cracks are enlarged rapidly by the combined effect of tensile stress induced by radial compression and by pneumatic wedging. When the mass in front of the borehole yields and moves forward. the high compressive stresses within the rock unload in much the same way as a compressed coil spring being suddenly released. The effect of unloading is to induce high tension stresses within the mass which complete the breakage process started in the second stage. The small fractures and threshold fracture conditions created in the second stage serve as zones of weakness to initiate the major fragmentation reactions. (Figure 11-13c)

f. **NUCLEI OR STRESS WAVE—FLAW THEORY** (32. 33. 34. 35. 37. 38)

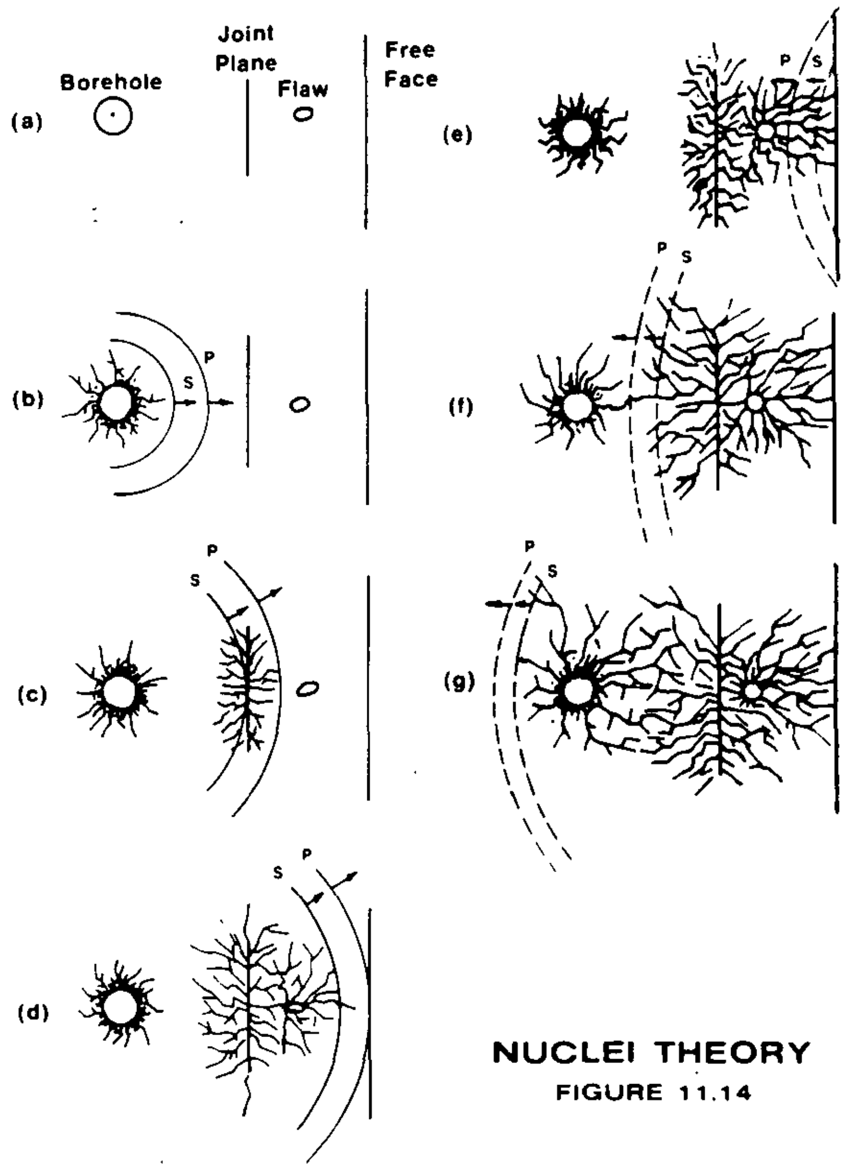
This relatively new theory was formulated at the University of Maryland in the fracture mechanics laboratory. Laboratory tests were conducted in homolite-100 models. both unflawed and flawed. by simulating many of the geologic structures and discontinuities (joints. fractures. bedding planes) typically found in large scale bench blasting. Results showed that stress waves were quite important in the fragmentation process and caused a substantial amount of crack initiation at regions rather remote from the borehole. These regions consisted of small or large flaws. joints. bedding planes. and other discontinuities that acted as a nuclei for crack formation. development or extension. This new stress wave dominated mechanism of fragmentation is referred here as the nuclei theory.

The theory and actual mechanisms of stress wave propagation and interaction in a flawed medium are quite complex. They involve many phases such as: (40)

- detonation and crack nucleation around borehole
- crushed zone extension
- dynamic crack stability
- activation of flaws
- coalescence of wave velocities and strains
- branching of cracks
- interaction of cracks and reflected wave systems
- instability of crack direction
- random progressive failure

In more simple terms, the important points of the theory are explained with the illustration in Figure 11-14. A borehole is located behind a free face with two discontinuities, a joint plane and a small flaw, located between the borehole and free face. Assume all other areas in the medium to be homogeneous and flaw free.

In unflawed material, only 8 to 12 dominant cracks emerge from a dense radial network around the borehole. These dominant cracks can travel significant distances and consequently form large pie shaped segments, that alone are not conducive for good fragmentation. Stress waves continuing away from the fractured zone around the borehole result in no further damage.



NUCLEI THEORY

FIGURE 11.14

In flawed material or sections of the material which contain flaws. fragmentation is quite different Consider the P and S waves propagating away from the fracture network around the borehole in Figure 11-14b and 11-14c.. Refer to Chapter 12—Vibration/AirBlast section for a discussion on Seismic Waves. No fracturing takes place until the flaw (joint plane) is initiated by the P wave tail and the leading front of the S wave. (Figure 11-14c). The remainder of the S wave has sufficient energy to keep the crack from arresting. A similar effect occurs as the P and S waves move past the small flaw between the joint plane and the free face. (Figure 11-14d). It is important to note that cracks are initiated at flaw sites remote from the borehole region by the combined action of the P wave tail and the S wave front Flaws initiated in the immediate borehole vicinity of these waves have only a small effect Note also. that the outward directed P and S waves can initiate flaws anywhere independent of the presence of a free surface.

When a P wave encounters a free face (Figure 11-14d and 11-14e). it is reflected and travels back into the medium as a tensile wave to meet the outcoming S wave. At this stage. constructive interference can occur which allows for further crack initiation or extension of cracks previously formed. New wave systems (PP. PS. SP. SS. PP. and S. PS. and S) will also form from the original outgoing wave system upon reflection at a free surface or discontinuity. These new wave systems can also contribute to crack extensions. Figure 11-14f and 11-14g illustrate further crack extensions when all wave systems have been reflected back towards the hole.

The important points of the nuclei or stress-wave flaw theory are:

- **the fracture network spreads with the speed of the P and S waves, which initiate fracture around flaws remote from the borehole**
- **in highly flawed material, fragmentation results from the nucleation of new cracks at flaws and reinitiation of old cracks from the reflected stress wave systems**
- **gas pressurization does not contribute significantly to the fragmentation process**

Computational models incorporating stress wave/flaw interaction as a mechanism of nucleating and extending cracks is growing in popularity. (32-38. 40) Although the models differ in approach and/or details. the main idea is that shock and/or stress waves fragment material and gas pressure acts to displace the broken material. Stress wave functions not only to initiate fractures at or near the borehole wall, but also initiate fractures throughout the rock mass being blasted.

Recent work in full scale production shots and in large blocks added further insight into this phenomena. (35) Stress wave induced fracturing at flaws and discontinuities removed from the borehole was found to be considerably greater than either spalling or borehole radial tensile failure documented by earlier works. Gas pressurized radial fracturing, in typical bench blasting operation, was found to be only a minor contributor to the overall fragmentation of the rock mass.

Some key points of Winzer's theory and observations are:

i) new fractures are seen to form at the face at about twice the time it takes for the P wave to traverse the burden distance

ii) old fractures are the loci of new fractures or are re-initiated themselves early in the event; they continue to be active for several tens of milliseconds after detonation of the explosive

iii) fragmentation continues in blocks of rock, following detachment from the main rock mass, by trapped stress waves

iv) the fracture pattern on the free face is well developed prior to the expected time of arrival of radial cracks from the borehole

v) in blasted faces from production-scale shots, fractures are observed to have initiated at, and propagated from, joint and bedding planes, suggesting the same operating mechanism(s) as those observed in homolite models at the University of Maryland

vi) gas venting occurs through already open cracks relatively late in the event, indicating that the majority of fractures observed on the free face are not gas pressurized

vii) in more massive rock stress waves are transmitted with higher velocity and less attenuation, but fewer fractures will form because there are few fracture sites. However, more radial fractures will form in massive rock, while fewer fractures form at a distance from the borehole

viii) large fragments will form early in the event. and as they move and fractures open. large segments of the rock mass will be effectively isolated from further stress energy

ix) in more heavily fractured rock, the stress wave velocity will be lower and attenuation higher. but there are more fractures to serve as initiation sites

x) the stress wave takes longer to penetrate the mass. and movement of the rock can be expected to be slower as more energy is absorbed by the rock mass

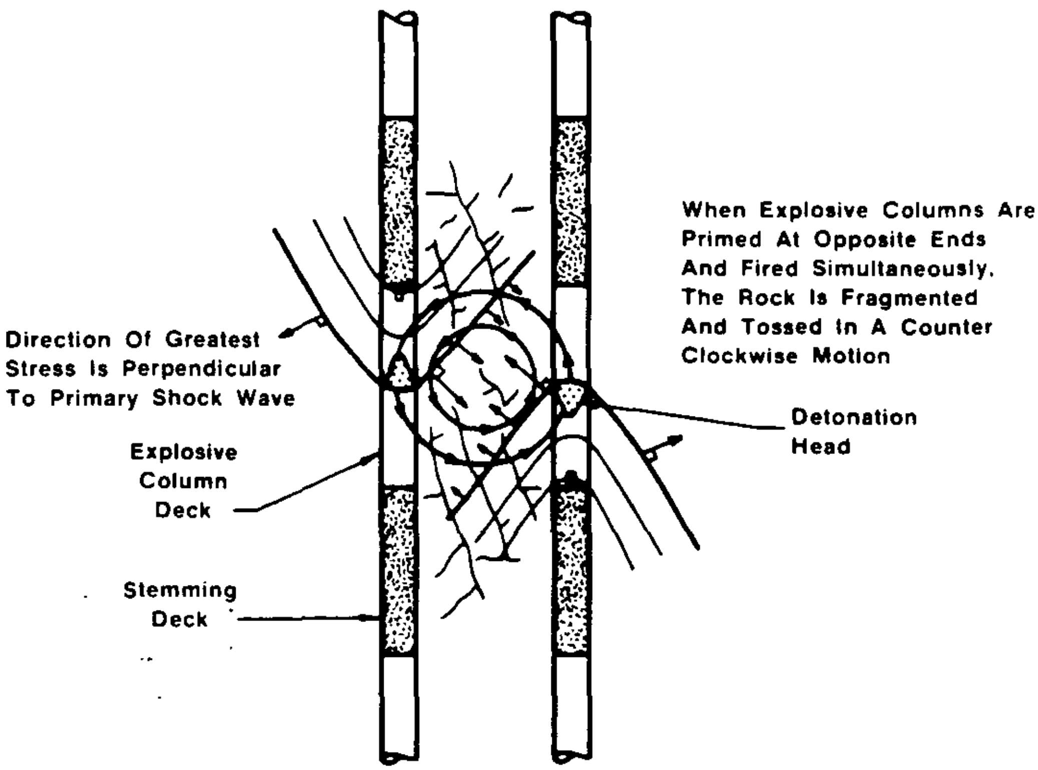
xi) cracks open more slowly, and smaller masses of rock are isolated early in the event, so that later arriving stress waves can continue to increase crack initiation and propagation

g. TORQUE THEORY

The success of this theory is totally dependent on the absolute, accurate timing of initiators. When two adjacent explosive columns are initiated simultaneously from opposite ends, a compressional shock wave from each column traveling parallel but in opposite directions is formed. (Figure 11-15) The greatest stress is always directed perpendicular to the primary shock front. This stress is also assumed to be greatest near the detonation head in the explosive and diminishes with distance away from the detonation head. An uneven stress distribution is formed between explosive columns when the columns are fired simultaneously and from opposite directions. This action tends to toss the fragmented rock between explosive columns in a counterclockwise motion. Reversing the primers of each explosive column will toss the material in a clockwise motion. This action is precisely what is needed to obtain uniform fragmentation and avoid tight muck piles such as in the case of In-situ retorting. For this theory to work, exact initiators are crucial; nothing less will do, especially when using explosives with very high velocity of detonation.

h. CRATERING THEORY (41-45)

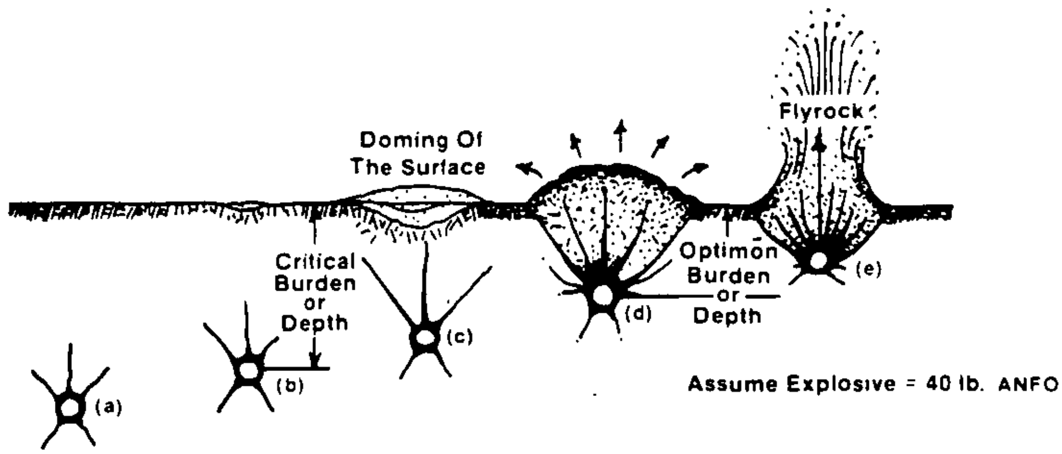
The concept of cratering, its development, and resulting applications were originally proposed by C.W. Livingston and later modified by others such as Lang and Bauer. (41) (43) (44) It involves a spherical charge of length to diameter ratio of less than or equal to 6 to 1, detonated at an empiracally determined distance beneath the sur-



APPLICATION OF NEW BLASTING THEORY TO IN-SITU RETORTS BLASTING

FIGURE 11.15

face to optimize the greatest volume of permanently fragmented material between the charge and free surface. This implies that given a specific explosive and material, there exists a burden distance between the charge and free surface which yields the largest crater (Figure 11-16d). This burden is referred to as the optimum burden or depth. Similarly, there exists another burden distance referred to as the critical distance, which is too far below the surface to result in any crater or expulsion of material at the surface, other than minor radial cracks This is the point where material at the surface just begins to show evidence of failure, (Figure 11-16b).



(a) B = 15'	(b) B = 12'	(c) B = 9'	(d) B = 6'	(e) B = 3'
Completely Contained. Only Failure is Pulverisation Near The Charge and Radial Tensile Failure Running Out From It.	Start of Surface Failure. Burden Not Broken. Some Doming of The Surface.	Surface and Subsurface Failure Almost Meet There Will Be A Shelf of Unbroken Rock Between The Two. Doming or Surface Bulging.	Full Crater. Burden Completely Broken Out. Surface And Subsurface Failures Run Through To The Surface.	Full Crater. Lower Volume Than Optimum Fine Fragmentation. Noise. Flyrock. Bowl Shaped Crater

SCHEMATIC OF THE EFFECT OF DECREASING THE BURDEN ON CHARGES FIRED IN ROCK
FIGURE 11.16

Livingston determined, experimentally and theoretically, that there was a constant factor between this critical burden distance and the cube root of the weight of explosive and expressed it as:

Strain Energy Equation

$$N = E \times W^{\frac{1}{3}}$$

where:

- N = critical distance in feet
- W = weight of explosive in pounds
- E = proportionality constant or the **strain energy factor** which has no units and is constant for one given explosive - rock combination

If a sufficient number of tests are performed as illustrated in Figure 11-16, then the strain energy factor could be calculated. For example if the critical burden was found to be 12 feet when using 40 pounds of ANFO, then

$$E = \frac{N}{W^{\frac{1}{3}}}$$

$$E = \frac{12}{(40)^{\frac{1}{3}}}$$

$$E = \frac{12}{3\ 42}$$

$$E = 3.51$$

Strain Energy Factor = 3 51

This strain energy factor, E, will differ if the same explosive is used in a different material or the same material is blasted with a different explosive. When rock gets more brittle, E increases and the optimum crater volume occurs at lower values of depth ratio. In softer material. E decreases and the optimum crater volume occurs at higher values of depth ratio.

The strain energy equation can be written in another form that relates the charge depth from surface to the depth ratio, strain energy and explosive weight as:

Upper Limit of Shock Range

$$d_c = \triangle \times E \times W^{\frac{1}{3}}$$

where:

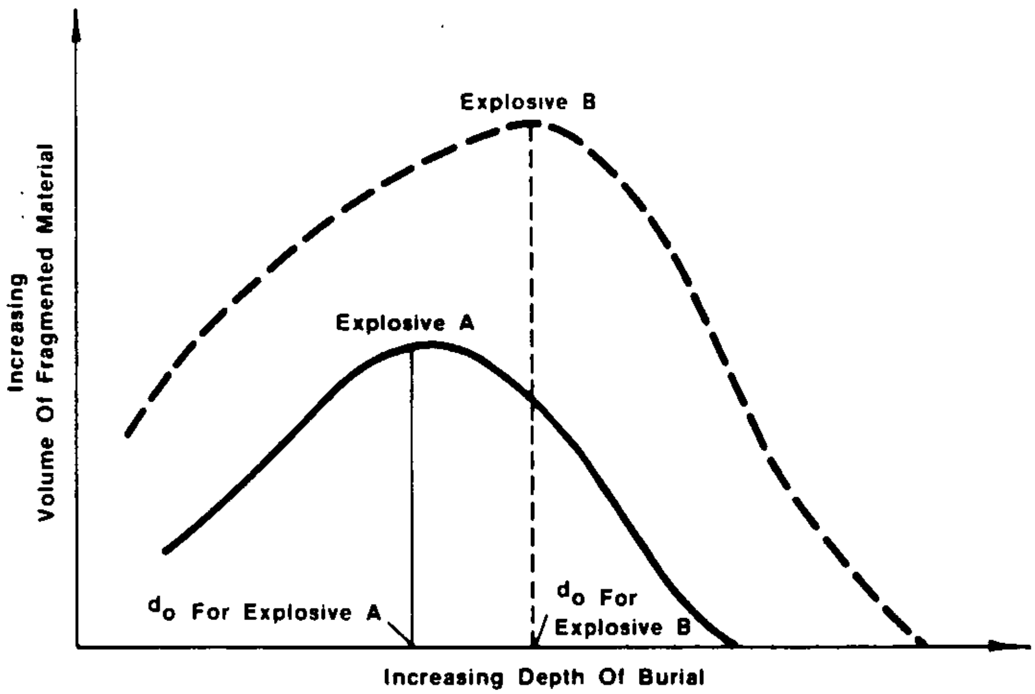
d_c = distance from surface to the center of gravity of the charge in feet

$$\triangle = \text{depth ratio} = \frac{\text{depth of burial}}{\text{critical depth}}$$

W = weight of explosive in pounds

If d_c is the optimum burden that yields the greatest volume of fragmented material, then it is referred to as d_o and the optimum depth ratio is referred to as $\triangle_o$.

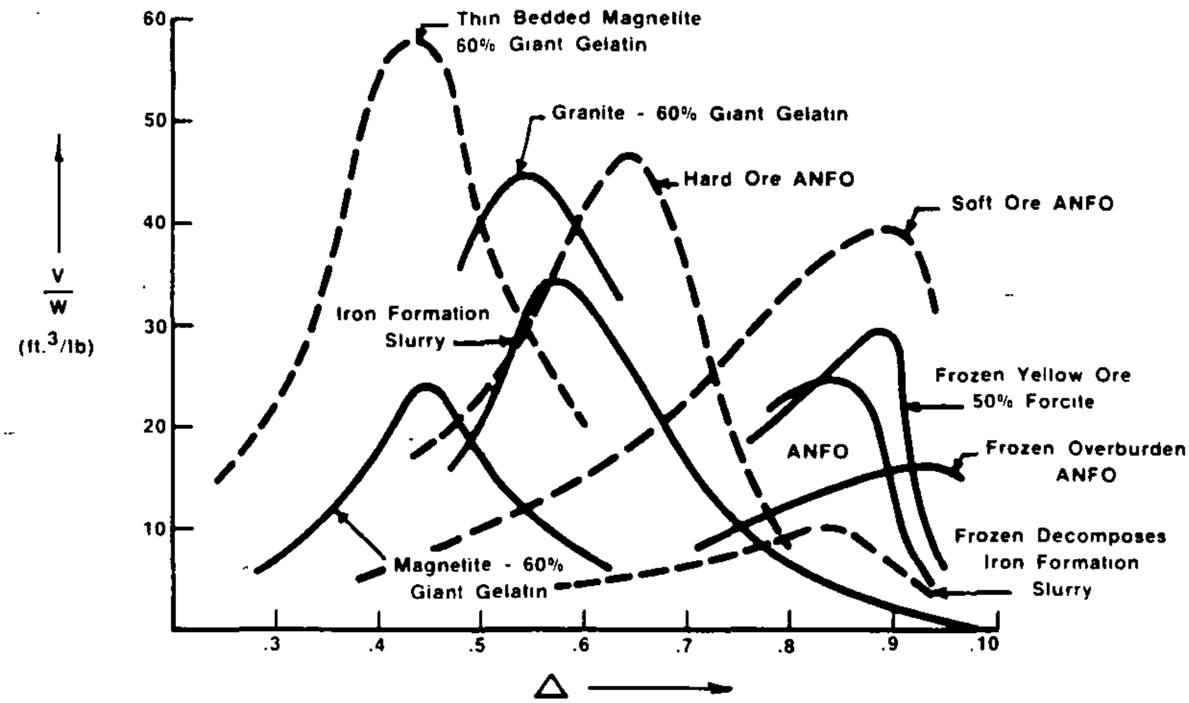
Crater data can be plotted in a number of different ways. Figure 11-17 illustrates the effect of two explosives. A and B on the amount of fragmented material that each is capable of achieving at different depths of burials Note that the higher energy explosive always fragments a greater volume of material at the same depth of burial as explosive A, but that the optimum depth of burial differs for each explosive.



VOLUME OF FRAGMENTED MATERIAL VERSUS DEPTH OF BURIED FOR TWO EXPLOSIVES IN THE SAME MATERIAL

FIGURE 11.17

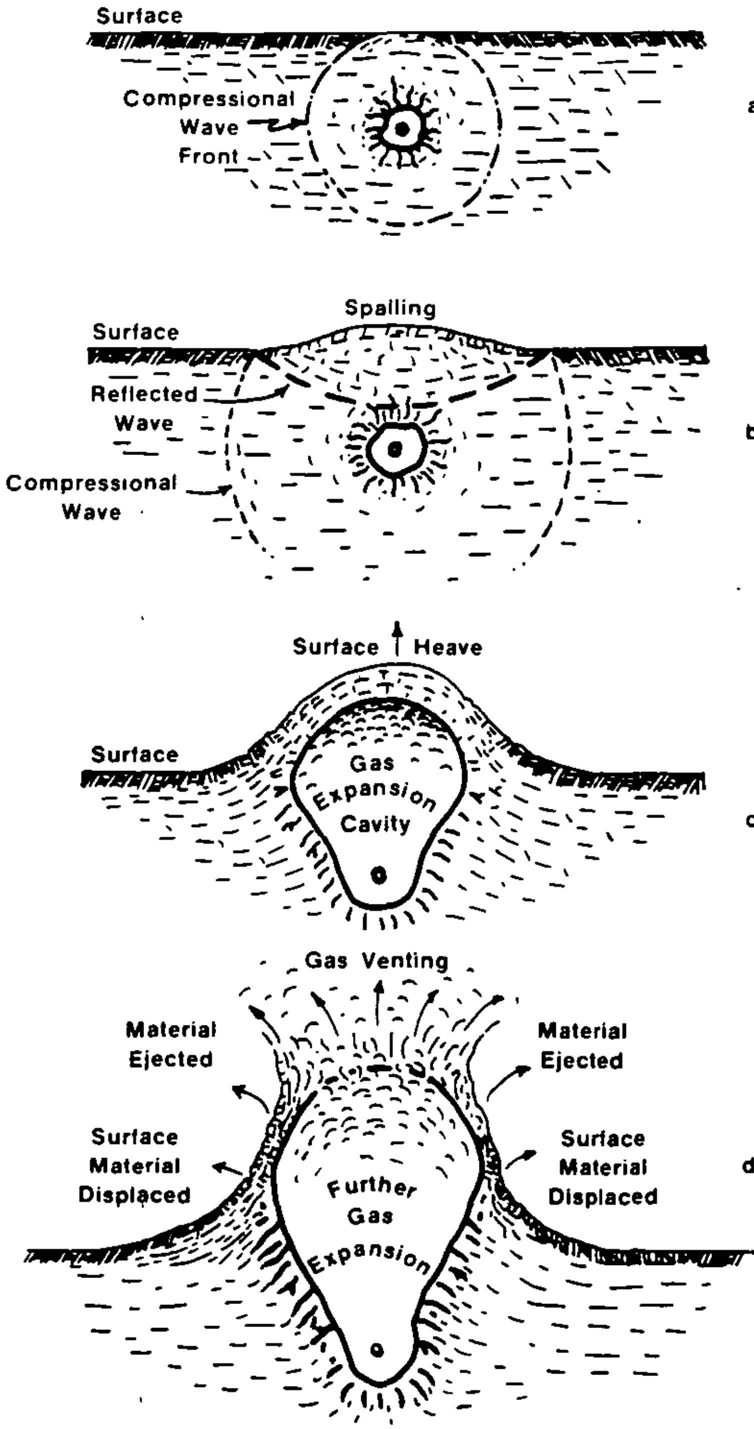
Another method of representing crater data on a common base is by plotting V/W on the y-axis and the depth ratio on the x-axis as shown in Figure 11-18. (44) V is the volume of broken material in cubic feet, W is the weight of explosive in pounds, and the depth ratio has been defined as the depth of burial divided by the critical depth The important thing to note is that the optimum depth ratio. (Δ_o), varies with each explosive-rock combination. The advantage of performing such field experiments is that one would obtain crater data specifically suited to the user environment for a number of different explosives. Although the curves in Figure 11-18 are fitted as smooth curves, one should remember that some scatter of data is always present and it is important to take this into account for crucial applications of cratering.



ROCK REMOVED IN CU. FT. PER LB. OF EXPLOSIVE VS DEPTH RATIO
FIGURE 11.18 (44)

I. CRATERING MECHANISMS (4) (45)

As the high pressure explosive gases expand against the medium immediately surrounding the explosion, a spherical shock wave is generated causing crushing, compaction and plastic deformation. (Figure 11-19a) For commercial explosives the initial shock pressures are on the order of 100 to 200 thousand atmospheres (one atmosphere = 14.7 pounds per square inch). As the shock front moves outward in a spherically diverging shell, the medium behind the shock front is put into radial compression and tangential tension. This results in the formation of radial cracks directed outward from the cavity. The peak pressure in the shock front becomes reduced due to spherical divergence and the expenditure of energy in the medium For shock pressures above the dynamic crushing strength



a) • Detonation
Crushing Around Charge
Stress Wave Reaches Surface

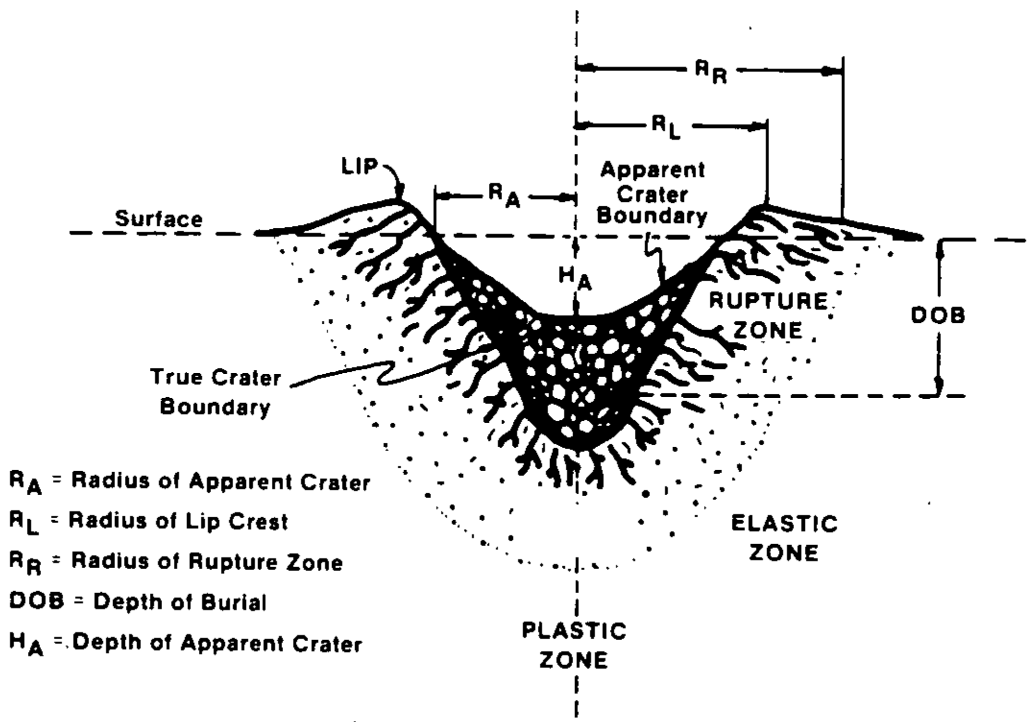
b) • Stress Wave Reflects
at Surface
Some Surface Spalling

c) • Gas Expansion And
Acceleration

d) • Gas Venting And
Material Ejection

CRATERING EVENTS AND MECHANISMS
FIGURE 11.19

of the medium, the material is crushed, heated and physically displaced, forming a cavity. In regions outside this limit the shock wave will produce permanent deformation by plastic flow, until the peak pressure in the shock front has decreased to a value equal to the plastic limit of the medium. This is the boundary between the plastic and elastic zones shown in Figure 11-20



EMPLOYMENT OF ATOMIC DEMOLITION MUNITIONS
DEPARTMENT OF THE ARMY, WASHINGTON, D.C. AUG. 1971
FIGURE 11.20

When the compressive shock front encounters a free face, it must match the boundary condition that the normal stress or pressure be zero at all times. This results in the generation of negative stress, or rarefaction wave which propogates back into the medium(Figure 11-19b). Thus the medium which was originally under high compression is put into tension by the rarefaction wave. This phenomenon causes the medium to break up and fly upward with a velocity characteristic of the total momentum imparted to it. In a loose soil material, this spalling makes almost every particle fly into the air individually, while in a rock medium the thickness of the spalled material is generally determined by the presence of pre-existing fracture patterns and zones of weakness. As the distance from surface increases, the peak negative pressure decreases until it no longer exceeds the tensile strength of the medium. The velocity of spalled material also decreases in proportion to the peak pressure. This breakage mechanism is predominant only for charges placed at very shallow depths of burial.

The two mechanisms described so far are short term, lasting only a few milliseconds. The gas acceleration mechanism, however, is a much longer lasting process which imparts motion to the medium around the detonation by the expansion of gases trapped in the explosion-formed cavity. (Figure 11-19c and 11-19d) These gases are produced in the surrounding material by vaporization and chemical changes induced by the heat and pressure of the explosion. Venting occurs because the material is no longer cohesive enough to contain the explosion gases. As the gases are released, fragments assume free ballistic trajectories. At depths of burial at which crater dimensions are maximum, the gases produced will give appreciable acceleration to overlying material during its escape or venting through cracks extending from the cavity to the surface. At shallow depth of burials the spall velocities are so high that the gases are unable to exert any pressure before venting occurs. For very deep explosions the weight of the overburden precludes any significant gas acceleration of the overlying material. Gas acceleration is the dominant mechanism at optimum depth of burial. With a constant weight of explosive, the optimum depth of burial varies with the surrounding material.

At deep depths of burial, the mechanism of overburden collapse (subsidence) becomes dominant. This effect is closely linked to the crushing, compaction and plastic deformation mechanism which produces an underground cavity. At these depths of burial, spall and gas acceleration will not impart sufficient velocity to the overlying material to physically eject it from the crater. Most throwout returns to the crater as fallback material. In a rock medium the bulking action of the rock, when it is disoriented from its original fracture pattern, could produce a volume greater than the underground cavity. This could result in no crater or a mound above the ground rather than a crater.

At even deeper depths of burial, about twice or deeper of that of optimum, another type of subsidence occurs. In this case the spall and gas acceleration has no significant effect on the overlying material. Only an underground cavity is formed. When the pressure in the cavity decreases below overburden pressure, the roof of the cavity begins to collapse. In most media this collapse will continue upward forming a chimney of collapsed material. In soil, where the density of the material will not significantly change after it has fallen, the volume of the undergrcund cavity will be transmitted to the surface.

Figure 11-21 illustrates surface time profiles after detonation of a 40 pound equivalent charge of ANFO, buried 8.0 feet in an unconsolidated, sedimentary type material. (46) High-speed photography was

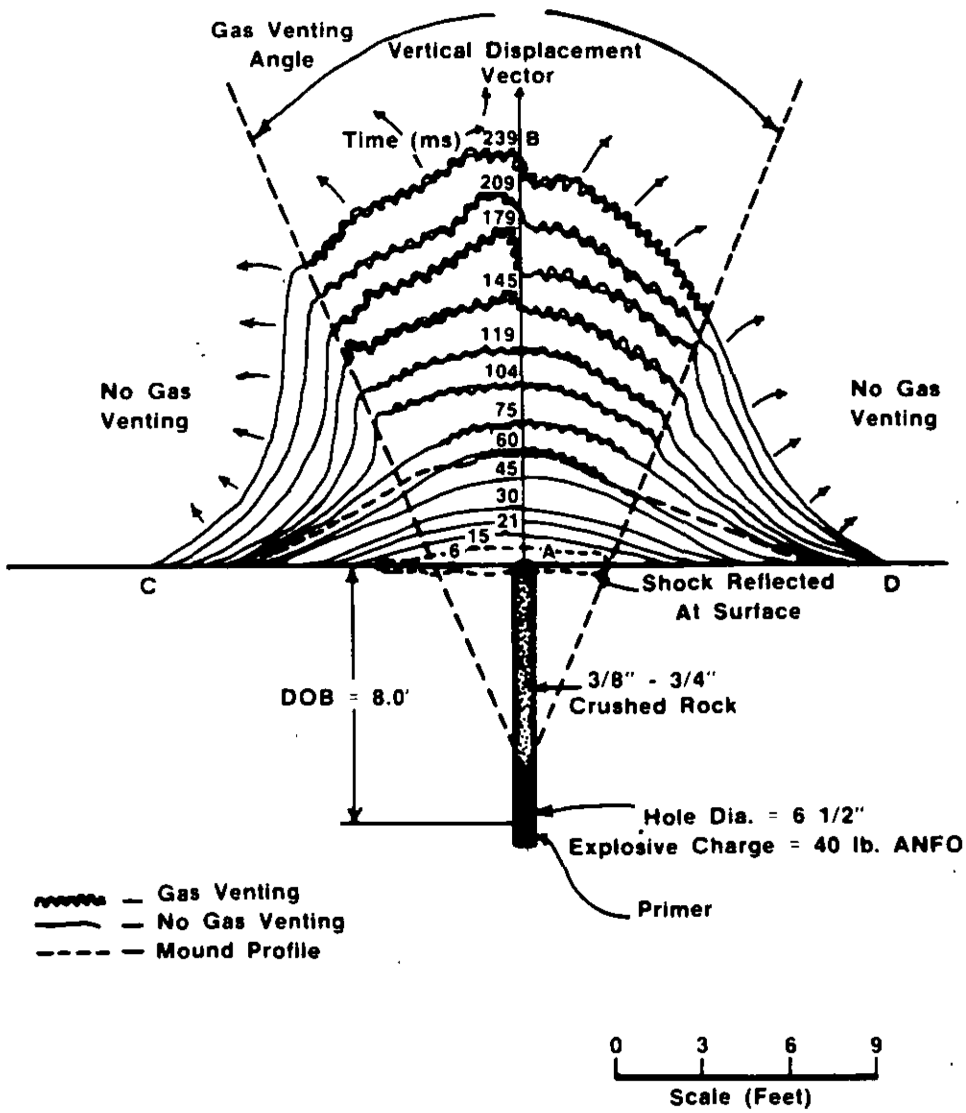
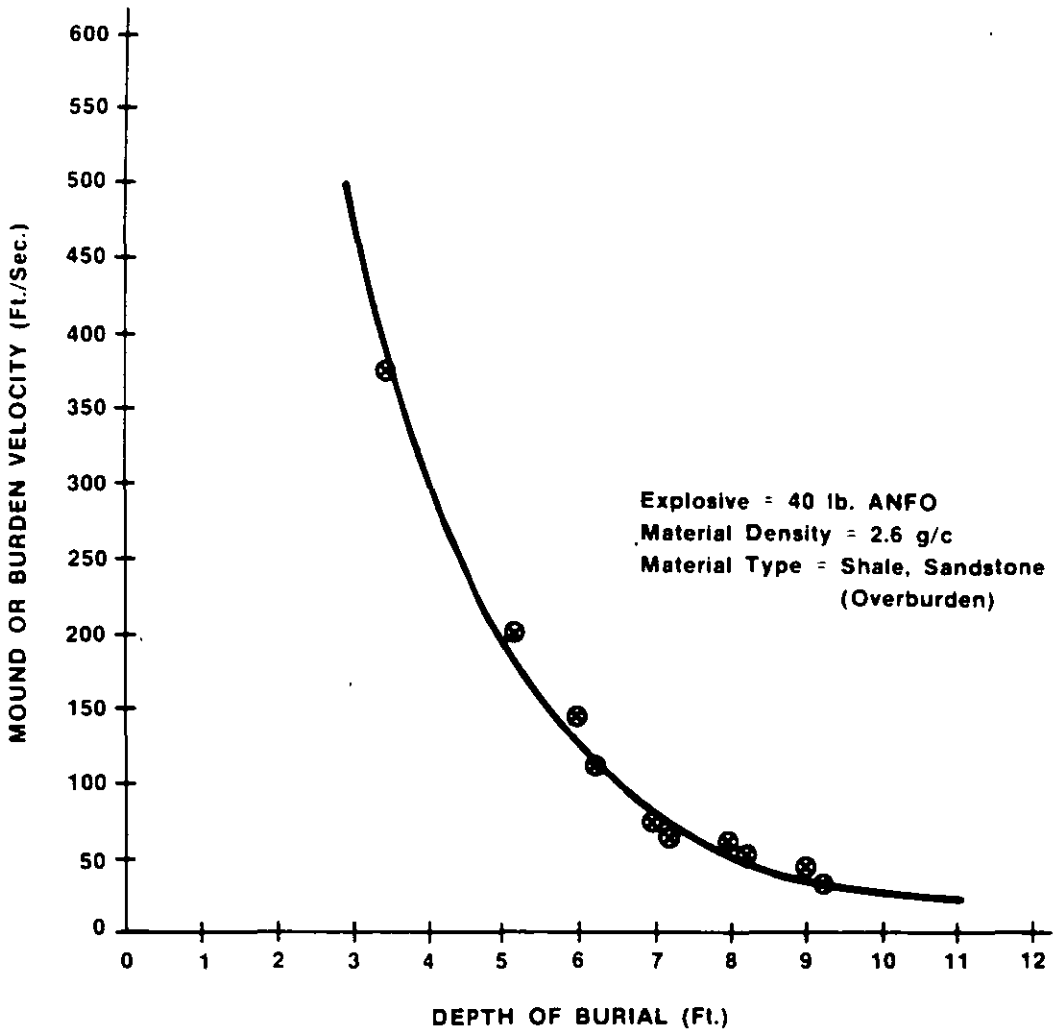


FIGURE 11.21

used to document the effects of shock and gas pressure The first observation was that of brisance or the reflection of the compressive shock at the surface a few milliseconds after detonation. This is indicated by the dotted eclipse immediately above the charge hole or surface With sufficient camera coverage and appropriate viewing angles, this shock ring can often be used to estimate, in rough, the degree of crater damage. In this case, sufficient viewing angles were not available and so only part of the total reflected shock could be resolved. Because the charge was placed at a depth significantly greater than the optimum depth of burial, no appreciable spalling occurred. Gas pressure was the dominant mechanism responsible for uplifting and ejecting material radially outward.

As gas expansion occurs around the charge cavity, the material above the charge is compacted and heaved upwards Between 0 to 45 milliseconds after detonation, the uplifted material is resiliant and compacted enough to maintain sufficient cohesion to contain all gases resulting from expansion. At 60 ms gas venting begins to occur directly above the charge and continues to expand in a well defined arc with respect to time. If the gas venting contacts at each end of each time profile are connected with straight lines, the lines will most always point toward the top or the center of the charge. In this case, the gas venting angle was measured to be approximately 45 degrees. The gas venting angle is useful in determining how much of the top part of a cylindrical charge, as found in production holes, actually contributes to gas venting, cratering and/or lost energy through lack of stemming confinement. At either side of the gas venting angle, no gas venting occurs, but material fragments are displaced and/or ejected outwardly Material fragments are also ejected from within the bounds of the gas venting angle Owing to a charge depth beyond optimum, the final result is a mound rather than a crater The mound is indicated by the shaded section underneath the 60 ms time profile

The initial instantaneous uplifting velocity above the charge is generally high but diminishes to zero when the material has reached its highest displacement. In reference to Figure 11-21, the average initial velocity along the vertical displacement vector up to 45 ms is 68 ft/sec. The average velocity from 60 ms to 239 ms is 54 ft/sec. The difference in velocity is attributed to the effects of gas venting and expansion beyond 60 ms. These velocities are dependent on material type and structure, explosive and depth of burial In general, the velocity will decrease exponentially with depth for a given explosive and material type as shown in Figure 11-22 (46)



MOUND OR BURDEN VELOCITY VS. DEPTH OF BURIAL FOR 40 POUND CHARGES OF ANFO
FIGURE 11.22

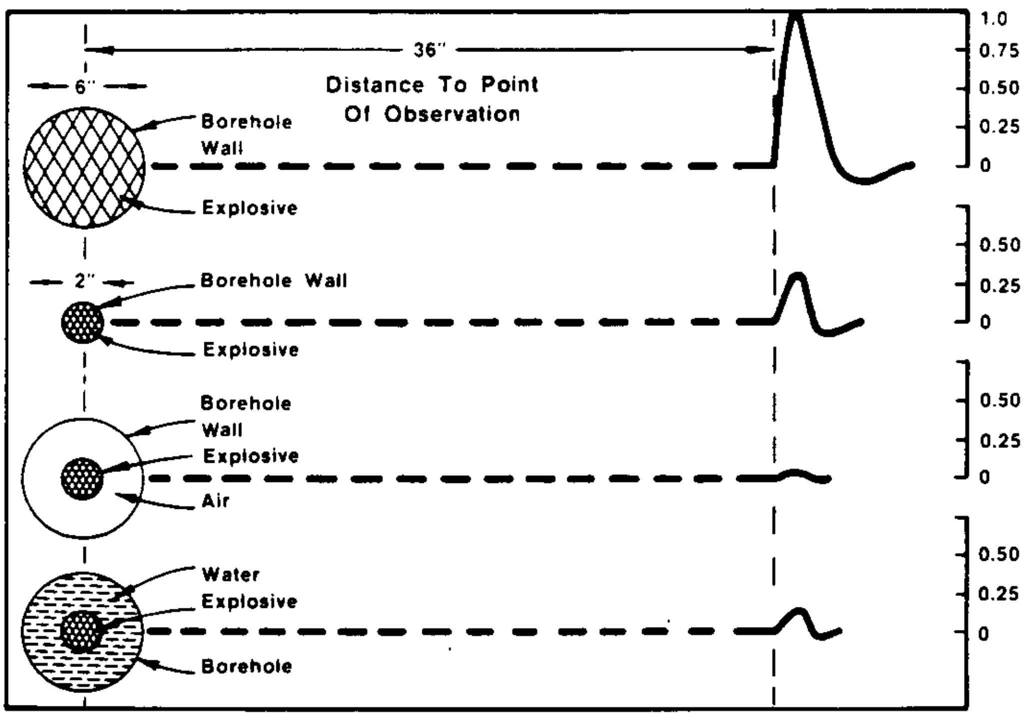
5. DECOUPLING

Decoupling is generally used as a control to reduce backbreak to the final planned excavation limit for pit wall slopes in open pit mines, shafts, drifts, ditches, road cuts and mine benches.

Since the borehole pressure is quite intense for a fully coupled borehole, exceeding many times that of the dynamic compressive strength of the rock, it must be reduced to avoid extensive damage. The three principal modes of rock failure occur by exceeding the dynamic compressive, shear or tensile strengths Ideally, the borehole pressure should be somewhere between the compressive and tensile strength of the rock, so as to avoid extensive crushing at the borehole wall, yet provide enough pressure to extend a single predominant crack between any two perimeter holes in the control line of holes.

A good example of decoupling in air and water in relation to fully coupled holes is illustrated in Figure 11-23. (47) The pressure imparted in the rock mass at 36" away for the same explosive is shown for four conditions

i) a 6" diameter explosive in a 6" hole
ii) a 2" diameter explosive in a 2" hole
iii) a 2" diameter explosive in a 6" hole (air decoupled)
iv) a 2" diameter explosive in a 6" hole (water decoupled)



EFFECT OF AIR AND WATER DECOUPLING VS FULLY COUPLED HOLES
FIGURE 11.23 (47)

All measured stress levels are compared relative to the 6" diameter explosive in a 6" diameter hole. A number of important points are immediately evident. The greatest stress level was achieved with a fully coupled explosive in a 6" diameter hole. The next highest stress level was achieved, again, with a fully coupled explosive, even though the hole diameter was reduced three-fold to a 2" diameter. Water decoupling followed next and air decoupling produced the smallest stress level. Thus, an air decoupled charge is the most effective means of reducing borehole pressure and consequently the peak stress level within the rock mass.

A reasonably reliable method of calculating the borehole pressure is with the following formula which takes into account two decoupling ratios. (48) (49) (50)

$$P_b = 1.69 \times 10^{-3} \times \rho \times VOD^2 \times \left[\sqrt{c} \times \frac{de}{dh}\right]^{2.6}$$

where:

P_b = Borehole pressure in PSI.

ρ = Density of explosive in g/cc

VOD = Velocity of detonation in ft/sec

c = Percentage of explosive column loaded expressed as a decimal

de = Explosive diameter (in.)

dh = Hole diameter (in.)

This formula is best suited for explosives which contain no metallic elements or relatively small amounts, since the addition of energizing metals lowers the detonation velocity of the explosive and hence, the borehole pressure as calculated by this equation. Computer codes such as TIGER and EXPLODE are used to calculate borehole pressures from explosives containing metallic elements.

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7. UNDERGROUND BLASTING

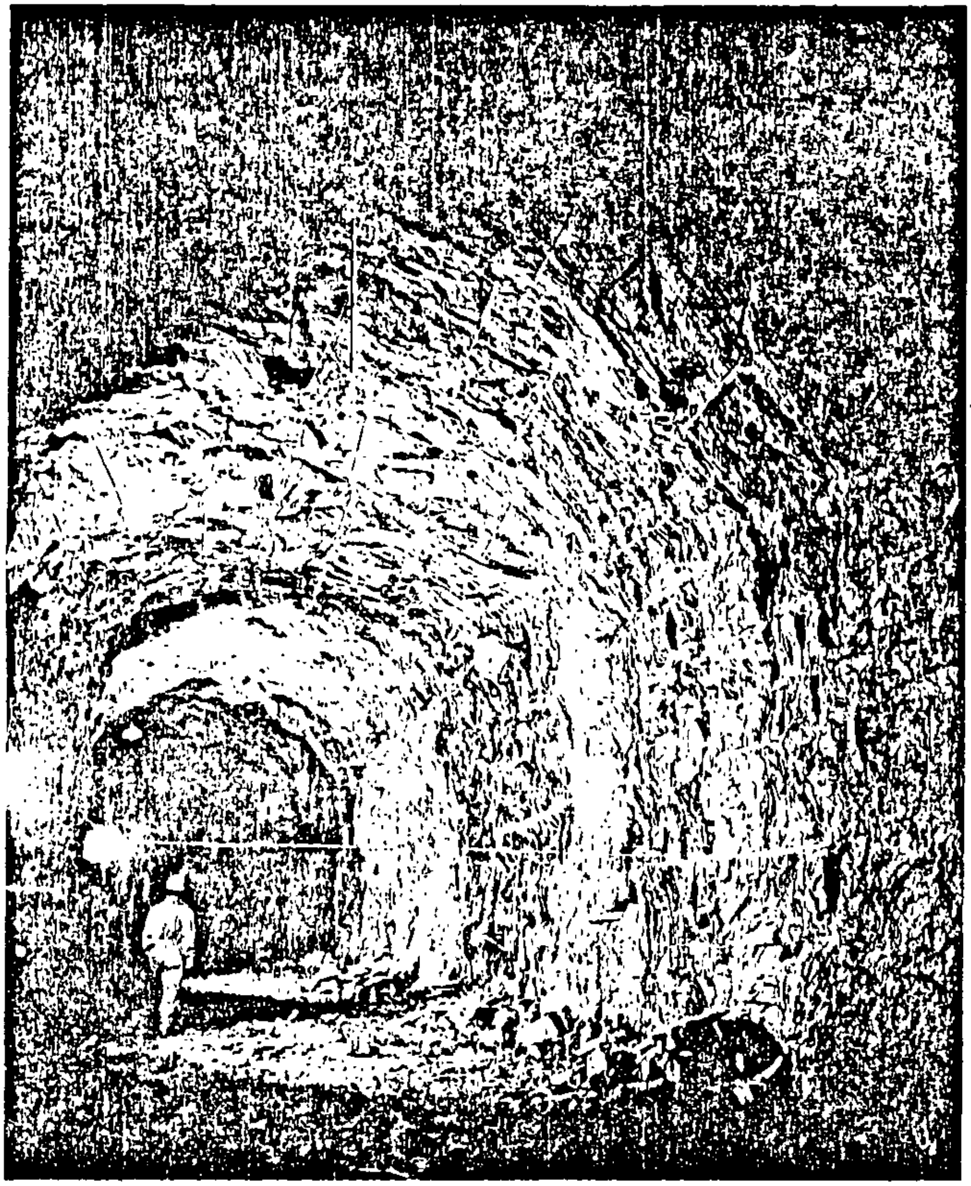

Fig. 7.1 Tunneling.

7.1 Tunneling.

There are two reasons to go underground and excavate:

- to use the excavated space, e.g. for storage, transport etc.
- to use the excavated material, e.g. mining operations.

In both cases tunneling forms an important part of the entire operation. In underground construction it is necessary to gain access to the construction site by

FACULTAD DE INGENIERIA U.N.A.M.
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CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

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ALTERNATE VELOCITY LOADING TECHNIQUES AND DETONATIONS IN A PRODUCTION ENVIRONMENT

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Mine Health and Health Administration
National Mine Health and Safety Administration
National Mine Health and Safety Academy
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ALTERNATE VELOCITY LOADING TECHNIQUES AND DETONATIONS IN A PRODUCTION ENVIRONMENT

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ABSTRACT

A simple and cost effective technique to increase fragmentation and burden velocities without making major modification to the overall blast design is with ALTERNATE VELOCITY LOADING OR BOOSTERING OF ANFO. The technique requires the placement of a cartridge or slug of explosive, having higher density and detonation velocity than ANFO, every few feet in an ANFO column. The greater the difference in density and detonation velocity of the Alternate Velocity Load to ANFO, the more pronounced are the results. Emulsions were selected as the Alternate Velocity test explosives because they detonate closer to ideal conditions than most other commercial explosives.

The emulsion explosives embedded in the ANFO column did not require additional boostering. Even a low order ANFO detonation, alone, acted as an effective primer on the Alternate Velocity emulsion explosive. It was also determined that ANFO efficiency suffers greatly, when ANFO is loaded in a dewatered hole.

Testing consisted of single and multi-hole production shots in full-scale environments. Analytical methods, testing procedures and a discussion of the breaking processes are described in detail.

TABLE OF CONTENTS

INTRODUCTION

The breaking and heaving processes resulting from single or multi-hole detonations encompass a complex array of phenomena not all of which are completely understood. However, with the advent of newer, more precise and sophisticated instrumentation, a better definition and explanation of what occurs within and around a borehole at close vicinity are now possible. Clarification of such short lived phenomena in and around a borehole environment is invaluable to us in our basic understanding of the breakage process.

The ATLAS POWDER COMPANY in association with other research organizations has invested heavily in researching specific areas of blasting in an attempt to produce more efficient, consistent and cost effective blasting techniques for the end user. One such technique is described in this paper as ALTERNATE VELOCITY LOADING OR ALTERNATE VELOCITY BOOSTERING OF ANFO. The technique consists of placing a cartridge or slug of explosive, with higher density and velocity of detonation than ANFO, every few feet in an ANFO column. In the last few years, the technique has been used in a wide variety of materials stretching from very soft overburdens to the hardest of granites and in hole diameters ranging from 2-1/2 to 12 inches. With a carefully designed blast utilizing the proper amount and distribution of energy, optimum selection of MS delay timing and ALTERNATE VELOCITY LOADING, excellent results can easily be realized in terms of fragmentation and mass movement of burdens. Thus, ALTERNATE VELOCITY LOADING is equally applicable to overburden casting in stripping operations and to bench blasting operations in quarries.

In order to understand some of the mechanisms responsible for the success of Alternate Velocity Loading, a review of the basic breakage process is essential. There are basically four time frames designated as T1 to T4 in which detonation, breakage and heaving of material occur during and after detonation of a confined charge. The time frames are defined as follows:

- T1 - Detonation
- T2 - Shock or Stress Wave Propagation
- T3 - Gas Pressure Expansion
- T4 - Mass Movement

Although each time frame is treated as a discrete event for conceptual clarity, it should be emphasized that in a typical shot hole or production blast, one event phase can occur simultaneously with another at specific time intervals. Each time frame is first discussed separately and then in a unified explanation and meshing of events in conjunction with some of the more commonly accepted blasting theories.

T1 - DETONATION

Detonation is the beginning phase of the fragmentation process. The ingredients of an explosive consisting of a fuel and oxidizer combination; upon detonation, are immediately converted to high pressure, high-temperature gases. Pressures just behind the detonation front or head are in the order of 9 Kbars to 275 Kbars, while temperatures range from 3000 to 7000 F. The detonation head is referred to here as the primary reaction zone for the fuel and oxidizer mixture.

Detonation pressure is generally expressed as a function of the velocity of detonation and density of the explosives as,

$$P = (2.325 \times 10^{-7}) \times \rho \times VOD$$

Where P = detonation pressure in Kbars

ρ = density in g/cc

VOD = velocity of detonation in ft/sec.

To change detonation pressure from Kbars to lb/in , multiply Kbars by 14,504. Generally, explosives yielding higher detonation pressures are required to fracture materials which are massive, fine grained, hard, tightly bonded and strongly consolidated with heavy burdens. Typical values of detonation pressure for selected explosives are presented in Table 1.

TABLE 1

DETONATION PRESSURES FOR SELECTED EXPLOSIVES

Explosive	Density (g/cc)	VOD (ft/sec)	Detonation Pressure (Kbars*)	Pressure (psi)
ANFO	0.81	12,000	27.00	391,600
POWERMAX 420	1.19	19,000	100.00	1,450,400
HI-PRIME	1.40	20,000	130.00	1,885,500
"G" BOOSTER	1.60	26,000	251.00	3,640,500

*1 Kbar = 14,504 psi

The detonation wave starts at the point of primer initiation in the explosive column and travels at supersonic speeds. Supersonic refers to velocities which are faster than the speed of sound in the explosive. Typical velocities of detonation for commercial explosives range from 8,000 to 26,000 ft/sec. This velocity, sometimes referred to as the steady-state velocity, remains fairly constant for a given explosive, but varies from one explosive to another, depending primarily on the composition, particle size and density of the explosive. To a lesser extent, the steady state velocity is also affected by the degree of confinement and explosive diameter.

Since the velocity of detonation is greater than the velocity of sound in the explosive, the explosive material directly in front of the detonation head is totally unaffected until the detonation head passes through it. In a typical 30 foot explosive column loaded with an explosive having a characteristic velocity of detonation of 10,000 ft/sec, complete detonation and energy release within the entire column would occur in about 3 milliseconds. For an explosive with a velocity of detonation of 20,000 ft/sec, detonation and energy release would be complete in 1.5 milliseconds. Detonations of this kind are self-sustaining due to the inertia of the explosive itself that provides confinement necessary to maintain conditions for fast chemical reaction rates. (1)

Figure 1 illustrates a typical hole load configuration. Velocity of detonation within the explosive column was measured with the SLIFER system developed at SANDIA NATIONAL LABORATORIES. For a continuous 11 foot column of cartridged ANFO, the velocity of detonation was measured to be 12,200 ft/sec as indicated by the slope of the straight line segment between points (a) and (b) in Figure 1. The straight line is indicative of a consistent explosive composition, constant density and a stable velocity of detonation. As detonation progresses along the column, not only is a shock wave imparted into the surrounding medium adjacent to the borehole wall, but is also imparted into the stemming as indicated by the slope of the straight line segment between points (b) and (c). In this case, the shock wave velocity through the stemming was measured to be 2,900 ft/sec, or approximately 1/4 that of the velocity of detonation.

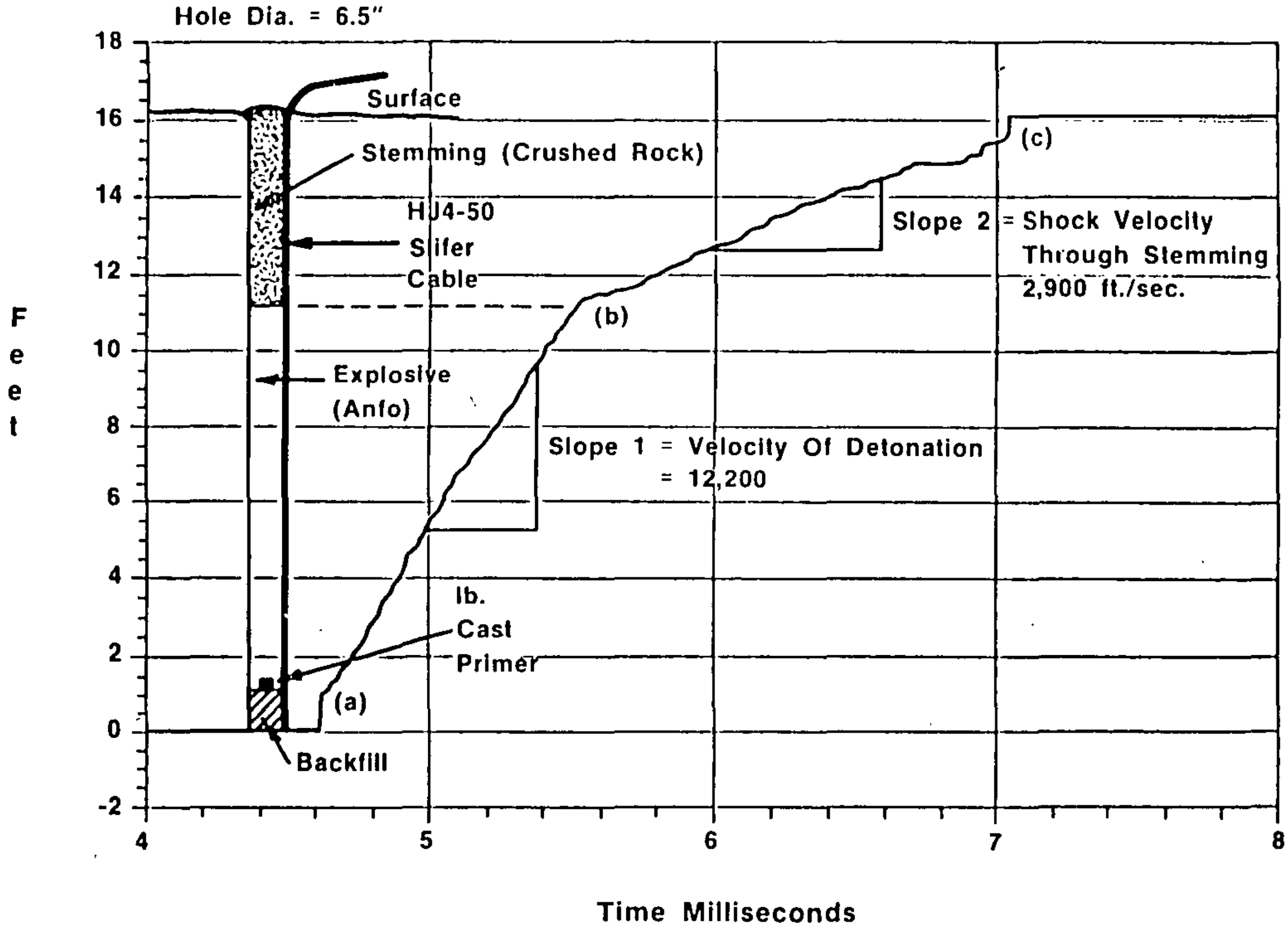


FIGURE 1

VELOCITY OF DETONATION MEASUREMENT USING THE SLIFER SYSTEM DEVELOPED AT SANDIA NATIONAL LABORATORIES

In a stable detonation the detonation wave travels through the column of explosive at a constant rate, (Figure 1). This rate is strictly dependent on the chemical energy released and the density and the diameter of the explosive column. Although stable, this steady-state velocity of detonation is not necessarily the "ideal" or theoretical maximum that is possible for ANFO in a larger diameter hole. If all the energy is liberated before the end of the detonation head, the detonation velocity is ideal. This is diagrammatically illustrated in Figure 2 for high explosives (TNT, RDX, etc.) where the thickness of the reaction zone is relatively small and thin and the detonation front is relatively flat.

The ideal velocity of detonation for any explosive can be calculated from the equilibrium thermodynamics and an appropriate equation of state for a given original density and chemical composition of the expected detonation products. When experimental results closely match with the predicted values, we can also say that the explosive is ideal.

Based on ideal performance calculation using the BKW code, (2), the detonation velocity for ANFO should be 17,700 ft/sec with a 73 Kbar (1,073,000 PSI) detonation pressure. However, experimental results in 3.9" ($\rho = 0.95$ g/cc) and 7.9" ($\rho = 0.90$ g/cc) diameter charges with ANFO gave detonation velocities of 11,400 ft/sec and 13,500 ft/sec, respectively. Finger et al (3) reported detonation velocities for 0.84 g/cc ANFO in the order of 15,400 ft/sec. Measurements made by

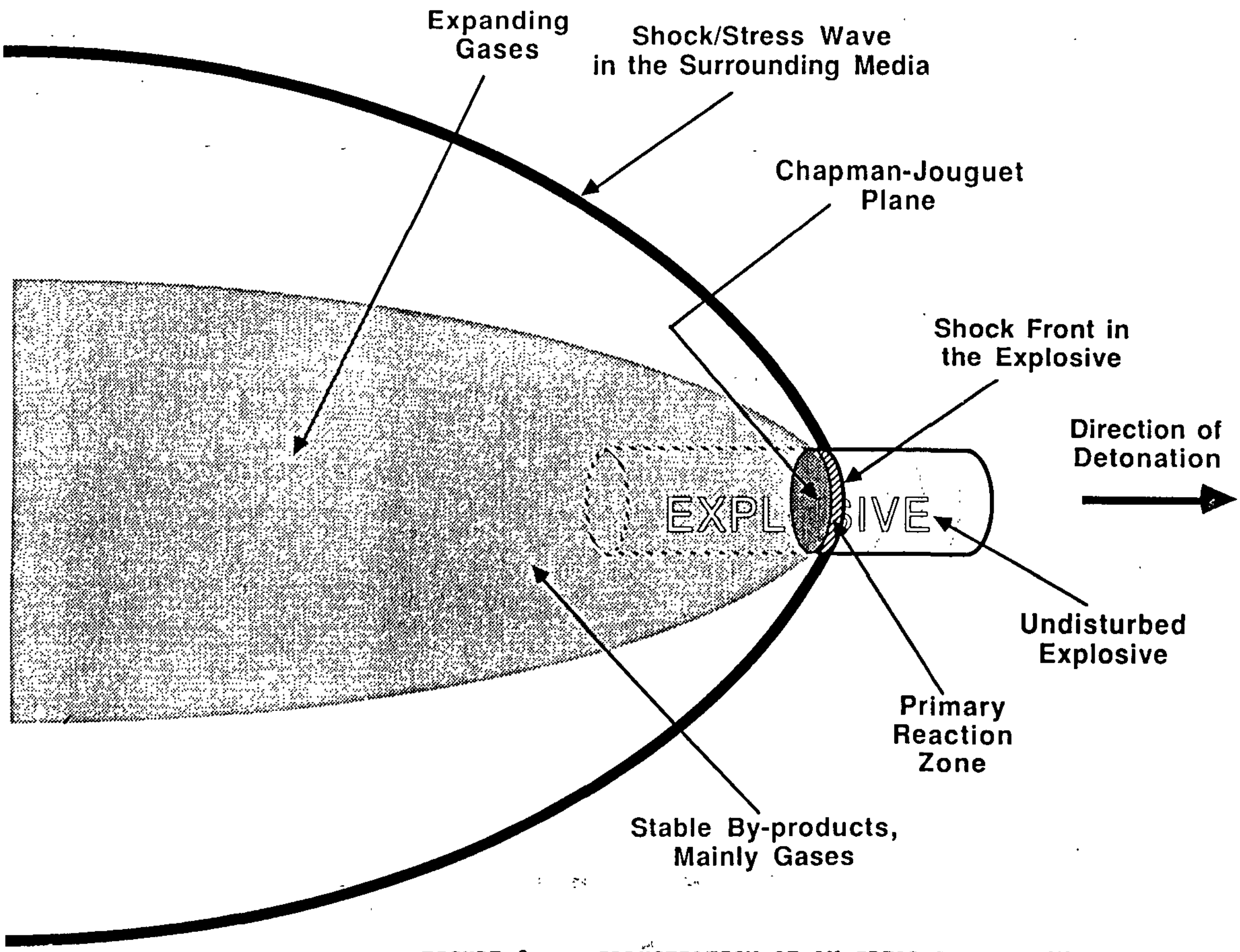


FIGURE 2 - ILLUSTRATION OF AN IDEAL DETONATION

Helm et al (4) also show that detonation velocities in ANFO are well below ideal conditions for large diameter holes up to 11.5" Persson(9) reported detonation velocities very close to the theoretical value of 17,800 ft/sec for 10.5" diameter holes when confined in rock. However, Atlas field studies for measurements in hole diameters up to 12" resulted in detonation velocities, at best, of 15,000 ft/sec. This is still well below our computer calculations of 17,000 - 19,000 ft/sec for ANFO when using different codes and equations of state.

Clearly then, detonation velocities for ANFO in hole diameters less than 17" are less than ideal. When this occurs, the reaction in the detonation head is said to be non-ideal and takes the general (exaggerated) shape as is illustrated in Figure 3. Compared to an ideal reaction zone, the non-ideal reaction zone is not flat, but rounded at the front and somewhat longer. At diameters less than that at which ideal detonation occurs (such as in most production holes) the non-ideal regime holds. Under these conditions, the detonation velocities are less than ideal, and it suggests that an ANFO prill entering the detonation is still not completely reacted by the time the tail end of the detonation has passed (refer to Figure 3). This accounts for the reduced detonation velocity.

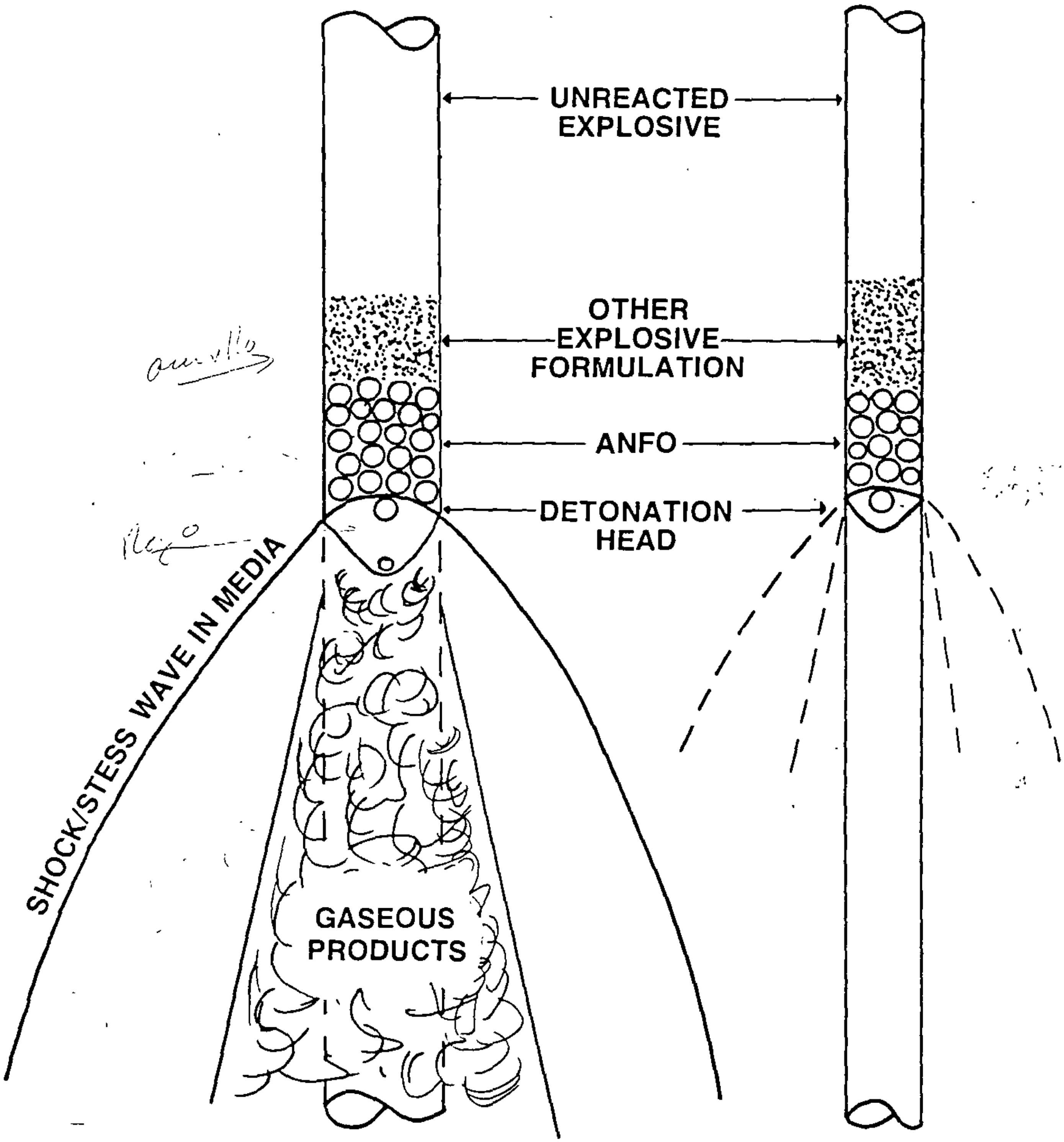


FIGURE 3

ILLUSTRATION OF A NON-IDEAL DETONATION

It also suggests that ANFO not consumed in the primary reaction zone may be reacting outside of the detonation head, that is, just behind the reaction zone at lower temperatures and pressures in the expanding gas zone.

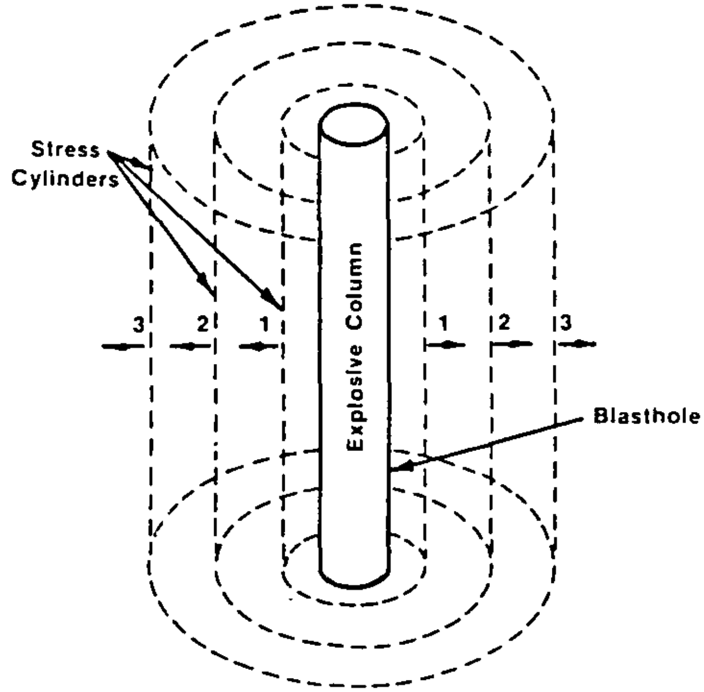
In addition to ANFO, most commercially used explosives are of the non-ideal type. This includes slurries, watergels and emulsions. Performance of slurries and watergels, as in the case of ANFO, is dependent upon charge geometry and confinement. Furthermore, a large fraction of the explosive energy comes from the reaction of the oxidizer and fuel, which are in discrete phases whose dimensions vary. The larger the dimensions of these phases, the larger become the reaction times and the reaction zones. In the case of emulsion explosives, which comprise a better intimate mixture on a smaller scale, actual detonation velocities are very close to ideal. <u>Thus, emulsion explosives allow a more efficient release of energy in the primary reaction zone, and for this reason have been selected as our test explosive as the alternate velocity load.</u>

T2 - <u>SHOCK AND STRAIN WAVE PROPAGATION</u>

The second phase, immediately following detonation or in conjunction with the detonation phase of T1, is the shock and strain wave propagations throughout the rock mass. This disturbance or emitted pressure wave(s) emitted into the rock mass results, in part, from the rapidly expanding high-pressure gas impacting the borehole wall. The geometry of dispersion depends primarily on the shape of the charge. If the charge is shot, with a length to diameter ratio of less than or equal to 6:1, then the disturbance is propagated in the form of an expanding cylinder, (Figure 4). However, in a typical, bottom primed, cylindrical shot hole normally encountered in bench blasting, the strain waves originally formed near the point of initiation are already in progress and propagating into the surrounding medium, while the detonation is still progressing within the explosive column. Thus, close to the shot hole, strain wave propagation is neither perfectly spherical nor cylindrical but more like that shown in Figure 5.

The pressure next to the borehole wall will rise instantaneously to its peak and then rapidly decay exponentially. The quick decay is due to cavity expansion around the borehole and increased gas cooling. Cavity expansion around the borehole can occur through crushing, pulverization, and/or displacement of material and can range anywhere from about one to three holes diameters depending on the medium and explosive used. Generally, extensive compressive, shear and tensile failure occur as a region of pulverized material, since the wave energy is at its maximum near the borehole wall.

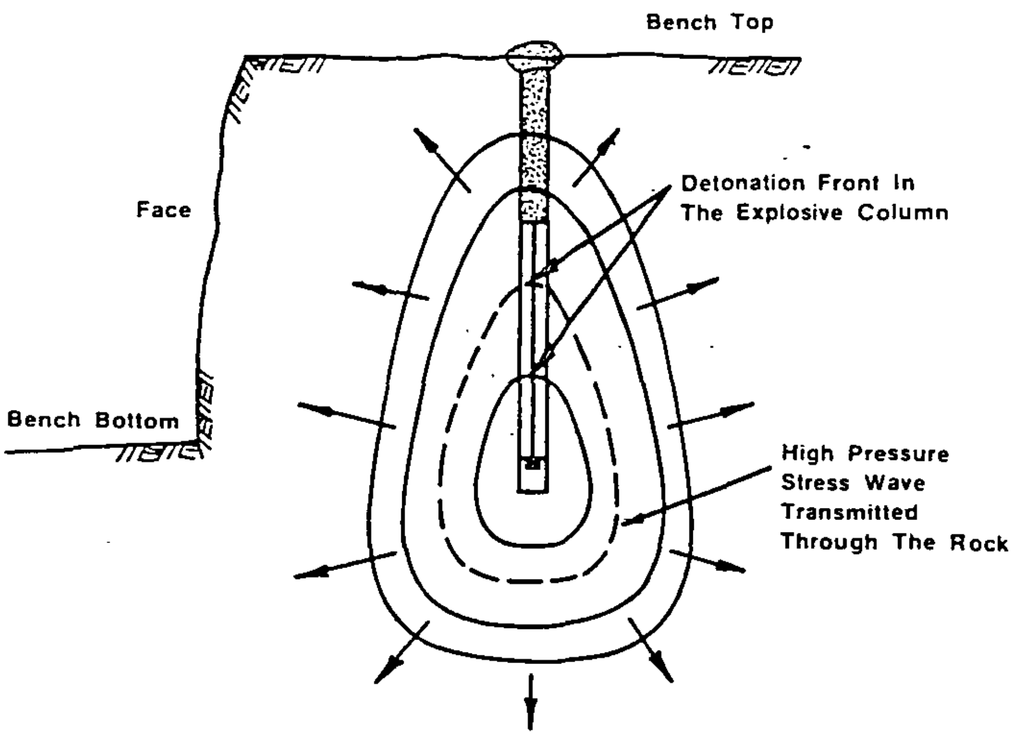
As the strain wave front proceeds outward, it has a tendency to compress the material at the wave front through a volume



1,2,3 Successive Positions Of Stress Wave

THEORETICAL POSITIONS OF THE OUTBOUND DISTURBANCE FROM A COLUMN CHARGE

FIGURE 4



SECTION THROUGH THE FACE DURING DETONATION SHOWING EXPANDING STRESS WAVE FRONT

FIGURE 5

change. At right angles to this compressive front, there exists another component referred to as the tangential or "hoop" stress. The tangential stress, if large enough, can cause tensile failures at right angles to the direction of propagation. The largest tensile failures are expected to occur close to the borehole where the tangential stress is high enough for failure to occur. Both the compressive and tensile components of the wave front decay with distance from the borehole.

When the compressive wave front encounters a discontinuity or interface, some of the energy is transferred across the discontinuity and some is reflected back to its point of origin.(5) For the most part, the partitioning of energy depends on the ratio of the acoustic impedance of the materials on either side of the interface, as illustrated in Figure 6. Acoustic impedance, Z, for any material is defined as:

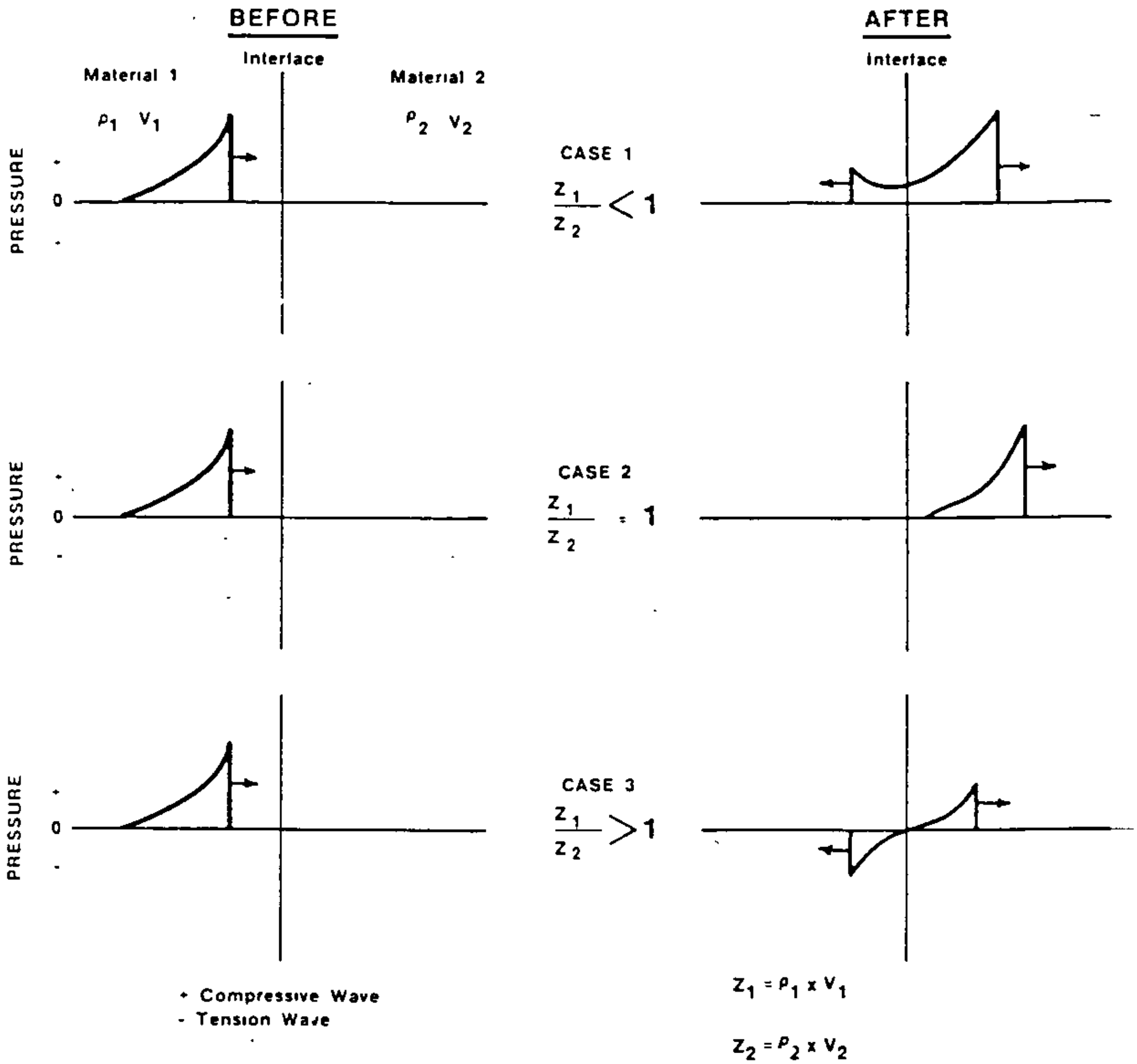
$$Z = \rho \times V$$

Where: Z = acoustic impedance

ρ = density of material

V = sonic velocity of material

In reference to Figure 6, where the ratio of the acoustic impedance of material 1 to material 2 is less than one, some of the wave energy is transferred into material 2 and some reflected back, but both waves remain compressional. When the acoustic impedance ratio is 1, all of the energy is



INTERACTION OF STRESS WAVES
AT AN INTERFACE
FIGURE 6

transferred into material 2 and no reflected wave occurs. When the impedance ratio is greater than 1, then some of the energy gets transferred into material 2 as a compressive wave and the remaining energy gets reflected at the interface as a tensile wave. When a compressive wave travelling through rock encounters an interface such as a free face, nearly all of the energy will be reflected back as a tensile wave. If the burden distance between the free face and explosive column is relatively small in contrast to normal burdens for a chosen explosive, then most of the energy is consumed in spalling at the free face.

The interaction of stress waves in the outgoing compressive and reflected tensile modes around discontinuities and flaws within the rock mass is an area of intense research and is considered to be quite important in some of the newer blasting theories. In order to effectively utilize the interaction of shock waves in a production environment, ultra-precise detonators with precisions in the order of a few hundred microseconds about the mean are suggested for the next generation of detonators.

T3 - GAS PRESSURE

During and/or after strain wave propagation, the high pressure, high temperature gases impart a stress field around the blasthole that can expand the original borehole, extend radial cracks and jet into any discontinuity. It is during this phase where some controversy exists as to the main mechanism of fragmentation. Some believe that the fracture network throughout the rock mass is completed while others believe that the major fracturing process is just beginning. In any case, it is the gases that have jetted into discontinuities and the fracture network that is either fully developed or being developed, which are responsible for the displacement of broken material. Past, current and newer blasting theories are listed as follows:(6)

1) Reflection Theory
2) Gas Expansion Theory
3) Flexural Rupture
4) Stress Wave & Gas Expansion Theory
5) Combined Theory
6) Nuclei or Stress-Wave/Flaw Theory
7) Torque Theory
8) Cratering Theory

It is not clear as to the exact travel paths that gases take within the rock mass, although it is agreed that they will always take the path of least resistance. This means that gases will first migrate into existing cracks, joints, faults, and discontinuities, in addition to seams of material which exhibit low cohesion or bonding at interfaces. If a discontinuity or seam between the borehole and free face is sufficiently large, the high pressure gases will immediately vent to the atmosphere, rapidly reducing the total confinement pressures, and results in reduced displacement of broken and fragmented material.

The confinement time of gas pressures within a rock mass vary significantly depending on the amount and type of explosive, material type and structure, fracture network, amount and type of stemming, and burden. Studies by Chiappetta et al (7) with the use of high-speed photography in full-scale bench blasts, have shown that gas confinement times before the onset of movement can vary from a few milliseconds to tens of milliseconds. To date, confinement times have been measured to range from 5 to 110 milliseconds for a variety of materials, explosives and burdens. Generally, but not always, confinement times can be decreased by employing higher energy explosives, decreasing the burden, or a combination of both. This applies equally to material at the bench face or at the bench top, as in the case of stemming blowouts or cratering. It is evident that only suitably burdened and well stemmed charges can deliver their full potential of additional gas extension fracturing and mass movement.

T4 - MASS MOVEMENT

Mass movement of materials is the last stage in the breaking process. The majority of fragmentation has already been completed through compressional and tensile stress waves, gas pressurization or a combination of both. However, some degree of fragmentation, although slight, occurs through in-flight collisions and also when the material impacts the ground. Generally, the higher the bench height, the greater is this type of breakage owing to the increased impact velocities of individual fragments when falling onto the bench floor. Similarly, material ejected from opposite rows of a "V-shot" design upon head-on collisions can result in increased fragmentation. This phenomenon was evidenced and documented with the use of high-speed photography of bench blasts.

Mass burden movement of fragmented material is shown in Figure 7 for a number of typical face conditions encountered in bench blasting operations. Face profiles and velocities are based on the results of high-speed photographic analysis performed at the ATLAS POWDER COMPANY. Where no subdrilling is utilized, (a and b), two types of face movement may be encountered. In Figure 7a the entire length of face burden, directly in front of the explosive column, moves out similar to a plane wave and the face velocity at any point is constant. This behavior is usually encountered where material is very competent, quite brittle, and structured with well defined, largely spaced joints much greater than the spacings or burdens employed in blast designs. When the material is soft, highly fissured, and/or closely jointed as might be found in coal and some sedimentary deposits, face profiles resembling that of flexural rupture are more likely. In this case, the greatest displacement and velocity occur adjacent to the center of the explosive column with the least amount of movement occurring at the toe and crest.

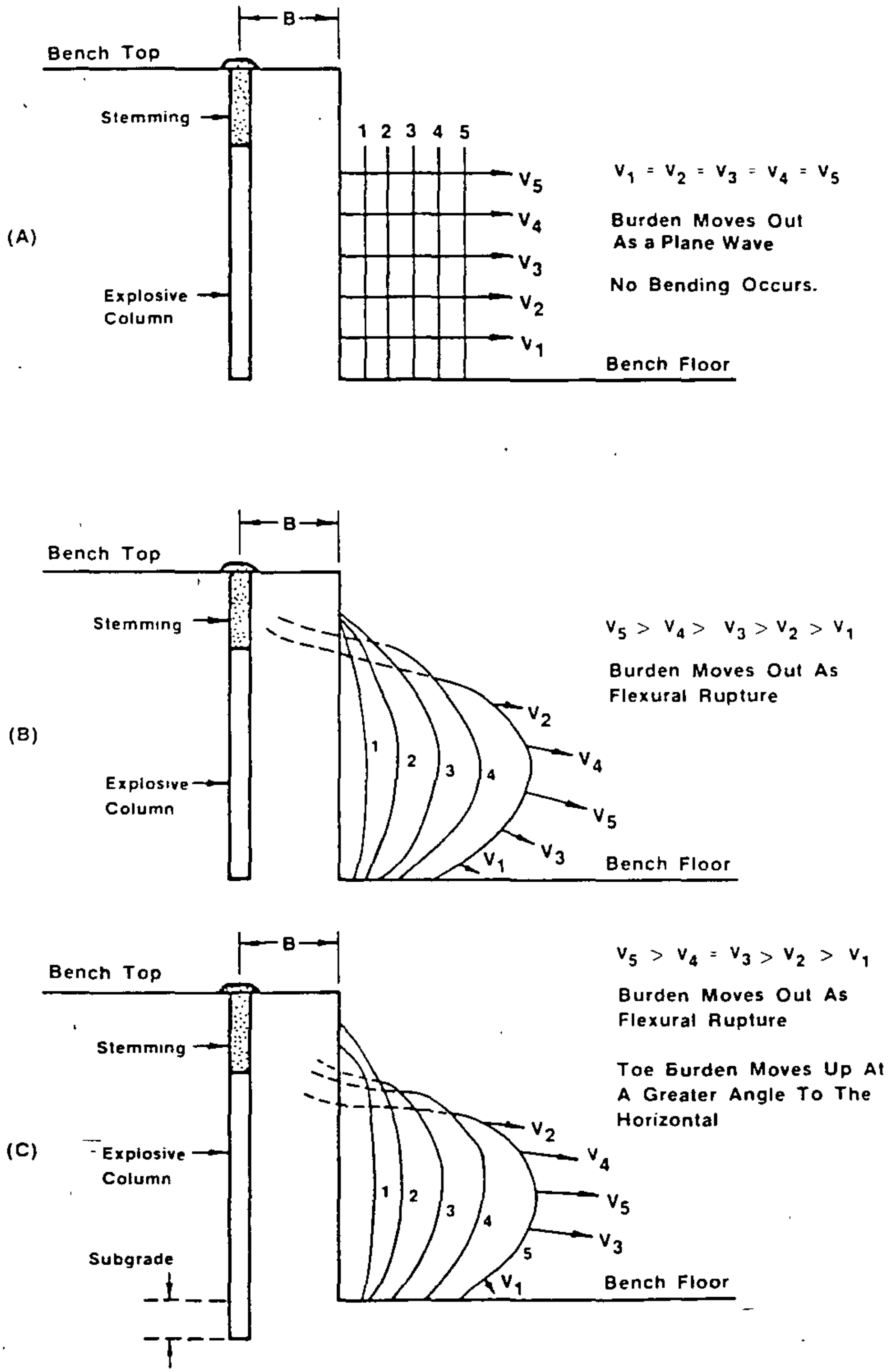
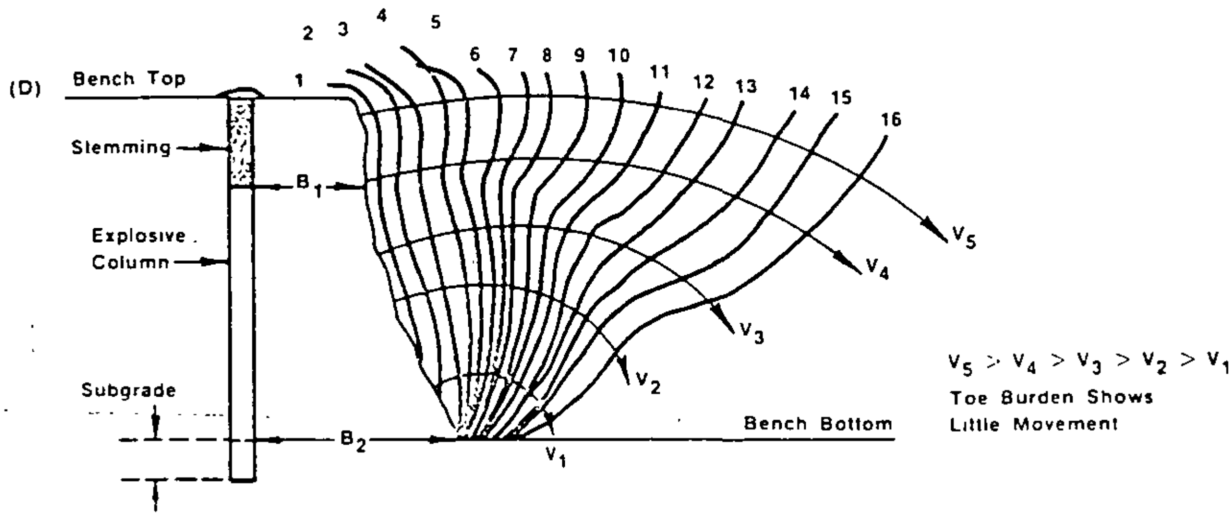

B
Bench Top
Stemming
Explosive Column
1 2 3 4 5
V5
V4
V3
V2
V1
(A)
V1 = V2 = V3 = V4 = V5
Burden Moves Out As a Plane Wave
No Bending Occurs.
Bench Floor
(B)
V5 > V4 > V3 > V2 > V1
Burden Moves Out As Flexural Rupture
(C)
V5 > V4 = V3 > V2 > V1
Burden Moves Out As Flexural Rupture
Toe Burden Moves Up At A Greater Angle To The Horizontal
Subgrade

FIGURE 7



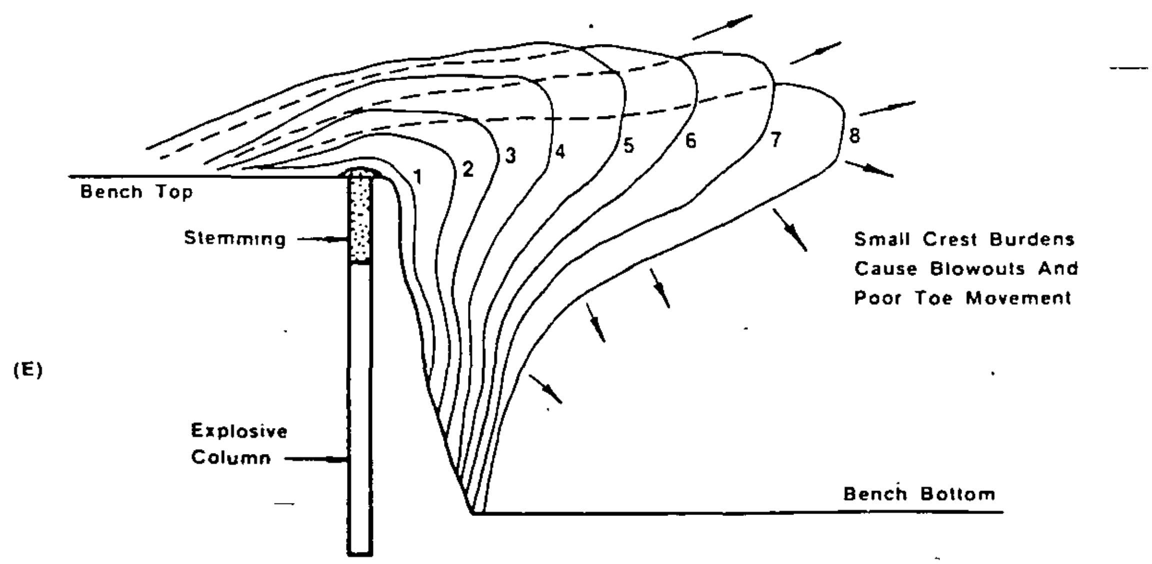
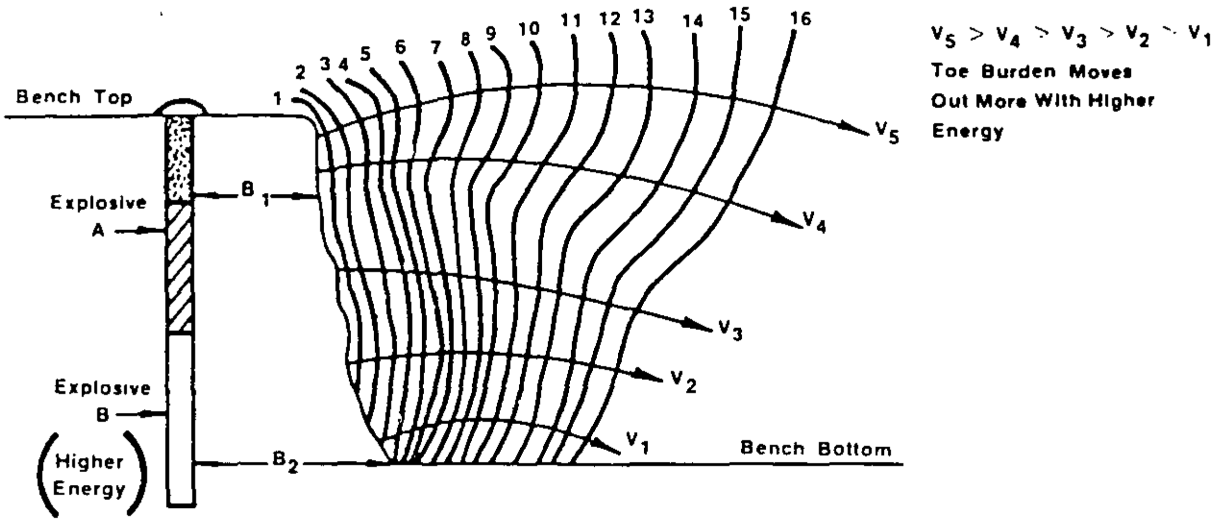
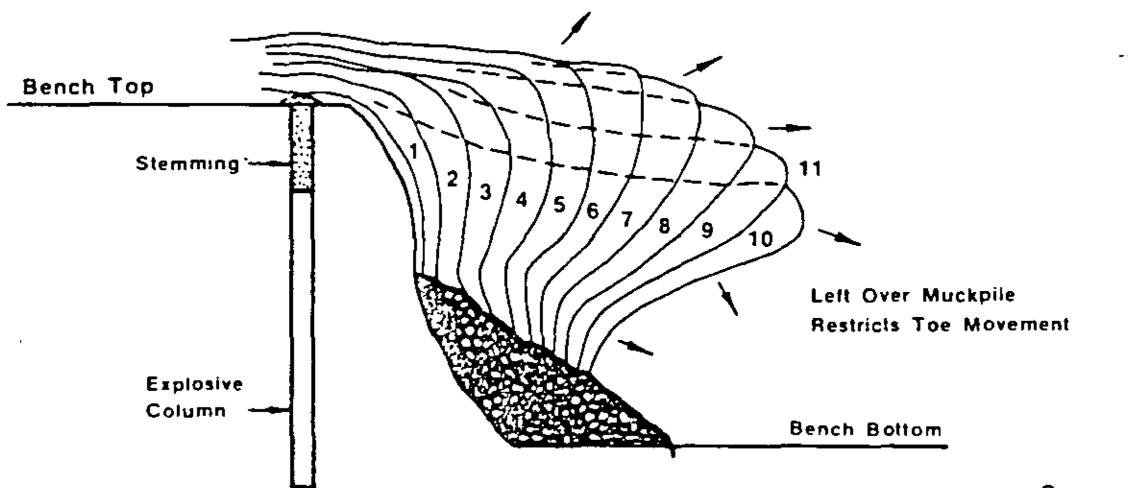


FIGURE 7 (Cont'd)

(F)



(G)



(H)

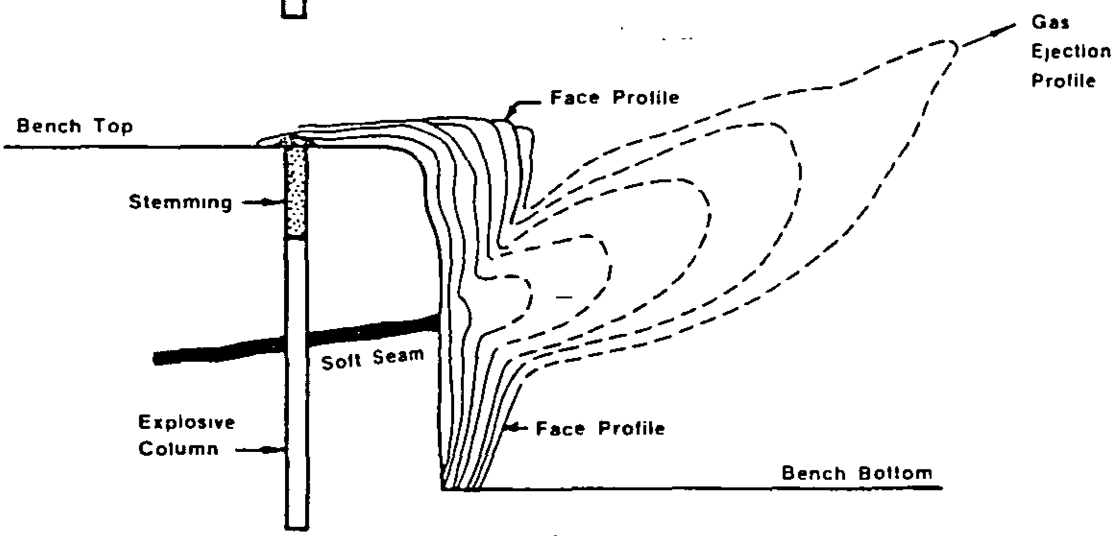


FIGURE 7 (Cont'd)

When identical conditions in Figure 7b are assumed and when subdrilling ~~is employed~~, face movement results in much the same way except that the toe burden is displaced upwards faster and at a greater angle to the horizontal. (Figure 7c)

The first three cases assumed a relatively straight face between the crest and toe, however, in many bench blasting operations, the condition is more like that illustrated in Figure 7d, where toe burden is considerably greater than the crest burden. The toe burden is too great for the explosive selected, hence, very little movement occurs at the toe while the greatest displacement results in the upper half of the bench.

Three options are available to increase toe movement:

* Employ angle drilling in an attempt to maintain constant burdens from the crest to the toe
* Use a higher energy bottom charge in the current vertical drill holes.
* Decrease the burden with the current drill holes.

In selecting the latter, care should be exercise so as not to decrease the burden to the point of obtaining the condition shown in Figure 7e. The toe burden is now correct for the explosive selected, but the crest burden is substantially reduced. This may bring about many adverse conditions near the crest burden such as flyrock, blowouts and increased airblast complaints. Because confinement pressures are released near the crest (in this case, a path of least resistance relative to the toe burden), restricted toe movement will result. It is better to use the same burden, but with a high energy bottom charge near the toe. This load configuration as shown in Figure 7 f tends to pressurize more of the burden mass for longer periods without adverse effects, and adequate toe movement generally results.

Where large leftover muckpiles are left against the face, Figure 7g, toe movement will be restricted and increased ground vibration levels are likely. Unless the situation requires a buffer, such as when blasting in the vicinity of mining equipment or to avoid dilution of an ore blast adjacent to a waste muckpile, it should be avoided.

Where seams are encountered in a blast, Figure 7h, tremendous gas ejections with velocities up to 600 ft/sec can occur. When such gas venting occurs, it will adversely affect other parts of the burden to displace adequately and inevitably leads to poor overall blasting results. A stemming deck immediately adjacent to the seam will give better results.

TIME EVENTS T1 - T4 COMBINED

Up to this point, events T1 to T4 have been discussed more or less as separate isolated events. However, in a real blasting environment, more than one event can occur at the same time.

Consider a single vertical hole in a quarry face with the primer located near the bottom of the hole as is illustrated in Figure 8. Assume the explosive used is 40 feet of ANFO with a velocity of detonation equal to 13,000 ft/sec, the material blasted is limestone with a sonic wave velocity of 15,000 ft/sec and a density of 2.3 g/cc. Upon initiation of the primer, it takes only a few microseconds and a distance of 2 to 6 hole diameters up the column to form a full detonation head. When a full detonation head is formed, it travels up the explosive column with a velocity characteristic of the steady-state velocity, (in this case 13,000 ft/sec). It takes approximately 3.0 msec for the 40 foot column of ANFO to be completely detonated.

Within this 3.0 msec, many other things have occurred. Starting at the bottom of the hole and progressing up the column, borehole expansion through crushing of the borehole walls has taken place. This produces compressive stress waves with tangential components emanating from the borehole walls and progressing outward in every direction with a velocity characteristic of the sonic wave velocity of limestone. It takes approximately 1.0 msec for the compressive strain wave to traverse 15 feet of burden to the free face. Behind the strain wave propagation some radial cracks start to develop in the crushed zone region of the borehole with a velocity ranging from 25 to 50% of the P-wave velocity for limestone. If the intensity of the compressive strain pulse is high enough, new crack and/or extensions of pre-existing cracks

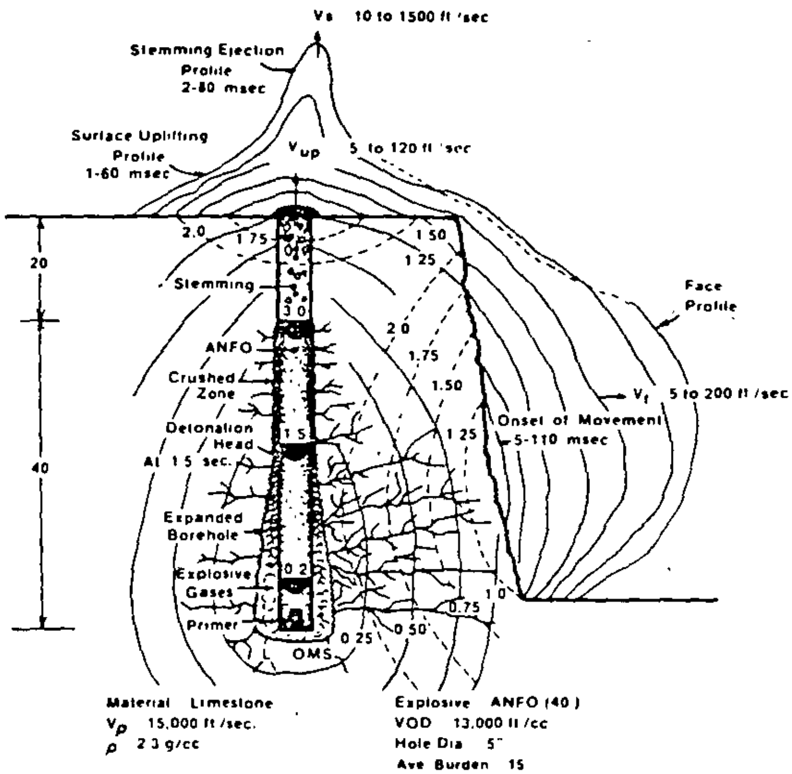


ILLUSTRATION SHOWING THE INTERACTION OF TIME EVENTS T1 TO T4 IN A TYPICAL QUARRY BENCH

FIGURE 8

and flaws can be initiated anywhere between the crushed zone next to the borehole and the free face. The greatest number of cracks are generally found closest to the borehole.

When the compressive wave strikes a free face, it is immediately converted to a tensile strain wave which starts at the free face and travels back through the rock mass towards the borehole. Owing to the new fractures created from the outgoing compressive strain wave, the tensile strain wave will take somewhat longer to travel the same burden distance of 15 feet. If the burden is small enough and the intensity of the reflected strain wave is large enough, then some spalling at the free face or bench top is expected, although no significant mass movement will occur.

At 3 msec after detonation and assumed complete reaction of ANFO, the original high temperature, high pressure gases have reached a new equilibrium due to borehole expansion. Both temperature and pressure have dropped significantly resulting in an energy reduction ranging from 25 to 60% of the theoretical energy originally available. This remaining energy acts on the surrounding "preconditioned" rock mass to displace it in the direction of least resistance. Further fragmentation can occur at this stage from gases entering and extending preexisting cracks or discontinuities. It is at this stage where some blasting theories are contradictory. Some believe that the major fracture network is completed within about 3 msec due to the interaction of stress waves on the surrounding material, while others believe that the major fracture network is just beginning.

Regardless of which time frame is responsible for the development of a fracture network, mass movement and displacement of material at the bench top or face occurs much later in time due to the confinement of gas pressure within the rock mass. The onset of mass movement depends on the material response in conjunction with the strain and gas pressure stimulus generated from the explosive. For typical stemming and burdens encountered in the field, bench top swelling occurs between 1 to 60 msec, stemming ejections between 2 to 80 msec and bench burdens between 5 to 110 msec. Surface uplifting velocities around the collar region of a hole occur between 5 and 120 ft/sec, stemming ejection between 10 to 1500 ft/sec and burden velocities between 5 to 200 ft/sec. Gas ejection velocities at discontinuities have been recorded as high as 700 ft/sec and often occur in less than 5 msec.

CHARACTERIZATION OF ALTERNATE VELOCITY LOADING WITH AN EMULSION EXPLOSIVE IN ANFO

It is evident from the discussion so far that the selection of conventional blast design variables can have a pronounced effect on overall results regarding fragmentation and mass movement of burdens. Without making major modifications in the blast design, a simple and cost effective technique to further improve fragmentation and/or increase burden velocities is with Alternate Velocity Loading. Alternate Velocity Loading is achieved by sparingly placing a cartridge or slug of emulsion explosive every few feet in the ANFO column (refer to Figure 9). Field trials conducted in a wide variety of materials have generally resulted in improved fragmentation and increased burden velocities. It appears that when an explosive of higher density and detonation velocity (i.e., emulsion) is embedded within the column of the main charge with a lower density and detonation velocity (i.e., ANFO), improvements in blasting results are the norm. Whether the material exhibits physical and strength properties characteristic of granites, limestones and dolomites, or overburden, unconsolidated type materials, the results are often dramatic and repeatable.

In the course of implementing the Alternate Velocity Technique, a few groups in the industry believed that ANFO did not possess sufficient detonation pressure to act as an effective primer on the emulsion explosive. They also stipulated that the technique could not be cost effective because each emulsion cartridge or slug required a primer and detonator assembly for it to succeed.

As an investigative approach to address these thoughts, the ATLAS POWDER COMPANY embarked on a joint research program with SANDIA NATIONAL LABORATORIES where over 100 holes were fired under controlled conditions in a field environment. Instrumentation consisted of continuous velocity probes (Slifer System), pressure sensors to measure gas pressure, accelerometers to measure shock in the surrounding rock mass, survey gear to quantify the extent of damage, mounds, craters and multiple high-speed cameras to quantify gross movement and gas venting.

An Alternate Velocity test was performed in a full-scale, 6-1/2" diameter, production hole, with ANFO (ρ = 0.81 g/cc) as the main charge, and an Apex grade of cartridged emulsion explosive (5" x 30lb, ρ = 1.25 g/cc) as the Alternate Velocity explosive. Borehole depth was approximately 40 feet and each emulsion cartridge was 3' in length. Continuous velocity of detonation measurements were performed by Sandia National Laboratories using the Slifer System for the entire length of the hole.(8) Results are illustrated as displacement versus time plots in Figures 9 and 10.

Explosive loading, starting at the bottom of the hole, consisted of a one pound ATLAS Cast G Booster, followed by a

3 foot cartridge of emulsion, 10 feet of ANFO, a 3 foot cartridge of emulsion, 11 feet of ANFO, a 3 foot cartridge of emulsion, and approximately 10 feet of 1/2" - 1-1/8" of crushed rock for stemming.

Figure 9 illustrates the actual, untouched and unfiltered results obtained from the field, while Figure 10 illustrates a filtered close-up of results in the bottom 20 feet of the borehole. The filtered close-up gives better resolution and allows for more accurate measurement of detonation velocity. Detonation velocity results are presented in Table 2.

TABLE 2

DETONATION VELOCITY RESULTS FOR THE ALTERNATE VELOCITY TEST (Emulsion and ANFO)

COLUMN POSITION	COLUMN LENGTH (ft)	AVERAGE VELOCITY OF DETONATION (ft/sec)	COLUMN EXPLOSIVE/MATERIAL
A - B	3	20,500	EMULSION
B - C	5	13,000 - 2,045	WET ANFO
C - D	5	2,045	WET ANFO
D - E	3	20,000	EMULSION
E - F	11	12,500	DRY ANFO
F - G	3	9,000 - 16,000	EMULSION ANFO STEMMING
G - H*	10	1,000	STEMMING

* Shock wave velocity through stemming.

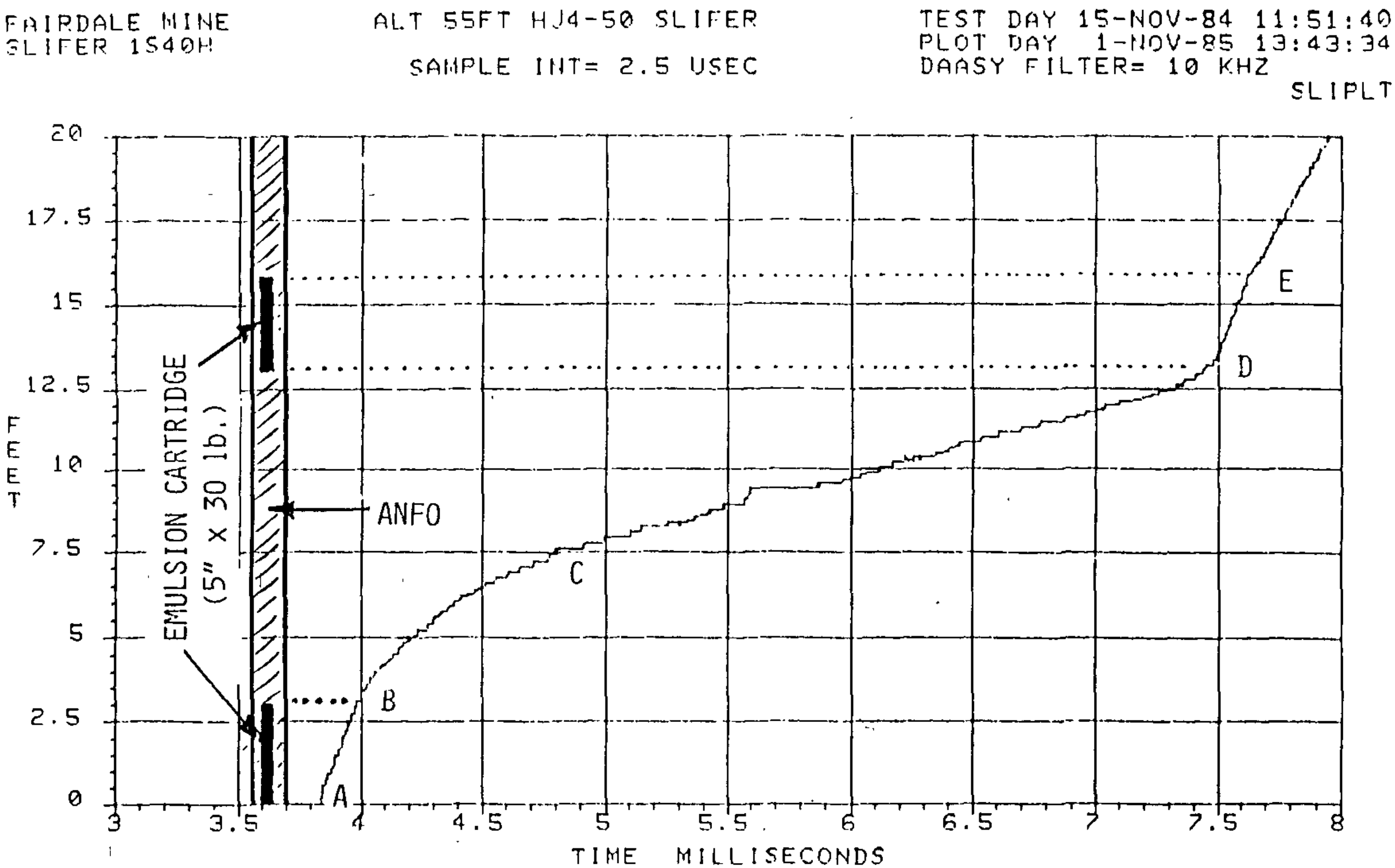


FIGURE 10

FILTERED CLOSE-UP OF FIGURE 9

(FAIRDALE QUARRY, Macklin Brothers Stone Company, Fairdale, IL)

Many important and interesting points are noteworthy in the results. Between points (a) and (b), the velocity of detonation for the 3 foot length of emulsion cartridge is 20,500 ft/sec. Between (b) and (c), the velocity of detonation in ANFO is quickly reduced from 13,000 ft/sec to 2,045 ft/sec and the ANFO detonation at this velocity is sustained until just before point (d) is reached. Just before point (d) is reached, or 6" below the second emulsion cartridge, ANFO is increasing in detonation velocity from 2,045 ft/sec to 4,900 ft/sec. ANFO with a detonation velocity of 4,900 ft/sec and a density of 0.81 g/cc is equivalent to a detonation pressure of approximately 4.5 Kbars. At point (d), the detonation head in ANFO encounters the second emulsion cartridge, which when detonated, at 20,000 ft/sec between points (d) and (e), brings ANFO immediately back up to its normally rated detonation velocity of 12,500 ft/sec. <u>Thus, even a low order ANFO detonation can act as a very effective primer for the emulsion cartridge without additional boostering on the emulsion cartridge.</u>

The decrease in the ANFO detonation velocity between points (b) and (c) is attributed to water trickling into the bottom 12-1/2 feet of the hole from the surrounding rock mass.

> The drill hole was drilled one week in advance of testing and it had accumulated approximately 3 feet of water. Prior to testing, the water in the hole was pumped out and it was determine that in one hour, the hole could accumulate about 2" of water in the

> bottom. The 2" of water was again pumped out and the hole was backfilled with 6" of crushed rock so that the bottom of the hole would be out of water.
>
> It took approximately one hour from the time of backfilling the hole to firing the shot hole. We feel that there was sufficient wetness and/or tiny amounts of water trickling into the borehole walls for the bottom 12-1/2 feet of the test hole to affect the performance of ANFO. Although the wet ANFO was not purposely planned as part of the test, it does illustrate that even a low order ANFO detonation of 4,900 ft/sec can act as an effective primer on an emulsion. It also illustrates that when water is pumped out of a hole and the hole is loaded with ANFO, ANFO performance can still be drastically affected just by the wetness on the sides of the borehole. Inadequate priming at the bottom of this hole would have probably resulted in a failure.

Although ANFO can tolerate up to a 10% water saturation level, it does so at the cost of blasting efficiency. If the center emulsion cartridge was not present, one of two things could have occurred. ANFO may have sustained a low order detonation throughout the remaining column until dry ANFO was reached, or it would have soon failed due to its unstableness. Between points (e) and (f), the detonation velocity in dry ANFO was 12,500 ft/sec, as expected. Between points (f) and (g), the average velocity of detonation in the top cartridge appeared to fluctuate between 9,000 - 16,000 ft/sec due to the combined effect of emulsion, ANFO and stemming. For optimum results, the top emulsion cartridge should have been placed one to two feet below the stemming column where it would have been completely embedded in the ANFO column. Between points (g) and (h), the average shock wave velocity through stemming was 1,000 - 1,100 ft/sec.

ALTERNATE VELOCITY FIELD TESTS IN FULL-SCALE PRODUCTION SHOTS

A study was conducted in a Southeastern granite quarry to evaluate relative explosive performance in terms of burden velocities for normal and Alternate Velocity Loading Techniques. Four full-scale production shots were made. Three shots were loaded conventionally, using a different type or grade of bottom charge, amounting to seven 2-3/4" x 16" cartridges or nine feet of bottom load and ANFO as a top load. The bottom loads in the conventional shots consisted of Powermax 440 (emulsion), Powermax 460 (emulsion) and a watergel explosive. The fourth shot was made with the same type and amount of bottom charge of Powermax 460 with four additional cartridges of Powermax 460 spaced at 6 foot intervals as the alternate velocity load in the ANFO column. All other blast design variables such as burden, spacing, hole depth, millisecond delay pattern, blasting direction, etc. were kept as constant as was reasonably possible in a production environment. High-speed motion picture photography was used to quantify, evaluate and compare results.

Average face velocities were calculated for three areas of the face; at the toe, 12 feet above the toe, and 24 feet above the toe as illustrated in Figures 11 to 14. Analysis was accomplished by designating the first face profile as occurring at zero time to produce a plot of displacement versus time.

Average burden velocities were calculated using linear regression analysis. Results are summarized in Table 3. Figure 15 illustrates the Table 3 values in graphical form. It is evident that the greatest average burden velocity is achieved with PMX 460 as a bottom load and PMX 460 as alternate velocity boostering of ANFO in the upper load. PMX 460 as a bottom load and straight ANFO as a top load resulted in the second highest burden velocities, followed by the watergel and ANFO, and PMX 440 and ANFO. An error analysis between the last two, (Watergel and ANFO, and PMX 440 and ANFO), revealed that, statistically, the results are equivalent or in other words, there is no significant difference owing to an experimental error of ±5 ft/sec.

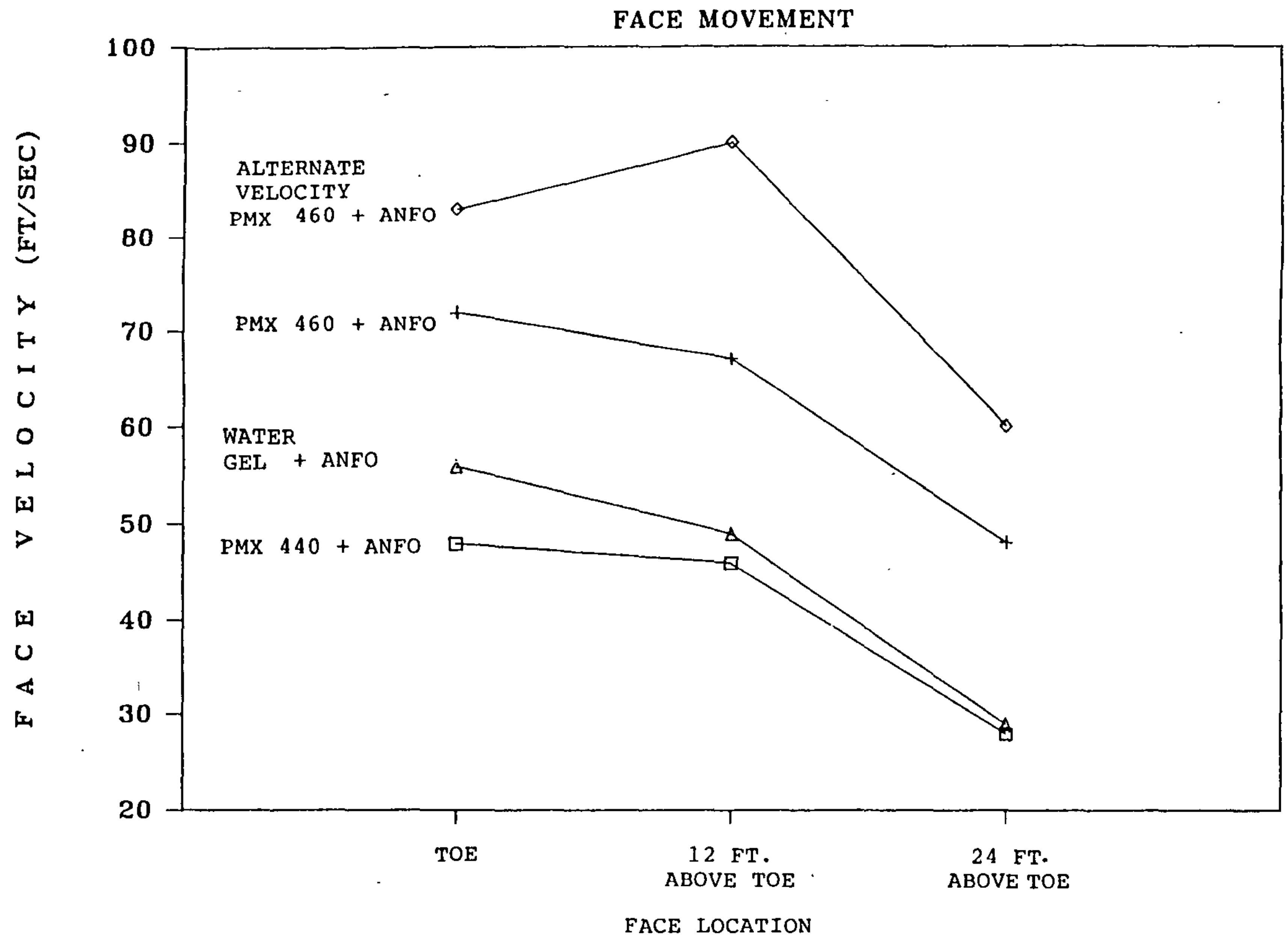
A more detailed analysis has also been performed to compute the instantaneous velocity at any point in the time domain for movement at the toe and 12 feet above the toe. Displacement and time data were fitted to a polynomial equation of degree five with a goodness of fit of no less than 99%. The first derivative of this equation forms a velocity equation, which when plotted, yields the graphs illustrated in Figures 16 and 17 for the instantaneous velocity at the toe and 12 feet above the toe, respectively. Not enough data was available to perform an analysis to the same degree of fit and accuracy for face movement 24 feet above the toe.

TABLE 3

AVERAGE BURDEN VELOCITIES

SHOT NO.	BOTTOM EXPLOSIVE LOAD	TOP EXPLOSIVE LOAD	FACE LOCATION AND VELOCITY (FT/SEC) TOE	12' ABOVE TOE	24' ABOVE TOE
1	POWERMAX 440	ANFO	48	46	28
2	POWERMAX 460	ANFO	72	67	48
3 *	POWERMAX 460	POWERMAX 460 EVERY 6' IN ANFO COLUMN	83	90	60
4	WATERGEL	ANFO	56	49	29

* POWERMAX 460 was used as the Alternate Velocity Explosive


FACE MOVEMENT
100
90
80
70
60
50
40
30
20
FACE VELOCITY (FT/SEC)
ALTERNATE
VELOCITY
PMX 460 + ANFO
PMX 460 + ANFO
WATER
GEL + ANFO
PMX 440 + ANFO
TOE
12 FT.
ABOVE TOE
24 FT.
ABOVE TOE
FACE LOCATION

FC.

FIGURE 15

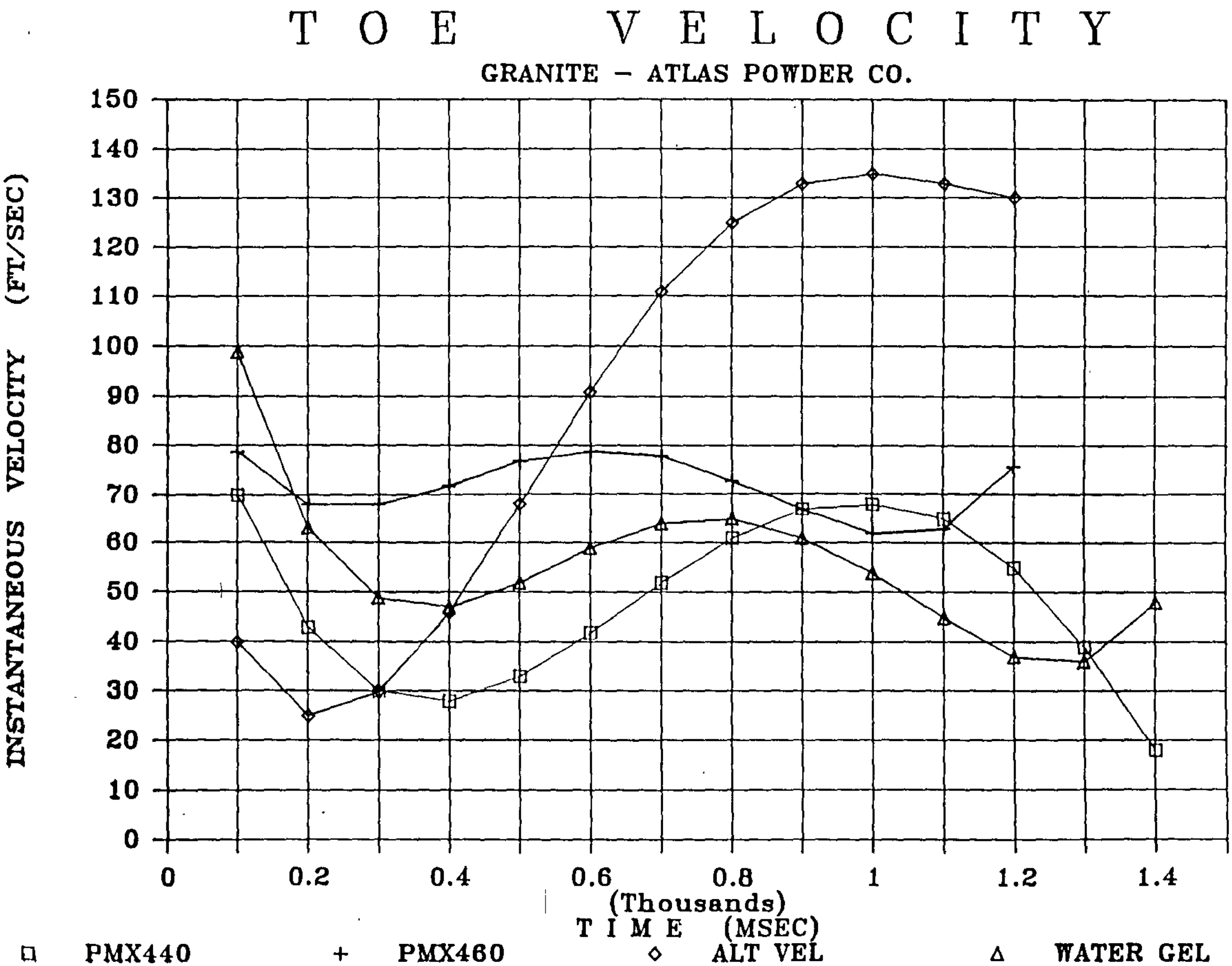


FIGURE 16 INSTANTANEOUS VELOCITY vs TIME FOR TOE MOVEMENT

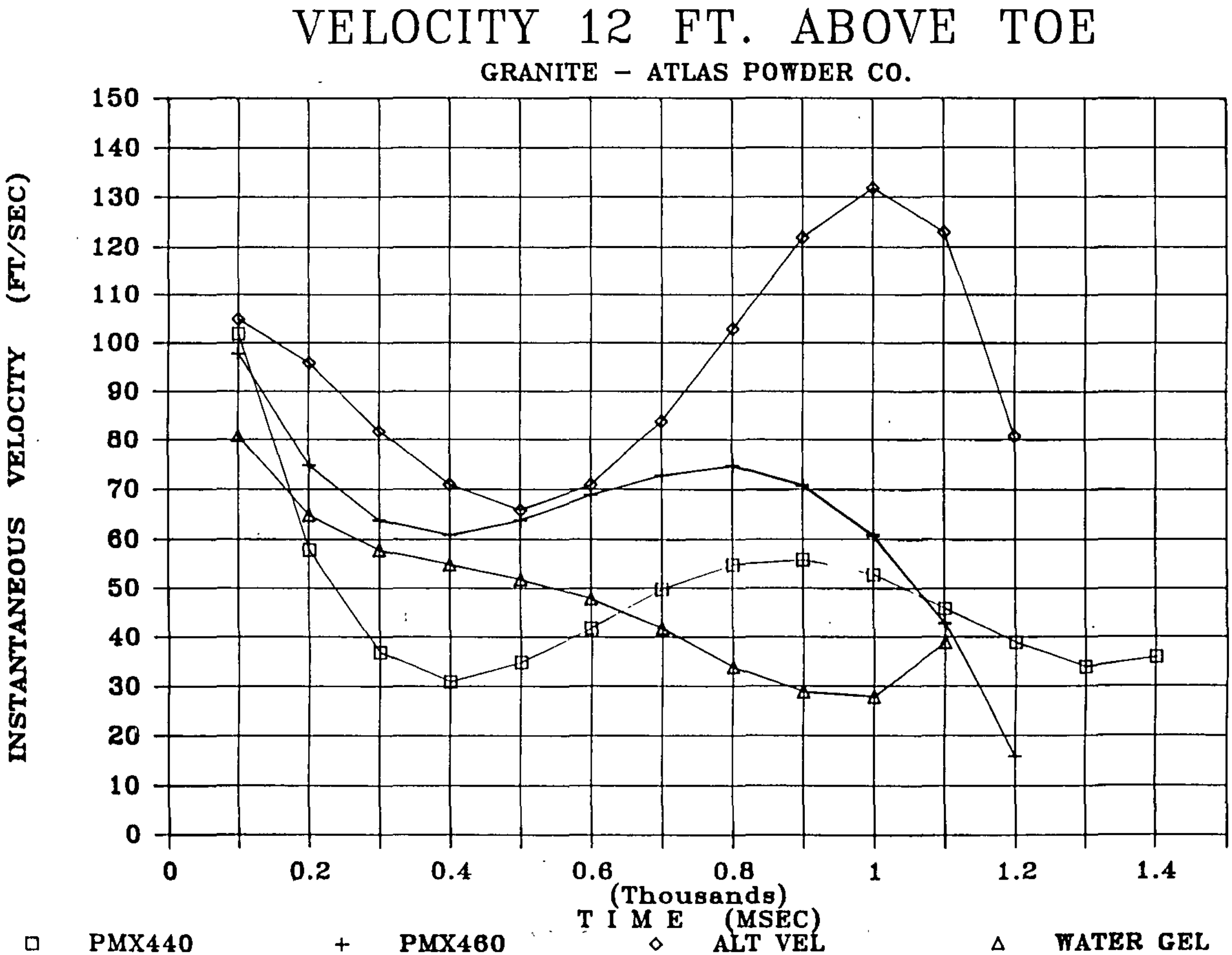


FIGURE 17 INSTANTANEOUS VELOCITY vs TIME FOR BURDEN MOVEMENT 12 FEET ABOVE TOE.

In Figure 16, the PMX 460 bottom load with alternate velocity boostering of ANFO in the upper load generated the most dramatic velocity increase between 200 to 1000 msec into the shot. Beyond 550 msec, velocity for this load configuration exceeded all others. Movement at the toe in this case was attributed to the bottom load and to the bottom third of the upper column. Thus, toe velocity in this case was affected by both bottom and top loads.

If the bottom load is one of much higher energy, it will generate a larger crushed zone and improve fragmentation in that vicinity. It also creates a larger cavity for explosive gases to immediately migrate into and fill the new formed cavity. Gas temperature and pressure drop quickly and it may be for this reason that the initial toe velocity is somewhat less until the alternate velocity boostering temporarily reverses the process by releasing more of the available energy in ANFO as higher temperature, higher pressure gases. This dramatic late increase in gas pressure is evidenced in Figure 16 for the PMX 460 alternate velocity boostering from 200 msec on and is sustained until well over 1000 msec. The key to moving burden material at higher velocities is to sustain these high pressures with appropriate energy and by the timed systematic and controlled release of such energy with precise MS delays.

Instantaneous velocities 12 feet above the toe (Figure 17) are attributed primarily to the upper column load. In this case, the PMX 460 with PMX 460 as alternate velocity boostering in the ANFO column achieved the greatest burden velocity for all times.

It is not definitely clear as to the exact mechanisms and/or processes which actually contribute to increased burden velocities without direct and continuous measurements of temperature and pressure within the borehole. However, based on direct measurement of burden velocities and the fact that it is gas pressure expansion in the borehole and in the surrounding rock mass that is responsible for mass displacement, certain logic can be suggested as follows:

1) Since ANFO at best is only 60 - 70% efficient, complete reaction of the ANFO is not occurring as predicted in the detonation head. The addition of alternate velocity boostering may contribute to a more complete reaction of ANFO just behind the detonation head in the partially reacted and expanding gas phase of ANFO.

2) The emulsion cartridges used as Alternate Velocity boostering may generate higher temperatures and pressures in their vicinity and thus raise the overall temperature and pressure of the combined emulsion cartridge and ANFO detonation in the new formed cavity. This would tend to pressurize the cavity at higher pressures and possibly for a longer period before the burden responds to mass movement.

3) More of the <u>available</u> energy in ANFO is utilized with the technique in production holes less than 17 inches in diameter.

SUMMARY AND RECOMMENDATIONS FOR FIELD USE

Alternate Velocity Boostering of ANFO with emulsion explosives has resulted in the following benefits:

1) The technique is simple to employ without major modifications to the overall blast design.

2) Improves overall fragmentation, especially in the vicinity of the emulsion cartridge or slug.

3) Definitely increases burden velocities and mass movement.

4) Increased burden velocities will result in increased cast distances, lower and looser muckpiles and minimal back break and spills.

5) A loose muckpile will reduce maintenance costs on excavation equipment through less wear and tear on buckets and tires.

6) The technique can be used in any application regardless of material properties or structure where the above mentioned benefits are desirable.

7) The Alternate Velocity Emulsion explosive does not require additional priming. ANFO is more than sufficient.

8) When ANFO is used in dewatered holes, the alternate velocity emulsion can enhance and increase the detonability of ANFO.

9) It appears that the greater the difference in density and detonation velocity of the alternate velocity explosive to ANFO, the more pronounced are the results.

10) In essence, the technique allows more of the available energy in ANFO to be converted to useful work. It is cost effective, and a productivity increase should be realized over the short and long terms.

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UNITED STATES BUREAU OF MINES - M. Staggs, J. Knopp, and M. Nutting.

MACKLIN BROTHERS STONE COMPANY - R. Macklin

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CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

THE VELOCITY OF DETONATION RECORDER A NEW BLAST AND SHOCK WAVE DIAGNOSTIC TOOL FOR COMMERCIAL USE

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

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THE VELOCITY OF DETONATION RECORDER

A NEW BLAST AND SHOCK WAVE DIAGNOSTIC TOOL FOR COMMERCIAL USE

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The knowledge of how and when your explosives go off can help you make intelligent decisions regarding future application of explosives thus removing some of the black magic associated with blasting. The net result will be intelligently set-up shots with timing, explosive charges, and placement more accurately done. Lower operational costs can be achieved when the correct combinations are applied.

This article describes a technique for accurately measuring explosive detonation time, burn rates, and shock wave propagation in a straightforward and simply implemented fashion. Further, application of this technique can be used to collect information from large scale blasting operations where there are multiple holes and sequenced firings of explosive.

The technique has been used for many years in underground nuclear tests. The technology (Time Domain Reflectometry, TDR) is well understood and is a standard diagnostic technique in many fields. In the nuclear field the technology fielded is called CORRTEX for Continuous Reflectometry for Radius versus Time Experiments. Developed at and fielded from Los Alamos National Laboratory, New Mexico, its major use has been to estimate the yield from underground nuclear explosive detonations both in the U.S. and in the Soviet Union. This technology also has a useful and important role in the explosive industry, however, and will be particularly useful when high timing accuracy detonators are readily available.

The concepts involved are similar to that of RADAR where a pulse of radio waves are sent out and a echo or reflected pulse is returned to give ranging information. The technique uses a coaxial cable to carry a fast rise-time electrical pulse back and forth. The time between the sending of the pulse and its return is accurately measured. Knowing how the time changes from pulse to pulse gives an accurate picture of the length of the cable in time. This is the underlying concept for the CORRTEX and of the Velocity of Detonation Recorder (VODR), the commercial version of the CORRTEX.

In the measurement of explosive performance the coaxial cable is laid out along the length of the explosive with the far end of the cable near the detonator (see Figure 1).

As the explosive is detonated (and burns), the cable becomes progressively shorter. The VODR accurately measures the time interval and therefore the length of the cable from moment to moment.

The collected information can then be processed into a useful form which may be tabular or graphical in nature (see Figure 2). From the displayed data both burn rates and timing information can be obtained and compared to expected information.

To further understand the technology involved you need to know something about pulses traveling along electrical cables. First they travel a little bit slower than the speed of light which travels at 299 million meters per second. This is many times faster than the fastest explosive burn rate. Secondly, the electrical pulses are reflected back by discontinuities in the transmission line. Thus, a pulse is reflected at the point where the cable is crushed or severed by the explosive shock wave. The return pulse then contains useful information about exactly where the shock wave is passing.

To measure distances very accurately using time you must have a high resolution timer. For the unit designed, the resolution is down to 125 picoseconds for a two-way transit distance on an electrical cable of about 0.6 inches. The fine resolution is useful for examining events occurring close to the detonator.

The unit sends pulses down the electrical cable up to 100,000 times a second at its maximum pulse rate. Consider an explosive with a burn rate of 6000 meters a second. In the 10 microseconds between pulses the explosive can burn about 2.4 inches. Thus, the time resolution of 125 picoseconds is about the correct resolution considering that most explosives have a significantly slower burn rate.

COMMERCIAL CORRTEX

The Commercial version of CORRTEX, the VODR is smaller and more highly integrated than some of the earlier units built for underground nuclear yield monitoring. The commercial unit is a self contained unit about the size of a large suitcase. An external 24 volt, 15 amp-hour battery is used for power allowing it to operate for about 5 hours. The external battery charger will allow the battery to be completely charged in about three hours from a 120 volt 60 Hz source.

An operator communicates with the VODR with a standard IBM personal computer keyboard. A graphics liquid crystal display permits both the display of alphanumeric characters as well as graphics. The combination of the IBM style keyboard and the display makes for an easy operator interface. A 5-inch wide thermal graphics printer allow easy hard copies of the data.

APPLICATION OF THE REPETITIVE TIME DOMAIN REFLECTOMETER

For explosions involving multiple detonations, the cabling is laid out so the end of the cable is located where the first detonation begins and is then laid out so the next detonation and shock wave sequences make the cable progressively shorter to where the VODR instrumentation is located (see Figure 3).

Often several different channels of data may be required to fully instrument a shot. Repetitive Time Domain Recorder has the capability to collect data from two different coaxial cables. The frequency at which each cable is sampled is reduced, however.

In addition to the time domain reflectometer capabilities the VODR has remote analog recording capabilities that may be used for various environmental measurements having an influence on shot performance parameters. Examples of such data are temperatures and pressures. It is not intended at this time that analog recordings be made at the same time as the operation of the time domain unit reflectometry unit.

FIBER OPTIC TIME DOMAIN REFLECTOMETRY

Fiber optic cables use light waves for the transmission of data down a thin thread of glass. As with electrical cables, part of the light is reflected off the end of a broken fiber. The time between the transmission of the light and the return of the reflected pulse is a strong indication of how long the fiber is.

Except for the fiber optic cables and the special interface board the fiber optic system operates in the same fashion as the coaxial cable unit.

At the present time the fiber optic version of the CORRTEX unit is under development.

In the system EG&G has developed, we have considered the needs of the blasting engineer. Data is available in the raw form, and in a graphical mode both processed and unprocessed. Hard copies of the data are available from a 5-inch wide thermal printer. Additionally, the data can easily be loaded into a lap top personal computer for permanent storage.

For more exacting technical information, see the technical description note.

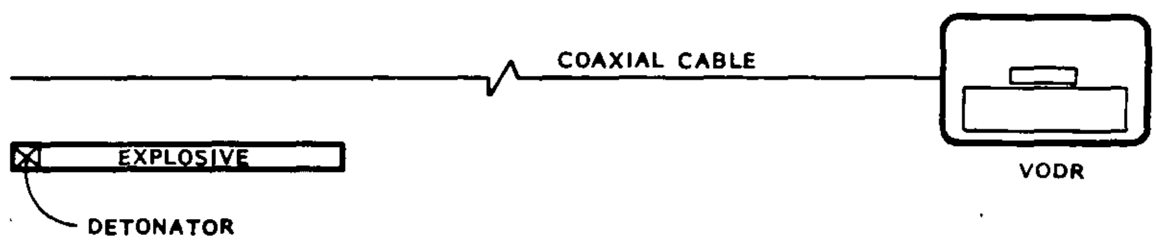

Figure 1. Principle of Operation

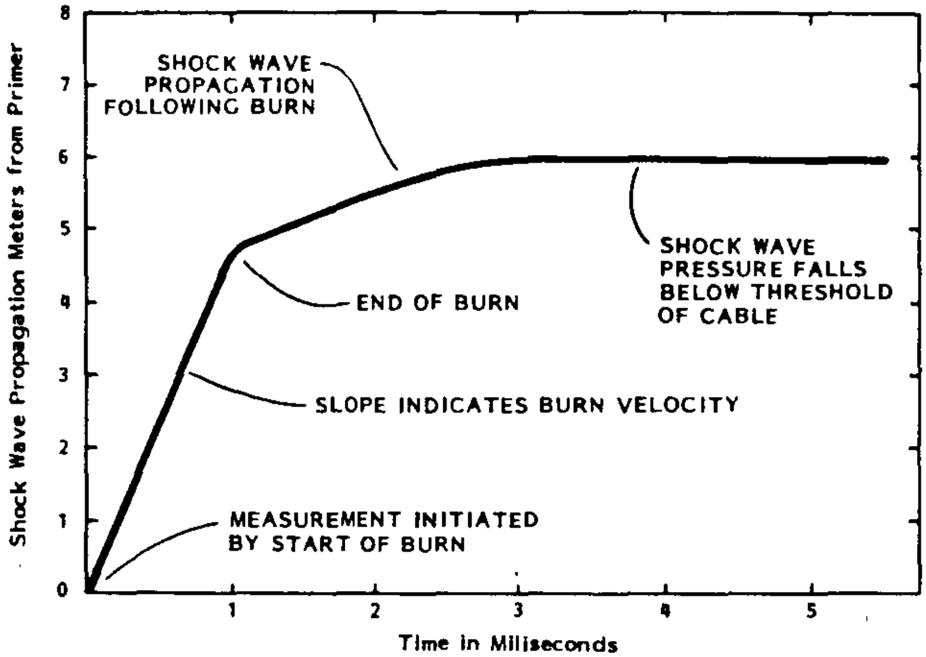

Figure 2. Display of Data

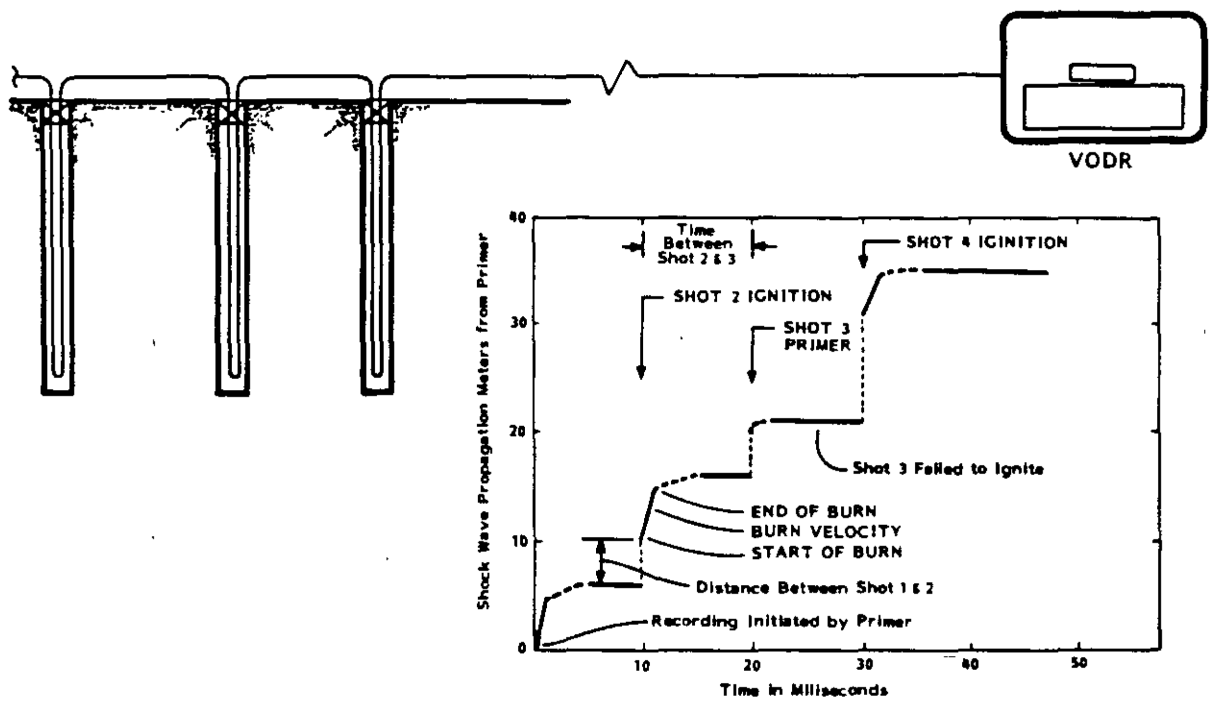

Figure 3. Example Application

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THE LAWS OF ENERGY DIVERGENCE

By Richard L. Ash, P.E.

Energy transmitted through a homogeneous isotropic medium from an explosion-induced confined source is diverged outward equally in all directions. The effect is to reduce the unit energy at any one position within the medium as the distances increase away from the explosion center. This phenomenum as applied to mass, volumetric, or weight considerations is described by what is known as the cube-root law: whereas, the decrease of the effective stress or pressure with distance in any single direction is expressed by the square-root law. Both of the two laws serve as the principle bases for all explosive-energy design criteria. It is most important, therefore, that the laws be clearly distinguished from one another as to their specific applications.

The first assumption made in the use of the laws is that the explosion source consists of a single concentric or point charge, i.e., one that is initiated in its center and whose length-to-diameter ratio (L_e/D_e) is not more than 6. However, the laws can be applied equally as well to the long-length cylindrical charge normally used for the bulk of industrial blasting, providing one recognizes that only in the planes of the charge length will the effects differ from those produced by the concentric charge.

To comprehend the unique characteristics of the long-length charge it is convenient to consider that it is essentially no more than a continuous series or succession of individual point charges. Thus, stress energy from each segment normallily will be released in a definite sequence or progressive order, from the point of initiation and then preceding to each adjacent portion of the charge remaining in the column. The resultant composite stress-form transmitted into the surrounding medium will vary in shape from that of a sphere to that of a cylinder with hemispherical ends, depending on the method of initiation used and the properties of both the explosives and containing material. For the cylindrical wave to form for example, all points along the column must be initiated simultaneously. This is highly improbable for most types of blasting except for very short columns that are initiated in the center or moderate length charges that contain closely-spaced primers which are all initiated by means of instantaneous electrical blasting caps. Instead, the composite wave-form usually is either some form of conical shape or spherical. The conical wave results when the reaction velocity through the explosive column exceeds the compressive wave velocity of the surrounding medium. In this instance stresses from the charge ends in each of the planes intersecting the axis of the explosive column will be diverged like those from the spherical charge with changing angles of incidence at free-face planes, while stresses in the planes of the axis from the central portion of the column will be directed with a constant angle of incidence. Stresses in the planes of the charge diameter and perpendicular to the column axis, however, will be diverged much in the same circular manner as those produced from the spherical point charge.

The Cube-Root Law

For the idealized condition of the single point charge, as would be also the case for each single segment within the long-length column charge, the divergence of energy through a material propagated from an explosion could be considered to be contained within a sphere of influence whose volume would be a function of the cube of the distance of propagation away from the explosion's center or $V_d = 4\pi d^3/3$. It follows, then, that for the energy to spread out within a volume whose radius is twice that of the charge, the explosive's energy would have been dispersed throughout a volume equal to 8 times that occupied by the explosive. In the event the radius of influence considered in the material is 5 times the charge radius or $d = 5D_e/2$, then the volume of the expanded sphere of influence would be 125 times that of the charge. In other words, the ideal magnitude of the energy contained within one unit volume of material at a distance of d from an explosion center (dQ_d) would be equal to the product of the unit energy released initially from the explosion (dQ_e) TIMES the specific ratio of the explosive's initial volume to that of the sphere of influence in the material affected. Mathematically, the general relationship would be

$$dQ_d = dQ_e(V_e/V_d).$$

Because the volume of the explosive would be a function of its charge diameter, or $V_e = (D_e/2)^3$, and the volume of the sphere of influence in the material being stressed by the explosive is proportional to the distance away from the explosion center, where $V_d = d^3$, it can be concluded that the fraction of the explosion's unit energy contained within a unit volume of the material will be equal to

$$dQ_d/dQ_e = (D_e/2d)^3.$$

In similar manner, it can be reasoned that for any specific spherical explosive, its total weight and amount of available energy would depend to a large extent on its unit density (SG_e) and charge diameter (D_e), providing velocity effects are excluded. Thus, a point charge with a 4-inch diameter could be expected to contain 8 times the energy and weigh 8 times that of a 2-inch diameter point charge of the same explosive. However, the total weights would differ between a concentric point charge and a long-length column charge with identical charge diameters by a factor equal to the ratio of their respective lengths; although the energy concentration PER FOOT OF CHARGE LENGTH would be identical. Therefore, it is generally preferred to use the unit density (SG_e) or loading density (d_e) of an explosive as the standard for comparison, rather than total charge weight. In the event the explosive's velocity does NOT remain constant, it is then necessary to account for its difference in order to consider the energy quantity for each condition. As a general rule, the relative energy (RE) would be a function of the product of the loading density and reaction velocity squared, or $d_e v_e^2$. Therefore, using the concept of relati energies from different explosives and under different blasting conditions, it is possible to make reasonable approximations of the energy capabilities of various kinds of explosives and the estimated level of the transmitted stresses at any distance d from the explosion center. With respect to the relative energies of explosives with different densities but having a constant charge diameter and velocity of reaction, the following expression can be used:

$$RE_2 = RE_1(SG_{e2}/SG_{e1})$$

In the case where explosives' densities, charge diameters, and reaction velocities all differ, the general relationship for making the comparison would be

$$RE_2 = RE_1 \ (d_{e2}v_{e2}{}^2/d_{e1}v_{e1}{}^2)$$

For all practical considerations, the comparisons would be reasonably valid for all center-initiated 1-ft long charges with diameters that would vary from 2 to 72 inches. This is because the range of values for L_c/D_e and its reciprocal, or D_e/L_c, would not exceed 6, which was defined earlier as the limitation for a point charge.

The significance of the foregoing relationships becomes apparent when one considers their application for cratering in materials. The problem, in gist, is one involving the accomplishment of mechanical work, whereby the energy supplied by the explosive (Q_e) is used for fracturing the materials by overcoming their strength properties and then displacing the broken particles. In general, the required diverged value of δQ_e, or δQ_d at distance d from the explosion's center, will be unique for any given type of material.

The specific depth of charge burial, which would correspond with the maximum limit for distance d, at which optimum crater results will be achieved is called the burden, B. The volume of the developed crater (V_c), in turn, will always be a function of B, as well as the explosive's Q_e. For example, V_c for a simple cone-type crater with one free surface is $\pi r^2 B/3$, but the value for the crater radius r is dependent on the material's properties and is related to B. Thus, as a general rule one can assume that $V_c \approx B^3 \approx Q_e$ for approximation purposes. From the previous discussions it was shown that $Q_e \approx RE_e \approx SG_e \approx D_e{}^3$. Therefore, for any confined explosive charge it can be concluded that

$$B \approx V_c{}^{1/3} \approx RE_e{}^{1/3} \approx SG_e{}^{1/3} \approx D_e,$$

and

$$B_2 \approx B_1(RE_2/RE_1)^{1/3} \text{ or } B_2 \approx B_1(D_{e2}/D_{e1}).$$

In summation, the cube-root law describes the effect of three-dimensional divergence in reducing the stress energy produced by a confined explosive charge as the energy propagates in all directions through a material. As a result, the ideal optimum depths of burial (or burden) and the resultant crater volumes produced by confined explosive charges in a given blasting environment will be proportional to the charge diameters and the cube root of their respective relative energies.

The Square-Root Law

Because the energy of an explosive is released as a pressure or stress, i.e., P_e or σ_e, it will exert itself over the entire surface of the charge. For the concentric or point charge the total outer surface at which the pressure acts is equal to $4\pi r_e^2 = \pi D_e^2$. Therefore, the total energy from the explosion (Q_e) will be equal to $\sigma_e(\pi D_e^2)$. By similar analysis, from the explosion center the stress transmitted through a material a distance d will be distributed over a surface area equaling $4\pi d^2$, in which the total energy at that location σ_d times the area would be related in the form of $Q_d = \sigma_d(4\pi d^2)$. Assuming there are no losses because of absorption, etc., or $Q_e = Q_d$, it follows then that

$$\sigma_e(\pi D_e^2) = \sigma_d(4\pi d^2),$$

or $$\sigma_d = \sigma_e(D_e/2d)^2.$$

If $d = B$ for the optimum crater produced, then

$$B = (D_e/2)(\sigma_e/\sigma_B)^{1/2}$$

Because the stresses transmitted into a material are proportional to the pressure released by the explosion, or $\sigma_e = P_e$, the transmitted stresses for the production of a crater must equal or exceed the material's strength properties, or σ_t or τ_s depending on which would be the most critical. Thus, one can conclude that

$$B = P_e^{1/2} = \sigma_t^{-1/2} \text{ or } \tau_s^{-1/2}.$$

In brief, then, the square-root law of energy divergence, when used to predict requirements for the production of craters, states that the optimum depths of burial (or burden) and the resultant crater volumes for confined explosive charges in any given blasting environment will vary directly with the square root of their respective explosion pressures and inversely with the square root of the pertinent materials' strengths.

SINGLE BLASTHOLE DESIGN PROBLEM

A deposit is quarried in 30-ft high benches for crushed stone. The rock is quite massive and has the following properties:

$$SG_r = 2.9, \quad v_p = 17{,}000 \text{ fps}, \quad \mu = 0.25, \quad S_f^s = 0.7,$$

$$\gamma = 45 \text{ deg}, \quad \sigma_c = 25{,}000 \text{ psi, and } \sigma_t = 1750 \text{ psi.}$$

Blasted rock is loaded by a 5 cy front-end loader. The blastholes are drilled vertically and bulk loaded ($D_e = D_h$) with an explosive having an SO = 117, D_c = 1 in., and confined velocities of 12,500 fps at 3 in. and 15,000 fps at 5 in. and larger charge diameters. The relationship between v_e and D_e in the 1 to 5 in. range can be assumed to be in the form of

$$y = \frac{cx}{a + bx} .$$

Drainage at the operation is such that blastholes generally are always dry, and there is no free parting in the rock available that can serve as a floor. For estimating purposes the average blast area A of material cratored by a single blasthole would be equal to $1.4B^2$.

A. Considering the foregoing information, find the following properties for the intact rock:

(1) τ_s, and (2) E_r.

B. For charge diameters D_e of (a) 2 in., and (b) 4 in., determine each of the following estimates:

(1) v_e, (2) P_d, (3) P_e, (4) B, (5) T, (6) J,

(7) E, (8) W, (9) t_f, and (10) t_1.

C. At the given bench height L determine the respective D_e values that define each of the following conditions:

(1) The B' that insures all of the explosive column will react before any cracks will have propagated to any open face when using a single primer located at (a) Floor level, and at (b) The Center of the charge column.

(2) The B" at which overbreak quite likely may begin to occur when the primer is place at floor level.

SOLUTION TO SINGLE BLASTHOLE DESIGN PROBLEM

A(1) From Eq. 21,

$$\sigma_t = \sigma_c \left(\frac{1-\sin\phi}{1+\sin\phi}\right)$$

Then

$$\frac{1750}{25{,}000} = \frac{1-\sin\phi}{1+\sin\phi}$$

or

$$0.07(1+\sin\phi) = 1-\sin\phi$$

$$0.07+0.07\sin\phi = 1-\sin\phi$$

$$1.07\sin\phi = 0.93$$

or

$$\sin\phi = 0.87$$

Thus,

$$\phi = 60 \text{ deg}$$

From Eq. 22(b),

$$\tau_s = \frac{\sigma_c}{2}(\cos\phi)$$

$$= \frac{25{,}000}{2}(0.5)$$

or

$$\tau_s = 6250 \text{ psi}$$

A(2) From Eq. 14(b)

$$V_p = \left[\frac{E_r(1-\mu)}{\rho_r(1+\mu)(1-2\mu)}\right]^{1/2}$$

From Eq. 8(b)

$$\rho_r = 1.941\, SG_r$$

Thus, substituting given values of V_p, μ, and SG_r and squaring both sides of Eq. 14(b),

$$(17{,}000)^2 = \frac{E_r(1-0.25)}{1.941(2.9)(1+0.25)(1-2\cdot 0.25)}$$

Rearranging

$$E_r = \frac{1.7^2\times10^8(1.941)(2.9)(1.25)(0.5)}{0.75}$$

or

$$E_r = 13.5\times10^8 \text{ psf} = 9.4\times10^6 \text{ psi}$$

SOLUTION (CONT.)

B (1) (CONT.)

Therefore, basic velocity equation for the explosive is

$$V_e = \frac{5000\,(D_c - 1)}{0.26 + 0.27\,(D_c - 1)}$$

with the D_c range of values from 1 to 5 inches.

Check: @ $D_c = 3$ in.

$$V_e = \frac{5000\,(3-1)}{0.26 + 0.27\,(3-1)} = \frac{10,000}{0.26 + 0.54}$$

$$= 12,500 \text{ fps} \qquad \underline{OK}$$

@ $D_c = 5$ in.

$$V_e = \frac{5000\,(5-1)}{0.26 + 0.27\,(5-1)} = \frac{20,000}{0.26 + 1.08}$$

$$= 14,900 \text{ fps} \qquad \underline{OK}$$

(a) $D_c = 2$ in.

$$V_e = \frac{5000\,(2-1)}{0.26 + 0.27\,(2-1)} = \frac{5000}{0.26 + 0.27} = 9450 \text{ fps} \Leftarrow$$

(b) $D_c = 4$ in.

$$V_e = \frac{5000\,(4-1)}{0.26 + 0.27\,(4-1)} = \frac{15,000}{0.26 + 0.81} = 14,000 \text{ fps} \Leftarrow$$

B (2) From Eq. 4 (a)

$$P_d = \frac{6.06 \times 10^{-3}\, V_e^2\,(SG_e)}{1 + 0.80\,(SG_e)}$$

From Eq. 1

$$SG_e = \frac{141}{SC} = \frac{141}{117} = 1.2$$

Then

$$P_{d_{max}} = \frac{6.06 \times 10^{-3} \times 1.5^2 \times 10^8 \times 1.2}{1 + 0.80\,(1.2)} = \frac{6.06 \times 2.25 \times 10^5 \times 1.2}{1.96}$$

or $\quad P_{d_{max}} = 835,000$ psi

SOLUTION (CONT.)

<u>5 (1)</u> First determine relationship of V_c with D_e from $y = \frac{cx}{a + bx}$ where $y = V_c$ and $x = D_e - D_c$

Then $$V_0 = \frac{c(D_e - D_c)}{a + b(D_e - D_c)}$$

It is given that $D_c = 1$ in., $V_c = 12{,}500$ fps @ $D_e = 3$ in., and $V_0 = 15{,}000$ fps @ $D_e = 5$ in.

Then @ $D_e = 3$ in., $$12{,}500 = \frac{c(3-1)}{a + b(3-1)} = \frac{2c}{a + 2b}$$

Assume $c = 5000$,

Then $$a + 2b = \frac{2(5000)}{12{,}500} = \frac{4}{5} = 0.80 \quad (I)$$

For $D_e = 5$ in., $$15{,}000 = \frac{c(5-1)}{a + b(5-1)} = \frac{4c}{a + 4b}$$

or $$a + 4b = \frac{4(5000)}{15{,}000} = \frac{4}{3} = 1.33 \quad (II)$$

Regrouping
$$a + 2b = 0.80 \quad (I)$$
$$a + 4b = 1.33 \quad (II)$$

Subtracting I from II,
$$2b = 0.53$$
or $$b \cong 0.27$$

Substituting value of b in I and II,
$$a + 2(0.27) = 0.80 \quad (I)$$
or $$a = 0.26$$
and $$a + 4(0.27) = 1.33 \quad (II)$$
or $$a = 0.25$$

For all practical purposes, then, $a = 0.26$ and $b = 0.27$ when $c = 5000$.

SOLUTION (CONT.)

B (2) (CONT.)

(a) $D_e = 2$ in.

$$P_d = P_{d_{max}}\left(\frac{9450}{15,000}\right)^2 = 835,000\,(0.397)$$

or $$P_d = 331,000 \text{ psi} \Longleftarrow$$

(b) $D_e = 4$ in.

$$P_d = P_{d_{max}}\left(\frac{14,000}{15,000}\right)^2 = 835,000\,(0.87)$$

or $$P_d = 730,000 \text{ psi} \Longleftarrow$$

B (3) From Eq. 4(b),

$$P_e = P_{d_{max}}/2$$

Thus, (a) $D_e = 2$ in., and (b) $D_e = 4$ in.,

$$P_e = 835,000/2 = 417,500 \text{ psi} \Longleftarrow$$

B (4) From Eq. 35,

$$K_B = 30\left(\frac{160}{d_r}\right)^{\frac{1}{3}}\left(\frac{SG_e}{1.3}\right)^{\frac{1}{3}}\left(\frac{V_c}{12,000}\right)^{\frac{2}{3}}$$

From Eq. 7,

$$d_r = 62.4\,(SG_r) = 62.4\,(2.9) = 181 \text{ pcf}$$

Substituting for values of d_r and SG_e, then

$$K_B = 30\left(\frac{160}{181}\right)^{\frac{1}{3}}\left(\frac{1.2}{1.3}\right)^{\frac{1}{3}}\left(\frac{V_c}{12,000}\right)^{\frac{2}{3}}$$

$$= 30\,(0.96)(0.97)\left(\frac{V_c}{12,000}\right)^{\frac{2}{3}}$$

or $$K_B = 28\left(\frac{V_c}{12,000}\right)^{\frac{2}{3}}$$

But from Eq. 34,

$$B = \frac{K_B D_e}{12} = \frac{28}{12}\left(\frac{V_c}{12,000}\right)^{\frac{2}{3}} D_e$$

SOLUTION (CONT.)

B (4) (CONT.)

In general form, therefore, $B = 2.33\,D_c \left(\frac{V_c}{12{,}000}\right)^{\frac{2}{3}}$, ft.

(a) $D_c = 2$ in.

$$B = 2.33(2)\left(\frac{9450}{12{,}000}\right)^{\frac{2}{3}} = 4.66\,(0.86) = 4.0 \text{ ft} \Leftarrow$$

(b) $D_c = 4$ in.

$$B = 2.33\,(4)\left(\frac{14{,}000}{12{,}000}\right)^{\frac{2}{3}} = 9.32\,(1.11) = 10.3 \text{ ft} \Leftarrow$$

Also, for $D_c = 3$ in.

$$B = 2.33\,(3)\left(\frac{12{,}500}{12{,}000}\right)^{\frac{2}{3}} = 7\,(1.02) \simeq 7 \text{ ft}$$

And for. $D_c = 5$ in. and larger,

$$B = 2.33\,D_c\left(\frac{15{,}000}{12{,}000}\right)^{\frac{2}{3}} = 2.33\,D_c\,(1.16) = 2.7\,D_c$$

Thus, at $D_c = 5$ in.,

$$B = 2.7\,(5) = 13.5 \text{ ft.}$$

At $D_c = 6$ in.,

$$B = 2.7\,(6) = 16.2 \text{ ft}$$

B (5) From Eq. 33, $T \simeq 2B/3$

Thus, (a) $D_c = 2$ in.,

$$T \simeq 2(4)/3 = 2.7 \text{ ft} \Leftarrow$$

(b) $D_c = 4$ in.,

$$T \simeq 2(10.3)/3 = 6.9 \text{ ft} \Leftarrow$$

B (6) From Eq. 32, $J \simeq B/3$

Thus, (a) $D_c = 2$ in.,

$$J \simeq 4/3 = 1.33 \text{ ft} \Leftarrow$$

(b) $D_c = 4$ in.,

$$J \simeq 10.3/3 = 3.43 \text{ ft} \Leftarrow$$

SOLUTION (CONT.)

B (7) From Eq. 3 where $d_c = 0.34 D_0^2 (SG_c)$ and combining Eq's. 29 through 33, we obtain when $L = 30$ ft,

$$E = d_c (PC) = 0.34 D_c^2 (SG_c)(L + J - T)$$
$$= 0.34\, D_c^2 (1.2)(30 - B/3) = 0.41\, D_c^2 (30 - B/3), \text{ lb.}$$

(a) $D_c = 2$ in.

$$E = 0.41(2)^2(30 - 1.33) = (1.64)(28.7) = 47 \text{ lb} \Leftarrow$$

(b) $D_c = 4$ in.

$$E = 0.41(4)^2(30 - 3.43) = (6.56)(26.6) = 174 \text{ lb} \Leftarrow$$

B (8) If $A = 1.4B^2$ and $W = \frac{ALd_r}{2000} = \frac{1.4B^2(30)(:\overline{61})}{2000}$,

then $W = 3.8\, B^2$

(a) $D_c = 2$ in., $W = 3.8(4)^2 = 61$ tons $\Leftarrow$

(b) $D_c = 4$ in., $W = 3.8(10.3)^2 = 402$ tons $\Leftarrow$

B (9) From Eq. 19, $V_f = V_p/3 = 17{,}000/3 = 5670$ fps

If $t_f = B/V_f$, sec, (a) $D_c = 2$ in., $t_f = 4/5670 = 0.6$ ms $\Leftarrow$

(b) $D_c = 4$ in., $t_f = 10.3/5670 = 1.5$ ms $\Leftarrow$

B (10) If $t_i = 0.001\, B$, sec.

(a) $D_c = 2$ in., $t_i = 0.001(4) = 4.0$ ms $\Leftarrow$

(b) $D_c = 4$ in., $t_i = 0.001(10.3) = 10.3$ ms $\Leftarrow$

#

SOLUTION (CONT.)

C.(1) From Eq. 36, $K_v = \frac{V_e}{V_p}$. Thus, from part E and determining K_v for the respective V_e values for each D_e from 1 to 6 inches, inclusive, the following summary table can be prepared:

D_e, in.	B, ft.	V_e, fps	K_v
1	0	0	0
2	4	9450	0.56
3	7	12,500	0.74
4	10.3	14,000	0.82
5	13.5	14,900	0.88
6	16.2	15,000	0.88

(a) Floor Priming

From Eq. 38(a), $B' = \frac{3L}{9K_v + 2}$

1. At $D_e = 5$ in.,

$$B' = 3(30)/(9 \times 0.88 + 2) = 90/9.9 = 9.1 \text{ ft.}$$

From above Table, $B = 13.5$ ft.

Thus, $B > B'$ or $13.5 > 9.1$. Diameter can be reduced.

2. At $D_e = 4$ in.,

$$B' = 3(30)/(9 \times 0.82 + 2) = 90/9.4 = 9.6 \text{ ft.}$$

From Table, $B = 10.3$ ft.

Thus, $B > B'$ or $10.3 > 9.6$. Diameter can be reduced.

3. At $D_e = 3$ in.,

$$B' = 3(30)/(9 \times 0.74 + 2) = 90/8.66 = 10.4 \text{ ft.}$$

From Table, $B = 7$ ft.

Thus, $B < B'$ or $7 < 10.4$. Diameter too small.

Note that at $D_e = 4$ in., optimum burden B and the minimum burden B' at which misfire might occur are approximately equal. Therefore, use $D_0 = 4$ in. ⟸

SOLUTION (CONT.)

B(1) (CONT.)

(b) Primer at Center of Charge Column.

From Eq. 38(b), $B' = \frac{3L}{18K_v+1}$

1. At D_e = 5 in.,

$B' = 3(30)/(18 \times 0.88+1) = 90/16.8 = 5.4$ ft.

From Table B = 13.5 ft.

B is much greater than B' indicating diameter can be much smaller, i.e., $B > B'$ or $13.5 > 5.4$.

2. At D_e = 3 in.,

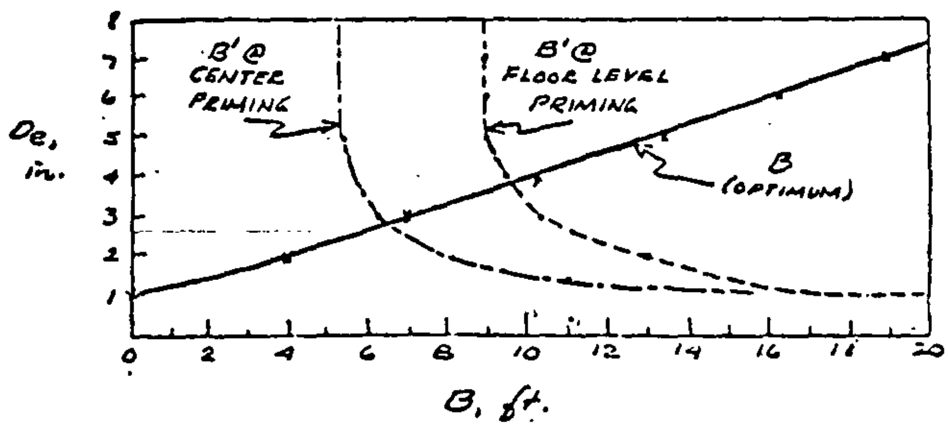
$B' = 3(30)/(18 \times 0.74+1) = 90/14.3 = 6.3$ ft.

From Table B = 7 ft.

Values of B and B' are approximately equal with $B > B'$ a small amount, which is desirable.

Therefore, use D_e = 3 in. ⟸

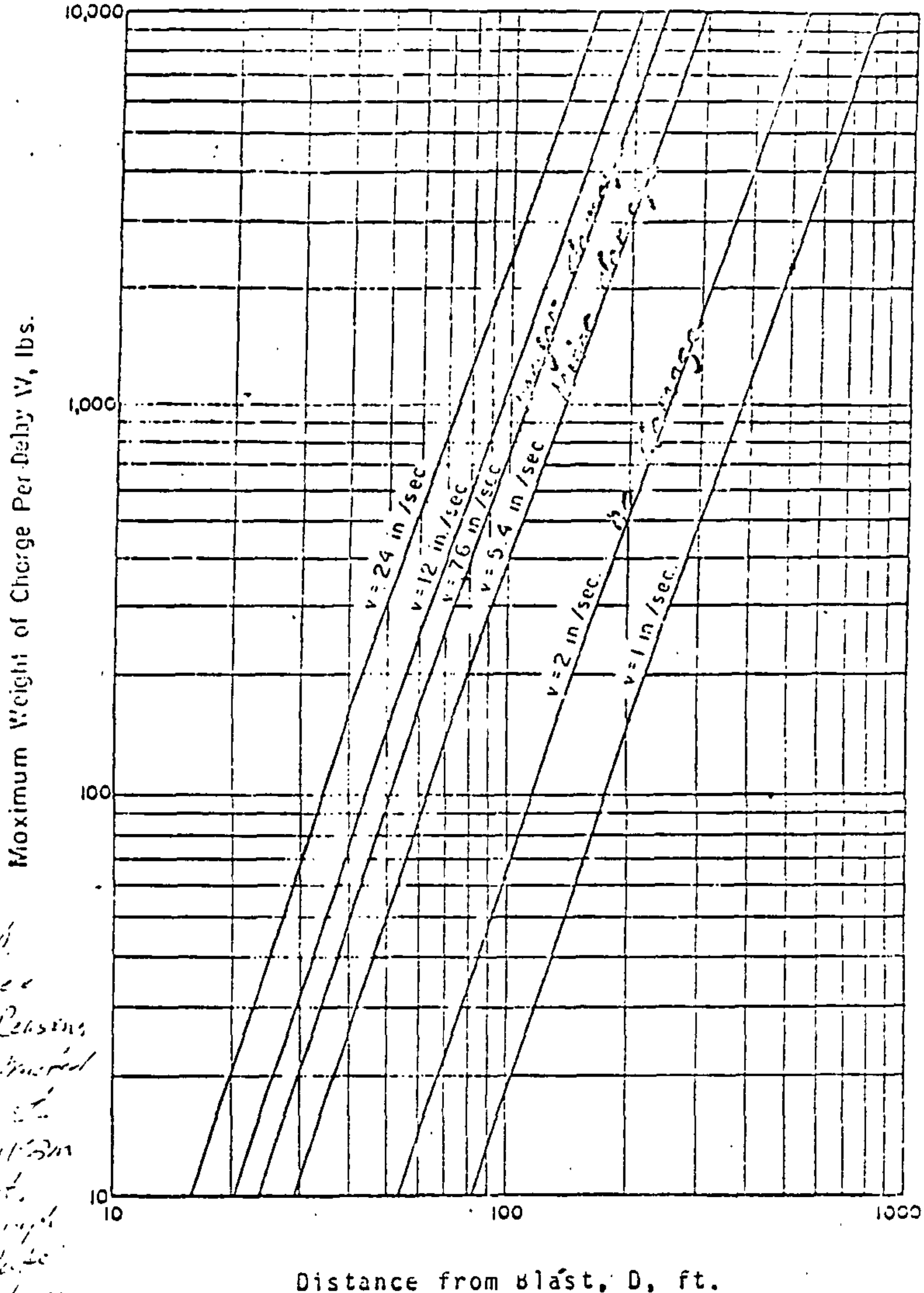
NOTE The previous solutions can be solved quite simply by plotting values as shown below:



C (2)

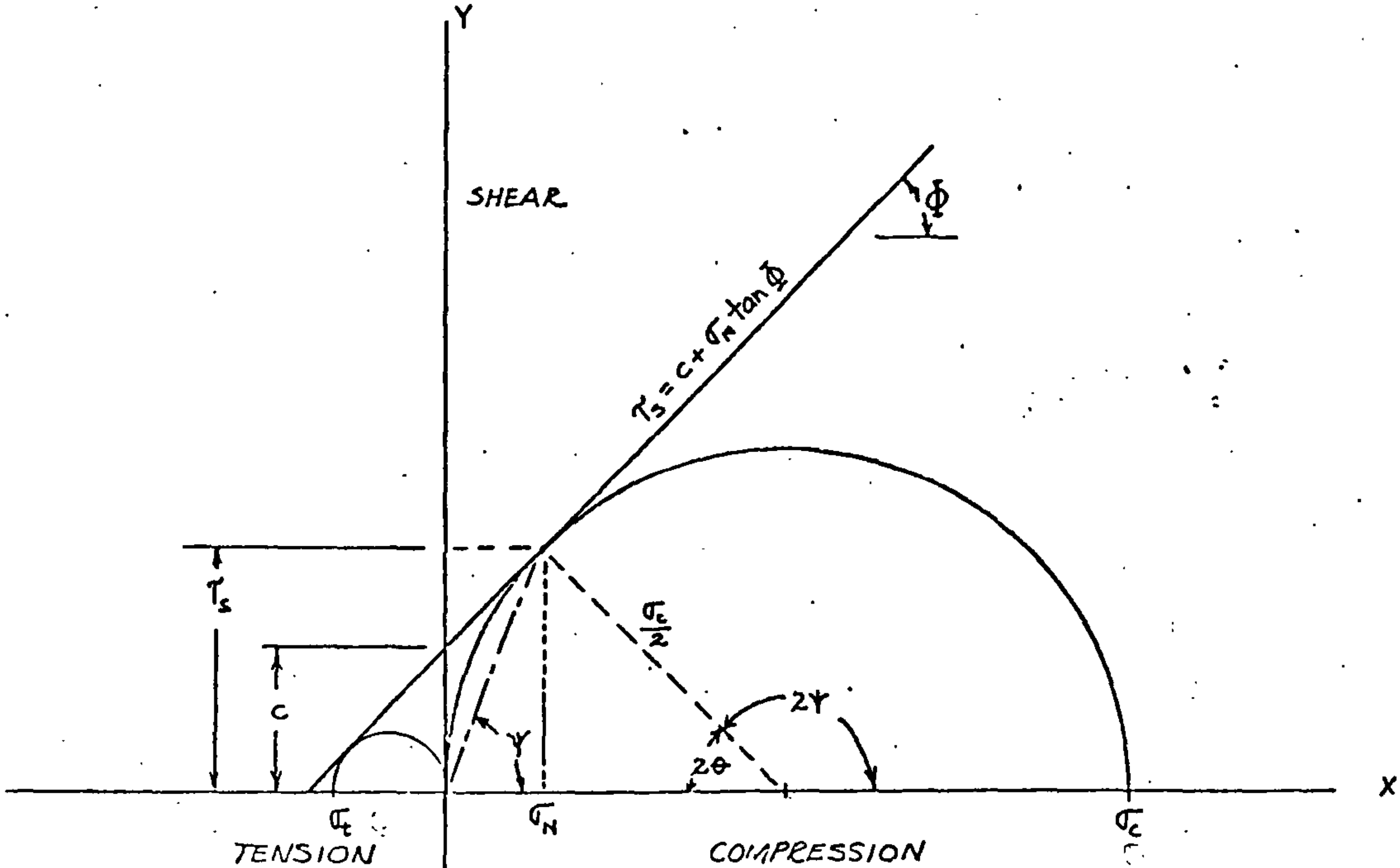
From Eq. 39, $B'' = 0.62L = 0.62(30) = 18.6$ ft ⟸

This burden would be for $D_e = \frac{B''}{2.7} = \frac{18.6}{2.7} = 6.9$ in.



minor damage: fine plaster cracks, opening of old cracks

major damage: serious new cracking, plaster falls



MOHR'S FAILURE ENVELOPE

NOTE: @ $\Phi = 36.8°$, $\tau_s = \sigma_t = \sigma_c / 4$; AND $\Psi = (90° + \Phi) / 2$

2LA

TABLE I. Strength Data For Some Competent Rocks (13).

Rock Type	Compressive Strength psi x 10^3	Elasticity Modulus (Compression) psi x 10^6	Tensile Strength psi	Tens./Comp.	C psi	ϕ deg.	Equation of Mohr's Envelope (ref. Fig. 3)	C/Compress. Strength
Chert	29.3	8.15	820	0.0280	2550	71.5	$Y = 2550 + 3.0\ x$	0.0870
Coal	6.2	–	–	—	1600	38.5	$Y = 1600 + 0.8\ x$	0.258
Granite	28.0	3.17	410	0.0146	1720	76.5	$Y = 1720 + 4.2\ x$	0.0614
Green Stone	29.1	8.82	380	0.0130	1700	77.5	$Y = 1700 + 4.5\ x$	0.0584
Greywacke	7.9	1.80	700	0.0886	1200	59.5	$Y = 1200 + 1.7\ x$	0.1519
Limestone	21.3	9.50	350	0.0164	1320	75.5	$Y = 1320 + 3.9\ x$	0.0620
Marble	30.8	7.15	863	0.0279	2650	71.0	$Y = 2650 + 2.9\ x$	0.0860
Salt Rock	2.2	1.35	85	0.0386	210	72.5	$Y = 210 + 3.2\ x$	0.0954
Sand Stone	14.8	2.00	230	0.0155	900	76.0	$Y = 900 + 4.0\ x$	0.0608
Shale	5.2	1.09	1538	0.2957	1420	31.0	$Y = 1420 + 0.6\ x$	0.2731
Silt-Stone	5.0	12.60	440	0.0880	750	59.5	$Y = 750 + 1.7\ x$	0.1500

C – Cohesion

ϕ – Angle of Internal Friction

Y – Shear Stress, τ_s

x – Normal Stress, σ_n

TABLE II. Strength Data for Some Altered or Fragmented Materials (14, 15).

Rock Type	Cohesion, c, psi	Friction Angle, ϕ, deg.	Equation of Mohr's Envelope (Ref. Fig. 3)
Decomposed Limestone	7.0	18	$Y = 7.0 + 0.30\,x$
Altered Quartzite	5.0	34	$Y = 5.0 + 0.67\,x$
Altered Schist	6.0	40	$Y = 6.0 + 0.84\,x$
Dense Sand and Gravel	7.0	32	$Y = 7.0 + 0.61\,x$
Medium Clay	7.0	20	$Y = 7.0 + 0.36\,x$
Soft Clay	2.8	15	$Y = 2.8 + 0.27\,x$
Liquid Clay	0.7	15	$Y = 0.7 + 0.27\,x$

Y = Shear stress, τ_f

x = Normal stress, σ_n

MATERIAL'S PROPERTIES PROBLEM

A certain ore deposit has been core-drilled and results from laboratory tests on the specimens were as follows:

SG_r for solid ore = 3.0, SG_- for broken ore = 2.4,
σ_c = 24,000 psi, σ_t = 2,000 psi, Porosity = 3 per cent,
ϵ_c = 417 microinches/inch at 3,000 psi compressive load,
ϵ_t = 104 microinches/inch at 3,000 psi compressive load

If one can assume that the material's dynamic and static properties were similar, determine the following constants expressed in the proper units:

(a) μ, (b) E_r, (c) G_r, (d) K_m, (e) K_i, (f) S_f,
(g) v_p, (h) v_s, (i) ϕ, (j) c, (k) τ_s, and (l) γ,

(m) Construct a graph of Mohr's Failure Envelope on the assumption of a straight line relationship.

(n) Based on the assumption the above values all apply to dry rock, estimate the possible effect water saturation might have on the values of the various constants.

SOLUTION TO MATERIAL'S PROPERTIES PROBLEM

(a) By definition, $\mu = \frac{\epsilon_t}{\epsilon_c} = \frac{104}{417} = 0.25$ ⟸

(b) From Eq. 9, $E_r = \frac{\sigma_c}{\epsilon_c} = \frac{3000}{417 \times 10^{-6}} = 7.18 \times 10^6 \text{ psi}$ ⟸

(c) From Eq. 10(a), $G_r = \frac{E_r}{2(1+\mu)} = \frac{7.18 \times 10^6}{2(1+0.25)} = 2.87 \times 10^6 \text{ psi}$ ⟸

(d) From Eq. 11, $K_m = \frac{E_r}{3(1-2\mu)} = \frac{7.18 \times 10^6}{3(1-0.5)} = 4.78 \times 10^6 \text{ psi}$ ⟸

(e) From Eq. 18(b), $K_i = \left[\frac{2(1-\mu)}{1-2\mu}\right]^{\frac{1}{2}} = \left[\frac{2(1-0.25)}{1-0.50}\right]^{\frac{1}{2}} = \left[\frac{1.5}{0.5}\right]^{\frac{1}{2}}$

or $K_i = (3)^{\frac{1}{2}} = 1.732$ ⟸

(f) By definition, $S_f = \frac{SG_r\,(\text{broken})}{SG_r\,(\text{solid})} = \frac{2.4}{3.0} = 0.80$ ⟸

(g) From Eq. 8(b), $\rho = 1.941\,(SG_r) = 1.941\,(3) = 5.823 \text{ lb-sec}^2/\text{ft}^4$

Then, from Eq. 14(b),

$$V_p = \left[\frac{E_r(1-\mu)}{\rho_r(1+\mu)(1-2\mu)}\right]^{\frac{1}{2}}$$

$$= \left[\frac{7.18 \times 10^6 \times 144\,(1-0.25)}{5.82\,(1+0.25)(1-0.5)}\right]^{\frac{1}{2}}$$

or

$V_p = 14{,}600 \text{ fps}$ ⟸

(h) From Eq. 18(a), $V_S = \frac{V_p}{K_i} = \frac{14{,}600}{1.732} = 8430 \text{ fps}$ ⟸

(i) From Eq. 21, $\sigma_t(1+\sin\phi) = \sigma_c(1-\sin\phi)$

$$2000\,(1+\sin\phi) = \cancel{24{,}000}\ 12{,}000\,(1-\sin\phi)$$

$$\sin\phi = \frac{11}{13} = 0.846$$

or $\phi = 58 \text{ deg}$ ⟸

SOLUTION (CONT.)

(j) From Eq. 23, $c = \frac{\sigma_c}{2}\left[\cos\phi - (1-\sin\phi)\tan\phi\right]$

$$= \frac{24{,}000}{2}\left[0.530 - (1-\overset{0.152}{0.848})(1.60)\right]$$

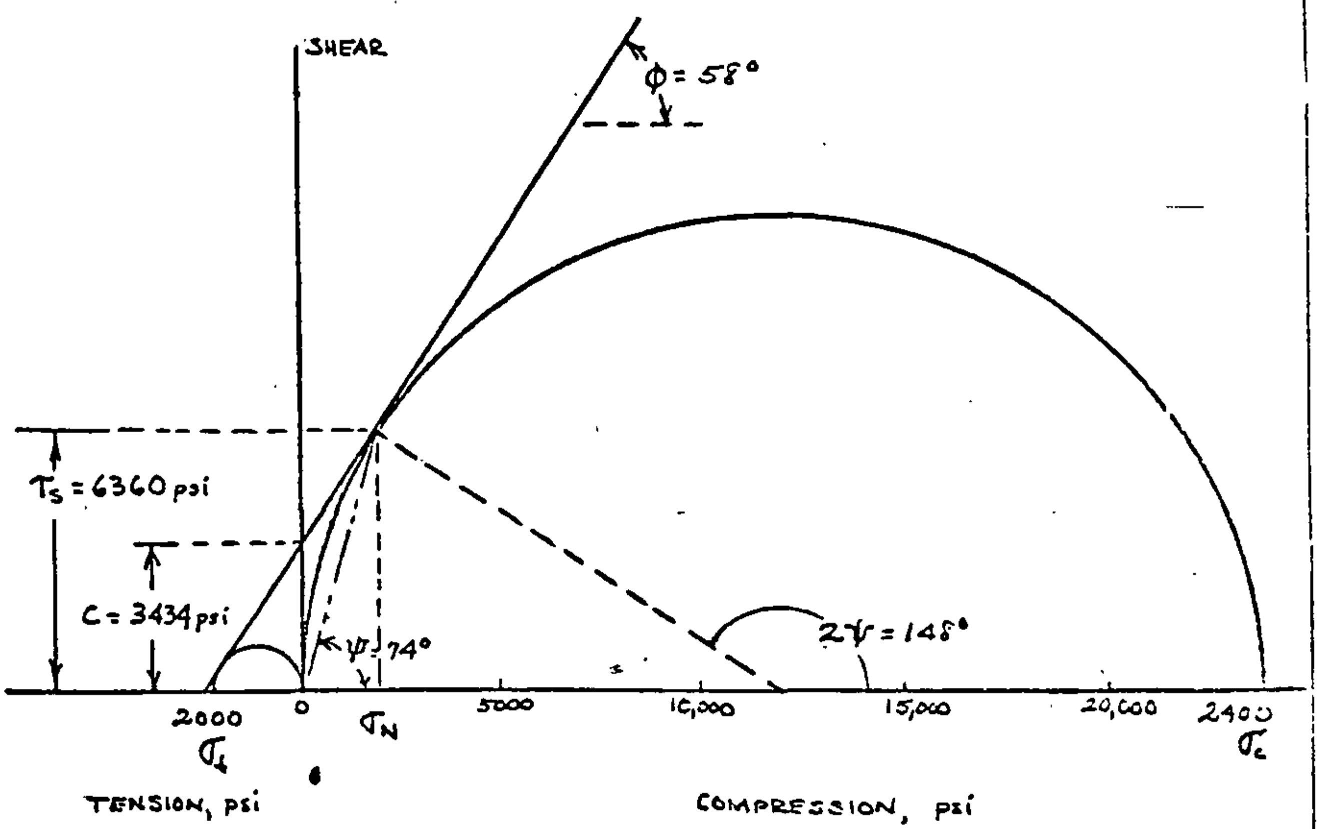
$$= 12{,}000\,(0.530 \overset{0.287}{-} 0.243)$$

or $c = 3434$ psi ⇐

(k) From 22(b), $\tau_s = \frac{\sigma_c}{2}(\cos\phi) = 12{,}000(0.530) = 6360$ psi ⇐

(l) From Eq. 28, $\psi = (\phi + 90°)/2 = \frac{58+90}{2} = 74$ deg ⇐

(m) Mohr's Failure Envelope:



SOLUTION (CONT.)

(n) If the rock were completely saturated, there would be an increase of weight equal to $0.03(3) = 0.09 SG_r$, or new $SG_r = 3.09$. Also, if μ for water is 0.50, then μ for rock should increase slightly because of increased rigidity. Similarly, a slight increase in E_r should be anticipated as an increased pore pressure development when rock is loaded. Because of the latter there is less ability to deform with an accompanying reduction in effective stress, causing the material to have a lower shear strength. From these arguments one can conclude the following changes:

(1) μ increases slightly.
(2) E_r increases slightly
(3) G_r remains nearly constant
(4) K_m increases
(5) K_i increases e.g. if $\mu = 0.27$, $K_i = 1.78$
(6) S_f remains constant
(7) V_p increases
(8) V_s decreases slightly
(9) ϕ decreases
(10) c remains constant
(11) T_s decreases
(12) γ decreases

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TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

THE MECHANICS OF ROCK BREAKAGE

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The Mechanics of
ROCK BREAKAGE

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The Mechanics of

Part I

IN quarrying, the profitability of an operation is directly controlled by the blasting, because it is at the face that the production cycle begins. Poor blast results invariably will lead to economic difficulties. In addition, the frequent changes and complexity of operating conditions force operators to struggle continually with their problems, often without reaching satisfactory solutions. The usual trial-and-error approach as such is expensive and often hazardous, and it rarely leads to complete success because of a lack in flexibility of application. Also, information that is generally available on blasting is not usually applicable from the practical viewpoint.

For these reasons certain basic standards have been developed to assist producers in the design and evaluation of their blasting. It is the purpose of this discussion, therefore, to describe those guidelines and show how they can be applied, in order that normal blasting difficulties might be reasonably avoided.

There are two fundamental effects from blasting that must be controlled: fragmentation and displacement. For the first effect, uniformity of particle-size distribution and the limits of actual sizing are the two important qualities. Usually reasonably uniform sizing is preferred, too many fines or too many slabs being undesirable. Similarly, for the second effect, rock movement, too little or too much displacement is not wanted for economic and safety considerations. The two effects always become problems if overbreak occurs. Air blast and objectionable ground vibration are also problems that can lead to serious difficulties if uncontrolled. Thus, to direct these effects properly and apply the basic standards successfully, one should first have a working knowledge of the blasting process itself.

THE MECHANICS OF ROCK BREAKAGE

Rocks are normally more resistant to failure by compression, or crushing, than they are to being separated by tension. For example, limestones as a group may have compressive strengths of 3,500 to 25,000 psi, but they may have tensile strengths as low as 500 to 2,500 psi. In addition, the ordinary high explosives and blasting agents normally used in blasting produce very high pressures at extremely rapid reaction velocities, which may be from 8,000 to

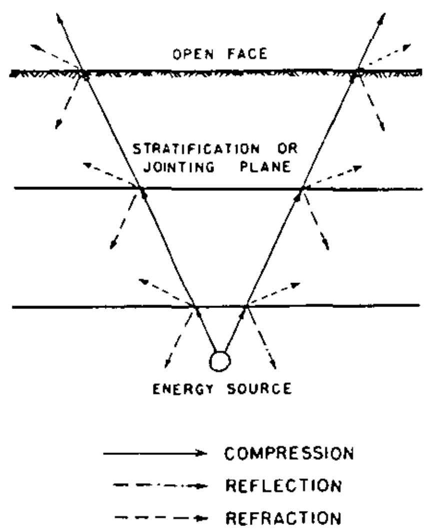

Figure 1—Energy reflection and refraction force components at density interfaces.

26,000 fps (5,300 to 17,000 mph). The rapidly developed pressures in blastholes may be as low as 250,000 psi or in excess of 2,000,000 psi, depending on the particular type of explosive and the conditions under which it is used. The effect of explosives reacting on rocks, then, is one of impact, or impulse, from a quickly applied blow of extremely high intensity.

When explosive charges are used in circular blastholes, the sudden application of high pressures into the surrounding rock is exerted equally in all directions along the blasthole perimeter. The rock in that region is quickly compressed, usually crushing the rock for a limited distance. The sudden application and following quick release of high pressure introduces a compressive stress-wave that quickly spreads throughout the rock mass as an elastic wave. This action results because most rocks are characterized by some brittleness and are therefore somewhat elastic. The particular speed at which the energy travels through the rock is a function of the rock's density, denser materials transmitting compressive-wave energy at high rates and the porous or lighter rocks at relatively low speeds.

For simplicity, one might visualize the wave effect as being similar to that achieved by dropping a pebble into a pond of water. As with the waves in water when they encounter a shoreline, some of the compressive-wave energy from the explosive transmitted through the rock is reflected and refracted (bent) at all changes of density or structural discontinuities (Figure 1). Any open face, change of rock type, etc., will produce this effect. The remainder of the energy, however, tries to continue along its original travel direction. The angle of travel

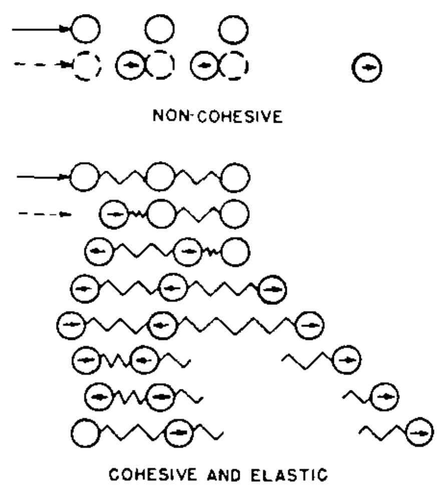

Figure 2—Energy transmission in materials from impulsive loads.

ROCK BREAKAGE

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direction of the reflected energy is the same in value but opposite to the direction of the energy imparted at the boundary, the direction of energy refracted into the next material being a function of the characteristics of both materials. Thus, at every change of density some of the impulsive energy is reflected and refracted, the balance continuing to travel in its initial direction through the second material.

The action of energy transmission is more easily understood if one first considers the material being blasted as being made of many small particles (Figure 2). If a blow is exerted on one particle, we could expect the energy to be transmitted in the direction of the applied blow to adjacent particles, until the energy is eventually consumed as a result of work-performing effects such as friction, dampening, fragmentation, etc. Particles in a pile of sand are noncohesive; so there is little or no attraction between the particles, even though each may have a certain amount of elasticity within itself. Most rocks, however, are cohesive as well as somewhat elastic, thus promoting a different effect from that occurring in loose materials.

For the noncohesive particles, the one on the outside of the pile, on receiving a blow from an adjacent one inside, would endeavor to keep traveling outward, since there are no particles remaining to impede its movement. The cohesive material, on the other hand, would have the outer particles held to adjacent ones, as if by springs. If the blow is sufficiently strong, the inertia of the outer particles will tend to keep them moving outward, once the energy has been applied to them, the springs then being placed in tension. If the tensile strength of the springs is exceeded, they will break. The sudden release of tension will in turn cause the adjacent particles toward the inside of the mass to rebound. As each particle is acted upon in this fashion, beginning at the open face, the springs will be broken in subsequent order back to the source of the initial blow, provided that there is enough energy remaining to exceed the tensile strength of all of the springs.

Thus, the stressing action of breaking rock begins at a free surface, or change in density, and moves back in toward the explosive charge. The problem for proper fragmentation, then, is to be certain there is sufficient applied energy to permit travel outward from the explosive charge and return, with sufficient strength to exceed the tensile strengths of the rocks along the entire path of travel.

Since blastholes are circular, the energy propagation will spread out in distance from the source, or as a fan. This action causes the energy travel in particles to move in different directions. In addition, stresses developed in the walls of blastholes will decrease rapidly as the energy pulses travel away from the charges. There will be only one direction, that perpendicular to a free face and usually called the burden, where energy will be the strongest and first to reach the boundary surface. Energy from the explosive charge will continually weaken and will reach outer particles along the face at later intervals in progressive order.

Fly rock velocity will be greatest at the center point, where the energy travel distance is least; on either side, particles will have less energy imparted to them and will have a progressively greater lateral action as distance is increased from the center. The appearance of the face assumes the shape of a large bubble opposite the charge, with the outermost edge stretched in lateral tension (Figure 3). As a result of this action a crater forms, caused by the combination of tensile effects *along* the energy travel paths from the charge outward *and* those between particles *laterally* because of the diverging action imposed by the differences in energy travel directions.

The outline of the excavation and fracture pattern within the cratered portion are influenced strongly by the structural planes of weakness in the rock mass, such as slips and joints. Whether or not there is enough energy to travel outward *and*

About the Author

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The author of this article is a third-generation mining engineer, who received his formal education in this field at the Pennsylvania State University and his extensive experience in working for Atlas Chemical Industries, Inc. (Explosives Division) and later with firms in the construction, mining, quarrying, and seismic prospecting industries. He served as a naval engineering officer in the Pacific theatre from 1942 to 1946.

Since 1960 he has been a member of the faculty of the University of Missouri School of Mines and Metallurgy. He is also working on research for the department of mining engineering. In addition, he is a consultant on industry problems and is engaged in sponsored research projects and special field assignments. Specialties include excavation techniques, rock mechanics and explosives technology, and blasting problems, with relation to cost and legal aspects.

Mr. Ash is a reserve officer with the United States Civil Engineers Corps and a member of the following: AGI, AIME, ASEE, MSPE, NSPE, SAME, SGE, and SME. He has published articles and presented addresses on equipment performance, explosives, and blasting applications.

return must be determined for each blasting situation. If the amount of initial explosive energy is inadequate for the total travel distance, so that the tensile strengths are not exceeded both outward and on return, one can expect to find the unbroken rock, or very coarse breakage, *inside* the broken rock pile, nearest the location of the blasthole.

Where excess energy is used, the broken rock will be thrown farther out from the face, and there may be some overbreak in back of holes and on the edges. On the other hand, if slabs or boulders are found on the *outside* of the pile after blasting, it is most likely because the ledge was cracked before the blast was made, from earlier overbreak, or because mud seams or similar density changes existed in the rock mass. Cracks, or density changes, serve to reflect and refract energy before it reaches the outer free face, with a subsequent reduction in energy levels passing through, the outside portions therefore being merely pushed out from the face.

For most field blasting, more than one free face will exist, i.e., a bench or ledge is present. The addition of a third free face, such as a corner, will alter the crater effect (Figure 4). Since the relative distances to open faces from a charge determines which face is stressed first, too large a difference in distances often gives humps, toes, or very coarse fragmentation in the area with the longest distance. Full cratering with overbreak will occur on the other side, where energy travel is the least, even though a corner may be present.

In that blastholes are much greater in length than they are in width, the effects from the explosive reaction along the blasthole must also be considered, cylindrical rather than spherical effects being the usual condition. Figure 5 illustrates blastholes in a ledge with pertinent terminology described, while Figure 6 gives wave forms in rock resulting from the cylindrical effect.

It is apparent from the sketches that the time when the compressive-energy wave in rock first arrives at an open face will be different for each blasting situation. The shape of the wave will vary from that of a sphere to a cone, the actual shape of which is a function of the explosive's reaction velocity (v_e) to that of energy travel in the rock (v_r), usually expressed as the K, or velocity ratio.

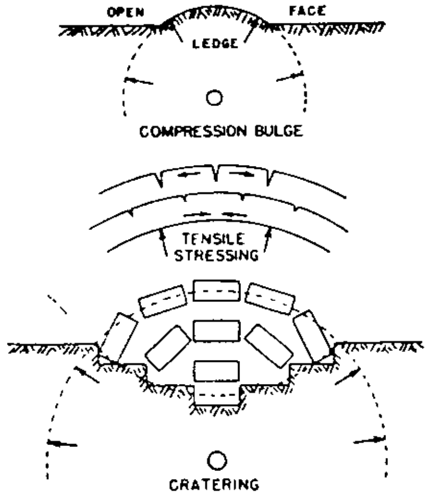


Figure 3—Sequence of actions in crater formation.

The primer location will determine that portion of the ledge which will be stressed and displaced first. As hole depths increase, the difference in blast effects will become greater. Collar priming usually promotes a waterfall effect, with the broken rock left in high piles directly against the vertical face. Bottom priming tends to scatter, or spread out, the broken rock over a larger floor area. Center priming, on the other hand, produces a compromise effect. Collar and bottom priming, when used together in the same blasthole, will tend to increase the stressing in the ledge center, thereby intensifying the fragmentation and displacement actions.

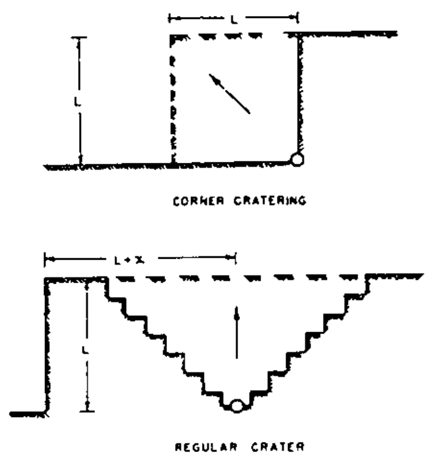
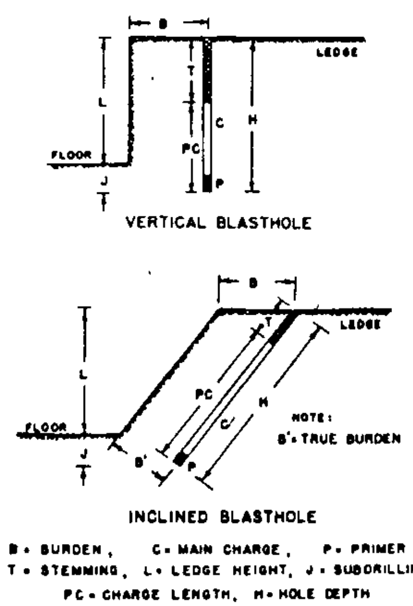


Figure 4—Influence of free-face locations on crater position.

The influence of gravity, or static loading, has little or no practical effect on fragmentation under most blasting conditions. However, for vertically drilled blastholes the higher the ledge, the proportionately greater the resistance to displacement of rock at ledge bottoms. Since the pressure waves produced in the rock from every point along an explosive column cannot reach the vertical and horizontal free faces at the same time, it is most often preferred that stressing begin at the base of the vertical free face. This is usually because of the need for adequate displacement to insure easy and safe digging.

Blastholes that are inclined (Figure 5) help to compensate for weight effects as well as to extend the effective area for stressing in the vicinities of hole collars and bottoms. Boulders most often come from those areas. It has been shown that the greater the angle of inclination the better geometrically proportioned becomes the stemming zone for cratering, thus reducing backbreak effects. But air blast and possible violence are more likely to occur since the volume of rock is appreciably reduced in the stemming region. Thus, less dense explosives would be preferred in collar areas. It should be noted, however, that stressing in portions of the ledge other than at the collar and floor level will be no different, regardless of the hole inclination, provided that the bench face parallels the charge column.

Figure 5—Blasthole terminology.



IT is not enough just to understand what happens during blasting. Probably the most important thing to the average person is to know how blast effects can be controlled to suit the requirements of his operation. In this respect there are available five basic standards upon which to evaluate blasts, all of which are unitless (dimensionless) ratios. They can be applied to both underground and surface blasting with equal success. For simplicity, however, their use will be discussed as applied to surface (open-pit) blasting conditions. The standards are defined as follows:

1. *Burden Ratio* (K_B)—the ratio of the burden distance in feet to the diameter of the explosive in inches, equal to 12 B/D.
2. *Hole-Depth Ratio* (K_H)—the ratio of the hole depth to the burden, both measured in feet, or H/B
3. *Subdrilling Ratio* (K_J)—the ratio of the subdrilling used to that of the burden, both expressed in feet, or J/B
4. *Stemming Ratio* (K_T)—the ratio of the stemming, or collar distance to that of the burden, both being in feet, or T/B
5. *Spacing Ratio* (K_S)—the ratio of the spacing dimension to that of the burden, both measured in feet, or S/B.

The Mechanics of ROCK BREAKAGE

STANDARDS FOR BLASTING DESIGN

Part II of a Series

Burden Ratio The most critical and important dimension in blasting is that of the burden. There are two requirements necessary to define it properly. To cover all conditions, the burden should be considered as the distance from a charge measured perpendicular to the nearest free face *and* in the direction in which displacement will most likely occur. Its actual value will depend on a combination of variables, including the rock characteristics, the explosive used, etc. But when rock is completely fragmented but displaced little or not at all, one can assume the critical value has been approached. Usually, an amount slightly less than the critical value is preferred by most blasters.

There are many formulas that provide approximate burden values, but most require calculations that are bothersome or complex to the average man in the field. Many also require knowledge of various qualities of the rock and explosives, such as tensile strengths and detonation pressures, etc. As a rule, the necessary information is not readily available, nor is it understood.

A convenient guide that can be used for estimating the burden, however, is the K_B ratio. Experience shows that when $K_B=30$, the blaster can usually expect satisfactory results for average field conditions (Table 1). Thus, for a 3-in. diameter explosive, a 7½-ft. burden (30 x 3/12) would be a reasonable approximation. To provide greater throw, the K_B value could be reduced below 30, and subsequent finer sizing is also expected to result.

Light density explosives, such as field-mixed AN/FO mixtures, necessarily require the use of lower K_B ratios (20 to 25), while dense explosives, such as the slurries and gelatins, permit the use of a K_B near 40. The final value selected should be the result of adjustments made to suit not only the rock and explosive types and densities but also the degree of fragmentation and displacement desired.

To estimate the desired K_B value, one should know that densities for explosives are rarely greater than 1.6 or less than 0.8 gm/cc. Also, for most rocks requiring blasting, the density in gm/cc rarely exceeds 3.2, nor is it less than 2.2, with 2.7 (165 lb. per cu. ft. in the solid) by far the most common value. Thus, by first approximating the burden make simple estimations toward 20 at a K_B of 30, the blaster can then

Table 1—Standard Blasting Ratios for Vertical Blastholes (All Types of Surface Blasting, 20 Different Rock Types, Hole Depths From 5 to 260 ft., and Hole Diameters From 1⅝ to 10⅝ in. for All Grades of Explosives)

All Operations				All Operations but Coal Strippings			
K_B Group	Frequency	K_H Group	Frequency	K_J* Group	Frequency	K_T* Group	Frequency
						0.10-0.19	0
		0.0-0.9	0			0.20-0.29	6
10-13	0	1.0-1.9	43			0.30-0.39	12
14-17	5	2.0-2.9	70	0.00-0.09	15	0.40-0.49	18
18-21	13	3.0-3.9	56	0.10-0.19	18	0.50-0.59	18
22-25	51	4.0-4.9	45	0.20-0.29	27	0.60-0.69	25
26-29	74	5.0-5.9	22	0.30-0.39	26	0.70-0.79	19
30-33	66	6.0-6.9	22	0.40-0.49	25	0.80-0.89	13
34-37	44	7.0-7.9	11	0.50-0.59	2	0.90-0.99	6
38-41	20	8.0-8.9	4	0.60-0.69	6	1.00-1.09	14
42-45	7	9.0-9.9	2	0.70-0.79	2	1.10-1.19	7
46-49	4	10.0-10.9	8	0.80-0.89	0	1.20-1.29	7
50-53	0	11.0-11.9	0			1.30-1.39	3
		12.0-12.9	1			1.40-1.49	2
						1.50-1.59	2
Total	284	Total	284	Total	125	Total	152
Mean	30	Mean	4.0	Mean	0.28	Mean	0.74
Mode	38	Mode	2.6	Mode	0.24	Mode	0.65
Median	29	Median	3.4	Median	0.27	Median	0.67

*Note—Rf: Ash, R. L., and Pearse, T. E.—"Velocity Hole Depth Related to Blasting Results," *Mining Engineering*—September, 1962, p. 75.

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(or 40) to suit the rock and explosive characteristics, densities for the latter exerting the greater influence.

Thus, for light explosives in dense rock, use $K_B=20$; for heavy explosives in light rock, use $K_B=40$; for light explosives in average rock, $K_B=25$; for heavy explosives in average rock, $K_B=35$, etc. Figure 7 illustrates the relationships between burdens and explosive diameters and can be used to approximate values for quick estimations. It should be noted, however, that the burden must be more carefully selected for small-diameter blastholes than for the larger charges, a fact well confirmed by field experience.

Hole-Depth Ratio As a rule, a blasthole should never be drilled to a depth *less* than the burden dimension, if overbreak and cratering are to be avoided. The primer location and the K_v ratio (Figure 6) have an important influence on the minimum required depth, in that the shape and direction of the wave form determines where and which face is stressed first. In practice, blastholes are generally drilled from 1½ to 4 times the burden dimension; and blasting is done most frequently with a K_H value of 2.6 (Table 1).

One could then presume that when using a 3-in. explosive of average density in normal rock with a 7½-ft. B, a hole depth from 10 to 30 ft. would normally give satisfactory results. As the depth increased beyond 30 ft., displacement problems could result, leaving toes or bootlegs (part of the hole left intact) because of the failure to pull the full ledge height. Inclined drilling will help to eliminate some of the difficulty. But a hole depth less than the burden, 8 ft., for example, could always be expected to be violent and to produce overbreak in back of holes.

Subdrilling Ratio The primary reason for drilling blastholes below floor level (or grade) is to insure that a full face will be removed. Uneven floors caused by humps and toes generally create problems for later blasting, as well as in loading and haulage operations. For most conditions, the required subdrilling (J) should never be less than 0.2 the burden dimension, a K_J of at least 0.3 being preferred for quite massive ledges (Table 1).

The amount of necessary overdrilling logically depends upon the structural and density characteristics of the ledge, but also on the direction of the blastholes, in that inclined holes require less subdrilling and horizontal holes no subdrilling whatsoever. Under certain conditions no subdrilling is required also for vertical holes, as would be the case for many coal strippings or rock quarries having a pronounced parting at floor level. However, for relatively massive rock drilling, at least

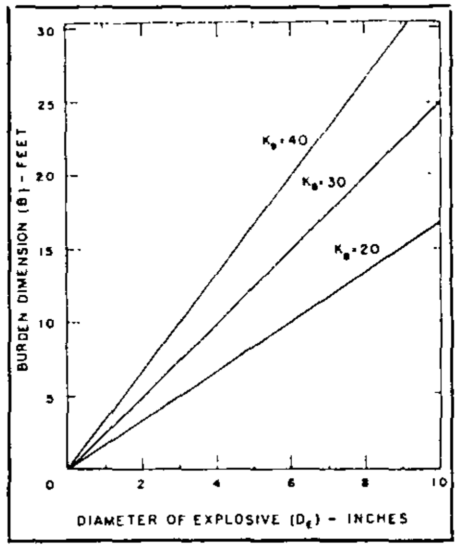


Figure 7—Relationships between burden dimension and explosive diameter.

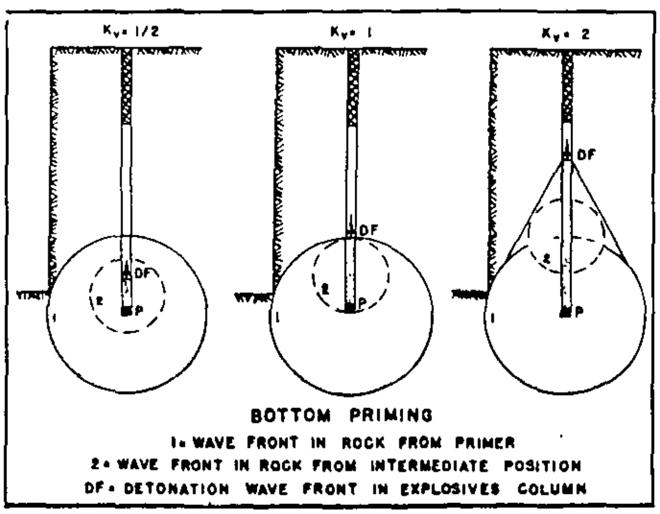
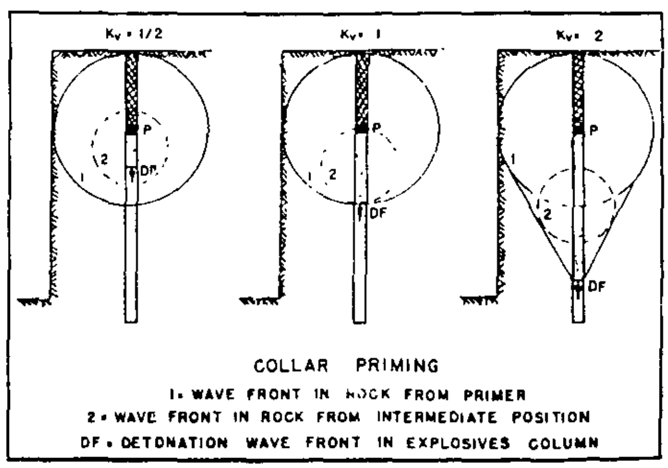


Figure 6—Compressive stress wave-forms in massive uniform rock.



0.3 the burden below the floor will insure that full ledge heights are obtained, provided, of course, that a proper K_H value is also used. Thus, for the 3-in explosive and 7½-ft burden, the blasthole should be drilled at least 2½ ft. below floor level.

Stemming Ratio Collar and stemming are sometimes used to express the same thing. However, stemming refers to the filling of blastholes in the collar region with materials such as drill cuttings to confine the explosive gases. But stemming and amount of collar, the latter being the unloaded portion of a blasthole, perform other functions in addition to confining gases. Since an energy wave will travel much faster in solid rock than in the less dense unconsolidated stemming material, stressing will occur much earlier in the solid material than compaction of the stemming material could be accomplished. Thus, the amount of collar that is left (T), whether or not stemming is used, determines the degree of stress balance in that region. The use of stemming material then assists in confining the gases by a delayed action that should be long enough in time duration to permit their performing the necessary work before rock movement and stemming ejection can occur. For stress balance in bench-blasting of massive material, the value of T should equal the B dimension (Figures 5 and 6).

Usually a K_T value of less than 1 in solid rock will cause some cratering, with back break and possible violence, particularly for collar priming of charges. However, if there are structural discontinuities in the collar region, reflection and refraction of the energy waves reduce the effects in the direction of the charge length. Thus, the K_T value can be reduced under such circumstances, the amount depending upon the degree of energy reduction at the density or structural interfaces. Field experience shows that a K_T value of 0.7 is a reasonable approximation for the control of air blast and stress balance in the collar region (Table 1). Thus, for the 3-in explosive using a 7½-ft. burden, 5 to 6 ft of collar with suitable stemming is generally satisfactory.

Spacing Ratio Commercial blasting usually requires the use of multiple blastholes, making it necessary for blasters to know whether or not there are any mutual effects between charges. If adjacent charges are initiated separately (in sequence), with a time-delay interval of sufficient length to permit each charge to complete its entire blasting action, there will be no interaction between their energy waves (Figure 8).

However, if the time interval for initiating adjacent charges is reduced complex effects will result. There might be reinforcement or cancellation of forces, depending upon the force magnitudes and directions at their point of interference. For charges initiated simultaneously, or at extremely short-delay intervals, the reinforcement action increases with larger angles of force collision. This action promotes greater ground vibration force-effects. However, as described earlier, the energy levels of stresses in the rock are reduced by the fan effect as distance from the source of energy increases. The mutual reinforcement action then tends partially to minimize the energy reduction because of fan effect reductions, thus permitting greater spacings to be used between blastholes initiated simultaneously than when delayed.

The manner in which the zone of rock between holes is broken depends then not only on the particular initiation-timing system used but also on the spacing dimension. Ideal energy balancing between charges is usually accomplished when the spacing dimension is nearly equal to double that of the burden (K_S=2) when charges are initiated simultaneously. For long-interval delays, the spacing should approximate the burden, or K_S=1. For short-period delays, the K_S value will vary from 1 to 2, depending upon the interval used. However, since structural planes of weakness such as joining, etc., are not actually perpendicular to one another, the exact value for K_S normally will vary from 1.2 to 1.8, the preferred value of which must be tailored to local conditions.

Most difficulties resulting from blasting can be attributed to the use of an unsuitable K_S relationship.

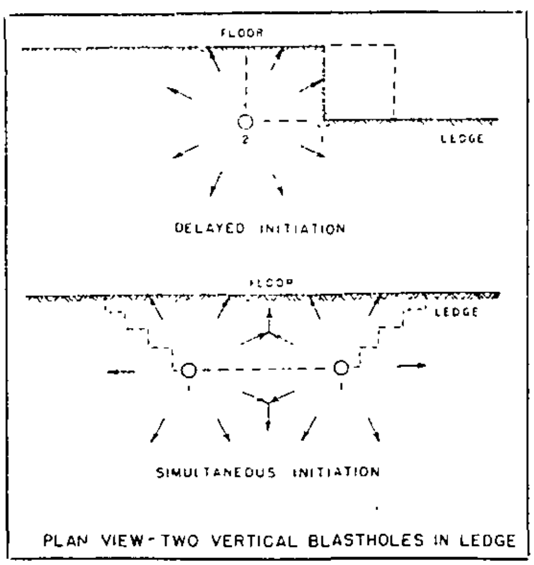
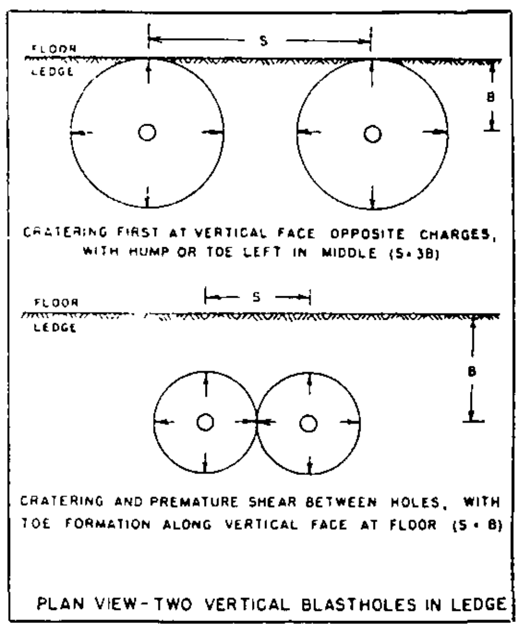


Figure 8 — Compressive-pulse force interactions for multiple-charge initiation (balanced K_S ratios).

Figure 9 — Compressive-pulse wave-positions at instant cratering begins. (Simultaneous initiation of multiple charges with unbalanced K_S ratios)



For example, from Figure 9, illustrating compressive-pulse wave positions, one can see that when fracturing begins for simultaneous initiation, extended spacings (K_S greater than 2) always lead to horizontal cratering. The action always leaves humps at floor level between the blastholes. Too close a spacing, on the other hand, causes premature shearing between holes. This condition produces finely broken rock between holes, providing all the explosive reacts, but with boulders or slabs formed in the burden zone.

Premature shear and related loss of confinement further promotes volume changes, with subsequent pressure drops in the blasthole region, which for the relatively insensitive blasting agents may kill the reaction completely and result in a misfire. The action also usually loosens stemming too early and permits the release of gases out through the collar regions. Unless deliberate shearing is desired, as for pre-splitting when charge loads should be reduced and fairly sensitive explosives are used, normal blasts exhibit vertical cratering, overbreak, violent fly rock, nonuniform breakage, and toes at floor level.

It can be generally assumed that uniformity of sizing is a direct result of the K_S ratio. If on firing a single hole the rock is satisfactorily broken and cleanly removed without excessive displacement, it may be assumed the burden is satisfactory. Too often blasters reduce the burden rather than extend the spacing in their desire to eliminate boulders or to make rock sizing more uniform.

The basic principles for spacing selection apply to all multiple-charge blasts, as long as all holes are drilled parallel and in the same direction relative to one another. Figure 10 illustrates the basic drill patterns for most field conditions and may be summarized as follows: (1) for sequence delays in the same row, the K_S should be near 1; (2) for simultaneous initiation of holes in the same row, the preferred K_S is near 2; (3) for sequence timing in the same row and simultaneous initiation *laterally* between holes in adjacent rows, the entire blast should be drilled in a square arrangement in order to avoid stress unbalance; and (4) staggered drill patterns are preferred between rows within which all charges are initiated simultaneously.

It should be noted that the actual (or true) burden may be different from that normally considered for each separate blasting condition, if we take into account the fact it should be measured *in the direction* in which displacement occurs. For example, in Figure 5 the true burden for an inclined hole is not actually the horizontal distance, since stressing from wave travel will occur earliest at a point on a line perpendicular to the free face (B′). Thus, the normally considered horizontal burden can be extended by inclining the blasthole even though the true burden would be the same as that discussed previously (K_S=20 to 40).

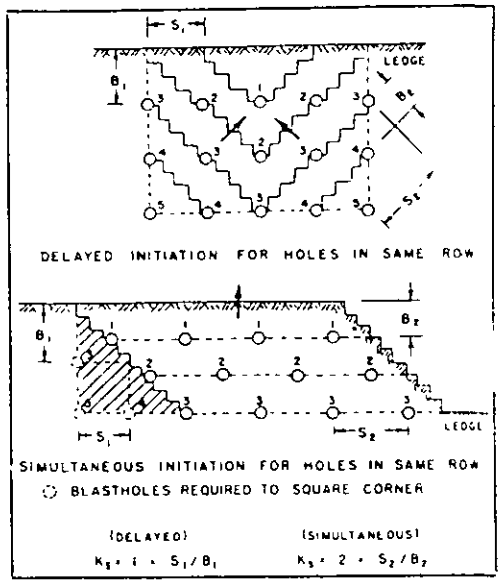


Figure 10—Basic drill-pattern relationships (Ideal blasting conditions.)

From Figure 10 one can see that the preferred K_S never changes, regardless of conditions, with a K_S near 1 for sequence and near 2 for simultaneous initiation patterns. Because movement is about 45 degrees with the open face for sequence timing, when holes in adjacent rows measured laterally are initiated at the same time, their true actual burden must be considered as measured laterally since movement is perpendicular to that direction. Thus, for different drill patterns but using the same K_B value, the actual area (or volume) of rock blasted should not change.

This can be explained by the example of the 7½-ft. burden for a 3-in. explosive, where a 10 x 10-ft. square pattern is desirable for sequence timing in the same row; but a 7½ x 13-ft. staggered pattern would work equally well when all holes in the same row are fired together. A typical 8 x 12-ft. pattern often followed in the field is merely a compromise between the two more desirable arrangements. However, the pattern invariably gives nonuniform breakage, particularly in massive rock, no matter what timing system is used because of stress unbalance, and resulting overbreak in corners.

Under certain conditions the K ratio controls displacement to an advantage. If the timing system is properly selected to give a desired blast effect, slight adjustments can

Table 2—Normal Drill-Pattern Dimensions for Average Blasting Conditions (All Values in Feet Except for Explosive Diameter)

D_E: Inches	B	J	T	L (Max.)*	Equivalent Patterns: Staggered (Simultaneous Timing)	Equivalent Patterns: Square (Sequence Timing)
1	2½	1	2	10	2½ x 4	3 x 3
2	5	2	4	20	5 x 9	7 x 7
3	7½	2½	5	30	7½ x 13	10 x 10
4	10	3	6	40	10 x 18	13 x 14
5	12½	4	8	50	12½ x 22	16 x 16
6	15	5	10	60	15 x 27	20 x 20
7	17½	5½	12	70	17½ x 31	23 x 23
8	20	6	14	80	20 x 36	26 x 27
9	22	7	15	88	22 x 40	29 x 30
10	24	7½	16	96	24 x 43	32 x 32
11	26½	8	18	106	26½ x 48	35 x 36
12	29	9	20	116	29 x 52	38 x 39

*Note—Minimum L=B

be made to the K_S ratio so as to place the broken rock in an other-than-normal position, but with some sacrifice in uniformity of rock sizing. For example, for a K_S of 0.7 to 0.9 (whereby the spacing actually becomes the burden) the use of sequence timing causes broken rock to move parallel or along the ledge face and not out onto the floor, as is the effect often desired in coal stripping. On the other hand, a K_S of 1.2 to 1.4 for delayed charges moves the broken rock farther away from the ledge.

SUMMARY

Most blasting difficulties occur because of a lack in understanding of how rock is broken and the use of improper charge-placement and initiation-timing practices. The clues as to what could be wrong are often revealed by observing how a blast performs: whether or not nonuniform breakage results, toes are left, overbreak and violence occur, and similar undesirable effects exist. Provided that the proper explosives are employed for the operating conditions, certain standards can be applied, to help in the evaluation of blasts. These standards can also assist in providing guidelines as to which direction adjustments should be made for correcting any difficulties. The standards are practical and simple to apply, being based on two fundamental, usually known qualities, explosive diameters and ledge height. The standards are as follows:

$$K_B = 20 \text{ to } 40 \text{ (30 avg.)}$$
$$K_H = 1\tfrac{1}{2} \text{ to } 4 \text{ (2.6 avg.)}$$
$$K_J = 0.3 \text{ minimum}$$
$$K_T = 0.5 \text{ to } 1 \text{ (0.7 avg.)}$$
$$K_S = 1 \text{ to } 2$$

As a rule, the K_B relationship is the first standard to apply, since it provides the burden dimension. An exception to this is for blasting extremely low or very high ledges. In such cases the ratio must be adjusted to suit the ledge height. For normal conditions and using a 2-in. explosive, for example, the burden will average near 5 ft. for hole depths not less than 7½ nor more than 20 ft., with subdrilling of at least 1½ ft. and stemming near 3½ ft. The ledge height (L) could then be from 5 to 6 ft. up to about 18½ ft. Table 2 lists data for normal operating conditions. However, the spacing value for adjacent charges will depend entirely on the timing system used and on the rock structural features; but it will vary from 5 to 10 ft. for the example given.

For ledge heights less than the minimum, smaller-diameter explosives should be used; otherwise, overloading and possible violence will occur. For very high faces, the burdens must be reduced or the explosive diameters increased. The latter can be accomplished by drilling larger vertical holes, springing or enlarging holes at their bottoms, using additional snake, or horizontally drilled, holes in the toe region, inclining the drill holes, etc.

An additional problem often present in blasting is that of cap rock, or hard massive layers at the top of a ledge. Using less than normal stemming does nothing but promote violence, since this solution only aggravates vertical cratering, with subsequent overbreak. Instead, an additional short hole should be drilled in the block center, with part of the normal explosive charge for the deeper holes divided equally between a small deck charge, loaded near the collar of the deep hole but separated from the main charge by stemming, and a small charge placed in the extra short hole. In this manner the ledge height limitations are satisfied, with the cap rock and remainder of the ledge then being considered as two separate benches, even though they are blasted at the same time.

The standards will be found to be quite convenient and useful, after very little practice, not only for the initial design of blasts but also in providing guidelines upon which to correct normal blasting difficulties which invariably occur from time to time. However, one must realize that the standards in themselves are not cure-alls, since blasting as such depends heavily on cost and safety considerations as well as on the explosive grades used, the material's characteristics, and blasting techniques employed.

The Mechanics of

CHARACTERISTICS OF EXPLOSIVES

Part III of a Series

IN selecting an explosive upon which to base a particular set of blasting standards, the choice will depend largely on the cost and properties of the explosive and its adaptability to the materials to be blasted. Since blasting effectiveness from any explosive is controlled by its chemical composition and the effects produced by the field conditions under which it is used, the user should have a working knowledge of the various explosives products available and their particular properties. In this manner he is then better able to make a practical choice to suit his own operating conditions.

An explosive can be considered simply as a tool for performing work, designed to accomplish a specific job. The work performed is made possible by the gas pressures produced when the explosive reacts. The ideal explosive would be one in which only gases are formed from the original ingredients. However, if some solids are also produced by the reaction, the gas pressures would be correspondingly reduced, with the explosive then being capable of producing less work. Since there are many different field conditions with which to contend, manufacturers offer many different types and grades, many of which are nonideal and designed to have their own qualities that make them differ from one another. Part of the differences are chemical, part are physical. However, since explosives are chemical compounds, it is from their original composition that all basic qualities are first determined.

Ingredients and Composition Most commercial explosives are mixtures of compounds containing four basic elements: carbon, hydrogen, nitrogen, and oxygen. Other compounds with additional elements such as sodium, aluminum, calcium, etc., may also be included to produce certain desired effects. As a rule, manufacturers design their products to be nearly oxygen-balanced. This means that there is the correct amount of oxygen available in the mixture so that during the reaction all of the hydrogen reacts to form only steam (H_2O), the carbon reacts to form only carbon dioxide gas (CO_2), and the nitrogen released forms only free nitrogen gas (N_2).

If there are other than the basic four elements, e.g., sodium, solids would be expected to be produced, and for these there must be included sufficient additional oxygen to combine with them. When there is an excess of available oxygen, however, certain other compounds are produced, among which are the highly poisonous nitrous-oxide fumes (NO/NO_2). These particular fumes are easily detectable by their obnoxious odor and red-brown color. On the other hand, if there is an oxygen shortage, the deadly carbon-monoxide fume (CO) will be formed, as well as certain other compounds, depending on the ingredients. Unfortunately, carbon monoxide cannot be detected by odor or sight. In addition to the formation of poisonous fumes, an excess or deficiency of oxygen will yield a lower heat of explosion, with a subsequent reduction in pressures produced.

It should therefore be recognized that if one is to expect safe and efficient results from explosives, there should be a suitable initial chemical balance, with thorough mixing of ingredients to ensure that all materials are in intimate contact, maintenance of the desired mixture while in storage, and then proper use on the job. The following chemical equations may help to illustrate the effects from oxygen balancing, using an AN-FO blasting agent for an example:

(1) *Balanced for oxygen:*

$$3NH_4NO_3 + CH_2 \rightarrow 7H_2O + CO_2 + 3N_2$$

(2) *Excess oxygen:*

$$5NH_4NO_3 + CH_2 \rightarrow 11H_2O + CO_2 + 4N_2 + 2NO$$

(3) *Deficient oxygen:*

$$2NH_4NO_3 + CH_2 \rightarrow 5H_2O + 2N_2 + \underline{CO}$$

It is not necessary for an explosive to contain nitroglycerin (NG), nitrostarch (NS), TNT, and similar explosive compounds. The individ-

Table 3—Some Ingredients of Explosives

Name	Chemical Symbol	Function
Nitroglycerin (NG)	$C_3H_5(NO_3)_3$	Explosive base
Trinitrotoluene (TNT)	$C_6H_2CH_3(NO_2)_3$	Explosive base
Dinitrotoluene (DNT)	$C_7N_2O_4H_6$	Explosive base
Ethylene glycol dinitrate (EGDN)	$C_2H_4(NO_3)_2$	Explosive base, antifreeze
Nitrocellulose	$C_6H_7(NO_3)_3O_2$	Explosive base, gelatinizing agent
Ammonium nitrate (AN)	NH_4NO_3	Explosive base and oxygen carrier
Potassium chlorate	$KClO_3$	Explosive base, oxygen carrier
Potassium perchlorate	$KClO_4$	Explosive base, oxygen carrier
Sodium nitrate (SN)	$NaNO_3$	Oxygen carrier, reduce freezing point
Potassium nitrate	KNO_3	Oxygen carrier
Wood pulp	$C_6H_{10}O_5$	Absorbent, combustible
Fuel oil	CH_2	Fuel
Paraffin	CH_2	Fuel
Lampblack	C	Fuel
Chalk	$CaCO_3$	Antiacid
Zinc oxide	ZnO	Antiacid
Aluminum metal	Al	Catalyzer
Magnesium metal	Mg	Catalyzer
Kieselguhr	SiO_2	Absorbent, anti-caking material
Liquid oxygen	O_2	Oxygen carrier
Sulphur	S	Fuel
Salt	NaCl	Flame depressant
Organic nitro compounds		Explosive base, but used primarily to sensitize, reduce freezing point, and as anti-caking material

ROCK BREAKAGE

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ual characteristics of each ingredient determine whether it may be desirable for use in a mixture. Table 3 gives a partial listing of the many ingredients that might be included in an explosive. It can be recognized that certain compounds may be highly explosive by themselves or may be normally inert; but when combined, the entire mix may form an explosive. For this reason the compounding of explosives should not be attempted by the average person.

Explosive Reactions To be an explosive, the change in form from liquid or solid, or a combination of both, to that of a gas, or gas and solid, must be an exothermic reaction, or one from which heat is released. For most explosives, the quantity of heat released is quite large (Table 4). The gases formed, in turn, quickly produce very high pressures, with the reaction being called either deflagration or detonation.

The distinction between the two types of reaction is that deflagration consists of a burning action at a high rate of speed, the chemical reaction of which causes gaseous formation and pressure expansion *along* with the burning. Thus, a heaving action from the pressures produced is experienced at nearly the same rate as that of the burning. This type of reaction is characteristic of low explosives, of which black powder is one particular type.

Detonation, on the other hand, consists of the propagation of a shock wave through the explosive, accompanied by a chemical reaction that furnishes energy to sustain the shock-wave propagation in a stable manner, with gaseous formation following shortly thereafter. The shock wave is characterized by a very sharp rise in pressure (Figure 11), in front of which there is a zone in which all immediate matter is ionized. The pressures developed by detonation (shock) are nearly double those produced by the gaseous expansion that follows. All high explosives are designed to detonate,

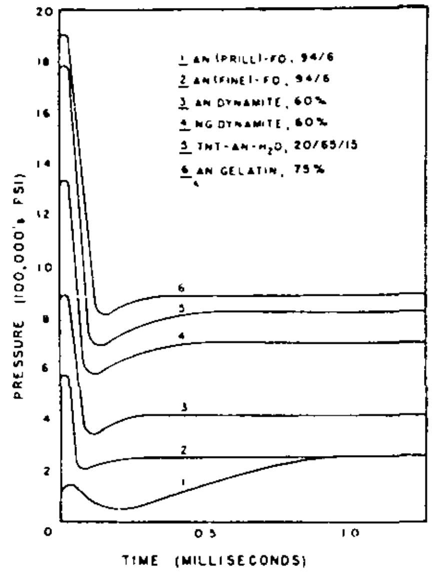

Figure 11—Curves of calculated pressure developed by some selected explosives under perfect confinement.

all low explosives will deflagrate; and blasting agents may exhibit one or the other type of reaction, according to their specifications and conditions of use. The important thing to remember about the reactions is that the effects of one type are very much different from those of the other, detonation producing higher energy and much higher velocities.

To accomplish a desired reaction, certain temperature and pressure conditions must be met, most explosives being designed for use under confinement, e.g., in blastholes. If the temperature required for a proper reaction is not present, no detonation may occur, with only burning or possible deflagration resulting. In practical terms, this means that even though the designed chemical composition calls for detonation, inadequate initial heat from an initiator or primer or a loss in confinement conditions can result in lower blast energy being developed from the explosive charge, or even in complete failure, causing a misfire.

For this reason, control over the confinement and the selection of primers with adequate heat energy and initiating power are particularly important. One should recognize then, which of the explosives need strong priming and which need very little heat for initiating their reactions, not only for reasons of blasting efficiency but for safety considerations as well. *(Turn page)*

Table 4—Available Heat Energies (Q) for Certain Selected Explosives

Explosive	SG	SC	Q (Cal/gm)
Nitroglycerin (NG)	1.6	88	1420
PETN	1.6	88	1400
RDX	1.6	88	1320
Composition B	1.6	88	1140
Tetryl	1.6	88	1010
NG gelatin 40%	1.5	94	820
Slurry (TNT-AN-H_2O, 20/65/15)	1.5	94	770
NG gelatin 100%	1.4	101	1400
NG gelatin 75%	1.4	101	1150
AN gelatin 75%	1.4	101	990
NG dynamite 40%	1.4	101	930
AN gelatin 40%	1.4	101	800
NG dynamite 60%	1.3	109	990
PETN	1.2	118	1200
Semigelatin	1.2	118	940
Extra dynamite 60%	1.2	118	880
Amatol, 50/50	1.1	128	890
RDX	1.0	141	1280
DNT	1.0	141	960
TNT-AN, 50/50	1.0	141	900
TNT	1.0	141	870
AN-FO, 94/6	0.9	157	890
AN low-density dynamite	0.8	176	880
AN	0.8	176	350

To better understand the requirements just described, Table 5 illustrates the approximate temperature characteristics of two basic ingredients used in many commercial explosives. It should be noted that at a very low temperature NG begins to decompose, boiling occurring shortly thereafter. Flame from a fuse, heat released by blasting caps, a relatively warm blasthole (such as one just recently drilled), friction from metal objects, and similar effects can all provide quite easily the relatively low temperature needed to provide dangerous conditions. If the NG is confined, e.g., in a blasthole, the initial decomposition will be accelerated to result in detonation.

On the other hand, AN requires a fairly high temperature before it will begin to decompose and fume, necessitating a large amount of initial heat. However, once decomposition begins, detonation or deflagration will follow with a very small temperature rise. By combining the two ingredients, as is done in the ammonia dynamites, a compromise effect is achieved, the grades having the most NG being the easier to initiate.

Important Properties Of Explosives Most manufacturers supply catalogs and other information concerning the specifications of their products. However, certain properties are particularly important to quarry blasting. A review and explanation of their practical aspects should therefore be of special interest to the operator.

Water Resistance For all explosives, the presence of water in blastholes tends to promote chemical unbalance, as well as retard the heating reaction. Water supplies additional hydrogen and oxygen and requires additional heat to be vaporized into steam. If water is flowing through the ground, a leaching action can occur, whereby certain salts that may be easily dissolved could be removed from the explosive mixture. Explosives may be protected internally from water action by gelatinizing the mix or externally by cartridging. The ingredients added for gelatinizing are usually included in the chemical balance, as with the use of nitrocellulose in the gelatin grades.

Table 5—Comparison of Approximate Reaction Temperatures (°F) of NG and AN

	NG	AN
Detonate	420	460
Boil	290	—
Decompose	140	410
Freeze	50	340

Similarly, the paper, wood fiber, paraffin, or polyethylene used for external cartridging are generally included in the chemical balance. For this reason explosives that are made for use in cartridges should not be removed if preservation of the oxygen balance is to be maintained.

If an explosive is properly compounded initially, but detrimental effects occur from water, the action will be noticeable by the formation of brown nitrous-oxide fumes and a low blasting action. If these effects are observed, the explosive grade should be changed or other appropriate action taken. Primers must of necessity possess unlimited water resistance.

Fumes Most explosives are given a fume rating, the classification of which is based on the amounts of poisonous gases produced by the explosive reaction. Limits are set by many of the states, the U. S. Bureau of Mines, and certain other agencies. Where inadequate ventilation and exposure of personnel to toxic gases may exist, care must be taken to ensure that the explosives used give amounts below the established limits.

This property is particularly important for underground blasting; but for open-cut operations the problem could also be quite serious. Fumes may lie inside piles of broken rock. Such material, when stirred up by loading equipment, will release the fumes, to contaminate the air in which men are working. The problem may be aggravated by atmospheric conditions, deep cuts, and similar factors that hinder air circulation. Men will become ill and nauseated if this situation is present.

A person should understand the distinction between fumes and smoke, the latter of which is composed of liquid or solid particles suspended in the air. Usually when white smoke is observed from blasts, it is quite likely composed primarily of the steam from the reaction.

Sensitivity This property actually refers to two related characteristics. It defines the relative ease with which an explosive reaction can be initiated and the relative ease with which the reaction is propagated through an entire charge. Several tests are used to rate sensitivity, the most common of which is the minimum booster required for initiation. Usually the total number of No. 6 strength blasting caps required for initiation is used to classify sensitivity.

However, an explosive may initiate easily but in small diameters the reaction may not propagate and dies out. For this reason explosives may not be manufactured below specific diameters. A critical diameter, or that below which propagation of a reaction will not continue, exists for practically all commercial products. Some blasting agents have a large critical diameter; most high explosives have a small one. By definition, blasting agents cannot be sensitive to initiation by a single No. 6 blasting cap, while high explosives all are one-cap sensitive.

On the other hand, an explosive may be quite insensitive to initiation but propagate easily when above the critical diameter. For safety reasons this situation is the more desirable; it is a definite advantage offered by many of the blasting agents. However, adequate priming is mandatory for their use. If propagation is difficult or impossible through a column of explosives, boosters may be used to sustain the reaction. But it should be recognized that both boosters and primers must be sensitive to initiation.

The sensitivity of an explosive is a function of the ingredients, their particle sizing, the charge diameter, the degree of confinement, and certain other factors. For example, ammonium-nitrate explosives may become quite sensitive in time by particle degradation due to the process of cycling. AN has the characteristic whereby it will change its crystalline form with changes in temperature; two of the changes often encountered in normal field blasting are at 0 and 90 deg. F. Constant changes through those temperatures causes the particles to break into smaller sizes. The smaller particles offer more contact surfaces between ingredients, making it easier for particles to be consumed by the explosive reaction. The result is to permit easier initiation and subsequent more rapid propagation through a charge. Blasting agents that would normally be insensitive become quite sensitive to initiation by a single No. 6 blasting cap, similar to that expected of high explosives.

Larger charge diameters also propagate reactions more easily because of the greater surface area available. Confinement tends to concentrate the reaction's force along the charge length rather than permit the action to spread.

Certain hydrocarbons have an adverse effect on some types of explosives, principally those with free NG, as do the straight and extra grades of dynamites (Table 6). Since some of the blasting agents have liquid hydrocarbons as one of their ingredients, e.g., FO, one should be particularly cautious in his choice of primer explosive. Under certain conditions there could be an accumulation of the hydrocarbon in the blastholes, particularly at the bottoms, which in turn may lead to misfires when charges are bottom-primed. This situation can be avoided by using gelatins or simigelatins or high explosives containing no NG for priming. Furthermore, it is simply good practice to avoid the use of excessive FO in any blasting agent, to avoid upsetting the oxygen balance.

Density Explosives are manufactured and sold on a weight basis, the densest explosives usually being the strongest. The density, or weight per unit volume, of an explosive is therefore one of its most important properties. In industry this property may be specified in three ways: (a) by specific gravity (SG) expressed as a unitless number *or* in gm/cc; (b) by stick count (SC) or the number of 1¼ x 8-in. cartridges per 50-lb. box; and (c) by loading density (d_e) or the pounds of explosive per foot of charge length. The value for the loading density, however, is a function of the explosive's charge diameter

Table 6—Percent by Weight of Diesel FO Additive Where Detonation Fails

Explosive	Pct. Add.	Qt. FO/lb. of Expl.
Extra dynamite 40%	1.5	0.008
Extra dynamite 60%	2.5	0.014
Low-density dynamite (SC 120)	4.0	0.022
AN gelatin 60%	8.0*	0.05*
NG gelatin 60%	39.0*	0.21*

*Amounts applied, but detonation successful, no failures.

(D_e), which should then also be specified easily for clarity.

The various measures for density can be calculated easily for rapid use in the field, provided that the charge diameter (D_e), expressed in inches, and one of the density values are known. The relationships are as follows:

$$d_e = 48D_e^2/SC \qquad (1)$$

$$d_e = 0.34D_e^2(SG) \qquad (2)$$

$$SG = 141/SC \qquad (3)$$

These formulas provide a very convenient means for estimating explosive quantities, in that most explosive manufacturers supply the SC or SG for their products. For example, if a free-flow AN-FO mixture with an SC of 176 were to be used in a 10-in. diameter blasthole, one would expect slightly in excess of 27 lb. per foot of hole (or $d_e = 48 \times 10^2$ divided by $176 = 27$ lb./ft.). (The relationships are illustrated graphically by Figure 12.)

It will be noted that an SC of 176 corresponds to an SG of 0.8, which could also be determined from

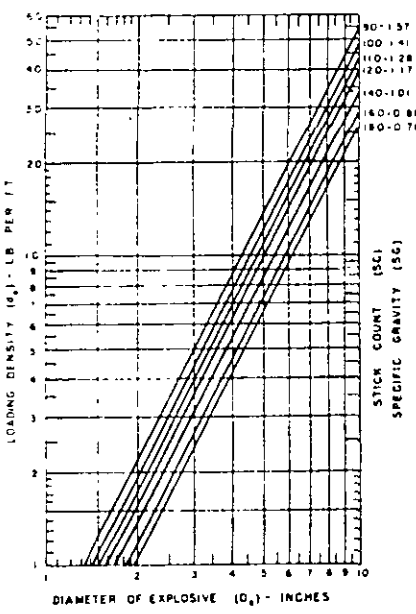

Figure 12—Relationships between densities of explosives.

Expression (3), above. Since the SG of water is 1 and its equivalent SC value is 141, any explosive with an SG *greater* than 1 or an SC *less* than 141 could be expected to sink in wet blastholes. It should be pointed out, however, that D_e is the diameter of the explosive, *not* that of the blasthole. These diameters are equal only in the case of free-flowing explosives or charges composed of cartridges that are thoroughly tamped.

Because certain ingredients may be included in explosives that do not contribute to the energy produced, there is no distinct relationship between density and pressures developed. In fact, some manufacturers make a 40 percent Extra type dynamite, for example, that is denser than the 60 percent of the same type of explosive. Similarly, a 90 percent gelatin is lighter than a 30 percent gelatin. But as a general rule it is reasonably approximate to relate the energy developed by explosives to their relative densities. This is because explosives are characterized by general density groups that correspond to their various types, e.g., gelatins, dynamites, etc. The denser types as a group produce more energy than the lighter ones, even though there may be exceptions to the rule between grades within the same type.

Velocity The rate, usually expressed in feet per second (fps), at which a reaction propagates through an explosive is considered by many as the most important quality of an explosive. It is often called the detonation velocity, but this is not always technically correct. Its importance can be better appreciated when it is understood that the energy produced by any explosive is a function of the *product* of its density and velocity characteristics. Since the initial reaction for most explosives used in commercial blasting is detonation with subsequent gaseous expansion, the action would be considered dynamic.

Thus, impulsive and momentive forces are produced as a result of the kinetic energy of the reaction, which can be expressed by the relationship $KE = \frac{1}{2}Mv_e^2$, where M is the mass and v_e is the velocity of the explosive's reaction. The relationship is given to illustrate that the value of the velocity is squared. Thus, energy releases are affected much more by changes in velocity than by changes in density. For example, if one of two different explosives has *double the density* of the other but both have the *same velocity*, the denser one could be expected normally to produce twice the work. However, if both explosives have the *same density*, but one has *double the velocity* of the other, the faster explosive would produce *four* times the work possible from the other.

Contrary to common belief, all high explosives do *not* react with high velocities, which may vary from about 24,000 fps to as low as 5,000 fps. The velocity of an explosive is related to the sensitivity in some respects, being dependent on the particular ingredients used, their particle sizing, the density, the charge diameter, and the degree of confinement under which it is used. As explained earlier, the smaller the particles the greater the density, which in turn usually increases the amount of energy-producing material per unit of volume and the number of contact surfaces between particles, thereby increasing the over-all rate of reaction. The combined effect is to increase the energy potential of the explosive.

Explosives are given two velocity ratings, one for use in the open or unconfined, the other if it is confined. For many grades and types, the unconfined velocities are 20 to 30 percent lower than those achieved under confinement. In a practical sense one could then assume that an explosive would produce only 60 to 70 percent of the total work possible if used unconfined. It is, therefore, particularly important to know which velocity value is specified for a product.

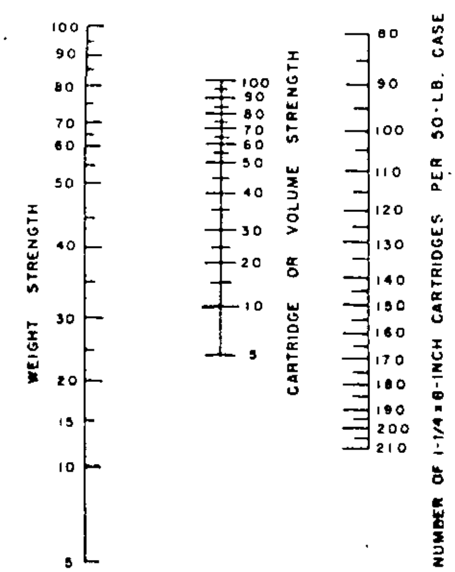

Figure 13—Chart for correlating explosive strengths.

The technique known as cushion blasting utilizes the principle of reduced velocities resulting from less confinement. It can be used to prevent shattering. In this method an annular air space is left around the explosive, if used in cartridges, or air pockets are left at prescribed intervals between deck charges placed along the length of a blasthole.

Strength The least understood and often the most improperly specified property for describing an explosive is its strength. It is usually expressed as a percentage, and it was originated when all commercial high explosives contained NG as the primary energy-producing ingredient. In the beginning, the percentage meant the actual amount of NG in the total weight of explosive, which would be applicable for most of the straight dynamites. However, for all other types of explosives other ingredients may be used to supply a part or all of the energy. In addition, there are two strength ratings given to explosives; and unless this is clearly understood by users, it can lead to very serious difficulties.

The first method for rating—*weight strength*—means that a pound of a particular explosive can do the same work as a pound of straight NG dynamite of equivalent strength when used under certain specified conditions. Since densities of explosives vary considerably although the explosive or blasthole diameter may not be changed, a method for rating strength on an equal volume basis would be necessary.

The bulk, cartridge, or *volume strength* rating provides the necessary comparison, but its value is determined by calculation. The two strength ratings, by weight and by volume, are considered equal when the stick count (SC) is near 100, as it would be for most straight dynamites. To assist in the correlation between the two ratings, the nomograph in Figure 13 can be used.

If the weight strength of an explosive having an SC of 150 is 60 percent, a pound of it will provide energy equivalent to that of a pound of 60 percent straight dynamite. However, from Figure 13, the cartridge strength is indicated as only 30 percent, which means that if the explosive was used on an equal volume basis, it would have the energy of only a 30 percent straight dynamite. Unfortunately, some explosives are sold and designated by weight strength, and others by bulk or volume strength; and still others are specified by letter or number, with a weight strength given for the general class or type of explosive in which it is but one of the grades

The operator can understand that he could be badly mistaken if he were not careful to distinguish between the two strengths in using this property as a primary basis for selecting an explosive. To avoid confusion and possible serious difficulties, it is generally much simpler to judge an explosive's relative strength according to its density and velocity characteristics. The quantities of both are usually available from the manufacturer's information.

Correlating Explosive's Properties to Blasting Standards Since the burden is the most important single dimension for successful blasting, and that upon which the design standards are based, its determination must take into account the individual characteristics of the particular explosive selected for use on a job. A convenient method for estimating its value is to employ the relative-energy comparison technique. Because all properties may be considered relative for comparison purposes, an explosive with an SG of 1.2 and a v_c of 12,000 fps could be considered the standard, or one with characteristics near that, for 40 percent to 60 percent Extra dynamites, which long have been considered appropriate explosives for quarry blasting. However, it should be understood that any standard might be used for making a comparison.

To estimate the relative energy potential of an explosive, the diameter (D_e), density (SG), and velocity

(v_e) must be known, or approximated. Furthermore, to simplify calculations, one can assume blastholes would be filled across their entire diameter, or $D_e = D_H$. This condition ensures little or no energy losses, or dampening, for a complete energy transfer from the explosive's reaction into the surrounding rock to be blasted.

The relative energy (RE) and that exerted to the rock could then be expressed by a simplified kinetic-energy relationship, or $RE = a(SG)v_e^2$. The "a" is a conversion factor to permit the use of specific gravity instead of mass, and it assumes that the explosives will be used in the same diameter. For any set of similar field conditions the "a" will be a particular constant number, making it then possible to omit it from the relationship when explosives are compared under identical field conditions. Thus, the following expression can be used for comparing two or more explosives, based on their energies:

$$RE_2/RE_1 = (SG_2)(v_{e2})^2/(SG_1)(v_{e1})^2$$

If Explosive No. 1 represented the average explosive ($SG_1 = 1.2$ and $v_{e1} = 12{,}000$ fps) and Explosive No. 2 had $SG_2 = 1.5$ and $v_{e2} = 18{,}000$ fps, the relative energy of the second compared to the first according to Expression (4) would be as follows.

$$RE_2/1 = (1.5)(18{,}000)^2/(1.2)(12{,}000)^2 = 2.8$$

The RE value shows then that the second explosive has 2.8 times the energy potential of the standard explosive. Since the comparison is made between explosives used for blasting the same material, the comparative blast results in the rock would vary as the cube root of their relative energy value. The cube root is used rather than the direct ratio because of the spherical fan effect for energy propagation through homogenous materials. This relationship then tells us that the K_B ratios and therefore the burdens will vary in proportion to the cube root of the explosives' relative energies. To provide a simple formula for illustrating the relationship, the following may be used:

$$K_{B2} = K_{B1}(RE_2/RE_1)^{1/3}$$

If one assumes that average rock will be blasted, a K_B value of 30 would represent the average explosive (Figure 7). The burden used

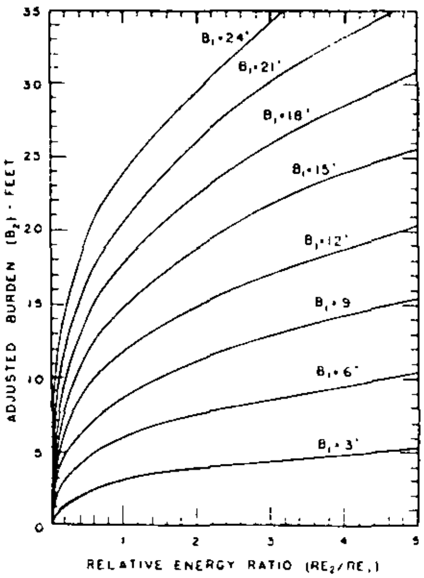

Figure 14—Relationships between burden dimensions for explosives according to their relative energy and when used under field conditions.

would be 7½ ft. for a 3-in. diameter explosive, since $K_{B1} = 30 = 12B_1/D_e$, which gives $B_1 = 30D_e/12 = 30 \times 3/12$ or 7½ ft.

For Explosive No. 2, then, using Expression (5), one can approximate that $K_{B2} = 42$, or $K_{B2} = 30(2.8)^{1/3}$. The burden for the second explosive would then be 10½ ft., since $B_2 = 42D_e/12 = 3½ \times 3$. For direct calculation of the burdens for explosives used in the same diameters and under identical field conditions the following may be used.

$$B_2 = B_1(RE_2/RE_1)^{1/3}$$

The relationships given by Expressions (5) and (6) are shown on Figure 14, which permits one to determine the approximate new burden for any explosive as compared to the average explosive when used under identical field conditions.

Although the example given illustrates ideal conditions and one should recognize that many variables enter into making the final selection of a K_B ratio and its related subsequent burden dimension, the relative-energy comparison technique gives a realistic approximation. As a matter of interest, for most explosives used in blasting the maximum density variation is from 0.7 to 1.6, with a velocity variation from 8,000 to 20,000 fps, the heavier densities having the higher reaction rates. Therefore, the weakest explosives possess only 26 percent of the energy available, while the strongest have 370 percent of the energy available, as compared to that available from the average explosive. Converted to K_B values and using a $K_B = 30$ for the average explosive in average rock, the lower and upper limits for K_B values would be 19 and 46, respectively. From Table 1 it can be seen that these values satisfy results from actual field experiences.

The Mechanics of ROCK BREAKAGE

MATERIAL PROPERTIES, POWDER FACTOR, BLASTING COST

Part IV of a Series

MATERIALS PROPERTIES AND INFLUENCE

MOST materials requiring blasting are not homogeneous nor are their properties the same throughout. Of all the physical properties, there are essentially five that predominantly influence blasting results. These include in order of their importance the following characteristics: (1) structure, (2) resilience, (3) strength, (4) density, and (5) velocity of energy propagation. Blastability, elasticity, hardness, toughness, and other terms may also be used to describe a material, but often such expressions are too indefinite and difficult for the ordinary quarry man to understand. Drillability, or ease of drilling, should in no way be confused with the manner in which a material can be blasted.

Structure The structural features of a material usually have the greatest influence on blast effects. To better understand their importance one should recognize that rock, as we think of it, is essentially an accumulation of small particles bonded together. The constituents are oriented in definite structural patterns, established during the formation and alteration processes. Of primary importance to blasting is compression jointing, existing within all rocks (igneous, sedimentary, and metamorphic) and composed of planes along which there is no resistance to separation. Igneous rock may also have tension jointing, formed during the cooling process.

Sedimentary rocks are unique in that they have stratification planes (in addition to joints), which were originally horizontal and formed by interruptions in the initial deposition of sediments. Stratification and jointing are not the same thing. For metamorphic rocks, the relationship of their jointing to schistosity is similar to that between jointing in sedimentary rocks and their stratification, both in angular position and mechanical development.

Jointing is usually easily detected, the planes being generally smooth and often short distances apart. One set of planes is parallel with the dip and strike of the rock formation, with two or more sets being nearly perpendicular to the first set. Rocks when broken will separate into blocks of a shape characteristic of their particular jointing pattern, and the new faces produced from blasting tend to follow the jointing directions. (See Figures 3, 4, 8, 10, and 15.)

For the sedimentary rocks there is one particular direction along which jointing is the most pronounced, the other planes being less dominant. The horizontal angles between the vertical jointing planes are usually near 75 and 105 degrees, which form rhombohedrons when the rock is broken. Igneous rocks, however, have jointing planes of uniform strength, the angles between planes being most often near 60 degrees. The fragments produced from blasting are generally hexagons or pyramids in shape.

Jointing directions can be found quite easily if it is recognized that most faults, cliffs, mud seams, caves, etc., produced by weathering and the other geologic actions tend to follow the jointing planes. It is particularly important that the blaster endeavor to locate the planes before laying out a drill pattern. Blastholes located in tight corners will generally overbreak, opening large cracks in the ledge. Subsequent blasts will usually do no more in those areas than give large boulders, and possibly be quite violent. It can be seen from Figure 15, which illustrates a representative quarry in a sedimentary rock formation, that there are tight (75-degree) and open (105-degree) corners. This means that normal blasts under those conditions should be directed out of the open angles in so far as possible, or toward the east or west. If blasting is done in the other directions, or to the north or south, cracking of the solid ledge will occur along the planes forming the tight angles.

Another structural feature that is

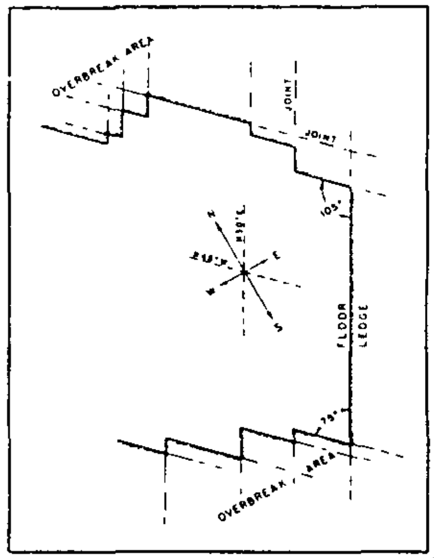

Figure 15—A representative plan sketch of a quarry in a sedimentary rock formation, showing tight (75-degree)—and open (105-degree) corners.

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very important, particularly to rock fracturing, is the type and strength of the bonding between individual grains. For example, rock may have pronounced jointing at widely separated distances, but the material between joint planes may be strongly bonded, or massive in character. Large boulders invariably result when blasting is carelessly done under this condition. On the other hand, rocks may be highly laminated or stratified, or the bond between grains may be very weak, so that fragmentation is always easily accomplished by merely moving the material from its original place.

Table 7—Properties of Various Selected Materials

Name and Location	Compressive Strength (psi)	Modulus of Rupture (psi)	Specific Gravity (SG)	Density (d_r) (ton/cu. ft.)	Longitudinal Velocity (v_l) (fps)
Amphibolite (fine grain, India)	61,400	7,400	3.12	0.097	19,000
Basalt (New York)	46,600	8,000	2.94	0.092	18,700
Basalt (Michigan)	33,400	3,800	2.85	0.089	15,200
Basalt glass	—	—	2.81	0.088	21,000
Diabase (fine grain, Michigan)	44,200	5,300	2.94	0.092	16,700
Dolomite (Missouri)	8,800	1,000	2.80	0.087	—
Dolomite (Tennessee)	46,700	3,800	2.84	0.089	17,900
Gabbro (altered, New York)	40,200	5,400	2.93	0.091	17,600
Granite (Georgia)	28,000	2,000	2.64	0.082	8,900
Granite (Vermont)	33,200	2,900	2.66	0.083	11,100
Granite (Nevada)	39,500	3,900	2.63	0.082	14,500
Granite (North Carolina)	30,400	1,600	2.60	0.081	8,000
Greenstone (Michigan)	45,500	3,300	3.30	0.103	16,600
Gypsum (Indiana)	3,200	1,200	2.32	0.072	—
Limestone (Ohio)	28,500	2,900	2.69	0.084	15,400
Limestone (Utah)	28,000	2,200	2.78	0.087	15,900
Limestone (fossiliferous, Indiana)	10,900	1,600	2.31	0.072	12,400
Limestone (West Virginia)	23,000	1,900	2.68	0.084	16,400
Marble (Maryland)	30,800	2,800	2.37	0.074	13,700
Marble (New York)	18,400	1,700	2.72	0.085	14,500
Obsidian	—	—	2.35	0.073	16,100
Quartzite (taconite, Minnesota)	91,200	3,400	2.75	0.086	18,200
Rock salt (Louisiana)	5,000	Negligible	2.50	0.078	—
Sandstone (Ohio)	10,400	500	2.06	0.064	5,600
Sandstone (West Virginia)	19,400	3,400	2.50	0.078	12,900
Sandstone (Utah)	11,500	620	2.17	0.068	8,400
Sandstone (Alabama)	26,800	2,200	2.76	0.086	12,500
Shale (Utah)	31,300	2,500	2.81	0.088	14,900
Shale (West Virginia)	11,600	4,200	2.40	0.075	13,600
Syenite (New York)	34,300	2,800	2.72	0.085	14,500
Alluvium, broken rock, loess	—	—	1.3-1.5	0.044	2,300
Clay	—	—	2.58	0.081	5,900
Air	—	—	0.0012	—	1,080
Water	—	—	1.00	0.031	4,750

Resilience This property, sometimes called sponginess or toughness, refers to the elasticity of a material. It is used to express the capability of a rock to resist shock and recover its original position and shape without being ruptured. If a rock on being dropped, for example, makes a dull thud and does not rebound, it would be very difficult to break by impact. Brittle rocks, however, shatter easily, particularly those types having a high silica (quartz) content. A blaster can generally determine quite easily whether or not a material will break into small sizes or large coarse fragments by conducting a simple drop test. Furthermore, the test provides a clue as to the energy absorption power of the material, which is important for estimating the amount of additional charge, or energy, that would be necessary to overcome expected energy losses.

Strength Of the characteristic strengths of materials, blasting is normally concerned only with that of tension. Most rocks are very weak in tension, more resistant to shear, and strongest in compression, having approximately only one-tenth the resistance to tensile rupture that they have to failure by compression (Table 7). However, shear is not actually a force by itself but rather the result of two forces, either two tensile or two compressive forces, or a combination of one of each, which act along different lines and directions.

To know the actual strengths of a material, samples must be tested in a laboratory. (Regular tensile-strength tests are usually difficult to conduct.) However, tests for what is known as the modulus of rupture are much easier to perform; yet they provide information that is just as useful in providing tensile-strength data of equal practical value. In fact, the laboratory test for the modulus breaks samples in tension by bending test slabs until they fracture, much in the same manner that rock is stretched and broken at an open face during blasting (Figure 3).

Quite often it is impossible or quite impracticable for quarry operators to have tests conducted. Also, test results on samples may not necessarily provide information on the over-all strength of a rock deposit, except when the material is homogeneous and very massive. Nevertheless, if tests could be made, the data would aid greatly in determining the stress levels (psi) required for fracture. It is the resistance to tensile rupture that must be exceeded by the energy pulses at the free faces, and thus, if known, could also give an approximation of the required burden dimension and the explosive pressures needed for proper breakage. In the event specific test data cannot be obtained, the operator may find the information in Table 7 quite useful. From the various moduli listed for many of the representative rock-types, a practical estimate can be made that will approximate the characteristics of his particular deposit.

Density Denser materials require greater amounts of work energy to be satisfactorily broken and displaced, and heavier explosives or large charges will therefore be needed. However, from Table 7 it can be concluded that for most rocks there is a very narrow range of density differences, with SG values varying from 2.3 to 3.3 in most instances. The materials generally requiring blasting have densities confined to the 2.5-2.9 SG range. This can be interpreted to mean that the influence of rock density alone has a limited effect on blasting, the extreme conditions being within 15 percent of the average 2.7 SG. One may then reasonably assume that rock density by itself is of little importance to blasting and would not appreciably affect a K_B value or burden dimension.

Its importance, however, lies in the fact that it does influence costs and the other physical properties. Although densities are most often given by specific gravity, for calculations in costing and powder factor determinations it is more convenient to use the density ratio, d_r, expressed in units of tons/cu. ft. of solid material. If the d_r value is not known, one can utilize the following expression for converting any SG that may be given:

$$d_r = SG(62.4/2000) = 0.0312(SG), \text{ tons/cu. ft.} \quad (7)$$

Velocity The velocity of energy transmission in rock, v_r, is like the reaction velocity for explosives, v_e, in that it increases as rock density becomes greater. The denser rocks are often the least porous and are generally composed of small grains, which permit easier propagation of energy through the material. For this reason most dense rocks have smaller energy losses due to dampening, and they often have a tendency to shatter rather than break into slabs. Most brittle rocks also transmit energy at very high rates, except in the unique case of certain sandstones. The characteristic low velocities of many of the sandstones are due to a peculiarity in their composition: the matrix bonding the sand grains may be clay, lime, or other energy-absorbing substances. However, if the matrix is silica, the velocity is quite high.

Velocities for materials are usually specified as longitudinal velocities, v_l, as are also those given in Table 7. But these values are normally slightly lower than the velocity of energy propagation, v_r. The two velocities are related by the following expression:

$$v_r = v_l[(1-\mu)/(1+\mu)(1-2\mu)]^{1/2} \quad (8)$$

Because μ, or Poisson's Ratio, is usually considered as 0.25 for estimations, it is more convenient to convert velocities by using $v_r = 1.095 v_l$ for approximations. However, it is more practical and will not introduce any great error if the two velocities are considered equal.

The importance of velocity in rocks on blasting is that it has a strong influence on the amount and manner in which a material will be stressed. In order that the momentive forces be conserved, there should be nearly perfect coupling of the energy from an explosive's reaction with the surrounding material. The matching of the momentive energies is considered necessary theoretically for the most efficient blasting results. This condition is known as acoustical coupling. Since the energy required for stressing strong and dense rocks would be relatively large compared to that needed for lighter materials, the use of denser, fast-reacting explosives is generally advisable.

The velocity of a rock will determine the time it takes the stress energy to reach free faces and return. The velocity of an explosive, on the other hand, will determine the total time it takes for an entire charge to complete its reaction. The relationship of the two velocities, called the velocity ratio or $K_v = v_e/v_r$, has a very important influence on the manner in which an entire blast will function. This is because the K_v ratio defines the shape of the composite wave produced by all the individual stresses introduced into the rock from each point along a charge column (see Figure 6, PIT AND QUARRY, September, 1963, page 119) the primer positions thus controlling which faces are fractured first and the direction in which the composite wave will travel in the rock.

The K_v ratio, primer location, and general design features of a blast must follow certain definite relationships, if results are to be satisfactory. In particular, the influence of rock velocity is such that there will be a certain optimum of critical hole depth for each blasting situation. For example, when a charge is bottom-primed, there will be a specific *minimum* hole depth. If the depth is less than the minimum value, blast effects will begin near the collar region, which quite likely may promote violence and air blast. In some instances, toe will be left

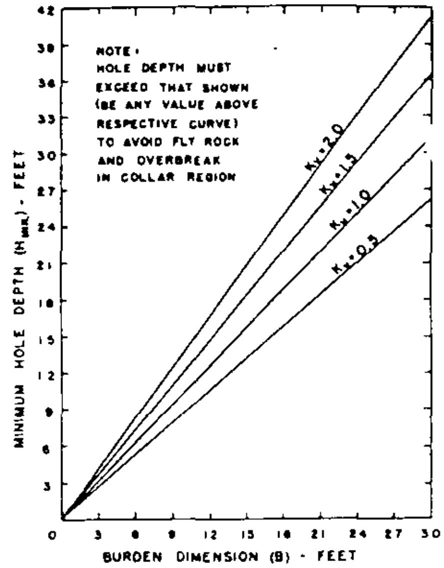


Figure 16—A graphic presentation of the relationship between minimum hole depth and burden dimension.

at the floor. However, when holes are deeper than the minimum value, stressing and rock movement will always begin at the ledge bottom before action occurs in the collar region. The particular minimum required depth of hole can be determined from the following expression:

$$H_{min} = K_v[(B^2+J^2)^{1/2} - T] + T \quad (9)$$

The relationship is illustrated graphically in Figure 16, in which $K_T = 0.7$ and $K_J = 0.3$ are considered average conditions. The values for the H_{min} represent balanced stressing at both the toe and collar regions.

If charges are collar-primed, stressing will always begin in the collar region, unless the amount of stemming used exceeds the burden dimension. Even under that condition, collar overbreak and air blast may occur, with possible toes resulting, if a particular *maximum* hole depth is exceeded. This limiting condition can be determined from the following relationship:

$$H_{max} = K_v(T - B) + T \quad (10)$$

From a practical viewpoint, the expression shows that under no circumstances should the stemming dimension be less than that for the burden in blasting massive rock. Otherwise, collar cratering and air blast can be expected. The condition becomes particularly critical when detonating fuse is used and initiation is done on the surface, since the fuse on detonating has the tendency to loosen the stemming. For deep holes, collar priming would definitely be undesirable under conditions where massive cap rock occurs in the collar region and where column loading is practiced; i.e., the charges are continuous from just below the stemming to the hole bottoms.

An unusual situation exists when the K_v is less than 1, or when the rate of travel of the compressive stress-wave in the rock exceeds the speed of the detonation wave in the charge column (Figure 6). Stress waves will reach free faces before the explosive has completed its reaction, with rock at the faces being repeatedly stressed by the pressures produced by the still reacting explosive column. The action reinforces the stresses and reduces the resistance of the rock to fracture, giving the impression that the explosive is stronger than it actually is. Under certain conditions, blasts are extremely efficient, but they are usually difficult to control, producing greater heave or throwing action.

Since there are critical hole depths for each blasting condition, the best results can often be insured by first estimating the particular K_v value for the conditions present, and then placing primers accordingly. Control for very deep holes, for example, is achieved by using primers both near the collars and in the hole bottoms; or primers may be placed at strategic intervals throughout the columns, with or without the use of deck charges. Either detonating fuse or close-interval delay blasting caps can be used for initiating the primers, those near the collar being preferably of a longer delay. The composite effect of using primers at both the collar and hole bottom is that it extends the optimum hole depth and better distributes the stresses in the ledge, notably in the toe and collar regions.

POWDER FACTOR AND ITS SIGNIFICANCE

A guideline used by many for estimating and evaluating blasting is the Powder Factor, Pf, an expression which relates the yield of material blasted to the quantity of explosives used. For quarry work and mining, the Pf is most often stated in tons/lb., or vice versa, while for most construction excavation it is customarily expressed in lb./cu. yd or cu. yd./lb. The latter ratio is also commonly used for much of the work in overburden removal for coal and metal-ore operations. Of all the different ratios in common use, only those utilizing weights, e.g., tons/lb., take into account any of the properties of the materials being blasted.

Because of its extremely variable character Pf is not normally a sound index upon which to judge blasting efficiency or design blasts, as many believe. Different values will be obtained by merely changing the blasthole pattern or configuration, and values will also change for other reasons, such as variable hole depths and deck loading. Also, the many different standards employed tend to confuse rather than assist persons in evaluating results. The most practical value of Pf is in cost analysis, because explosives are sold by weight, and payment for materials mined or removed is generally made on a weight or volume basis.

One of the ways in which the powder factor can vary is shown by the examples given in Figure 17. These sketches illustrate four possible ways of blasting with a single charge and six different patterns utilizing a V-cut arrangement for multiple charges. All the blasts are conducted under identical conditions except for the relative positions of open faces. Pertinent data for Figure 17 are given in Table 8. The information there given is merely representative and used for comparative purposes. It may or may not fit actual blasting situations.

In determining the possible yields given in Table 8 for the various blasts shown in Figure 17, the surface blast areas, A, were approximated based on the locations of open faces, assumed rock structural features, and the particular mechanics of how each specific blast would be expected to function. The excavation volume would then be the product of the blast area and the ledge height, L, not the hole depth, H, as some might assume. Simple conversion to tonnage yield, W, was accomplished by multiplying the volume by the material density, d_r, using the following relationship:

$$W = AL(d_r), \text{ tons} \qquad (11)$$

The quantity of explosives used, E,

Figure 17—These sketches show four possible ways of blasting with a single charge and six patterns utilizing a V-cut arrangement for multiple charges.

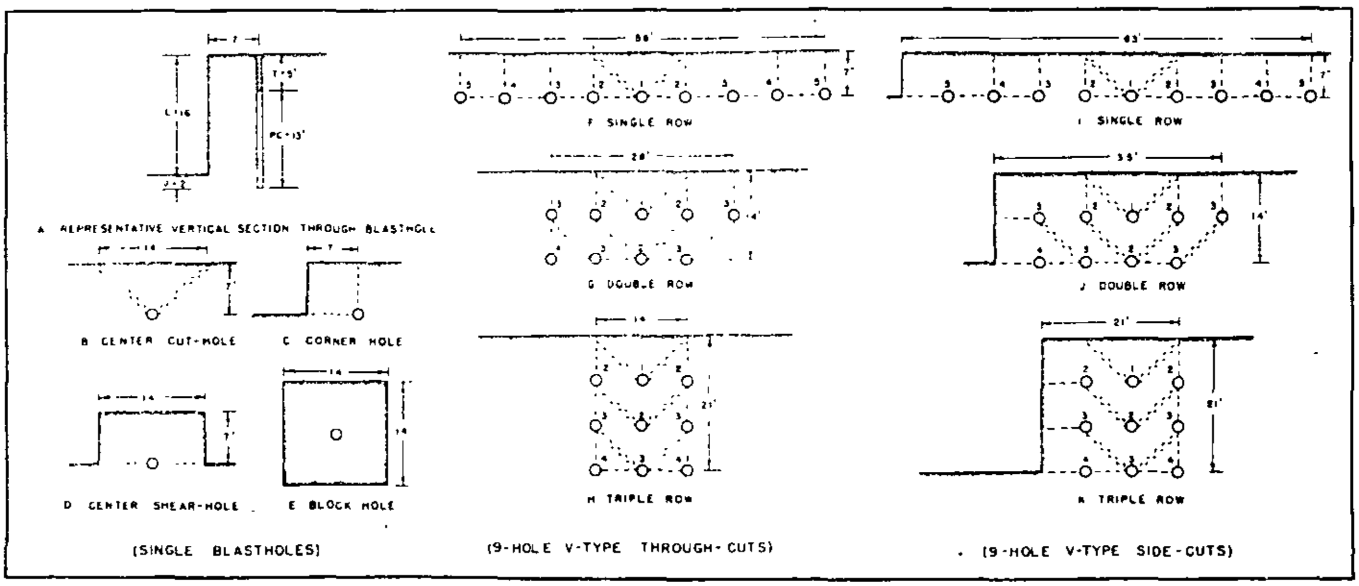


Table 8—Change in Powder Factor (Pf) With Variation In Drill-Pattern Configuration [a]

(For blasting limestone with $d_e = 0.084$ ton/cu. ft. [b] by Extra 60% dynamite, D_e=2 inches[c], and blastholes located according to average K_B ratio of 30 [d].)

	Total No. Blastholes	Total Yield (tons)	Total Expl. Used (lb.)	Powder Factor (tons/lb.)
Single charges:				
(B) Center cut-hole (2 free faces)...	1	66	22.7	2.91
(C) Corner hole (3 free faces)	1	66	22.7	2.91
(D) Center shear-hole (4 free faces) .	1	132	22.7	5.82
(E) Block hole (5 free faces)	1	264	22.7	11.64
Multiple charges: V-type through-cut				
(F) Single row	9	528	205	2.58
(G) Double row	9	462	205	2.25
(H) Triple row	9	396	205	1.93
Multiple charges V-type side-cut				
(I) Single row	9	594	205	2.90
(J) Double row	9	627	205	3.06
(K) Triple row	9	594	205	2.90

Notes: [a]—See Figure 17 for design specifications
[b]—Rf. Table 7
[c]—Rf Figure 12
[d]—Rf. Table 2

would be the product of the explosive's loading density, d_e, the total average length of one charge, PC, and the total number of blastholes, N, calculated as follows:

$$E=(d_e)(PC)N, \text{lb.} \qquad (12)$$

The powder factor, Pf, would then be the ratio of the above two expressions, or

$$Pf=W/E, \text{tons/lb.} \qquad (13)$$

In studying Figure 17 and Table 8, it will be noticed that the number of free faces has a very pronounced influence on the value of the Pf. For multiple-hole blasts, when there is a free face added on one side, the over-all Pf's for all blasts will usually be the same as that for a single corner or cut hole. However, the Pf may be affected by the initiation-timing pattern employed, which may change the blast area outline, as shown in Figure 17J and line J of Table 8. For the particular blast in point, the additional tonnage results from overbreak in the tight corner of the second row of holes. If a later-interval initiation delay were used in the corner hole, the blast would then be expected to cut squarely without any overbreak, to give the same yield as for the other two examples (Figures 17I and 17K).

Estimating or evaluating an entire blast on a single-hole Pf basis can be very misleading, but unfortunately it is a practice often followed. For the design and evaluation of underground face-blasting, the errors produced would be even more serious and costly when based on a single-hole Pf. This is because there is an automatic elimination of potential tonnage for one complete row of holes. The row may be considered as serving merely to shear the cut out of the solid without achieving any effective production It is also very important to recognize that in all blasting, when rows are added into the solid, with a subsequent reduction in the number of open faces, the Pf value will continue to change toward lower yields even though all other fundamental blasting relationships and the resulting rock fragmentation may remain substantially the same.

In surface or open-pit blasting the hole depths may vary within a particular cut or excavation, with no other changes being made in any of the other design dimensions. If column loading is practiced, the Pf will change with the hole-depth variations. The trend is illustrated by data given in Table 9, in which the values represent conditions for the 9-hole blast shown in Figure 17F. The cause for the Pf variations is the result of changes in the ratio of the amount of hole used for stemming relative to the total hole depth. To counteract the lowering of yields, deck loading could be used, a practice commonly followed for deep holes particularly This practice produces no detrimental effects on fragmentation when the decking is done properly.

Blasters should be cautioned regarding difficulties that may result from reducing the explosive loading density as a means for improving their Pf, or use of lighter grades or smaller diameter explosives. Attempts to extend drill-pattern dimensions by increasing burdens, etc., will produce similar difficulties for the same reason. Rather than sacrifice good fragmentation and displacement effects by decreasing the explosive energy, adjusting the blasthole arrangement is generally preferred. This can be done by redesign, so that more free faces are made available and charges are located more advantageously.

COST OF BLASTING

The primary concern of the quarry operator is to make a profit. To do this, costs must be kept to the minimum. Some costs, however, are interdependent, so that no single cost reduction may necessarily guarantee an over-all decrease in production expenses. It is the composite effect with which one must be

Table 9—Change in Powder Factor (Pf) With Variation of Hole Depth (H)

(9-hole single-row V-type through-cut, using Extra 60% dynamite with 2-in. D, column loaded and drill pattern dimensions* constant for blasting limestone with SG of 2.69)

Avg. H (ft.)	Avg. PC (ft.)	Avg. L (ft.)	Total Expl. Used (lb.)	Yield (tons) Total	Pf (tons/lb.)
10	5	8	79	264	3.34
12	7	10	110	330	3.00
14	9	12	142	396	2.79
16	11	14	173	462	2.67
18	13	16	205	528	2.58
20	15	18	236	594	2.52
22	17	20	268	660	2.47
24	19	22	300	726	2.42

Note: *See Figure 17 for drill pattern specifications.

concerned. In this respect many different costs and their effects on one another must be considered, some of which include the following: drilling, primary blasting, secondary breakage, loading, haulage, crushing, screening, stockpiling and reclaiming, loading and weighing for delivery to customers, supervision and engineering, maintenance, equipment and materials purchases and replacements, insurance, depletion and depreciation allowances, sales and other administrative services, royalties, stripping expenses (including ground breaking and removal), and taxes Of all costs or expenses, the first seven (and in some instances those for stripping) generally constitute the major portion of costs for quarry production

The percentage of total production costs attributed to drilling and blasting may be as low as 10 percent or as high as 40 percent The relative importance of primary and secondary breakage costs to loading, haulage, crushing, etc., will depend largely on the properties of the deposit, equipment and plant operating characteristics, and results achieved from the primary blasting. Studies on quarry efficiency show that in most cases hourly production rates for well-blasted material are nearly double that achieved for poorly blasted rock. Similar results are obtained in the other types of mining and in heavy construction work. Crushing and screening costs are likewise appreciably reduced if the material is well blasted at the very beginning. Because of these effects the trend today is to spend more for primary blasting, because the savings realized from all the other production phases more than compensate for the initial added cost for blasting. This fact is evidenced by the lower powder factor yields obtained in a great many operations.

Primary blasting expense is normally considered to be composed of costs for both drilling and explosives, including all charges for labor and material used. Before the advent of the new high-speed highly mobile drills, the respective costs for drilling and blasting were about equal. But with the new types of drilling equipment, drilling costs of many operations are only half as much as the explosive with conventional high explosives.

Table 10—Blasting Cost Analysis showing Effects from Changing the Type of Explosive (V-type side-cut[a] for vertical holes in a limestone ledge with constant Pf)

A Assumed Conditions:

(1) Kept constant are $K_T = 0.7$, $K_J = 0.3$, $K_S = 1.0$, $D_e = D_H = 3$ in., $L = 20$ ft., and $d_r = 0.084$ ton/cu. ft[b]
(2) E_1 = Extra 60% dynamite with SG = 1.28 and $v_e = 12{,}200$ fps[c]
(3) E_2 = field-mixed AN-FO, 94/6 with SG = 0.85 and v = 11,100 fps[d]
(4) All holes drilled with 4½-in. hammer track-mounted air-drill with 500 cfm compressor at average drilling rate of 400 ft per 8-hour shift[e]

B Unit Costs[n]

(1) Drilling at \$0.363/ft[e]
(2) Extra 60% at \$0.22/lb.
(3) AN-FO, 94/6 at \$0.05/lb
(4) 30-ft MS delay EBC at \$0.62
(5) 6-ft. instant EBC at \$0.17
(6) Regular Primacord at \$0.32/ft.
(7) MS delay Primacord connector at \$0.50
(8) Cast booster (½-lb primer) at \$0.50

C Blasting Data Calculations:

E_1 (Extra 60% dynamite)

$RE_1 = (1.28)(12{,}200)^2 = 191 \times 10^6$
If $K_{B_1} = 30$, then $B = 7\frac{1}{2}$ ft for equivalent drill pattern of 10 x 10 ft[f]
$T_1 = K_TB_1 = (0.7)(7.5) = 5$ ft
$J_1 = K_JB = (0.3)(7.5) = 2\frac{1}{2}$ ft
$H_1 = L + J_1 = 20 + 2\frac{1}{2} = 22\frac{1}{2}$ ft
$PC_1 = H - T = 22\frac{1}{2} - 5 = 17\frac{1}{2}$ ft
Since the blast consists of 3 rows of 3 holes each, or $N_1 = 9$ holes then $W_1 = A\,L(d_r) = 10(10)(9)(20)(0.084)$[j]
or $W_1 = 1510$ tons
If $d_{e1} = 3.9$ lb/ft[k] and $E = d_e(PC_1)N_1$[l], then $E_1 = (3.9)(17\frac{1}{2})(9) = 615$ lb.
Thus, if $Pf_1 = W_1/E_1$[m], then $Pf_1 = 1510/615 = 2.46$ tons/lb.
The total required drill footage, or $H_1N_1 = (22\frac{1}{2})(9) = 203$ ft.

E_2 (Field-mixed AN-FO, 94/6)

$RE_2 = (0.85)(11{,}100)^2 = 105 \times 10^6$
$RE_2/RE_1 = 105/191 = 0.55$,[g] or $K_{B_2} = 24\frac{1}{2}$.[h]
Thus, $B_2 = 6$ ft[i] for equivalent square drill pattern of 8 x 8½-ft.[f]
$T_2 = K_TB_2 = (0.7)(6) = 4$ ft.
$J_2 = K_JB_2 = (0.3)(6) = 2$ ft.
$H_2 = L + J_2 = 20 + 2 = 22$ ft.
$PC_2 = H_2 - T_2 = 22 - 4 = 18$ ft.
To drill a complete pattern there should be 4 rows of 4 holes each, or $N_2 = 16$ holes.
Thus, $W_2 = A_2L(d_r) = 8(8\frac{1}{2})(16)(20)(0.084)$[j]
or $W_2 = 1830$ tons
If $d_{e2} = 2.6$ lb./ft[k] and $E_2 = d_{e2}(PC_2)N_2$[l], then $E_2 = (2.6)(18)(16) = 750$ lb.
Thus, if $Pf_2 = W_2/E_2$[m], then $Pf_2 = 1830/750 = 2.44$ tons/lb.
The total required drill footage, or $H_2N_2 = (22)(16) = 352$ ft.

D Blasting Cost Comparison (Calculated from B and C, above):

Method of Initiation:	E_1 (Extra 60% dynamite) Electric		E_1 Nonelectric		E_2 (Field-mixed AN-FO, 94/6) Electric		E_2 Nonelectric	
Drilling:	(203')	\$ 73.69	(203')	\$ 73.69	(352')	\$127.78	(352')	\$127.78
Explosives:								
Dynamite	(615#)	135.30	(615#)	135.30		—		—
AN-FO		—		—	(750#)	37.50	(750#)	37.50
Primers		—		—	(16)	8.00	(16)	8.00
Initiators								
30' MS EBC.	(9)	5.58		—	(16)	8.12		—
6' Inst EBC		—	(2)	0.34		—	(2)	0.34
Primacord		—	(300')	9.60		—	(505')	16.16
Primacord MS connectors		—	(9)	4.50		—	(16)	8.00
Misc.:								
Connecting wire		1.25		—		1.25		—
Labor for loading and firing blast		2.00		1.80		3.50		3.00
Total blasting cost		\$217.82		\$225.23		\$186.15		\$200.78
Cost per ton:		0.144		0.149		0.102		0.109

E Percentage Distribution of Blasting Costs:

	E_1 Electric	E_1 Nonelectric	E_2 Electric	E_2 Nonelectric
Drilling	33.6	32.8	68.6	63.7
Explosives (Excl. primers)	62.2	60.1	20.1	18.7
Primers	—	—	4.3	4.0
Initiators	2.6	6.3	4.4	12.1
Misc	1.4	0.8	2.6	1.5
Total	100.0	100.0	100.0	100.0

Special Notes—Table 10

[a]—See Figure 17K for general drill pattern and initation-timing system
[b]—Rf. Table 7.
[c]—Rf Table 1, p 63, *Blasters' Handbook*, 14th edition, E. I. duPont de Nemours & Co.
[d]—Rf. Figure 6, p. 8, Technical Bulletin AG-2, Nov. 1960 Monsanto Chemical Co
[e]—Rf. *A Field Man's Guide to Drilling Costs*, A. W. Foster, Atlas Chemical Industries, Inc.
[f]—Rf. Table 2
[g]—Rf. Formula (4)
[h]—Rf Formula (5)
[i]—Rf. Formula (6) and Figure 14
[j]—Rf. Formula (11)
[k]—Rf Formula (2) and Figure 12
[l]—Rf Formula (12)
[m]—Rf. Formula (13)
[n]—Explosive unit costs based on schedule 1960 prices

With the introduction of inexpensive AN blasting agents, however, the drilling-explosive cost ratio has been reversed. Even though the less dense blasting agents appreciably increase the cost of drilling because of the increased number of blastholes required, the over-all drilling and blasting cost in most instances has been materially reduced This is because of the tremendous savings in costs of explosives. Such blasting agents often cost only 20 to 30 percent as much as the conventional high explosives.

To illustrate the effects of the various components that determine primary drilling and blasting cost, Table 10 presents representative data for a typical quarry blast Only the type of explosive has been changed, with the powder factor, drill-pattern general arrangement, and initiation-timing system kept the same. It should be noted from the data, however, that for conventional dynamite, i.e., Extra 60 percent, a typical 10- by 10-ft. pattern is used.

In order to use a regular AN-FO 94/6 blasting agent (field-mixed), the pattern dimensions are changed to an 8- by 8½-ft. arrangement. This is done according to the principles outlined earlier in the discussion on correlating the properties of explosives to the blasting standards. In this instance, the net result is that 16 blastholes are required for the AN-FO blast, compared to only nine holes for when Extra 60 percent is used. Because of the difference in the required true-burden dimension, other design dimensions necessarily must be adjusted to give a properly balanced blast. However, the basic K_T, K_J, K_S, and K_H ratios are kept closely to the same values for both blasts, only the K_B ratios being adjusted to suit the various characteristics of the explosives.

From the costs indicated in Table 10, one would logically conclude that everyone should change to AN-FO blasting agents. However, it must be kept in mind that individual circumstances may greatly change the over-all cost relationships. The factors that have the greatest influence on the final values would be the unit costs for drilling and explosives materials used and the particular properties of the explosives themselves, since the la[illegible] [illegible]ide the final required drill p[illegible] [illegible]imensions, i.e., the K_B. Fu[illegible], some explosives simply w[illegible] not be suitable for use under certain quarry operating conditio[illegible] One should, therefore, recogniz[illegible] for making a cost analysis, t[illegible] [illegible]ual values for expenses and [illegible] of materials peculiar to th[illegible] local circumstances should be used, not general estimates, as was [illegible] for Table 10 data.

The influence of the pro[illegible] of explosives on final costs cannot be overemphasized, this is tr[illegible] [illegible]icularly of the velocity of the [illegible], since it has a very promin[illegible] effect on the most desirable dri[illegible] [illegible]tern. As described earlier, the [illegible]ufacturer's specifications may [illegible] [illegible]early define whether the ve[illegible] for unconfined or confined b[illegible] [illegible] which charge diameter ap[illegible] As one can see from Table 11 specifications vary considerabl[illegible] fact which in turn greatly aff[illegible] estimates for designing blast[illegible] [illegible] on energy potential (RE) of [illegible] [illegible]od-

Table 11—A Comparison of Published Explosives Specification[illegible]

(For competitive grades equivalent to 60% ammonia dynamit[illegible] when used with D.=3 in. and based on published data)

	American[a] Ammonia Dynamite	Apache[b] Standard Dynamite	Atlas[c] Extra Dynamite	Du Pont[d] Red Cross Extra	Hercules[e] Extra Dynamite	Olin[f] Special Dynamite	Trojan[g] Standard [illegible]
Velocity (fps)	10,800	12,800	10,000	12,200	12,450	13,60[illegible]	[illegible],600
Open (O) or confined (C): ..	Not given	(0)	(0)	Not given	(0)	Not given	(0)
Charge diameter (inches): .. .	Not given	1¼	1¼	1¼	1¼	Not given	[illegible]
Stick count:	110	110	110	110	110	10[illegible]	116
Specific gravity	1.28	1.28	1.28	1.28	1.28	1.[illegible]	[illegible]22
RE factor (X10^6)	149	210	128	191	198	2[illegible]	193
Relative energy Ratio (RE_2/RE_1)[h]	0.78	1.10	0.67	1.00	1.04	1.2[illegible]	1.01
Adjusted burden (B_2 in feet):[i] .	6.5	8.0	6.2	7.5	7.7	8.2	7.5
Equivalent drill Pattern (square):	8x9	10x11	8x8	10x10	10x11	11x[illegible]	[illegible]x10

References

[a]—p 2. Ammonia Dynamites specification sheet A-2141-300-4/61, American Cyanamid Co.
[b]—p 16. Apache Explosives catalog, third revision Apache Powder Co.
[c]—p. 21. Atlas Explosives Products. Catalog No 13, 1957, Atlas Chemic[illegible] [illegible]dustries, Inc
[d]—p. 63, Table 1, *Blasters' Handbook*, 14th edition, 1958, E. I. du Pon[illegible] [illegible]emours & Co
[e]—p. 4. Hercules Explosives, Blasting Agents and Blasting Supplies, [illegible] 1959, Hercules Powder Co.
[f]—p 9 Olin Explosives Products catalog, fourth edition, 1955, Olin Mathie[illegible]n Chemical Co.
[g]—p. 4. Trojan Explosives and Blasting Supplies, Catalog No 101, Trojan [illegible]der Co.
[h]—Relative energy ratios calculated on basis of Du Pont Red Cross [illegible] 60% as unity.
[i]—Figures 7 and 14, with KB=30 for Du Pont Red Cross Extra 60%

uct. The suggested drill-pattern arrangements will not give the same powder factor yields but should produce comparable blast results, if the published specifications are not in error.

The expenses for primers and initiators may have a greater influence on final costs than one might expect, from the data indicated in Table 10. For blastholes with deck charges and those having extremely short depths, the costs for primers and initiators may constitute a considerable share of the over-all cost. Nevertheless, under such conditions the inherent savings resulting from higher powder factor yields usually compensate for the added costs. As experience has clearly shown, it is simply good practice always to use the best primers available. As a rule, the total required quantity of powerful high-energy primers is much smaller than that needed when cheaper low-energy explosives are used for priming. Initiator costs are also normally relatively low; so if improved blasting results can be insured by using additional initiators, the added expense could be considered insignificant, as compared to the benefits received.

As powder factor yields are reduced, costs will be increased proportionately. But irrespective of the actual powder factor value, blasts should always be designed to give the yield most suitable for maximum production at the least expense. In this respect, the percentage of usable material from a blast must also be given consideration. Well-blasted rock does not mean it must necessarily be pulverized. On the contrary, the required particle sizing and its uniformity must be such that maximum recovery is achieved. If, for example, 10 percent of the production is lost due to spoiling or waste, which in quarrying is quite common, the loss must be included in the final cost analysis. If recovery is reduced in order to increase rates of production, the value of the wasted material should logically be less than the savings accomplished from the lower operating costs for the material salvaged.

CONCLUSIONS

Effective blasting depends largely on a knowledge of how materials fracture, the particular characteristics of those materials, qualities of the various explosives that may be used, and recognition that the secret of efficient, economical, and safe results lies essentially in the suitable placement of charges where they will do the most good. Since explosives are merely very powerful tools for performing work, they should always be used accordingly.

As has been shown by these discussions, there are no easy, simple methods for solving blasting problems. The mechanisms and factors involved are too complex and numerous to permit clear-cut solutions. Each situation must be handled according to its own requirements, with the prudent use of one's best judgment However, with a reasonable amount of study and understanding of operating conditions, blasters can evaluate results and make adjustments toward improvements by using certain basic standards. It has been the purpose of this article, therefore, to outline those standards and explain how they can be adjusted to apply to on-the-job conditions. But it must be realized that there can be no substitute for initial tests to ascertain what may be expected.

The burden dimension is the most critical of the important factors in blasting. Its value must suit the characteristics of the material being blasted and the properties of the explosives, and it must produce the desired degree of fragmentation and displacement. All other blasting standards are controlled by the burden value, and they should be designed on that basis. It should be, therefore, of primary concern to all blasters first to establish the best burden for their particular needs.

It has been shown that the powder factor as such has little meaning except as a relative basis for cost comparisons. For many years it has been used all too frequently, and unfortunately, as a means of judging blast efficiency. But under no circumstances can it be used as a reliable index for judging what one can expect in rock breakage or control of throw. Its value in costing is even questionable under many conditions.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

EJERCICIO PARA DISEÑO DE VOLADURAS DE BANQUEO

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

SINGLE BLASTHOLE DESIGN PROBLEM

A deposit is quarried in 30-ft high benches for crushed stone. The rock is quite massive and has the following properties:

$SG_r = 2.9$, $v_p = 17{,}000$ fps, $\mu = 0.25$, $S_f = 0.7$,

$\gamma = 45$ deg, $\sigma_c = 25{,}000$ psi, and $\sigma_t = 1750$ psi.

Blasted rock is loaded by a 5 cy front-end loader. The blastholes are drilled vertically and bulk loaded ($D_e = D_h$) with an explosive having an SG = 117, $D_c = 1$ in., and confined velocities of 12,500 fps at 3 in. and 15,000 fps at 5 in. and larger charge diameters. The relationship between v_e and D_e in the 1 to 5 in. range can be assumed to be in the form of

$$y = \frac{cx}{a + bx} .$$

Drainage at the operation is such that blastholes generally are always dry, and there is no free parting in the rock available that can serve as a floor. For estimating purposes the average blast area A of material cratered by a single blasthole would be equal to $1.4B^2$.

A. Considering the foregoing information, find the following properties for the intact rock:

(1) τ_s, and (2) E_r.

B. For charge diameters D_e of (a) 2 in., and (b) 4 in., determine each of the following estimates:

(1) v_e, (2) P_d, (3) P_e, (4) B, (5) T, (6) J,

(7) E, (8) W, (9) t_f, and (10) t_1.

C. At the given bench height L determine the respective D_e values that define each of the following conditions:

(1) The B' that insures all of the explosive column will react before any cracks will have propagated to any open face when using a single primer located at (a) Floor level, and at (b) The Center of the charge column.

(2) The B" at which overbreak quite likely may begin to occur when the primer is place at floor level.

SOLUTION (CONT.)

B (1) First determine relationship of V_o with D_o from $y = \frac{cx}{a+bx}$ where $y = V_o$ and $x = D_o - D_c$

Then $$V_o = \frac{c(D_o - D_c)}{a + b(D_o - D_c)}$$

It is given that $D_c = 1$ in., $V_o = 12{,}500$ fps @ $D_o = 3$ in., and $V_o = 15{,}000$ fps @ $D_o = 5$ in.

Then @ $D_o = 3$ in., $$12{,}500 = \frac{c(3-1)}{a + b(3-1)} = \frac{2c}{a+2b}$$

Assume $c = 5000$,

Then $$a + 2b = \frac{2(5000)}{12{,}500} = \frac{4}{5} = 0.80 \quad (I)$$

For $D_o = 5$ in., $$15{,}000 = \frac{c(5-1)}{a + b(5-1)} = \frac{4c}{a+4b}$$

or $$a + 4b = \frac{4(5000)}{15{,}000} = \frac{4}{3} = 1.33 \quad (II)$$

Regrouping
$$a + 2b = 0.80 \quad (I)$$
$$a + 4b = 1.33 \quad (II)$$

Subtracting I from II,

$$2b = 0.53$$

or $$b \cong 0.27$$

Substituting value of b in I and II,

$$a + 2(0.27) = 0.80 \quad (I)$$

or $$a = 0.26$$

and $$a + 4(0.27) = 1.33 \quad (II)$$

or $$a = 0.25$$

For all practical purposes, then, $a = 0.26$ and $b = 0.27$ when $c = 5000$.

SOLUTION (CONT.)

B (2) (CONT.)

(a) $D_e = 2$ in.

$$P_d = P_{d_{max}}\left(\frac{9450}{15{,}000}\right)^2 = 835{,}000\,(0.397)$$

or $$P_d = 331{,}000 \text{ psi} \quad \Longleftarrow$$

(b) $D_e = 4$ in.

$$P_d = P_{d_{max}}\left(\frac{14{,}000}{15{,}000}\right)^2 = 835{,}000\,(0.87)$$

or $$P_d = 730{,}000 \text{ psi} \quad \Longleftarrow$$

B (3) From Eq. 4(b),

$$P_e = P_{d_{max}}/2$$

Thus, (a) $D_e = 2$ in., and (b) $D_e = 4$ in.,

$$P_e = 835{,}000/2 = 417{,}500 \text{ psi} \quad \Longleftarrow$$

B (4) From Eq. 35,

$$K_B = 30\left(\frac{160}{d_r}\right)^{\frac{1}{3}}\left(\frac{SG_e}{1.3}\right)^{\frac{1}{3}}\left(\frac{V_e}{12{,}000}\right)^{\frac{2}{3}}$$

From Eq. 7,

$$d_r = 62.4\,(SG_r) = 62.4\,(2.9) = 181 \text{ pcf}$$

Substituting for values of d_r and SG_e, then

$$K_B = 30\left(\frac{160}{181}\right)^{\frac{1}{3}}\left(\frac{1.2}{1.3}\right)^{\frac{1}{3}}\left(\frac{V_e}{12{,}000}\right)^{\frac{2}{3}}$$

$$= 30\,(0.96)(0.97)\left(\frac{V_e}{12{,}000}\right)^{\frac{2}{3}}$$

or $$K_B = 28\left(\frac{V_e}{12{,}000}\right)^{\frac{2}{3}}$$

But from Eq. 34,

$$B = \frac{K_B D_e}{12} = \frac{28}{12}\left(\frac{V_e}{12{,}000}\right)^{\frac{2}{3}} D_e$$

SOLUTION (CONT.)

B (7) From Eq. 3 where $d_c = 0.34 D_c^2 (SG_c)$ and combining Eq's. 29 through 33, we obtain when $L = 30$ ft,

$$E = d_c (PC) = 0.34 D_c^2 (SG_c)(L + J - T)$$

$$= 0.34\ D_c^2 (1.2)(30 - B/3) = 0.41 D_c^2 (30 - B/3),\ \text{lb.}$$

(a) $D_c = 2$ in.

$$E = 0.41 (2)^2 (30 - 1.33) = (1.64)(28.7) = 47\ \text{lb} \Leftarrow$$

(b) $D_c = 4$ in.

$$E = 0.41 (4)^2 (30 - 3.43) = (6.56)(26.6) = 174\ \text{lb} \Leftarrow$$

B (8) If $A = 1.4 B^2$ and $W = \dfrac{A L d_r}{2000} = \dfrac{1.4 B^2 (30)(181)}{2000}$,

then $W = 3.8\ B^2$

(a) $D_c = 2$ in, $W = 3.8 (4)^2 = 61$ tons $\Leftarrow$

(b) $D_c = 4$ in, $W = 3.8 (10.3)^2 = 402$ tons $\Leftarrow$

B (9) From Eq. 19, $V_f = V_p/3 = 17{,}000/3 = 5670$ fps

If $t_f = B/V_f$, sec, (a) $D_c = 2$ in., $t_f = 4/5670 = 0.6$ ms $\Leftarrow$

(b) $D_c = 4$ in., $t_f = 10.3/5670 = 1.5$ ms $\Leftarrow$

B (10) If $t_i = 0.001\ B$, sec.

(a) $D_c = 2$ in., $t_i = 0.001 (4) = 4.0$ ms $\Leftarrow$

(b) $D_c = 4$ in, $t_i = 0.001 (10.3) = 10.3$ ms $\Leftarrow$

SOLUTION (CONT.)

B(1) (CONT.)

(b) Primer at Center of Charge Column.

$$\text{From Eq. 38(b),}\quad B' = \frac{3L}{18K_v+1}$$

1. At D_e = 5 in.,

$$B' = 3(30)/(18 \times 0.88+1) = 90/16.8 = 5.4 \text{ ft.}$$

From Table B = 13.5 ft.

B is much greater than B' indicating diameter can be much smaller, i.e., $B > B'$ or $13.5 > 5.4$.

2. At D_e = 3 in.,

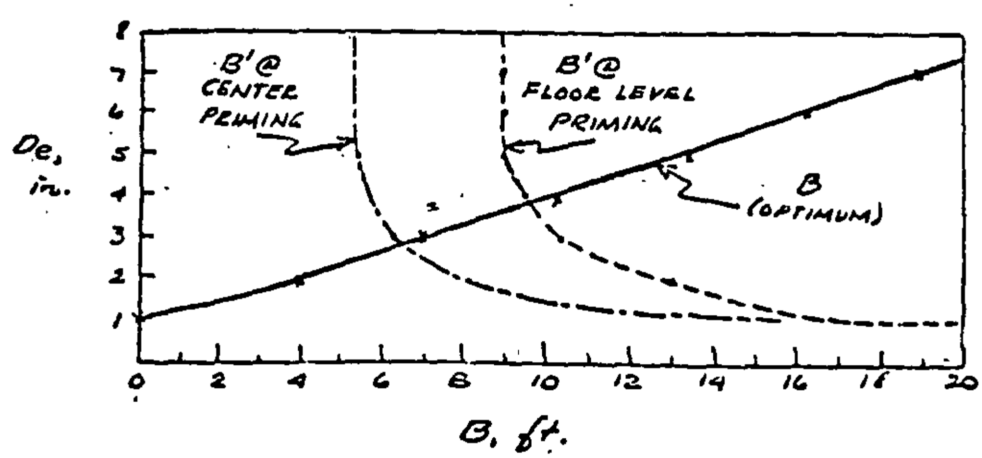
$$B' = 3(30)/(18 \times 0.74+1) = 90/14.3 = 6.3 \text{ ft.}$$

From Table B = 7 ft.

Values of B and B' are approximately equal with $B > B'$ a small amount, which is desirable.

Therefore, use D_e = 3 in. ⟸

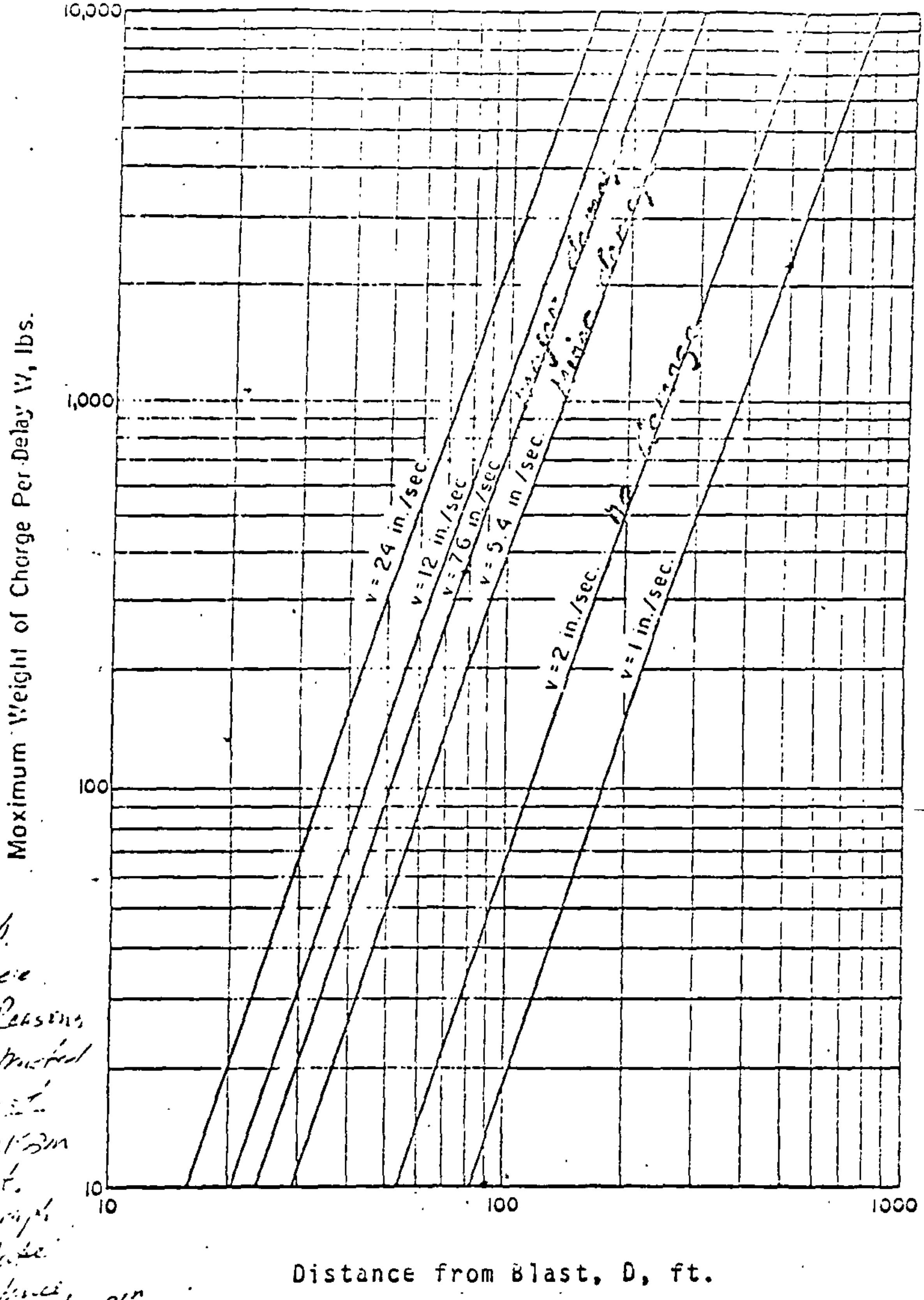
NOTE The previous solutions can be solved quite simply by plotting values as shown below:



C (2)

From Eq. 39, $B'' = 0.62L = 0.62(30) = 18.6$ ft ⟸

This burden would be for $D_e = \frac{B''}{2.7} = \frac{18.6}{2.7} = 6.9$ in.



...mograph
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...r correlate
...caled Distance
...which is based upon

minor damage: fine plaster cracks, opening of old cracks

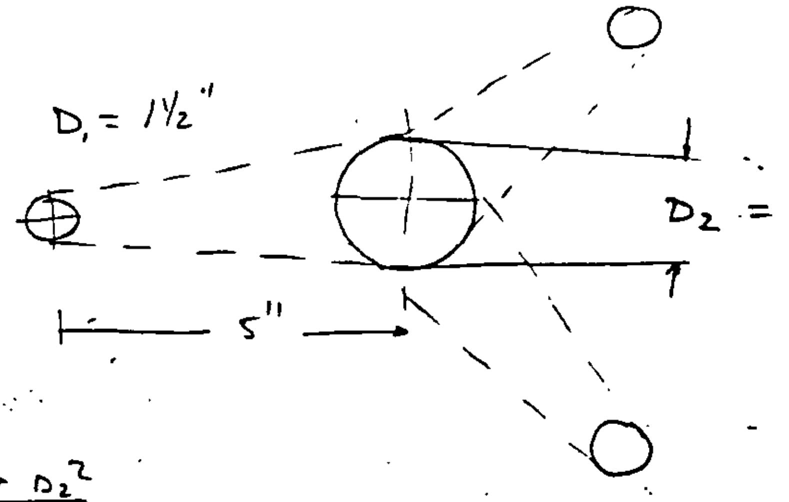
major damage: serious new cracking, plaster falls

TABLE I. Strength Data For Some Competent Rocks (13).

Rock Type	Compressive Strength psi x 10^3	Elasticity Modulus (Compression) psi x 10^6	Tensile Strength psi	C psi	ϕ deg.	Equation of Mohr's Envelope (ref. Fig. 3)
Chert	29.3	8.15	820	2550	71.5	Y = 2550 + 3.0 x
Coal	6.2	–	–	1600	38.5	Y = 1600 + 0.8 x
Granite	28.0	3.17	410	1720	76.5	Y = 1720 + 4.2 x
Green Stone	29.1	8.82	380	1700	77.5	Y = 1700 + 4.5 x
Greywacke	7.9	1.80	700	1200	59.5	Y = 1200 + 1.7 x
Limestone	21.3	9.50	350	1320	75.5	Y = 1320 + 3.9 x
Marble	30.8	7.15	863	2650	71.0	Y = 2650 + 2.9 x
Salt Rock	2.2	1.35	85	210	72.5	Y = 210 + 3.2 x
Sand Stone	14.8	2.00	230	900	76.0	Y = 900 + 4.0 x
Shale	5.2	1.09	1538	1420	31.0	Y = 1420 + 0.6 x
Silt-Stone	5.0	12.60	440	750	59.5	Y = 750 + 1.7 x

C – Cohesion
ϕ – Angle of Internal Friction
Y – Shear Stress, τ_s
x – Normal Stress, σ_n

$$Swell = \frac{\frac{\pi D_2^2}{4}}{3\left[\frac{(1.5+D_2)5}{2} - \frac{\pi D_2^2}{4\times 2} - \frac{\pi (1.5)^2}{4\times 2}\right]}$$



$$0.60 = \frac{\frac{\pi D_2^2}{4}}{\left(\frac{7.5 + 5 D_2}{2} - 0.393 D_2^2 - 0.884\right)3}$$

$$0.60 = \frac{0.786 D_2^2}{(11.25 + 7.5 D_2) - 1.179 D_2^2 - 2.65}$$

$$6.75 + 4.5 D_2 - 0.71 D_2^2 - 1.59 = .786 D_2^2$$

$$-1.5 D_2^2 + 4.5 D_2 + 5.16 = 0$$

$$D_2 = \frac{-4.5 \pm \sqrt{4.5^2 - 4(-1.5)(5.16)}}{2(-1.5)}$$

$$D_2 = \frac{-4.5 \pm 7.16}{-3.0} = \frac{2.66}{3.0}$$

$$D_2 = \frac{-4.5 - 7.16}{-3.0} = 3.89'' \approx 3\ {}^{14}/_{16}''$$

$$D_2 = \frac{-b \pm \sqrt{b^2 - 4ac}}{2a}$$

Table IX.2. Static Elastic Moduli, Poisson Ratios, Velocities c_ℓ and c_t and Strengths of Some Solids.

Solid	Reference	(Average) Density (g/cc)	Modulus (dynes/cm$^2 \cdot 10^{-10}$) Bulk K	Shear G	Lame λ	Young's E	Poisson Ratio ν	c_ℓ (km/sec)	c_t (km/sec)	Average Static Strength (kb) C_o	S_o	T_o
Aluminum	a	2.7	78	25	61	69	0.36	6.4			1.2	2.0
Copper	a	8.85	102	46	131	124	0.37	5.0			2.0	3.0
Steel	a	7.8	168	80	120	200	0.3	5.9			10 (5.4)	10
Lead	a	11.35	37	5	33	16	0.43	1.96			0.1 (0.15)	0.1
Plexiglas	a	1.18	6.6	1.4	5.6	4	0.4	2.68			(0.8)	
Glass	b	2.5	58	21.5	40	10	0.3	6.1			(0.5)	
Pyrex	a	2.32	39.7	25	23	62	0.24	5.64				
Marble	c,d	2.7 - 2.9		20-35		60 ± 40		5.3-6.4	3.3	0.6-2.5		[0.02, 0.06]
Limestone	c,d	2.3 - 2.5		35-40		40 ± 30	0.24-0.32	2.9-5.0	2.0-3.0	0.3-2.5	0.15 (0.2-0.5)	[0.03, 0.08]
Granite	c,e	2.6 - 2.7		10-31		70 ± 20	0.2 - 0.33	5.6		1.5-2.9	0.22 (0.3-0.6)	0.07
Sandstone	c	2 1 - 2.6				25 ± 20		3.05		0.3-2.4	0.17 (0.2)	0.09
Shale	c	2.7				70 - 100	0.26			0.7-2.3	0.08	
Greenstone	c	3 02		35		80	0.15	5.2		3.0		
Basalt	c	3.0		25		85	0.33	6.6	3.0	0 8-3.6	0.3	0.15
Taconite	e	3.23-3.44				91 - 102	0.26-0.24	5.1-5.9	3.2-3.7	3.3-4.4		0.2-0.3
Quartzite	e	2.17		29		69	0.28-0.15	5.0	3.3	3.8	(0.6)	0.18
Chalk	f	2.0		3		5		2.3		0.13		

a. Gray, D. W., *Am. Inst. Phys. Handbook*, McGraw-Hill, N.Y. (1957)
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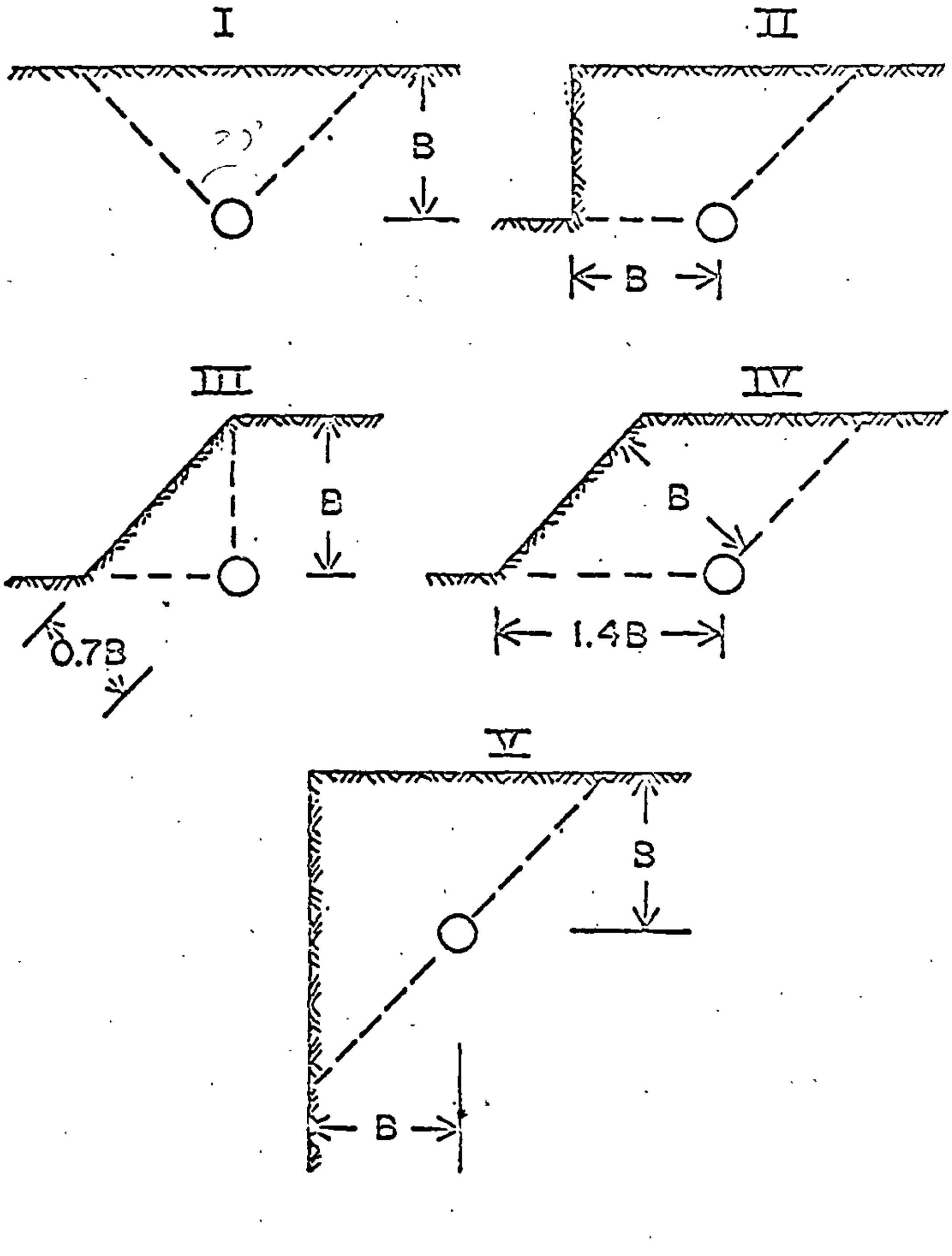
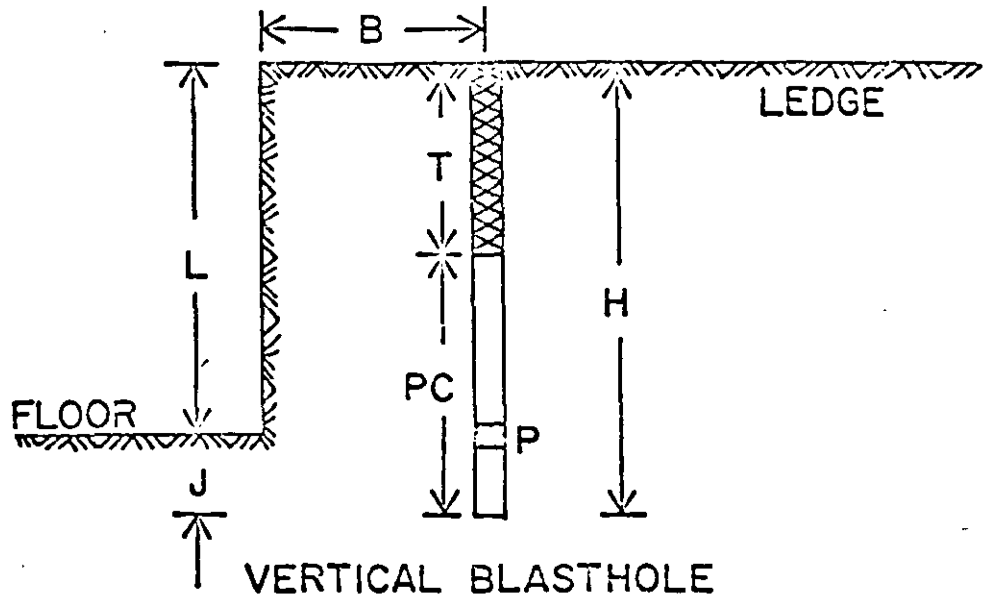
c. Obert, L. and W. I. Duvall, *Rock Mechanics*, , John Wiley & Co., N.Y. (1967)
d. Clark, G. B., *Min. Research*, Vol. 2, Pergamon Press, N.Y. (1962)
e. Paone, J., et al., *Drillability Studies*, Bur. Mines, RI6880 (1966)
f. Duvall, W. I., and T. C. Atchison, ibid., RI5356 (1957)

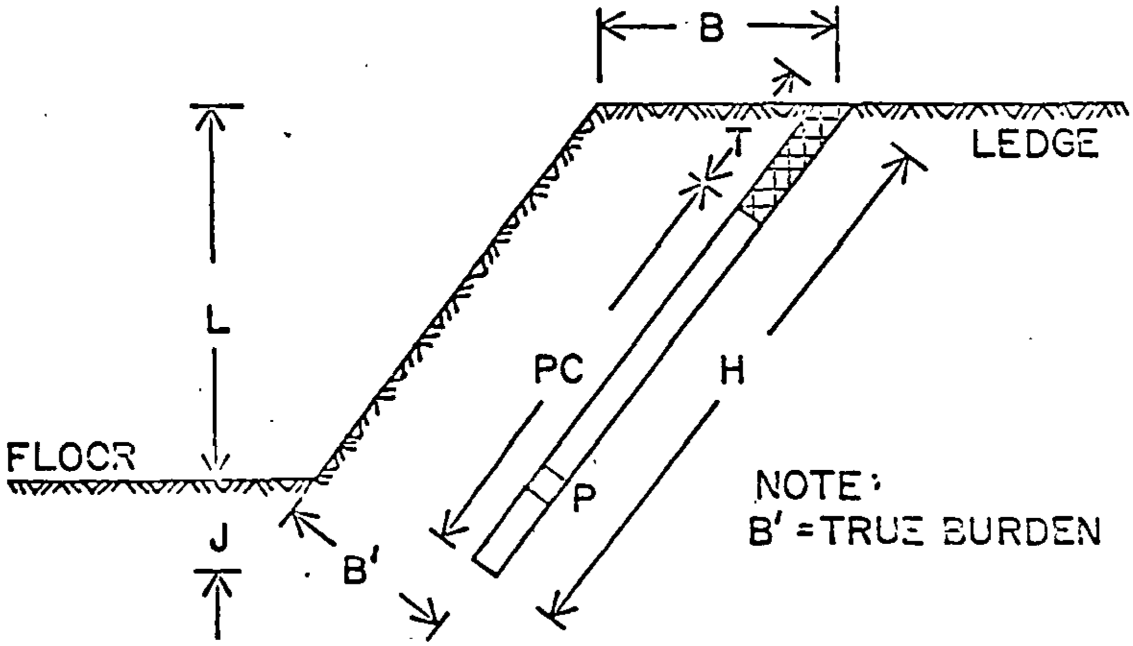
Values in ( ) for S_o were taken from reference 10 for zero confining pressure



BASIC CRATER FORMS IN PLANE OF CHARGE DIAMETER

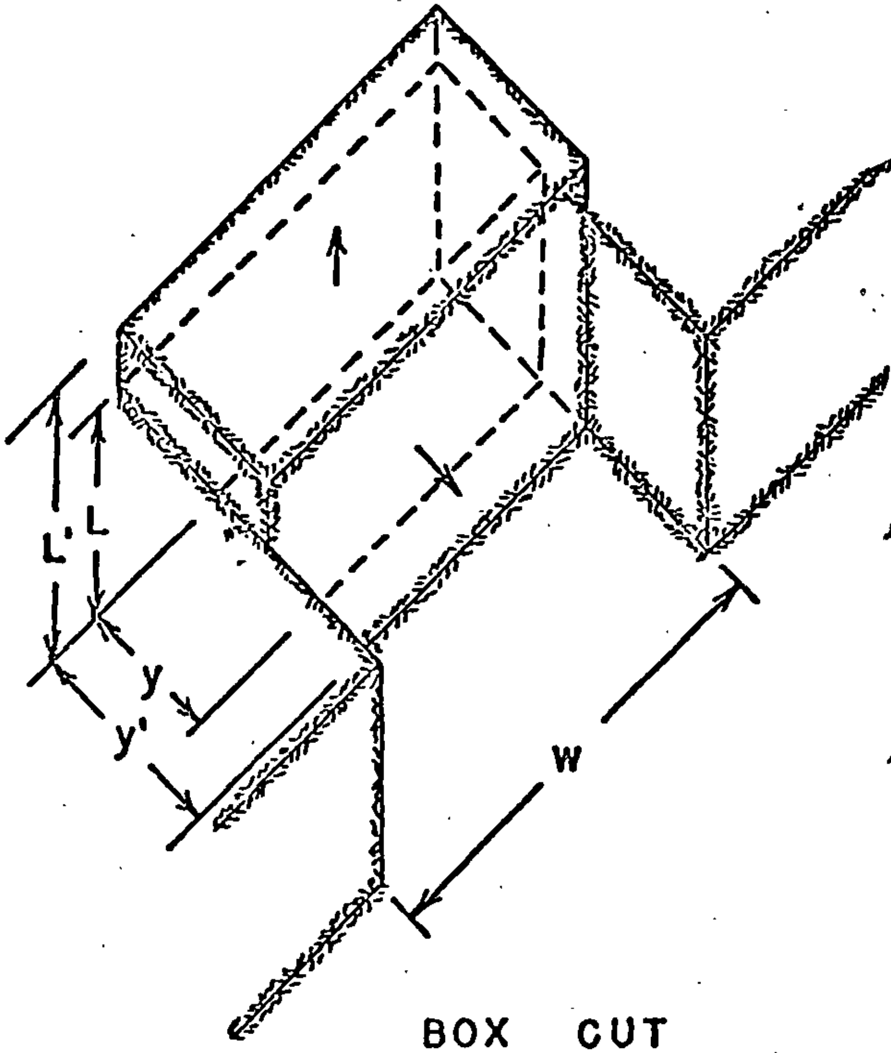


VERTICAL BLASTHOLE

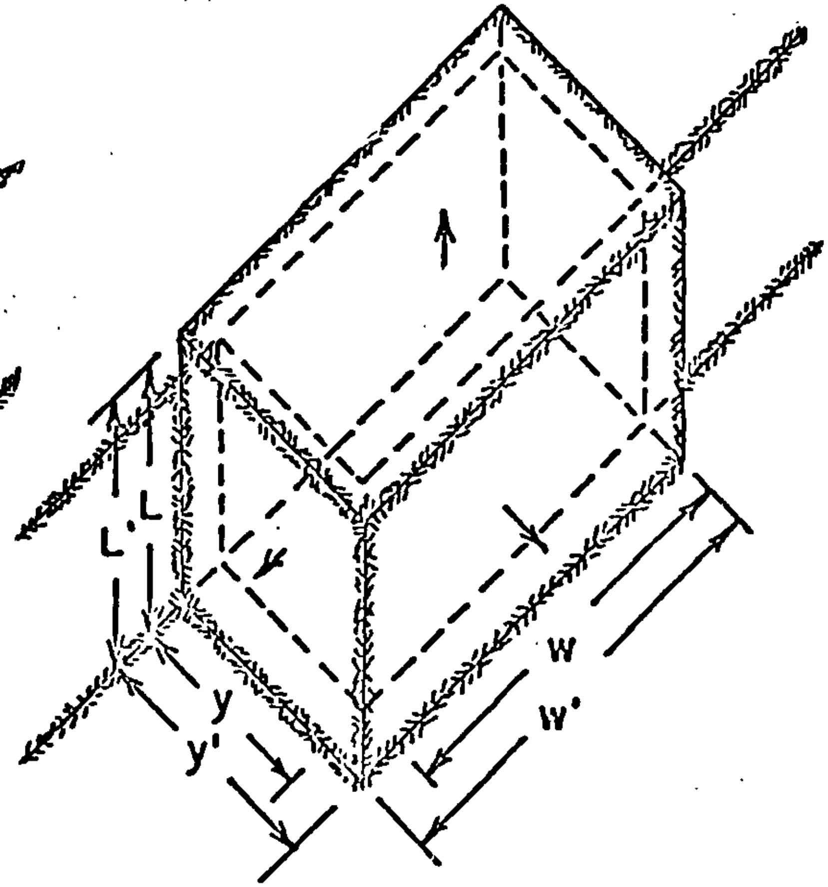


INCLINED BLASTHOLE

B = BURDEN, P = PRIMER, T = STEMMING,
L = BENCH HEIGHT, J = SUBDRILLING,
PC = CHARGE LENGTH, H = HOLE DEPTH



BOX CUT



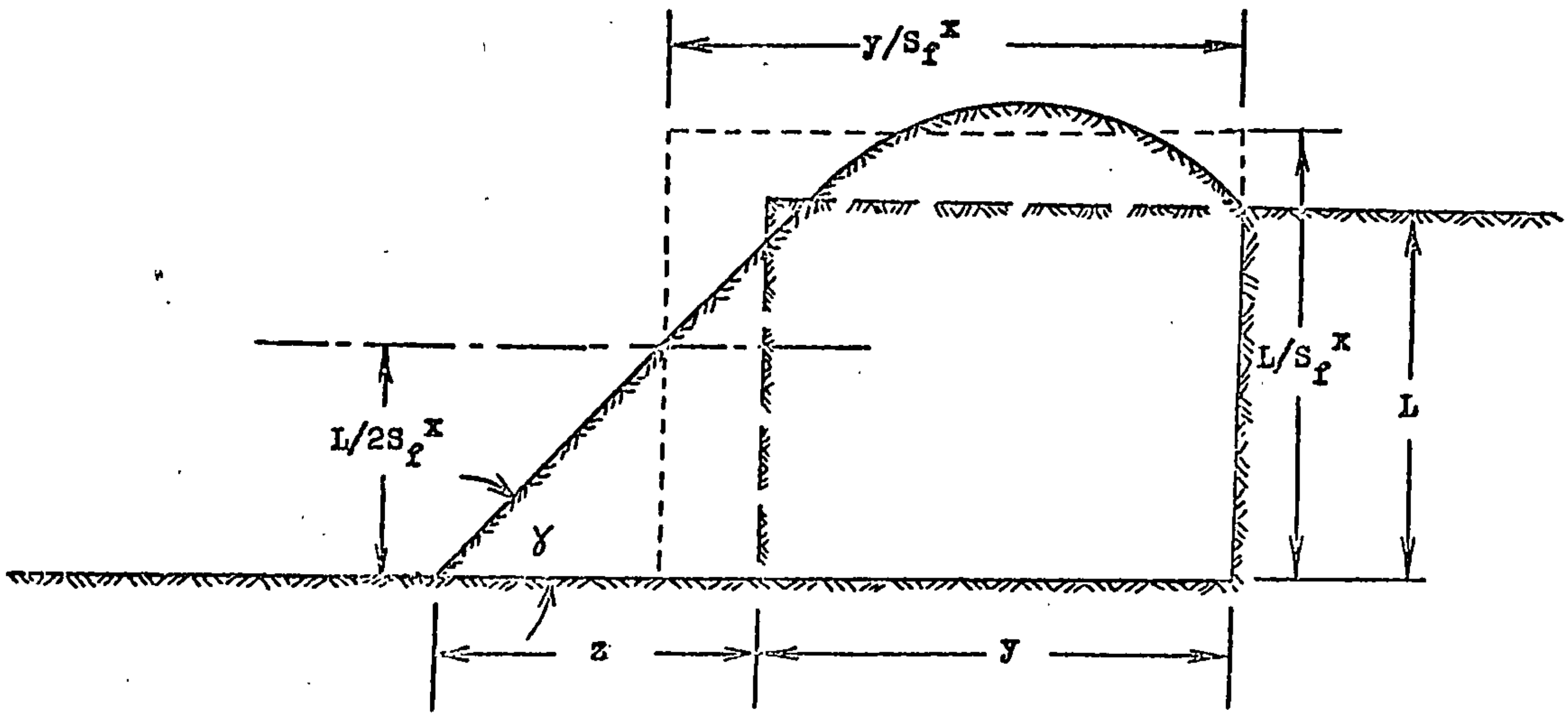
CORNER CUT

$$C_f = \frac{V_S}{V_B} = \frac{y\,W\,L}{y'\,W'\,L'}$$

$$(y'\,W'\,L')\,C_f = y\,W\,L$$

THE SWELL AND COLLAPSE OF BROKEN MATERIAL

WHEN NEITHER HEAPED OR THROWN DELIBERATELY FOR A BENCH BLAST

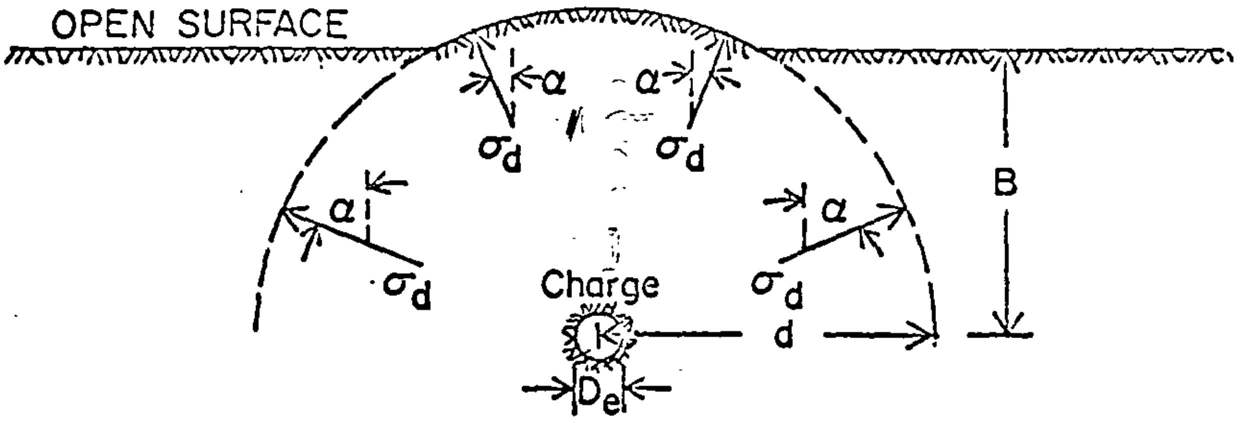


RLA

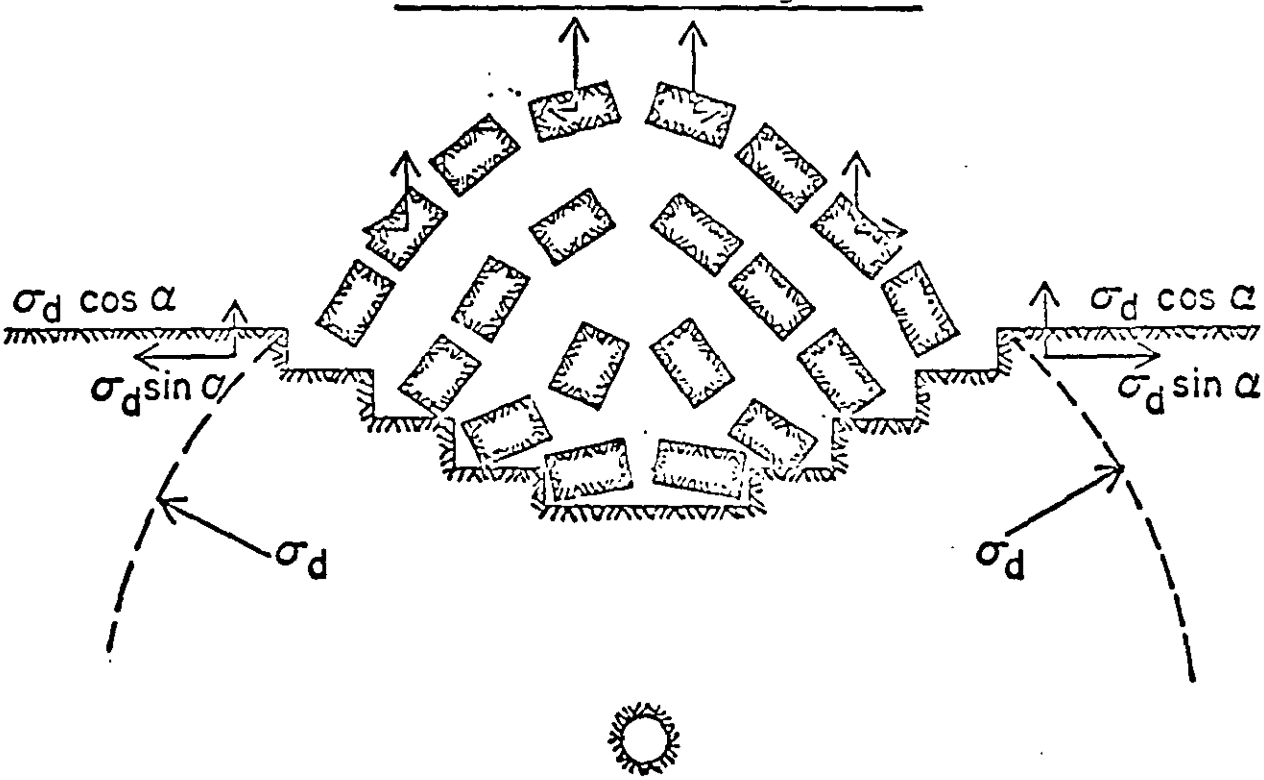
$$z = (1/2S_f^x)\left[L\cot\gamma + 2y(1 - S_f^x)\right]$$

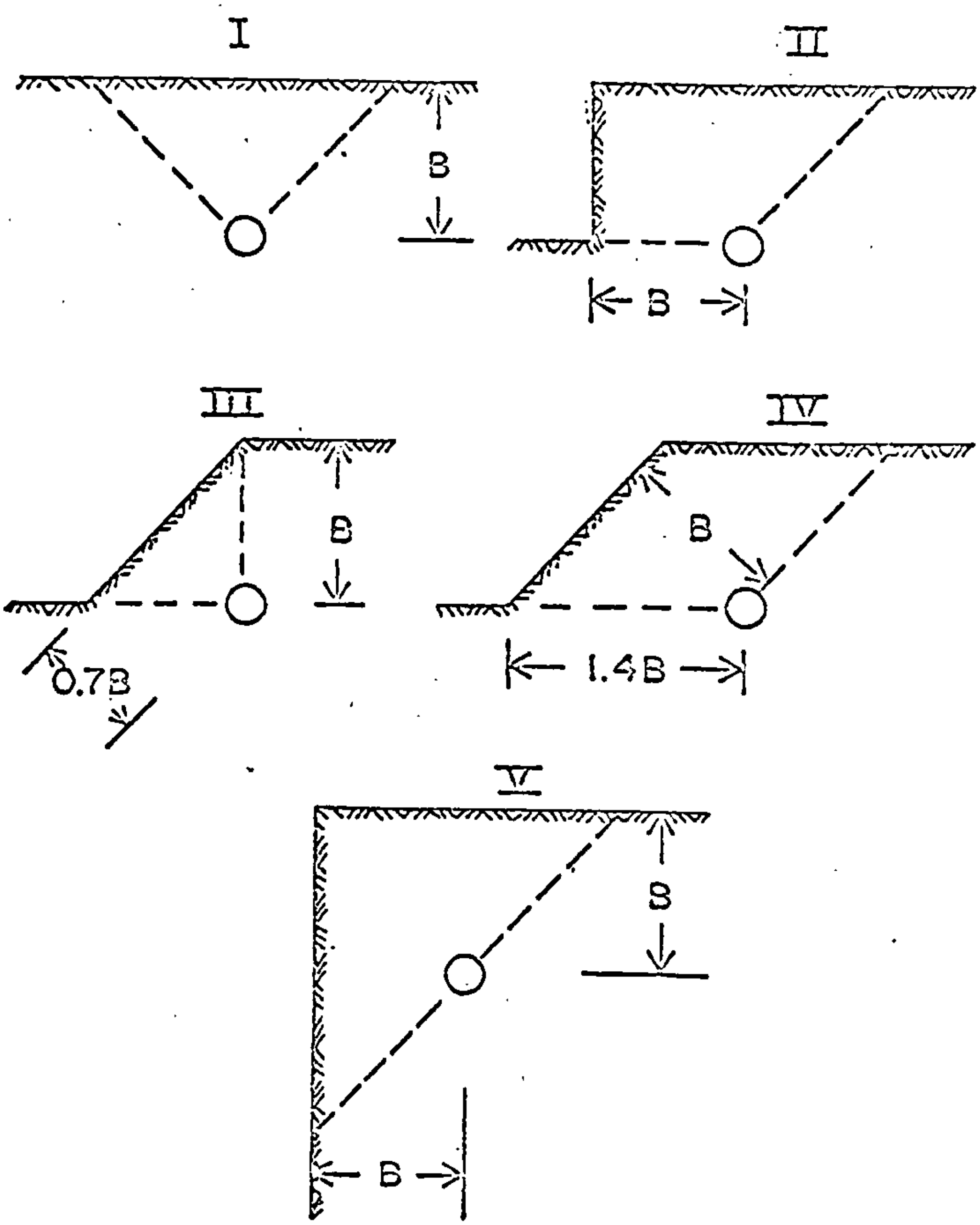
MECHANICS OF CRATER DEVELOPMENT

Formation of Compression Bulge



Intermediate Cratering Action

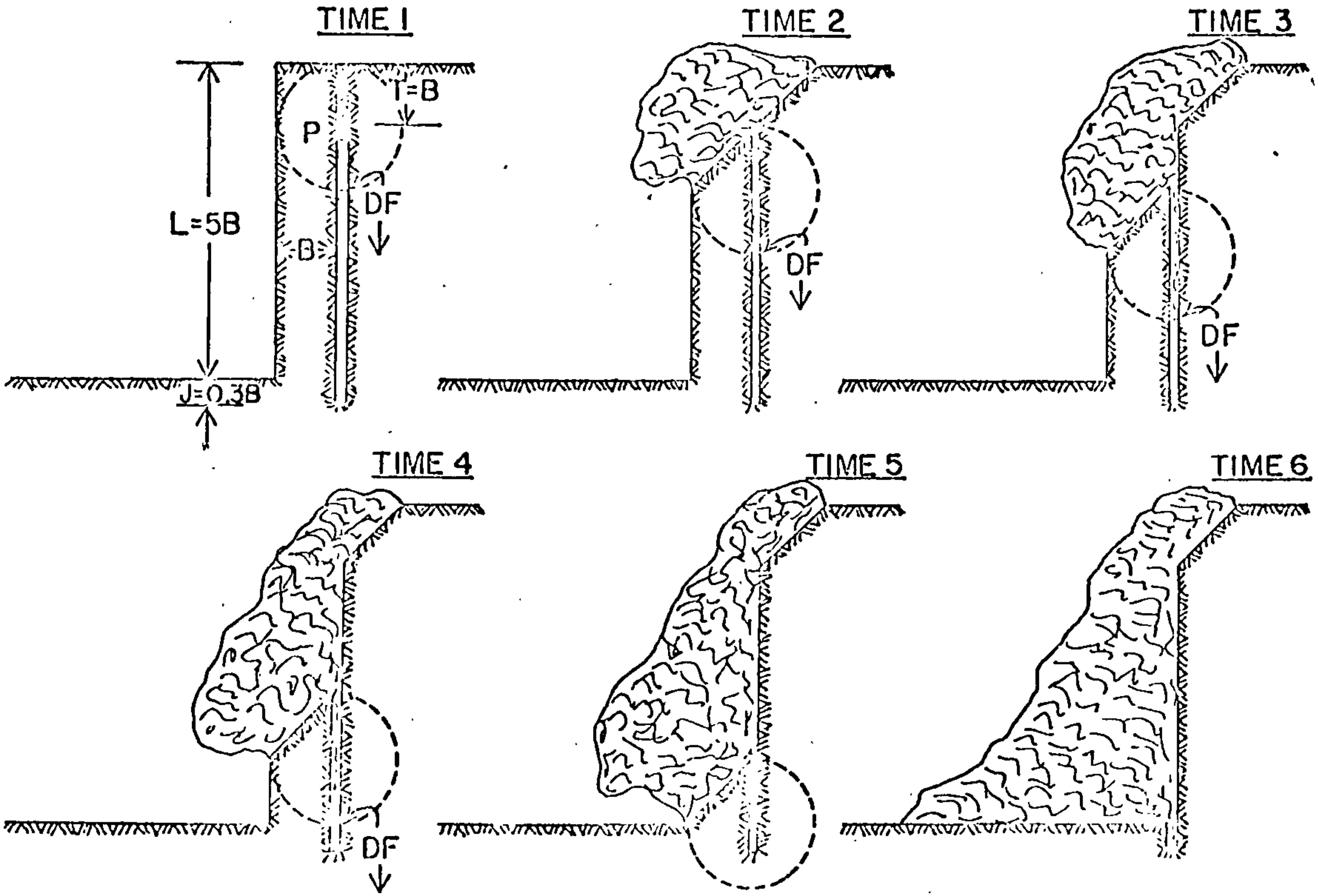




BASIC CRATER FORMS IN PLANE OF CHARGE DIAMETER

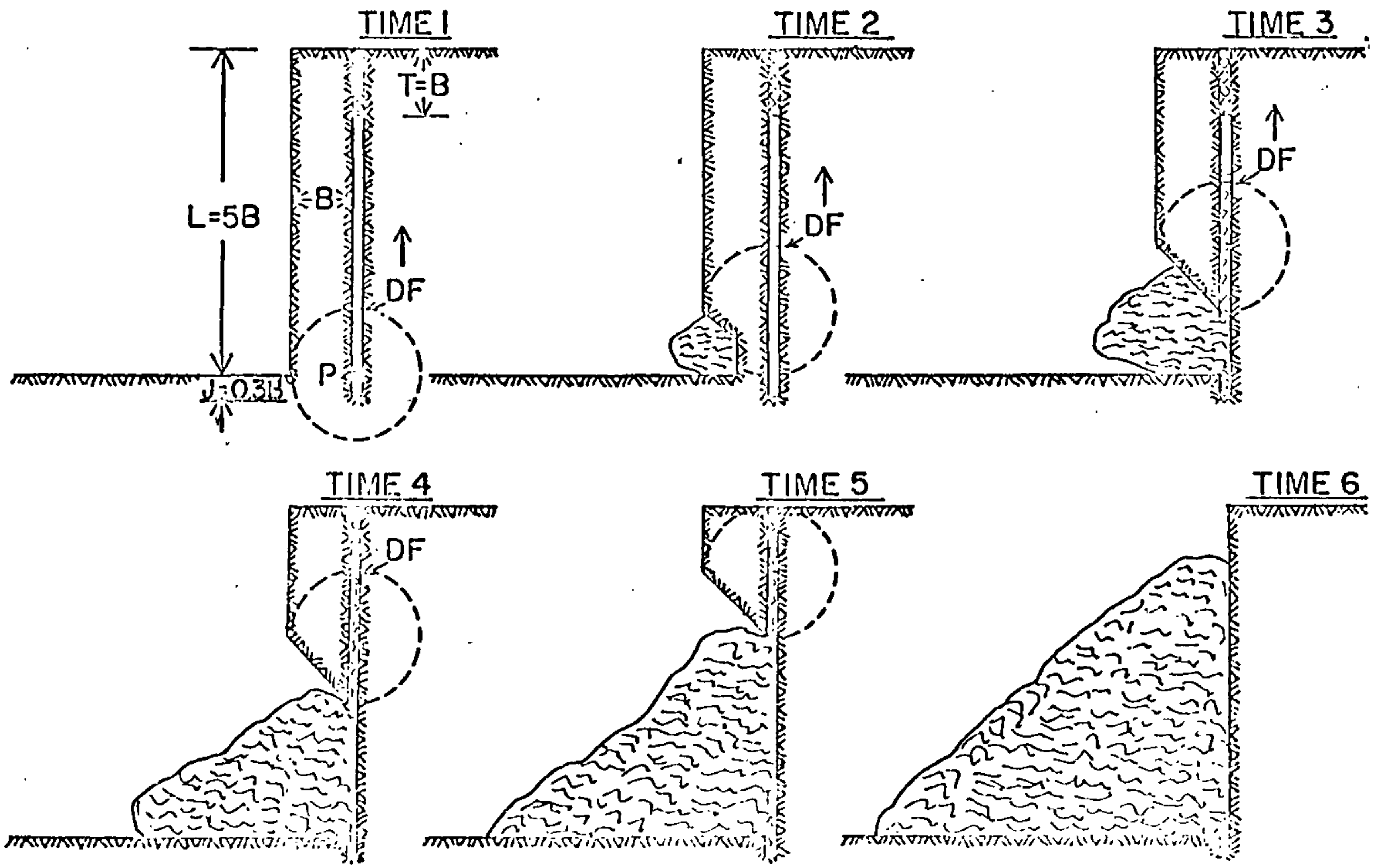
$V_e = V_p$

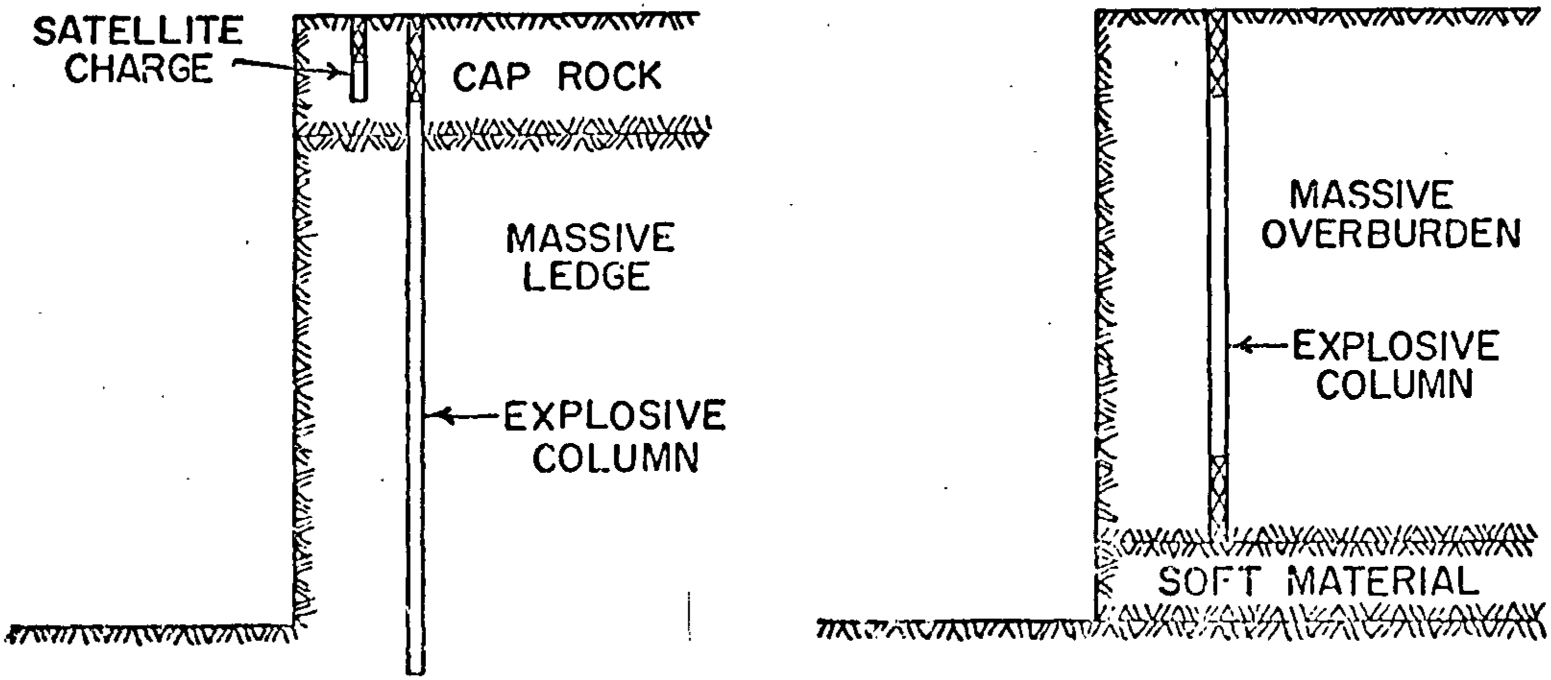
COLLAR PRIMING



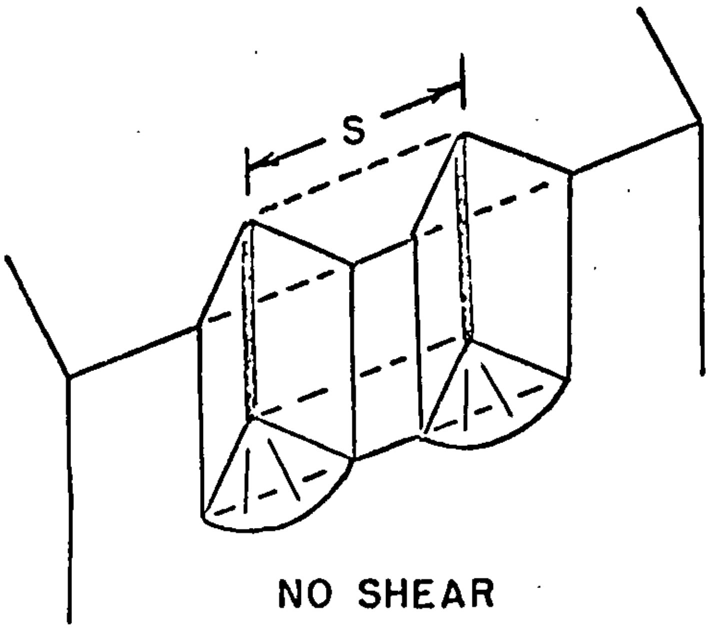
$V_e = V_p$

FLOOR PRIMING

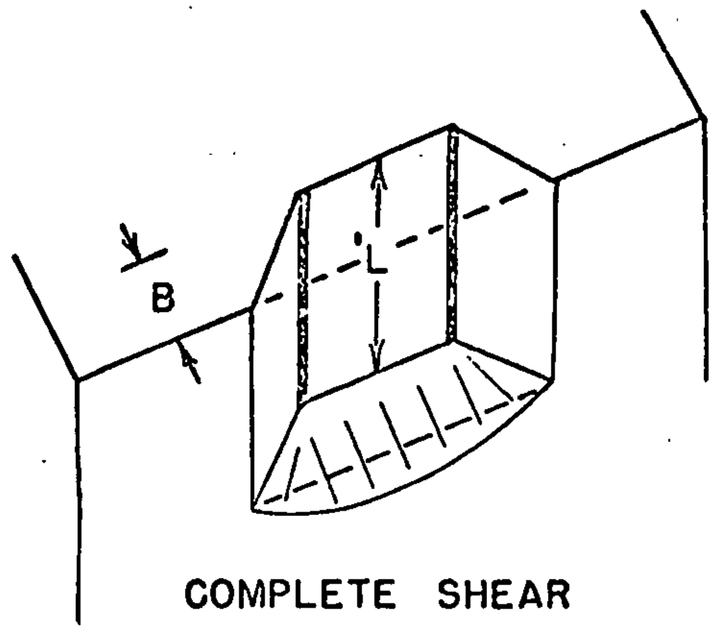




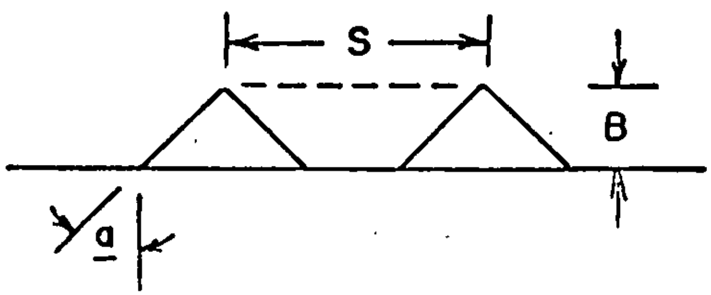
LOADING OF BLASTHOLES FOR SPECIAL END CONDITIONS



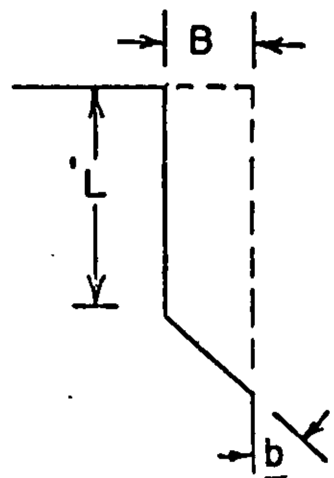
NO SHEAR



COMPLETE SHEAR



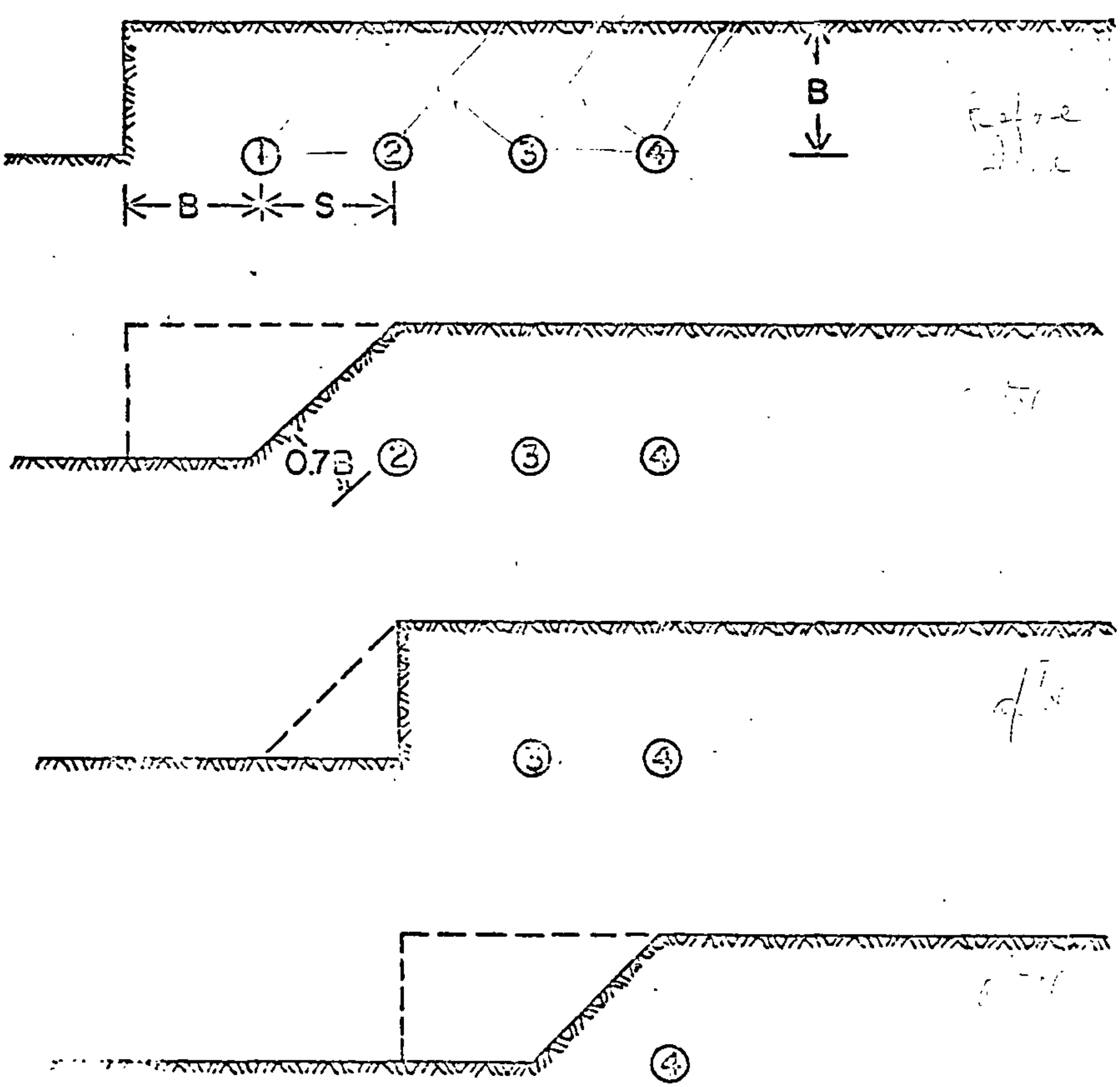
PLAN VIEW



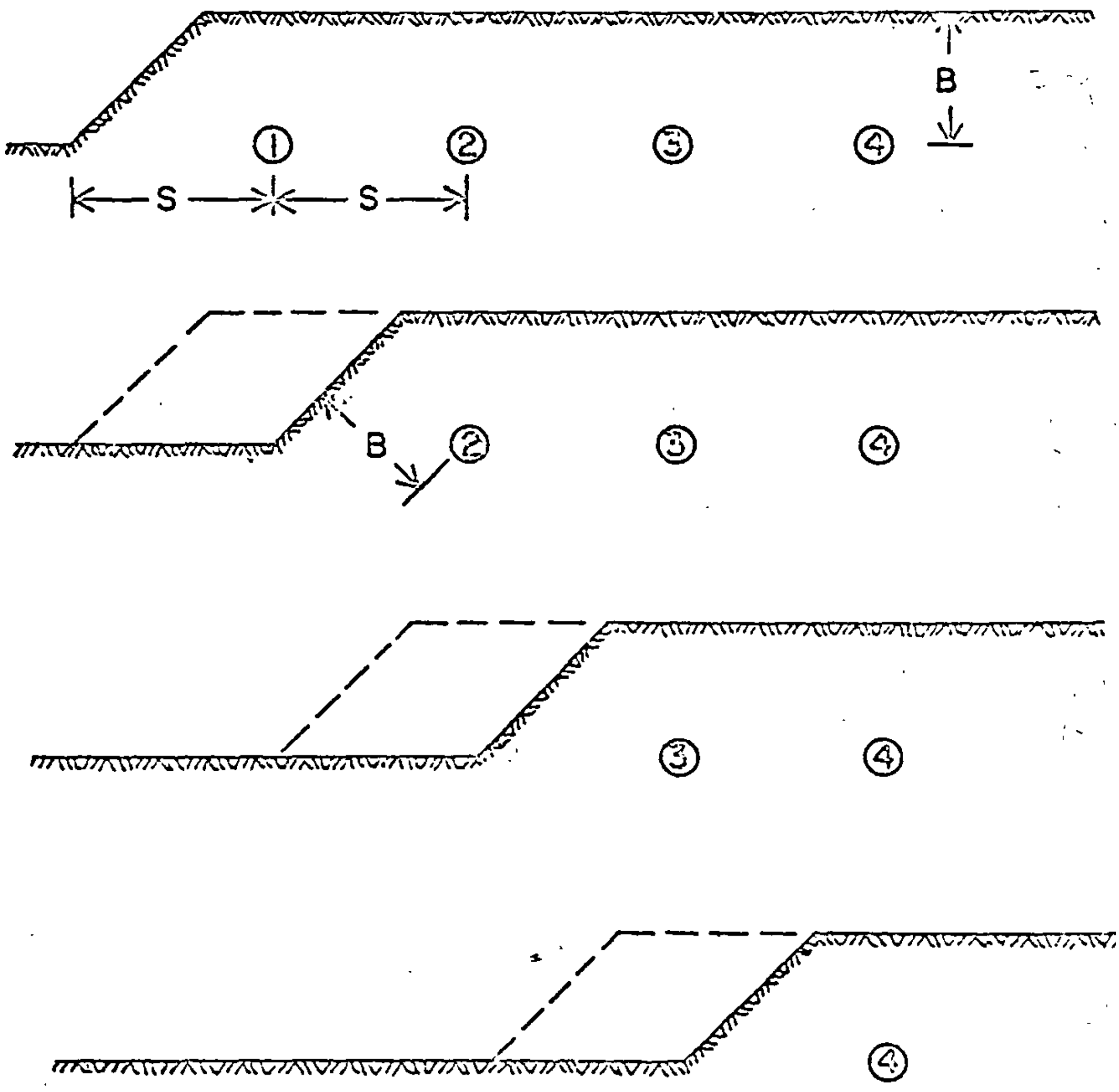
SECTION VIEW

RLA

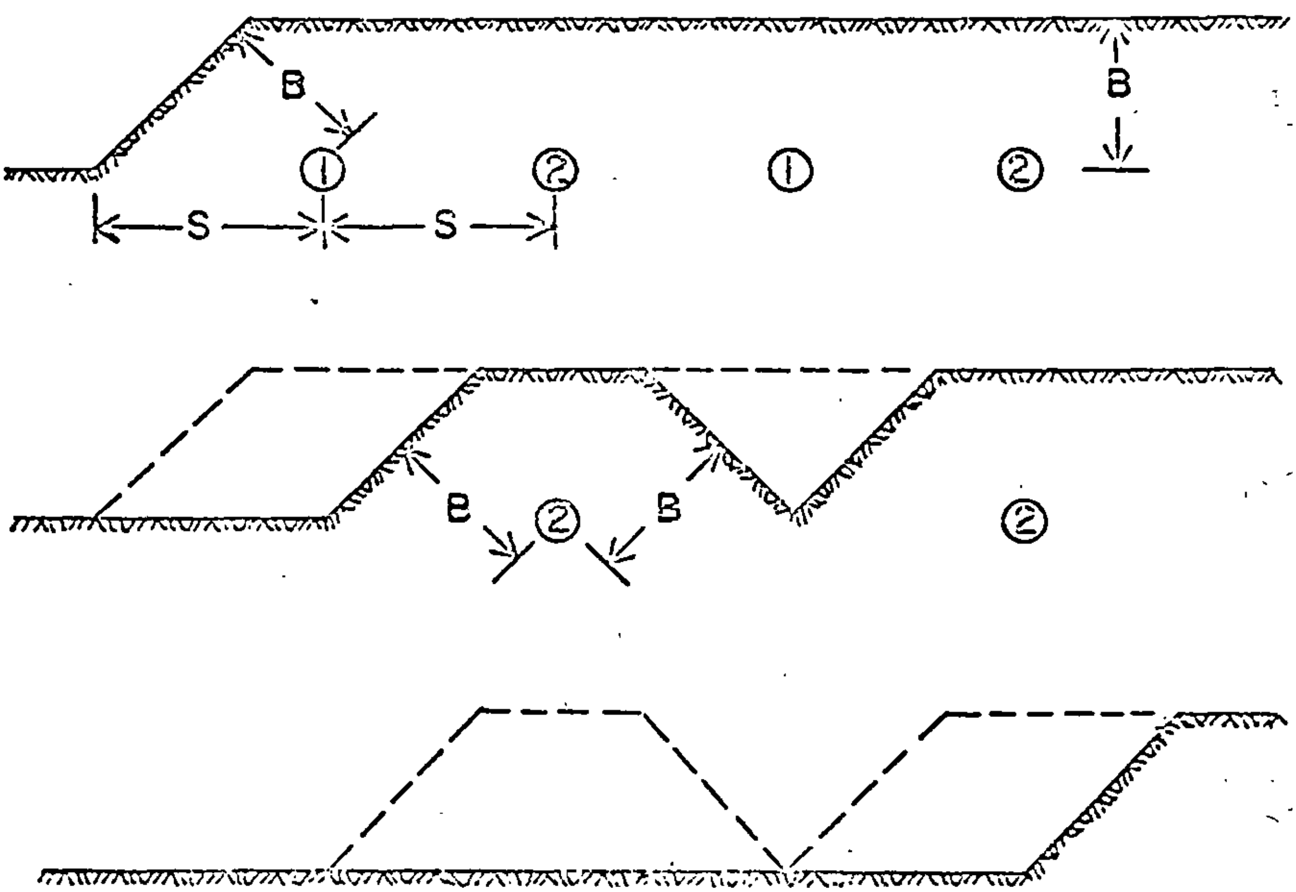
PROGRESSIVE TIMING, S=B



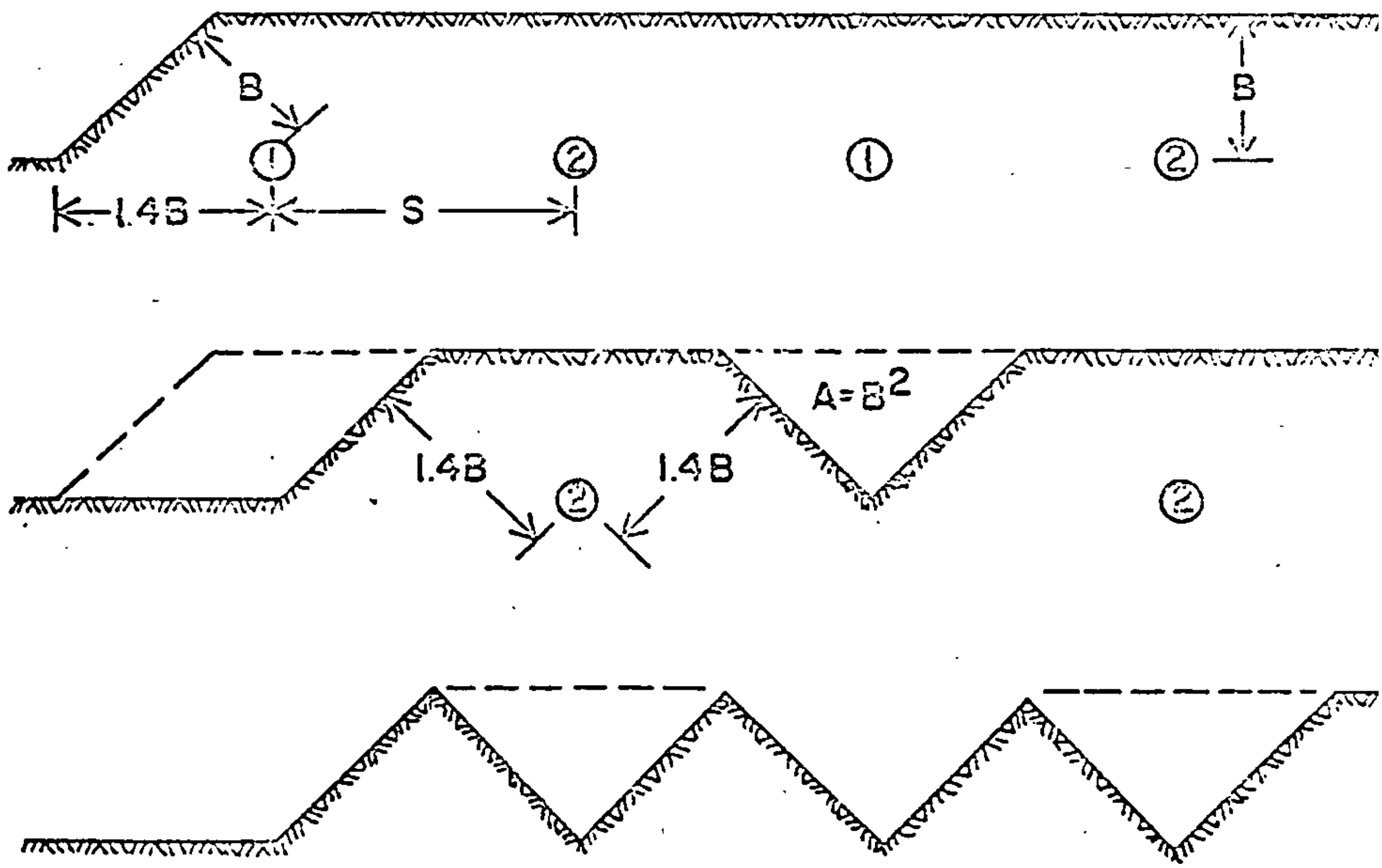
PROGRESSIVE TIMING, S=1.4B



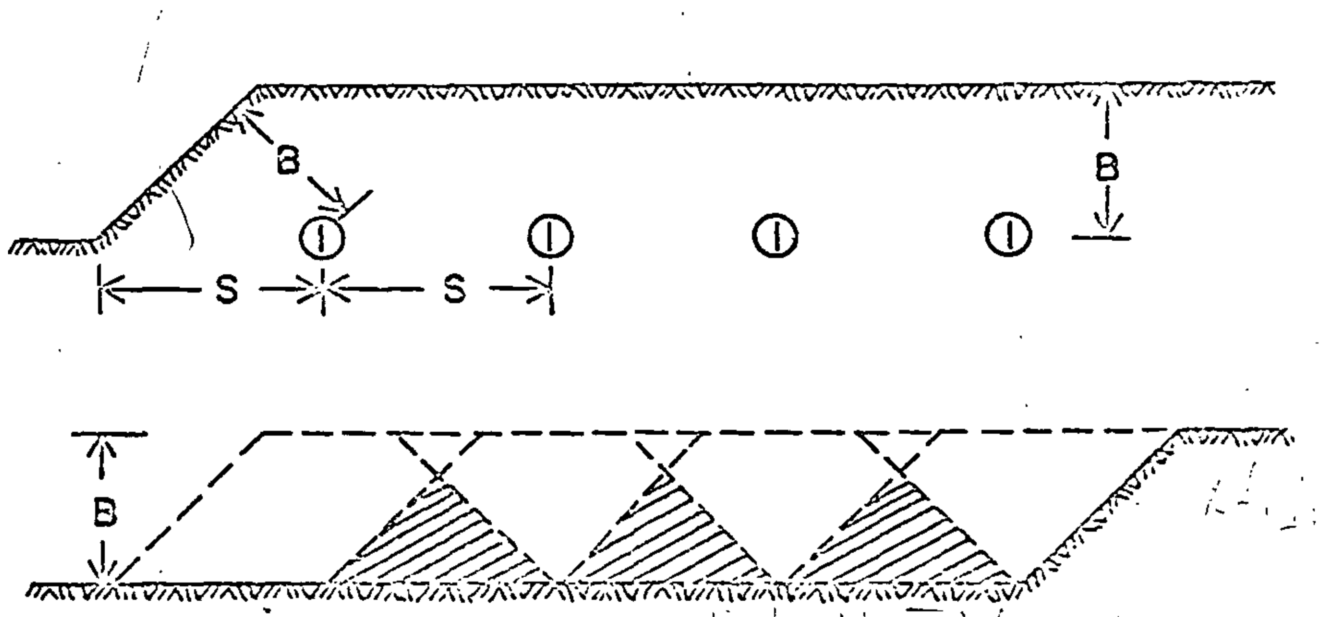
ALTERNATE TIMING, S=1.4B



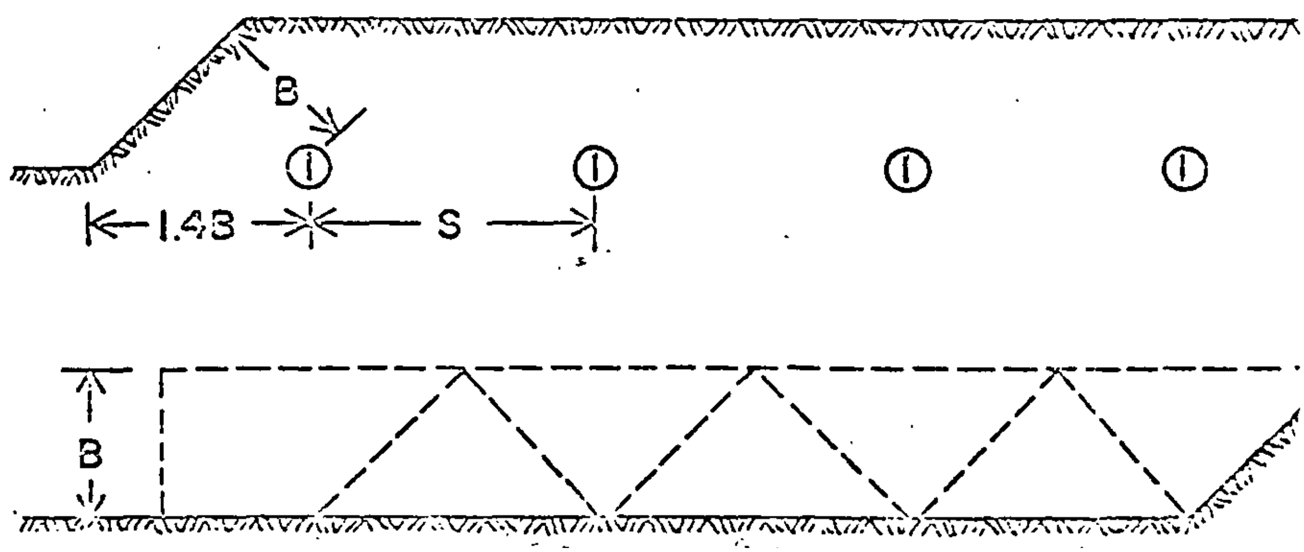
ALTERNATE TIMING, S=2B

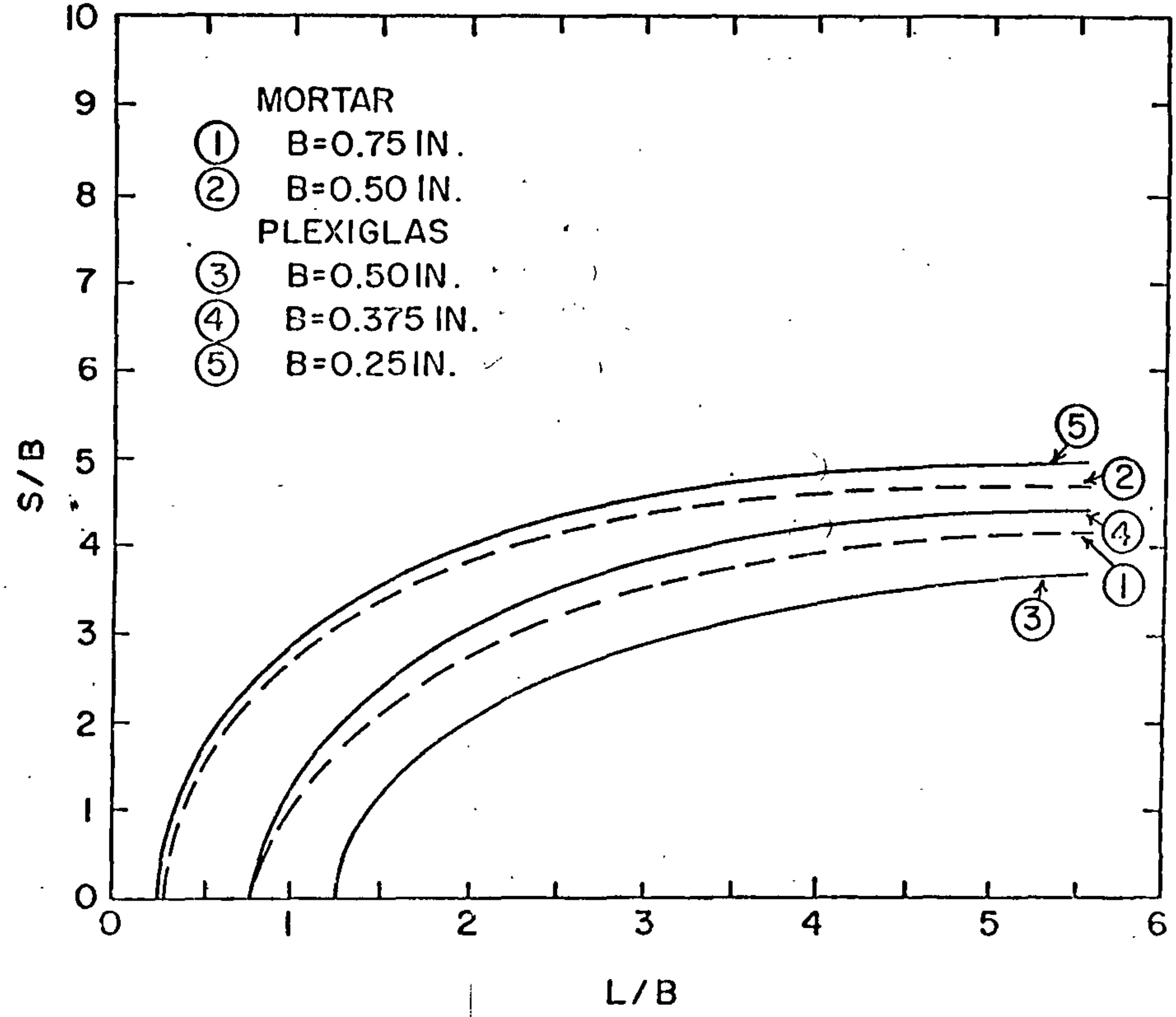


SIMULTANEOUS TIMING, S=1.4B

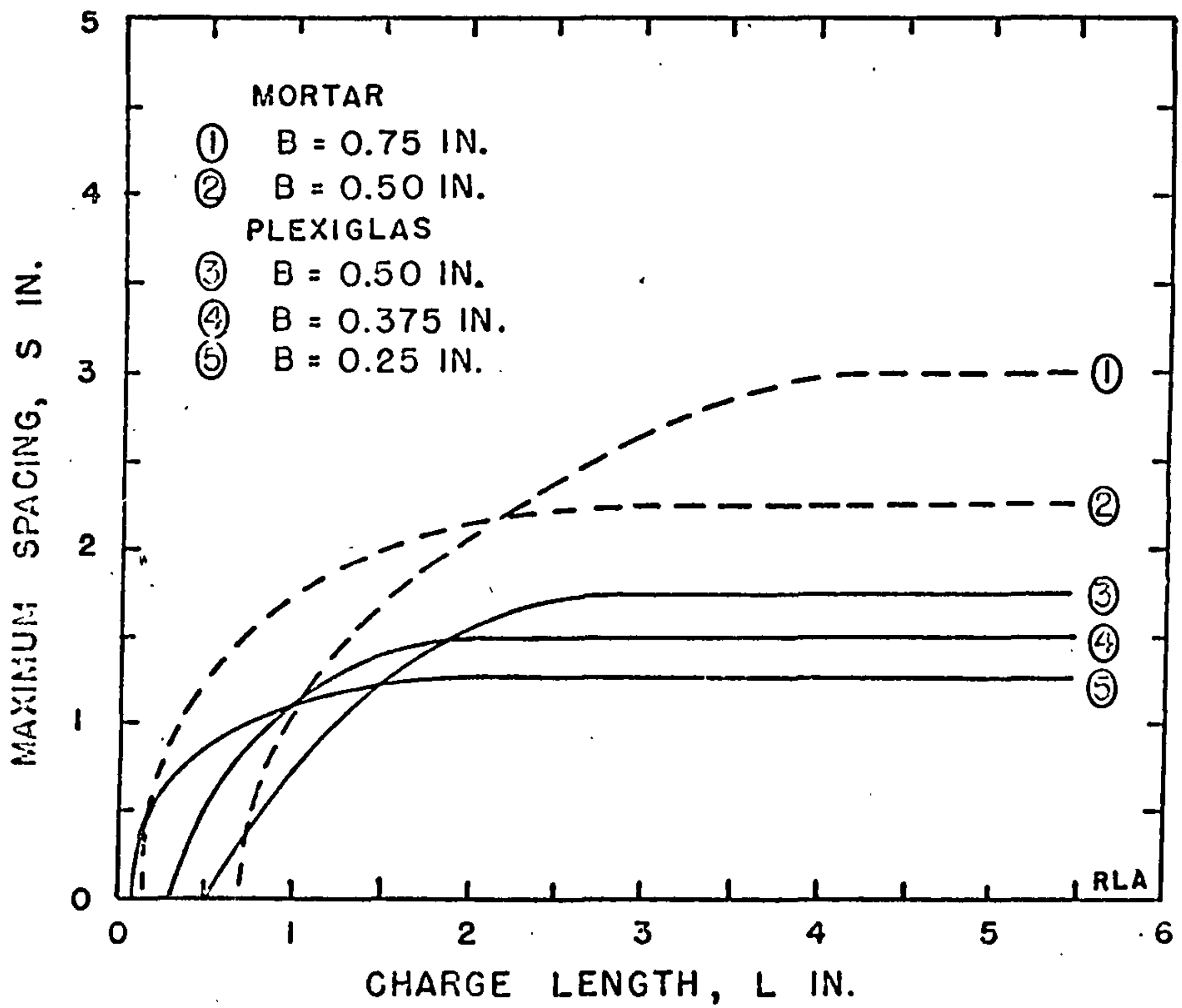


SIMULTANEOUS TIMING, S=2B

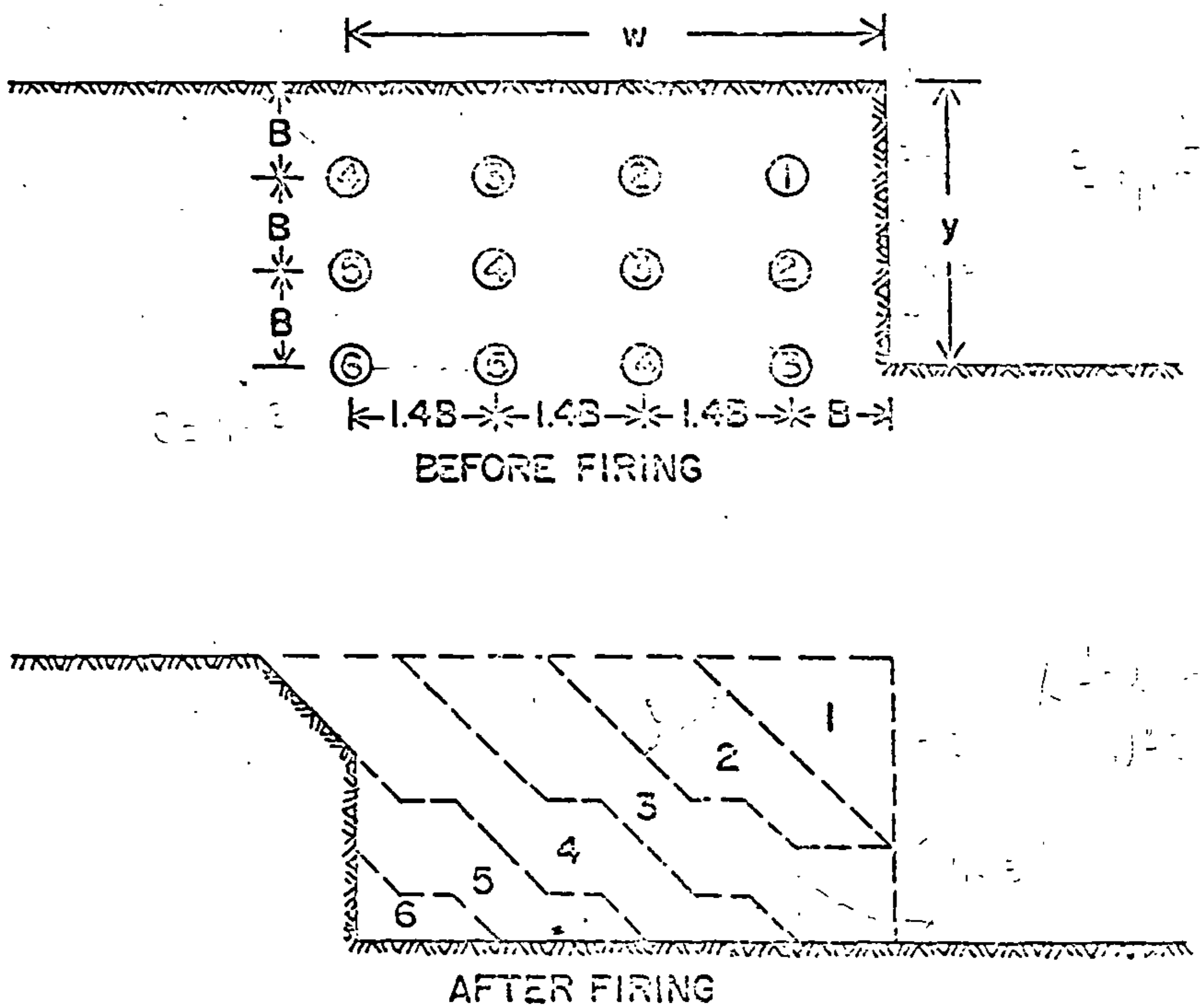




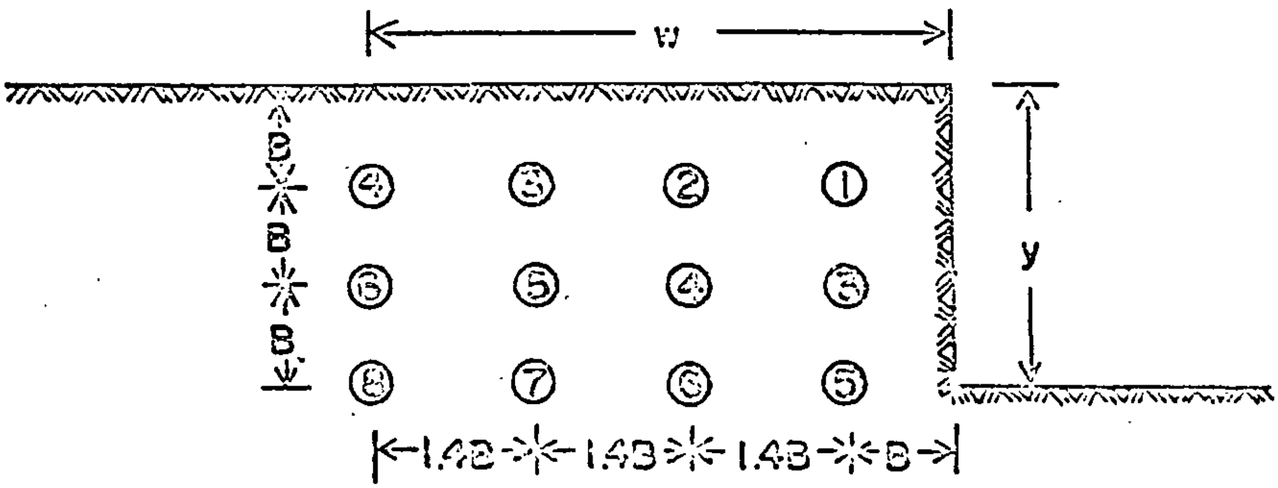
MORTAR
① B=0.75 IN.
② B=0.50 IN.
PLEXIGLAS
③ B=0.50 IN.
④ B=0.375 IN.
⑤ B=0.25 IN.
S/B
L/B


MORTAR
① B = 0.75 IN.
② B = 0.50 IN.
PLEXIGLAS
③ B = 0.50 IN.
④ B = 0.375 IN.
⑤ B = 0.25 IN.
MAXIMUM SPACING, S IN.
CHARGE LENGTH, L IN.
RLA
0
1
2
3
4
5
6

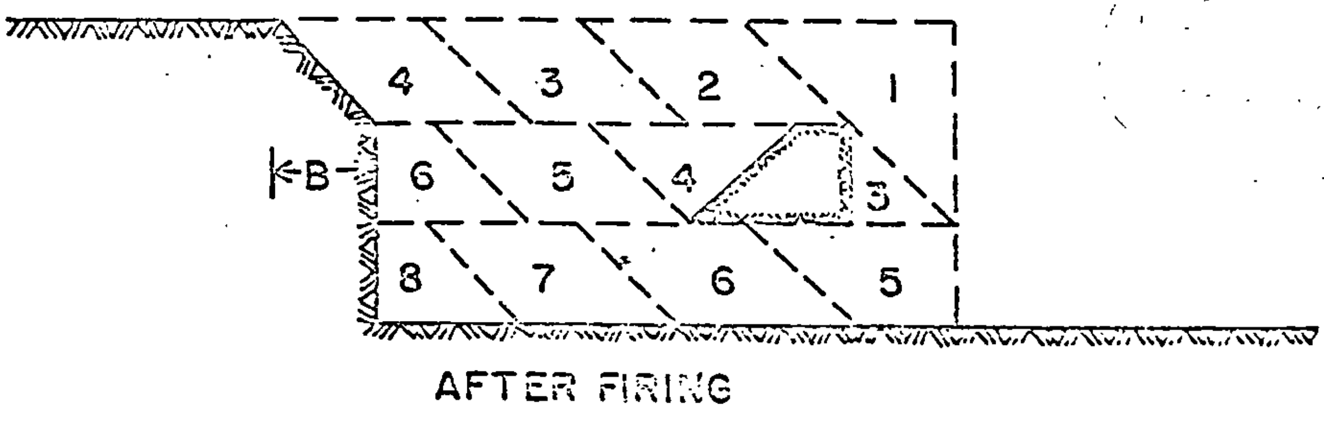
90-DEG. CRATERING, PROGRESSIVE DELAY, SQUARE (A)



90-DEG. CRATERING, PROGRESSIVE DELAY, SQUARE (B)

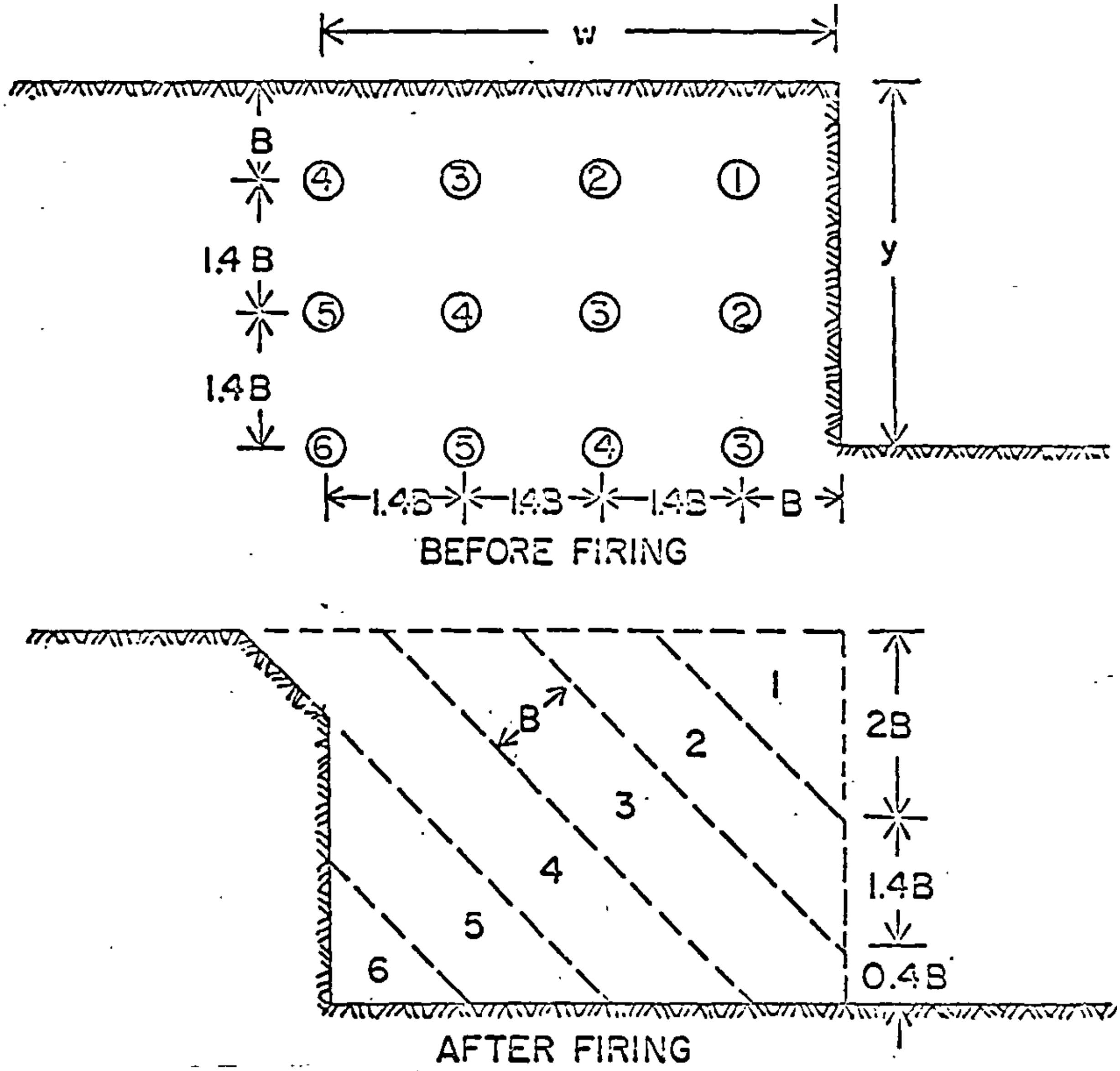


BEFORE FIRING

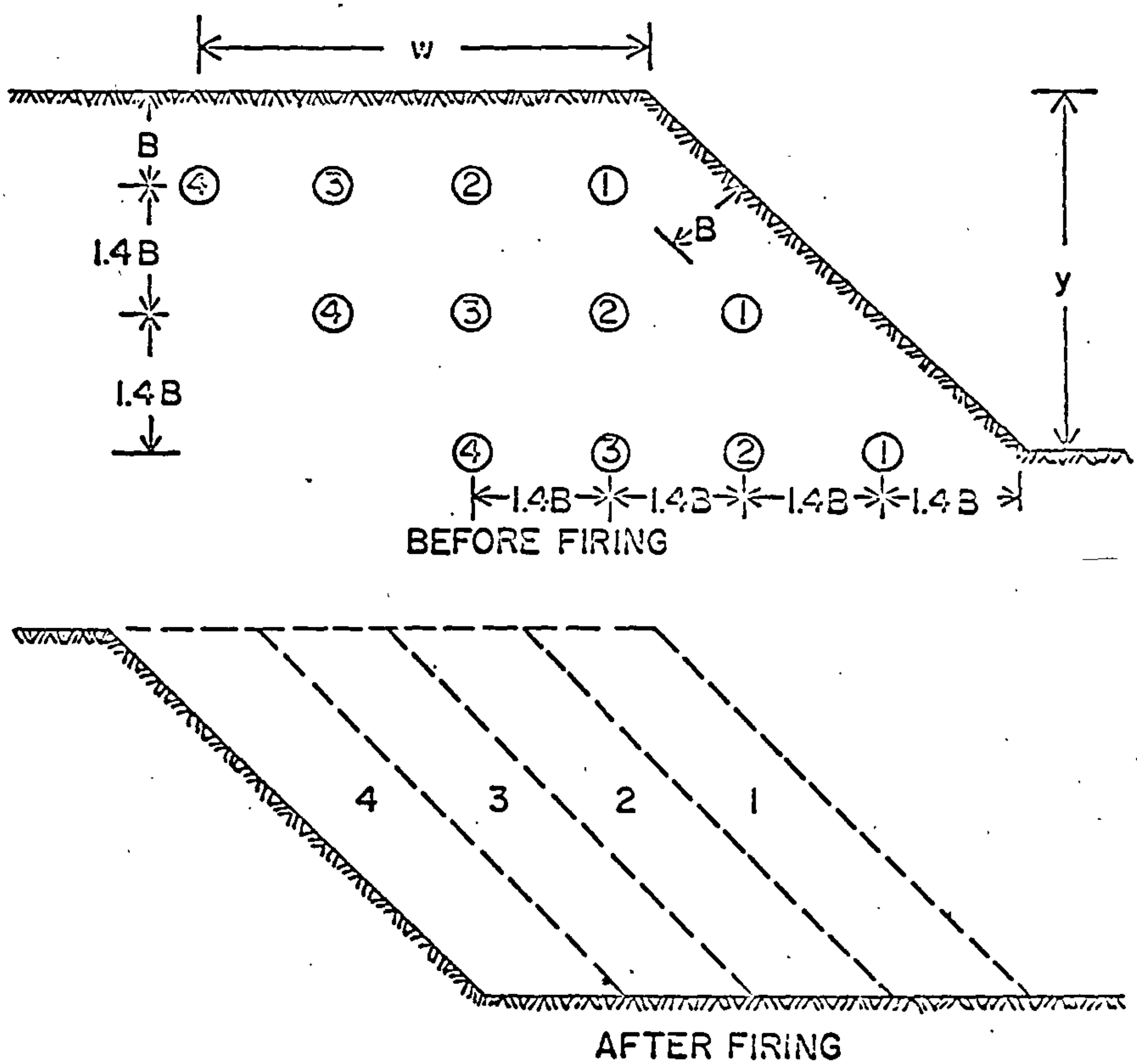


AFTER FIRING

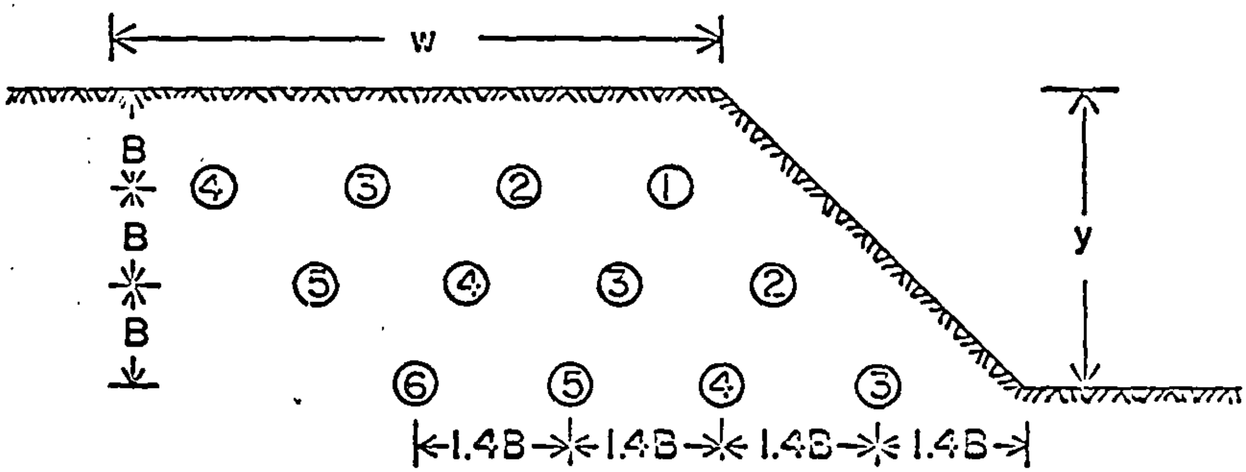
90-DEG. CRATERING, PROGRESSIVE DELAY, SQUARE (C)



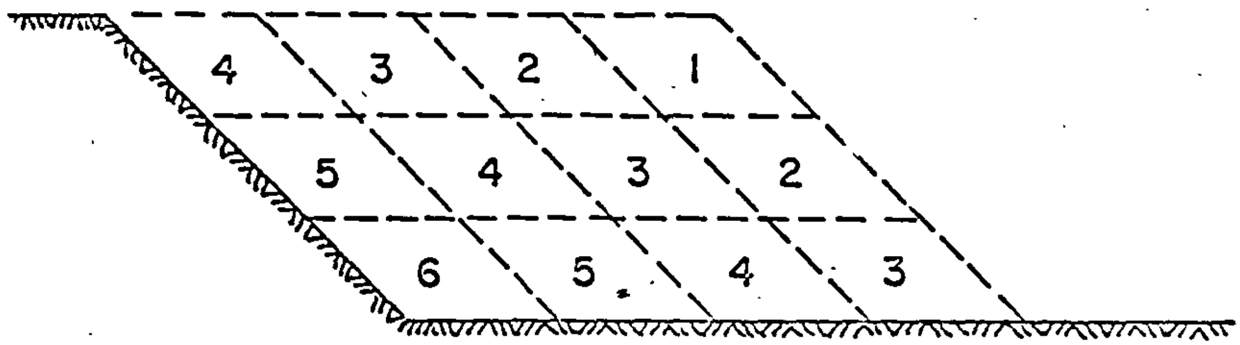
90-DEG. CRATERING, PROGRESSIVE DELAY, SQUARE (D)



90-DEG. CRATERING, PROGRESSIVE DELAY, STAGGERED (E)

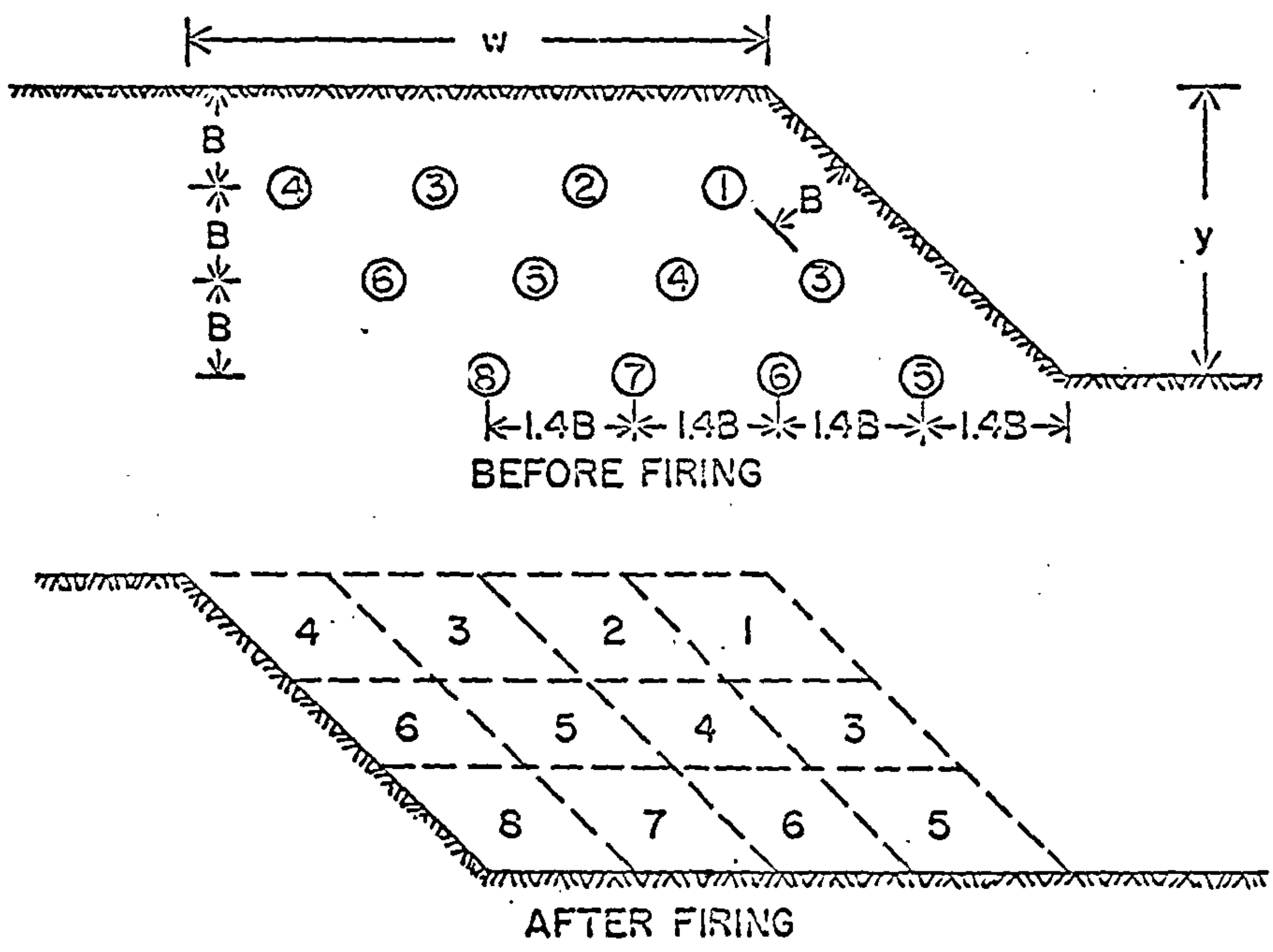


BEFORE FIRING

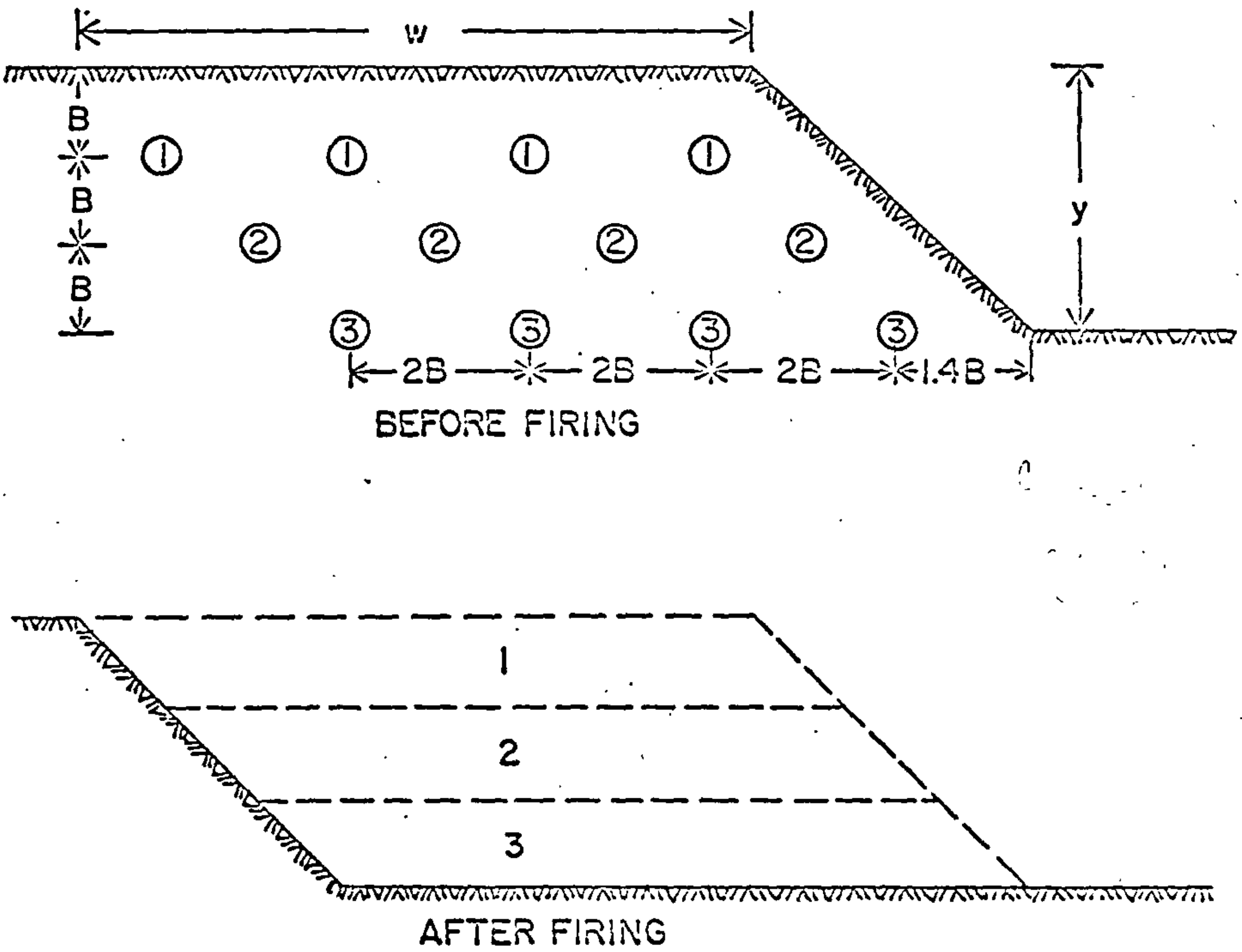


AFTER FIRING

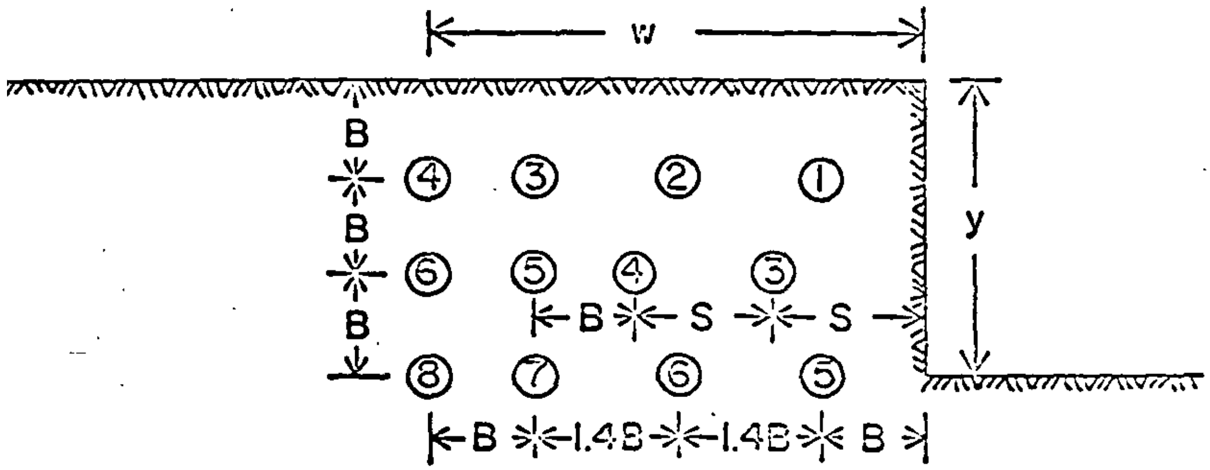
90-DEG. CRATERING, PROGRESSIVE DELAY, STAGGERED (F)



90-DEG. CRATERING, SIMULTANEOUS, STAGGERED



90-DEG. PROGRESSIVE DELAY, STAGGERED (G)



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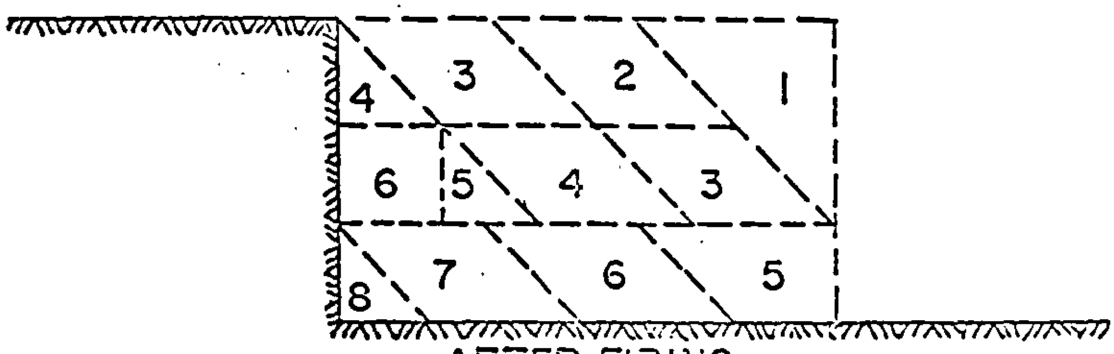
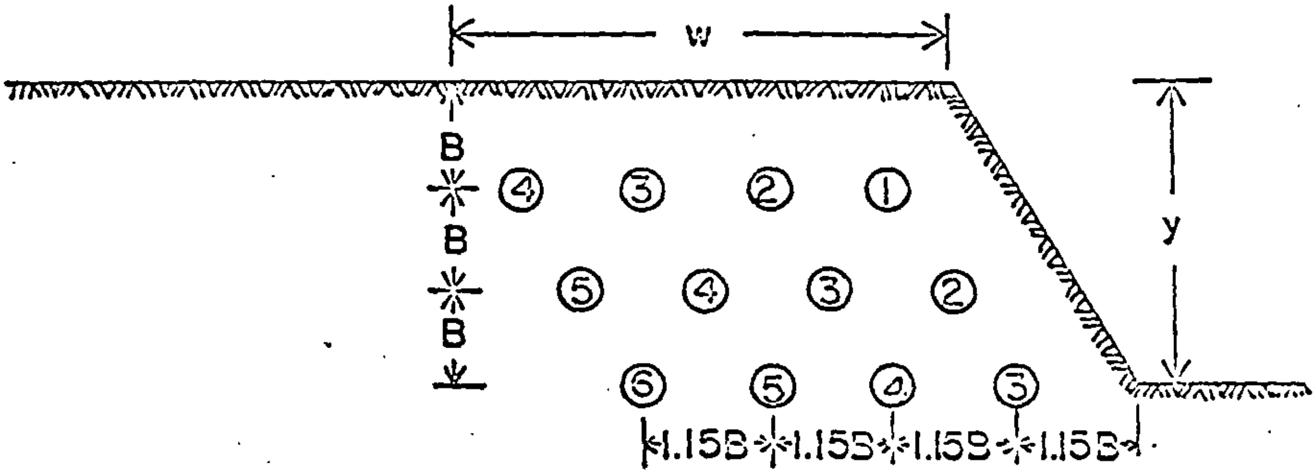
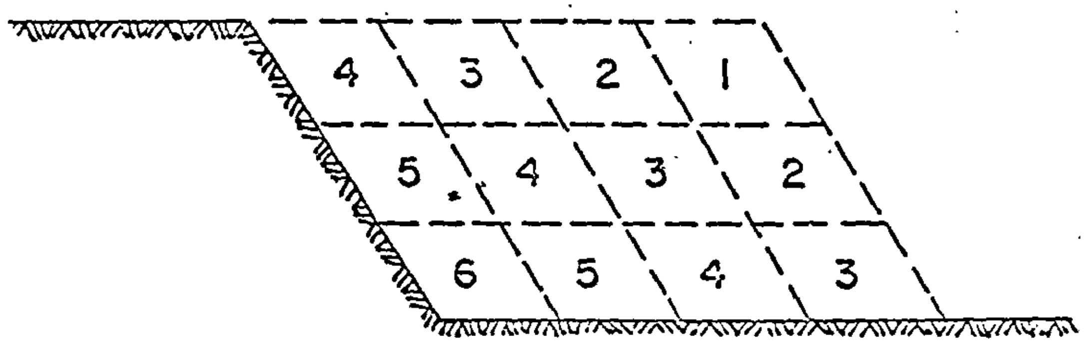


AFTER FIRING

120-DEG. CRATERING, PROGRESSIVE DELAY, STAGGERED (H)

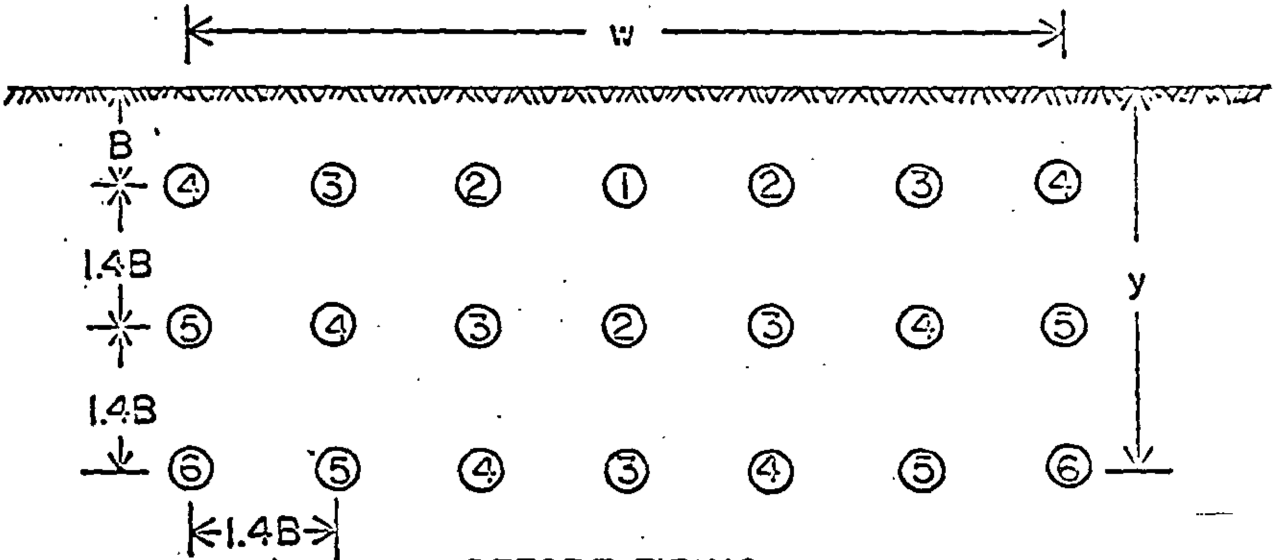


BEFORE FIRING

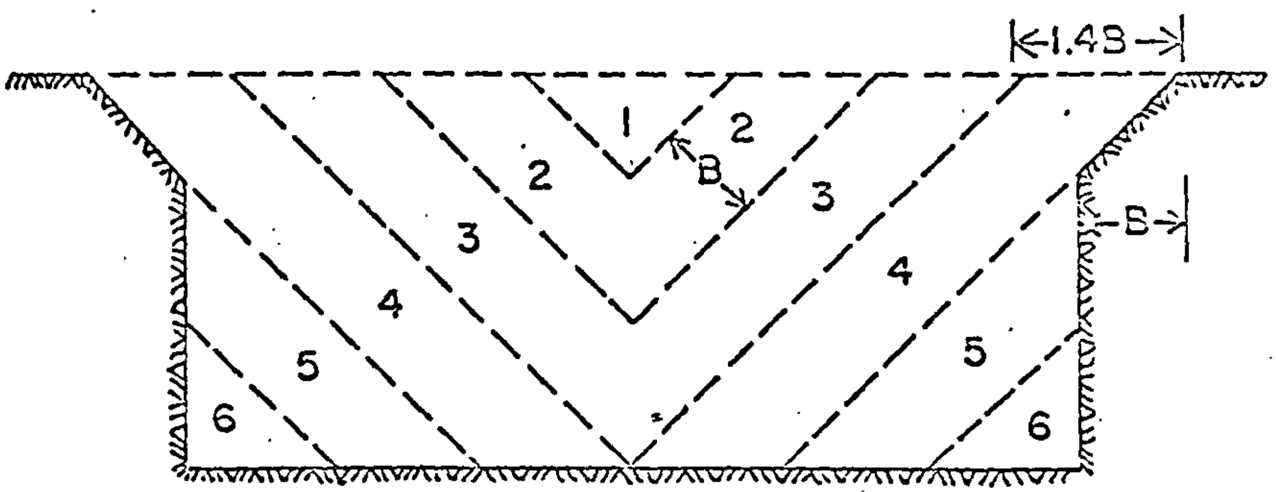


AFTER FIRING

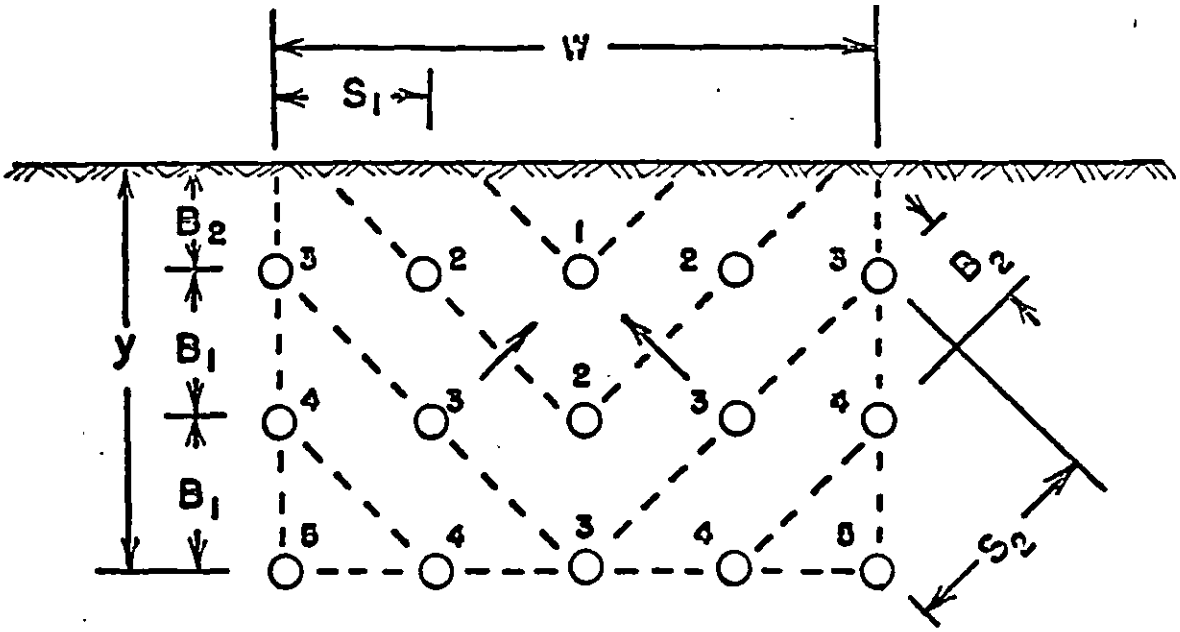
V CUT, 90-DEG. CRATERING, PROGRESSIVE DELAY (A)



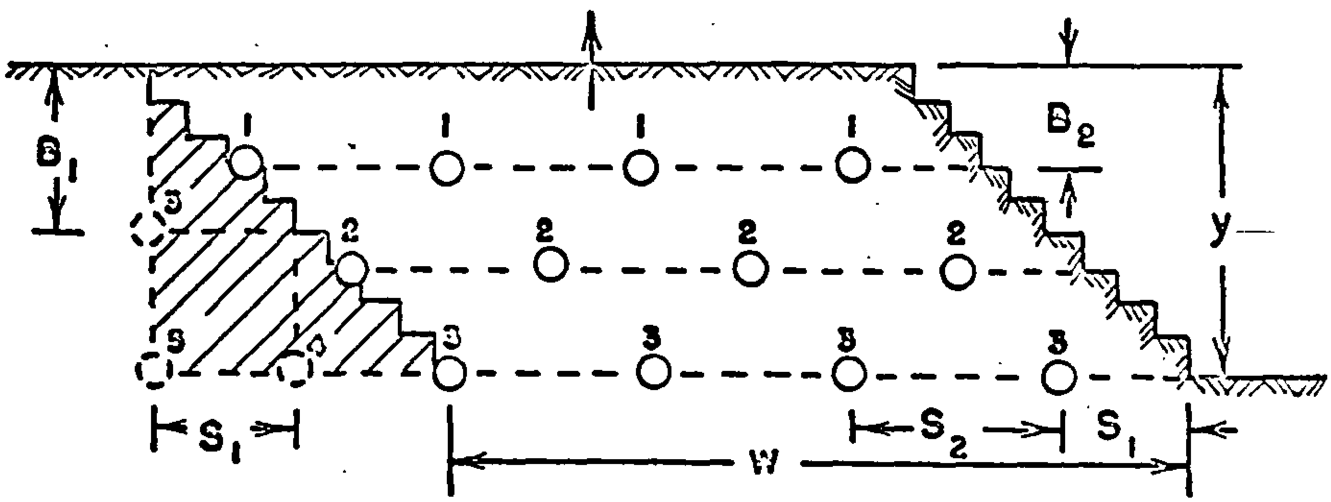
BEFORE FIRING



AFTER FIRING



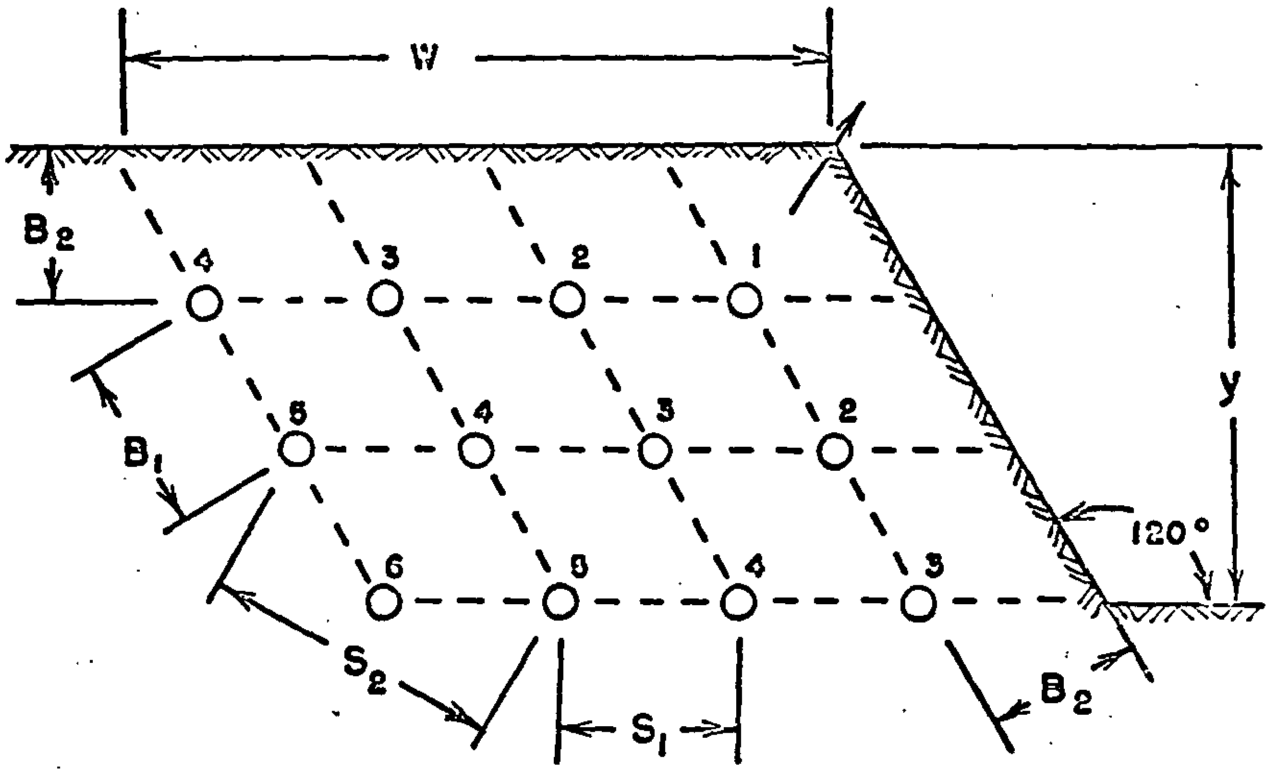
DELAYED INITIATION FOR HOLES IN SAME ROW



SIMULTANEOUS INITIATION FOR HOLES IN SAME ROW

$S = B_1 \qquad S_1 = 1.4B_2 \qquad S_2 = 2B_2$

RLA



DELAYED INITIATION FOR HOLES IN SAME ROW, MODIFIED FOR 120-DEG CRATERING ANGLE

$S_1 = B_1$ $\qquad$ $S_2 = 2B_2$ $\qquad$ $S_1 = 1.15B_2$

RLA

MULTIPLE BLASTHOLE OPEN-CUT DESIGN PROBLEM

A magnetite ore deposit with a tight nearly equilateral vertical jointing system is mined by open-pit benching. The ore is quite abrasive, massive, and tough with $\mu = 0.30$ and $v_p = 18{,}000$ fps. Horizontal jointing is present but tight.

About 30,000 tons is blasted each day for each shovel, which supplies a 10 % overrun of broken material for loading over 3 shifts. A power shovel works an estimated 6 hours per 8-hr shift with an average operating cycle time of 45 sec. Trucks are used for haulage to the primary crushing unit.

Blastholes are drilled by rotary units, which work 7 hr per 8-hr shift. Each shovel normally is matched with one drill. For estimating the penetration and footage rates for a drill the following performance characteristics can be used when using an 8-inch diameter bit and 30-ft sectional steels:

DRILLING TIMES IN MINUTES

Operation	Total Depth of Hole Drilled (H, ft)							
	10	20	30	40	50	60	70	80
D	2	8	18	32	50	72	98	128
AS	-	-	-	2	2	2	4	4
RS	1	2	3	5	6	7	9	10
M	3	3	3	3	3	3	3	3
Total Time	6	13	24	42	61	84	114	145

The explosives used consist of ONE cast booster per hole to be located at the floor level. Each blasthole contains a bottom 5-ft load of slurry having an SG_e of 1.4 and a maximum v_e of 18,000 fps. With $D_e = 3$ in. the $v_e = 15{,}000$ fps; at a 5-in. and larger the maximum v_e is attained. For the balance of the powder load AN-FO with an SG_e of 0.9 is used. The AN-FO exhibits a v_e of 11,000 fps at $D_e = 3$ in., reaching its maximum v_e of 15,000 fps at $D_e = 6$ in. and larger. Both the AN-FO have a D_c of 1 in. For estimating purposes the relationship of v_e with D_e in the respective variable ranges can be assumed to be of the following form;

$$y = \frac{cx}{a + bx} .$$

For initiation of the cast booster Primadet MS Delays will be used in the usual manner.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

BLASTING THEORY

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

4. BLASTING THEORY

Fig. 4.1 Bench blasting with EMULITE and NONEL.

The rock is affected by a detonating explosive in three principal stages.

In the first stage, starting from the initiation point, the blasthole expands by crushing the blasthole walls. This is due to the high pressure upon detonation.

In the second stage, compressive stress waves emanate in all directions from the blasthole with a velocity equal to the sonic wave velocity in the rock.
When these compressive stress waves reflect against a free rock face, they cause tensile stresses in the rock mass between the blasthole and the free face. If the tensile strength of the rock is exceeded, the rock breaks in the burden area, which is the case in a correctly designed blast.

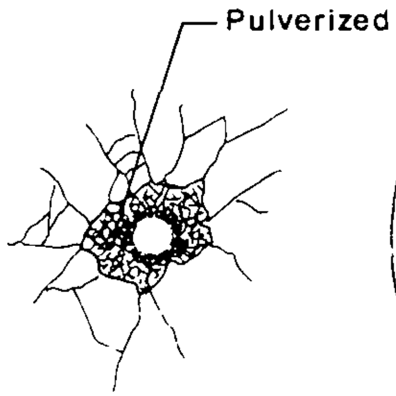


Fig. 4.2 Radial crack formation

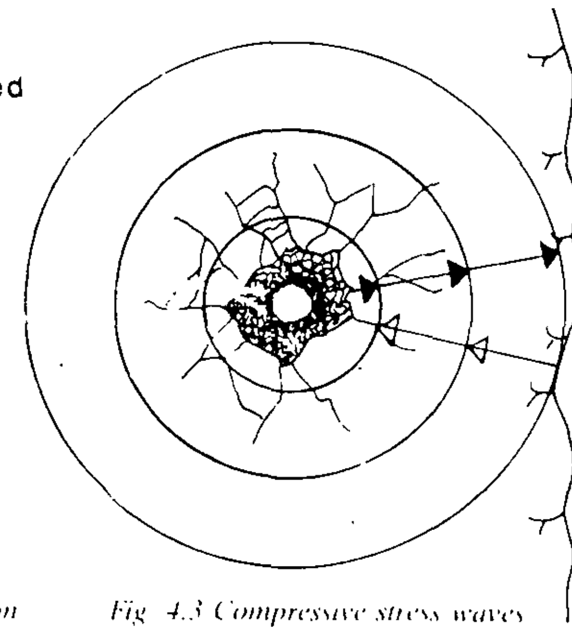


Fig. 4.3 Compressive stress waves

In the third stage, the released gas volume "enters" the crack formation under high pressure, expanding the cracks. If the distance between the blasthole and the free face is correctly calculated, the rock mass between the blasthole and the free face will yield and be thrown forward.

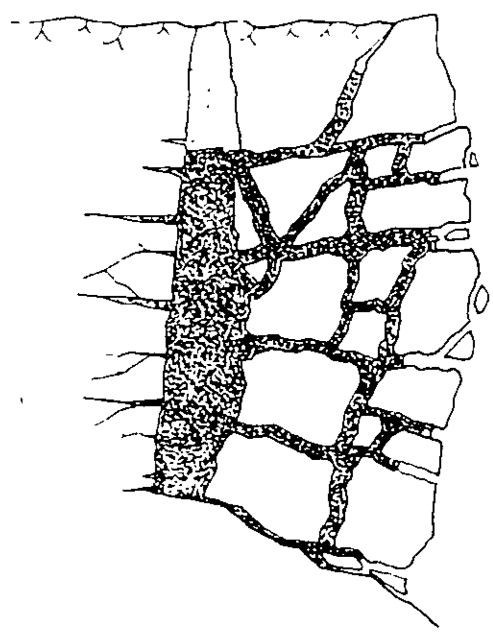


Fig. 4.4 Gas penetration of crack formation

The explosives reaction in the blasthole is very fast and the effective work of the xplosive is considered completed when the blasthole volume has expanded to 0 times its original volume which takes approx. 5 ms.

he following graph shows how the expansion of the blasthole is related to time

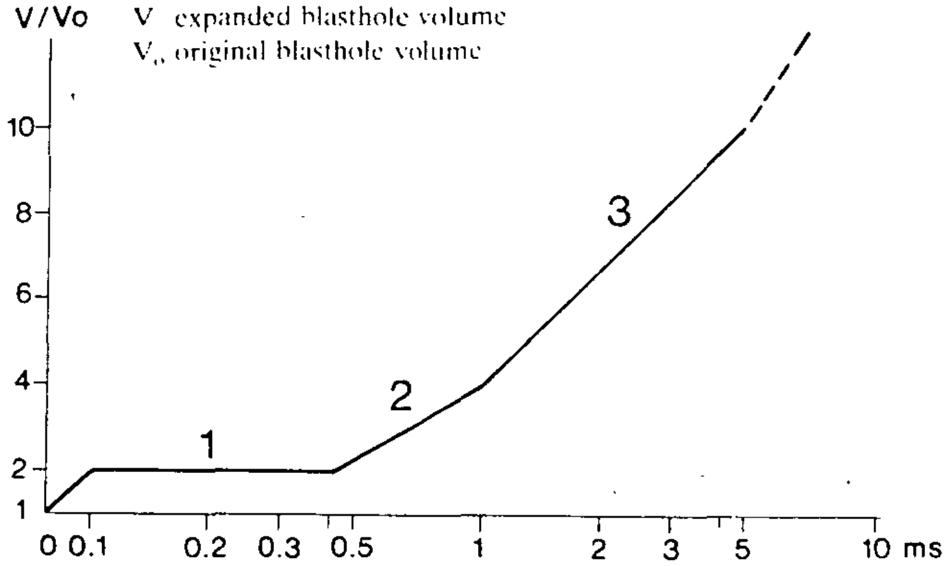


Fig. 4.5 Blasthole expansion in relation to time

1. Initiation of shockwave in rock crushing. The blasthole expands to double its original volume ($2V_o$). The blasthole will stay at this volume for relatively long time (0.1 to 0.4 ms) before radial cracks start to open
2. Besides the natural cracks are new cracks formed mainly by interaction between the stress field around the blasthole and tensile stresses formed by reflection of the outgoing shockwave at the free face. Reaction products expand from blasthole (which volume now is quadrupled) into the cracks. Fragmentation starts.
3. Gas expands further and accelerates the rock mass

5. BENCH BLASTING

Fig. 5.1 Quarry blasting in Sweden

5.1 General

Bench blasting is the most common kind of blasting work. It can be defined as blasting of vertical or close to vertical blastholes in one or several rows towards a free surface. The blastholes can have free breakage or fixed bottom

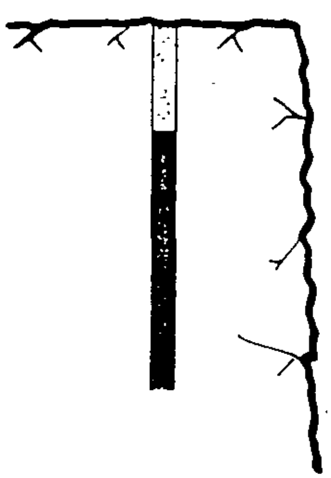


Fig. 5.2 Free breakage

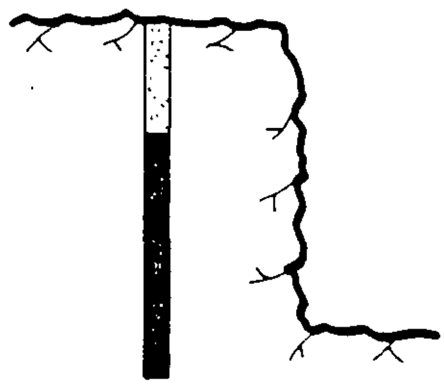


Fig. 5.3 Fixed bottom

Most types of blasting can be considered as bench blasting.

Trench blasting for pipelines is also a kind of bench blasting, but as the rock is more constricted, it requires a higher specific charge and more closely spaced drilling.

In tunneling, after the cut has been blasted, the stoping towards the cut is a type of bench blasting.

Rock is a material with widely varying properties. Its tensile, compressive and shearing strengths vary with different kinds of rock and may vary within the same blast. As the rock's tensile strength has to be exceeded in order to break the rock, its geological properties will affect its blastability.

Rock formations are rarely homogeneous. The rock formation in the blast area may consist of different types of rock. Furthermore, faults and dirt-seams may change the effect of the explosive in the blast. Faulty rock containing voids where the gases penetrate without giving full effect, may be difficult to blast even though the rock may have a relatively low tensile strength.

The requisite specific charge, (kg/cu m.) provides a first-rate measure of the blastability of the rock. By using the specific charge as a basis for the calculation, it is possible to calculate the charge which is suitable for the rock concerned.

The distribution of the explosives in the rock is of the utmost importance. A closely spaced round with small diameter blastholes gives much better fragmentation of the rock than a round of widely spaced large diameter blastholes, provided that the same specific charge is used. (See Chapter 5.6 Fragmentation.)

The following calculations are based on a specific charge of 0.4 kg/cu m. of EMULITE 150 in the bottom part of the round. In the constricted bottom part of the blasthole, this specific charge is needed to shatter the burden, but in the column part of the hole considerably less explosives are needed to break the rock. The average specific charge of the round (hole) will be less than 0.4 kg/cu m.

The value applies to burdens between 1.0 and 10.0 m and can be used for most kinds of rock. The basis of the computations of bench blasting will be Langefors'* formula:

$$B_{max} = \frac{d}{33}\sqrt{\frac{p \cdot s}{\bar{c} \cdot f \cdot E/V}}$$

where

B_{max}	= maximum burden	(m)
d	= diameter in the bottom of the blasthole	(mm)
p	= packing degree (loading density)	(kg/liter)
s	= weight strength of the explosive	(EMULITE 150 = 0.95)
c	= rock constant	(kg/cu m)
$\bar{c}$	= c + 0.05 for B_{max} between 1.4 and 15.0 meters	
f	= degree of fixation, 1.0 for vertical holes and 0.95 for holes with inclination 3:1	
S/B	ratio of spacing to burden	

* The Modern Technique of Rock Blasting, Langefors/Kihlström

In the following calculations, Langefors' formula is simplified to:

$B_{max} = 1.47\sqrt{l_b}$ for Dynamex M
$B_{max} = 1.45\sqrt{l_b}$ for Emulite 150
$B_{max} = 1.36\sqrt{l_b}$ for ANFO

where l_b is the requisite charge concentration (kg/m) of the selected explosive in the bottom part of the blasthole.

The hole inclination is assumed to be 3:1 and the rock constant c is 0.4. The bench height K is $\geq 2 \times B_{max}$.

For other values of hole inclination and rock constant correction factors are used.

The charge concentration depends on the diameter of the blasthole and the utilization of the hole.

Explosives in paper cartridges, which are normally tamped with a tamping rod in small diameter blastholes, can be tamped to an utilization of up to 90 % of the blasthole if tamping is carried out after the introduction of each cartridge. If tamping is carried out after every two or three cartridges, the charge concentration will be considerably lower. Pneumatic charging machines give good tamping of paper cartridges with high utilization of the blasthole volume.

Explosives in plastic hoses were introduced for the convenience of fast charging and easy handling. Dropped into the blasthole, they are intended to fill up the hole well. However, different tamping characteristics of different explosives give varying results. Emulite cartridges in plastic hoses, which are cut along the side, fill up the hole almost completely by impact, while dynamites and watergels with their stiffer consistency do not fill up the hole that well, especially in the winter. It is important when charging wet blastholes that the holes are flushed and cleaned before charging. If the blastholes contain water, the packing of the explosive will be almost nil and the charge concentration of the cartridges should be used for the calculations. Bulk explosives which are pumped, augered or poured into the blasthole utilize the blasthole volume to 100 %.

The calculations that follow will involve the following explosives

Dynamex M
Emulite 150
ANFO

which are explosives with differing characteristics regarding weight strength and density.

As the maximum burden, B_{max}, is also dependent on the fixation degree at the bottom part of the blasthole, the computations will involve drilling with inclination 3:1, which decreases the constriction in the bottom part of the hole. For other inclinations correction factors can be used.

The packing degree (utilization of the blasthole) of the explosive in the bottom part of the blasthole is assumed to be 95 % for Emulite 150 in plastic hoses and 90 % for Dynamex M. Poured ANFO and pumped Emulite fill up the hole to 100 %.

It is very important for the blasting result that the charge concentration obtained by the calculations is achieved in practice.

The formulae used in the calculations are empirical, but are based on information from thousands of blasts. The experience of Langefors' calculations is so good that it could be considered unnecessary in most blasting operations to make trial blasts. However, local conditions may make it necessary for the practical operator to test the theoretical calculations in the field.

ERRATA

The formula on page 64 should be

$$B_{max} = \frac{d}{33}\sqrt{\frac{p \cdot s}{\overline{c} \cdot f \cdot S/B}}$$

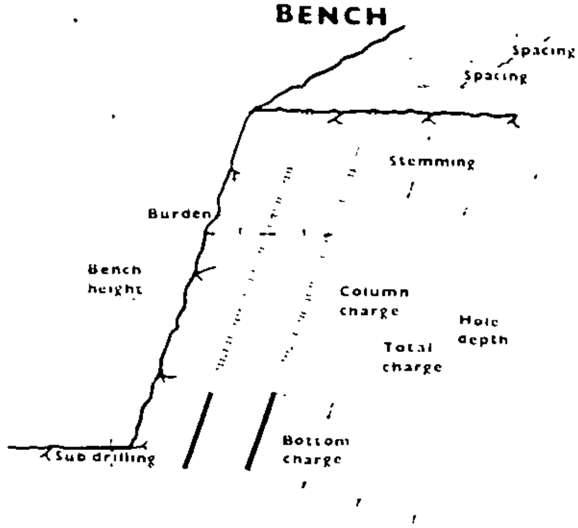
We regret the inconveniences the error may cause.

Yours sincerely
Stig O Olofsson

5.2 Charge calculations.

Bench height $\geq 2 \times B_{max}$

d	Diameter of the blasthole in the bottom	(mm)
K	Bench height	(m)
B_{max}	Maximum burden	(m)
U	Subdrilling	(m)
H	Hole depth	(m)
F	Error in drilling	(m)
B	Practical burden	(m)
S	Practical spacing	(m)
b	Specific drilling	(m/m³)
l_b	Concentration of bottom charge	(kg/m)
h_b	Height of bottom charge	(m)
Q_b	Weight of bottom charge	(kg)
h_0	Height of stemming	(m)
l_c	Concentration of column charge	(kg/m)
h_c	Height of column charge	(m)
Q_c	Weight of column charge	(kg)
Q_{tot}	Total charge weight per hole	(kg)
q	Specific charge	(kg/m³)

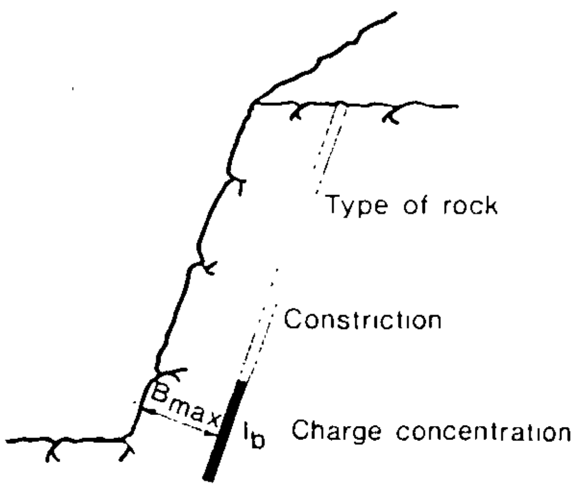


The following assumptions are made for the calculations.

K is equal to, or more than $2 \times B_{max}$

Explosive	Emulite 150	Dynamex M	ANFO
Packing degree	95 %	90 %	100 %
	1.15 kg/l	1.25 kg/l	0.8 kg/l
Rock constant c:	0.4	0.4	0.4
Hole inclination:	3:1	3:1	3:1

Calculation procedure:
The **maximum burden** in the bottom of the blasthole depends on:
- weight strength of the actual explosive (s)
- charge concentration (l_b)
- rock constant (c)
- constriction of the blasthole (R_1)



As mentioned before, the maximum burden B_{max} is calculated from Langefors' formula, which has been simplified to:

for Dynamex M $\quad B_{max} = 1.47\sqrt{l_b} \times R_1 \times R_2$ (m)
for Emulite 150 $\quad B_{max} = 1.45\sqrt{l_b} \times R_1 \times R_2$ (m)
for ANFO $\quad B_{max} = 1.36\sqrt{l_b} \times R_1 \times R_2$ (m)

where l_b = charge concentration, kg/m as per point 1, as follows
R_1 = correction for hole inclination other than 3:1 as per point 2, as follows
R_2 = correction for rock constant other than 0.4 as per point 3, as follows

1. Determination of charge concentration, l_b.
a/ For tamped cartridges and bulk explosives

$$l_b = 7.85\, d^2 \times P$$

where d = blasthole diameter, dm
P = packing degree, kg/liter

Charge concentration for different blasthole diameters and different explosives:

Blasthole diameter (mm)	51	64	76	89	102	127	152
ANFO, kg/m	1.6	2.6	3.6	5.0	6.5	10.1	14.5
Emulite 150 (cut and dropped into dry blastholes), kg/m	2.3	3.7	5.0	7.1	9.3	—	—
Bulk emulite, kg/m	2.4	3.9	5.3	7.5	9.9	15.3	21.9
Dynamex M (charged with pneumatic charging machine and ROBOT), kg/m	2.6	4.0	5.6	7.8	10.2	—	—

Charge concentration, kg/m, for drill series 11 and 12. Tamped explosives.

	Blasthole diameter, mm												
	27	28	29	30	31	32	33	34	35	36	37	38	39
Em150	0.66	0.71	0.76	0.81	0.87	0.92	0.98	1.04	1.11	1.17	1.24	1.30	1.37
Dx M	0.69	0.74	0.79	0.85	0.91	0.96	1.03	1.09	1.16	1.22	1.29	1.36	1.43

b/ Charge concentration, l_b, for explosives in plastic hoses at different degrees of compression.

Hose diameter (mm)	DYNAMEX M			EMULITE 150			
	Charge concentration kg/m at a compression of			Charge concentration kg/m at a compression of			
	0 %	5 %	10 %	0 %	10 %	20 %	25%
43	1.95	2.05	2.15	1.75	1.90	2.10	2.20
50	2.65	2.80	2.90	2.35	2.60	2.80	2.90
55	3.20	3.35	3.50	2.85	3.10	3.40	3.55
60	—	—	—	3.40	3.75	4.05	4.25
65	4.40	4.60	4.80	4.00	4.40	4.80	5.00
75	—	—	—	5.30	5.80	6.35	6.60
80	6.50	6.80	7.10	—	—	—	—
90	8.00	8.40	8.80	—	—	—	—
125	14.50	15.20	16.00	—	—	—	—

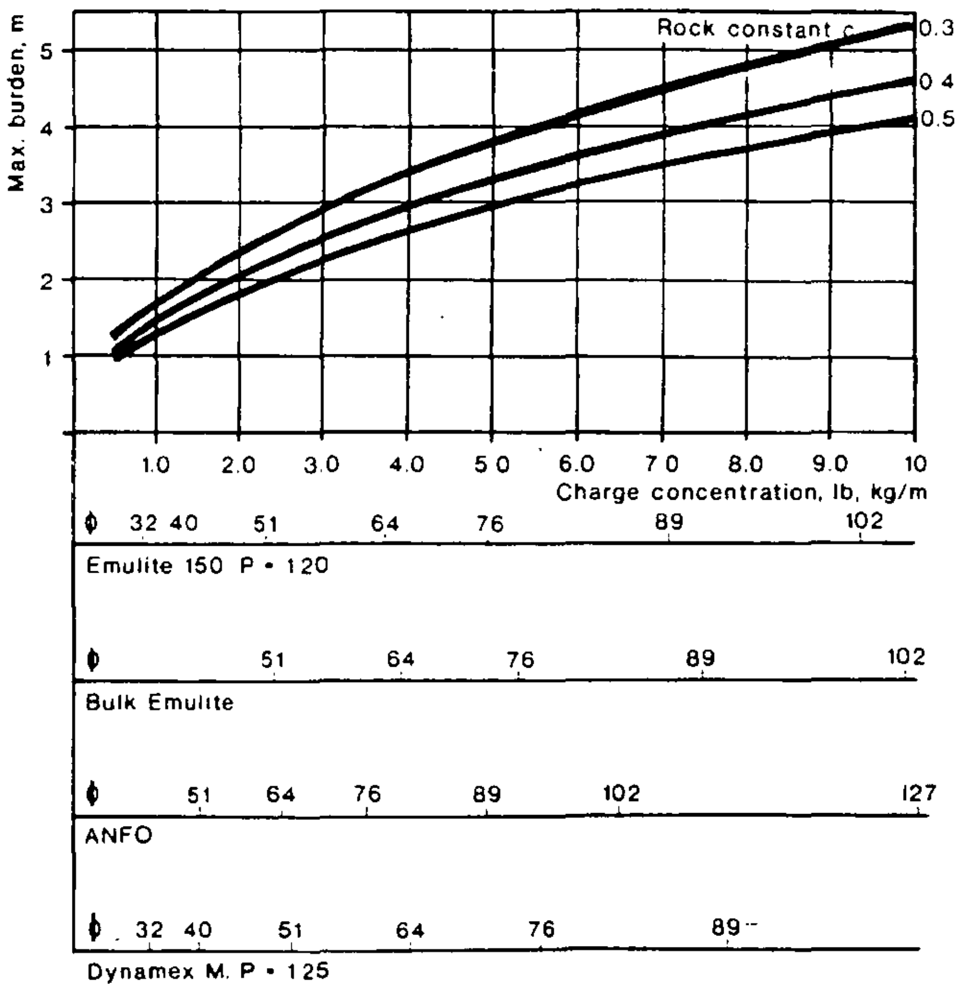
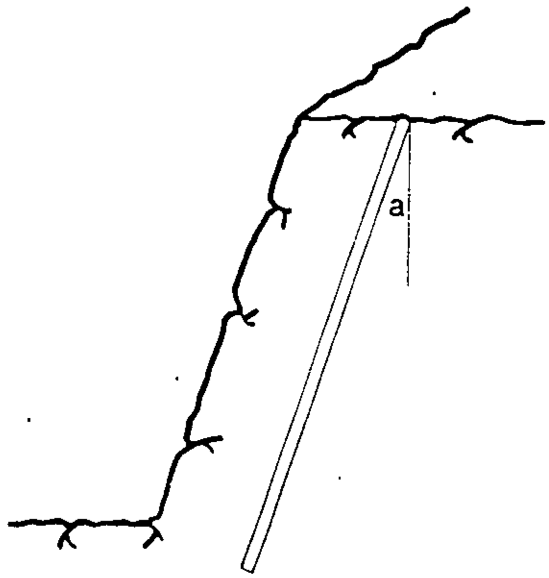


Fig. 5.4 The influence of charge concentration on maximum burden, B_{max}.

2. Correction of B_{max} for different hole inclinations.

Incli-nation	Vertical	10:1	5:1	3:1	2:1	1:1
R_1	0.95	0.96	0.98	1.00	1.03	1.10



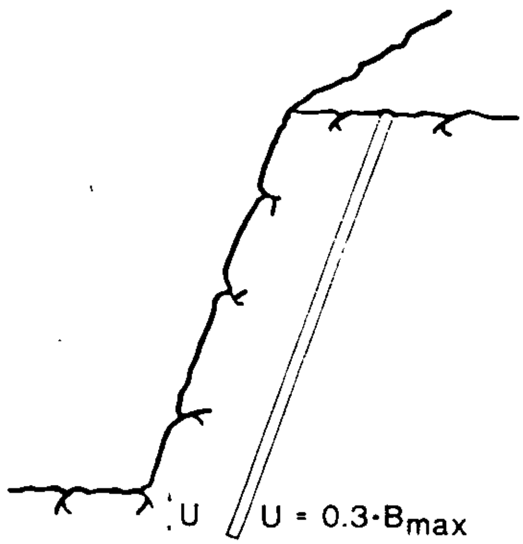
3. Correction of B_{max} for different rock constant c.

c	0.3	0.4	0.5
R_2	1.15	1.00	0.90

Subdrilling = 0.3 × maximum burden, at least 10 × d.

$U = 0.3 \times B_{max}$ (m)

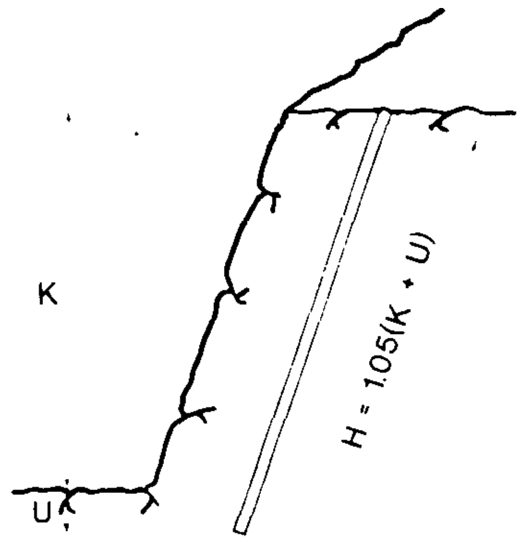
The subdrilling is necessary to avoid stumps above the theoretical grade.



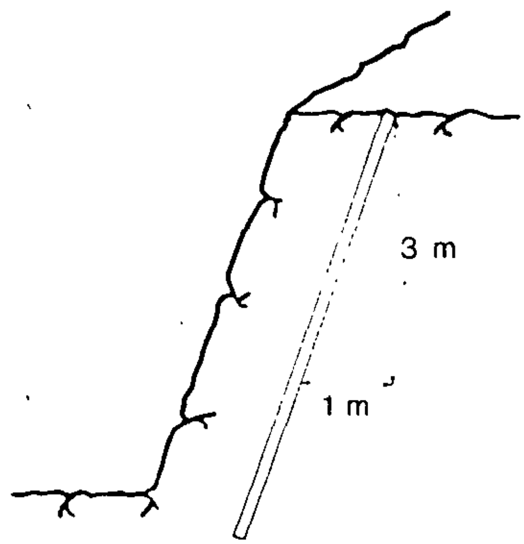
Depth of the hole = bench height + subdrilling + 5 cm/m of the depth of the blasthole due to 3:1 inclination.

$$H = K+U+0.05(K+U)$$

$$\mathbf{H = 1.05(K+U)} \qquad \mathbf{(m)}$$



Inclined holes have a favorable angle of breakage between the holes and the intended bottom, thus decreasing the constriction in the bottom part of the holes.



Faulty drilling consists of:

* Collaring error = d (in mm)
* Alignment error = 0.03 m/m of the blasthole depth.

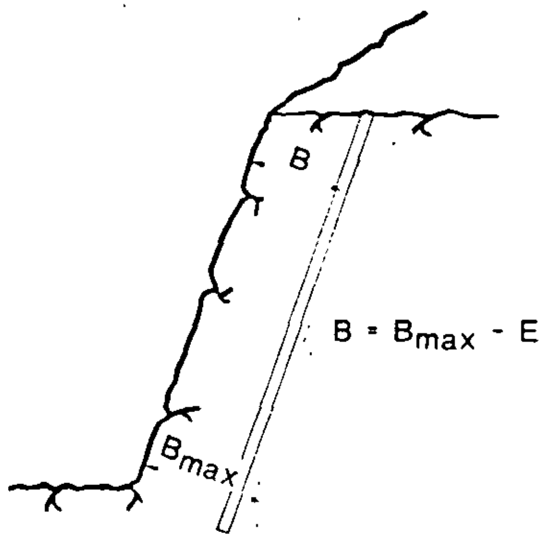
$$E = \frac{\mathbf{d}}{\mathbf{1000}} + \mathbf{0.03 \times H} \qquad \mathbf{(m)}$$

It has to be taken into account that it is impossible to drill a hole exactly in accordance with theoretical computations.
Both the machines used and the skill of the operator affect the accuracy of the drilling.
The error should not be allowed to exceed E as calculated in accordance with the above formula.

When the **practical burden** is calculated, the error in drilling has to be deducted.

$B = B_{max} - E$ (m)

The rule of thumb, B = d, where B (burden) is expressed in meters and d (blasthole diameter) is expressed in inches, can be used to check the calculations.



The burden is the distance from the blasthole to the nearest free face at the instant of detonation. In multiple row blasts new faces are created at each detonation.

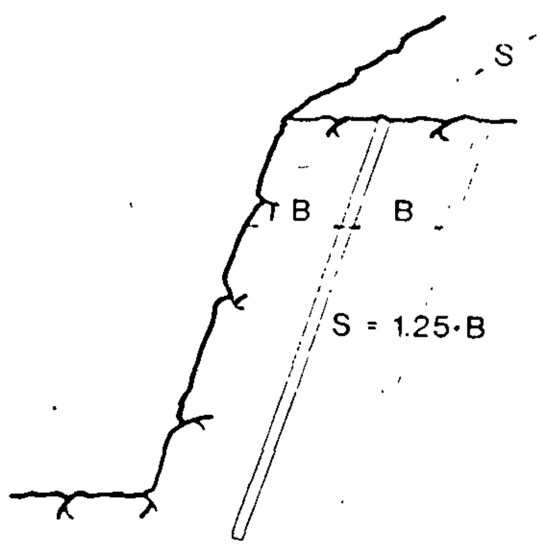
Practical hole spacing S is calculated from the relation

$S = 1.25 \times B$ (m)

Spacing is the distance between the adjacent blastholes in a row.

If the ratio S/B is changed without the specific drilling or the specific charge being changed it will result in the following:

S/B>1.25, finer fragmentation
S/B<1.25, coarser fragmentation



Specific drilling is the drilling needed to blast 1 cu.m. of rock and can be expressed as:

$$b = \frac{n \times H}{n \times B \times S \times K} \quad \text{(m/cu.m)}$$

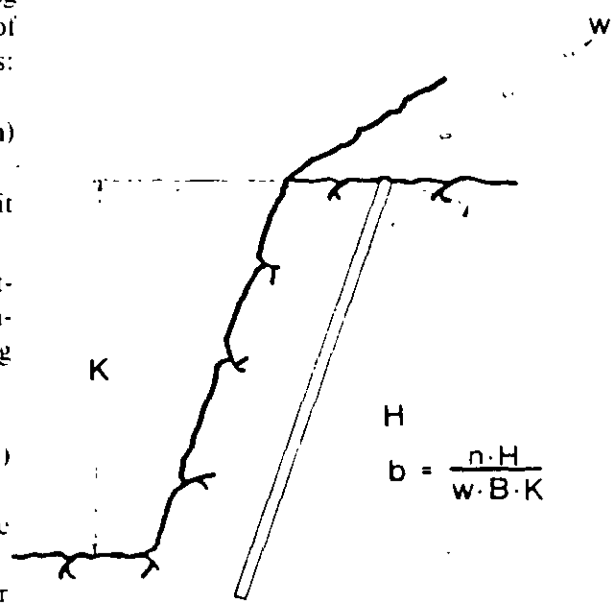
for quarries and open pit mines.

In road cuts etc. where blasting is performed within a limited area, the specific drilling is calculated per row:

$$b = \frac{n \times H}{w \times B \times K} \quad \text{(m/cu.m.)}$$

where w is the width of the round.

The latter value will be higher due to the influence of the edge holes.



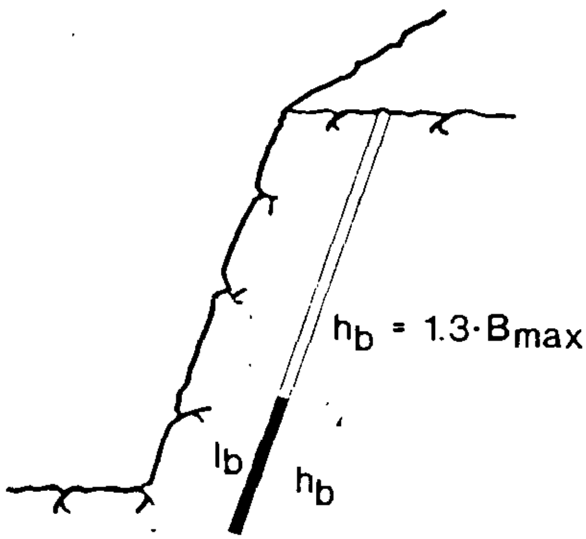
Charging the blasthole.

In order to loosen and break the rock in the constricted bottom part of the blasthole, the charge concentration used for the calculation of B_{max} should be used:

l_b = charge concentration used for determination of B_{max}.

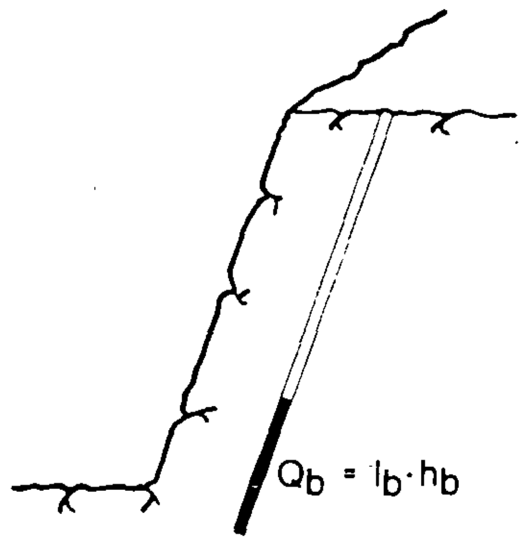
Height of the bottom charge:

$$h_b = 1.3 \times B_{max} \quad \text{(m)}$$



The **bottom charge** will then be:

$\mathbf{Q_b = l_b \times h_b}$ (m)



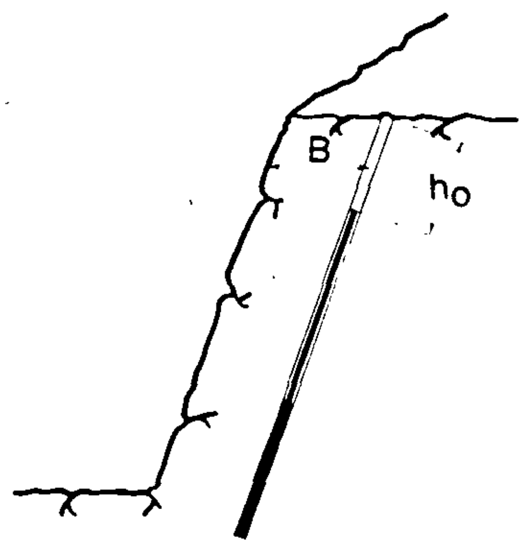
Stemming.

The unloaded part of the blasthole, the stemming, is normally equal to the burden:

$\mathbf{h_0 = B}$ (m)

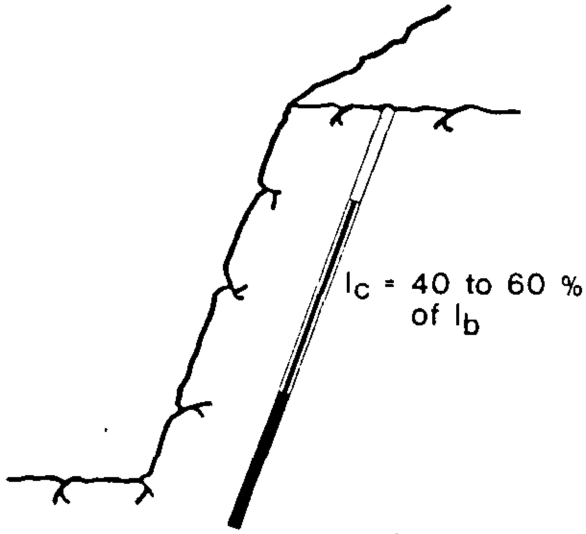
The stemming should consist of sand or gravel with a particle size of 4 to 9 mm Research has shown that this size gives the best confinement of the explosives gases. Drillfines should be avoided.

If $h_0 < B$, the risk of flyrock from the upper surface increases, but the amount of boulders decreases. On the other hand, $h_0 > B$, it will give more boulders but superficial throw will be less or eliminated.



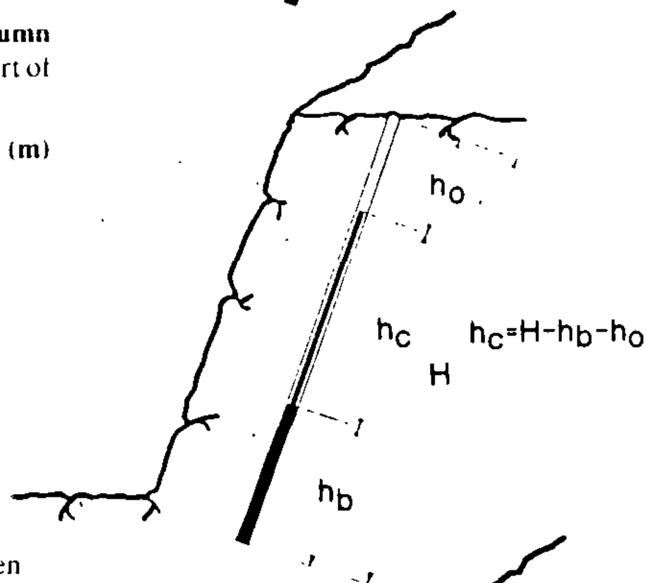
To break the rock above the bottom charge, a **column charge** is applied. As this part of the blasthole is less constricted, the **charge concentration** may be less.

$\mathbf{l_c = 40\ to\ 60\ \%\ of\ l_b}$ **(kg/m)**



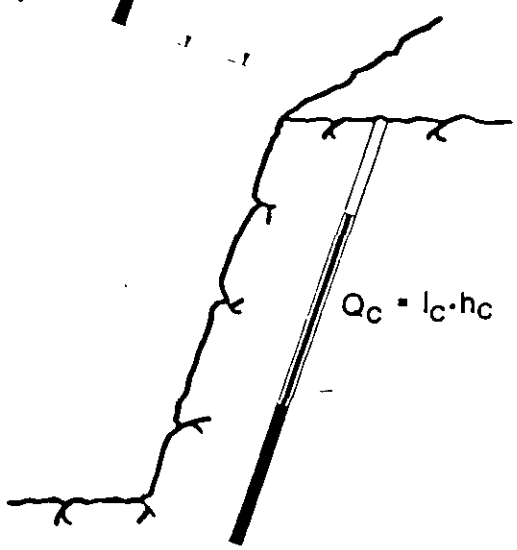
The **height of the column charge** is the remaining part of the blasthole.

$\mathbf{h_c = H - h_b - h_o}$ **(m)**



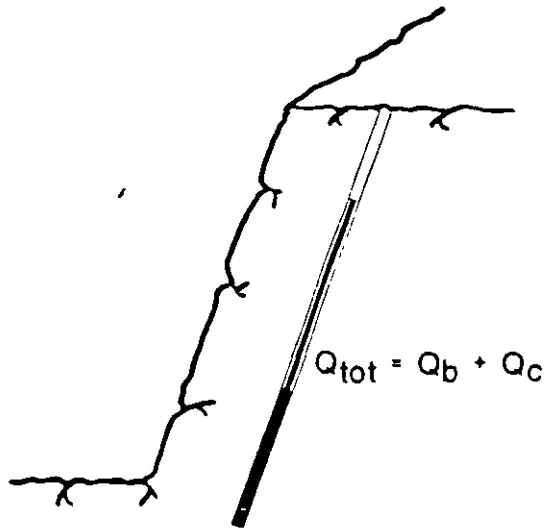
The **column charge** is then

$\mathbf{Q_c = l_c \times h_c}$ **(kg)**



The **total charge** per hole is the bottom charge plus the column charge.

$$Q_{tot} = Q_b + Q_c \qquad \text{(kg)}$$



The **specific charge** may be calculated in the same manner as specific drilling (b).

$$q = \frac{n \times Q_{tot}}{n \times B \times S \times K} \text{ (kg/cu.m.)}$$

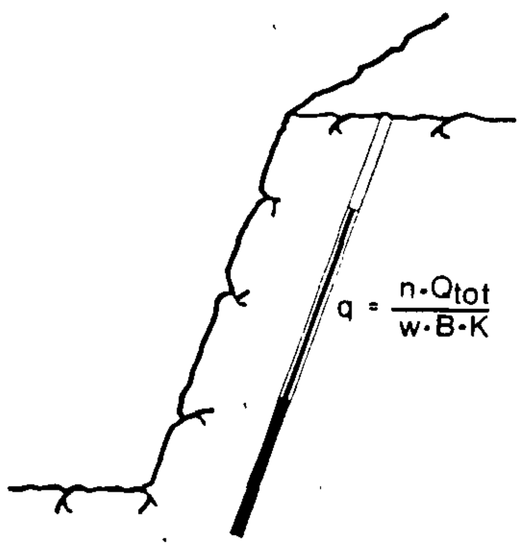
for quarries and open pit mines.

In road cuts etc. where blasting is performed within a limited area, the specific charge is calculated per row:

$$q = \frac{n \times Q_{tot}}{w \times B \times K} \qquad \text{(kg/cu.m.)}$$

where w is the width of the round.

The value of the specific charge will be higher due to the influence of the edge holes.



EXAMPLE SHOWING HOW THE CALCULATION FOR BENCH BLASTING IS CARRIED OUT:

Conditions:

Bench height:	$K = 15$ m
Width of the round:	$w = 26$ m
Blasthole diameter:	$d = 76$ mm
Rock constant:	$c = 0.4$
Hole inclination:	3:1
Explosive:	Emulite 150 in 65 mm plastic hoses dropt into the hole.
Charging condition:	Dry holes

Calculation of drilling pattern.

1. Maximum burden.

$B_{max} = 1.45\sqrt{l_b}$

Charge concentration, l_b, is found in table 1a and is in this case 5.0 kg/m. No correction for hole inclination or rock constant.

$B_{max} = 1.45\sqrt{5.0} = 3.24$ m

2. Subdrilling.

$U = 0.3 \times B_{max}$ $\qquad$ $U = 0.3 \times 3.24 = 0.97$ m

3. Depth of blasthole.

$H = 1.05(K+U)$ $\qquad$ $H = 1.05(15.0+0.97) = 16.76$ m

4. Error in drilling.

$$E = \frac{d}{1000} + 0.03 \times H \qquad E = \frac{76}{1000} + 0.03 \times 16.76 = 0.58 \text{ m}$$

5. Practical burden.

$B = B_{max} - E$ $\qquad$ $B = 3.24 - 0.58 = 2.66$ m

6. Practical spacing.

$S = 1.25 \times B$ $\qquad$ $S = 1.25 \times 2.66 = 3.32$ m

7. Adjustment for width of the round.

$$\frac{w}{s} \qquad \frac{26.0}{3.32} = 7.83 = 8 \text{ spaces}$$

$$S_{adj} = \frac{\text{width}}{\text{No. of spaces/row}} \qquad S_{adj} = \frac{26.0}{8} = 3.25 \text{ m}$$

Note that the number of holes in a row is the number of spaces +1.

8. Specific drilling.

$$b = \frac{n \times H}{B \times K \times w} \qquad b = \frac{9 \times 16.76}{2.66 \times 15.0 \times 26.0} = 0.145 \text{ m/cu m.}$$

Calculation of charges.

9. Concentration of bottom charge.

$l_b = l_b$ for the determination of B_{max} In accordance with table 1 a. $\qquad l_b = 5.0$ kg/m

10. Height of the bottom charge.

$h_b = 1.3 \times B_{max} \qquad h_b = 1.3 \times 3.24 = 4.20$ m

11. Weight of bottom charge.

$Q_b = l_b \times h_b \qquad Q_b = 5.0 \times 4.20 = 21.0$ kg

12. Stemming.

$h_o = B \qquad h_o = 2.66$ m

13. Concentration of column charge.

$l_c = 40$ to 60% of $l_b \qquad l_c = 0.5 \times 5.0 = 2.50$ kg/m

14. Height of column charge.

$h_c = H - h_b - h_o \qquad h_c = 16.76 - 4.20 - 2.66 = 9.90$ m

15. Weight of column charge.

$Q_c = l_c \times h_c \qquad Q_c = 2.50 \times 9.90 = 24.75$ kg

16. Total charge.

$Q_{tot} = Q_b + Q_c \qquad Q_{tot} = 21.00 + 24.75 = 45.75$ kg

17. Specific charge.

$$q = \frac{n \times Q_{tot}}{B \times K \times w} \qquad q = \frac{9 \times 45.75}{2.66 \times 15.0 \times 26.0} = 0.40 \text{ kg/cu.m.}$$

If the blast is not limited to a certain area, the specific drilling and specific charge will be lower.
In the above example, the specific charge will then be:

$$q = \frac{Q_{tot}}{B \times S \times K} \qquad q = \frac{45.75}{2.66 \times 3.32 \times 15.0} = 0.35 \text{ kg/cu.m.}$$

In quarrying there is no need to adjust the spacing between the holes and number of holes in accordance with the width of the cut.

Summary of important data:

Bench height m	Hole depth m	Burden m	Spacing m	Bottom charge kg	Column charge kg	Specific drilling m/cu.m	Specific charge kg/cu.m
15.0	16.8	2.65	3.25	21.0	24.8	0.145	0.40

Drilling and charging tables.

The following tables give the computed key data for different blasthole diameters.

The tables give the values for bench heights higher than 2 × B_{max}, lower benches will be dealt with separately (See Leveling)

The first two tables give the key data for drill series 11 and 12. These series are developed mainly for manual drilling.
The series is a series of drill rods with carbide-tipped bits, increasing 0.8 m in length and decreasing 1 mm in diameter between one drill rod and the next one.

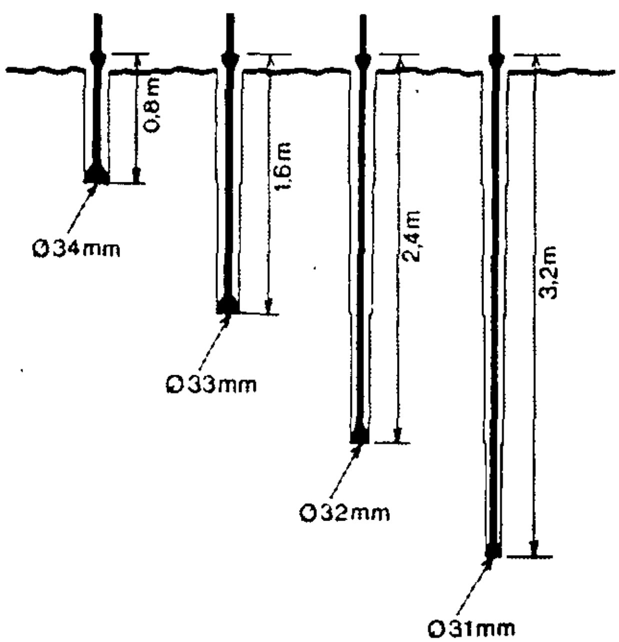


Fig. 5.5 Drill series 11.

The drilling and charging tables for Emulite 150 may also be used for Dynamex M

Drilling and charging table for drill series 11
Blasthole diameter 34 – 26 mm.

Explosive: Emulite 150
Hole inclination: 3:1

Bench height	K (m)	2.0	2.5	3.0	3.5	4.0	4.5	5.0	5.5
Hole diameter	d (mm)	31	31	30	29	29	28	28	27
Hole depth	H (m)	2.50	3.05	3.55	4.10	4.60	5.10	5.60	6.15
Practical burden	B (m)	1.10	1.20	1.15	1.10	1.10	1.05	1.00	0.95
Practical spacing	S (m)	1.35	1.50	1.45	1.40	1.35	1.30	1.25	1.20
Stemming	h_0 (m)	1.10	1.20	1.15	1.10	1.10	1.05	1.00	0.95
Bottom charge									
Concentration	l_b (kg/m)	0.87	0.87	0.81	0.76	0.76	0.71	0.71	0.66
Height	h_b (m)	1.40	1.70	1.70	1.65	1.65	1.60	1.60	1.55
Weight	Q_b (kg)	1.20	1.50	1.40	1.25	1.25	1.15	1.15	1.00
Column charge									
Concentration	l_c (m)	0.44	0.41	0.41	0.38	0.38	0.36	0.36	0.33
Height	h_c (m)	0.00	0.15	0.70	1.35	1.85	2.45	3.00	3.65
Weight	Q_c (kg)	0.00	0.10	0.30	0.50	0.70	0.90	1.05	1.20
Total charge	Q_{tot} (kg)	1.20	1.60	1.70	1.75	1.95	2.05	2.20	2.20
Specific drilling	b (m/cu m)	0.812	0.678	0.710	0.761	0.776	0.830	0.896	0.980
Specific charge	q (kg/cu m)	0.40	0.36	0.34	0.33	0.33	0.33	0.35	0.35

The reduction of the diameter of the blasthole for each drill rod used has to be taken into account in the drill and charge calculations.

Drilling and charging table for drill series 12
Blasthole diameter 40 – 29 mm

Explosive: Emulite 150
Hole inclination: 3:1

Bench height	K (m)	3.0	3.5	4.0	4.5	5.0	5.5	6.0	6.5
Hole diameter	d (mm)	36	35	35	34	33	33	32	32
Hole depth	H (m)	3.65	4.20	4.70	5.20	5.70	6.25	6.75	7.25
Practical burden	B (m)	1.40	1.35	1.35	1.30	1.25	1.20	1.15	1.15
Practical spacing	S (m)	1.75	1.75	1.70	1.60	1.55	1.50	1.45	1.40
Stemming	h_0 (m)	1.40	1.35	1.35	1.30	1.25	1.20	1.15	1.15
Bottom charge									
Concentration	l_b (kg/m)	1.17	1.11	1.11	1.04	0.98	0.98	0.92	0.92
Height	h_b (m)	2.00	2.00	2.00	1.90	1.90	1.90	1.80	1.80
Weight	Q_b (kg)	2.30	2.20	2.20	2.00	1.90	1.90	1.70	1.70
Column charge:									
Concentration	l_c (m)	0.59	0.56	0.56	0.52	0.49	0.49	0.46	0.46
Height	h_c (m)	0.25	0.85	1.35	2.00	2.55	3.15	3.80	4.30
Weight	Q_c (kg)	0.15	0.50	0.75	1.05	1.25	1.55	1.75	2.00
Total charge	Q_{tot} (kg)	2.45	2.70	2.95	3.05	3.15	3.45	3.45	3.70
Specific drilling	b (m/cu m)	0.497	0.508	0.511	0.556	0.589	0.631	0.675	0.700
Specific charge	q (kg/cu m)	0.33	0.33	0.33	0.33	0.33	0.35	0.35	0.36

If excavation is not carried out between the rounds, it may be necessary to increase the specific charge.
This can be done either by denser drilling or by increasing the concentration of the column charge.
Normally the latter alternative will suffice.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter 51 mm

Explosive: Emulite 150
Hole inclination 3:1

Bench height	K (m)	4.0	4.5	5.0	5.5	6.0	6.5	7.0	7.5
Hole diameter	d (mm)	51	51	51	51	51	51	51	51
Hole depth	H (m)	4.90	5.40	5.90	6.50	7.00	7.50	8.00	8.60
Practical burden	B (m)	2.00	2.00	1.95	1.95	1.90	1.90	1.90	1.90
Practical spacing	S (m)	2.50	2.45	2.45	2.40	2.40	2.40	2.35	2.35
Stemming	h_0 (m)	2.00	2.00	1.95	1.95	1.90	1.90	1.90	1.90
Bottom charge:									
Concentration	l_b (kg/m)	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30
Height	h_b (m)	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90
Weight	Q_b (kg)	6.70	6.70	6.70	6.70	6.70	6.70	6.70	6.70
Column charge									
Concentration	l_c (m)	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15
Height	h_c (m)	0.00	0.50	1.05	1.65	2.20	2.70	3.20	3.80
Weight	Q_c (kg)	0.00	0.60	1.20	1.90	2.50	3.10	3.70	4.40
Total charge	Q_{tot} (kg)	6.70	7.30	7.90	8.60	9.20	9.80	10.40	11.10
Specific drilling	b (m/cu m)	0.245	0.245	0.247	0.252	0.256	0.253	0.256	0.257
Specific charge	q (kg/cu m)	0.33	0.33	0.33	0.33	0.33	0.33	0.33	0.33

Drilling and charging table for blasthole diameter 64 mm

Explosive: Emulite 150
Hole inclination 3:1

Bench height	K (m)	5.0	6.0	7.0	8.0	9.0	10.0	11.0	12.0
Hole diameter	d (mm)	64	64	64	64	64	64	64	64
Hole depth	H (m)	6.10	7.20	8.20	9.30	10.30	11.40	12.40	13.50
Practical burden	B (m)	2.55	2.50	2.45	2.45	2.40	2.40	2.35	2.30
Practical spacing	S (m)	3.15	3.15	3.10	3.05	3.00	2.95	2.95	2.90
Stemming	h_0 (m)	2.55	2.50	2.45	2.45	2.40	2.40	2.35	2.30
Bottom charge:									
Concentration	l_b (kg/m)	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70
Height	h_b (m)	3.60	3.60	3.60	3.60	3.60	3.60	3.60	3.60
Weight	Q_b (kg)	13.30	13.30	13.30	13.30	13.30	13.30	13.30	13.30
Column charge:									
Concentration	l_c (m)	1.85	1.85	1.85	1.85	1.85	1.85	1.85	1.85
Height	h_c (m)	0.00	1.10	2.15	3.25	4.30	5.40	6.45	7.60
Weight	Q_c (kg)	0.00	2.00	4.00	6.00	8.00	10.00	12.00	14.00
Total charge	Q_{tot} (kg)	13.30	15.30	17.30	19.30	21.30	23.30	25.30	27.30
Specific drilling	b (m/cu m)	0.152	0.152	0.154	0.156	0.159	0.161	0.162	0.169
Specific charge	q (kg/cu.m)	0.33	0.32	0.32	0.32	0.33	0.33	0.33	0.34

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter 76 mm

Explosive: Emulite 150
Hole inclination 3:1

Bench height	K (m)	6.0	8.0	10.0	12.0	14.0	15.0	16.0	18.0
Hole diameter	d (mm)	76	76	76	76	76	76	76	76
Hole depth	H (m)	7.30	9.40	11.50	13.60	15.70	16.80	17.80	19.90
Practical burden	B (m)	2.95	2.85	2.80	2.75	2.70	2.65	2.60	2.55
Practical spacing	S (m)	3.65	3.60	3.55	3.45	3.35	3.30	3.30	3.20
Stemming	h_0 (m)	2.95	2.85	2.80	2.75	2.70	2.65	2.60	2.55
Bottom charge									
Concentration	l_b (kg/m)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Height	h_b (m)	4.20	4.20	4.20	4.20	4.20	4.20	4.20	4.20
Weight	Q_b (kg)	21.00	21.00	21.00	21.00	21.00	21.00	21.00	21.00
Column charge									
Concentration	l_c (m)	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50
Height	h_c (m)	0.15	2.35	4.50	6.65	8.80	9.95	11.00	13.15
Weight	Q_c (kg)	0.40	5.90	11.30	16.60	22.00	24.90	27.50	32.90
Total charge	Q_{tot} (kg)	21.40	26.90	32.30	37.60	43.00	45.90	48.50	53.90
Specific drilling	b (m/cu m)	0.113	0.115	0.116	0.119	0.124	0.128	0.130	0.135
Specific charge	q (kg/cu m)	0.33	0.33	0.33	0.33	0.34	0.35	0.35	0.37

Drilling and charging table for blasthole diameter 89 mm.

Explosive Emulite 150
Hole inclination 3:1

Bench height	K (m)	8.0	10.0	12.0	14.0	15.0	16.0	18.0	20.0
Hole diameter	d (mm)	89	89	89	89	89	89	89	89
Hole depth	H (m)	9.60	11.70	13.80	15.90	17.00	18.00	20.10	22.20
Practical burden	B (m)	3.45	3.40	3.35	3.30	3.25	3.20	3.15	3.10
Practical spacing	S (m)	4.30	4.30	4.20	4.10	4.10	4.05	4.00	3.90
Stemming	h_0 (m)	3.45	3.40	3.35	3.30	3.25	3.20	3.15	3.10
Bottom charge.									
Concentration	l_b (kg/m)	7.10	7.10	7.10	7.10	7.10	7.10	7.10	7.10
Height	h_b (m)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Weight	Q_b (kg)	35.50	35.50	35.50	35.50	35.50	35.50	35.50	35.50
Column charge									
Concentration	l_c (m)	3.55	3.55	3.55	3.55	3.55	3.55	3.55	3.55
Height	h_c (m)	1.15	3.30	5.45	7.60	8.75	9.80	11.95	14.10
Weight	Q_c (kg)	4.10	11.70	19.40	27.00	31.00	34.80	42.40	50.00
Total charge	Q_{tot} (kg)	39.60	47.20	54.90	62.50	66.50	70.30	77.90	85.50
Specific drilling	b (m/cu m)	0.081	0.081	0.082	0.084	0.085	0.087	0.089	0.092
Specific charge	q (kg/cu.m)	0.33	0.33	0.33	0.33	0.33	0.34	0.34	0.35

EXAMPLE OF CHARGE CALCULATION FOR BENCH BLASTING WITH ANFO IN VERTICAL HOLES.

Conditions:

Bench height:	$K = 18$ m
Width of the round:	$w = 40$ m
Blasthole diameter:	$d = 102$ mm
Hole inclination:	Vertical
Explosive:	ANFO, poured into the blasthole
Charging condition:	Dry holes

Calculation of drilling pattern.

1. Maximum burden.

$B_{max} = 1.36\sqrt{l_b} \times R_1$

Charge concentration, l_b, is found in table 1a and is in this case 6.5 kg/m. Correction for vertical drilling is found in table 2 and is 0.95. No correction for rock constant.
$B_{max} = 1.36 \times \sqrt{6.5} \times 0.95 = 3.29$ m

2. Subdrilling.

$U = 0.3 \times B_{max}$ $\qquad$ $U = 0.3 \times 3.29 = 0.99$ m

3. Depth of blasthole.

$H = K + U$ $\qquad$ $H = 18.0 + 0.99 = 19.00$ m

4. Error in drilling.

$$E = \frac{d}{1000} + 0.03 \times H \qquad E = \frac{102}{1000} + 0.03 \times 19.00 = 0.67 \text{ m}$$

5. Practical burden.

$B = B_{max} - E$ $\qquad$ $B = 3.29 - 0.67 = 2.62$ m

6. Practical spacing.

$S = 1.25 \times B$ $\qquad$ $S = 1.25 \times 2.62 = 3.27$ m

7. Adjustment for width of round.

$$\frac{w}{s} \qquad \frac{40}{3.27} = 12.23 = 13 \text{ spaces}$$

$$S_{adj} = \frac{\text{Width}}{\text{No. of spaces/row}} \qquad S_{adj} = \frac{40}{13} = 3.07 \text{ m}$$

Note that the number of holes in a row is the number of spaces +1.

8. Specific drilling.

$$b = \frac{n \times H}{B \times K \times w} \qquad b = \frac{14 \times 19.00}{2.62 \times 18 \times 40} = 0.141 \text{ m/cu.m.}$$

Calculation of charge.

In the case of ANFO the charge cannot be divided into bottom charge and column charge as it consists of only one column of charge with the same charge concentration

9. Stemming.

$h_o = B \qquad h_o = 2.62$ m

10. Charge concentration.

$l_b = l_b$ for determination of B_{max}. According to table 1a. $\qquad l_b = 6.5$ kg/m

11. Height of charge.

$h = H - h_o \qquad h = 19.00 - 2.62 = 16.38$ m

12. Weight of charge.

$Q = l_b \times h \qquad Q = 6.5 \times 16.38 = 106.5$ kg

13. Specific charge.

$$q = \frac{n \times Q}{B \times K \times w} \qquad q = \frac{14 \times 106.5}{2.62 \times 18 \times 40} = 0.79 \text{ kg/cu.m.}$$

Summary of important data.

Bench height m	Hole depth m	Burden m	Spacing m	Charge kg	Specific drilling m/cu.m.	Specific charge kg/cu.m.
18.0	19.0	2.62	3.07	106.5	0.141	0.79

The high specific charge depends to a large extent on the overcharged column part of the blast.

Drilling and charging tables for blastholes charged with ANFO.

For hole inclinations other than 3:1, the correct hole burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 64 mm

Explosive: ANFO

Hole inclination: 3:1

Bench height	K (m)	6.0	7.0	8.0	9.0	10.0	11.0	12.0	14.0
Hole diameter	d (mm)	64	64	64	64	64	64	64	64
Hole depth	H (m)	7.00	8.00	9.10	10.10	11.20	12.20	13.30	15.40
Practical burden	B (m)	1.90	1.90	1.85	1.80	1.80	1.75	1.70	1.65
Practical spacing	S (m)	2.40	2.35	2.30	2.30	2.25	2.20	2.20	2.10
Stemming	h_0 (m)	1.90	1.90	1.85	1.80	1.80	1.75	1.70	1.65
Charge ANFO									
Concentration	l_b (kg/m)	2.60	2.60	2.60	2.60	2.60	2.60	2.60	2.60
Height	h (m)	4.60	5.60	6.75	7.80	8.90	9.95	11.10	13.25
Weight	Q (kg)	12.00	14.60	17.60	20.30	23.10	25.90	28.90	34.50
Primer Emulite 150	50×550 mm	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30
Total charge	Q_{tot} (kg)	13.30	15.90	18.90	21.60	24.40	27.20	30.20	35.80
Specific drilling	b (m/cu m)	0.256	0.256	0.267	0.271	0.277	0.288	0.296	0.317
Specific charge	q (kg/cu m)	0.49	0.51	0.56	0.58	0.60	0.64	0.67	0.74

The specific charge increases with the hole depth as the uncharged part of the hole (stemming) is long compared to the bench height in lower benches.
The use of primer will not effect the burden or spacing unless the primer is of considerable size.

Drilling and charging table for blasthole diameter of 76 mm

Explosive: ANFO

Hole inclination: 3:1

Bench height	K (m)	8.0	10.0	12.0	14.0	15.0	16.0	18.0	20.0
Hole diameter	d (mm)	76	76	76	76	76	76	76	76
Hole depth	H (m)	9.20	11.30	13.40	15.50	16.60	17.60	19.70	21.80
Practical burden	B (m)	2.20	2.20	2.10	2.05	2.00	2.00	1.90	1.85
Practical spacing	S (m)	2.80	2.70	2.65	2.55	2.50	2.45	2.40	2.30
Stemming	h_0 (m)	2.20	2.20	2.10	2.05	2.00	2.00	1.90	1.85
Charge, ANFO									
Concentration	l_b (kg/m)	3.60	3.60	3.60	3.60	3.60	3.60	3.60	3.60
Height	h (m)	6.50	8.60	10.80	12.95	14.10	15.10	17.30	19.45
Weight	Q (kg)	23.40	31.00	38.90	46.60	50.80	54.40	62.30	70.00
Primer Emulite 150,	65×550 mm	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total charge	Q_{tot} (kg)	25.60	33.20	41.10	48.80	53.00	56.60	64.50	72.20
Specific drilling	b (m/cu m)	0.187	0.190	0.200	0.212	0.221	0.224	0.240	0.256
Specific charge	q (kg/cu m)	0.52	0.56	0.61	0.67	0.71	0.72	0.79	0.85

Drilling and charging tables for blastholes charged with ANFO.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 89 mm

Explosive ANFO
Hole inclination: 3:1

Bench height	K (m)	8.0	10.0	12.0	14.0	15.0	16.0	18.0	20.0
Hole diameter	d (mm)	89	89	89	89	89	89	89	89
Hole depth	H (m)	9.40	11.50	13.60	15.70	16.70	17.80	19.90	22.00
Practical burden	B (m)	2.70	2.60	2.55	2.50	2.45	2.40	2.35	2.30
Practical spacing	S (m)	3.30	3.30	3.15	3.10	3.10	3.00	2.95	2.85
Stemming	h_0 (m)	2.70	2.60	2.55	2.50	2.45	2.40	2.35	2.30
Charge: ANFO									
Concentration	l_c (kg/m)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Height	h (m)	6.20	8.40	10.55	12.70	13.75	14.90	17.05	19.20
Weight	Q (kg)	31.00	42.00	52.80	63.50	68.70	74.50	85.30	96.00
Primer: Emulite 150,	65×550 mm	2.20	2.20	2.20	2.20	2.20	2.20	2.20	2.20
Total charge	Q_{tot} (kg)	33.20	44.20	55.00	65.70	70.90	76.70	87.50	98.20
Specific drilling	b (m/cu.m)	0.132	0.134	0.141	0.145	0.147	0.155	0.159	0.168
Specific charge	q (kg/cu.m)	0.47	0.52	0.57	0.61	0.62	0.67	0.70	0.75

Drilling and charging table for blasthole diameter of 102 mm

Explosive ANFO
Hole inclination: 3:1

Bench height	K (m)	10.0	12.0	14.0	15.0	16.0	18.0	20.0	22.0
Hole diameter	d (mm)	102	102	102	102	102	102	102	102
Hole depth	H (m)	11.60	13.70	15.80	16.80	17.90	20.00	22.10	24.20
Practical burden	B (m)	3.00	2.95	2.90	2.85	2.85	2.75	2.70	2.65
Practical spacing	S (m)	3.80	3.70	3.60	3.60	3.55	3.50	3.40	3.30
Stemming	h_0 (m)	3.00	2.95	2.90	2.85	2.85	2.75	2.70	2.65
Charge, ANFO:									
Concentration	l_c (kg/m)	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
Height	h (m)	8.10	10.25	12.40	13.45	14.55	16.75	18.90	21.05
Weight	Q (kg)	53.00	67.00	81.00	88.00	95.00	109.00	123.00	137.00
Primer: Emulite 150,	75×550 mm	2.70	2.70	2.70	2.70	2.70	2.70	2.70	2.70
Total charge	Q_{tot} (kg)	55.70	69.70	83.70	90.70	97.70	111.70	125.70	139.70
Specific drilling	b (m/cu m)	0.102	0.104	0.108	0.109	0.111	0.115	0.120	0.126
Specific charge	q (kg/cu m)	0.49	0.53	0.57	0.59	0.60	0.64	0.68	0.73

Drilling an charging tables for blastholes charged with ANFO

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 127 mm

Explosive: ANFO
Hole inclination 3:1

Bench height	K (m)	10.0	12.0	14.0	15.0	16.0	18.0	20.0	22.0
Hole diameter	d (mm)	127	127	127	127	127	127	127	127
Hole depth	H (m)	11.90	14.00	16.10	17.10	18.20	20.30	22.40	24.50
Practical burden	B (m)	3.85	3.80	3.70	3.70	3.65	3.60	3.50	3.45
Practical spacing	S (m)	4.80	4.70	4.65	4.60	4.55	4.50	4.40	4.35
Stemming	h_0 (m)	3.85	3.80	3.70	3.70	3.65	3.60	3.50	3.45
Charge, ANFO									
Concentration	l_c (kg/m)	10.10	10.10	10.10	10.10	10.10	10.10	10.10	10.10
Height	h (m)	7.55	9.70	11.90	12.90	14.05	16.20	18.40	20.55
Weight	Q (kg)	76.00	98.00	120.00	130.00	142.00	164.00	186.00	208.00
Primer Emulite 150	75×550 mm	2.70	2.70	2.70	2.70	2.70	2.70	2.70	2.70
Total charge	Q_{tot} (kg)	78.70	100.70	122.70	132.70	144.70	166.70	188.70	210.70
Specific drilling	b (m/cu m)	0.064	0.065	0.067	0.067	0.069	0.070	0.073	0.074
Specific charge	q (kg/cu m)	0.43	0.47	0.51	0.52	0.54	0.57	0.61	0.64

Drilling and charging table for blasthole diameter of 152 mm

Explosive: ANFO
Hole inclination 3:1

Bench height	K (m)	12.0	14.0	15.0	16.0	18.0	20.0	22.0	24.0
Hole diameter	d (mm)	152	152	152	152	152	152	152	152
Hole depth	H (m)	14.20	16.30	17.40	18.40	20.50	22.60	24.70	26.80
Practical burden	B (m)	4.60	4.55	4.50	4.45	4.40	4.35	4.30	4.20
Practical spacing	S (m)	5.75	5.65	5.60	5.60	5.50	5.45	5.35	5.30
Stemming	h_0 (m)	4.60	4.55	4.50	4.45	4.40	4.35	4.30	4.20
Charge, ANFO									
Concentration	l_c (kg/m)	14.50	14.50	14.50	14.50	14.50	14.50	14.50	14.50
Height	h (m)	9.10	11.25	12.40	13.45	15.60	17.75	19.90	22.10
Weight	Q (kg)	132.00	163.00	180.00	195.00	226.00	257.00	289.00	320.00
Primer 2×Emulite 150,	75×550 mm	5.40	5.40	5.40	5.40	5.40	5.40	5.40	5.40
Total charge	Q_{tot} (kg)	137.40	168.40	185.40	200.40	231.40	262.40	294.40	325.40
Specific drilling	b (m/cu m)	0.045	0.045	0.046	0.046	0.047	0.048	0.049	0.050
Specific charge	q (kg/cu m)	0.43	0.47	0.49	0.50	0.53	0.55	0.58	0.61

The main problem when blasting with ANFO is the breakage in the bottom part of the blast as the bottom part of the blasthole, more often than not contains water. If the ANFO is contained in plastic hoses, the diameter of the explosives column will be reduced and the charge concentration will not be big enough to break the constricted bottom part. If the ANFO is poured into the wet blasthole it will deteriorate rapidly and the effect will be the same.

A shortened bottom charge to reinforce the priming of the ANFO has shown to be effective. The height of the reinforced priming should be $0.4 \times B_{max}$, giving a bottom charge which is extended above the theoretical grade. The reinforced primer makes it possible to increase the burden and spacing with 7 % each. The savings in costs of drilling and secondary blasting of the toe are sufficient to justify the higher consumption of high explosives in the blast.

The following charging tables give the guide lines for blasting with Emulite 150 as reinforced primer and ANFO as column charge.

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 64 mm

Explosive: Reinforced primer Emulite 150, height $0.4 \times B_{max}$
Column charge ANFO

Hole inclination: 3:1

Bench height	K (m)	6.0	7.0	8.0	9.0	10.0	11.0	12.0	14.0
Hole diameter	d (mm)	64	64	64	64	64	64	64	64
Hole depth	H (m)	7.10	8.10	9.20	10.20	11.30	12.30	13.40	15.50
Practical burden	B (m)	2.10	2.05	2.05	2.00	1.95	1.95	1.90	1.85
Practical spacing	S (m)	2.65	2.60	2.55	2.50	2.50	2.40	2.40	2.30
Stemming	h_0 (m)	2.10	2.05	2.05	2.00	1.95	1.95	1.90	1.85
Primer Emulite 150:									
Concentration	l_b (kg/m)	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70
Height	h_b (m)	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.20
Weight	Q_b (kg)	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40
Charge, ANFO									
Concentration	l_c (kg/m)	2.60	2.60	2.60	2.60	2.60	2.60	2.60	2.60
Height	h_c (m)	3.80	4.85	5.95	7.00	8.15	9.15	10.30	12.45
Weight	Q_c (kg)	9.90	12.60	15.50	18.20	21.20	23.80	26.80	32.40
Total charge	Q_{tot} (kg)	14.30	17.00	19.90	22.60	25.60	28.20	31.20	36.80
Specific drilling	b (m/cu m)	0.213	0.217	0.220	0.227	0.232	0.239	0.245	0.260
Specific charge	q (kg/cu m)	0.43	0.46	0.48	0.50	0.53	0.55	0.57	0.62

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 76 mm

Explosive	Reinforced primer Emulite 150, height $0.4 \times B_{max}$ Column charge ANFO								
Hole inclination	3:1								
Bench height	K (m)	8.0	10.0	12.0	14.0	15.0	16.0	18.0	20.0
Hole diameter	d (mm)	76	76	76	76	76	76	76	76
Hole depth	H (m)	9.30	11.40	13.50	15.60	16.65	17.70	19.80	21.90
Practical burden	B (m)	2.45	2.40	2.30	2.25	2.20	2.20	2.10	2.05
Practical spacing	S (m)	3.05	2.95	2.90	2.80	2.75	2.70	2.65	2.50
Stemming	h_0 (m)	2.45	2.40	2.30	2.25	2.20	2.20	2.10	2.05
Primer Emulite 150									
Concentration	l_b (kg/m)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Height	h_b (m)	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35
Weight	Q_b (kg)	6.80	6.80	6.80	6.80	6.80	6.80	6.80	6.80
Charge, ANFO									
Concentration	l_c (kg/m)	3.60	3.60	3.60	3.60	3.60	3.60	3.60	3.60
Height	h_c (m)	5.50	7.65	9.85	12.00	13.10	14.15	16.35	18.50
Weight	Q_c (kg)	19.80	27.50	35.50	43.20	47.20	51.00	59.90	66.60
Total charge	Q_{tot} (kg)	26.60	34.30	42.30	50.00	54.00	57.80	66.70	73.40
Specific drilling	b (m/cu m)	0.156	0.161	0.169	0.177	0.184	0.186	0.198	0.214
Specific charge	q (kg/cu m)	0.45	0.48	0.53	0.57	0.60	0.61	0.67	0.72

Drilling and charging table for blasthole diameter of 89 mm

Explosive	Reinforced primer Emulite 150, height $0.4 \times B_{max}$ Column charge ANFO								
Hole inclination	3:1								
Bench height	K (m)	8.0	10.0	12.0	14.0	15.0	16.0	18.0	20.0
Hole diameter	d (mm)	89	89	89	89	89	89	89	89
Hole depth	H (m)	9.50	11.60	13.70	15.80	16.80	17.90	20.00	22.10
Practical burden	B (m)	2.90	2.85	2.80	2.75	2.70	2.65	2.60	2.50
Practical spacing	S (m)	3.70	3.60	3.50	3.40	3.35	3.35	3.20	3.15
Stemming	h_0 (m)	2.90	2.85	2.80	2.75	2.70	2.65	2.60	2.50
Primer Emulite 150									
Concentration	l_b (kg/m)	7.10	7.10	7.10	7.10	7.10	7.10	7.10	7.10
Height	h_b (m)	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60
Weight	Q_b (kg)	11.40	11.40	11.40	11.40	11.40	11.40	11.40	11.40
Charge, ANFO									
Concentration	l_c (kg/m)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Height	h_c (m)	5.00	7.15	9.30	11.45	12.50	13.65	15.80	18.00
Weight	Q_c (kg)	25.00	35.80	46.50	57.30	62.50	68.30	79.00	90.00
Total charge	Q_{tot} (kg)	36.40	47.20	57.90	68.70	73.90	79.70	90.40	101.40
Specific drilling	b (m/cu m)	0.111	0.113	0.117	0.121	0.124	0.126	0.134	0.140
Specific charge	q (kg/cu m)	0.42	0.46	0.49	0.52	0.54	0.56	0.60	0.64

For hole inclinations other than 3:1, the correct burden B and spacing S are obtained by multiplying by the appropriate reduction factor in table 2, page 69.

Drilling and charging table for blasthole diameter of 102 mm

Explosive: Reinforced primer Emulite 150, height 0.4 × B
Column charge ANFO

Hole inclination: 3:1

Bench height	K (m)	10.0	12.0	14.0	15.0	16.0	18.0	20.0	22.0
Hole diameter	d (mm)	102	102	102	102	102	102	102	102
Hole depth	H (m)	11.70	13.80	15.90	17.00	18.00	20.10	22.20	24.20
Practical burden	B (m)	3.30	3.25	3.20	3.15	3.10	3.05	3.00	2.90
Practical spacing	S (m)	4.15	4.05	4.00	3.90	3.90	3.80	3.70	3.60
Stemming	h_0 (m)	3.30	3.25	3.20	3.15	3.10	3.05	3.00	2.90
Primer Emulite 150									
Concentration	l_b (kg/m)	9.30	9.30	9.30	9.30	9.30	9.30	9.30	9.30
Height	h_b (m)	1.85	1.85	1.85	1.85	1.85	1.85	1.85	1.85
Weight	Q_b (kg)	17.20	17.20	17.20	17.20	17.20	17.20	17.20	17.20
Charge ANFO									
Concentration	l_c (kg/m)	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50
Height	h_c (m)	6.55	8.70	10.85	12.00	13.05	15.20	17.35	19.45
Weight	Q_c (kg)	42.60	56.60	70.60	78.00	85.00	99.00	113.00	126.00
Total charge	Q_{tot} (kg)	59.80	73.80	87.80	95.20	102.20	116.20	130.20	143.20
Specific drilling	b (m/cu m)	0.085	0.087	0.089	0.092	0.093	0.096	0.100	0.105
Specific charge	q (kg/cu m)	0.44	0.47	0.49	0.52	0.53	0.56	0.59	0.62

Drilling and charging table for blasthole diameter of 127 mm

Explosive: Reinforced primer Emulite 150, height 0.4 × B
Column charge ANFO

Hole inclination: 3:1

Bench height	K (m)	10.0	12.0	14.0	15.0	16.0	18.0	20.0	22.0
Hole diameter	d (mm)	127	127	127	127	127	127	127	127
Hole depth	H (m)	12.00	14.40	16.20	17.30	18.30	20.40	22.50	24.60
Practical burden	B (m)	4.20	4.15	4.10	4.05	4.00	3.95	3.90	3.80
Practical spacing	S (m)	5.25	5.20	5.10	5.05	5.00	4.90	4.80	4.75
Stemming	h_0 (m)	4.20	4.15	4.10	4.05	4.00	3.95	3.90	3.80
Primer Emulite 150:									
Concentration	l_b (kg/m)	14.40	14.40	14.40	14.40	14.40	14.40	14.40	14.40
Height	h_b (m)	2.30	2.30	2.30	2.30	2.30	2.30	2.30	2.30
Weight	Q_b (kg)	33.00	33.00	33.00	33.00	33.00	33.00	33.00	33.00
Charge ANFO									
Concentration	l_c (kg/m)	10.10	10.10	10.10	10.10	10.10	10.10	10.10	10.10
Height	h_c (m)	5.50	7.65	9.80	10.95	12.00	14.15	16.30	18.50
Weight	Q_c (kg)	56.00	77.00	99.00	111.00	121.00	143.00	165.00	187.00
Total charge	Q_{tot} (kg)	89.00	110.00	132.00	144.00	154.00	176.00	198.00	220.00
Specific drilling	b (m/cu m)	0.054	0.054	0.055	0.056	0.057	0.059	0.060	0.062
Specific charge	q (kg/cu m)	0.40	0.42	0.45	0.47	0.48	0.51	0.53	0.55

As can be observed in the charge calculations, more drilling and explosives are needed in the case of blasting with ANFO compared with blasting operations with Emulite 150

The higher consumption of explosives does not affect the overall economy of the blasting operation, as ANFO is an inexpensive blasting agent, but the increased use of accessories (cord, detonators, primers etc.) has to be considered.
The drilling cost will also increase considerably due to denser drilling pattern.

It is always a good habit to sit down and analyze the blasting operation before selecting the explosive, taking into consideration the following parameters.

* The explosive, cost per ton or cu m. of blasted rock	
* Detonators,	–"–
* Cord,	–"–
* Primers,	–"–
* Drilling,	–"–
* Blasting,	–"–
* Secondary blasting,	–"–
* Mucking,	–"–
* Hauling,	–"–
* Crushing,	–"–

Besides the above parameters, consideration has to be given to weather conditions and ground water levels. Blasting agents of ANFO type have poor water resistance properties

5.3 Low benches, leveling.

Leveling is the kind of blasting where the bench height is below $2\times B_{max}$, but normally applies to bench heights under 1.0 m.

The blastholes for leveling generally have a small diameter, drill series 11 (34, 33, 32 ... mm), or smaller

The design of the firing pattern is important in leveling. Due to the low bench heights and small burdens, the rock moves faster than in normal bench blasting, which calls for shorter delay times between the holes. If the delay time is too long, the protective effect of short delay firing will not occur, increasing the risk of flyrock

The risk of flyrock is however always great in leveling and this is why the round should be given a heavy cover and several layers of splinter protective covering on top. (See also Chapter 5.9 Covering.)

To reduce drilling costs, 22 mm blastholes are now more widely used. This is because blasting with small diameter holes decreases the drilling cost and also the explosives cost due to better utilization of energy of the explosive compared with the conventional technique
Using small diameter blastholes also reduces the risk of flyrock and ground vibrations.
For the charging of small diameter blastholes, Nitro Nobel produces a tube charge, Primex 17×150 mm which can be cut in suitable lengths.

Leveling with drill series 11.

Drilling and charging table for drillseries 11.
Blasthole diameters 34–26 mm

Explosive	Emulite 150						
Hole inclination	3:1						
Bench height	K (m)	0.20	0.30	0.40	0.60	0.80	1.00
Hole diameter	d (mm)	34	34	34	34	33	33
Hole depth	H (m)	0.60	0.60	0.60	0,90	1.10	1,40
Practical burden	B (m)	0.40	0.40	0,40	0.50	0.60	0,80
Practical spacing	S (m)	0.50	0.50	0.50	0.65	0.75	1,00
Bottom charge							
Concentration	l_b (kg/m)	1.00	1.00	1.00	1.00	1.00	1.00
Height	h_b (m)	0.05	0,05	0.05	0.10	0.20	0,40
Weight	Q_b (kg)	0.05	0.05	0.05	0.10	0.20	0,40
Column charge	Q_c (kg)	0.00	0.00	0.00	0.00	0.00	0,00
Total charge	Q_{tot} (kg)	0.05	0.05	0.05	0.10	0.20	0.40
Stemming	h_0 (m)	0.50	0,50	0,50	0.80	0.90	1,00
Specific drilling	b (m/sq m)	3.00	3.00	3.00	2.80	2.45	1.75
Specific charge	q (kg/sq m)	0.25	0.25	0.25	0.31	0.45	0,50
Specific charge	q' (kg/cu m)	1.25	0.83	0.63	0,51	0.56	0.50

The specific drilling and specific charge are calculated in relation to the blasted area expressed in m/sq m. and kg/sq m

The consumption of detonators and explosives is high in low bench blasting. The cost of drilling is high due to the close drilling pattern

If the bottom of the cut is not restricted to a certain level, it is economically favorable to increase the subdrilling and subsequently increase the spacing between the blastholes.

As a consequence of the deeper drilling, the risk of flyrock will be reduced.

A suitable drill depth is 1.6 m which is the length of the second drillrod in the integral series No. 11. The diameter of the drillbit is 33 mm.

Comparison of consumption of explosives and detonators as well as specific drilling for a bench height of 0.4 m when drilled conventionally and with increased subdrilling.

	Conventional	Increased subdrilling
Bench height	0.4 m	0.4 m
Hole depth	0.6 m	1.6 m
Practical burden	0.4 m	0.9 m
Practical spacing	0.5 m	1.1 m
Charge of explosives per hole	0.05 kg	0.5 kg
Charge of explosives per sq m	0.25 kg	0.5 kg
Number of detonators per sq m	5 pcs	1 pc
Drilling per sq.m.	3.0 m	1.6 m

The higher specific charge in the case of increased subdrilling is well compensated by the lesser consumption of detonators and the decreased specific drilling.

In certain cases larger blasthole sizes must be used for low benches due to availability of equipment and drillbits.

The following tables are recommendations for blasting of low benches with blasthole diameters of 51, 64 and 76 mm.

Drilling and charging table for low benches
Blasthole diameter 51 mm

Explosive: Emulite 150
Hole inclination: 3:1

Bench height	K (m)	1.00	1.50	2.00	2.50
Hole diameter	d (mm)	51	51	51	51
Hole depth	H (m)	1.40	2.00	2.60	3.20
Practical burden	B (m)	0.80	1.00	1.30	1.50
Practical spacing	S (m)	1.00	1.20	1.60	1.90
Bottomcharge					
Concentration	l_b (kg/m)	1.50	1.50	1.50	1.50
Height	h_b (m)	0.20	0.50	1.00	1.60
Weight	Q_b (kg)	0.30	0.75	1.50	2.40
Explosives dimension	(mm)	40	40	40	40
Column charge	Q_c (kg)	0.10	0.10	0.20	0.30
Total charge	Q_{tot} (kg)	0.40	0.85	1.70	2.70
Stemming	h_0 (m)	1.10	1.20	1.30	1.50
Specific drilling	b (m/cu m)	1.75	1.11	0.63	0.45
Specific charge	q (kg/cu m)	0.50	0.47	0.41	0.38

Drilling and charging table for low benches.
Blasthole diameter 64 mm

Explosive: Emulite 150
Hole inclination: 3:1

Bench height	K (m)	1.00	2.00	3.00	4.00
Hole diameter	d (mm)	64	64	64	64
Hole depth	H (m)	1.40	2.70	3.80	4.90
Practical burden	B (m)	0.80	1.30	1.60	2.10
Practical spacing	S (m)	1.00	1.60	2.00	2.60
Bottom charge					
Concentration	l_b (kg/m)	2.20	2.20	2.20	2.20
Height	h_b (m)	0.15	0.80	1.50	2.70
Weight	Q_b (kg)	0.30	1.80	3.30	6.00
Explosives dimension	(mm)	50	50	50	50
Column charge	Q_c (kg)	0.10	0.10	0.50	0.50
Total charge	Q_{tot} (kg)	0.40	1.90	3.80	6.50
Stemming	h_0 (m)	1.10	1.50	1.60	2.00
Specific drilling	b (m/cu m)	1.75	0.65	0.40	0.22
Specific charge	q (kg/cu m)	0.50	0.46	0.40	0.30

Drilling and charging table for low benches
Blasthole diameter 76 mm

Explosive	Emulite 150
Hole inclination:	3:1

Bench height	K (m)	1.00	2.00	3.00	4.00	5.00	6.00
Hole diameter	d (mm)	76	76	76	76	76	76
Hole depth	H (m)	1.60	2.60	3.80	5.00	6.20	7.40
Practical burden	B (m)	1.10	1.30	1.50	1.70	2.00	2.60
Practical spacing	S (m)	1.30	1.60	1.80	2.10	2.50	3.20
Bottom charge							
Concentration	l_b (kg/m)	1.51	1.50	2.40	2.40	3.50	4.50
Height	h_b (m)	0.38	1.00	1.10	2.00	2.00	3.00
Weight	Q_b (kg)	0.57	1.50	2.60	4.80	7.00	13.50
Explosives dimension	(mm)	40	40	50	50	60	70
Column charge	Q_c (kg)	0.00	0.20	0.60	0.80	3.00	4.60
Total charge	Q_{tot} (kg)	0.57	1.70	3.20	5.60	10.00	18.10
Stemming	h_o (m)	1.20	1.30	1.50	1.70	2.00	2.60
Specific drilling	b (m/cu m)	1.12	0.63	0.47	0.35	0.25	0.15
Specific charge	q (kg/cu m)	0.40	0.41	0.40	0.39	0.40	0.36

Leveling with mini-hole technique.

Nitro Nobel has developed a new technique for rock blasting in sensitive environments. The new technique, blasting with 22 mm blastholes and small charges, is presented in depth in Chapter 12.10 Mini-hole blasting.

5.4 Secondary blasting.

Secondary blasting means the treatment of boulders that are still considered big enough to cause obstruction in subsequent operations like excavation, transport and crushing

The handling of boulders is normally very expensive and the aim in all blasting operations must be to avoid secondary blasting. Careful planning and execution of a blast may decrease the need of secondary blasting to a minimum. (See Chapter 5 6 Fragmentation.)

As it is impossible to completely avoid boulders in blasting operations, the problem has to be taken care of by blasting.

The most widely used method of blasting boulders is by drilling one or more blastholes in them. The blasthole/s is/are drilled so that the explosive can be placed in the center of the mass of the boulder, which requires a hole slightly deeper than half the thickness of the boulder.

As boulders resulting from blasting have been exposed to stresses during the blast, they contain a lot of microscopic cracks, thus making them relatively easy to blast compared with natural stones. (See Chapter 12.1 Blasting of natural boulders.)

For blasting of boulders resulting from blasts, a specific charge of approx. 0 06 kg/cu.m. is needed

Secondary blasting.

Size of boulder cu.m.	Thickness of boulder m	Depth of blasthole m	Number of blastholes	Charge kg/hole
0 5	0 8	0 45	1	0.03
1.0	1 0	0 55	1	0.06
2 0	1 0	0 55	2	0.06
3 0	1 5	0 80	2	0.09

When several blastholes are used in a boulder, the initiation should be carried out with instantaneous detonators or detonators with the same interval number.

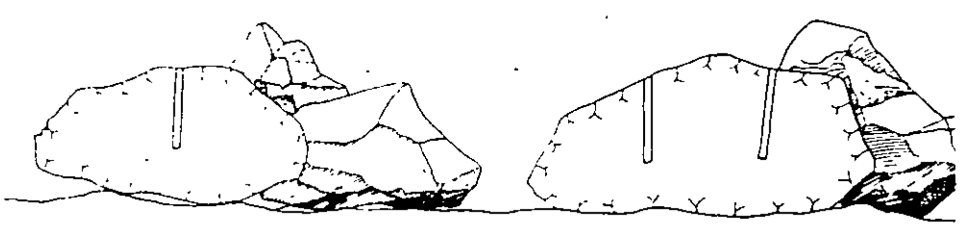


Fig 5 6 Secondary blasting.

5.5 Opening of the bench.

Generally, bench blasting is understood to be blasting of vertical, or close to vertical, blastholes towards a free face.
Occasionally, no free face is available and the conditions for bench blasting have to be created.

The simplest way to do this is by using a fan cut of the type shown below.

The drilling pattern and charging of the blasthole depend upon the blasthole diameter and explosive used. Guide values for drilling and charge calculations are found in Chapter 5.2 Charge calculations.

Note that the burden must be calculated in relation to the charge concentration in the bottom of the hole.
The fan cut creates a certain risk of flyrock. Care must therefore be taken close to inhabited areas, especially if the blasthole diameter is greater than 40 mm

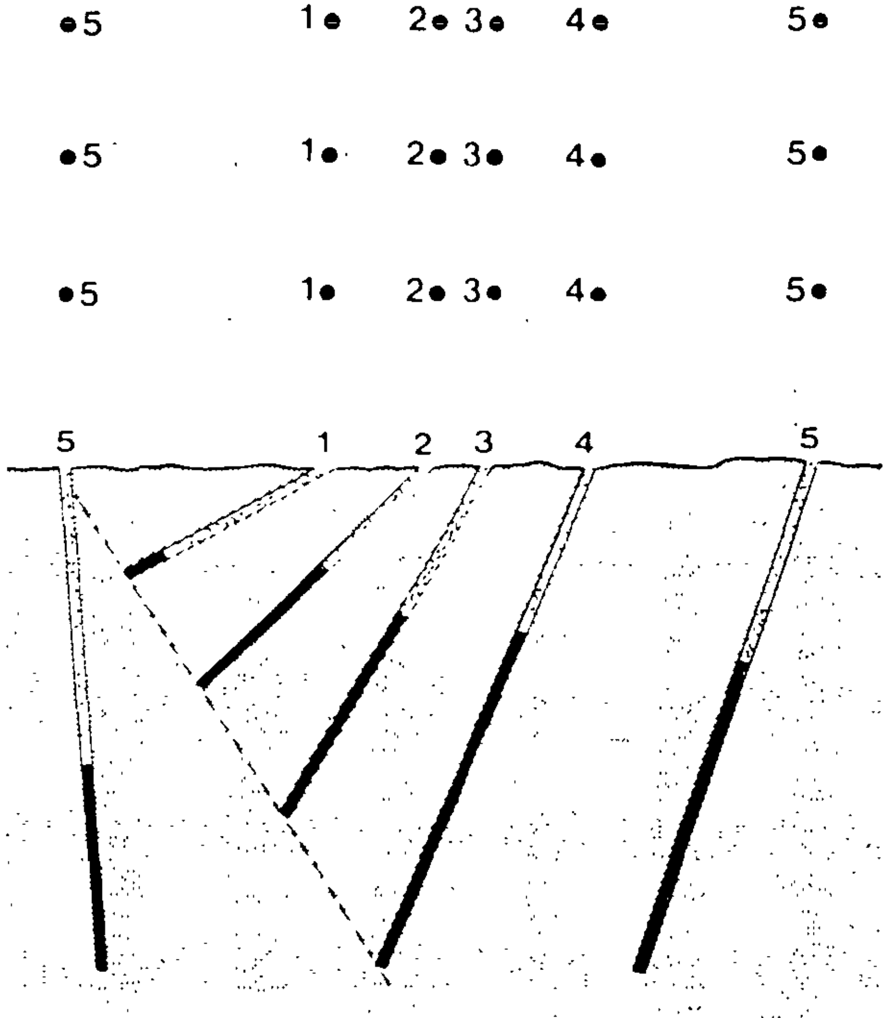


Fig 5.7 Opening of the bench

5.6 Rock fragmentation.

Fig. 5.8 Blast with good rock fragmentation, result of blast in Fig. 4.1.

5.6.1 Small rock fragmentation.

Good rock fragmentation is a subjective matter and depends on the end use of the rock.

The desired degree of fragmentation also depends upon of the type and size of equipment which is used for the subsequent handling of the rock.

Large loaders, trucks and crushers can allow larger fragmentation, but it is a common misconception that larger fragmentation can be allowed because large loading, transport and crushing equipment is used. The large size equipment is designed to handle large volumes of material, not large size material.

The ideally fragmented rock is the rock that needs no further treatment after the blast. Therefore, the parameters for the subsequent operations are the guide lines for deciding on the desired fragmentation of the rock. If the rock is just to be transported to a dumping area, it should be easy to load and transport. If the rock is intended for crushing, the size of the largest boulders should not exceed 75 per cent of the length of the shortest side of the opening of the primary crusher, thus allowing a free flow through the plant.

Since the size of the broken rock is of the utmost importance for the subsequent operations, all possible efforts have to be made to keep the size down.

In bench blasting, the fragmentation is influenced by the following factors:

* The geology of the rock (faults, voids etc.)
* Specific drilling
* Specific charge
* Drilling pattern
* Firing pattern
* Hole inclination
* Hole deviation
* Size of the round

By considering the above factors during the drilling and blasting operation, it is possible to influence the fragmentation. However, it is not possible to make a completely reliable calculation beforehand. Test blasting of some rows is a good way to obtain some impression of the blasting characteristics of the rock.

The **geology** of the rock frequently affects the fragmentation more than the explosive used in the blast. The properties that influence the result of the blast are compression strength, tensile strength, density, propagation velocity, hardness and structure.

Most rocks have a **tensile strength** which is 8 to 10 times lower than the **compression strength.** This property is an important factor in rock blasting. The rock's tensile strength has to be exceeded, otherwise the rock will not break.

Compression and tensile strengths of different rocks.

	Compression strength kg/sq.cm.	Tensile strength kg/sq.cm
Granite	2000–3600	100–300
Diabase	2900–4000	190–300
Marble	1500–1900	150–200
Limestone	1300–2000	170–300
Sandstone (hard)	approx. 3000	approx. 300

Rock with high **density** is normally harder to blast than a low density rock because the heavier rock masses require more explosives for the displacement of the rock.

The **propagation velocity** varies with different kinds of rock. Field tests have shown that hard rocks with high propagation velocity are best fragmented by an explosive with high velocity of detonation (VOD). In consequence a rock with low propagation velocity may be blasted with explosives with low VOD. EMULITE and DYNAMEX with a VOD of 5000 to 6000 m/sec. are suitable for blasting granite, marble and diabase (propagation velocity 4000 to 7000 m/sec.) while ANFO is suitable for limestone, sandstone etc with low propagation velocities.

The **hardness** or brittleness of the rock can have a great effect on the blasting result. Soft rock is more "forgiving" than hard rock. If soft rock is somewhat undercharged, it will still be muckable and if it is somewhat overcharged, excessive throw rarely occurs. On the other hand, undercharging of hard rock frequently results in a tight and blocky muckpile that is tough to excavate. Overcharging of hard rock may cause flyrock and airblast. The design of blasts in hard rock requires tighter control than in soft rock

Granite, gneiss and marble represent the hard rock while soft limestone and shale are considered soft

The structure of the rock should be documented before the blasting works start. The direction, severity and spacing between the joint sets should be mapped out so that drilling and firing patterns can be adjusted to the prevailing conditions.

The planning of the drilling with respect to the direction of the joints is very important

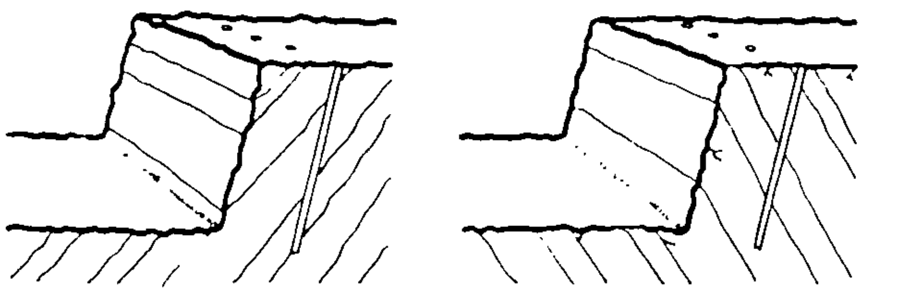


Fig. 5.9

Advantages
Effectively exploits the energy in the explosive. Good displacement of the blasted rock giving good digging conditions. Normally no stumps in the bottom part
(no toe-problem)

Disadvantages
Back break

Advantages
Reduced over break.

Disadvantages
Bad displacement with tighter muckpile. Risk for stumps in the bottom part. Overhang may occur in the back row

When the rock is full of faults and incompetent zones, much of the explosive's energy is lost in the faults instead of being used to break the rock. Alternate zones of competent and incompetent rock normally result in too blocky fragmentation. Higher specific charge will rarely correct this problem; it will only increase the risk of flyrock. The best way to lessen the problem is to use smaller blastholes with a closer drilling pattern in order to obtain better distribution of the explosives in the rock. The explosive charges should be concentrated in the competent rock while the faults and incompetent zones should be stemmed if possible.

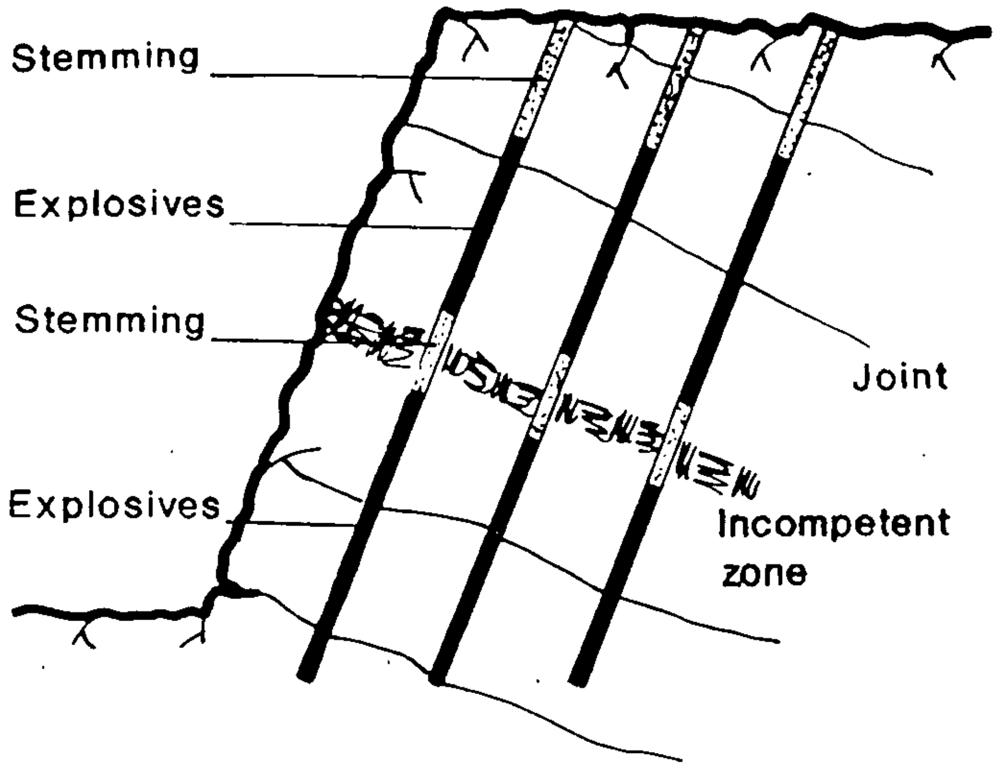


Fig. 5.10 Stemming of incompetent zones

The collar part of the blasthole, which contains the stemming, has an unfavorable effect on the rock fragmentation.
As a general rule a collar distance equal to the burden is left uncharged. In tough rock and rock with horizontal planes, the uncharged part of the hole will cause problems in form of an increased amount of boulders. To improve the blasting result, the following steps may be taken:

* Shorten the collar distance, thus charging higher in the blasthole.
* Drilling of relievers in the stemming section of the blast.

Higher charges in the blasthole can only be recommended when a large area can be evacuated and no delicate structures are in the vicinity of the blast, due to the increased risk of flyrock.
Occasionally a small charge in the stemming section of the blasthole can improve the result of the blast.

The drilling of relievers between the main blastholes helps to break the upper part of the round.

The relievers are short holes, normally with smaller diameter than the main blastholes.
The drilling of relievers between the blastholes does not usually make economic sense. More often than not, it is advisable to tolerate a certain amount of boulders from the blast and break them by secondary blasting or other method.

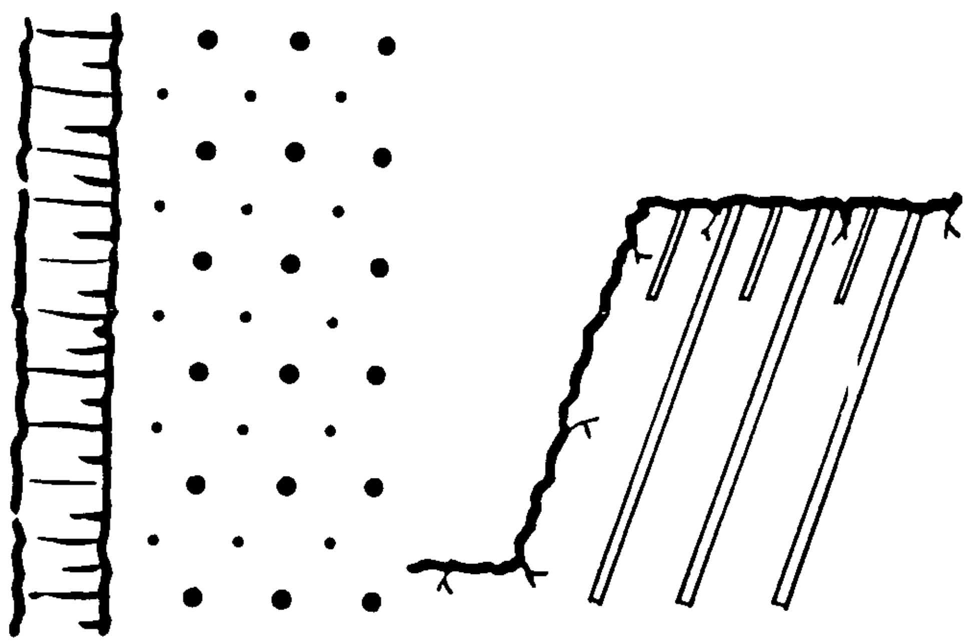


Fig. 5.11 Relievers between the main blastholes.

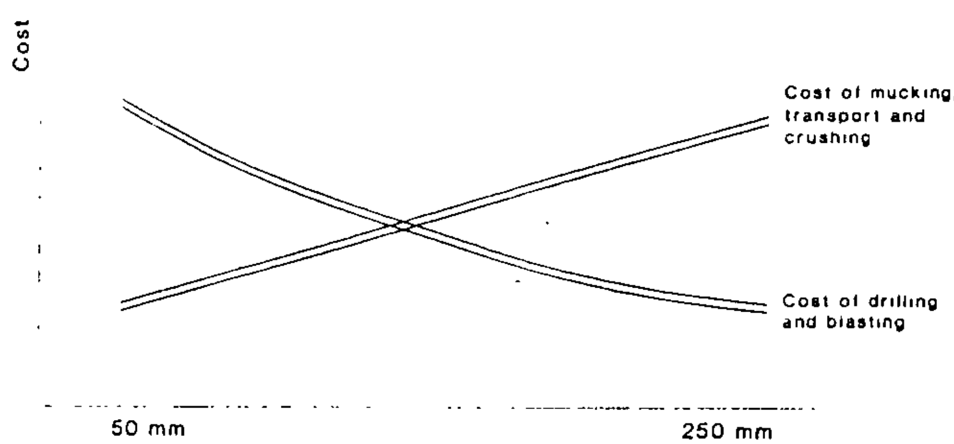


Fig. 5.12 Effect of small and large diameter blastholes on cost.

Specific drilling.

The size of the blasthole is the first consideration of any blast design. The blasthole diameter together with the explosive used will determine burden, spacing and hole depth

Practical hole diameters for bench blasting range from 30 mm to 400 mm.

Generally the cost of large diameter drilling is cheaper per cubic meter rock than small diameter drilling. Furthermore, cheaper blasting agents can be used in large diameter blastholes.

The large diameter blasthole pattern gives a relatively low drilling and blasting cost However, in geologically difficult situations the blasted material will be blocky, resulting in high mucking, transport and crushing cost as well as requiring more secondary breakage.

The geological structure is a major factor in determining the blasthole diameter. Joints and planes tend to isolate large blocks of rock in the burden area. The larger the drilling pattern, the greater the risk that these are left unbroken.

Higher specific drilling with smaller diameter blastholes distributes the explosives better in the rock resulting in better rock fragmentation

Lately, button bits have replaced insert bits to a great extent thanks to their excellent drilling characteristics and convenience in use (easier sharpening of the bit and longer intervals between the grindings). Added to the above advantages is a life span which is twice that of an insert bit

Because of the construction of the button bit, the diameter decreases with wear and when the bit is worn out, the diameter is up to 15 mm smaller than that of a new bit.

If the same burden and spacing is maintained during the entire life span of the bit, the fragmentation tends to be blockier at the end of the life span, due to the smaller diameter resulting in a smaller specific charge.

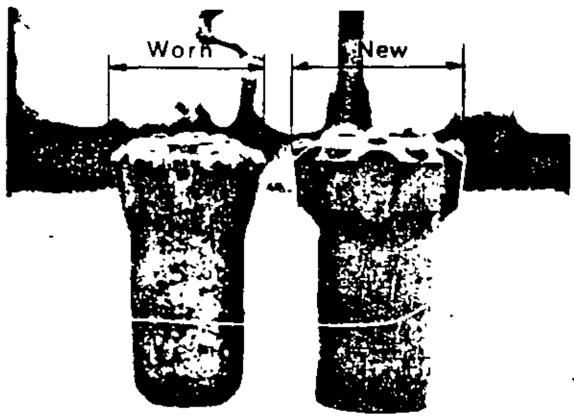


Fig. 5.13 Comparison of diameter of new and worn button bit

Specific charge.

Rock will be broken up more if the specific charge is increased and the drilling pattern maintained

The bottom part of the blast usually has the optimal specific charge and the fragmentation in this part is normally satisfactory.

The increase in specific charge can only be done in the column and stemming parts of the blast. The fragmentation will then be better but a greater forward movement of the rock has to be expected as well as an increased risk of flyrock.

Drilling pattern.

The typical drilling pattern has a spacing/burden ratio of 1.25 (S/B = 1.25), which has proved to give good rock fragmentation in multiple row blasting.

In the 70s, tests were carried out in Sweden with wide-space hole blasting with S/B ratios greater than 1.25. The results of the tests showed improved fragmentation up to an S/B ratio of 8. The method is now common practice in Swedish quarries.

The burden and spacing must be normal in the first row, otherwise the burden will be too small, increasing the risk of flyrock.

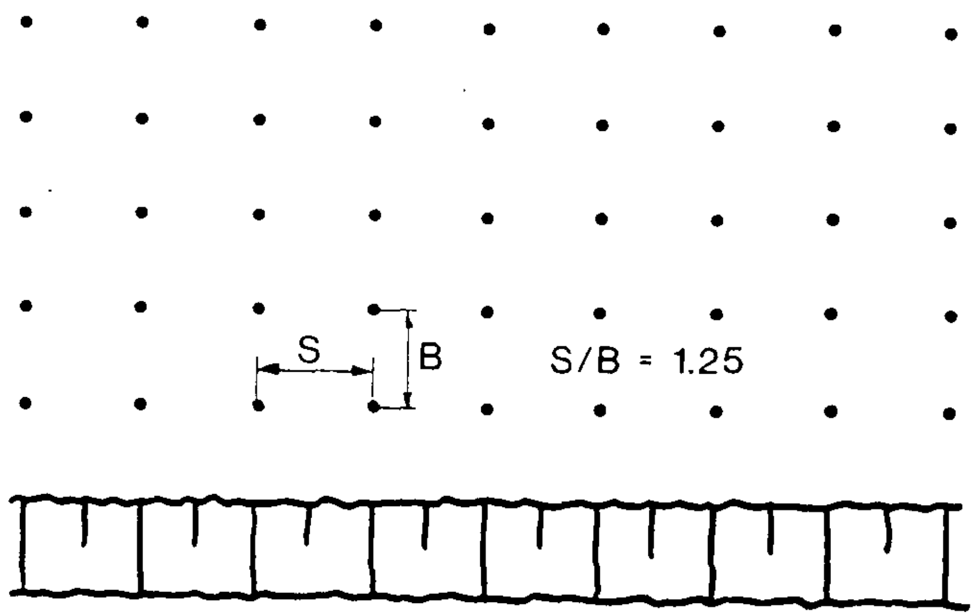


Fig. 5.14 Normal drilling pattern.

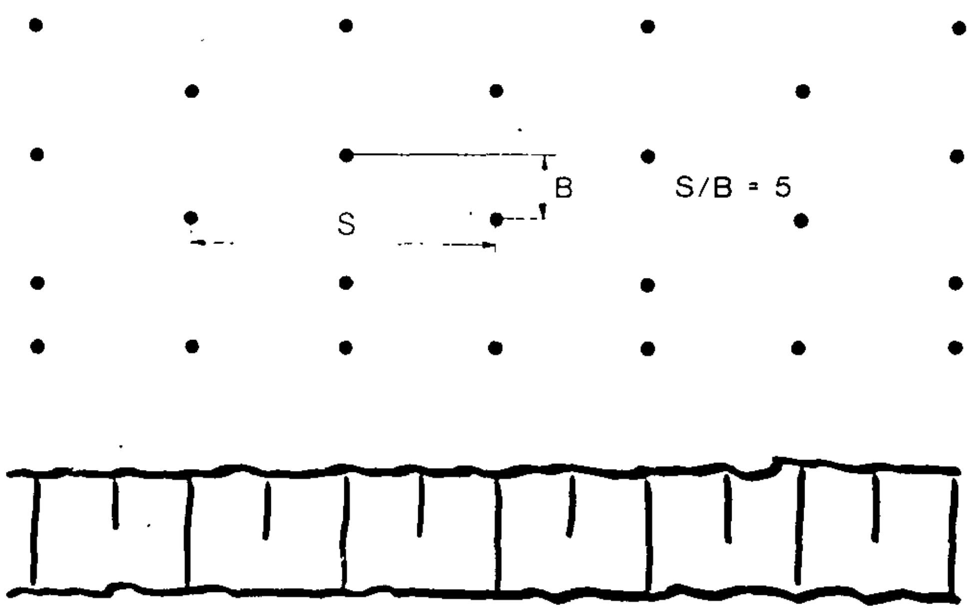


Fig. 5.15 Wide-space blasting.

Firing pattern.

Bench blasting is normally carried out as short delay blasting. The firing pattern has to be designed so that each blasthole has free breakage.

The delay time between blastholes and between rows has to be long enough to create space for the blasted rock from the succeeding rows.

Bernt Larsson of Nitro Nobel has studied the effect of the delay time on multiple row blastings. He states that the rock must be allowed to move 1/3 of the burden distance before the next row is allowed to detonate. The delay time between the rows may vary from 10 ms/m (hard rock) to 30 ms/m (soft rock) but generally 15 ms/m of the burden distance is a good guide value.

This length of delay gives good fragmentation and controls flyrock. It also gives the burden from the previously fired holes enough time to move forward to accommodate the broken rock from subsequent rows.

If the delay between the rows is too short, the rock from the back rows tends to take an upward direction instead of a horizontal. On the other hand, too long a delay may cause flyrock, airblast and boulders, as the protection from previously fired rows disappears due to too great a rock movement between detonations. The increase in boulders is due to the fact that the blast in this case may be compared with a single row blast.

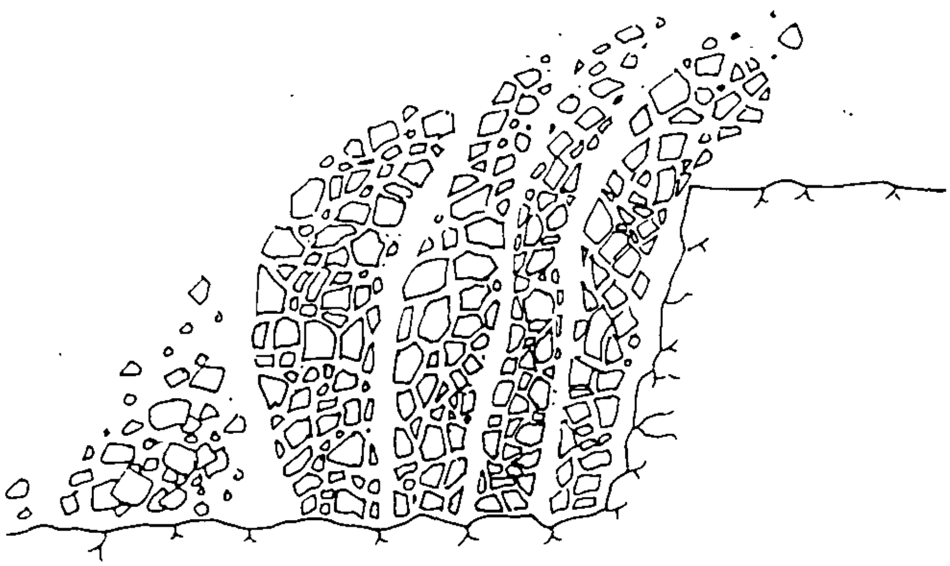


Fig. 5.16 Too short a delay between rows

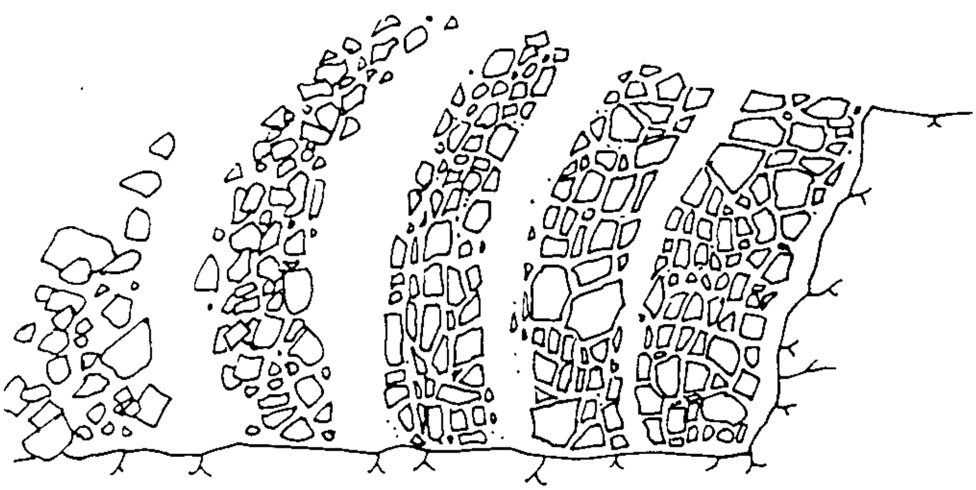


Fig. 5.17 Perfect delay between rows.

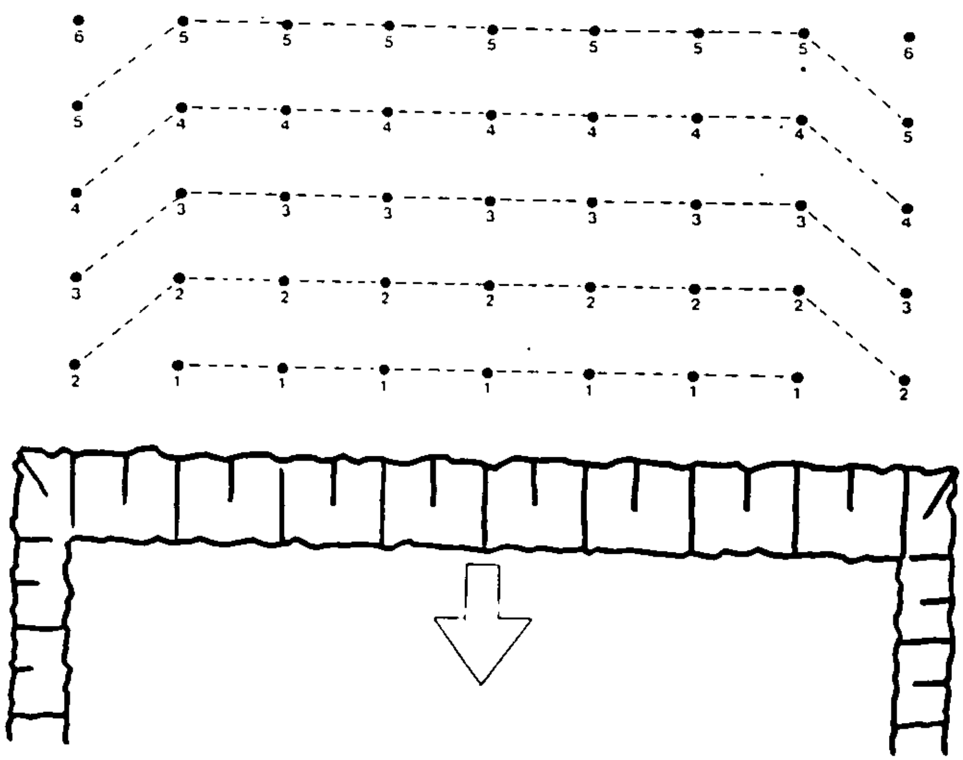


Fig. 5.18 Firing pattern, multiple row blasting.

Simple firing pattern for a laterally constricted multiple row round. All holes in the row have the same delay, except the perimeter holes, which are delayed one interval number to avoid excessive overbreak outside the limits of the excavation.

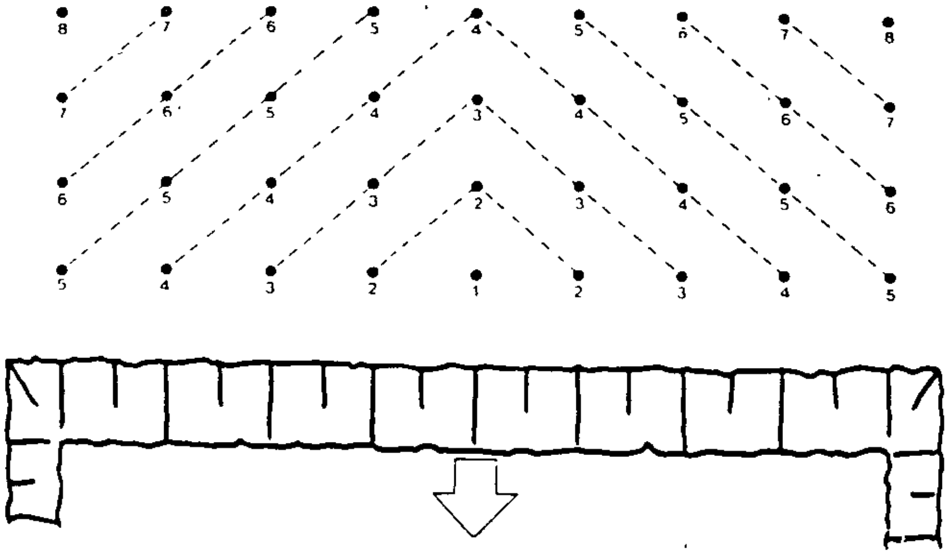


Fig. 5.19 Firing pattern.

This firing pattern gives better fragmentation. The ratio between true spacing and true burden, S/B, becomes more favorable (See wide-space drilling pattern.)

One disadvantage with the above firing pattern is the risk that the center hole in the second row of the blast may detonate before the detonators in the front row with the same delay number, due to the scatter within the delay interval. The hole will then be quite constricted causing incomplete breakage which will form boulders and possible butts above the theoretical grade.

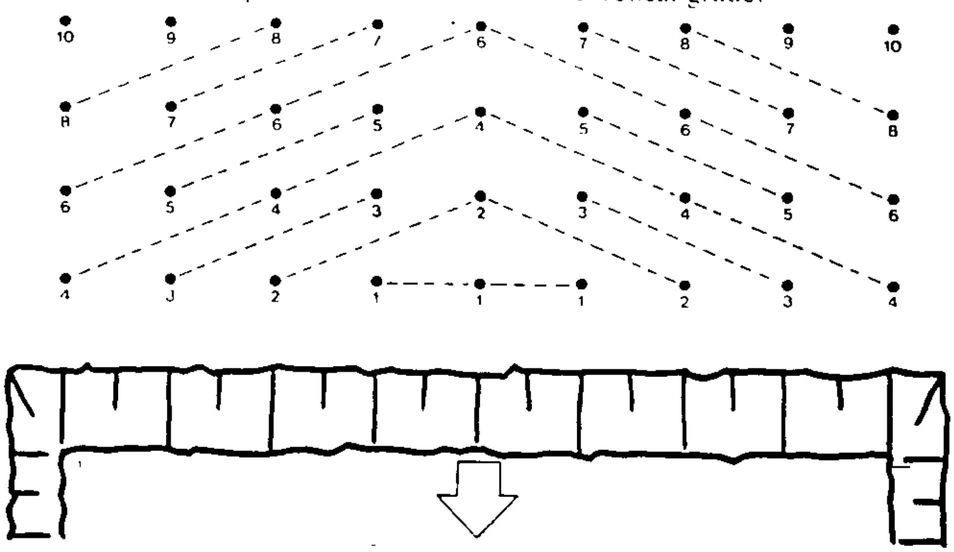


Fig 5.20 Firing pattern.

This firing pattern provides separate delay time for practically all blastholes and gives good fragmentation as well as good breakage in the bottom part of the round.

Hole inclination.

Inclined holes with an inclination of approx. 3:1 reduce the back break and the amount of boulders from the upper part of the blast.

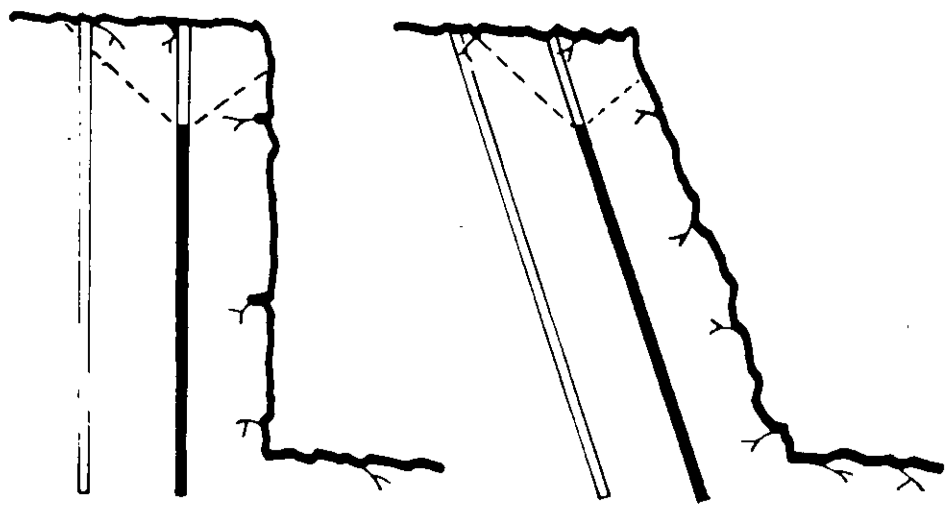


Fig 5.21

Hole deviation.

Precision in drilling is important for the blasting result.

Poor precision in drilling will form boulders due to irregular burdens and spacings.

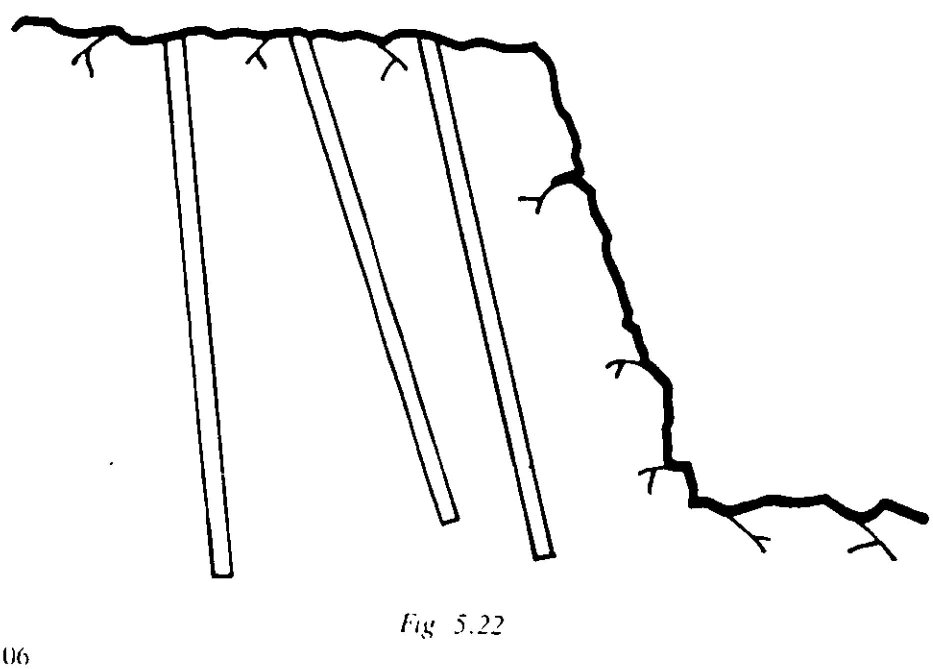


Fig 5.22

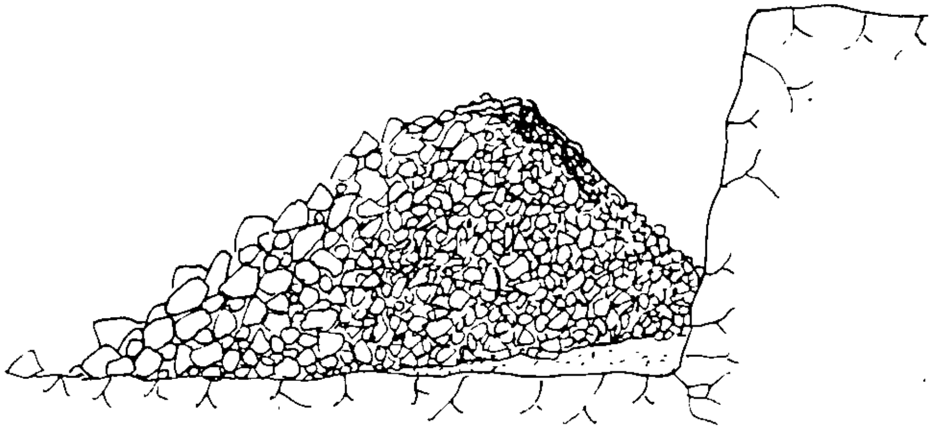


Fig. 5 23

Size of the round.

It is a known fact that most of the boulders in a blast come from the front row. Therefore, multiple row blasts give proportionally fewer boulders than single row blasts

However, the length of the blast should not be greater than 50 % of the width

5.6.2 Large size fragmentation.

Frequently large size fragmentation is required In the construction of ports, large size rock is used for the construction of breakwaters

The blasting to produce large size rock may be as difficult as producing small size fragmentation. The geology of the rock may form the greatest obstacle to obtain a good blasting result A homogenous rock is preferable in large fragmentation blasting to a fissured rock.

The method of blasting large sized blocks is somewhat different from normal bench blasting.

- The specific charge has to be low.
- The spacing/burden ratio (S/B) should be less than 1.
- Blasting of one row at a time, preferably instantaneously.

The specific charge should be decreased down to 0.20 kg per cubic meter (and occasionally lower) which may be sufficient to loosen the rock but not move it forward. Rock fragmentation will not occur to any large extent. The charge should be well distributed in the blasthole with a reasonable bottom charge although smaller than normal Due to the smaller bottom charge, a certain secondary blasting of the bottom has to be tolerated.

The choice of larger burden than spacing will definitely give blockier fragmentation with an optimum blasting result when the S/B ratio is between 0.5 and 1.0.

The instantaneous firing results in larger fragmentation than short delay blasting, as the tearing between the blastholes becomes less.

A combination of low specific charge, S/B ratio of 0.5 to 1.0, and single row instantaneous blasting, normally results in large fragmentation.

The following sketch gives an example how to blast to obtain large blocks

* Blasthole diameter	76 mm
* Bench height	15 m
* Burden	3.2 m
* Spacing	1.6 m
* Charge	
Bottom	5.0 kg
Column	11.0 kg
* Specific charge	0.21 kg/cu m.

Emulite 150, 50×550 mm as bottom charge.

Emulite 150, 32×550 mm taped on a 10 gram/meter detonating cord as column charge.

Initiation: Detonating cord.

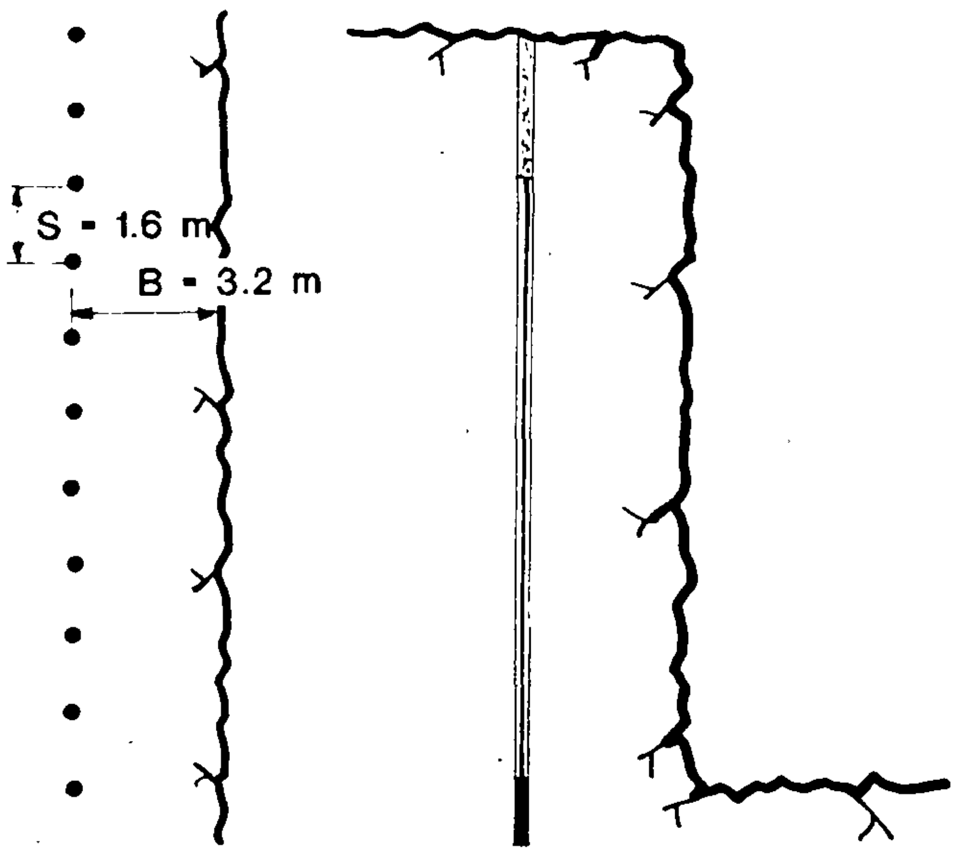


Fig. 5.24

5.7 Swelling.

When the rock is fragmented by a blast, its volume increases considerably, up to 50 %, this is known as the swelling.

The increased rock volume needs more space and if there is not space enough in front of the round, the rock must move upwards

The same applies to long blasts where the rock piles up in front of the round as the blast proceeds row by row.

As mentioned in Chapter 5.2 Charge calculation, the specific charge should be increased if the blastings are carried out without mucking between the blasts

According to Langefors, in his book Rock blasting, the requisite extra specific charge to compensate for the elevation of the blasted rock masses is $0.04 \times K$ (bench height) if the inclination of the blasthole is 2:1. If the hole inclination is steeper, the compensation of the specific charge has to be increased and is $0.08 \times K$ at a hole inclination of 3:1.

When no excavation is carried out between the blasts, the hole inclination must not be less than 3:1. Furthermore, the bench must not be too high. High benches must have such a high specific charge to compensate for the swelling that the risk of flyrock makes it prohibitive.

For long blasts, the rule of thumb is that the elevation of the swelling has to be considered when the length of the blast exceeds 50 % of the width. Experienced blasters usually compensate for the swelling by increasing the charge in the back rows.

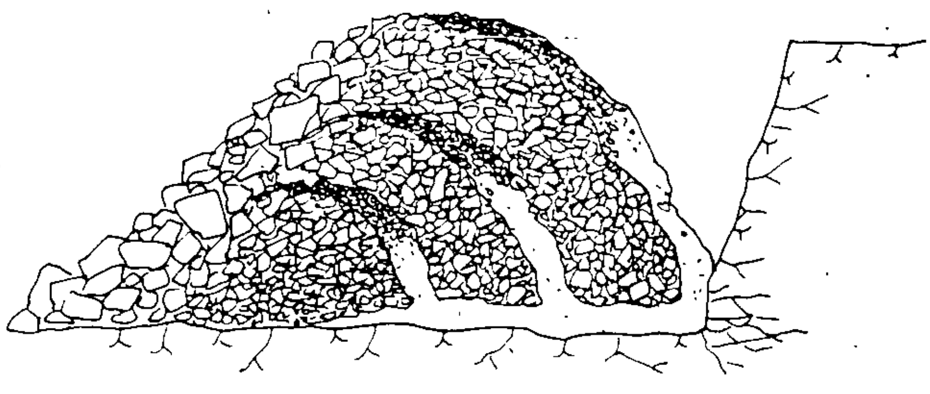


Fig. 5.25

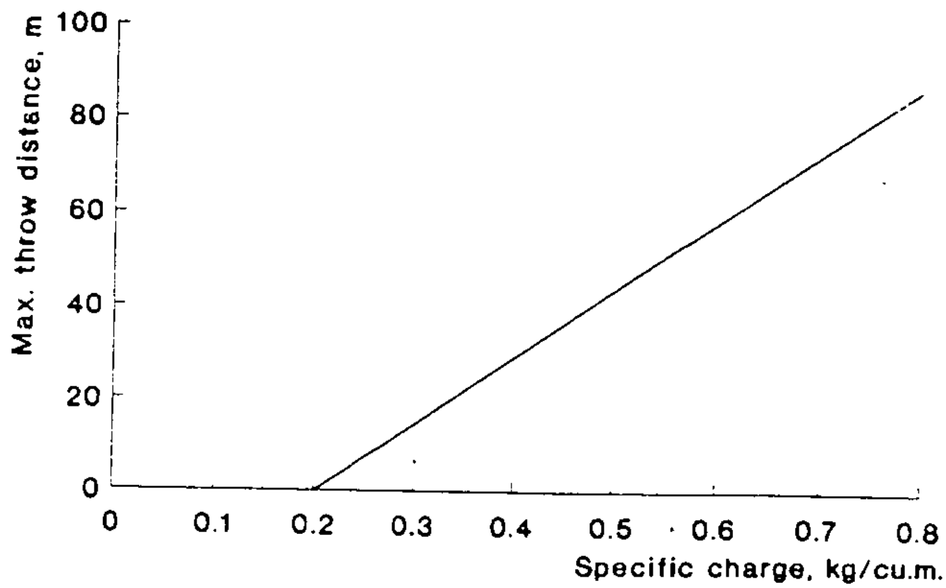


Fig. 5.26 Maximum forward movement as a function of specific charge.

5.8 Throw, flyrock.

In bench blasting we have to differentiate between two types of rock movement.

Firstly we have the forward movement of the entire rock mass which is mainly horizontal.
Secondly, the flyrock, which is scatter from the rock surface and the front of the blast.

The forward movement of the rock mass is dependent on the specific charge. Research performed by SVEDEFO* shows that a specific charge of 0.2 kg/cu.m. does not cause any forward movement of the rock, but only breaks the rock.
A normal specific charge of 0.4 kg/cu.m. moves the rock forward 20 to 30 m, which is the expected normal displacement of the rock mass.
Too short a forward movement causes a tight muckpile which is hard to excavate, while excessive forward movement spreads the muckpile, resulting in higher loading costs

The forward movement of the rock mass rarely represents any hazard in the blasting operation but may cause inconvenience when miscalculated.

Flyrocks are rocks ejected from the blast. They tend to travel long distances and are the main cause of on-site fatalities and damage to equipment.

Flyrock is mostly caused by an improperly designed or improperly charged blast.

Research performed by SVEDEFO shows that the maximum ejection of flyrock is a function of the blasthole diameter.

$L = 260\ d^{2/3}$

which is valid for a given specific charge. Fig. 5.27 shows the ejection distances

* Swedish Detonic Research Foundation.

of flyrock as a function of the specific charge at blasthole diameters of 25 to 100 mm.

A burden of less than 30 times the diameter of the blasthole gives too high a specific charge, especially if the explosive is poured or pumped into the blasthole. The excessive explosives content in the blasthole may result in rocks traveling long distances.
Too large a burden may cause flyrock if the explosive cannot break the burden and the gases vent through the collar of the hole creating crater effects.

An inappropriate firing pattern may cause the same effect as a too large burden. The gases from the blast tend to vent through the collar if the blasthole does not have free breakage.

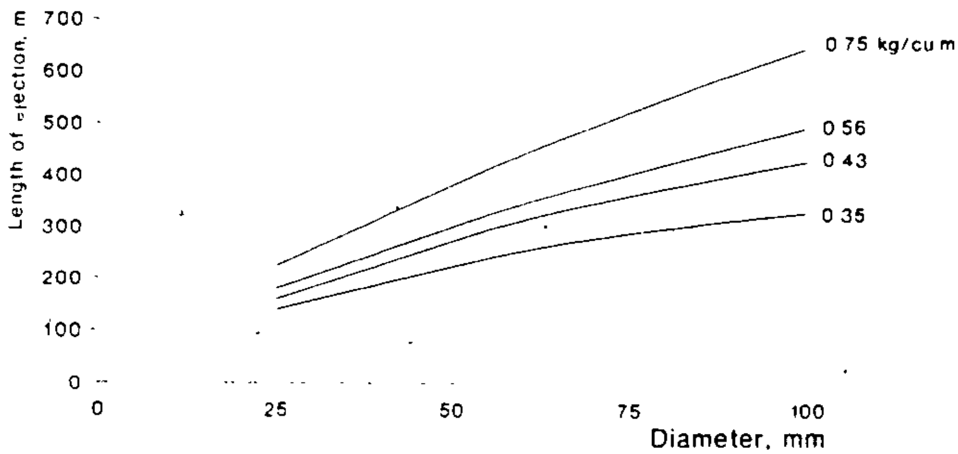


Fig. 5.27 Maximum traveling distance of flyrock as a function of blasthole diameter for different specific charges

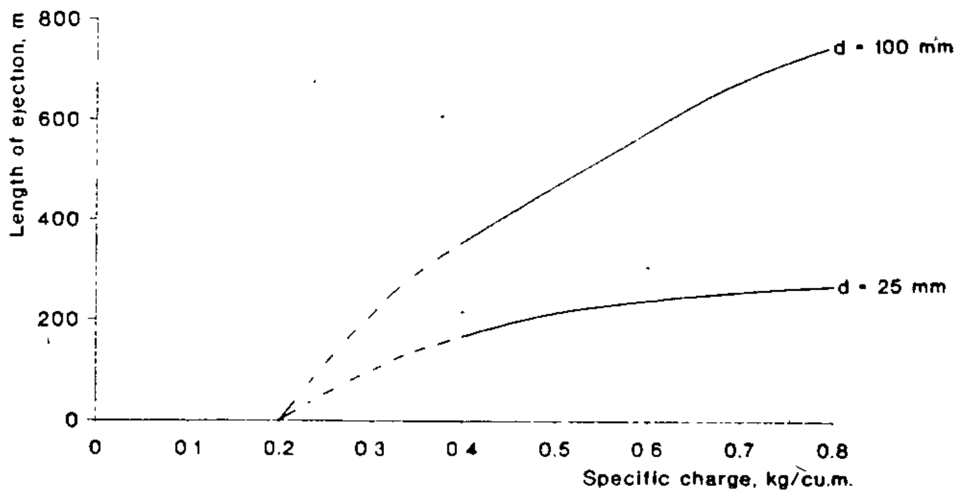
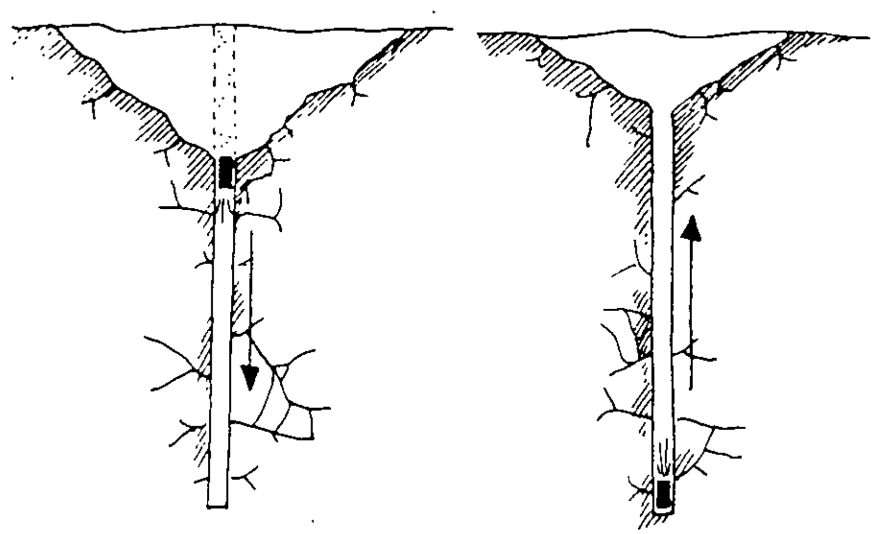


Fig. 5.28 Maximum traveling distance of flyrock as a function of the specific charge for 50 and 100 mm blasthole diameters.



Top-initiation *Bottom-initiation*

Fig. 5.29 Crater effect, vertical holes without free breakage.

Too short or too long a delay time between the blastholes may also cause flyrock. Too short a delay time creates an effect shown in fig. 5.16. The traveling distance is relatively limited. A more serious hazard appears when the delay time between the blastholes is too long. In a correctly designed firing pattern, the rock is held together and the rock from the front rows acts as a protection when the charges in the following rows detonate. If the delay between the rows or single blastholes is too long, the protective effect is not achieved.

Delay times between adjacent blastholes must not exceed 100 ms if the burden is less than 2 m.

When large diameter blastholes are used, longer delay between rows/holes must be used due to the sluggish movement of the large rock mass between the rows/holes.

Blasting of low benches, leveling, normally causes flyrock because of the fast movement of the rock mass. The low benches and the short burdens make it necessary to use short delay times between the blastholes. Leveling blasts should always be covered with heavy cover as well as light splinter-protective covering.

Various investigations, both in the U.S.A. and Sweden indicate that flyrock is more frequent when the blasthole is top-initiated than when it is bottom-initiated. (See Fig. 5.29.) Detonating cord with high core load top-initiates the explosive and tends to blow part of the stemming material out of the hole thus lowering the confinement of the explosive.

As mentioned in Chapter 5.2 Charge calculations, the stemming should have a particle size of 4 to 9 mm for best confinement. The best material for stemming is crusher run.

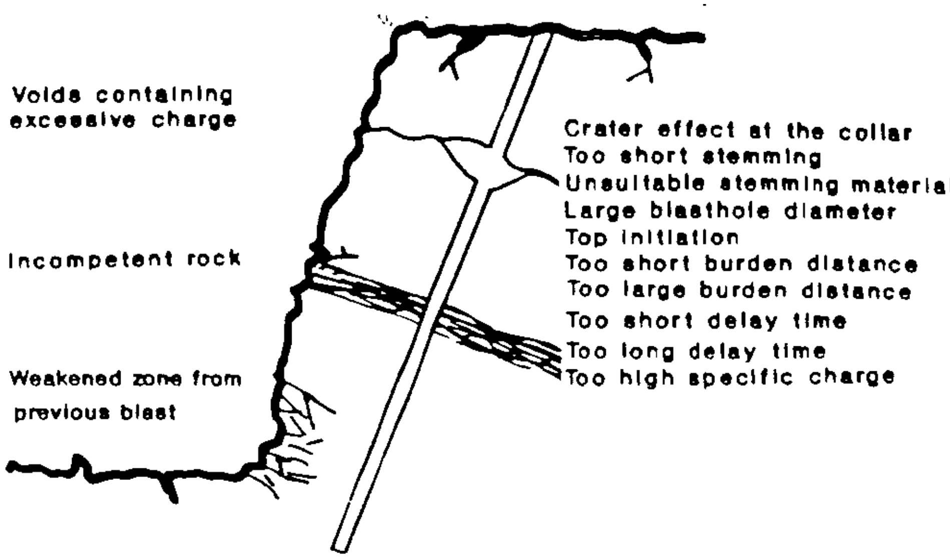


Fig 5.30 Causes of flyrock.

Flyrock is often caused by incompetent rock where the gases may break through easily due to less resistance than in the more competent parts of the rock. Necessary care must be taken during the charging of the blast, especially the first row.

The incompetent zones may be natural but also caused by the previous blast, especially in the heavily charged bottom part of the hole

If the blastholes are drilled with poor precision so that the burden is considerably smaller than the calculated one in the first row, the specific charge will locally be high. A deviation of 1 m for a 76 mm blasthole decreases the burden from 2.70 m (Emulite 150) to 1.70 m The specific charge will locally be increased from 0.40 kg/cu m. to 0.63 kg/cu m as an average for the hole The bottom part of the hole will have even higher specific charge The possible result of such an overcharge may be evaluated from Figures 5.27 and 5.28

How to avoid flyrock.

* Clean the rock surface from loose stones which may eject easily if gases vent through the collar of the blasthole.
* Avoid stemming shorter than the burden distance. Too short stemming may create crater effects
* Use good stemming material. No drillfines.
* Check that the drilling pattern is correct and that the blastholes are drilled with correct inclination
* Design the firing pattern so that each blasthole has free breakage and adequate delay time between each other.
* Look out for incompetent zones and voids and charge with care, stem incompetent parts of the holes.
* Charge the first row carefully Look out for back break which shortens the burden.

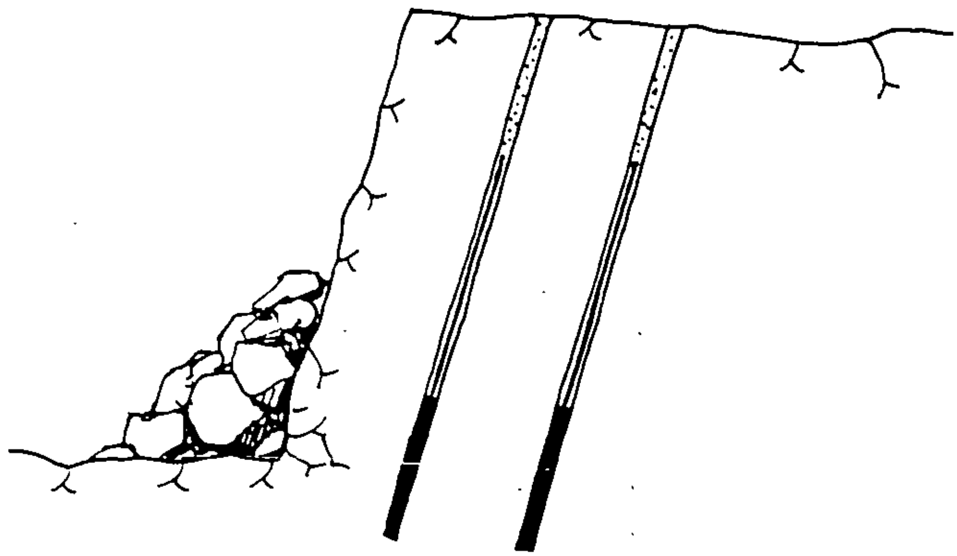


Fig. 5.31 Rock from previous blast in front of the face prevents flyrock from the heavily charged bottom part of the hole

* Check that the right amount of explosives is used. When flyrock is a problem, do NOT use free flowing explosives unless confined in plastic hoses and weighted
* Leave rock from the previous blast in front of the face, up to 1/3 of the bench height
* In built-up areas, cover the blast.

5.9 Covering.

To further protect against flyrock, the blast may be covered by energy-absorbing coverings which are placed on the top of the blast. This measure can be used for smaller blasts with small diameter blastholes, less than 76 mm.
Blasts with larger diameter blastholes are practically impossible to cover efficiently. However, large diameter blastholes are rarely used where flyrock is a problem, i.e. close to populated areas. Other limiting factors such as ground vibration levels and airblast levels will restrict the amount of explosive that may be detonated in each blasthole thus making large diameter blastings impractical.

The general rule for covering a blast is that the covering material should have the same weight as the blasted rock. This is valid for low benches, leveling, when small rock masses are loosened and the distance from the charge to the rock surface is short.
For normal bench blasting, where the bench height is more than twice the maximum burden ($K \geq 2 \times B_{max}$), it is hardly possible to use such a heavy covering
What we have to strive for in this case is to shorten the forward movement of the rock mass and to avoid flyrock.

The forward movement may be shortened by well-balanced charging of the blast and by leaving blasted rock from the previous blast in front of the rock face. The flyrock can be stopped by placing covering material over the blast and by well-poised stemming

Two types of covering are used and should be used together:

* Heavy covering.
* Splinter protective covering.

The heavy covering is intended to hold the blast together so that no part of it escapes when the round is fired.

The splinter protective covering is intended to prevent flyrock from the surface section of the round.

Heavy covering material.

* Rubber blasting mats made of scrap tires which are cut into sections and twined together with steel wires.
  The size of the blasting mat should be approximately 3×4 meters and have a weight of around 1 ton. Smaller mats which can be connected together to larger units can also be used.
* Mats made of logs shackled together

These blasting mats are heavy and energy absorbing (at least the rubber mats). The gases produced by the blast vent through the mats without displacing them to any greater extent

Splinter protective covering material:

* Industrial felt
* Tarpaulins
* Mesh nets.

The heavy covering should be placed closest to the rock surface with the lighter splinter protective covering on top

The covering with heavy mats should start from the back of the blast and work forward, each mat overlapping the previous
When the blast is fired, the mats ripple and do not follow the blast forward, which may happen if the blast is covered from the opposite direction, leaving the back rows without cover.

The covering work with heavy covering material has to be carried out with a crane or a retro excavator. For smaller-scale works small rubber mats may be used, but must be connected together with hooks to form covering units that are large enough.

The splinter protective material is then placed on top of the heavy covering, starting from the back and working forward.

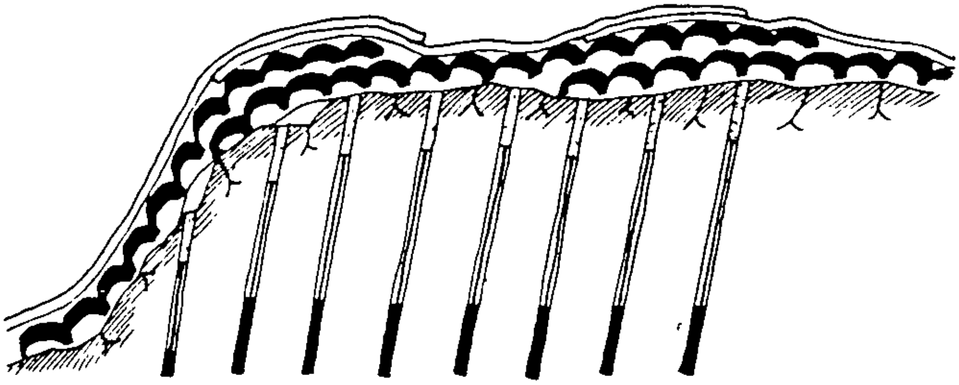


Fig. 5.32 Principle of covering.

Fig. 5.33 Covering with heavy tire mats.

Fig 5.34 Blasting with heavy covering in Abha, Saudi Arabia.

Note that no rock travels any great distance. All rock remains within 5 m of the blast.

5.10 Blasting economy.

Economic aspects on blasting operations.

In order to evaluate the cost of the blasting operation, it is not rational to isolate the drilling and blasting operations from the subsequent operations in the work cycle.

All operations in the work cycle have to be considered:

- drilling
- charging and blasting
- boulder blasting
- mucking (excavation)
- transport
- crushing

If the cost of the drilling and blasting operation is minimized, there is a great risk of increased costs in subsequent operations, which may give an increased total cost.

The factor that affects most on the operations following the blasting operation is that of **rock fragmentation**, which has to be considered when the cost of drilling and blasting is calculated

To quantify the rock fragmentation in relation to the blasting operation and the subsequent operations (boulder breakage, loading, transport and crushing) is quite a problem.

Harries and Mercer have visualized the relationship between blasting cost, transport and crushing to rock fragmentation in the following way:

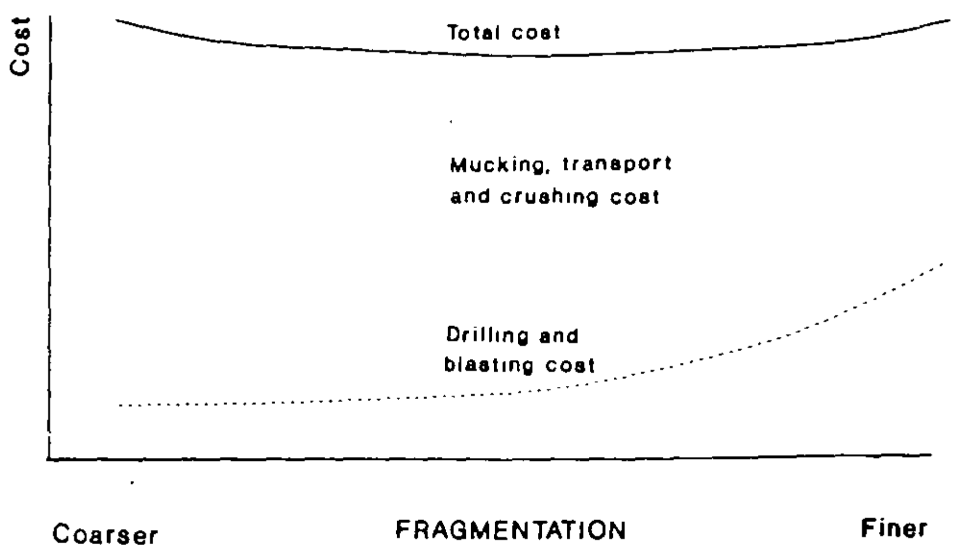


Fig. 5.35 Relationship between blasting cost and subsequent costs to rock fragmentation.

Despite the above, we will look into the cost of the drilling and blasting operation and ways of lowering the blasting costs

The drilling and blasting costs may be lowered by using bigger blasthole diameters

The cost per meter blasthole increases with the diameter

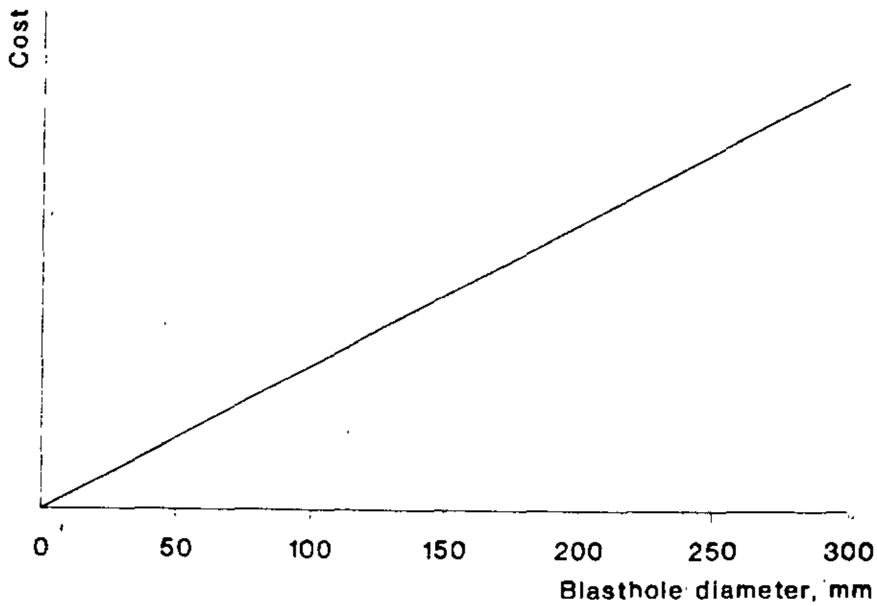


Fig. 5.36 Relative drill cost per meter blasthole.

The cost per volume of the blasthole decreases with larger diameter blastholes.

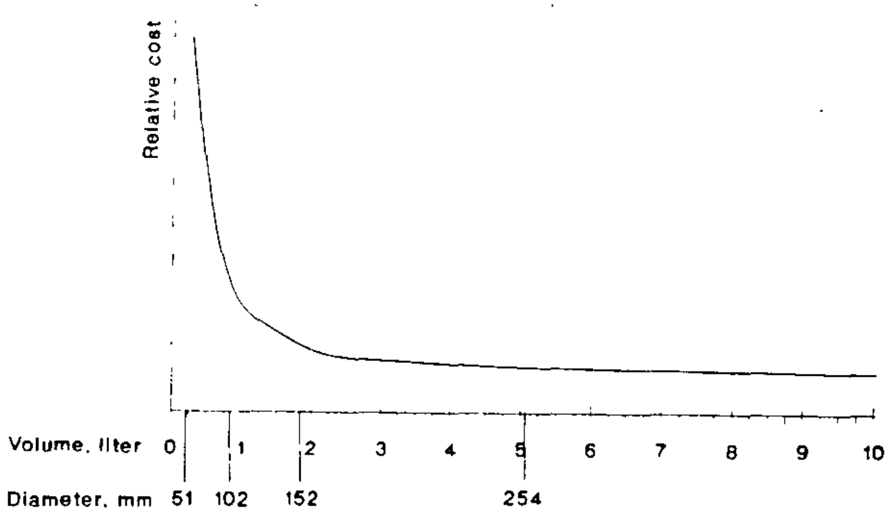


Fig. 5.37 Relative drill cost per liter of blasthole.

When large diameter blastholes are used, less expensive blasting agents may be used in the blastholes, lowering the total cost of the drilling and blasting operation.

As mentioned in Chapter 5 7 Rock fragmentation, large diameter blastholes tend to give a tight and blocky muckpile, resulting in more difficult loading and transport. Furthermore, the handling of boulders has a detrimental effect on the work cycle.

More often than not, the crushing operation is the bottleneck of the total workcycle The free flow through the crusher can in such cases have such a great influence, that extra expenditure on the drilling and blasting operation may be the only way to assure full capacity in the plant and improved overall economy.

The most effective utilization of the explosive charge is obtained when the maximum burden is 35 (ANFO) to 45 (Dynamex M) times the blasthole diameter. As the blasthole diameter is increased, the burden approaches the dimensions of the bench height, making the explosives work less efficiently. Such large burdens also increase the risk of flyrock and airblast as a stemming distance with the same dimension as the burden is impossible to obtain.

In the 1980s it has been a trend among the rock producers to abandon the use of large diameter blastholes to medium size holes. The trend is opposite in crushing equipment, the manufacturers are going for larger equipment. Larger equipment has been installed, but as mentioned earlier: **"Big crushing equipment is designed to handle large amounts of rock material, not large size material".**

The bench height will also influence the economics of the blasting operation. We know from experience that drilling deviates from the theoretical drill line. The magnitude of the deviation depends on the skill and care of the operator as well as the drilling equipment used. The geological characteristics of the rock may also influence the drilling precision

In normal blasting operations we calculate with a deviation in drilling of 3 cm per meter of blasthole depth The deviation has to be compensated for in the drilling pattern.

The higher the bench — the more closely spaced the drilling pattern.

The higher specific drilling increases the costs of the operation.

If the drilling pattern is not adjusted for the deviation in drilling, the bottom part of the blast will most certainly not be blasted to the intended level (toe-problem). This will add extra costs to the subsequent mucking operation, as the loading equipment will have a problem loading because of stumps. The secondary blasting of the toe adds unnecessary cost to the operation.

It is of vital importance to the economics of the blasting operation that the right explosive is used on each occasion. In dry conditions, cheap blasting agents can be used with excellent results. ANFO has become the most used blasting agent in the world due to its availability and economy. In dry conditions no explosive beats ANFO in overall economy including drilling, blasting, secondary blasting, loading, transport and crushing.

When the blastholes contain water, ANFO should not be used, at least not in the bottom part of the hole. ANFO deteriorates fast and should be replaced by Emulite 150 or Dynamex M which have excellent water resistance properties. If the bottom part of the hole is charged up over the theoretical grade with Emulite 150 or Dynamex M instead of ANFO, the risk of stumps in the blast will be practically eliminated. Furthermore, the rock fragmentation will be improved because of greater effectiveness of the explosive.
Tests carried out by Langefors (The modern technique of Rock Blasting) show that it is possible to increase the burden and spacing with 7 % each if the bottom charge of ANFO is replaced by a more potent explosive such as Emulite 150 to a height of $0.4 \times B_{max}$.

There are overall economic benefits from such a potent charge in the bottom part of the blast. The charge should be considered a reinforced primer rather than a reduced bottom charge. The advantages are clear:

* Reliable initiation of the blasting agent.
* Water resistant explosive in the water contaminated bottom part of the blast.
* Good breakage in the constricted bottom part of the blast.
* Less stumps above the intended grade, simplifying the loading operation.
* Less drilling.
* Lower consumption of firing devices.

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CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

UNDERGROUND BLASTING

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7. UNDERGROUND BLASTING

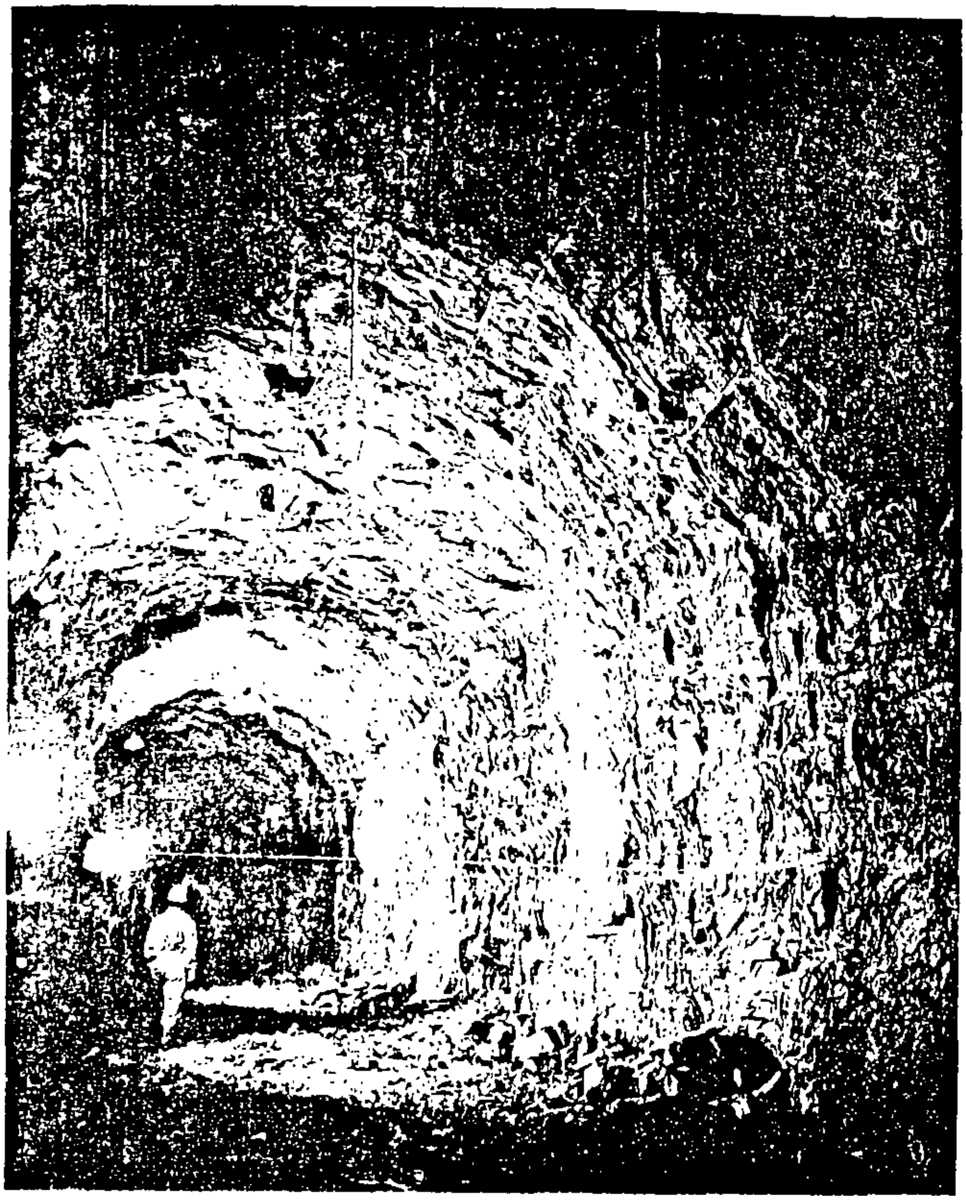


Fig. 7.1 Tunneling.

7.1 Tunneling.

There are two reasons to go underground and excavate:

- to use the excavated space, e.g. for storage, transport etc.
- to use the excavated material, e.g. mining operations.

In both cases tunneling forms an important part of the entire operation. In underground construction it is necessary to gain access to the construction site by

tunneling, but the tunnel can be a purpose in itself e.g. road, water, cable tunnels etc.
In mining operations tunnels are used as adits to the mining site and for preparatory work as well as for internal communication.
Tunnels are driven mainly in horizontal or close to horizontal directions but also inclined, from vertically upwards to vertically downwards. In the following, tunneling, raise shafts and sink shafts will be dealt with in detail while storage in rock caverns and mining will be dealt with more briefly.

Tunneling is the most frequently ocurring underground operation which also forms part of the construction of rock chambers etc. and is normally an integral part of mining operations.
The development of tunnel driving techniques has been tremendous during the last few years. The drilling techniques have developed from pneumatic drilling machines to electro-hydraulic drilling jumbos with a very high capacity. The charging of the blastholes can be carried out quickly either manually with plastic pipe charges or mechanically with pneumatic charging equipment.
The development of explosives has moved in the direction of safer products with better fumes characteristics. Modern explosives like Emulite and Dynamex M are well oxygen-balanced with a minimum of noxious fumes.
Initiating systems like NONEL have shortened the charging time and added further safety to the blasting operation due to their insusceptibility to electrical hazards.

The modern drilling equipment has shortened the drilling time, the NONEL system has made connecting of the detonators safer and faster and Emulite, with its excellent fumes characteristics, has shortened the ventilation time.
All the above contribute to a faster work cycle:

- drilling
- charging
- blasting
- ventilation
- scaling
- grouting (if necessary)
- loading and transport
- setting out for the new blast

The shorter work cycle calls for better work planning as well as better precision and accuracy in the different operations of the work cycle.
In the following, the drilling, charging and blasting operations will be dealt with.
It is obvious that it is of the utmost importance that the holes should be drilled at the right locations and with the right inclination. The marking of the holes on the rock face as well as collaring and drilling must be carried out accurately.
Langefors in "The modern technique of Rock Blasting", says about drilling precision: "**The scattering of the drill holes as a quantitative factor is often disregarded. It is included quite indefinitely in the technical margin together with the rock factor. In discussing blasting as a whole it would be a great advantage if**

attention could be paid to the drilling precision in calculating the charges and in constructing the drilling pattern; for the blasting of the cut it is essential."

The main difference between tunnel blasting and bench blasting is that tunnel blasting is done towards one free surface while bench blasting is done towards two or more free surfaces. The rock is thus more constricted in the case of tunneling and a second free face has to be created towards which the rock can break and be thrown away from the surface. This second face is produced by a cut in the tunnel face and can be either a parallel hole cut, a V-cut, a fan-cut or other ways of opening up the tunnel face.

After the cut opening is made, the stoping towards the cut will begin. The stoping can be compared with bench blasting, but it requires a higher specific charge due to higher drilling deviation, desire for good fragmentation, and absence of hole inclination. In addition, overcharge of a tunnelblast does not have the same disastrous effect as in an open air blast, where high precision in calculation is a must.

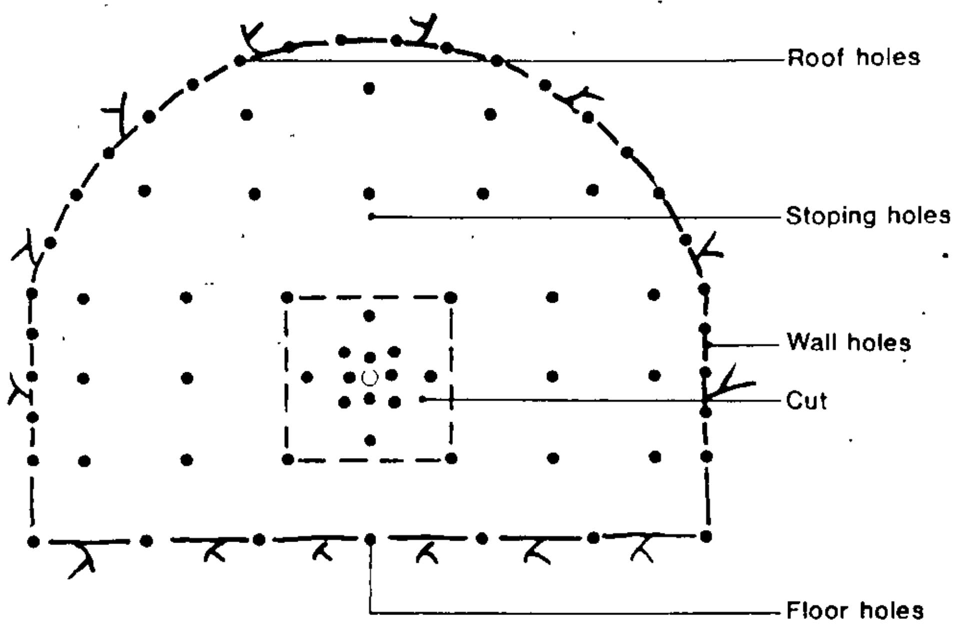


Fig 7.2 Nomenclature

In the case of V-cuts and fan cuts, the cut holes will occupy the major part of the width of the tunnel.

The contour holes – roof holes, wall holes and floor holes – have to be angled out of the contour, "look-out", so the tunnel will retain its designed area. The "look-out" should only be big enough to allow space for the drilling equipment for the coming round. As a guide value, the "look-out" should not exceed:

10 cm + 3 cm/m holedepth

which keeps the "look-out" to around 20 cm.

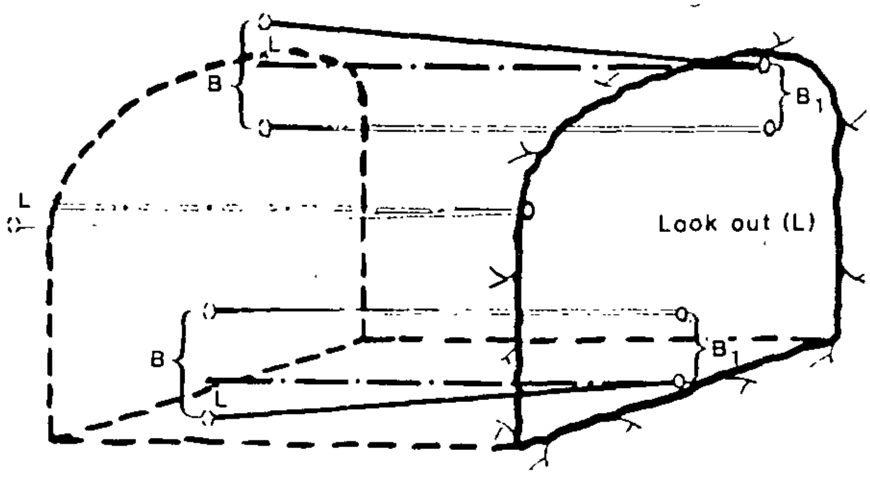


Fig. 7.3 Look-out.

The consumption of explosives in tunnel blasting is higher than in bench blasting. The specific charge is 3 to 10 times higher than that for bench blasting, depending mainly on reasons mentioned above like large drilling scatter, higher fixation of the holes, heave of lower rock upwards to ensure swell and lack of cooperation between adjacent blastholes.

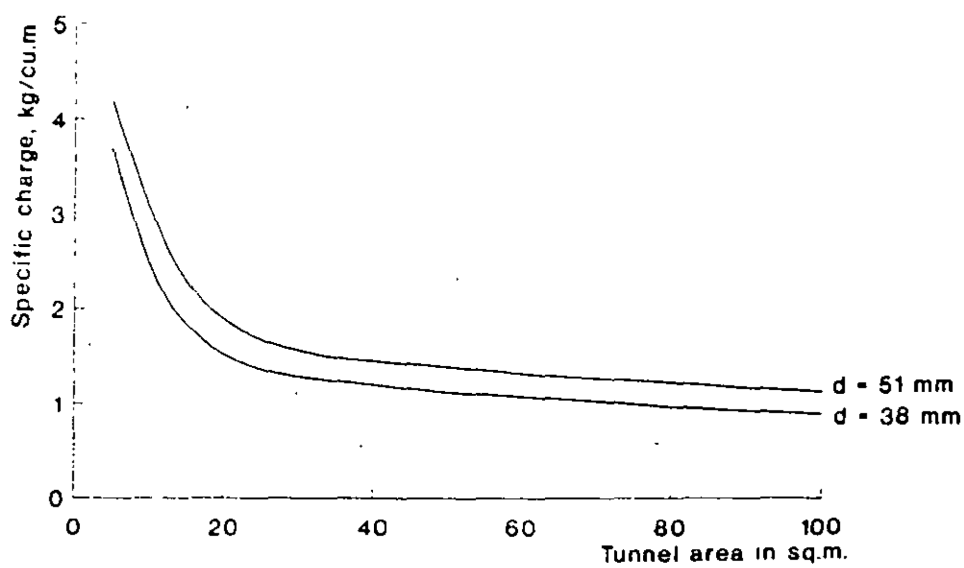


Fig. 7.4 Specific charge for different tunnel areas.

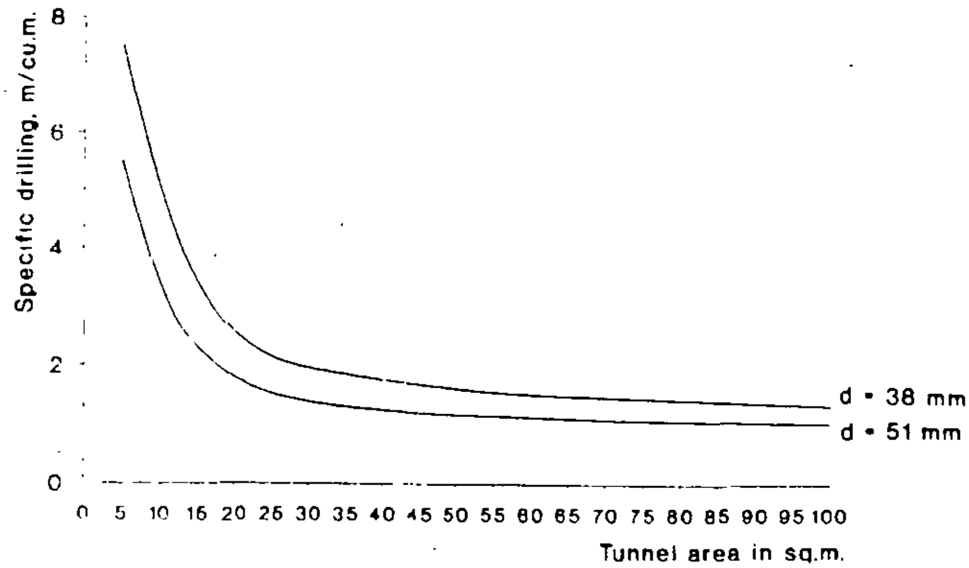


Fig 7.5 Specific drilling for different tunnel areas.

The consumption of explosives will be greatest in the cut area of the blast. A 1×1 m area around the empty hole/s in a parallel cut will consume approx. 7 kg/cu.m. and the specific charge will decrease with the distance from the cut until it reaches a minimum value of about 0.9 kg/cu.m.

7.1.1 The cut.

The most commonly used cut in tunneling today is the **circular cut** or **large hole cut** as most of the modern drilling equipment is designed for horizontal drilling perpendicular to the rock face. (Other cuts will be dealt with in the end of this chapter.)

All cut holes in the large hole cut are drilled parallel to each other and the blasting is carried out towards an empty large drill hole which acts as an opening. The parallel hole cut is a development of the **burn cut**, where all the holes are parallel and normally of the same diameter. One hole in the middle is given a heavy charge and the four holes around it are left uncharged, in other cases the middle hole is left uncharged and the four holes are charged.

However, the burn cuts generally result in less advance than the large hole cuts. The burn cut will therefore be disregarded and only the **large hole cuts** will be dealt with.

The cut may be placed at any location on the tunnel face, but the location of the cut influences the throw, the explosives' consumption and generally the number of holes in the round.

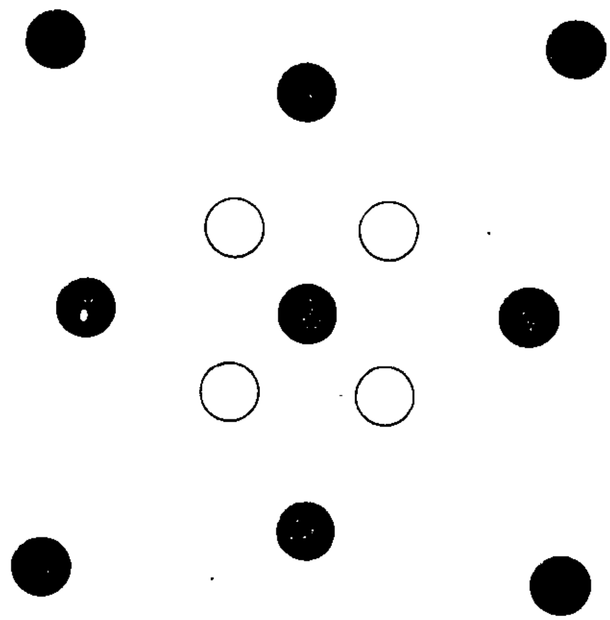


Fig. 7.6 Burn cut

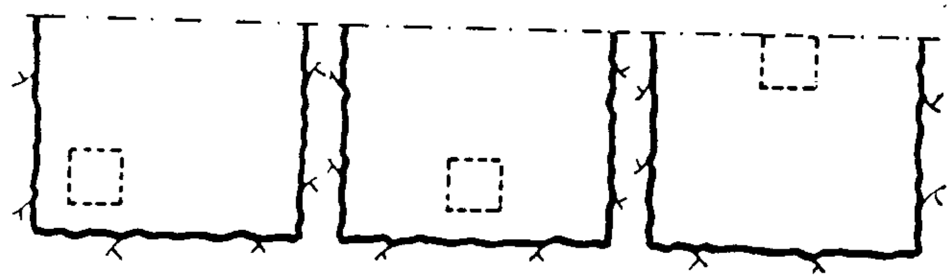


Fig. 7.7 Location of the cut.

If the cut is placed close to a wall, there is a probability of better exploitation of the drilling pattern with less holes in the round. Furthermore, the cut may be placed alternatively on the right or left side thus placing the cut in relatively undisturbed rock. To obtain good forward movement and centering of the muckpile, the cut may be placed approximately in the middle of the cross section and quite low down. This position will give less throw and less explosives' consumption because of more stoping downwards. A high position of the cut gives an extended and easily loaded muckpile, but higher explosives' consumption and normally more drilling due to more upwards stoping.

The normal location of the cut is on the first helper row above the floor.

As mentioned before, the large hole cut is the most common cut today. The cut is composed of one or more uncharged large diameter holes which are surrounded by small diameter blastholes with small burdens to the large hole/s. The blastholes are placed in squares around the opening.

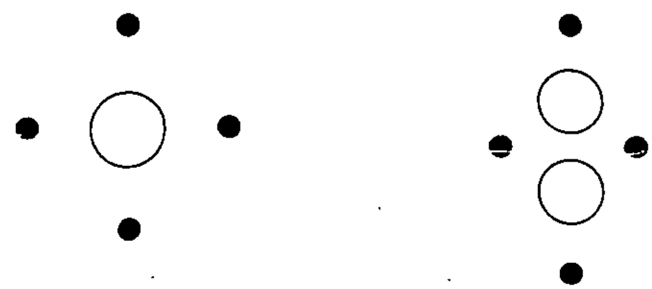


Fig. 7.8 Typical designs of large hole cuts.

The number of squares in the cut is limited by the fact that the burden in the last square must not exceed the burden of the stoping holes for a given charge concentration in the hole.

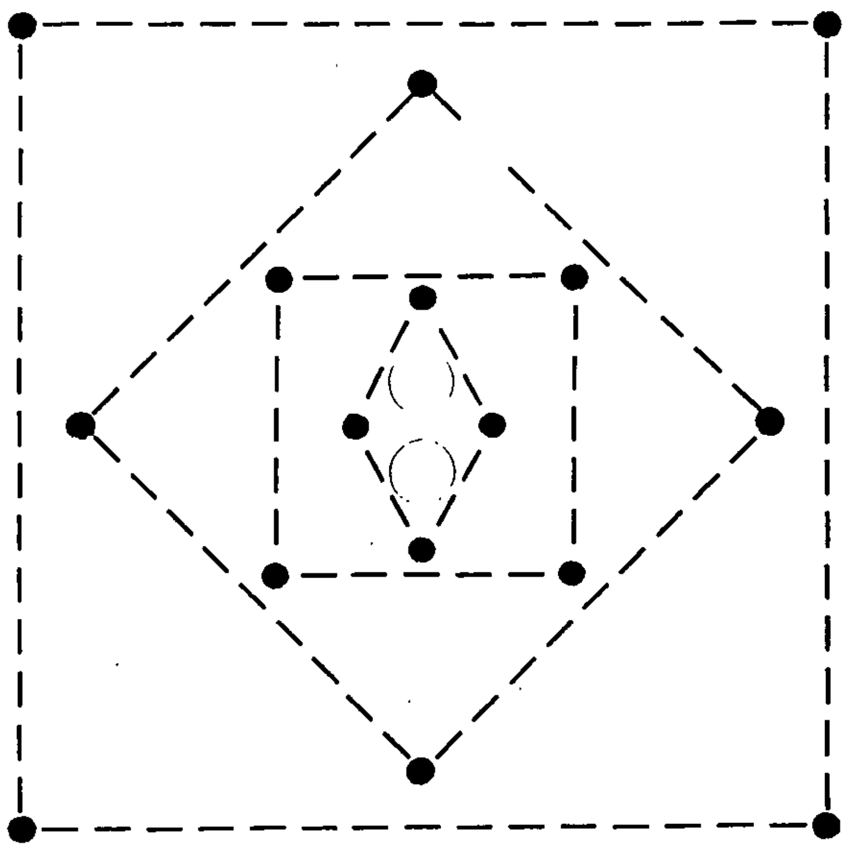


Fig. 7 9 The complete cut.

The cut holes occupy an area of approx. 2 sq.m. (Small tunnel areas, as a matter of fact, consist only of cut holes and contour holes.)

When designing the cut, the following parameters are of importance for a good result:

* the diameter of the large hole
* the burden
* the charge concentration

In addition, the drilling precision is of the utmost importance, especially for the blastholes closest to the large hole/s. The slightest deviation can cause the blasthole to meet the large hole or the burden to become excessively big. Too big a burden will only cause breakage or plastic deformation in the cut, resulting in a smaller or greater loss in advance

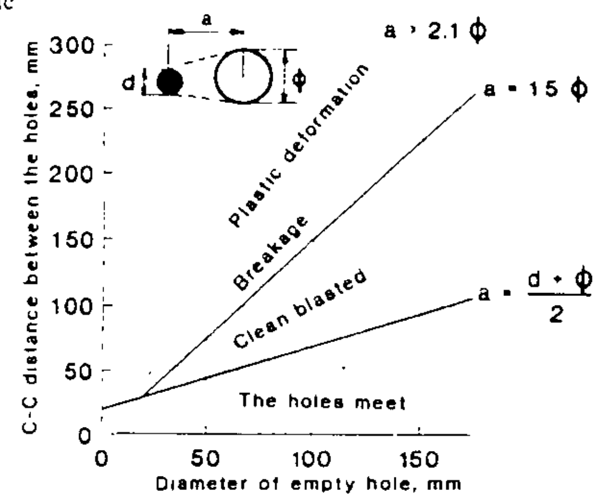


Fig 7.10 Result when blasting from varying distances towards an empty hole of varying diameter
(The Modern Technique of Rockblasting)

One of the parameters for good advance of the blasted round is the diameter of the large empty hole. The larger the diameter, the deeper the round may be drilled and a greater advance can be expected.
One of the most common causes of short advance is too small an empty hole in relation to the hole depth.

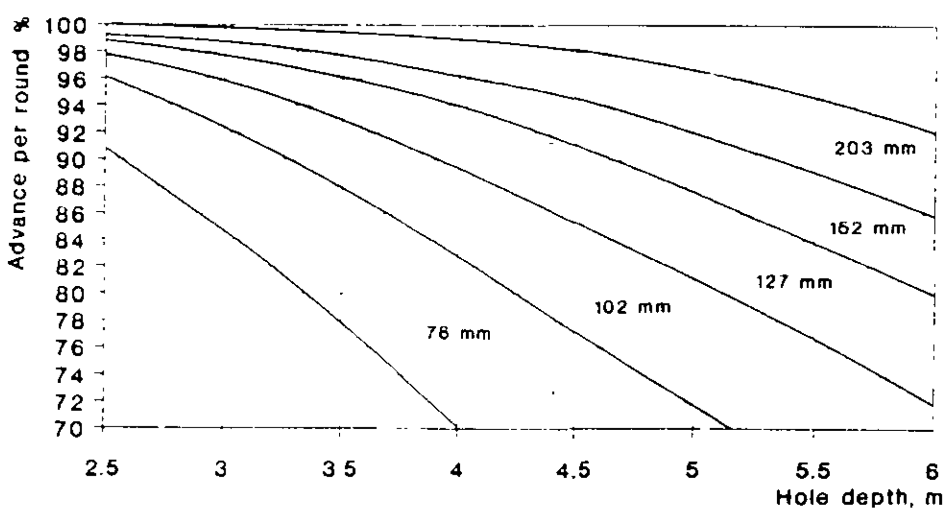


Fig 7.11 The relation between advance in per cent of the drill depth and different empty hole diameters.

As can be seen from the graph, an advance of approx. 90 % can be expected for a hole depth of 4 m and one empty hole with 102 mm diameter.
If several empty holes are used, a fictitious diameter has to be calculated. The fictitious diameter of the opening may be calculated in accordance with the following formula:

$$D = d\sqrt{n}$$

where D = fictitious empty large hole diameter
d = diameter of empty large holes
n = number of holes

In order to calculate the burden in the first square, the diameter of the large hole is used in the case of one large hole and the fictitious diameter in the case of several large holes

Calculation of the 1st square.

If we look at the graph 7.10 we find that the distance between the blasthole and the large empty hole should not be greater than 1.5 Ø for the opening to be clean blasted. If the distance is longer, there is merely breakage and when the distance is shorter, there is a great risk that the blasthole and empty hole will meet.

So the position of the blastholes in the 1st square is expressed as:

$$a = 1.5\,\varnothing$$

Where a = C–C distance between the large hole and the blasthole
$\varnothing$ = diameter of the large hole

In the case of several large holes, the relation is expressed as:

$$a = 1.5\,D$$

Where a = C–C distance between the center point of the large holes and the blasthole
D = fictitious diameter

Charging of the holes in the 1st square.

The holes closest to the empty hole/s must be charged carefully. Too low a charge concentration in the hole may not break the rock, while too high a charge concentration may throw the rock against the opposite wall of the large hole with such high a velocity that the broken rock will be recompacted there and not blown out through the large hole. Full advance is then not obtained.

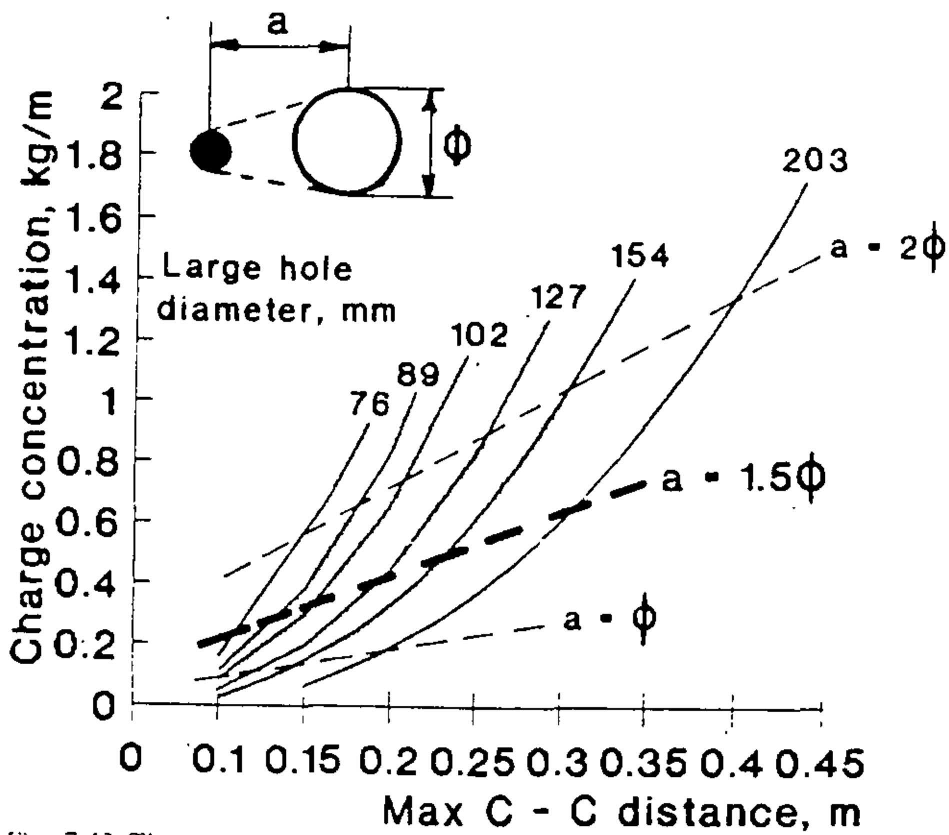


Fig 7.12 The minimum required charge concentration (kg/m) and maximum C–C distance (m) for different large hole diameters.

The requisite charge concentration for different C–C distances between the large hole and the nearest blasthole/s may be found in graph 7.10 for different large hole diameters. The normal relation for the distance is **a=1.5 Ø.** An increase in the C–C distance between the holes will cause subsequent increment of the charge concentration.

The cut is often somewhat overcharged to compensate for error in drilling which may cause too small an angle of breakage. However, too high a charge concentration may cause recompaction in the cut.

Calculation of the remaining squares of the cut.

The calculation method for the remaining squares of the cut is essentially the same as for the 1st square, with the difference that the breakage is towards a rectangular opening instead of a circular.

As is the case of the 1st square, the angle of breakage must not be too acute as small angles of breakage can only be compensated to a certain extent with higher charge concentration.

Normally the burden (B) for the remaining squares of the cut is equal to the width (W) of the opening. **B=W.**

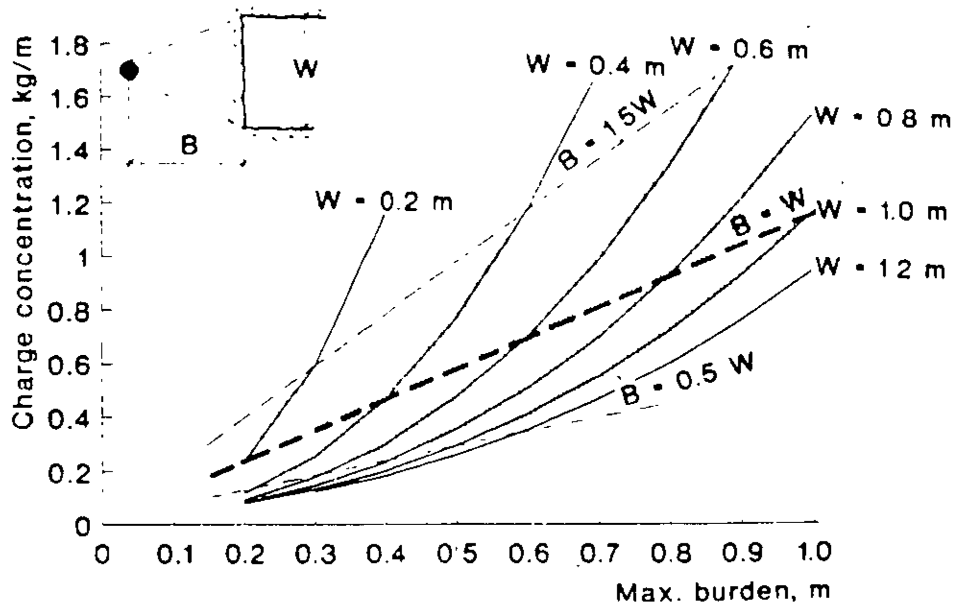


Fig. 7.13 The required minimum charge concentration (kg/m) and maximum burden (m) for different widths of the opening

The charge concentration obtained in graph 7.12 is that of the column of the hole. In order to break the constricted bottom part, a bottom charge with twice the charge concentration and a height of $1.5 \times B$ should be used. The stemming part of the hole has a length of $0.5 \times B$.

Design of cut.

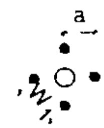
The following formulae are used for the geometric design of the cut area:

The cut:

1st square:

$a = 1.5\,\emptyset$

$W_1 = a\sqrt{2}$



$\emptyset$ mm	=	76	89	102	127	154
a mm	=	110	130	150	190	230
W_1 mm	=	150	180	210	270	320

2nd square:

$B_1 = W_1$

$C-C = 1.5W_1$

$W_2 = 1.5W_1\sqrt{2}$

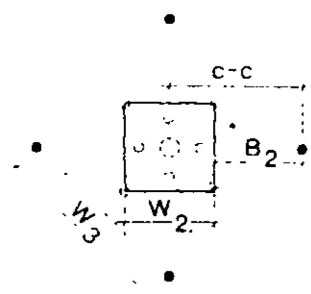
$\emptyset$ mm	=	76	89	102	127	154
W_1 mm	=	150	180	210	270	320
C−C	=	225	270	310	400	480
W_2 mm	=	320	380	440	560	670

3rd square:

$B_2 = W_2$

$C-C = 1.5W_2$

$W_3 = 1.5W_2\sqrt{2}$



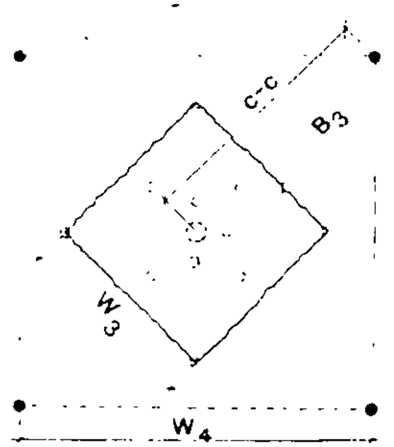
$\emptyset$ mm	=	76	89	102	127	154
W_2 mm	=	320	380	440	560	670
C−C	=	480	570	660	840	1000
W_3 mm	=	670	800	930	1180	1400

4th square:

$B_3 = W_3$

$C-C = 1.5W_3$

$W_4 = 1.5W_3\sqrt{2}$



$\emptyset$ mm	=	76	89	102	127
W_3 mm	=	670	800	930	1180
C−C	=	1000	1200	1400	1750
W_4 mm	=	1400	1700	1980	2400

The above distances apply to 38 mm blastholes. If larger blastholes are used which can accommodate more explosives, the values can be adjusted. However, an increased amount of explosives in the cut holes may not increase the burden to any greater extent

7.1.2 Stoping.

When the cut holes have been calculated, the rest of the tunnel round may be calculated.

The round is divided into:

* floor holes
* wall holes
* roof holes
* stoping holes with breakage upwards and horizontally
* stoping holes with breakage downwards

To calculate burdens (B) and charges for the different parts of the round the following graph (7.14) may be used as a basis.

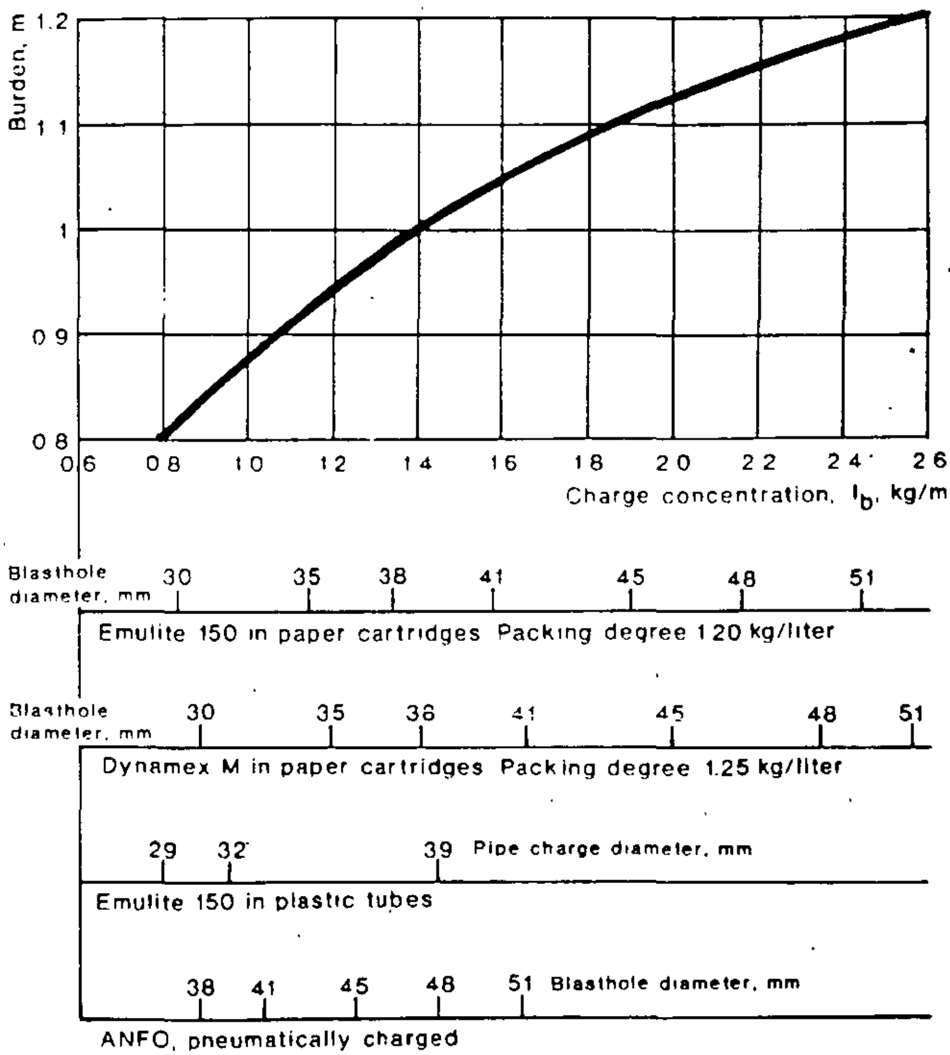


Fig. 7.14 The burden B in relation to the concentration of the bottom charge for different hole diameters and different explosives

For Emulite 150 in paper cartridges, the uppermost blasthole diameter table is used as input data.
For Emulite 150 and Dynamex M in plastic pipe cartridges, the pipe diameter is used as input data and for ANFO the lowest blasthole diameter table is used as input data.

When the burden (B), the hole depth (H) and the concentration of the bottom charge (l_b) are known, the following table will give the drilling and charging geometry of the round

Part of the round	Burden (m)	Spacing (m)	Height bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	1×B	1.1×B	1/3×H	l_b	1.0×l_b	0.2×B
Wall	0.9×B	1.1×B	1/6×H	l_b	0.4×l_b	0.5×B
Roof	0.9×B	1.1×B	1/6×H	l_b	0.3×l_b	0.5×B
Stoping						
Upwards	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
Horizontal	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
Downwards	1×B	1.2×B	1/3×H	l_b	0.5×l_b	0.5×B

The design of the drilling pattern can now be carried out and the cut located in the cross section in a suitable way

7.1.3 The contour.

The contour of the tunnel is divided into floor holes, wall holes and roof holes. The burden and spacing for the floor holes are the same as for the stoping holes. However, the floor holes are more heavily charged than the stoping holes to compensate for gravity and for the weight of the rock masses from the rest of the round which lay over them at the instant of detonation.
For the wall and roof holes two variants of contour blasting are used, **normal profile blasting** and **smooth blasting.**
With **normal profile blasting** no particular consideration is given to the appearance and condition of the blasted contour. The same explosives as in the rest of the round are utilized (but with a lesser charge concentration) and the contour holes are widely spaced. The contour of the tunnel becomes rough, irregular and cracked. The **smooth blasting** technique has been developed to obtain a smoother and stronger tunnel profile.
Smooth blasting is carried out by drilling the contour holes rather close to each other and using weaker explosives. (Gurit 17×500 mm and Gurit 11×460 mm have been specially developed for the requirements of smooth blasting.)
Smooth blasting is today a common technique in underground rock excavation as it produces tunnels with a regular profile, requiring substantially less reinforcement than if normal profile blasting is used.
Smooth blasting is dealt with in detail in Chapter 8.4 Smooth blasting, where charging tables for smooth blasting can be found

7.1.4 The firing pattern.

The firing pattern must be designed 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°

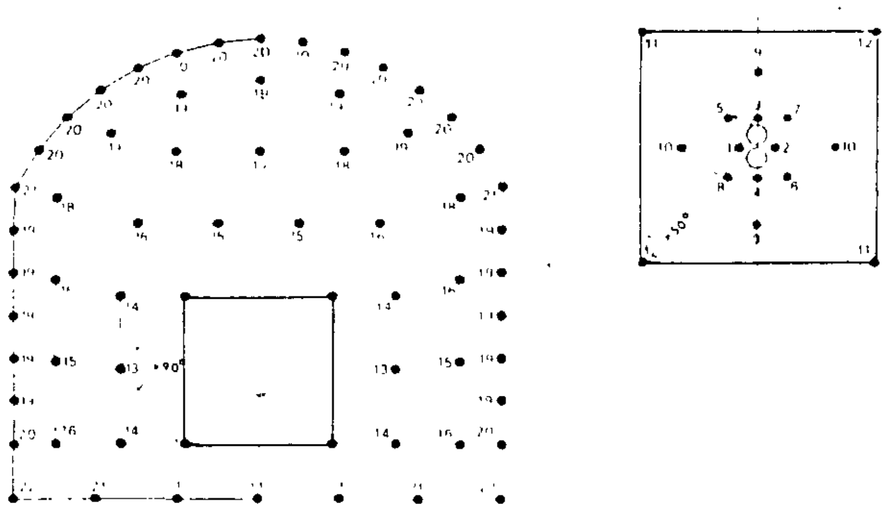


Fig. 7.15 Firing sequence for tunnel in numerical order.

It is important in tunnel blasting to have long enough time delay between the holes. In the cut area, the delay between the holes must be long enough to allow time for breakage and throw of rock through the narrow empty hole. It is proved that the rock moves with a velocity of 40 to 60 meters per second. A cut drilled to 4 m depth would thus require a delay time of 60 to 100 ms to be clean blasted. Normally delay times of 75 to 100 ms are used in the cut.

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

For the contour holes the scatter in delay between the holes should be as small as possible to obtain a good smooth blasting effect. Therefore, the roof should be blasted with the same interval number, normally the second highest of the series. The walls are also blasted with the same period number but with one delay lower than that of the roof.

Detonators for tunneling can be electric or non-electric.
The electric detonators are manufactured as MS (millisecond) and HS (half-second) delay detonators.

The non-electric detonators are manufactured as deci-second and half-second delay detonators.

Recommended detonators for tunneling:

Electric detonators:

	Interval No.	Delay time
VA/MS	1	25 ms
VA/MS	4	100 ms
VA/MS	7	175 ms
VA/MS	10	250 ms
VA/MS	13	325 ms
VA/MS	16	400 ms
VA/MS	18	450 ms
VA/MS	20	500 ms
VA/HS	2	1.0 sec
VA/HS	3	1.5 sec
VA/HS	4	2.0 sec
VA/HS	5	2.5 sec
VA/HS	6	3.0 sec
VA/HS	7	3.5 sec
VA/HS	8	4.0 sec
VA/HS	9	4.5 sec
VA/HS	10	5.0 sec
VA/HS	11	5.5 sec
VA/HS	12	6.0 sec

The MS and HS series give 19 periods which is sufficient in most cases. The VA/MS and VA/HS detonators may be used in the same round, as the electric characteristics of the VA detonators are the same, independent of the delay times.

Recommended legwire lengths for a 4 m hole depth are 5.0 and 6.0 m.

Non-electric detonators:

	Interval numbers	Delay time	Delay time between intervals
Nonel GT/T	0	25 ms	
Nonel GT/T	1–12	100–1200 ms	100 ms
Nonel GT/T	14, 16 18, 20	1400–2000 ms	200 ms
Nonel GT/T	25, 30, 35 40, 45, 50 55, 60	2500–6000 ms	500 ms

This tunnel series gives 25 different periods and is thus even more versatile than the electric tunnel series.

7.1.5 Cuts with angled holes.

The V-cut.

The most common cut with angled holes is the **V-cut.**

A certain tunnel width is required in order to accommodate the drilling equipment. Furthermore, the advance per round increases with the width and an advance of 45 to 50 % of the tunnel width is achievable.

The angle of the cut must not be too acute and should not be less than 60°. More acute angles require higher charge concentration in the holes.

The cut normally consists of two V:s but in deeper rounds the cut may consist of triple or quadruple V:s

Each V in the cut should be fired with the same interval number using MS detonators to ensure coordination between the blastholes with regard to breakage. As each V is blasted as an entity one after the other, the delay between the different V:s should be in the order of 50 ms to allow time for displacement and swelling.

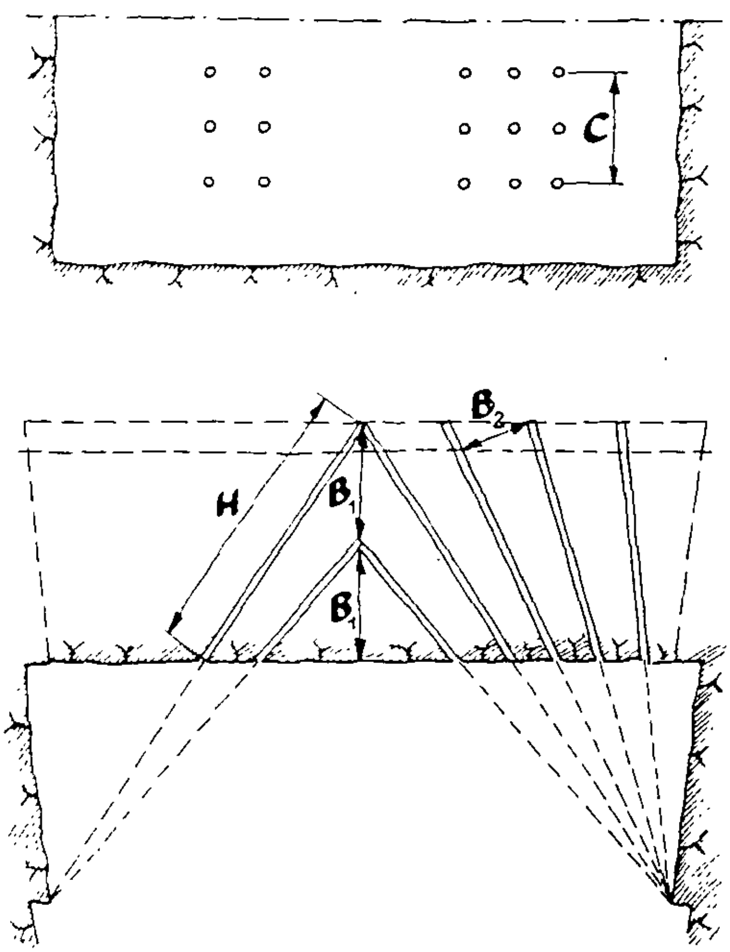


Fig 7.18 V-cut.

Recommended tube lengths for bunch blasting with Nonel are 6.0 to 7.8 m

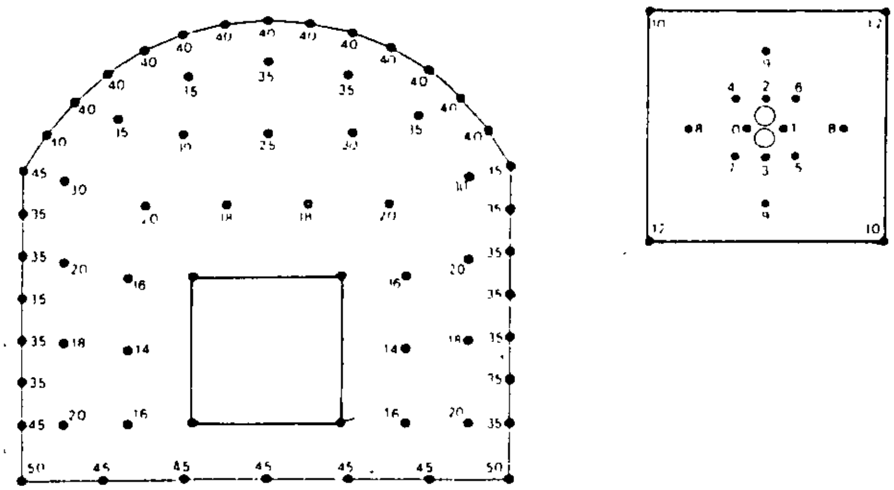


Fig. 7.16 Typical firing pattern for NONEL GT/T.

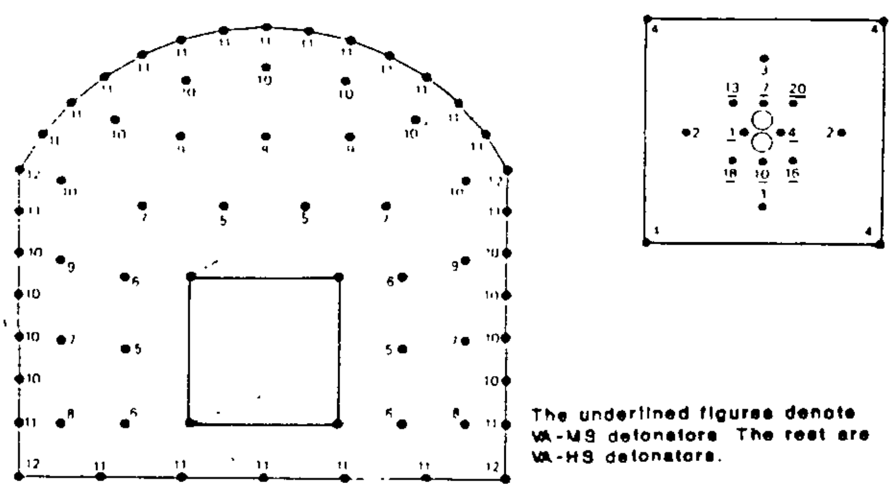


Fig. 7.17 Typical firing pattern for VA/MS and VA/HS detonators.

In the 4th square of the cut, four units of VA/HS interval No. 4 are used. This is made possible by wide range of scatter (±200 ms) within the interval for HS detonators.

Calculation of the V-cut.

The following graph (7.19) gives the height of the cut (C) and the burdens B_1 and B_2 for the cut.

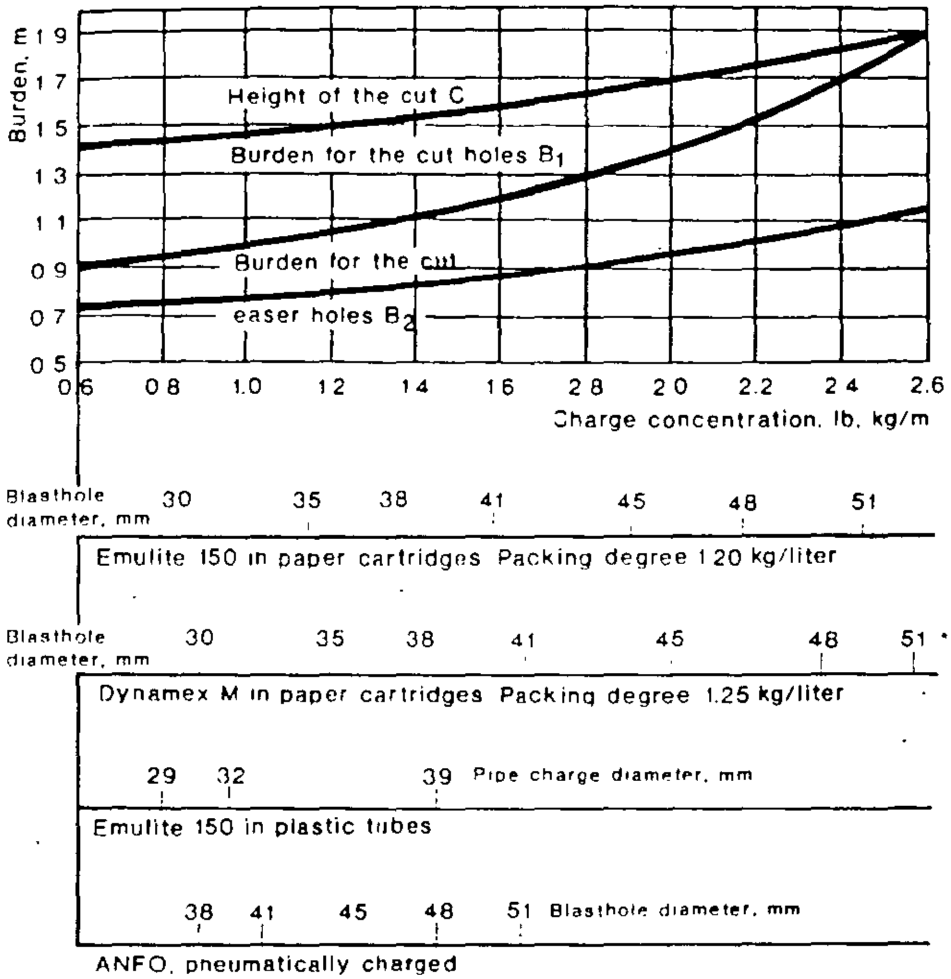


Fig 7.19 The burdens B_1, B_2 and the cut height C in relation to the bottom charge for different blasthole diameters and different explosives

Charging the cut holes.

The charge concentration in the bottom of the cut holes (l_b) can be found in graph 7.19.

The height of the bottom charge (h_b) for all cut holes is:

$$h_b = \frac{1}{3} \times H \quad \text{where } H = \text{hole depth (m)}$$

The concentration of the column charge (l_c) is:

l_c = 30 to 50 % of l_b

The uncharged part (stemming) of the holes in the cut (h_0) is:

$h_0 = 0.3 \times B_1$

The uncharged part for the rest of the cut is:

$h_0 = 0.5 \times B_2$

For the rest of the round, the method of calculation is the same as that in Chapter 7.1.2 Stoping.

The fan cut.

The **fan cut** is an other example of angled cuts. Like the V-cut, a certain width of tunnel is required to accommodate the drilling equipment to attain acceptable advance per round

The principle of the fan cut is to make a trench like opening across the tunnel and the charge calculations are similar to those in Chapter 5.6 Opening the bench. Due to the geometrical design of the cut the constriction of the holes is not large, making the cut easy to blast

The drilling and charging of the holes are similar to that of the cut holes in the V-cut

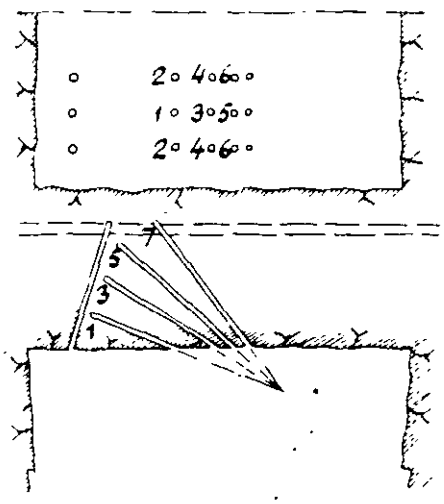
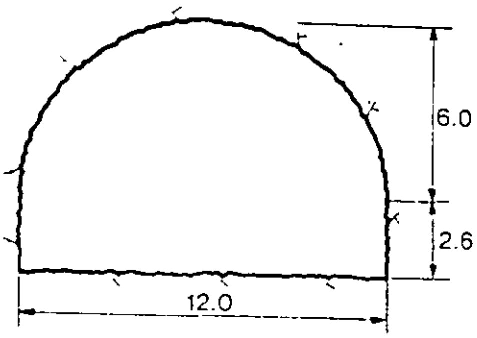


Fig 7.20 Fan cut.

7.1.6 Example of calculation.



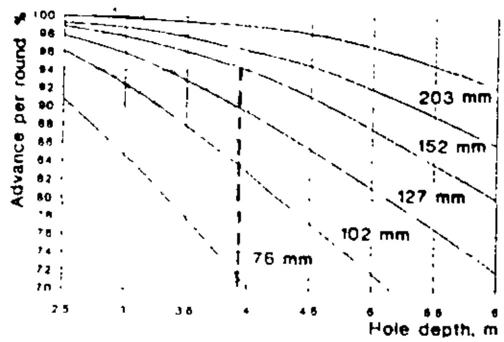
The project is a 1,500 m long road tunnel with a cross section area of 88 sq.m.

A blasthole diameter of 38 mm is chosen as the tunnel contour is to be smooth blasted. A larger blasthole diameter might cause overbreak from the stoping part of the round.

The drilling equipment is an electro hydraulic jumbo with 4.3 m steel length and feed travel of 3.9 m.

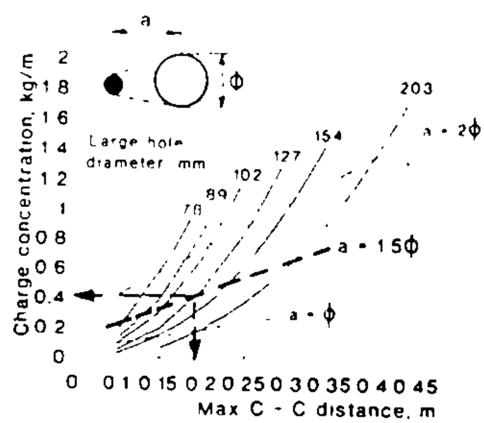
The expected advance is 95 % of the blasthole depth.

The explosive is Emulite 150 in 29 and 25 mm cartridges for the cut, stoping and floor. Gurit 17×500 mm in plastic cartridges is used for the contour. Nonel GT/T is used for initiation.



To attain an advance of more than 90 % of the blasthole depth, 3.9 m, a large hole diameter of 127 mm should be chosen.

2×89 mm large holes can be an alternative.



1st square.

The distance from the center of the large hole to the center of the closest blasthole is:

$a = 1.5\ \varnothing$

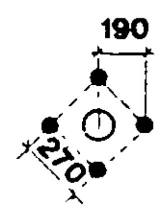
$a = 1.5 \times 127 = 190$ mm

The width of the 1st square is:

$W_1 = a\sqrt{2}$

$W_1 = 190\sqrt{2} = 270$ mm

The requisite charge concentration for the holes in the 1st square is 0.4 kg/m of Emulite 150. For practical reasons Emulite in 25×200 mm cartridges are used giving a charge concentration of 0.55 kg/m.



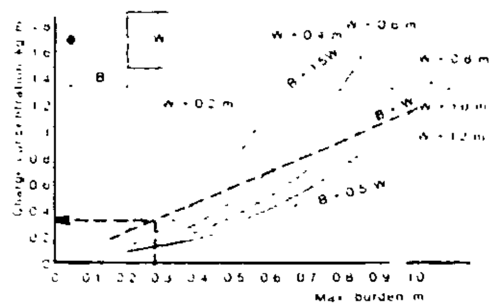
An overcharge of this magnitude does not cause any inconvenience.
The uncharged part of the hole is equal to the C−C distance $h_o = a$.
The charge of the hole is the length of the charge $H - h_o$ times the actual charge concentration.

$Q = l_c(H - h_o)$
$Q = 0.55(3.9 - 0.2)$
$Q = 2.0$ kg

Key data for the 1st square:
$a = 0.19$ m
$W_1 = 0.27$ m
$Q = 2.0$ kg.

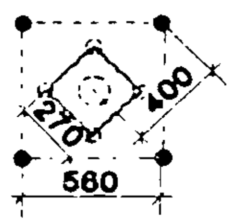
2nd square.

The blasting of the 1st square created an opening of 0.27×0.27 m. The burden in the 2nd square is equal to the width of the opening created.



$B_1 = W_1$
$B_1 = 0.27$ m
$C-C = 1.5W_1$
$C-C = 0.40$ m

$W_2 = 1.5W_1\sqrt{2}$
$W_2 = 0.56$ m



The requisite charge concentration for the holes in the 2nd square is approx. 0.37 kg/m.
Emulite 150 in 25×200 mm paper cartridges is used making the practical charge concentration 0.55 kg/m.
The uncharged part of the hole is 0.5×B.

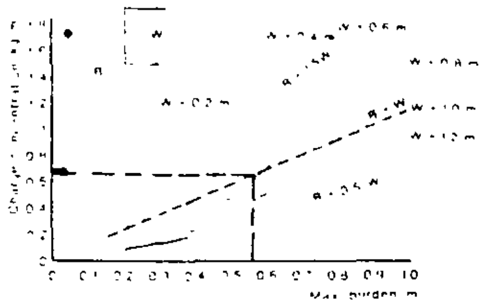
$Q = l_c(H - h_o)$
$Q = 0.55(3.9 - 0.15)$
$Q = 2.0$ kg.

Key data for the 2nd square:
$B = 0.27$ m
$W_2 = 0.56$ m
$Q = 2.0$ kg

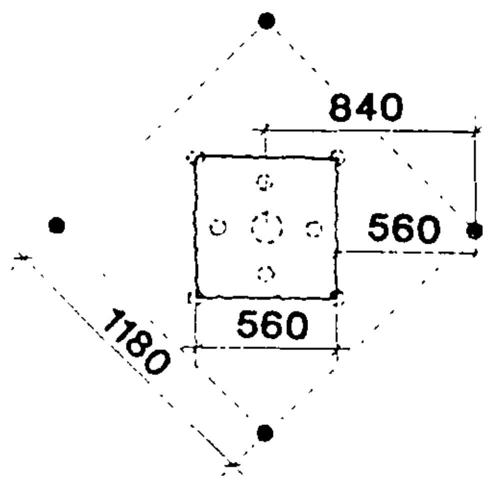
3rd square.

The opening has now a width W=0.56 m. The burden B is equal to W_2.

$B_3 = W_2$
$B_3 = 0.56$ m
$C-C = 1.5W_2$
$C-C = 0.84$ m
$W_3 = 1.5W_2\sqrt{2}$
$W_3 = 1.18$ m

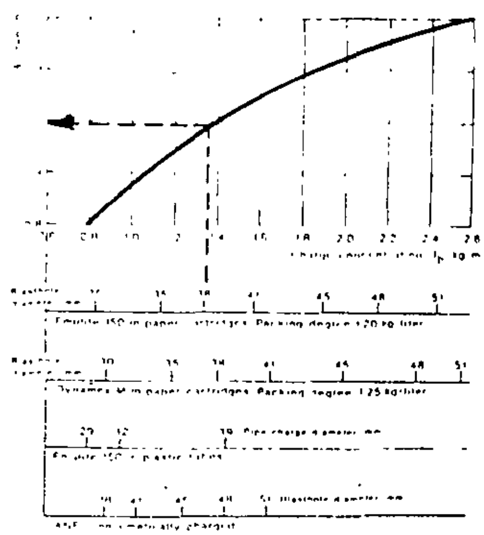


The requisite charge concentration is approx. 0.65 kg/m. Now the 25×200 mm cartridges do not provide sufficient charge concentration to ensure breakage. A larger dimension of Emulite 150 must be used unless the cartridges are tamped.
Emulite 29×200 mm in paper cartridges give a charge concentration of 0.90 kg/m. The hole will thus be overcharged.
The uncharged part of the hole is 0.5×B



$Q = l_c(H-h_0)$
$Q = 0.90(3.9-0.3)$
$Q = 3.2$ kg

Key data for the 3rd square:
B = 0.56 m
W_3 = 1.18 m
Q = 3.2 kg.



4th square.

The width of the opening is now 1.18 m. If B is chosen equal to W, the burden will be greater than that of the stoping part of the round. Therefore, the burden must be adjusted to that of the stoping part and the charge calculations are made as for stoping holes.

The burden is chosen from the graph 7.14 to 1.0 m.

The charge concentration of the bottom charge is found in the same graph to be 1.35 kg/m.
From the adjoining table the charge of the hole can be calculated.

$l_b = 1.35$ kg/m
$h_b = 1/3H$
$h_b = 0.33 \times 3.9$
$h_b = 1.3$ m
$Q_b = l_b \times h_b$
$Q_b = 1.35 \times 1.3$
$\mathbf{Q_b = 1.75}$ **kg**

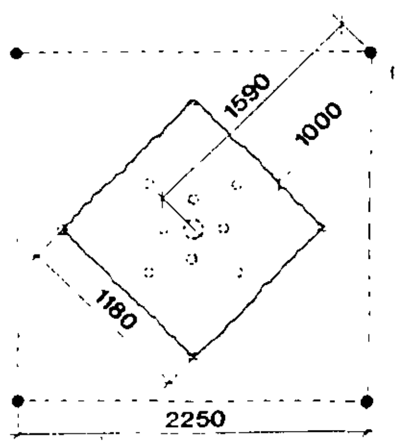
Part of the round	Burden (m)	Spacing (m)	Height of bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	1 B	1.1 B	1/3 H	l_b	1.0 l_b	0.2 B
Wall	0.9 B	1.1 B	1/6 H	l_b	0.4 l_b	0.5 B
Roof	0.9 B	1.1 B	1/6 H	l_b	0.3 l_b	0.5 B
★ Stoping						
Upwards	1 B	1.1 B	1/3 H	[illegible]	0.5 l_b	0.5 B
Horizontal	1 B	1.1 B	1/3 H	l_b	0.5 l_b	0.5 B
Downwards	1 B	1.2 B	1/3 H	l_b	0.5 l_b	0.5 B

In the bottom charge Emulite in paper cartridges with 29 mm diameter is used and tamped well

The column charge is:
$l_c = 0.5 \times l_b$
$l_c = 0.5 \times 1.35$
$l_c = 0.67$ kg/m

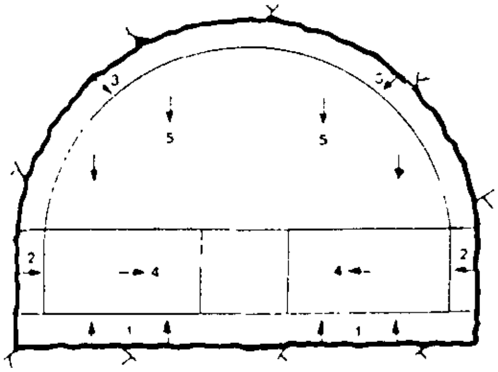
The product with dimensions closest to this is Emulite 150, 29×200 mm with an $l_c = 0.90$ kg/m

Practical $l_c = 0.90$ kg/m
$h_o = 0.5B$
$h_o = 0.5 \times 1.0 = 0.5$ m
$h_c = H - h_b - h_o$
$h_c = 3.9 - 1.3 - 0.5$
$h_c = 2.1$ m.
$\mathbf{Q_c = l_c \times h_c}$
$Q_c = 0.90 \times 2.1$
$Q_c = 1.9$ kg
$Q_{tot} = Q_b + Q_c$
$Q_{tot} = 1.75 + 1.9$
$\mathbf{Q_{tot} = 3.65}$ **kg**



Key data for the 4th square:
$\mathbf{B = 1.0}$ **m**
$\mathbf{W_4 = 2.2}$ **m**
$\mathbf{Q = 3.65}$ **kg.**

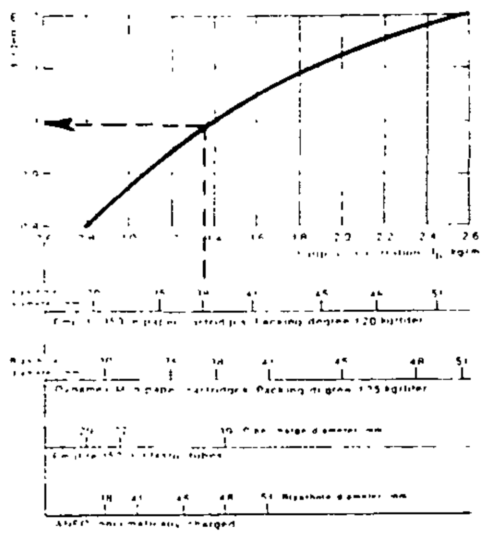
After the cut has been designed, the rest of the round is calculated.



This is most simply done in the following order:

1. Floor holes.
2. Wall holes
3. Roof holes.
4. Stoping, upwards and horizontal.
5. Stoping downwards

The reason for starting with the perimeter holes is to decide the burdens and spacings for the outer boundaries of the round.
When these calculations are completed the cut and the stoping holes may be located in accordance with the parameters which apply to them.



1. The floor holes.

In the calculation of all perimeter holes, the "**look-out**" has to be taken into account. As mentioned earlier, the "look-out" should not exceed 10 cm + 3 cm/m of hole depth. In this case the "look-out" should be limited to 20 cm.
The **burden** is 1.0 m according to the graph and the **spacing** is 1.1×B.
Due to "look-out", the holes above the floor holes are set out 0.8 m above the floor. The spacing is 1.1 m.

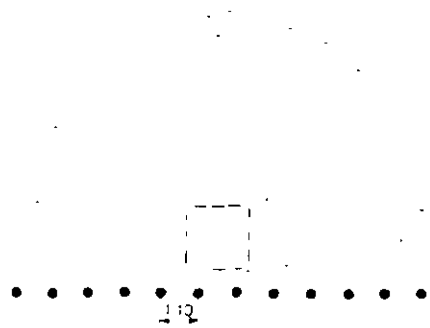
Bottom charge:
l_b = 1.35 kg/m
h_b = 1/3×3.90=1.30 m
Q_b = 1.35×1.3=1.75 kg

Column charge:
l_c = l_b=1.35 kg/m
h_o = 0.2×B=0.2 m
h_c = H−h_b−h_o=2.4 m
Q_c = 1.35×2.4=3.25 kg

Total charge:
Q = 1.75+3.25=5.0 kg

Part of the round	Burden (m)	Spacing (m)	Height of bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	1×B	1.1×B	1/3×H	l_b	1.0×l_b	0.2×B
Wall	0.9×B	1.1×B	1/6×H	l_b	0.4×l_b	0.5×B
Roof	0.8×B	1.1×B	1/6×H	l_b	0.3×l_b	0.5×B
Stoping						
Upwards	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
Horizontal	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
Downwards	1×B	1.2×B	1/3×H	l_b	0.5×l_b	0.5×B

Key data for floor holes:
B = 1.0 m
S = 1.1 m
Q = 5.0 kg.



2. The wall holes.

In this particular case the walls are very low and do not make a good example for the design of the drilling and charging pattern

The drilling pattern is taken from the **smooth blasting** table and the burden is chosen to 0.8 m and the spacing to 0.6 m.

The uncharged part of the hole is 0.2 m

The charge concentration for Gurit 17×500 mm is 0.23 kg/m The holes will be charged with 7 tube charges and 1 stick of Emulite 150, 25×200 mm in the bottom.

Bottom charge
$Q_b = 0.11$ kg

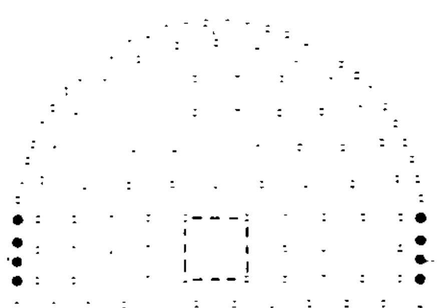
Column charge
$Q_c = 7 \times 0.115 = 0.81$ kg

Total charge.
$Q = 0.11 + 0.81 = 0.92$ kg

The "look-out" has to be considered, so the burden to be set out on the face is $0.8 - 0.2 = 0.6$ m.

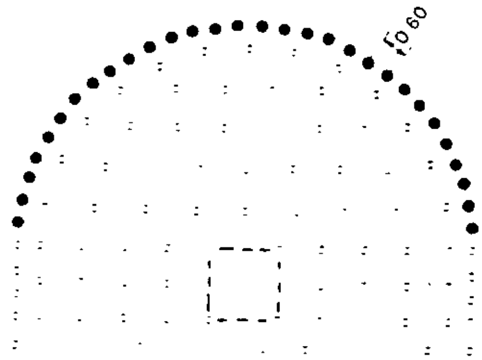
Key data for the wall holes:
B = 0.8 m
S = 0.6 m
Q = 0.92 kg

[illegible]	[illegible]	Charge type	Burden m	Spacing m
[illegible]		[illegible] mm Gurit	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	1.0 [illegible]	[illegible]
[illegible]	[illegible]	22 mm Emulite	[illegible]	0.80 [illegible]



3. The roof holes.

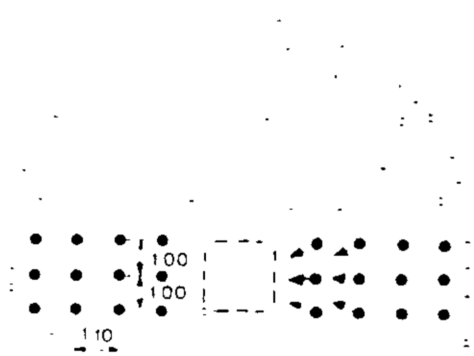
The conditions for the roof holes are equal to those of the wall holes. The burden is chosen to 0.8 m and the spacing to 0.6 m.
The charge concentration is the same as for the wall holes.
The "look-out" must be considered in this case as well.



Key data for the roof holes:
$B = 0.8$ m
$S = 0.6$ m
$Q = 0.92$ kg.

4. Stoping upwards and horizontally.

The stoping holes are calculated in a similar way to the floor holes, but less explosives are needed. While the floor holes must be charged to compensate for gravity and heavage of broken rock, the stoping holes can normally contain less explosives as the direction of breakage is horizontal or close to horizontal.
Charge: Bottom, tamped Emulite 29 mm, $l_b = 1.35$ kg/m.
Charge: Column, Emulite 29 mm in paper cartridges with $l_c = 0.90$ kg/m
The burden B is 1.0 m, according to the graph 7.14.



The spacing S will be 1.1 m according to adjoining table.

Bottom charge:
$l_b = 1.35$ kg/m
$h_b = 1/3 \times 3.90 = 1.30$ m
$Q_b = 1.35 \times 1.3 = 1.75$ kg

Column charge:
$l_c = 0.90$ kg/m
$h_o = 0.5 \times B = 0.5$ m
$h_c = H - h_b - h_o = 2.1$ m
$Q_c = 0.90 \times 2.1 = 1.9$ kg

Total charge:
$Q = 1.75 + 1.9 = 3.65$ kg

Part of the round	Burden (m)	Spacing (m)	Height bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	1×B	1.1×B	1/3×H	l_b	1.0×l_b	0.2×B
Wall	0.9×B	1.1×B	1/6×H	l_b	0.4×l_b	0.5×B
Roof	0.9×B	1.1×B	1/6×H	l_b	0.3×l_b	0.5×B
Stoping						
★ Upwards	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
★ Horizontal	1×B	1.1×B	1/3×H	l_b	0.5×l_b	0.5×B
Downwards	1×B	1.2×B	1/3×H	l_b	0.5×l_b	0.5×B

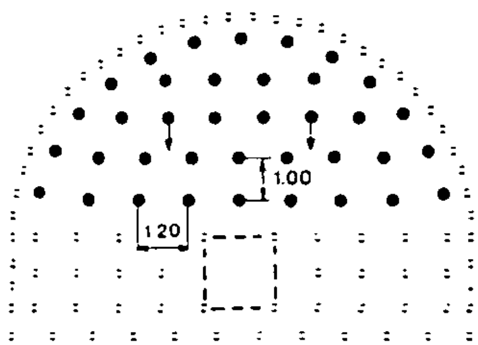
Key data for stoping holes upwards and horizontal:
B = 1.0 m
S = 1.1 m
Q = 3.65 kg

Part of the round	Burden (m)	Spacing (m)	Height bottom charge (m)	Charge concentration Bottom (kg/m)	Charge concentration Column (kg/m)	Stemming (m)
Floor	1 × B	1.1 × B	1/3 × H	l_b	1.0 × l_b	0.2 × B
Wall	0.9 × B	1.1 × B	1/6 × H	l_b	0.4 × l_b	0.5 × B
Roof	0.9 × B	1.1 × B	1/6 × H	l_b	0.3 × l_b	0.5 × B
Stoping						
Upwards	1 × B	1.1 × B	1/3 × H	l_b	0.5 × l_b	0.5 × B
Horizontal	1 × B	1.1 × B	1/3 × H	l_b	0.5 × l_b	0.5 × B
Downwards	1 × B	1.2 × B	1/3 × H	l_b	0.5 × l_b	0.5 × B

5. Stoping downwards.

The design of the drilling pattern for stoping downwards is similar to stoping in other directions with the difference that larger spacing may be permitted. The charge of the holes is the same in all stoping.

Key data for stoping holes downwards:
B = 1.0 m
S = 1.2 m
Q = 3.65 kg



SUMMARY

The round consists of 127 blastholes with 38 mm diameter and 1 large hole with 127 mm diameter.

The round is charged as follows:

Part of the round	No. of holes	Kind of explosive	Weight per hole (kg)	Total (kg)
Cut				
1st square	4	Emulite 150, 25 mm	2.0	8.0
2nd square	4	Emulite 150, 25 mm	2.0	8.0
3rd square	4	Emulite 150, 29 mm	3.2	12.8
4th square	4	Emulite 150, 29 mm	3.65	14.6
Floor holes	12	Emulite 150, 29 mm	5.0	60.0
Wall holes	8	Emulite 150, 25 mm	0.11	0.9
		Gurit 17 mm	0.81	6.5
Roof holes	30	Emulite 150, 25 mm	0.11	3.3
		Gurit 17 mm	0.81	24.3
Stoping:				
Upwards	8	Emulite 150, 29 mm	3.65	29.2
Horizontal	16	Emulite 150, 29 mm	3.65	58.4
Downwards	37	Emulite 150, 29 mm	3.65	135.1

Consumption per round:	Emulite 150, 25×200 mm	20.1 kg
	Emulite 150, 29×200 mm	310.1 kg
	Gurit	30.8 kg
	Nonel GT/T	127 units

The expected advance per round is over 90 %. It is assumed to be 3.55 m.

Specific charge: $\dfrac{361.1}{3.55\times 88.0} = 1.16$ kg/cu.m.

Explosives consumption for the whole project:
Number of rounds: 1500/3.55=425
Consumption of

Emulite 150, 25×200 mm	**20.2×425**	**= approx. 9 tons**
Emulite 150, 29×200 mm	**310.1×425**	**= approx. 132 tons**
Gurit	**30.8×425**	**= approx. 13 tons**
Nonel GT/T	**127×425**	**= approx. 54000 units.**

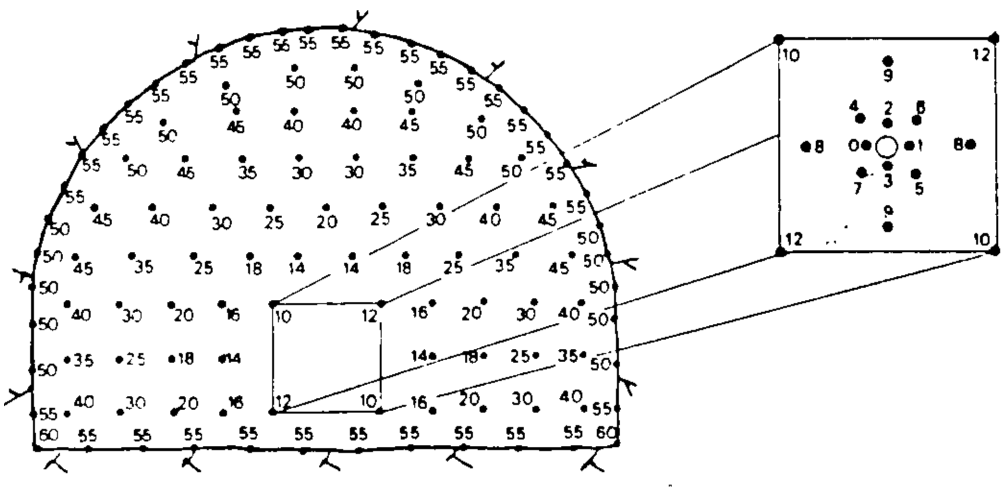


Fig. 7.21 Drilling and firing pattern.

7.2 Shafts.

In mining, shafts form a system of vertically or inclined passageways which are used for transportation of ore, refill, personnel, equipment, air, electricity, ventilation etc.

In underground construction, shafts are driven for the building of penstocks, cable shafts, ventilation and elevator shafts, surge chambers etc. In addition, shafts are driven as "glory holes" for transportation of material which is not accessible by other means than vertical or close to vertical tunnels.

Shafts are either driven downwards, sink shafts, or upwards, raise shafts.

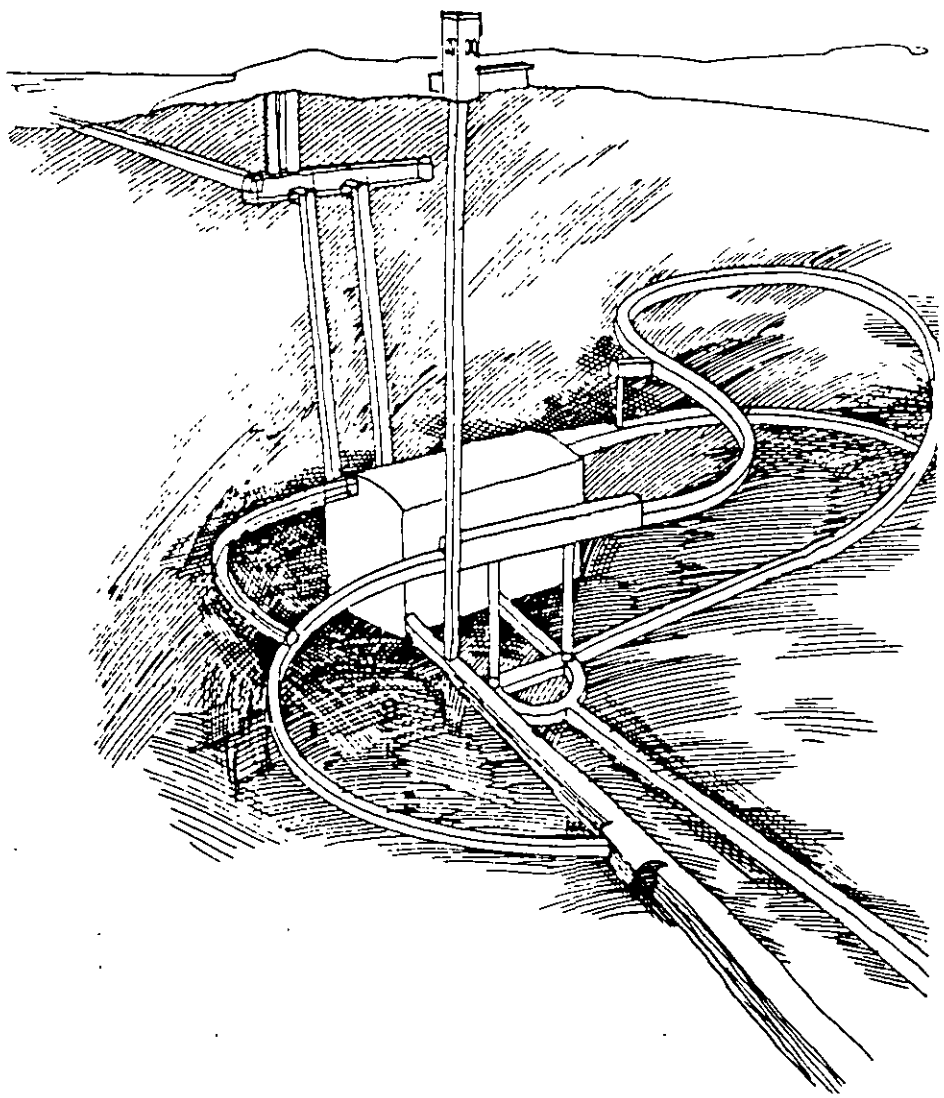


Fig. 7.22 Typical tunnel system in a hydroelectric power plant.

7.2.1 Sink shafts.

Sink shafts are passageways sunk from the surface downwards or underground from one level to a lower one. The majority of the sink shafts are driven vertically.

Shaft sinking is one of the most difficult and risky blasting jobs as the work area is normally wet, narrow and noisy. Furthermore, the drilling and blasting crews are exposed to falling objects.

The advance is slow as the rock has to be removed between each blast with special equipment which has limited digging capacity. The blasted rock must be well fragmented to suit the excavation equipment.

The design of the cross section of the shaft principally depends on the quality of the rock. Nowadays most of the shafts are made with a circular cross section which gives better distribution of the rock pressure, thus decreasing the need for reinforcement, especially in deep shafts.

The most common drilling and blasting methods are benching and blasting with pyramid cut.

The **benching** method is a fast and efficient method as the time-consuming cleaning of the floor between the blasts can be minimized. It is also easy to keep the shaft free from water as a pump can always be placed in the lower blasted part of the shaft. The drilling and charging pattern is similar to that of smaller surface blastings.

The burden and spacing vary with the hole diameter but the drilling pattern is more closely spaced than for surface blasting due to higher constriction.

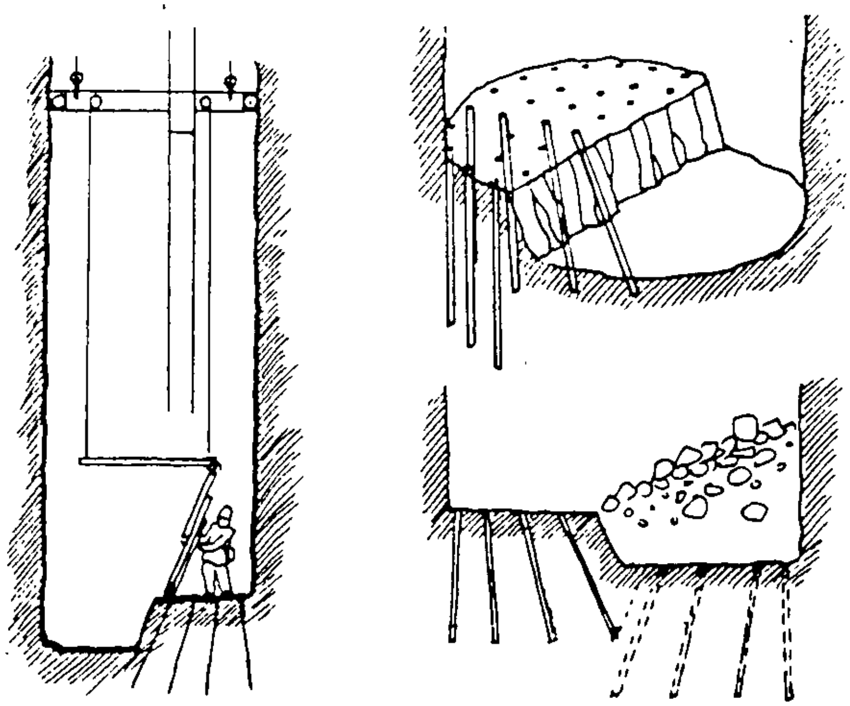


Fig. 7.23 Shaftsinking by benching.

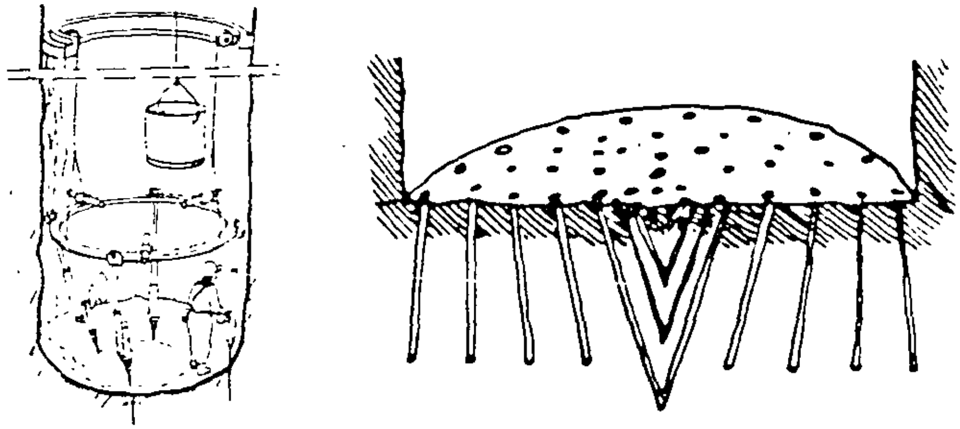


Fig. 7.24 Shaft sinking with pyramid cut.

Shaft sinking with **pyramid cuts** is similar to tunnel blasting with V-cuts. The drilling is done with a "drill-ring" which is composed of a circular I-beam to which the drilling machines are fixed. The "drill-ring" may be fixed to the shaft walls with bolts. Due to the construction of the "drill-ring", the cut will be conical.

The explosives used in shaft sinking must always be water resistant. Even if the ground is dry, the flushing water from the drilling will always stay in the blastholes.
For this reason explosives with excellent water resistance properties are preferred. Emulite 150 and Dynamex M are easily tamped to utilize the hole volume well, thus decreasing the number of holes and the drilling and charging time. The specific charge in shaft sinking is rather high, ranging from 2.0 kg/cu.m. to 4.0 kg/cu.m.

The initiation of the blast may be done with electric detonators or non-electric detonators. As a sink shaft is a small confined area, thunderstorms are a particular hazard as stray currents tend to be transmitted down the shaft on pipes and cables. To avoid problems with evacuation of the blasting crew during a thunderstorm, NONEL detonators should be used.

7.2.2 Raise shafts.

The drifting of raise shafts – shafts which are driven from blasted underground chambers or tunnels, vertically or inclined upwards – is one of the most difficult, most costly and most dangerous undertakings in mining and construction.
As the drifting of raise shafts has increased in the world, new methods have been developed to make the work more mechanized, cheaper and safer.
Raise shafts were drifted in more or less the same way for decades until the 1950's when new types of raise shaft elevators were taken into use.

Various raise shaft drifting methods where blasting is part of the method.

Older methods:

* Timbered shafts
* Open shafts

Modern methods:

* Boliden elevator type Jora
* Alimak Raise Climber
* Longhole drilling

To start with the older methods, the timbered shaft method was the most common method in Sweden until some 40 years ago and is still occasionally used for shorter shafts. The raise shaft is driven vertically and divided into two sections by a timber wall which is extended before each blast. When the round is fired, one section is filled with rock. The blasted rock will then act as a working platform for the next round. In order to maintain the working height at the face some rock has to be excavated after each blast. The second section is used as a ladderway and for transportation of equipment, drill steel, explosives and timber. The ventilation is also placed in this section which is covered during blasting.

Timbered raise shafts have been driven up close to 100 m, but normally the maximum height should not exceed 60 m. The cross section area is usually 4 sq.m. and the advance per round approx. 2.2 m.

Fig. 7.25 Timbered raise shaft.

The timbered shaft method was replaced by open shaft methods when the cost of timber became too high. In one of these methods a working platform of planks is laid on timber which is supported by bolts in the shaft walls. New bolt holes are drilled in the shaft walls when the round is drilled so the platform can be moved upwards as the work proceeds.

Another open shaft method is to use steel tubes instead of timber. The steel tubes are bolted to the shaft walls and the tubes support the platform.

The open shaft methods are rarely used and when used, only for short raises, up to 25 m. From a safety point of view none of the open shaft methods is to be recommended.

The cross section is normally 4 sq.m. and the advance approx. 2.2 m.

The JORA lift method.

Raise shafting using a lift cage hanging on a wire which runs through a large drillhole has been used in Sweden and other countries since the 1940's, but it was not until the 1950's when Boliden AB developed the JORA lift, that the method came into wider use.

A large hole, diameter 110 to 150 mm, is drilled from an upper level in the center of the intended shaft. Through the hole a wire is sunk down to the lower level and a working platform with a lift cage is fastened to it. By a lifting gear the platform is elevated up to the shaft face by remote control from the lift cage. The drilling and charging are carried out from the platform on the top of the lift cage and some scaling can be done from the cage with the protection of the platform. During the scaling, drilling and charging operations the platform is fixed with bolts to the shaft walls. Before blasting the platform is lowered down and placed on a sledge like vehicle and towed aside. The wire is lifted up through the large hole before blasting. The large hole is used as cut hole in the blasting of the round. Due to the large size of the cut hole, advances of up to 4 m are obtained. The area is approx. 4 sq.m. and the maximum height is 100 m. In this method it is necessary to have free space above the shaft for the drilling of the large hole and for the placing of the lifting gear.

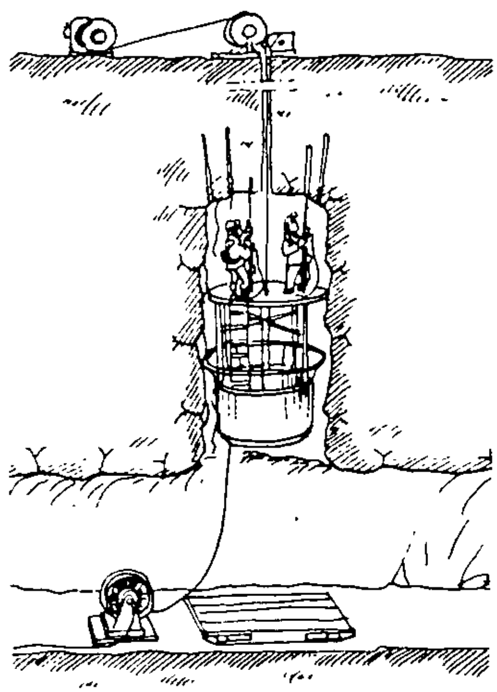


Fig. 7.26 The JORA lift.

The ALIMAK Raise Climber.

The Alimak raise shaft driving method was introduced in 1957 and became the most utilized system in the world because of its flexibility, safety, economy and speed.

The equipment consists of a raise climber with a working platform, which covers practically the entire area of the shaft. Under the platform there is a cage for the transport of personnel, material and equipment. The raise climber is propelled by a rack and pinion system along a special guide rail. The rail system incorporates a tube system for the air and water supply to the drilling equipment. The system also provides air for the blasting with NONEL and to ventilate the raise after the blasting.

The platform is equipped with a protective roof under which the blaster stands during scaling and drilling operations. If the inclination of the raise shaft is 60° or less the scaling may be done gradually during the ascent under the protection of the previously scaled hanging wall.

The Alimak method can be used for vertical as well as inclined shafts. The lower limit of the inclination depends on the angle of repose of the rock.

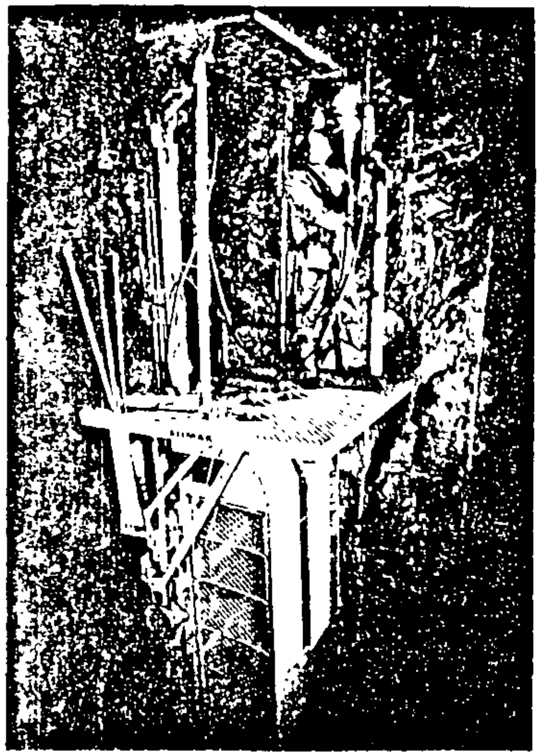


Fig 7.27 The ALIMAK Raise Climber.

Unlike other modern methods for raise shafting, the Alimak needs only one point of attack, the lower one. The upper break-through point may be prepared while the raise is driven.

The lengths which may be driven are only limited by the time which is at the blasting crews' disposal for ascent, scaling, drilling, charging, descent and blasting. For an 8 hour shift, the upper limit should be around 2,000 m. The lengths are also limited by the type of drive. The air-driven raise climber may be used for up to 150 m shaft length, electric drive up to 900 m. For longer shafts diesel-hydraulic driven climbers are used.

The area is normally 4 sq.m., but inclined shafts have been driven full face upp to 36 sq.m.

Drilling and charging patterns are the same for all above mentioned raise shafting methods. Normally a raise shaft of 4 sq.m. is driven upwards and then the shaft is stoped to its final area. However, sometimes the shaft is driven "fullface" and as mentioned earlier areas up to 36 sq.m. have been successfully blasted.

The drilling and firing pattern for a raise shaft does not differ from that of a horizontal tunnel of the same size.

The Alimak work cycle:

Drilling:

The drilling and charging is carried out from the raise climber's platform under a specially designed protective roof. Both air and water to the drilling machines are supplied through tubes in the guide rail sections.

Blasting:

After drilling and charging the round, the raise climber is driven to the bottom and under the roof of the drift. During the blast, the climber is therefore well protected from falling rock.

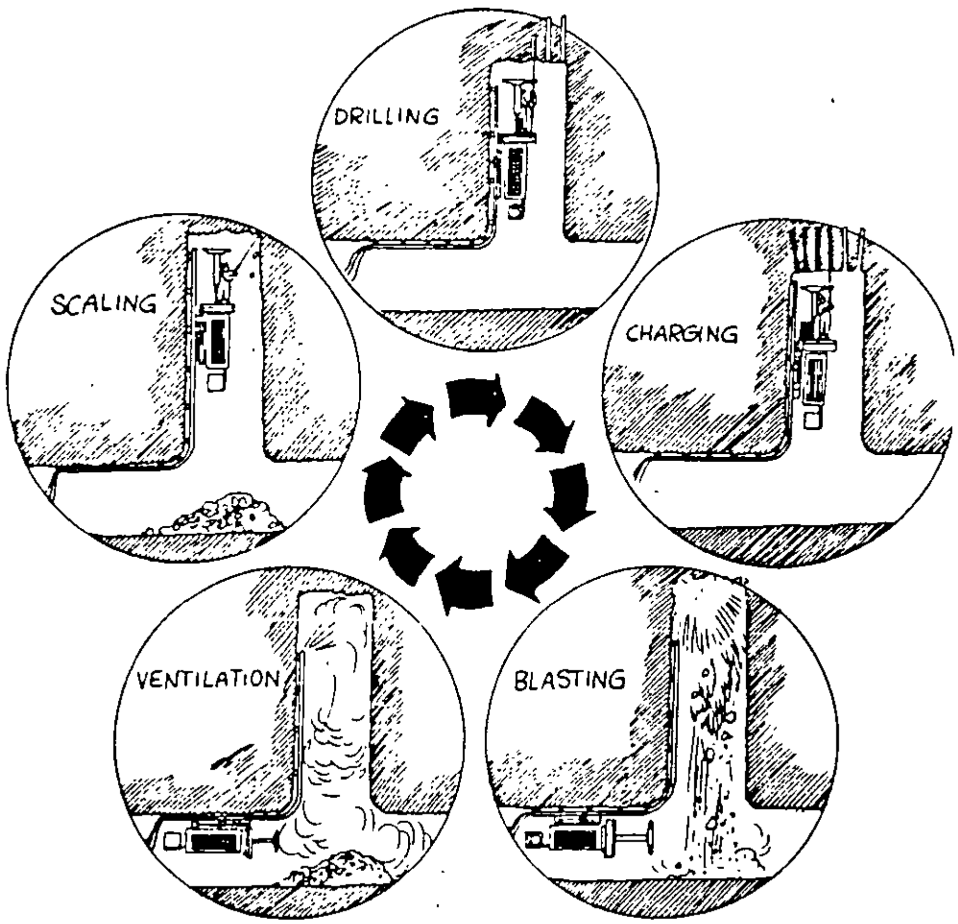


Fig. 7.28 The ALIMAK work cycle.

Ventilation:

After blasting the raise is ventilated and sprayed with water. The top of the guide rail is protected by a header plate which also acts as a water diffuser during the ventilation phase.

Scaling:

Scaling of the roof and walls of the raise is done from under the protective roof which gives the workmen good protection.

Generally large hole cuts are used and the design of the cut varies with the diameter of the large hole. (See 7.1.1 The cut, in Chapter Tunneling.)

The normal hole depth is 2.4 m and the expected advance 2.1 to 2.2 m.
The drilling is done with stopers, which are designed for raise driving, overhead drilling and roof bolting or drilling machines with jack legs
For the blastholes drill series 11 (34 to 32 mm) is used and the large hole diameter is normally 75 mm.
For the stability of the walls and to avoid overbreak, the walls of the raise are normally smoothblasted. . The smooth blasting method is also used if the shaft is to be widened at a later stage in order to avoid excessive scaling and to decrease the risk of rockfall.

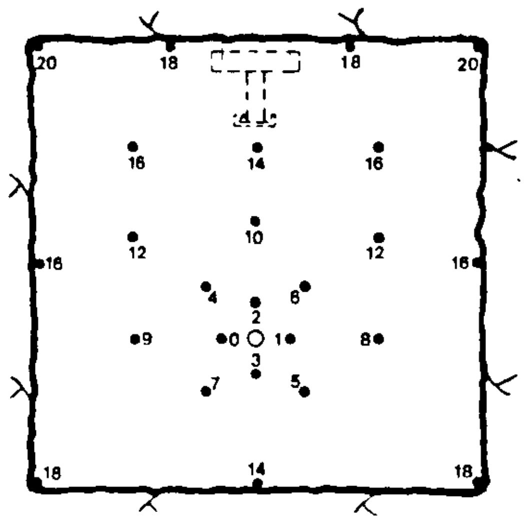


Fig 7.29 Drilling and firing pattern for 4 sq.m. raise shaft

A normal pilot shaft has an area of 4 sq.m. Normally one round is drilled and blasted per shift with an advance of 2.2 m. Working 2 shifts per day, the advance should be 4.4 m but taking disturbances in the work cycle into account, the long term advance is approx. 3.5 m/day or 70 to 90 m per month.

Shaft raising by long hole drilling.

In this method, all drilling is done downwards with parallel holes and the whole area is drilled at the same time:
Great precision in drilling and charging is a must and the lack of precision has earlier limited the practical height to 25 to 30 m. Now, with new drillrigs e.g. Atlas Copco Simba, the drilling can be carried out with great precision in any direction from vertical to 50°. With the Simba the deviation can be kept under 0.5 % for holes up to a length of 50 m

The long hole drilling method is also advantageous from a safety point of view as all drilling and charging work is carried out from a safe location.

Two different cuts are used:

- large hole cut (blasting towards a large hole).
- crater cut (blasting towards the lower free face of the raise).

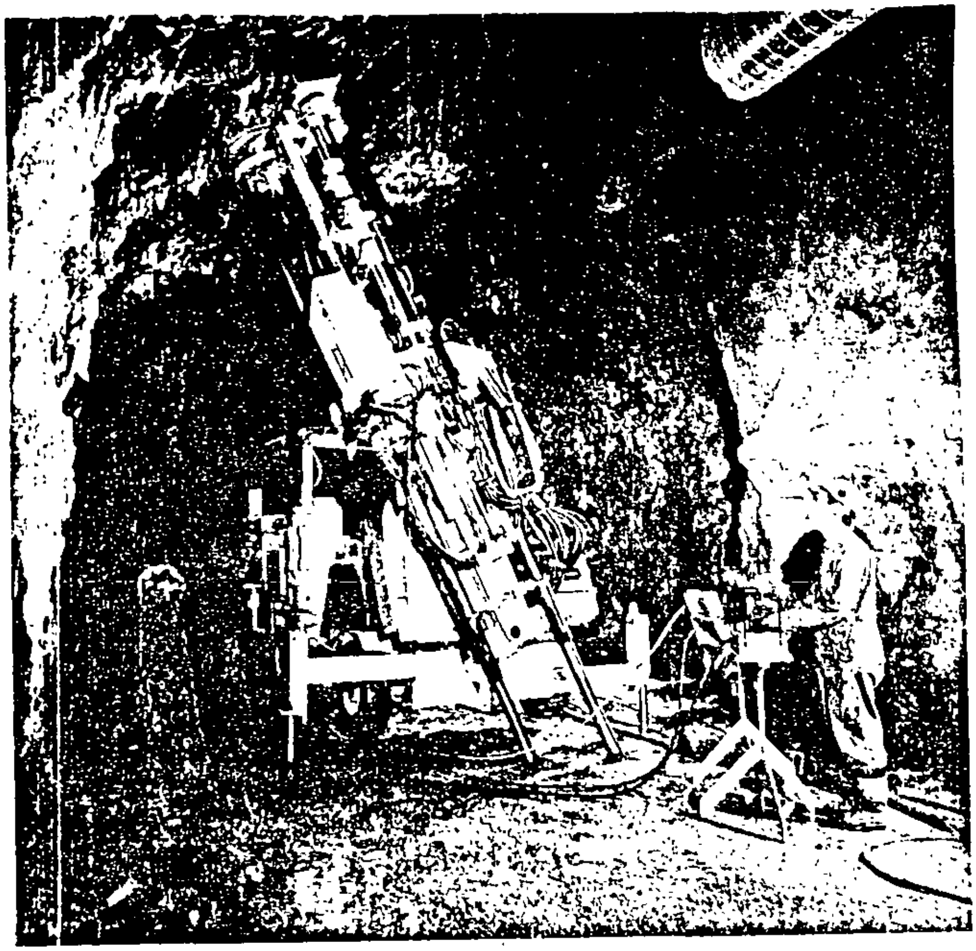


Fig. 7.30 Simba.

The large hole cut came first and is still the most common one.
The drill holes in the round have a diameter of 50 to 75 mm and the central large hole is reamed to a diameter of 102 to 203 mm.

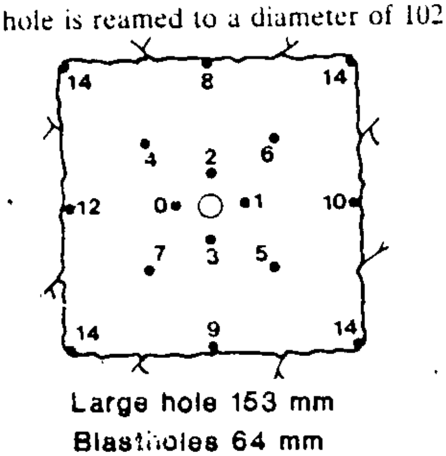


Fig. 7.31 Firing sequence for 4 sq.m. raise.

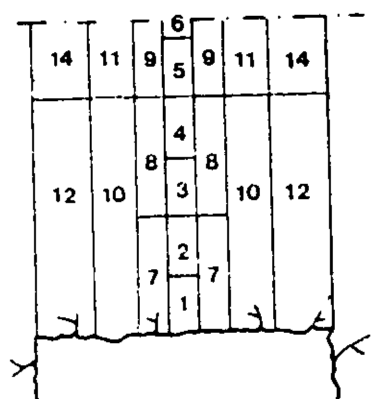


Fig. 7.32 Round sequence for raises with larger cross section.

The design and charging of the cut follow the same principles as described in Chapter 7 11 Tunneling. The cut. The firing sequence depends on the faulty drilling so the hole with the smallest real burden is fired with the lowest period number. It is therefore necessary to map every hole with regard to the faulty drilling.

The charging is done from the upper level. A piece of wood is lowered down on a rope and when the wood passes the lower mouth of the hole the rope is tightened and the piece of wood forms a plug for the lower part of the hole. The charges are lowered to the bottom of the hole. The hole should not be stemmed as the stemming may sinter and block the hole for the subsequent blast. The holes may be relatively overcharged compared with a tunnel cut as the charges are not confined at either end. Furthermore, the blastholes are normally of larger diameter than those used in tunnels. The risk of recompaction of the rock in the cut section can be considered as low even if the holes are considerably overcharged.

Crater blasting.

The blasting of a long hole drilled raise can also be carried out towards the free lower surface of the raise with a crater cut. No large diameter center hole is needed but the blastholes normally have a larger diameter than in the previous method. The crater blasting method is used only for the cut section to open a hole of approx. 1 sq.m., then normal stoping will follow.

The crater cut consists of five holes, one center hole and four edge holes. The center hole is blasted first whereupon the edge holes are blasted one by one with different delays.

Before charging, the holes are plugged with a piece of wood which is lowered down from the upper surface on a rope and secured to the lower rock surface. The hole is then filled with sand to the calculated level of the explosives charge. The charge should have a diameter close to that of the hole.

The charge is then stemmed with water. (Any other stemming may sinter and block the hole, making subsequent blasting operations impossible.)

The requisite charge weight and depth of the charge are calculated from Livingstone's theories as follows:

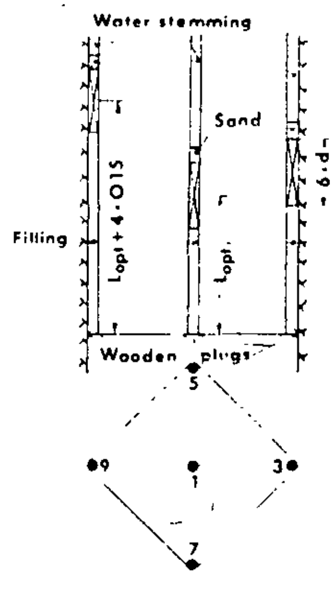


Fig. 7.33 Drilling, charging and firing pattern for crater cut.

1. The length of the charge shall be 6 times the blasthole diameter.

$$l = 6\times d \qquad \text{(mm)}$$

2. The optimum depth of the charge is 50 % of the critical depth.

$$L_{opt} = 0.5\times L_{crit} \qquad \text{(mm)}$$

3. The critical depth depends on the charge weight.

$$L_{crit} = S\times Q^{1/3} \qquad \text{(mm)}$$

where S = the strain energy factor approx. 1.5 (depending on the explosive used and the type of rock)
Q = charge weight in kg.

4. The charge weight is then

$$Q = \frac{3\times d^3\times \pi\times p}{2} \qquad \text{(kg)}$$

where p = charging density (1.2 kg/liter for Emulite 150 and 1.35 kg/liter for Dynamex M)

5. The optimum charge depth is then related to charge weight, explosives density, blasthole diameter and strain energy factor as follows:

$$L_{opt} = 0.5\times S\times \sqrt[3]{\frac{3\times \pi\times d}{2}}\times d\times 10 \qquad \text{(mm)}$$

The crater theory is valid only for the center hole. The charge of the edge holes is placed so that the burden is less than the charge depth of the crater hole. The charge depth increases with 10 to 20 cm between each hole.

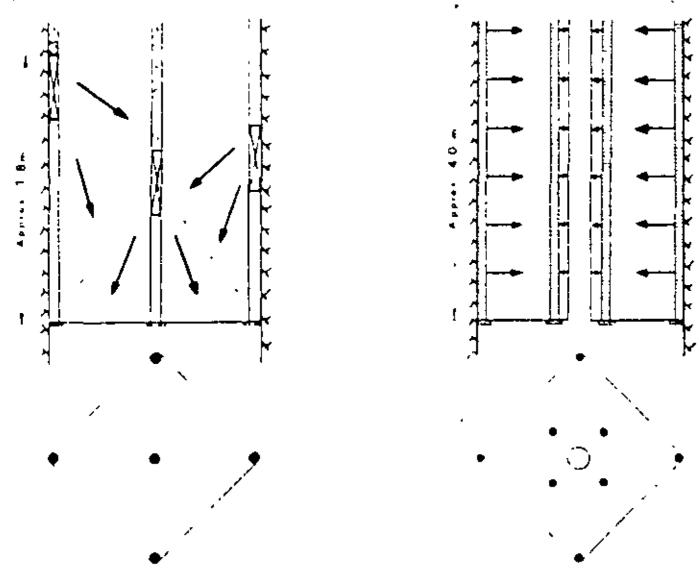


Fig. 7.34 Comparison of crater cut and standard large hole cut.

The advantages with crater cut compared to large hole cut are:

1. Lower cost for drilling and explosives as less holes are drilled in the cut. The same hole diameter is used in all holes.
2. Drilling precision is not as essential as for large hole cuts.
3. Simpler blasting practice with less need for well trained personnel.

The disadvantage with the crater cut method is the relatively short rounds that may be shot each time

7.3 Underground chambers.

The military defense forces started early to utilize solid rock for construction of fortifications which gave many advantages over surface construction. Solid rock is difficult to penetrate and underground chambers are difficult to discover and easy to guard.

The field of application is huge: Protection for guns, ammunition and soldiers, protection for submarines and smaller warships, storage for material, fuels and foodstuffs and not least as air-raid shelters for civilians

Oil was initially stored in surface tanks, but after WWII storage in unlined storage chambers has become the most common method. The increased exploitation of sub-surface storage has to a great extent been due to the rapid development of rock blasting techniques. The increased mechanization of the operations has resulted in relatively unchanged construction costs over a number of years, while at the same time the price of land has increased considerably.

Common to all types of underground chambers is that they are well protected from a military point of view. They are well camouflaged and more difficult to damage than surface storage facilities if attacked from the air or overland. They require little land: surface space is only needed for access roads, ventilation etc. From an environmental point of view sub-surface storage is safer, as leakage does not often occur from underground chambers. It is safer than surface storage in case of fire, as the supply of oxygen is often insufficient to allow a bigger fire to develop.

Underground chambers have many fields of application:

- storage for different products
  cold storage for food, wines, water, oil etc.
- garages, telephone exchanges, swimming pools
- military and civil stores and workshops
- air-raid shelters for people
  aircrafts
  warships
  archives
- storage for lightly contaminated nuclear waste
- storage of nuclear residue
- hydro-electric powerstations

Some of the applications may be combined. In wartime, the space which is normally used for garages, workshops or swimming pools can be utilized as air-raid shelters.

The basis for underground chambers is a qualitative sound rock to build in. Some economic aspects have to be considered. If the chamber is located at too shallow a level, the cost of reinforcing the rock may be high as the quality of the surface rock is normally poorer than rock at deeper levels. However, deep location results in long access roads, which may cause problem both during construction and when the chambers come into use.

From the point of view of rock blasting techniques, the construction of underground chambers does not differ from that of tunnels of the same magnitude. The width of underground chambers cannot be too great due to the inability of the rock to support the roof by its own strength. For oil storage chambers and machine halls for hydro-electric power-plants, widths of 20 to 24 m have been constructed without need for heavy reinforcement. The height of the chambers may be up to 40 m.
Small underground chambers, with a height of less than 8 m are blasted as tunnels. In larger chambers, the operation has to be divided into several stages of drilling and blasting in which different methods are used:

- pilot tunnel with side stoping
- horizontal benching
- vertical benching.

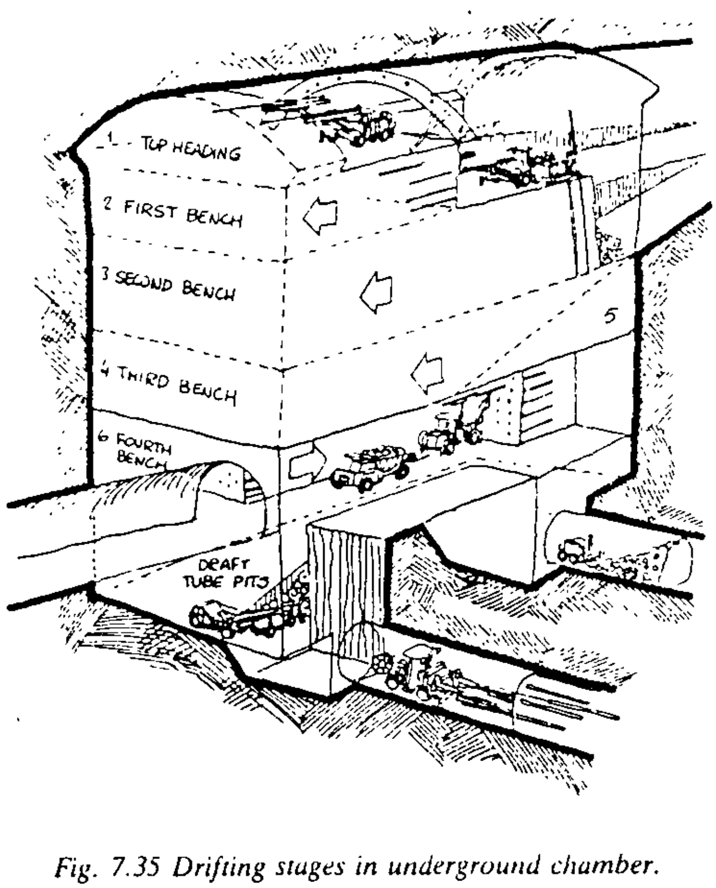


Fig. 7.35 Drifting stages in underground chamber.

The pilot tunnel is drifted at the roof of the chamber to facilitate scaling and reinforcement. The side stoping to full width is then carried out. Scaling and, if necessary, bolting and shotcreting of the roof are done simultaneously to avoid future expensive reinforcement work.
Then blasting is carried out in one or several benches. It is common for the first bench to be a horizontal bench utilizing the drilling equipment for the tunnel. Some rock chambers are also designed in such way that no space is available close to the wall for the boom of the vertical drilling equipment. The disadvantage with horizontal benching is that the height and depth of the round depends on the drilling equipment. The height is normally limited to 8 m and the depth of the round to 4 m. Other limitation on the blast design is that the blasthole diameter can rarely exceed 51 mm.
Excavation of the blasted material must be carried out between each blast.
Vertical benching is the dominant method for benching in rock chambers. The advantages with vertical benching is that drilling and excavation may be carried out simultaneously. The bench height may be varied within a wide range and larger blastholes may be used, often with better economy as a consequence. It is also easier to obtain a smoother contour with vertical benches than with horizontal.
The charge calculations for the pilot tunnel, side stoping and horizontal benching are the same as presented in Chapter 7 Tunneling, where the side stoping is calculated as stoping holes with horizontal breakage and the vertical bench as stoping holes with upwards breakage.
The vertical benching is calculated in accordance with Chapter 5 Bench blasting. If excavation is not carried out between the blasts, the specific charge has to be increased in order to compensate for movement of rock from previous rounds. See 5.8 Swelling.
Access tunnels are required for each bench for the transport of rock and equipment.

In certain cases, restrictions due to geological reasons, ground vibrations etc., may affect the execution of the work.
In Fig. 7.36 the roof must be bolted with 8 m long bolts and sprayed with concrete before any side stoping can be done.
The vertical bench is limited to a height of 4 m which makes it feasible to make a raise shaft, "glory hole", for the transport of the blasted rock. The raise shaft is a long hole drilled one, from the upper level and the blasting starts at the lower level. See Chapter 7.2.2.

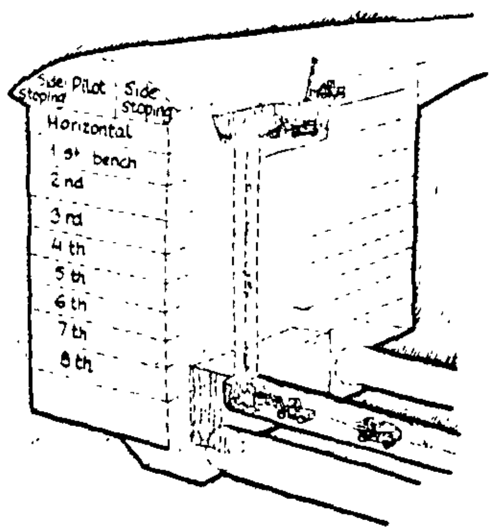


Fig. 7.36 Drifting stages for machine hall in hydro-electric power plant.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

CAUTIOUS BLASTING

CONFERENCISTA
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10. CAUTIOUS BLASTING

Fig. 10.1 Cautious blasting.

10.1 General.

The development of blasting techniques has made it possible to carry out advanced blasting operations close to and under existing structures.

In the last decades, blasting activities in populated areas have increased due to the need for better communications like metros, tunnels for telephone cables as well as tunnels for water supply, sewerage, electric cables etc.

Another area of cautious blasting is the expansion of existing hydroelectric and nuclear power plants, where it is of the utmost importance that power production is not disturbed during the construction period.

The increased prices of land in urban areas have also made it feasible to utilize the "below street level" space for various purposes such as garages, offices, bomb shelters etc.
In all these blasting operations, ground vibrations and to a certain extent air shock waves and flyrock constitute a threat to property and life and it is therefore necessary to control these hazards to avoid damage.
It is primarily the ground vibrations which affect neighboring structures but special attention has to be given to the possible occurrence of flyrock, which is the main cause of on-site fatalities and damage to equipment.

10.2 Ground vibrations.

10.2.1 The theory of ground vibrations.

Ground vibrations are seismic movements in the ground caused by rock blasting, piling, traffic, excavation, vibration compaction etc.
Ground vibrations, which are a form of energy transport through the ground, may damage adjacent structures when they reach a certain level.
Some of the energy released from a blast propagates in all directions from the hole as seismic waves with different frequencies. The energy from these seismic waves is damped by distance and the waves with the highest frequency are damped fastest. This means that the dominant frequencies from the blast are high at short distances and lower at longer distances.

The size of the ground vibrations depends on:

* quantity of co-operating charges
* constriction
* characteristics of the rock
* distance from the blasting site
* geology of covering earth deposits

By selecting the right blasting method and correct drilling and firing patterns the size of the ground vibrations can be controlled.

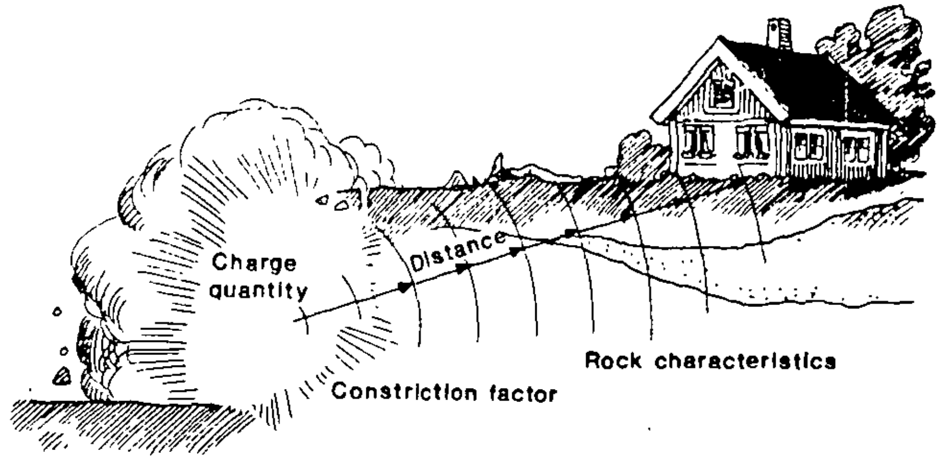


Fig. 10.2

Ground vibrations are a complicated type of seismic waves and consist of different kinds of waves:

* **P-wave.** The P-wave is also called the primary or compressional wave. It is the fastest wave through the ground. The particles in the wave move in the same direction as the propagation of the wave. the density of the material will change when the wave passes.

* **S-wave.** The S-wave is also called the secondary or shear wave. It moves through the medium at right angle to the wave propagation but slower than the P-wave. The S-wave changes the shape of the material but not the density.

Abbreviations:
SH = Shear wave, horizontal
SV = Shear wave, vertical
R = Rayleigh wave
P = Compressional wave

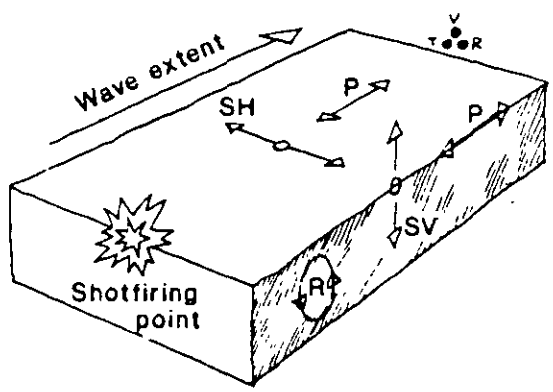


Fig. 10.3 Seismic waves.

The common denomination for P-waves and S-waves is body waves.

* **R-wave.** The R-wave (Rayleigh wave) is a surface wave which fades fast with depth. It propagates more slowly than the P and S waves and the particles move elliptically in the vertical plane and in the same direction as the propagation. At the surface the movement is retrograde to the movement of the wave.

The measuring of the ground vibrations is usually done at one or several points at ground level. For a total analysis, the practice is to measure in three directions: vertical, longitudinal and transverse. Normally the vertical component is dominant at shorter distances. It is therefore, usually sufficient to measure in the vertical direction. For vibration analysis of the measured values, the vibration phenomenon may be recorded as a function of time – time history. Then the displacement, particle velocity and acceleration can be recorded.
The basic rule is that the vibration velocity is measured on structures (buildings etc.) with a geophone and the acceleration on installations (computers etc.) with an accelerometer.
If the vibration velocity is measured, the acceleration can be calculated and vice versa. Which of these parameters that is the most interesting depends on the damage criterion for the structure to be protected. If this is known, it is normally sufficient to measure the peak value of the desired parameter.

10.2.2 Damage criteria and recommendations.

Experience over many years of measuring has shown that the particle velocity of the ground vibrations affecting a foundation constitutes the best parameter for the risk criterion for damage. As ground vibrations is approximately a sine formed vibration, the particle velocity can be calculated in accordance with the following formula:

$$v = 2\pi f A$$

where v = particle velocity (mm/sec)
f = frequency (periods/sec)
A = displacement in mm

From the above formula, the acceleration of the vibration can be calculated:

$$a = 4\pi^2 f^2 A$$

where a = acceleration in g (9.81 m/sec^2)
A = displacement in mm

Control of the particle velocity is important, as it has been shown to be directly proportional to the stress to which the building material is exposed.
The relationship between particle velocity and stress in an ideal case, when a plane shock wave passes through an infinite elastic medium can be expressed as follows:

$$\gamma = \frac{v}{c}$$

where γ = shearing angle (mm/m)
v = particle velocity (mm/sec)
c = propagation velocity (m/sec)

To recommend realistic permitted levels of ground vibrations for buildings, engineers with extensive experience of rock blasting and vibration measurement evaluation should be consulted. Any restriction in the form of reduced vibration levels will increase the cost of drilling and blasting considerably.
For that reason it is important to start all blasting operations in populated areas with an inspection of surrounding buildings. This will be followed by a risk analysis in order to assess the sensitivity of the buildings and foundations to ground vibrations.

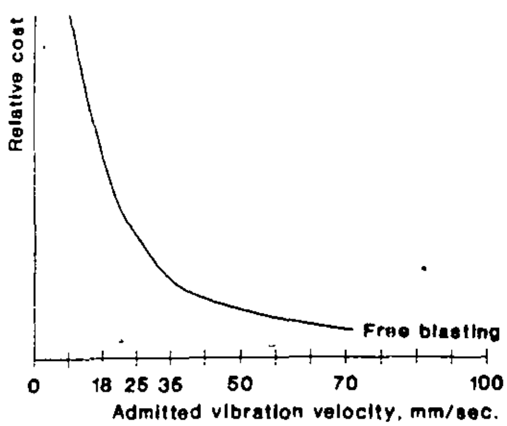


Fig. 10.4 The effect on cost of different levels of vibration velocity.

The most important parameters are:

- Vibration resistance of the building materials.
- The general condition of the building.
- Duration and character of the ground vibration.
- Presence of equipment sensitive to ground vibrations within the building.
- How the foundation is constructed.
- The quality of the foundation.
- The velocity of the wave propagation in rock, soil and construction material.

The following table shows the values that are normally permitted and which are used to evaluate the potential damage risk through ground vibration to standard residential housing.

Although the vibration velocity is stated as the permitted value it is the shearing angle which determines the dimensions. The accuracy of the values in the table has been confirmed by hundreds of thousands of readings over more than 40 years.

Vibration velocities normally recommended in appraising ground vibration damage risk to residential buildings with respect to the foundation of the building.

Wave velocity c m/sec	1000–1500 Sand, gravel clay and ground water	2000–3000 Moraine slate, soft limestone	4500–6000 Granite, gneiss, hard limestone, diabase quartzite, sandstone	Result in typical housing structures	Level at c=4500 to 6000 m/sec.
Vibration velocity v mm/sec	9	18	35	No	0.008
	13	25	50	visible	0.015
	18	35	70	cracking	0.03
	30	55	100	Fine cracks, falling plaster	0.06
	40	80	150	Noticeable cracking	0.12
	60	115	225	Severe cracking	0.25

In the case of older buildings of poorer quality, it is customary to decrease the permissible vibration velocity from 70 mm/sec. to 50 mm/sec., in buildings of light concrete it should be decreased to 35 mm/sec. Conversely, there have been occasions where velocity values of more than 100 mm/sec. were attained without damage to buildings. In the case of individual blasting operations, sturdy concrete structures can stand values exceeding 150 mm/sec.

If the limit values in the table above regarding "no visible cracking" are transferred into a three-party graph, the curve will look like in Fig. 10.5 curve 3. However, in the curve the limit value for buildings founded on rock has been reduced from 70 mm/sec. to 50 mm/sec. Curve 3 can be said to represent the recommended limit values for normal residential areas. For frequencies exceeding 40 Hz the particle velocity is the criterion for damage but at lower frequencies the displacement represents the criterion.

The dominant frequencies for vibrations passing through soft kinds of rock, moraine, sand, gravel, clay etc. are lower than for example granite. This is shown in the above table and curve 3 reflects this for lower frequencies where displacement is used as the criterion. Curve 2 in Fig. 10.5 represents values at which buildings receive direct damage. (Langefors and Kihlström, 1967.)

It must be pointed out that curve 3 only indicates the recommended limiting value and expert judgment is needed to determine more accurately which upper limit values should be set for structures adjacent to blasting operations.

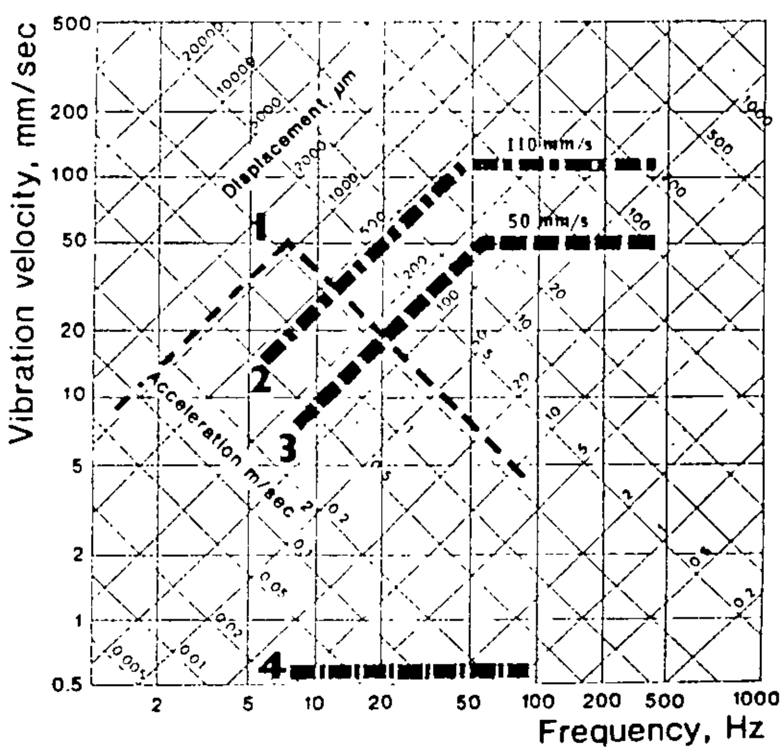


Fig. 10.5

Criteria for damage and recommendations:

Curve 1: Recommended upper limit for IBM computers with a duration of vibration less than 5 sec.
Curve 2: Direct damage from vibrations to buildings during blasting.
Curve 3: Recommended upper limit for blasting.
Curve 4: Vibrations disturbing to human beings.

In connection with blasting operations close to telephone and relay stations or buildings containing other sensitive equipment such as computers, electron microscopes, turbines etc. consideration must be given to the acceleration in order to avoid disturbances.
The recommended permissible values for ground vibration close to this type of equipment are:

* Telephone – relay stations
  v=50 mm/sec. and a=0.1−3.0 g depending on type of station.
* TV-stations
  v=35 mm/sec. and a=3.0 g.
* Computers
  a=0.25 g. (For certain parts of the computer.) .

Blasting close to computer installations (not micro computers or PC:s), where the manufacturer prescribes a maximum acceleration of 0.25 g, is difficult and under certain circumstances impossible, if special arrangements are not made. Nitro Consult AB, a subsidiary of Dyno Industries, Norway, has therefore developed a special method to dampen these installations, thus reducing the vibrations coming into the equipment. Dampening should always be followed up with vibration measurement.

The size of the ground vibrations depends on:

* number of co-operating charges
* the constriction of the blast
* the characteristics of the rock
* the distance from the blasting site
* the geology of the surrounding ground

For the planning of blasting operations where ground vibration problems occur, it is important to be aware of the relationship between distance, charge and ground vibration

Using Langefors' formula for determining the charge level the vibration velocity can be calculated:

$$\text{Charge level} = \frac{Q}{R^{3/2}}$$

where Q indicates the charge in one hole in kg or several instantaneously fired charges at the same distance R in meters.

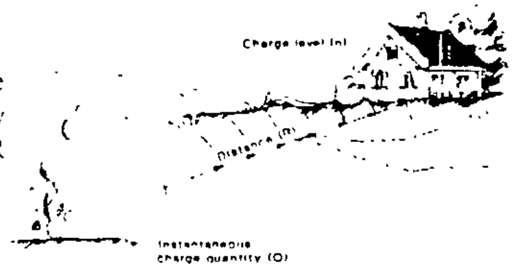


Fig. 10.6.

Vibration velocity:

$$v = K\sqrt{\frac{Q}{R^{3/2}}}$$

where Q = instantaneously detonating charge (kg)
R = distance (m)
v = vibration (particle) velocity (mm/sec)
K = transmission factor, constant depending on the homogeneity of the rock and the presence of faults and cracks. For hard Swedish granite it is approx. 400 but it is normally lower.

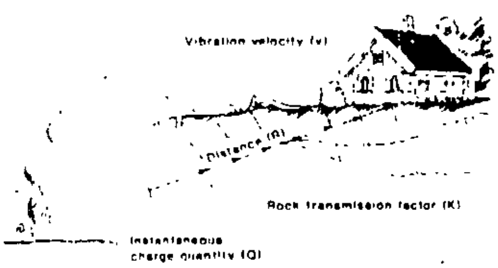


Fig. 10.7.

The relationship between charge/distance and ground vibration can be used to make a simple table which may serve as ready-reckoner for the planning of blasting operations.

Distance	Charge in kg (instantaneous detonation)						
m	Level: 0.008	0.015	0.03	0.06	0.12	0.25	0.50
0.5				0.02	0.04	0.08	0.16
1	0.008	0.015	0.03	0.06	0.12	0.25	0.50
2	0.023	0.04	0.08	0.17	0.34	0.68	1.35
3	0.04	0.08	0.16	0.32	0.65	1.30	2.60
4	0.06	0.12	0.24	0.48	1.0	2.0	4.0
5	0.09	0.17	0.35	0.70	1.4	2.8	5.6
6	0.12	0.22	0.44	0.88	1.8	3.7	7.3
7	0.15	0.28	0.56	1.1	2.2	4.6	9.2
8	0.18	0.34	0.68	1.35	2.7	5.7	11.3
9	0.22	0.4	0.8	1.6	3.2	6.7	13.5
10	0.25	0.5	1.0	2.0	4.0	8.0	16.0
12	0.3	0.6	1.2	2.5	5.0	10.4	20.8
14	0.4	0.8	1.6	3.1	6.3	13.0	26.0
16	0.5	1.0	1.9	3.8	7.7	16	32
18	0.6	1.2	2.3	4.6	9.2	19	38
20	0.7	1.3	2.7	5.4	10.7	22	44
25	1.0	1.9	3.8	7.5	15	31	62
30	1.3	2.5	4.9	9.8	20	41	82
40	2.0	3.8	7.6	15	30	63	126
50	2.8	5.3	10.6	21	42	88	176
60	3.7	7.0	14	28	56	116	232
70	4.7	8.8	18	35	70	146	292
80	5.7	10.7	21	43	86	178	358
90	6.8	12.8	25	51	102	213	427
100	8.0	15.0	30	60	120	250	500
120	10.5	19.7	39	79	158	328	657
140	13.2	24.8	50	100	200	410	820
160	16.2	30	60	120	240	500	1000
180	19.3	36	72	145	290	600	1200
200	22.6	42	85	170	340	700	1400

The charge levels in the previous table correspond to the following vibration velocities at the rock transmission factor K=400.

Level	Vibration velocity	
$Q/R^{3/2}$	mm/sec.	
0.008	35	
0.015	50	
0.03	70	
0.06	100	(threshold value granite)
0.12	140	
0.25	200	
0.50	280	

The relationship charge/distance and vibration velocity can also be expressed graphically:

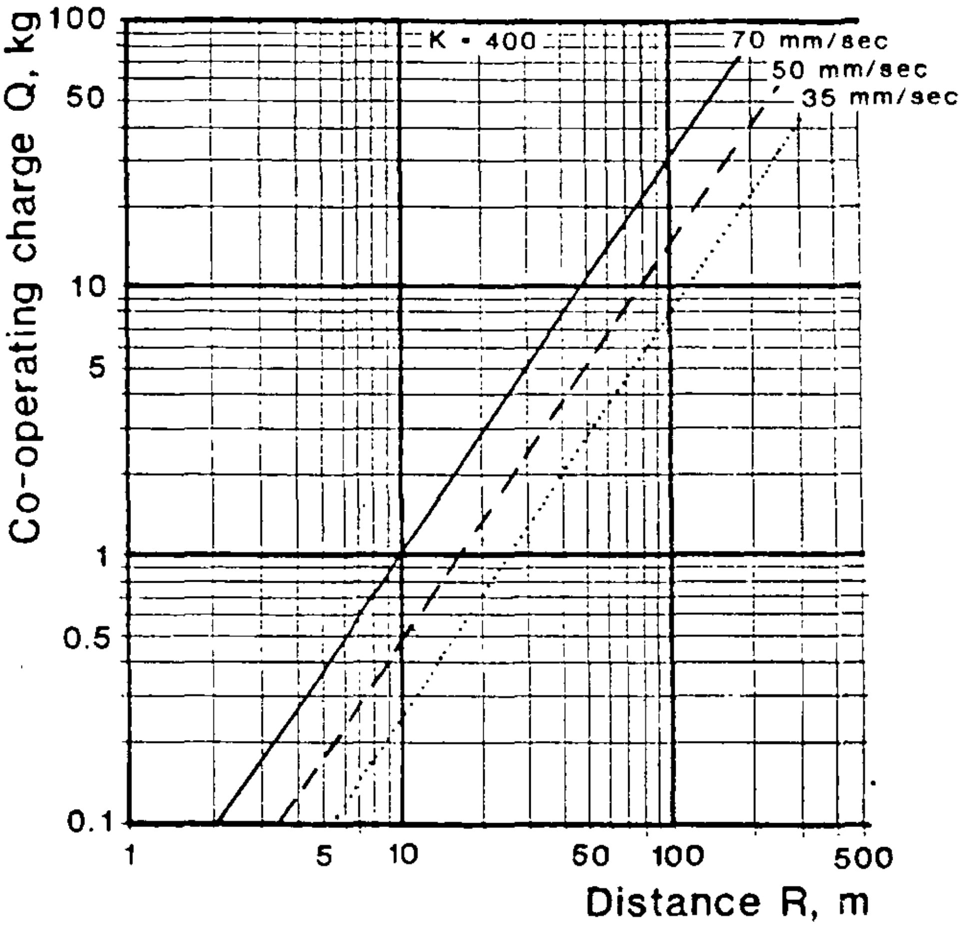


Fig. 10.8 Charge (Q) as a function of distance (R) for different levels of vibration velocity. Rock transmission factor K = 400. At a distance of 20 m, the charge must not exceed 1.3 kg to ensure a vibration velocity of less than 50 mm/sec.

The distance and charge tables which are based on the determined rock transmission factor K should be used with care close to buildings where the foundation is unknown e.g. buildings built partly on rock and partly on soil and buildings founded on wooden piles in clay etc. The value of the rock transmission factor K will also change depending on the characteristics of the ground and the distance. Looser materials such as moraine and clay have lower K values than homogeneous hard rock. The rock transmission factor K is also lower in weathered and fissured rocks.

The actual value of the factor K is best determined by test blastings at the actual site, followed up by scrupulous vibration measurement.

To evaluate the test blasts, the constriction of the blast must be considered e.g. if the test hole has free breakage, if it just cracks the rock or if it does not affect the rock at all. To evaluate a test blast correctly, experience of test blastings and knowledge of the field of ground vibrations is necessary.

When the rock transmission factor K is determined, the graph in fig. 10.8 may be adjusted accordingly and the realistic relationship between charge/distance and vibration velocity adapted to the local conditions.

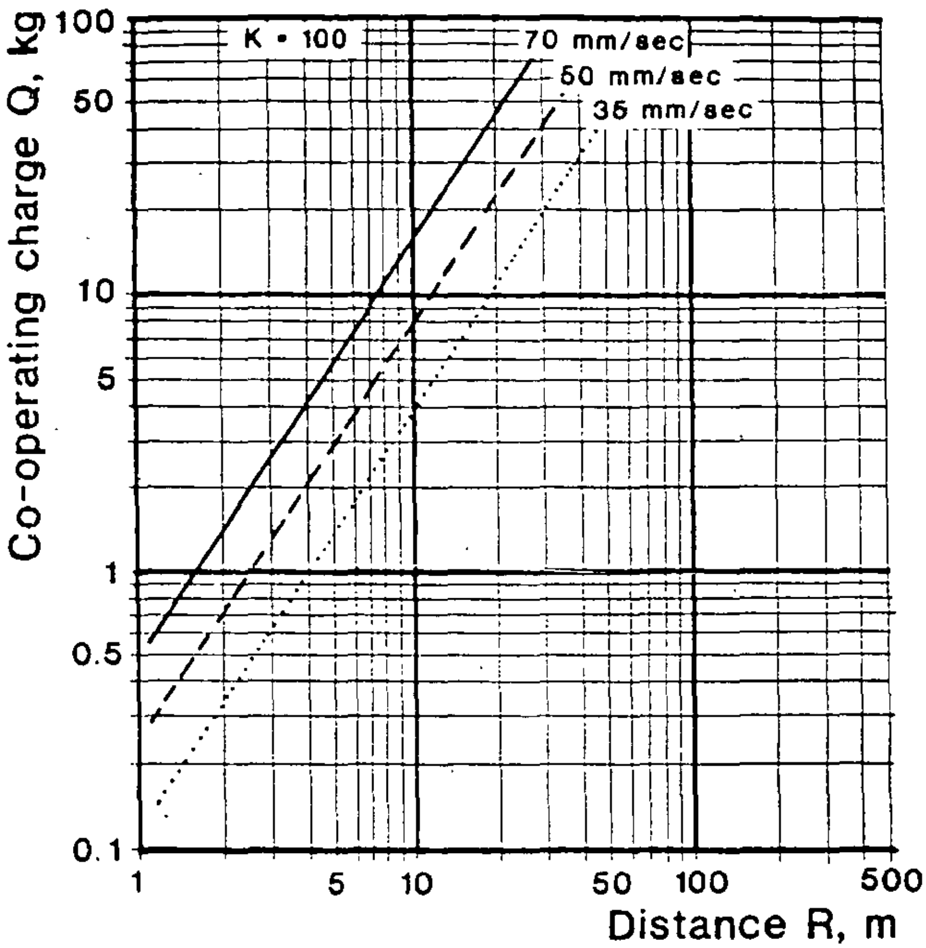


Fig. 10.9 Charge (Q) as a function of distance (R) for different levels of vibration velocity. Rock transmission factor K=100. At a distance of 20 m, the charge must not exceed 20 kg to ensure a vibration velocity of less than 50 mm/sec.

The comparison of the two graphs with rock transmission factors K=400 and K=100 respectively shows that the dampening effect is higher in the softer rock (K=100) and the vibration velocity is lower if the relationship charge/distance is maintained.

10.2.3 Geological factors influencing ground vibrations.

Soils and rocks are porous materials with a relatively rigid skeleton of particles. The pores are filled with water or air. In soil, the soil skeleton consists of mineral grains which are held together by frictional and cohesive forces In sedimentary rocks the mineral grains are cemented together and in magma rocks and metamorphous rocks the minerals have crystallized to a rockmass which usually contains water-bearing fissures and joints. In practice it may be difficult to state a precise propagation velocity of the seismic wave in different soils and rocks.

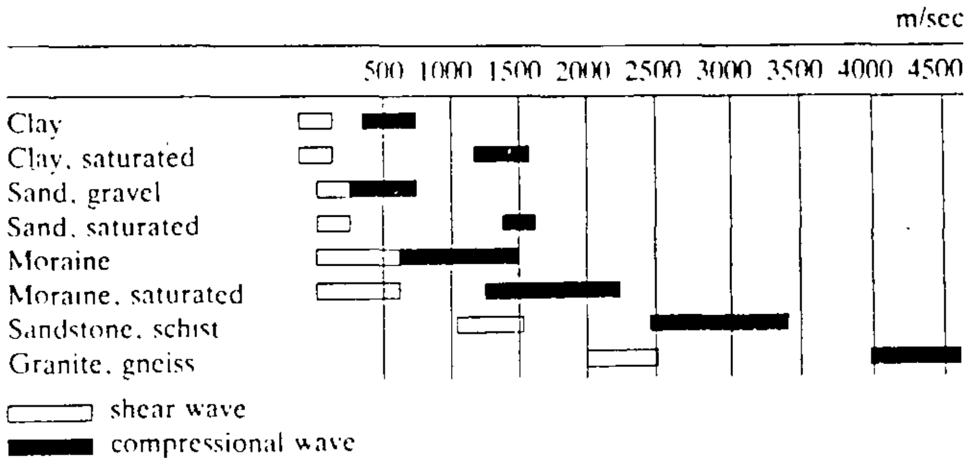


Fig. 10 10 The propagation velocity of compressional and shear waves through different soils and rocks.

The propagation velocities of the Rayleigh wave depend on the frequency and are lower than those of the shear wave

Every geological environment has its own ground vibration characteristics which affect the propagation of the vibration wave. The ground vibration characteristics depend on the following properties of the ground:

- the elastic constants of the ground (elastic and shearing moduli) which determine the propagation velocity of the waves.
- the type of soil and its depth which determine the predominant range of frequency and type of waves.
- the moistness of the soil and the ground water level.
- the topography and morphology, which may lead to focusing of seismic waves.
- the damping characteristics of the ground.

An example of geological factors influencing the rock blasting operation is the difference in permitted charges at different distances in Sweden and the U.S.A.

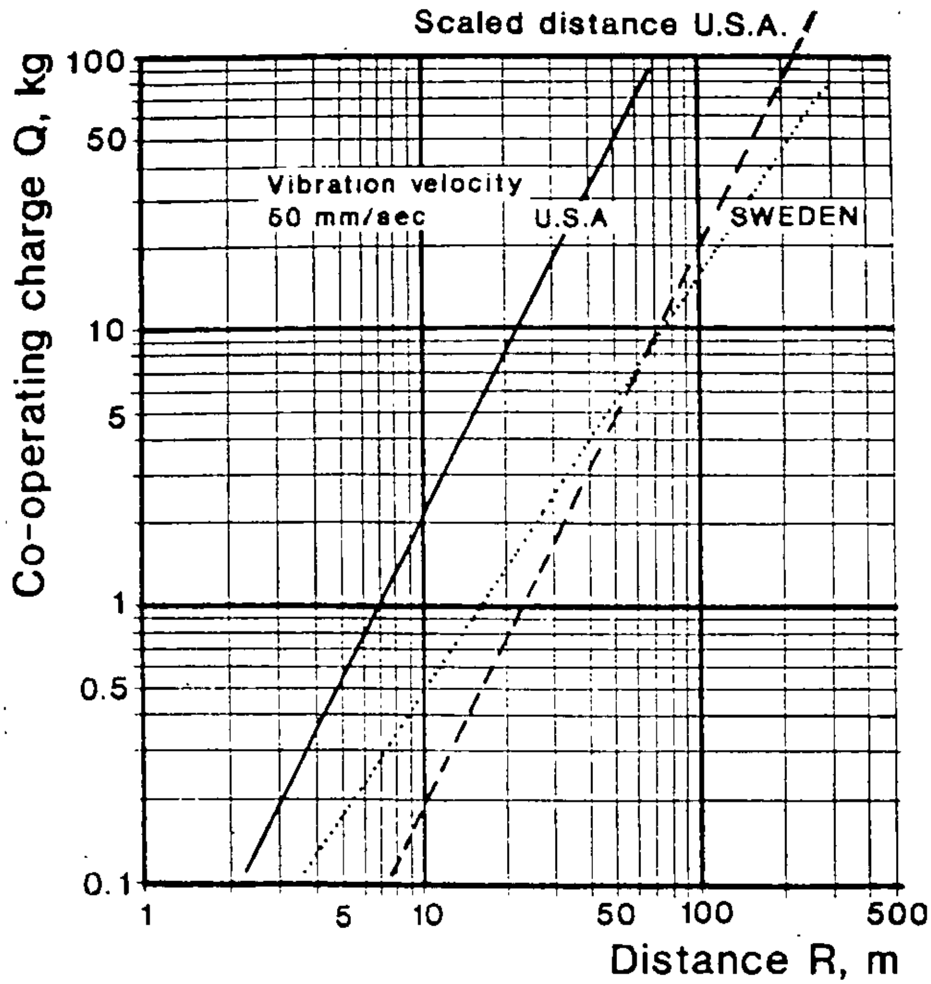


Fig. 10.11 Comparison of quantity-distance relationships for specified values of vibration velocity in Sweden and U.S.A

In U.S.A. the highest permitted vibration velocity is 50 mm/sec. and the blastings must be followed up with ground vibration measurement if the charge-distance graph is used. In the case of blasting without ground vibration measurement the regulations permit the blaster to use a scaled distance equation instead.

The scaled distance equation is as follows

$$S.D. = D/\sqrt{W}$$

where S.D. is the scaled distance

D is the distance in feet from blasting site to the structure in question.

W is the maximum charge weight in pounds per delay.

The regulations say that a scaled distance of 50 or more will protect against vibrations greater than 50 mm/sec As may be seen from Fig. 10.11, the scaled distance-equation gives rather conservative values.

Fig. 10.11 also shows that the maximum permitted charge in the U.S.A should be 19 kg at a distance of 30 m compared to 2.6 kg in Sweden. At a distance of 100 m the charge should be 195 kg in the U.S.A and 15.6 kg in Sweden. The US values are considerably higher than the Swedish, but on the other hand a vibration velocity of 70 mm/s is generally permitted in Sweden, which gives charge weights twice those permitted at 50 mm/sec.

The Swedish values are still considerably lower than those of the U.S.A. The reason for that is that the rocks in U.S.A are different from those in Sweden. They are generally softer and the propagation velocity of the vibration waves is lower. Furthermore, and equally important, the vibrations are damped faster and the vibration velocity is thus lowered.

10.2.4 Planning of blasting operations.

At the planning stage of the blasting operation, attention must be paid to the geological characteristics of the rock. If there are zones of weathered and fissured rock between the blasting site and objects sensitive to vibrations with a damping effect on the ground vibrations, the geological characteristics of the rock may change to more homogeneous rock as the work proceeds, increasing the ground vibrations. It may than be necessary to decrease the charge to avoid damage.

Therefore the test blastings should be measured to make a seismic profile where the seismic waves are measured at various points giving information on how the characteristics of the rock varies.

When planning and executing the blasting operation, it is important that the constriction of the round is minimized by correct drilling and firing patterns.

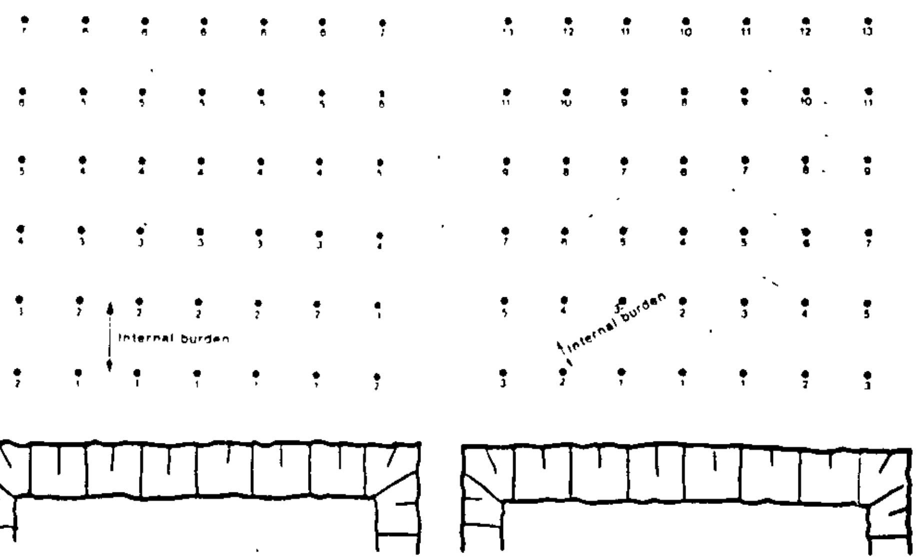


Fig. 10.12 By changing the firing pattern, the internal burden may be minimized and the constriction lowered.

The vibration velocity also depends on the inclination of the hole. Steeper hole inclinations or other conditions increasing the constriction of the blast (misfires etc.) may cause considerable increase of the vibration velocity.
The ground vibrations will also increase if the blast fails to break the rock down to the intended level.

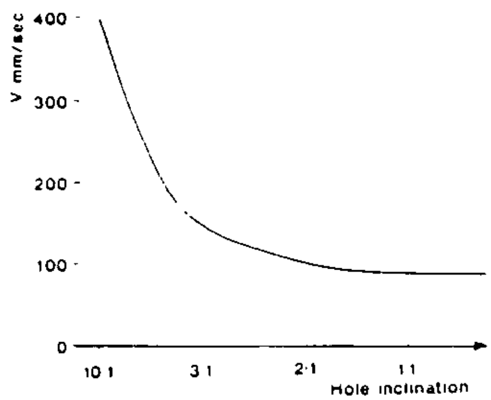


Fig. 10.13 Vibration velocity in relation to hole inclination with the same burden and explosive charge (trench blasting).

The first rounds blasted at a work site must be considered as test blastings and the vibration measurements should be used as a guidance for the planning of an optimum blasting operation. The results from the vibration measurements should be utilized during all blasting operations to find the most economic drilling and firing pattern. However, a certain margin to the permitted vibration velocity should always be maintained as the ground vibrations may increase sharply if the blast does not go according to plan. This can be difficult in cases when the drilling is far ahead of the blasting operation, but using the result of the initial risk analysis and a thorough follow-up during the blasting operation, the drilling pattern may be selected in such a way that several charges may be used in each hole if the vibration velocity values become too high.

Investigations show that people in general react to vibration values far below the limit for damage on buildings. It has also been demonstrated that blasting operations which are executed in a short time are better accepted by people in the area than operations lasting for a long time, even if there are long gaps between the blasts.
The best way to forestall complaints is if those responsible for the blasting operations give comprehensive information to the people affected.

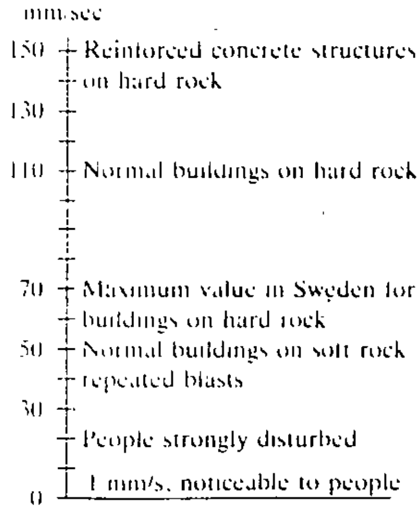


Fig. 10.14 Maximum permitted vibration velocities for residential buildings.

The electronic instruments available are:
- peak particle velocity instruments
- vibration time-history recorders
- combined instruments (ground vibration and air pressure)

ULTRALETTE is a multi-channel system for time-history recording which is used all over the world. The instrument is not automatic and thus has to be handled by personnel trained for the purpose. It consists of a light-weight, small size recorder with plug-in signal conditioning modules and external velocity transducers for vertical and horizontal measurements. The recorder can register peak vibration velocity, acceleration, frequency and amplitude.
The ULTRALETTE is also suitable for the measurement of the propagation velocity of the seismic wave, and evaluation of test blastings.

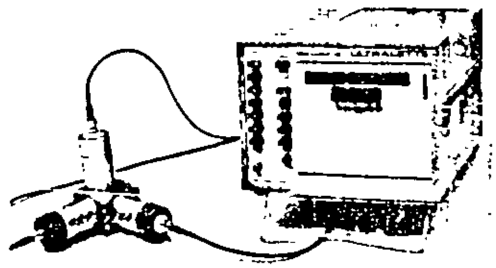


Fig. 10.16 Ultralette.

Combined instruments for the registration of ground vibrations and sound pressure are LOG 1 and UVS 1404. These two are computerized instruments for easy and accurate use.

The **LOG 1** is a portable seismograph for the monitoring and recording of seismic and sound signals from blastings and similar operations. Its design uses modern microcomputer technology, resulting in maximum accuracy and versatility. The LOG 1 can be used for on-the-spot measurements or for unattended operations in industries such as construction, mining, quarrying and transportation.
The LOG 1 records peak values in three directions and time-histories of seismic vibrations and sound pressure. A full alphanumeric keyboard is built in allowing the operator to insert any message into the computer memory for subsequent printout with the recorded data.

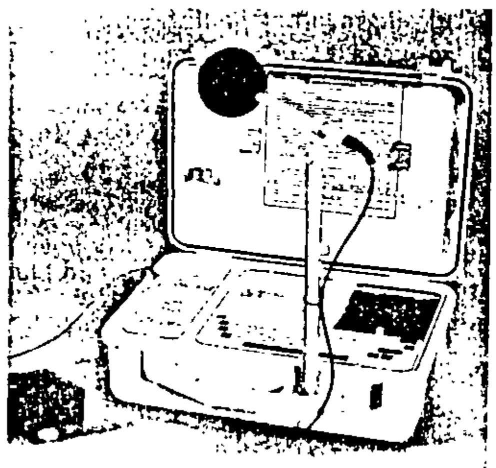


Fig. 10.17 LOG 1.

More often than not it is a good investment to employ a consultant at the beginning of a blasting operation. The consultant will take care of the primary risk analysis and before blasting starts point out the problems which are likely to occur. Using the knowledge gained from the risk analysis the blasting operation can be better planned both technically and economically. It is normally cheaper to prevent problems than take measures when they arise.

10.2.5 Instruments to measure ground vibrations.

Different types of instruments have been developed for ground vibration measurements.

The early instruments were mechanical. They were fixed to the object which was subjected to ground vibrations. The principle was that the instrument contained a heavy weight which was suspended in a spring, acting as an inert mass. During the vibration the instrument moved but the weight did not. The movement of the instrument was recorded on paper and the level of ground vibration could be evaluated. The mechanical instruments have nowadays been replaced by electronic ones.

In the electronic instrument, the mechanical vibration is sensed and converted to an electric signal by an electro dynamic transducer called a geophone. This transducer gives an electric signal which is direct proportional to the particle velocity of the vibration, which is the parameter being recorded.

The geophone consists of a spring (1) loaded – moving mass (2) system. A coil (3) is wound around the moving mass. The system moves freely in a magnetic field created by a permanent magnet (4). When the coil moves in the magnetic field an electric current is induced with a magnitude proportional to the velocity of the coil.

In the geophone in use, the coil is steady while the magnet mounted in the outer case moves in relation to the received mechanical vibration.

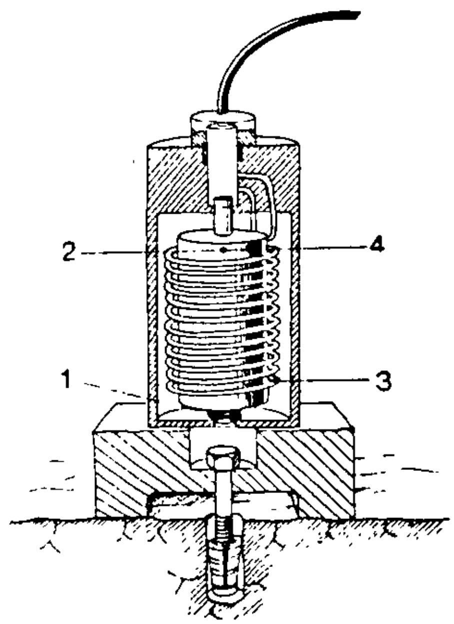


Fig. 10.15 Basic design of geophone.

There are instruments available for the measurement of vertical and horizontal components at one or several measuring points. As regards blasting it is the magnitude of the vertical component that is important. The instruments may also be supplemented with an accelerometer to measure the acceleration.

The printout is done on a dot matrix printer and can be used for further analysis if required.

The **UVS 1404** is a fully electronic instrument for continuous monitoring of vibrations and shock waves from mining, construction, traffic etc.
The UVS 1404 is a completely new concept for ground vibration monitoring. This fully electronic instrument has been referred to as the third generation of vibration monitors, following the mechanical and electromechanical generations.
Low temperature and high humidity have always been a problematic combination for electro-mechanical instruments, especially for the built-in printers. The UVS 1404 has no moving parts, which means that the main source of operational and maintenance problems is eliminated. Wide temperature range components (−20°C to +80°C) and a heavy-duty, waterproof aluminium case with built-in humidity absorbers contribute to make the UVS 1404 insensitive to harsh environments and rough handling, as well as normal wear and tear.
The four channel UVS 1404 measures and records peak values of particle velocity, acceleration, air shock waves etc., depending on the type of sensor used for each channel
The recorded data is presented on a dual LCD (Liquid Crystal Display) system. A continuous graphical diagram along a horizontal time-axis is given on the primary LCD, and alphanumeric information (date, time, graphical scale, battery condition, max. recorded value, channel number etc.) on the secondary LCD. The UVS 1404 has a complete 31 days' memory, which is continuously updated with the latest information. It is powered by 2 standard type batteries allowing unattended monitoring for one month.
The values recorded in each channel are stored in the memory in 2-minute periods. No trigger levels have to be set in advance, everything including zero readings is recorded. This eliminates the risk of the memory being cluttered with undesired information because of too low trigger levels or missed low readings because of too high levels.
In addition to the normal LCD presentation, a corresponding paper copy can be obtained at any time. By connecting a Hewlett Packard THINKJET printer to the digital outlet, any sequence of information within the total memory capacity can be printed on A4-sheets for detailed study and long-term documentation.
If large volumes of measurement records have to be handled and stored, an op-

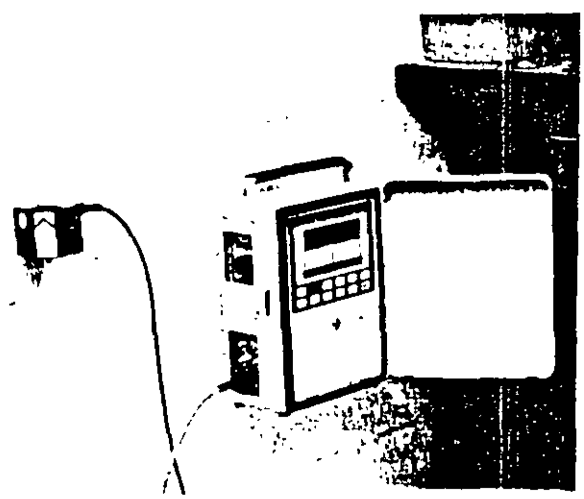


Fig. 10.18 UVS 1404.

tional system for automatic data collection and report printing is available. The memory is tapped from the UVS 1404 (on site or elsewhere) into a Hewlett Packard portable disk drive. One single disk can store data from four channels covering five complete 31-days' periods. Processed by a Hewlett Packard HP71C the data is automatically printed by the THINKJET in a table format report. Furthermore, a buffered analog outlet gives direct access to the input signals, thus making it possible to transfer full time-history records to a separate tape recorder or equivalent.
Most common geophones for velocity measuring as well as a variety of sensors such as accelerometers and air shock wave microphones can be directly connected to the UVS 1404. The instrument presents each reading directly in the pre-selected unit (g, MPa, dB etc.).

10.3 Charge calculations.

10.3.1 General.

When blasting close to buildings and other installations sensitive to vibrations it is not always possible to utilize the blastholes in the same way as in normal blasting operations. The ground vibrations which always occur in blasting operations depend on the maximum co-operating charge weight. Thus, the charge weight for each delay must be kept within certain limits for different distances. However, for big blasts and long distances the total amount of explosives may be a determining factor for the size of the vibrations.
The constriction of the blast is another factor which affects the size of the ground vibrations. A constricted charge gives higher vibrations than one with free breakage.

The maximum co-operating charge can be determined from the charge/distance graph. Knowing the distance to the sensitive object it is easy to determine the correct charge weight.
The charge weight depends on the permitted vibration velocity and the rock transmission factor

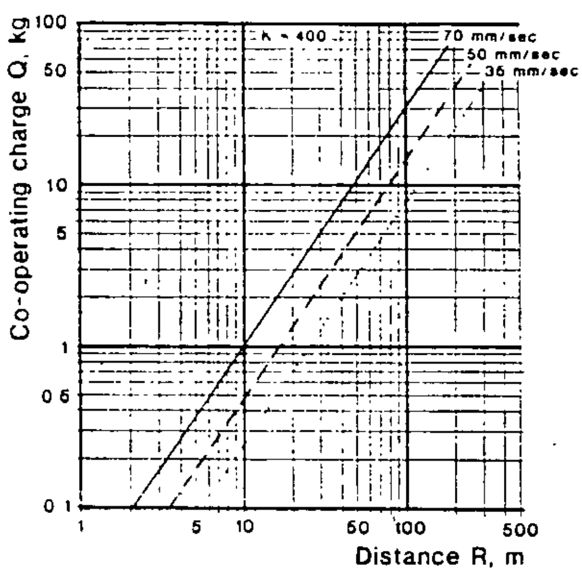


Fig. 10.19 Charge/distance graph for different vibration velocities

The maximum co-operating charge can be reduced in the following way:

- The firing pattern.
  The number of holes with the same period number is reduced so the maximum co-operating charge is not exceeded.
- Reduced drilling pattern.
  The blasthole volume is not utilized to the maximum for the explosives charge as in normal blasting. The drilling pattern is more closely spaced with less explosives in each hole.
- Divided charges.
  The requisite charge amount for the hole is divided into several partial charges fired with different delays. The charges are separated by sand stemming.
- Divided benches.
  The bench is not blasted to its full depth in one go but divided into several lower benches.

10.3.2 The firing pattern.

In cautious blasting it may be necessary to decrease the co-operating charge. This can be done by decreasing the number of detonators with the same period number.

Detonators with the same period number always have a certain scatter. In other words, the delay time of the delay element is not exactly the same for detonators with the same period number. This means that only some of the detonators within the period will co-operate.

The co-operation within the period or between various periods depends on the frequency of the ground vibrations. For hard homogeneous bedrock the frequency is normally over 60 Hz and here the following practical rule for co-operation will apply:

Detonator type	Period number	Co-operation within period (reduction factor)
VA/MS	1–10	1/2
Nonel GT	3–10	1/2
VA/MS	11–20	1/3
Nonel GT	11–20	1/3
Nonel GT/T	1–20	1/4
	25–60	1/6
VA/HS	1–12	1/6

The table is based on the scatter within the period which is lowest for MS detonators but may be as high as 200 ms for HS detonators.

According to Langefors, the risk of co-operation is greater at low frequencies.

Less than 60 Hz:		
VA/MS	1–10	1
VA/MS	11–20	1/2
VA/HS	1–12	1/6
Less than 20 Hz:		
VA/MS	1–20	1
VA/HS	1–12	1/3

The low frequencies occur in soft rocks and when blasting at relatively great distances.
At the lowest frequencies it may theoretically be co-operation between different period numbers.
In the U.S A., with its softer rocks, the charges are supposed to co-operate if the delay between them is shorter than 9 ms.

The following example shows the effect of the reduction factor in cautious blasting:

Conditions:

- Rock, granite
- Bench blasting
- Blasthole diameter, drill series 11 (34–29 mm)
- Bench height 4 0 m
- Charge per hole 1.95 kg
- Maximum permitted co-operating charge 5.0 kg

A. Blasting without considering the reduction factor.

2×1.95 kg = 3 9 kg

Conclusion: Maximum 2 blastholes may co-operate, which in this case implies 2 detonators per period number.

B. Blasting considering the reduction factor.

- MS detonators with period numbers 1 to 10 have a reduction factor of 1/2, which means that only half the detonators within the same period are likely to co-operate.

  If 4 detonators with the same period number are used only 2 will co-operate. The maximum charge which will detonate instantaneously is then $(4\times1.95)\times1/2=3.9$ kg. If for example 5 detonators are used in the same period there is a risk of over-charging, as 3 of the 5 detonators are likely to co-operate (3×1.95 kg=5.85 kg).

  Conclusion: 4 detonators MS 1 to 10 may be used in the same period without risk of excessive charge.

- MS detonators with period numbers 11 to 20 have a reduction factor of 1/3, meaning that one third of the detonators within the period are likely to co-operate.

If 6 detonators are used with the same period number only 2 will co-operate. The maximum charge which will detonate instantaneously is $(6\times1.95)\times 1/3=3.9$ kg.

Conclusion: 6 detonators MS 11 to 20 may be used in the same period without risk of excessive charge.

If a firing pattern starting with 4 pcs MS No. 1 is changed to one starting with 6 pcs MS No. 11, it will not increase the co-operating charge.
In the case of different amounts of explosives in the blastholes, the least favorable case has to be reckoned on, that is, the holes with the biggest amounts of explosives will co-operate.
When the number of detonators within each period is limited because of restricted ground vibrations, it may be a problem to obtain enough periods for the blast. In cases like this it is practical to use NONEL UNIDET with its unlimited number of delays (See Chapter 3b.2.4 NONEL).
In blasting operations very close to objects which are sensitive to ground vibrations, where vibrations over the permitted limit may result in severe damage, no reduction factor should be used and only the real number of detonators per period taken into account.

10.3.3 Bench blasting with reduced drilling pattern.

When the maximum permitted co-operating charge is smaller than the requisite charge for the blasthole, it is no longer possible to reduce the co-operating charge with the firing pattern. One possibility to reduce the charge is to reduce the blasthole diameter, which gives less explosives in each blasthole. Then normal drilling and charging tables may be used for the actual blasthole diameter.
However, it is often practically impossible to change the drilling equipment. In these cases a reduced drilling pattern is used, drilled with the existing equipment, where the permitted charge determines the drilling pattern.
The reduced drilling pattern increases the specific drilling which naturally increases the cost. To what degree the specific drilling may increase from an economic point of view must be decided upon in each case.
One basis for forming a judgment is to compare with the specific drilling when the blasthole is fully utilized:

Hole diameter mm	Specific drilling m/cu.m.
Drill series 11	
34–27	0.8–1.3
Drill series 12	
40–29	0.6–0.9
51	0.3
64	0.2
76	0.15

To calculate the reduced drilling pattern, the permitted co-operating charge must be known. The permitted co-operating charge may be found in the charge/distance table or in the graph 10.8. Knowing the vibration velocity which is permitted for the object in question and the distance to the blasting site, the actual permitted co-operating charge will be found.

The correct permitted vibration velocity is found in the table on page 204, which shows the vibration velocity that is normally permissible for residential buildings for the kind of material on which the buildings are built.

The basis for the calculations is that the specific charge should be 0.40 kg/cu.m. This value is the normal value and changes may be needed due to the blastability of the rock. The change of specific charge does not change the calculation procedure.
By carrying out test blasts followed by analysis of the vibration measurement results, it is often possible to use more explosives than indicated in the graph thus lowering the blasting cost.

Charge calculation procedure.

Specific charge	q (kg/cu.m.)
Permitted co-operating charge	Q_{per} (kg)

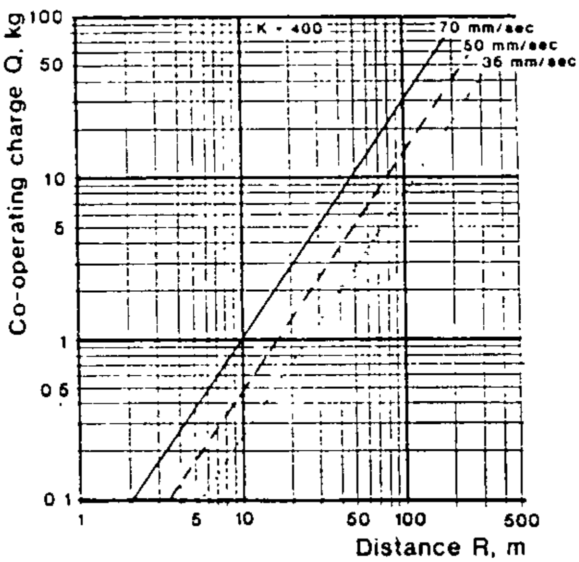
Drilling pattern.

1. The volume of rock which is blasted by each hole:

$$\text{Volume} = \frac{Q_{per}}{q} \quad \text{(cu m.)}$$

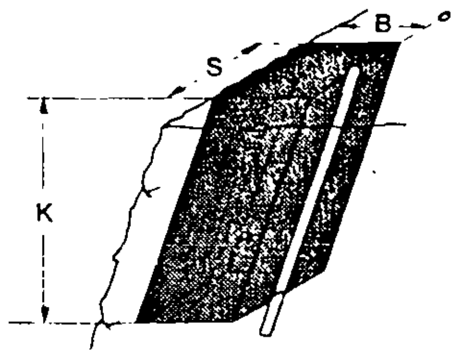
2. When the volume is known, the drilling pattern can be calculated.
The surface area each blasthole can cover:

$$\text{Area} = \frac{\text{volume}}{K} \quad \text{(sq.m.)}$$



3. Practical drilling pattern. When the surface area for the blasthole is known the practical burden is:

$$B = \sqrt{\frac{\text{Area}}{1.25}} \qquad (m)$$



Practical spacing:

$$S = 1.25 \times B \qquad (m)$$

The practical spacing should be adjusted to the width of the bench if necessary.

4. Hole depth
The hole depth H may be estimated from tables in Chapter 5.2 Charge calculations.

5. Specific drilling

$$b = \frac{H_{est}}{B \times S \times K} \qquad (m/cu.m.)$$

An estimate should be carried out to see if the specific drilling is acceptable. If not, the use of divided charges in the holes or several lower benches should be considered

If the specific drilling is acceptable, the calculations continue as follows:

6. Drilling error.

$$E = \frac{d}{1000} + 0.03K \qquad (m)$$

7. Subdrilling

$$U = 0.3(B+E) \qquad (m)$$

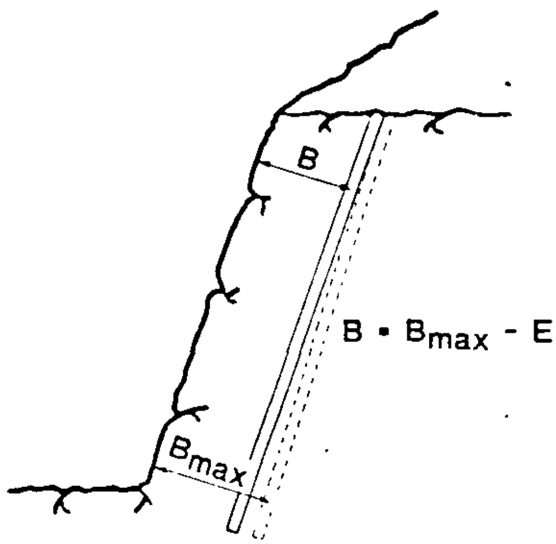
8. Hole depth

$$H = a(K+U) \qquad (m)$$

a = 1.05 for inclination 3:1 and 1.0 for vertical holes.

9. Maximum burden

$$B_{max} = B + E \qquad (m)$$



Bottom charge.

10. Charge concentration

$$l_b = \frac{B_{max}^2}{2} \text{ for Dynamex M (kg/m)}$$

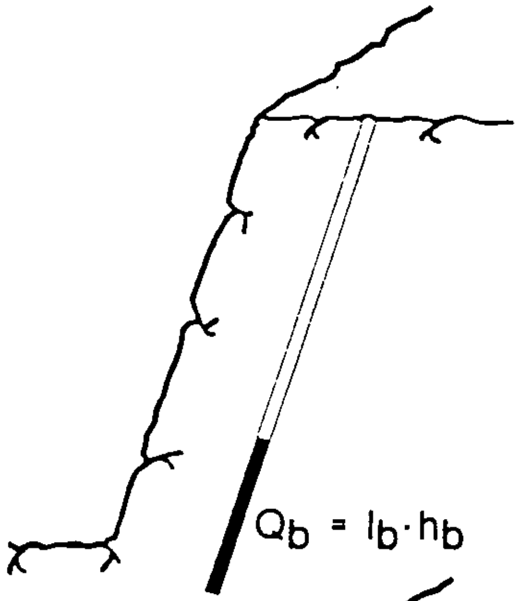
$$l_b = \frac{B_{max}^2}{2} \times \frac{1.15}{1.25} \text{ for Emulite 150}$$

11. Height

$h_b = 1.3 \times B_{max}$ (m)

12. Weight

$Q_b = l_b \times h_b$ (kg)



Column charge.

13 Weight

$Q_c = Q_{per} - Q_b$ (kg)

14. Stemming

$h_o = B$ (m)

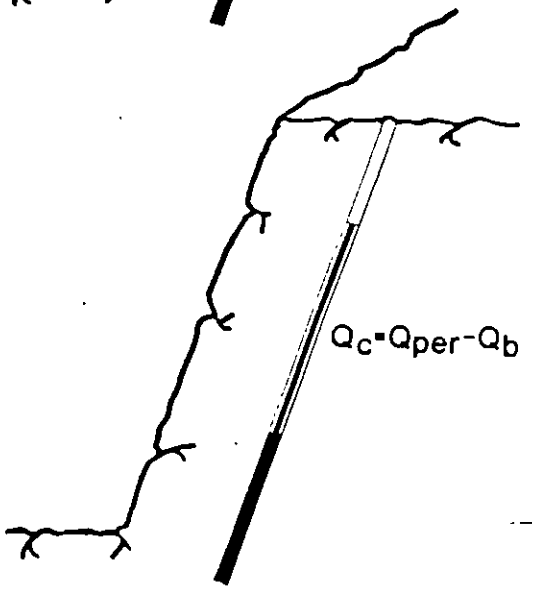
15. Height

$h_c = H - h_o - h_b$ (m)

16. Charge concentration

$$l_c = \frac{Q_c}{h_c} \quad \text{(kg/m)}$$

l_c should be at least 40 % of l_b. Explosives of suitable dimensions should be selected for the charge.



17. Total charge weight

$Q_{tot} = Q_b + Q_c$ (kg)

Check that $Q_{tot} \leq Q_{per}$. If this is not the case, the column charge should be reduced. If that is not possible the drilling pattern must be reduced further.

18. Specific drilling

$$b = \frac{\text{number of drilled meters per row}}{\text{volume per row}} = \frac{n \times H}{w \times B \times K} \quad \text{(m/cu.m.)}$$

19. Specific charge

$$q = \frac{\text{Total charge per row}}{\text{Volume per row}} = \frac{n \times Q_{tot}}{w \times B \times K} \quad \text{(kg/cu.m.)}$$

Calculation example.

Conditions: Blasting to be carried out close to a TV transmitting station. The permitted vibration velocity is 35 mm/sec. and the distance to the blasting site is 20 m.

Blasthole diameter:	Drill series 11, 34–27 mm in this case 31 mm.
Bench height.	2.5 m
Hole inclination:	3:1
Width of the round:	12 m
Rock transmission factor K:	400
Explosive:	Dynamex M
	Gurit
Specific charge q:	0.4 kg/cu.m.
Permitted co-operating charge	0.65 kg
(in accordance with graph Fig 10.8)	

1. Rock volume per hole.

$$\text{Volume} = \frac{Q_{per}}{q} = \frac{0.65}{0.4} = 1.63 \text{ cu.m}$$

2. Surface area per hole.

$$\text{Area} = \frac{\text{Volume}}{K} = \frac{1.63}{2.5} = 0.65 \text{ sq.m}$$

3. Practical drilling pattern.

Burden:

$$B = \sqrt{\frac{\text{Area}}{1.25}} = \sqrt{\frac{0.65}{1.25}} = 0.72 \text{ m}$$

Spacing:

$S = 1.25 \times 0.72 = 0.90$ m

Adjustment of the spacing to the width of the bench.

$$\text{Number of hole spaces } \frac{12.0}{0.90} = 13.33, \text{ that is } 14.$$

$S_{adj} = 12/14 = 0.86$ m

Number of holes per row $14+1=15$.

4. Estimated hole depth

From table in Chapter 5.2 Charge calculations

$H_{est} = 3.05$ m

5. Specific drilling

$$b = \frac{H_{est}}{B \times S \times K} = \frac{3.05}{0.72 \times 0.86 \times 2.50} = 1.97 \text{ m/cu.m.}$$

The specific drilling for drill series 11 is 0.8 to 1.3 m/cu.m. when the blasthole is fully utilized. The specific drilling is somewhat high in this case, but may be accepted.

6. Drilling error.

$$E = \frac{d}{1000} + 0.03K = \frac{31}{1000} + 0.03 \times 2.5 = 0.11 \text{ m}$$

7. Subdrilling

$U = 0.3(B+E) = 0.3(0.72+0.11) = 0.25$ m

8. Hole depth

$H = a(K+U) = 1.05(2.5+0.25) = 2.89$ approx. 2.90 m

9. Maximum burden

$B_{max} = B+E = 0.72+0.11 = 0.83$ m

Bottom charge.

10 Charge concentration

$$l_b = \frac{B_{max}^2}{2} = \frac{0.83^2}{2} = 0.35 \text{ kg/m}$$

11. Height

$h_b = 1.3 \times B_{max} = 1.3 \times 0.83 = 1.08$ approx. 1.10 m

12. Weight

$Q_b = l_b \times h_b = 0.35 \times 1.10 = 0.39$ kg

The bottom charge may consist of 4 cartridges of Dynamex M, 22×200 mm with a weight of 0.1 kg each = 0.4 kg.

The practical height of the bottom charge, h_b, will be 0.8 m.

Column charge

13. Weight

$Q_c = Q_{per} - Q_b = 0.65 - 0.40 = 0.25$ kg

14. Stemming

$h_o = B = 0.72$ m

15. Height

$h_c = H - h_b - h_o = 2.90 - 0.80 - 0.72 = 1.38$ m

16. Charge concentration

$$l_c = \frac{Q_c}{h_c} = \frac{0.25}{1.38} = 0.18 \text{ kg/m}$$

The concentration of the column charge should be at least 40 % of the concentration of the bottom charge, which is found to be the case.

The column may be charged with 2 cartridges of Gurit 17×500 mm with a cartridge weight of 0.115 kg each Total weight 0.23 kg.

The total length of the bottom charge, 0.8 m, and the column charge, 1.0 m, will leave a stemming length of 1.1 m which may cause some boulders from the upper part of the round.

17. Total charge

$$Q_{tot} = Q_b + Q_c = 0.41 + 0.23 = 0.64 \text{ kg}$$

18. Specific drilling

$$b = \frac{n \times H}{w \times B \times K} = \frac{15 \times 2.90}{12.0 \times 0.72 \times 2.5} = 2.01 \text{ m/cu.m.}$$

19. Specific charge

$$q = \frac{n \times Q_{tot}}{w \times B \times K} = \frac{15 \times 0.64}{12.0 \times 0.72 \times 2.5} = 0.44 \text{ kg/cu.m}$$

Summary of important data:

Bench height K (m)	Hole depth H (m)	Burden B (m)	Spacing S (m)	Bottom charge Q_b (kg)	Column charge Q_c (kg)	Specific drilling b (m/cu.m)	Specific charge q (kg/cm.m)
2.5	2.9	0.72	0.86	0.40	0.23	2.01	0.44

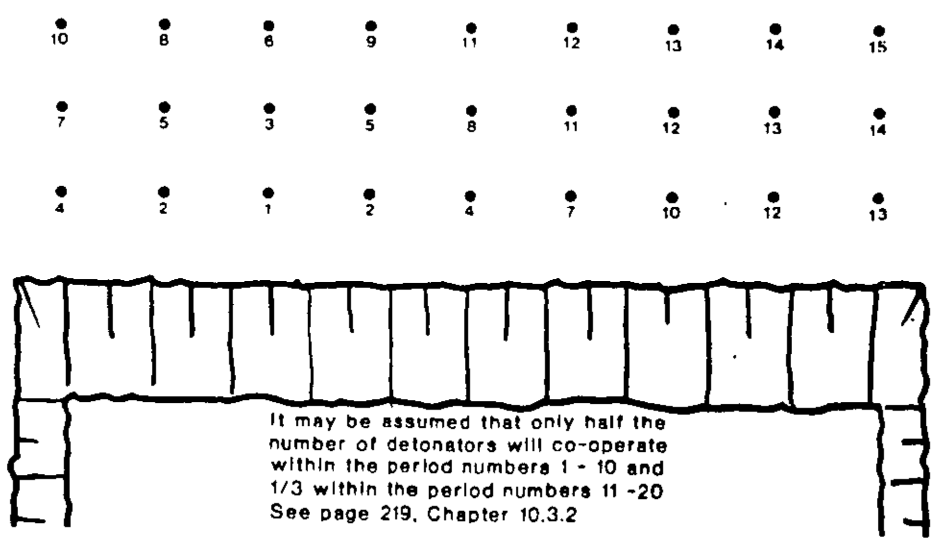


Fig. 10.20 Firing pattern.

10.3.4 Bench blasting with divided charges.

In blasting operations, it is common that the drilling is carried out well ahead of the blasting operation, with the result that the drilling pattern is fixed and cannot be changed if it is found that the requisite charge for the blasthole is higher than the permitted one.

In cases like this, the charge may be divided into two or more smaller charges in the hole which are shot with different period numbers. The upper charge must then always be initiated with the lower period number.

An intermediate sand stemming divides the charges from each other to avoid flash-over between the charges. An explosive's susceptibility to flash-over depends on parameters like age of the explosive, temperature, charge diameter, quality and length of the stemming. The length of the stemming needed between charges varies from 0.4 m for drill series 11 (34–26 mm) to 2.0 m for a blasthole diameter of 150 mm. Too long intermediate stemming could result in more difficult breakage for the lower bottom charge resulting in higher vibration values. The best stemming material has a particle size of 1/10 of the blasthole diameter (for diameters up to 100 mm).

Charge calculation procedure.

Drilling pattern.

1. Maximum burden
B_{max} depends on Q_{per} and is found in table P_1.

2. Charge concentration.
l_b depends on B_{max} and is found in table P_1. Chose suitable explosives units considering the l_b

3. Subdrilling.

$$U = 0.3 \times B_{max} \qquad \text{(m)}$$

4. Hole depth.

$$H = a(K+U) \qquad \text{(m)}$$

a = 1.05 for hole inclination 3:1 and 1.0 for vertical holes.

5. Error in drilling

$$E = \frac{d}{1000} + 0.03H \qquad \text{(m)}$$

TABLE P_1

Determination of B_{max} and l_b with regards to Q_{per}

Permitted charge kg	Maximum burden m	Charge concentration kg/m
Q_{per}	B_{max}	l_b
0.25	0.7	0.25
0.5	0.9	0.4
1.0	1.2	0.7
1.5	1.35	0.9
2.0	1.5	1.1
2.5	1.6	1.25
3.0	1.7	1.4
4.0	1.85	1.7
5.0	2.0	2.0
6.0	2.1	2.2
7.0	2.2	2.4
8.0	2.3	2.7
9.0	2.4	2.9
10.0	2.5	3.1
12.0	2.65	3.5
14.0	2.8	3.9
16.0	2.9	4.2
18.0	3.0	4.5
20.0	3.15	5.0
25.0	3.4	5.8

6. Practical drilling pattern.

Practical burden:
$B = B_{max} - E$ (m)

Practical spacing:
$S = 1.25 \times B$ (m)

The hole spacing is adjusted to the width of the round.

Charging.

Lower partial charge.

7. Weight

$Q_l = Q_{per}$ (kg)

8. Height
$h_l = 1.3 \times B_{max}$ (m)

9. Length of the intermediate stemming, h_s, is between 0.4 and 2.0 m depending on the hole diameter.

Upper partial charge.

10. Residual chargeable length of blasthole.

$h_r = H - h_l - h_s$ (m)

Upper bottom charge.

11. Weight.

$Q_{bu} = 0.75 \times Q_l$ (kg)

Less bottom charge is required as the charge has free breakage which is not the case in the lower charge

12. Height

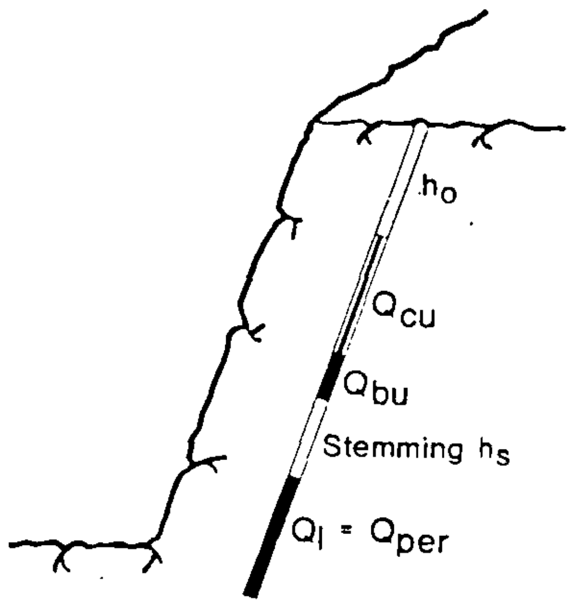
$h_{bu} = 1.3 \times B_{max}$ (m)

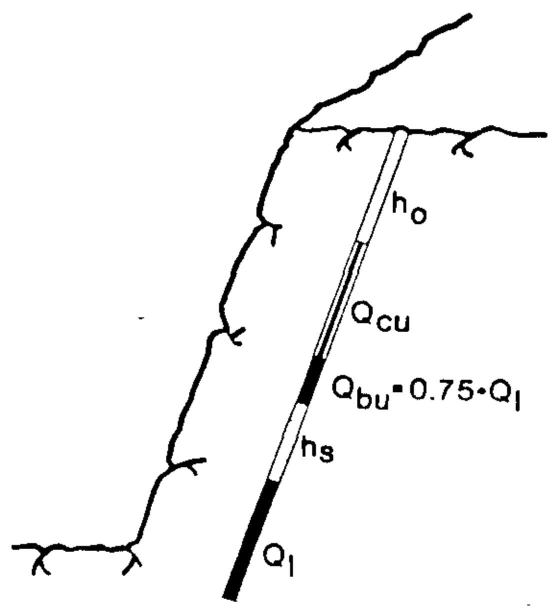
Upper column charge.

13. Stemming part.

$h_o = B$ (m)

The length of the stemming may be adjusted depending on the charge concentration in the column.





14. Height.

$$h_{cu} = h_r - h_{bu} - h_o \qquad \text{(m)}$$

15. Concentration of column charge.

$$l_{cu} = \frac{Q_{per} - Q_{bu}}{h_{cu}} \qquad \text{(kg/m)}$$

Judge if the calculated charge concentration in the column is sufficient in relation to the bottom charge. It should be at least 40 % of the concentration of the bottom charge. If the remaining charge weight is not large enough to obtain an acceptable charge concentration in the column, a third charge must be used in the hole.

16 Total charge weight – upper partial charge.

$$Q_u = Q_{bu} + Q_{cu} \qquad \text{(kg)}$$

Check that $Q_u \leq Q_{per}$.

10.3.5 Bench blasting with divided benches.

The blasting of the area closest to a building often means that the methods of reduced drilling pattern or divided charges will not suffice, but the bench heights have to be reduced.

Over short distances, under 5 m, the ground vibrations are only slightly damped so the values in the charge/distance graph should be followed. The blastings must also be continuously followed up with ground vibration measurement. Any change of the vibration velocity must be taken into account for the planning of subsequent blasts.

Faults and incompetent zones may cause unexpected problems in the immediate vicinity of buildings by displacement of surface rock or gas expansion under the building.

The rounds close to the building should have free breakage to avoid upward movement of the surface rock. The blastholes closest to the building should have weak column charges which cut off the rock thus preventing backbreak.

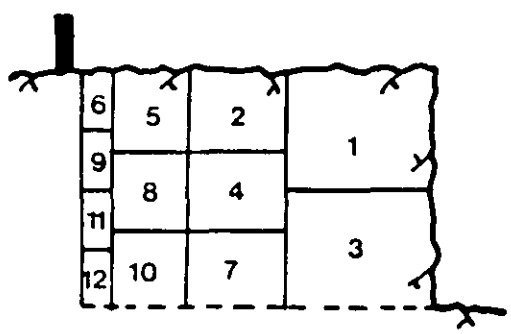


Fig. 10.21 Blasting order close to buildings.

10.3.6 The slot drilling method.

Lately SKANSKA (Major Swedish contractor) has patented a method of reduc-

Fig. 10.22 The SKANSKA slot drilling method.

ing ground vibrations from blasting close to existing buildings.

A slot is drilled which separates the building from the blasting site. The slot consists of holes drilled in parallel to form a fully open slot. The slot must be free from drill cuttings and water and extend a certain distance outside the object to be protected to give the best result.

Ground vibration measurements have shown that the slot acts as an effective damper of ground vibrations. This implies that the drilling and blasting costs can be reduced, as more effective drilling and charging patterns can be used.

Furthermore, the method may imply further savings as the need to reinforce the rock is reduced or in certain cases eliminated.

10.3.7 Trench blasting with reduced burden.

A lot of today's trench blasting is done in populated areas and consequently in the immediate proximity of buildings.

Due to the increased constriction of the rock in trench blasting, the ground vibrations increase and consequently the risk of damage. Therefore, it is of the utmost importance that the blasting operation is planned and executed in a scrupulous way.

The charge calculations for cautious trench blasting will generally follow the same pattern which is used for bench blasting. As in bench blasting, the basis for the calculation is the required specific charge in kg/cu.m.

Charge calculations procedure.

Drilling pattern.

1. Number of holes in each row, n, is found in the drilling and charging tables for trench blasting, Chapter 6.

2. Permitted charge is found in charge/distance graph and the charge per row is:

$$Q_{row} = n \times Q_{per} \quad (kg)$$

3. Hole depth.
H (m) is found in the drilling and charging table for trench blasting, Chapter 6.

4. Specific charge.
q (kg/cu.m.) is found in graph R_1.

5. Practical burden.

$$B = \frac{Q_{row}}{q \times H \times w} \quad (m)$$

Charging.

6. Bottom charge.
Q_b in accordance with graph R_2

7. Height of bottom charge

$$h_b = \frac{Q_b \times 1000}{d^2} \quad (m)$$

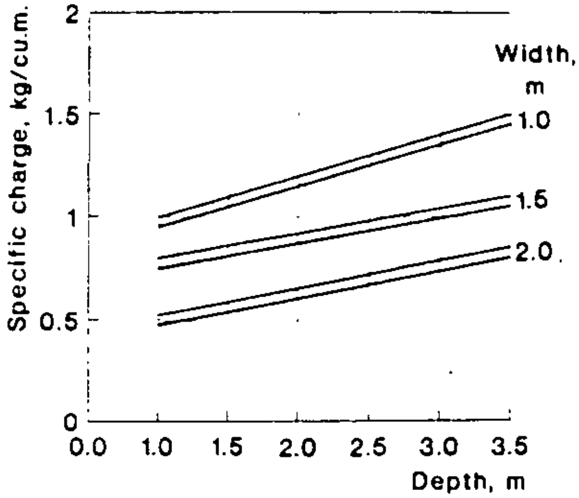
8. Height of stemming

$$h_o = B \quad (m)$$

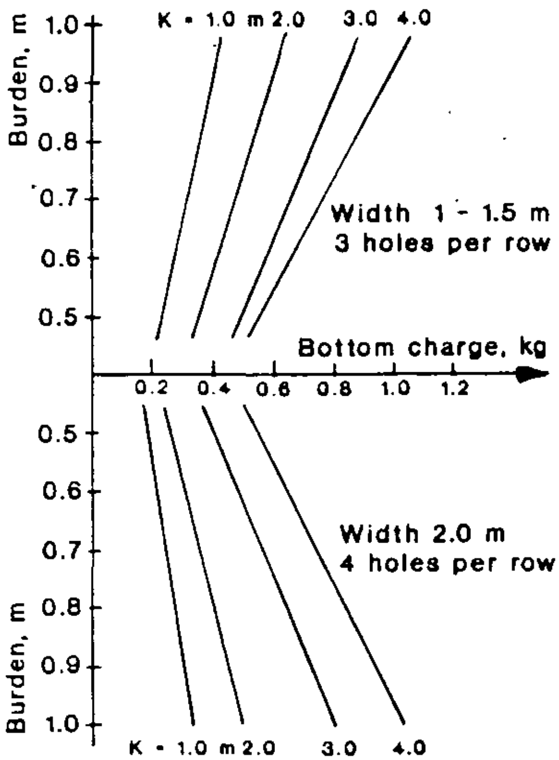
h_o should be adapted to the burden B and the concentration of the column charge. From a ground vibration point of view it is favorable to have short stemming (higher column charge) on condition that throw can be controlled by covering the blast and that the required charge concentration is obtained.

GRAPH R1

Specific charge as function of depth and width.



GRAPH R2



9. Weight of column charge

$Q_c = Q_{per} - Q_b$ (kg)

10. Height of column charge

$h_c = H - h_b - h_o$ (m)

11. Concentration of column charge

$$l_c = \frac{Q_c}{h_c} \quad \text{(kg/m)}$$

12. Weight of column charge

$Q_c = l_c \times h_c$ (kg)

13. Total charge weight

$Q_{tot} = Q_b + Q_c$ (kg)

Check that $Q_{tot} \leq Q_{per}$.

TABLE R₁

Recommended charge concentration of the column charge in relation to the practical burden in trench blasting.

Practical burden B (m)	Minimum required charge concentration l_c (kg/m)	Suitable explosive	Real charge concentration l_c (kg/m)
0.3	0.05	Gurit, 11×460 mm	0.08
0.4	0.07	Gurit, 11×460 mm	0.08
0.5	0.1	Gu 11* + 1/4 Em 150*	0.12
0.6	0.15	Gu 11* + 1/2 Em 150*	0.15
0.7	0.18	1/4 cartr. Em 150* + 10 cm wooden stick	0.20
0.8 0.9	0.22 0.25	1/2 cartr. Em 150* + 10 cm wooden stick	0.25 0.25

Gu 11* = Gurit, 11×460 mm
Em 150* = Emulite 150, 25×200 mm.

10.3.8 Trench blasting with divided charges.

The method of using divided charges in trench blasting can be used when the hole depth exceeds 2.0 m. This is because the intermediate stemming and the normal stemming occupy a minimum length of 1.0 m.

Charge calculation procedure.

Drilling pattern.

1. The number of holes in each row is found in the drilling and charging tables for trench blasting.

2. Hole depth.
H (m) is found in the drilling and charging tables for trench blasting.

3. Practical burden.
B (m) is found in graph R_2 if the lower bottom charge is equal to Q_{pcr}.

Charging.

Lower bottom charge.

4. Weight.

$Q_{bl} = Q_{pcr}$ (kg)

5. Height.

$$h_{bl} = \frac{Q_{bl} \times 1000}{d^2} \quad \text{(m)}$$

6. Height of intermediate stemming.
$h_s = 0.4$ to 1.0 m. The value depends on the circumstances, e.g. the blasthole diameter.

Upper partial charge.

7. Residual chargeable height of the blasthole.

$h_r = H - h_{bl} - h_s$ (m)

Upper bottom charge.

8. Only 60 % of charge Q_{bl} is needed for the breakage of the upper part as the hole has free breakage, i.e. no constriction.

$Q_{bu} = 0.6 \times Q_{bl}$ (kg)

9. Height.
h_{bu} is estimated from the chosen charge unit.

Upper column charge.

10. Charge concentration.

l_{cu} from table R_3 (kg/m)

11. Stemming.

$h_o \geq B$ (m)

Adjusted to the charge concentration in the column of the hole.

12. Height.

$h_c = h_r - h_{hu} - h_o$ (m)

13. Weight.

$Q_{cu} = l_{cu} \times h_{cu}$ (kg)

14. Total charge weight.
Upper partial charge.

$Q_u = Q_{hu} + Q_{cu}$ (kg)

Check that $Q_u \leq Q_{per}$

10.3.9 Cautious tunnel blasting.

An increasing number of tunnels are being constructed under built-up areas where they pass under inhabited buildings as well as buildings with equipment sensitive to ground vibrations.
Cautious blasting followed up with ground vibration measurement has subsequently become more common
The blastholes in a tunnel round are very constricted. To decrease ground vibrations, it is necessary not only to lower the co-operating charge, but also to endeavor to reduce constriction of the rock. This means that drilling pattern, hole depth, charge per hole and firing pattern have to be adjusted so that the permitted co-operating charge is not exceeded and that all holes have free breakage
Some of the most important points to consider are:

- Choice of blasthole diameter and explosive.
- Choice of large hole diameter, one or several large empty holes in the cut to decrease constriction and the risk for flash-over.
- Accuracy in drilling.
- Suitable firing pattern which minimizes the co-operating charge and guarantees most favorable angle of breakage.
- Dividing the round into partial rounds.
- Reduction of the distance between the holes so the charge in each hole can be reduced.
- Reduction of the hole depth.

The firing pattern is of the utmost importance in cautious tunnel blasting. The constriction of the blastholes can be decreased by using the right period number, so that each hole has an angle of breakage of at least 90° in the stoping part of the round (See Fig. 7 15 in Chapter 7 Underground blasting). If the tunnel round is of such a size that the number of periods does not suffice without exceeding the permitted co-operating charge, the blast must be divided into two or more partial blasts e.g. blast the constricted cut holes as a separate blast, then the stoping holes and finally the contour holes. In order to obtain the best result in the contour, the perimeter holes (except the floor holes) should be blasted with the same period number. This is normally no problem as the contour holes usually have very low charge concentration.

In cautious blasting, certain types of cuts like V-cuts are not suitable because of the risk of co-operation and flash-over between the large number of holes in the cut. Large hole cuts have earlier been considered to give rise to large ground vibrations but measurements of ground vibrations analyzed over a long period contradicts this. A parallel hole cut with two or more large holes is a very practical way of reducing constriction and unsuccessful breakage and is thus to be recommended.

It is often more advantageous to drill more holes in a round with reduced charges than to shorten the hole depth, thus maintaining normal advance of the round. However, sometimes the hole depth has to be reduced in order to keep the co-operating charge within the permitted limits.

A continuous follow-up of the blasting activities by ground vibration measurement may be beneficial by disclosing more favorable practical ground vibration values than those determined by theoretical calculations.

The adaptation of the blasting operation to the measured results means an optimum rate of drifting on the basis of vibration measurement.

Problems with flyrock and air shock waves do occur in the initial stage of the tunneling operation and constitute a risk when the work is started in populated areas, which is often the case nowadays. Therefore, it is important to investigate the rock with regard to fissures and incompetent zones and than cover the blast well. The air shock wave is troublesome not only in the initial stage of the work but also when the drifting has advanced further into the rock, especially in the direction of the tunnel.

10.4 Blasting close to hardening concrete.

Blasting works are often carried out simultaneously with construction work which give rise to problems with blasting close to hardening concrete.

The problem has been studied by the Ontario Hydro, Concrete and Masonry Research Section and the following recommendations are given for concrete with STD cement, without entering too deeply into theories and research results.

If it is presumed that concrete which hardens in a temperature of $+5°$ C can stand a peak particle velocity of 100 mm/sec. after 90 days, the following vibration velocity values are recommended:

Hardening time days	Maximum permitted vibration velocity mm/sec
2	8
3	11
7	35
28	80
90	100

Note: Up to 10 hours after casting, the concrete can stand ground vibrations of up to 100 mm/sec. Between 10 and 70 hours after casting no blastings should be undertaken closer than 30 meters.

On the other hand, if it is presumed that the concrete is hardening in a temperature of +21° C and that the concrete stands 100 mm/sec of ground vibration after 90 days, the following is recommended:

Hardening time days	Maximum permitted vibration velocity mm/sec
1	14
2	30
3	40
7	60
28	85
90	100

Note. The concrete can stand ground vibrations of up to 100 mm/sec up to 5 hours after casting. No blastings should be undertaken closer than 30 meters between 5 and 24 hours after casting.

10.5 Flyrock.

Cautious blasting does not only mean the control of ground vibration but also the control of flyrock.
The control of flyrock and its prevention has been dealt with thoroughly in Chapter 5.8 Throw, flyrock.

10.6 Air shock waves.

The immediate effect of blasting is not only to cause ground vibrations and throw, but also an air shock wave.
In most routine blastings, in which the explosives are enclosed in blastholes, and which are designed for ground vibration velocities of 70 mm/sec or less, the blasting does not cause air shock waves of the magnitude that may cause damage to buildings.
However, a low level of air shock wave overpressure does play an important role in distressing neighboring residents by rattling windows etc. Therefore, complaints may be reduced by taking actions to reduce overpressure from air shock waves.

Air shock waves are pressure waves which radiate in the air from a detonating charge. The intensity of the pressure depends on the size of the charge and on its degree of confinement. When a pressure wave passes a given position, the pressure of the air rises very rapidly to a value over the ambient atmospheric pressure. It then falls relatively slowly to a pressure below the atmospheric value before returning to the atmospheric pressure after a series of oscillations. The maximum pressure is known as the peak air overpressure. The air shock waves are within a wide range of frequencies, typically between 0.1 Hz and 200 Hz. In the portion of the spectrum lying over 20 Hz the air shock waves are audible and known as noise, while concussion is the portion under 20 Hz and inaudible.

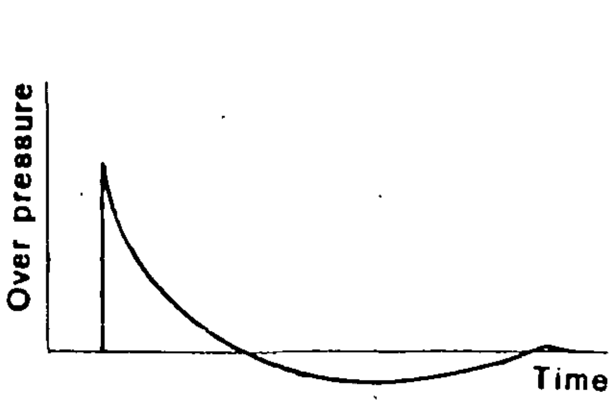


Fig. 10.23 Pressure/time curve for air shock wave.

The lower, inaudible, frequencies are damped more slowly than the higher, audible, frequencies and cause overpressure over greater distances. These low frequencies can occasionally cause direct damage onto structures, but can more commonly induce higher frequency vibrations which are noticed as noise in windows, doors, crockery etc. Under such circumstances it is impossible to determine whether the ground vibration or air shock wave is being perceived without monitoring the blast.

The air overpressure is measured as units of pressure and usually pressure unit millibar (mbar) is used. The units decibel (dB) and kilopascal (kPa) are also used.

The decibel unit is expressed as:

$$dB = 20 \log \frac{P}{Po}$$

where P is the measured pressure and Po the reference pressure of 0.00002 Pa.

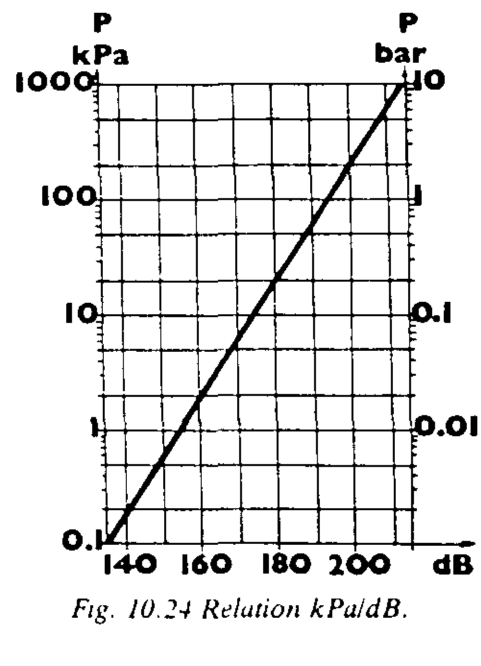


Fig. 10.24 Relation kPa/dB.

Knowing the charge weight Q (kg) and the distance R (m) to the charge, the overpressure can be calculated from the formula:

$$P = 700 \frac{Q^{1/3}}{R} \text{ (mbar)}$$

The relationship applies to TNT, which means that for civil explosives type Emulite 150 and Dynamex M the charge weight should be reduced by 25 % when used in the formula.
The relationship applies to unconfined charges.

The unconfined charges which cause problems in populated areas are concussion charges (mudcapping), trunklines of detonating cord, welding of powerlines with explosives, presplitting with unstemmed holes etc.
As can be seen in Fig 10.25, a trunkline consisting of 100 m 10 gr detonating cord can cause broken windows at a distance of up to 100 m.

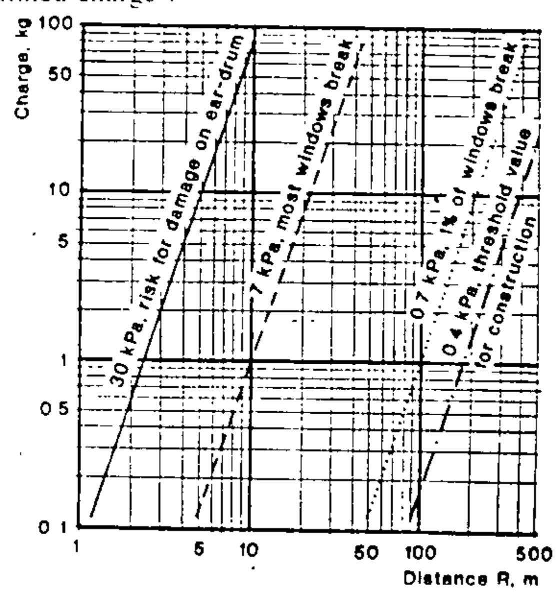
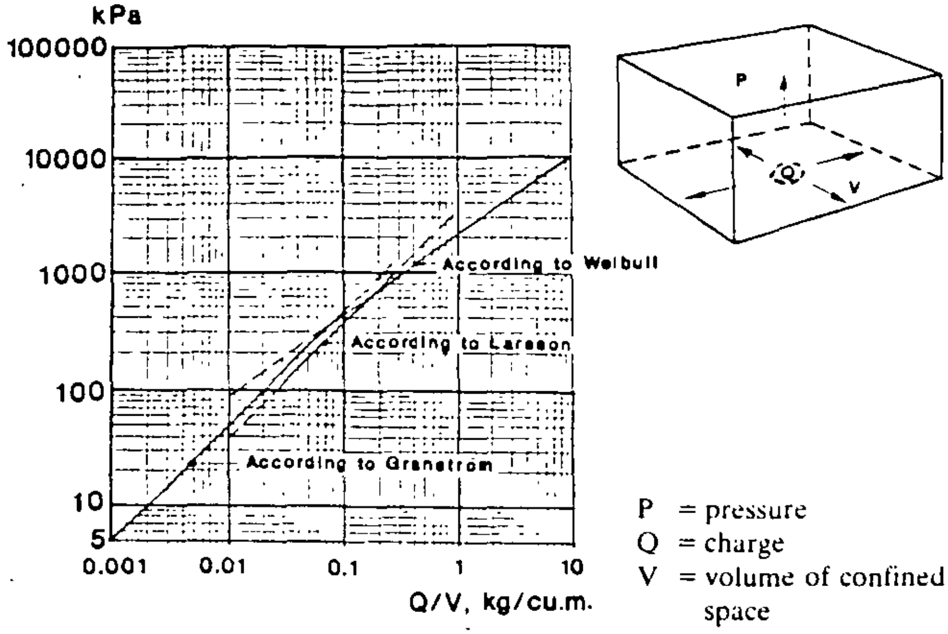


Fig. 10.25 Charge as a function of distance for different levels of air overpressure.

The propagation of the air shock waves is influenced by atmospheric conditions where the wind direction, wind velocity, temperature and air pressure have a great effect.
Reflexions in the atmosphere may be caused by temperature inversion, where the air shock wave is reflected against the boundary layer of air strata with different temperatures. Temperature inversion frequently occurs on cloudless evenings, nights and mornings. The phenomenon can cause local amplification of the air overpressure, which is greater than that which would normally have been expected at a certain distance.
Even if the air overpressure is kept under the threshold value for buildings (0.4 kPa), it is not always sufficient to safeguard against complaints. The blasts should therefore be designed for the minimum practical level.

Air overpressure in confined spaces.

In the case of blasting in underground chambers and tunnels, different conditions prevail as the pressure wave is confined and in the case of tunnels concentrated in one direction. This means that the pressure is amplified compared to blasts in an open space.



P = pressure
Q = charge
V = volume of confined space

Fig. 10.26 Pressure as a function of charge and volume from a detonation in a confined space.

The principal sources of air overpressure are:

- Detonation of unconfined charges. The most common are concussion charges (plaster charges), trunklines of unconfined detonating cord, blastwelding of powerlines and presplitting with unstemmed holes.
- Too short stemming and/or wrong stemming material. Inadequate stemming might not confine the explosive on detonation.
- Venting of high velocity gases may occur in poorly designed blasts where no consideration has been given to incompetent zones, principally in the burden area. Overcharging of a blasthole could cause the same effect.
- The sudden movement of the blasted rock mass towards the free face or faces will raise the air pressure.

In order to control the air shock waves, the following steps should be considered:

- Design the blast in such a way that the amount of explosives is in accordance with blasting requirements and minimum air overpressure.
- Pay particular attention to incompetent zones, overbreak from previous round, mudseams etc. through which gases may vent and cause overpressure.
- Accurate drilling is necessary to maintain the designed blasting pattern. Too big a burden could cause venting in the collar part of the hole. Use setbacks to determine the burden of the next round.
- Bottom initiation decreases venting in the stemming area. See Chapter 5.8 Throw, flyrock.

- Reduction of the size of the round tend to reduce the air overpressure.
- If possible, the development of the benches should be such that the blasted material is thrown away from residential areas.
- The stemming material should be of sufficiant quantity and quality to confine the explosives on detonation. Crushed stone material size 4 to 9 mm gives better confinement than drill fines.
- Check the rise of the explosives column during charging to minimize the risk of overcharging in any void or fault.
- Avoid excessive delays between holes to prevent underburdening the holes.
- In multiple row blasting the delay between the rows should be longer then the delays between the holes in the row. In deep rounds, this promotes forward rather than upward movement of the burden
- Do not use concussion charges in populated areas for secondary blasting and boulder blasting.
- If misfired underburdened holes have to be fired, use screening materials e g. sandbags or loose sand to cover. The thickness of the cover has to be sufficient both to avoid flyrock and to damp the air shock wave.
- Surface lines of detonating cord should be avoided in residential areas. If electric firing is not allowed or possible the non-electric firing system NONEL should be used. If detonating cord is the only firing device available, trunk-lines and connecting lines should be covered with at least 600 mm of absorbent material, preferably sand.
- As speed and direction of the wind are major influences on the magnitude of air overpressure, blasting should, if possible be avoided when the wind is blowing towards critical areas.
- Blasting should be avoided in early mornings, late afternoons and evenings when temperature inversions are likely to occur:
- Schedule blasts to times when the noise level from surrounding sources is at its highest and when the neighbors are busy or expect blasting to occur
- Employ an audible warning system immediately before every blast. If the blasting is an isolated occurrence, give specific warning indicating approximate time of the blast
- Maintain good public relations. Give good and adequate information about the work, duration and disturbances to be expected. The most stringent measures against air overpressure can be rendered useless without good relationship between the blasting crew and the neighbors.

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TEMA

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Finite Element Modelling of Crack Propagation in Presplit Blasting

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Abstract

Pressure-time histories were recorded for low density ammonium nitrate/fuel oil, detonated in long heavy walled steel cannons of various bores. These were then used in a finite-element model of a horizontally layered limestone rock mass to predict the crack propagation limits in presplit blasting for a range of borehole diameters. Apart from showing very good agreement with field results, the model clearly demonstrated the strong dependence of the results on the pressure-time curve. The important elements were the peak pressure, the rise time to it and its duration. Control of these characteristics offers the possibility for optimization of crack propagation distances and borehole spacings for various ground conditions. Also, the results to date provide the basis for investigating the characteristics of the explosives presently used in presplit blasting, and finding ways to modify them with the purpose of optimizing the field results.

Introduction

Some years ago in a comprehensive review article Mellor (1975) summarized the state of the art on presplitting in the form of graphs relating blasthole spacing to hole diameter. As pointed out in the review, the published data suffered from a lack of physical rock properties and structural detail. Later a static model was developed, CANMET (1977), Bauer (1982), which yielded the following expression for presplit hole spacing when multiple holes were fired simultaneously.

$$S \leq 2r\,(P_b + \sigma_t)/\sigma_t$$

where

r – borehole radius

P_b – pressure at the borehole wall

σ_t – rock tensile strength

If the pressure at the borehole wall is matched to or is less than the compressive strength of rock (σ_c) then localized crushing can be avoided. Figure 1 is a plot of the data presented in Mellor (1975) along with the static model predictions for various values of the ratio of the compressive to tensile rock strengths, σ_c/σ_t.

Whilst the use of this static model or the empirical rules of thumb often give good results they are nonetheless deficient when new

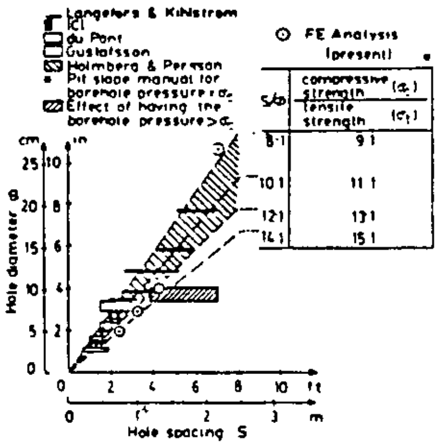

FIGURE 1 - RELATIONSHIP BETWEEN HOLE DIAMETER AND SPACING FOR PRESPLITTING

situations arise. More realistic models should be able to predict accurately the influence of changes in the pressure-time curve within the borehole, on ground response and cracking limits. In addition, it should be possible to quantify the effect of joint frequency, orientation, and properties on the presplit hole spacing.

The research described in this paper consisted of the measurement of pressure-time curves for low density AN/FOs detonating in long heavy walled small bore steel cannons. These curves had to be modified to represent the larger charge and borehole diameters employed in field. Then they were used in a two-dimensional finite element model to predict the changes in stress distribution with distance and time in a limestone rock mass. This allowed the crack limits radiating from a single borehole to be determined as a function of borehole diameter and driving force. Those factors which contributed strongly towards maximizing this distance were identified.

Model Definition

The dynamic behavior of rock under the action of time-dependent pressures was simulated by means of a versatile computer finite element code, Hibbitt et al (1982). The code had incorporated one of the modern incremental theories of plasticity, Chen and Chen (1975), which are based on a close relationship between the plastic strain increment, the current state of stress, and the stress increment. In the particular theory adopted, all stress distributions that can cause yielding are described by a single function which represents a surface in stress space (yield surface). In the same fashion, it is possible to determine a failure surface. The shapes of the above two surfaces are determined through experiments on specimens under different loading combinations. A succession of surfaces between the yield and failure surface represents the different stages of loading after yielding and before failure (loading surfaces). Such surfaces depend on the plastic strain history. The form of these loading surfaces, which is an evolution of the yield surface, is determined by the hardening rule best fitting the material behavior.

How the plastic strain increment is connected to the state of stress and stress increment is decided by the flow rule; its choice plays a very important role in the generation of reliable results and it is guided by experimental procedures.

The incremental theory of plasticity can be easily adopted in order to predict the response of materials with high compressive and low tensile strength such as plain concrete, rock, soils, etc, under the action of loads.

The model geometry, boundary conditions, stress distribution and characteristics of the analysis in the present study were dictated by the physical and technical aspects of presplit blasting and also by the physical rock properties.

The limestone considered was free of joints, isotropic, linear-elastic, strain-hardening, plastic-fracturing with high compressive and low tensile strength (Figure 2).

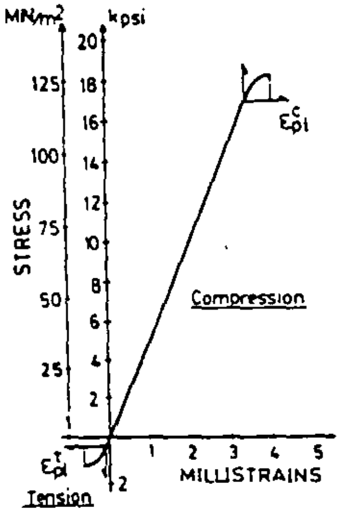

FIGURE 2 - STRESS-STRAIN CURVE FOR LIMESTONE

In the model, only one hole was considered. This is because in the field, between holes fired together, there is always delay time long enough to allow complete crack formation before any of the adjacent holes is initiated.

Because of the existing symmetry with respect to hole center and to every straight line passing through it, only a 10° wedge of the area surrounding the borehole was analyzed.

Borehole radii considered were equal to 1.0 in. (25.4 mm), 1.5 in. (38.1 mm) and 2 in. (50.8 mm). On the other hand the external radius considered depended on the expected crack propagation distance.

The area was divided into a number of elements. The number of these elements depended on limestone properties, Valliappan et al (1983), White et al (1979), on how fast the pressure changed, and how long the crack was expected to be. The elements were 0.10-in. (2.5 mm) thick and had 8 nodes. The stress distribution corresponded to plane strain. Up to 700 elements were used.

In compliance with the existing symmetry only displacements in the radial direction were allowed. The outer boundaries were restricted in both directions. The time-dependent pressure acted internally as a uniformly distributed load.

The results which were given in the form of stresses, displacements, velocities and elements cracked were computed every 0.25 - 10 μsec.

Comparison between plane strain and plane stress distribution in the wedge model yielded the same conclusions. The relative insensitivity to the type of stress distribution is attributed to the self-containing nature of the material - small Poisson's ratio 0.1.

Input Data Requirements

Apart from the model geometry, node location and division into elements the other required data were the limestone, mechanical and physical properties, and the pressure-time profile of low density AN/FO.

The limestone properties were determined in the laboratory from 5 in. (127 mm) long, 2-1/14 in. (57 mm) diameter rock cores, taken from rock blocks in three mutually perpendicular directions relative to the bench face from which the blocks were selected from regions uninfluenced by blasting. Uniaxial compression tests determined the yield stress and ultimate failure stress, while Brazilian tests gave the tensile strength. The measured physical properties were:

Density: 0.00025 $lb\text{-}sec^2/ft$ (2.7 g/cm^3)

Young's Modulus: 5×10^6 psi (34.5×10^6 kN/m^2)

Maximum uniaxial compressive stress at zero plastic strain: 18,000 psi (124,110 kN/m^2)

Uniaxial compressive strength: 18,300 psi (126,180 kN/m^2)

Maximum plastic strain at peak compressive strength: 0.5×10^{-3}

Uniaxial tensile strength: 1500 psi (10,343 kN/m^2)

Poisson's Ratio: 0.1

Since the finite element code determines failure when a certain material dependent surface is reached, additional parameters defining this failure surface were required. Some of them were assumed on the basis of similarly behaving materials, Chen et al (1975), Chen (1982), and some were measured as stated below.

These additional parameters were:

- Ratio of each biaxial compressive strength/uniaxial component: 1.16 (assumed).
- Ratio of uniaxial tensile/compressive strength: 0.082 (measured).
- Ratio of a plastic strain component at failure under biaxial compression to the plastic strain at failure under uniaxial compression: 1.28 (assumed).
- Ratio of plastic strain at failure under uniaxial tension to plastic strain at failure under uniaxial compression: 0.01 (assumed). This ratio was varied in one part of the analysis in order to study the effect of plasticity in tension on the final results.

The next step was the determination of the pressure-time profile of the explosive. This step involved the development of a new experimental technique and new instrumentation. Both these are described in the next section.

Experimental Pressure-Time History Determination

After determining the limestone properties in the laboratory, to determine the pressure time curves for AN/FO at densities of 0.16, 0.20 and 0.24 g/cm^3 a series of fully coupled, cylindrical charges was detonated. The charges were placed in 4-ft (1.22-m) long, thick walled, steel cylinders having internal diameters of 3/8 in. (9.5 mm), 5/8 in. (1 59 mm), and 1 in. (25.4 mm) with corresponding external diameters of 2-1/2 in. (63.5 mm), 3-1/2 in. (88.9 mm) and 4-1/2 in. (114.3 mm).

The explosive was ground to -100 mesh, for adequate sensitivity at these diameters and was blended with microbubbles to yield the required densities.

Four high-pressure quartz transducers, capable of measuring pressures up to 150 000 psi (862 000 kN/m^2) were placed in small diameter cylindrical holes drilled perpendicular to the cannon bore. The gauge tips were in contact with the explosive for direct pressure measurement (Figure 3).

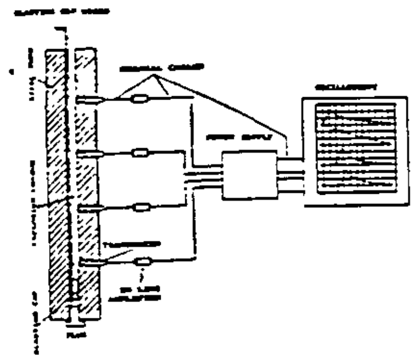

FIGURE 3 - SKETCH OF EXPERIMENTAL SETUP FOR PRESSURE-TIME RECORDING

The end of the cannon where the charge is initiated was closed airtight by means of a plug, while the other one was left open. The explosive was placed in the cannon and initiated from the closed end, with a No. 8 electric blasting cap.

Special mountings were developed for the transducer to eliminate self-induced vibrations caused by precursor waves in the cannon walls. The gauge fittings also included insulators to stop heat reaching the transducers and rubber and brass plugs to prevent gas leakage.

Results

A typical pressure-time profile is shown in Figure 4. The first 16 μsec of the profile a shown in Figure 5, to demonstrate its characteristics more clearly. Notwithstanding the many precautions that had been taken, mechanical vibrations set up in the transducer assembly itself generated output that was superimposed on that due to the explosive.

To determine the input pressure-time profile of the explosive, the transducer structure was modeled with three-dimensional finite elements, 171 in total (Figure 6). It was then subjected to a number of

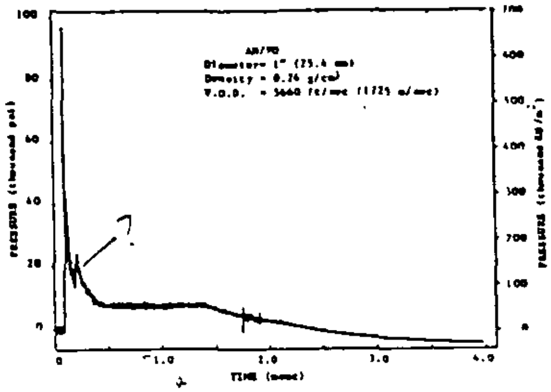

FIGURE 4 - PRESSURE-TIME PROFILE

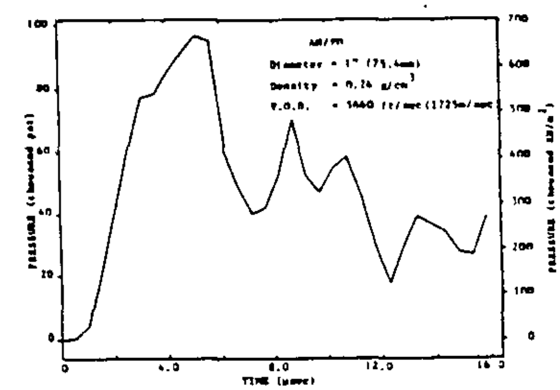

FIGURE 5 - PRESSURE-TIME PROFILE (FIRST 16μ SEC)

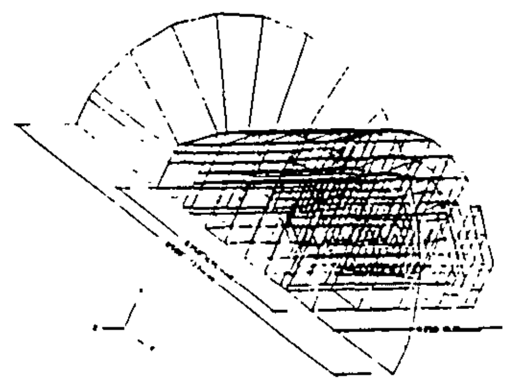

FIGURE 6 - 3-D FINITE ELEMENT MODEL OF THE TRANSDUCER STRUCTURE

different pressure-time profiles. The pressure was applied against the external surfaces of the transducer tip elements in a sequential fashion to simulate closely the true continuous application. Through trial and error it was possible to determine the pressure-time profile (Figure 7) which, when applied to

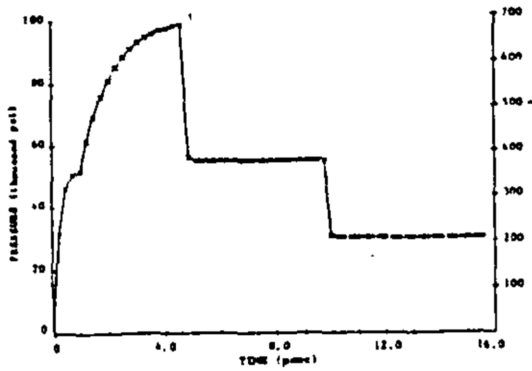

FIGURE 7 - INPUT PRESSURE-TIME PROFILE

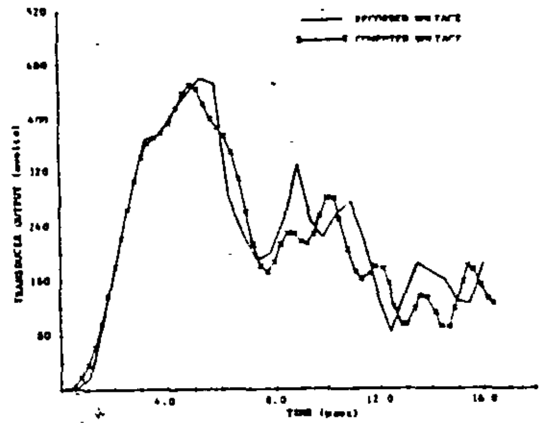

FIGURE 8 - COMPARISON BETWEEN COMPUTED AND RECORDED TRANSDUCER OUTPUT

the transducer tip, produced a pressure-time history very close to the recorded one (Figure 8). This is the input pressure-time profile of the explosive.

Through these experiments, it became possible to determine that the ideal or theoretical velocity of detonation for a density of 0.20 g/cm^3 is 4813 ft/sec (1467 m/sec) and this was obtained in diameters of 5/8 in. (15.9 mm) or greater; for 0.24 g/cm^3 it is 5660 ft/sec (1725 m/sec) and was given at diameters of 3/8 in. (9.5 mm) or greater.

The corresponding true pressure-time curves for ideal detonations were the ones used in the analysis after certain modifications.

Application of the Determined Input p-t Profile in the Finite Element Analysis

Before proceeding with the analysis, certain assumptions and modifications had to be made concerning the explosive and its pressure-time profile. These assumptions were:

First, the charges were assumed to have a density of 0.24 g/cm^3.

Second, because crushing around the borehole walls is of primary concern in presplitting, the peak pressure had to be reduced to a level (17 600 psi or 121 352 kN/m^2) not exceeding the compressive rock strength (18 300 psi or 126 180 kN/m^2). In the field, this is accomplished by decoupling the charge to various degrees. A constant coupling ratio of 0.92 was kept in all cases.

In addition to lowering the peak pressure, decoupling changes the characteristics of p-t profiles. Consequently, certain modifications had to be made in the determined input profile of the explosive to accommodate these changes.

These modifications were:

1 - The first peak was eliminated because it is associated with the detonation head and it does not exist even within the explosive behind it. With more reason it is not transmitted through the gap between the charge and the borehole walls. In support of that come recordings of decoupled charges (upper right corner Figure 9) having 0.340 in. (8.6 mm) diameter in 0.375 in. (9.5 mm) diameter cannon bores.

2 - The rise time (t_r) to peak was increased to 150 μsec (upper right corner Figure 9) and it is expected to increase with larger hole diameters, and

3 - Also, the stay at peak pressure (t_{df}) increased - in the neighborhood of 50 μsec (upper right corner Figure 9).

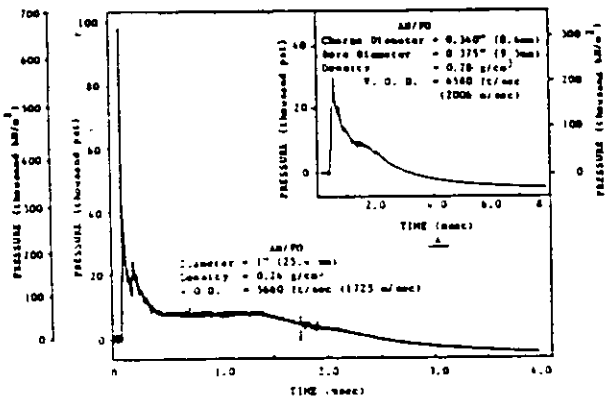

FIGURE 9 - PRESSURE-TIME CURVES FOR COUPLED AND DECOUPLED CHARGES

The modified input profile with already lowered peak pressure consisted the first part of the forcing function acting against the borehole walls during the numerical simulation. The rest of the forcing function was made of the recorded profile (Figure 4), after the part corresponding to the unmodified input profile was removed and the remaining part was translated to the right by $t_r + t_{df} = 200$ μsec.

The beginning of the descending part of the profile was also shifted to the right as a result of the borehole length. This shift location is dependent and equal to the time that the detonation needs to travel the distance between the point in question and the end of the charge, plus the time it takes the pressure drop to reach the same point. This shift is much longer than the time within which the crack is completed (= 750 μsec) and consequently does not affect the final results.

Drop in borehole pressure due to expansion of the walls was not considered at all because the failure is associated with an insignificant 0.035 in. (0.9 mm) increase in a 4-in. (102-mm) borehole diameter.

Larger hole diameters are expected to have a more pronounced effect on the p-t profiles of decoupled charges, demonstrated mainly through changes in rise time and stay at peak, even for the same coupling ratio. Consequently, the crack propagation distance is affected by these changes and their influence on it was demonstrated in this study.

Results - Predictions

The analysis was carried out in two parts. These dealt with the effect of rock and explosive properties on crack propagation distance (L_{cr}).

During the first part of the analysis, the rise time, stay at peak and hole diameter were kept constant ($t_r = 150$ μsec, $t_{df} = 50$ μsec, $\phi = 4$ in. or 102 mm). The rock tensile strength, the rock compressive strength, the modulus of elasticity and the degree of plasticity were varied. The findings are presented below.

(a) The effect of material tensile strength on crack length is illustrated in Figure 10. For tensile strength of 1500 psi (10 343 kN/m^3) the crack length is 22.6 in (57.7 cm). Reduction of the tensile strength by 10% ($\sigma_t = 1350$ psi or 9300 kN/m^2) resulted in a crack length increase to 24.5 in. (62.2 cm), (an 8% increase), while reduction by 33% ($\sigma_t = 1000$ psi or 6895 kN/m^2) yielded a 37% increase to 31.3 in. (79 cm).

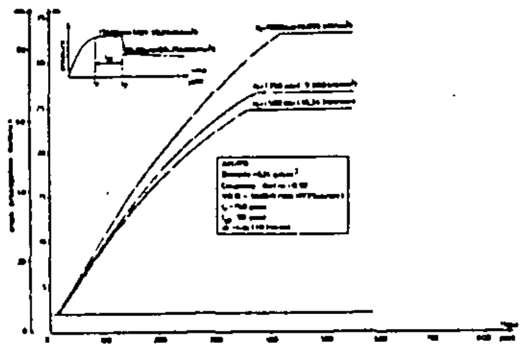

FIGURE 10 - EFFECT OF TENSILE STRENGTH ON CRACK PROPAGATION DISTANCE

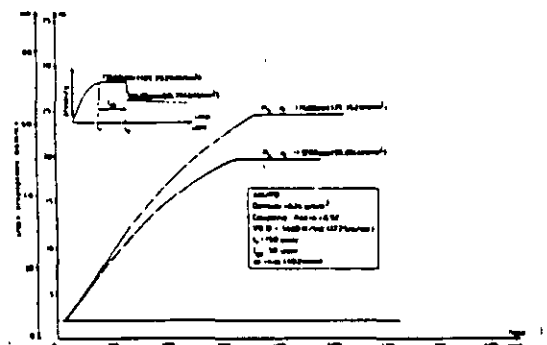

FIGURE 11 - EFFECT OF COMPRESSIVE STRENGTH ON CRACK PROPAGATION DISTANCE

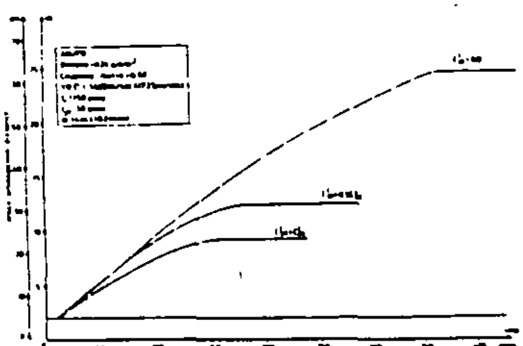

FIGURE 12 - EFFECT OF PLASTICITY ON CRACK PROPAGATION DISTANCE

(b) The magnitude of internal pressure or rock compressive strength affects the crack propagation limits in a similar fashion (Figure 11). A 25% decrease in the magnitude of the above quantities reduced the crack length by 22%.

(c) The final distance of crack propagation is rather insensitive to changes in modulus of elasticity (E) for stiff materials. A 10% decrease in E resulted only in a 2% decrease in L_{cr}.

(d) Departure from the linear elastic behavior in tension with the same Young's modulus as in compression and introduction of plasticity reduces L_{cr} (Figure 12). The relations between crack length, time and amount of tensile plastic strain at failure expressed as a percentage of compressive plastic strain at failure (Figure 2) are shown in Figure 12. A 50% tensile plastic strain at failure produced a 54% decrease in crack length, while a 100% produced a 66% decrease.

In the second part of the analysis the effects of rise time and pressure stay at peak on crack length were studied.

The importance of rise-time from zero to peak is demonstrated in Figure 13. The stay at peak was 50 μsec. The rise time changed from 10 μsec to 50, 150, 250, 350 and 450 μsec. The distance of crack propagation increased at a decreasing rate and when plotted as a function of rise time (Figure 13) it demonstrated an upper limit which is approached asymptotically. This limit states that for each material there is an optimum rise time that can maximize the crack length, all other explosive characteristics being constant.

Finally, the effect of pressure stay at peak is shown in Figure 14. The rise time was 150 μsec in all three cases, (t_{df} = 50, t_{df} = 150 and t_{df} = 250 μsec). A total increase

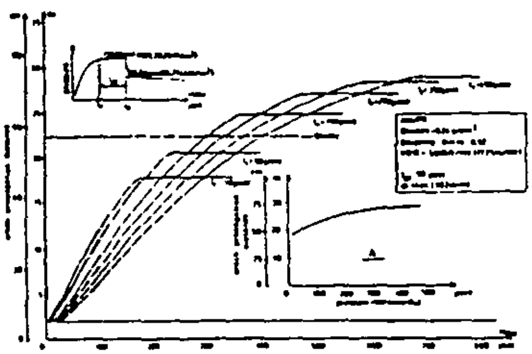

FIGURE 13 - EFFECT OF RISE TIME ON CRACK PROPAGATION DISTANCE

of 200 μsec can cause a 20% increase in crack length from 25 in. (63.5 cm) to 30 in. (76.2 cm). The crack length will increase at a decreasing rate up to a certain point beyond which further increase in stay at peak will not have any effect due to the barrier imposed by geometric damping.

Comparison to Field Results

The most common diameters in presplitting are 2, 3 and 4 in., or 51, 76 and 102 mm. In addition to these a 10-5/8-in. (270 mm) diameter was considered in the analysis to disclose the spacing-diameter relationship. The crack lengths associated with these diameters are plotted in Figure 1 .

The following table shows the ultimate crack propagation distance from the borehole center versus hole diameter.

Hole Diameter				
(in.)	2	3	4	10-5/8
(mm)	51	76	102	270
Crack Propagation Distance				
(in.)	15	20.5	26	42.4
(cm)	38.1	52.1	66.0	107.7

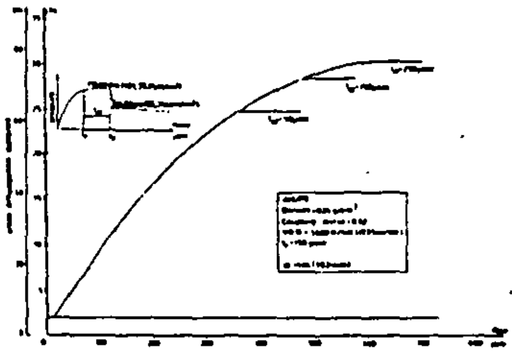

FIGURE 14 - EFFECT OF PRESSURE DURATION ON CRACK PROPAGATION DISTANCE

The results are plotted in Figure 1, comprising field data where hole spacing is correlated to hole diameter for various ratios of compressive to tensile strength. The agreement is very good and it becomes evident that the crack length per inch diameter is longer for smaller hole diameters. It is worth mentioning that the crack length or hole spacing does not depend only on the ratio between compressive and tensile strength as it is implied by the correlation of field data in the above figure, but also on the absolute magnitude of each of them.

Conclusions

The input p-t profile reveals the true behavior and the characteristics of an explosive charge in the same way the stress-strain curve does for a certain material.

This analysis shows clearly that the crack propagation distance depends firstly on the characteristics of the explosive and secondly on the material properties. Control of these characteristics through decoupling or change in sensitivity would make it possible to create the most suitable pressure-time profile for a given ground, in terms of optimum field results.

The explosives studied in this research had low density. However, the explosives generated pressures that were sufficient and of such profile that, when used in the finite element analysis, predicted crack propagation distances of the same magnitude as the ones achieved in the field. In the future, other explosives with similar low densities could be produced and used in presplitting.

The field practice employs explosives of higher density and unknown characteristics. Evaluation of these characteristics on the basis of the present findings will be instrumental in developing ways to modify them and optimize the results further. Increases in hole spacing and reductions in borehole diameter and amount of explosive used should yield considerable savings.

On the other hand the application of advanced rock failure theories and their implementation in finite element codes will make the behavior of rock more predictable and the design of slopes or underground openings safer.

Acknowledgments

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FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

VOLADORAS EN TÚNELES

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

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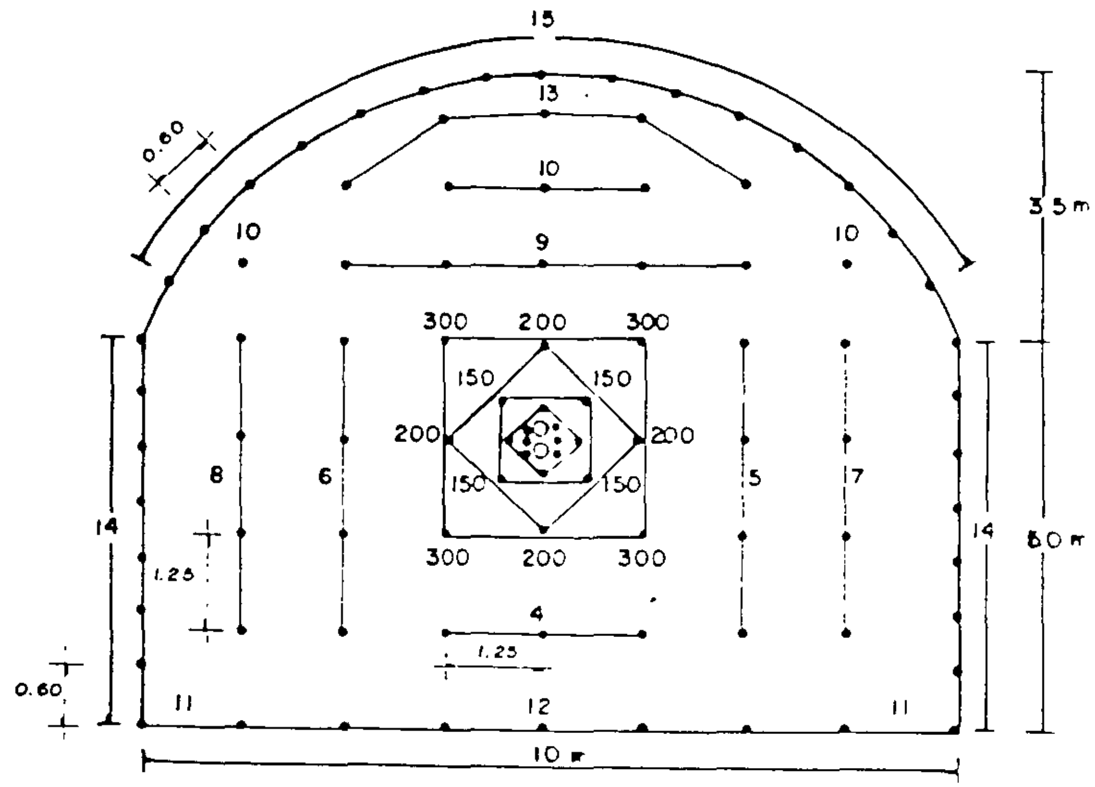
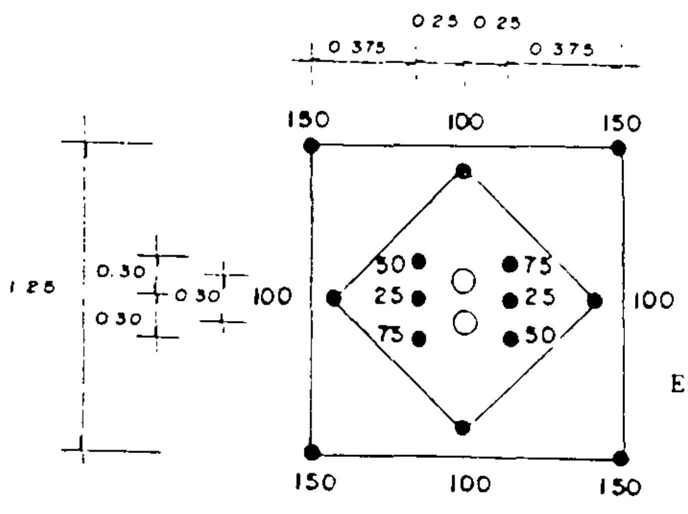
TUNEL ACCESO A CASA DE MAQUINAS

PLANTILLA DE BARRENACION Y CARGA



Acotaciones en metros



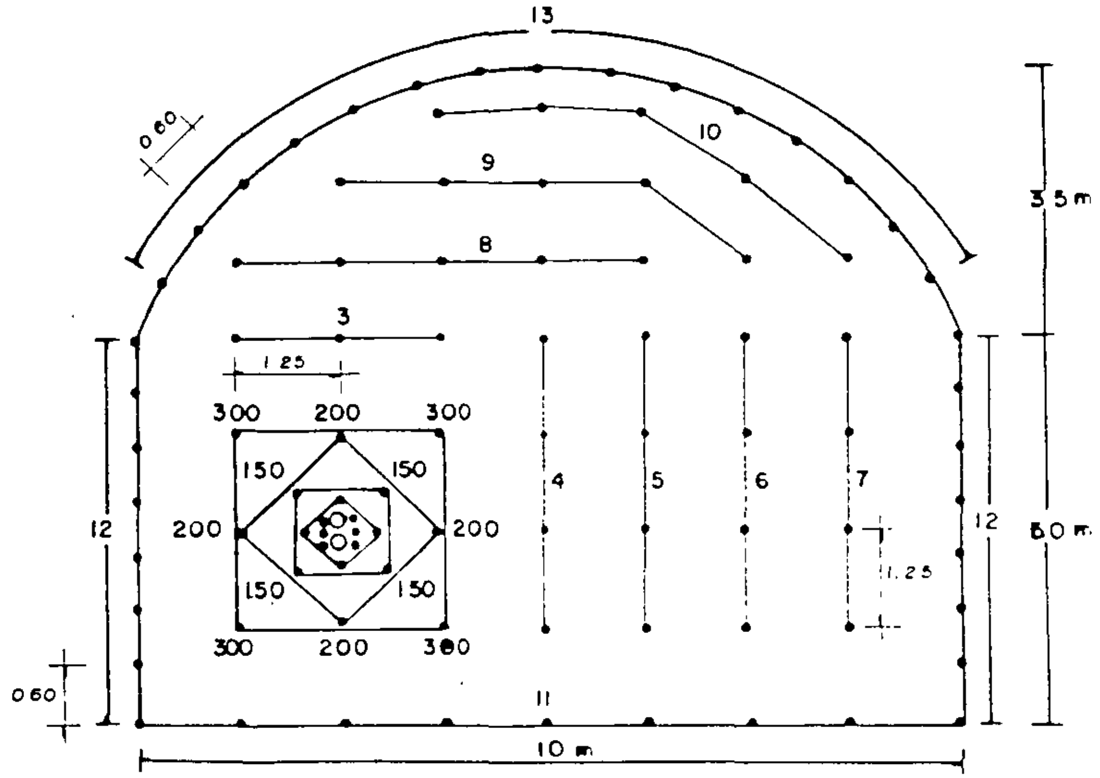
Estopines milisegundos

DETALLE DE CUÑA

FIG. 7

TUNEL ACCESO A CASA DE MAQUINAS
BIFURCACION

PLANTILLA DE BARRENACION Y CARGA



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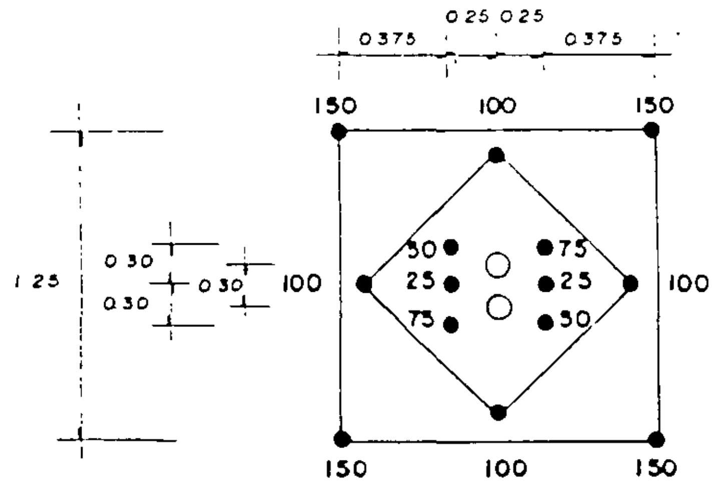


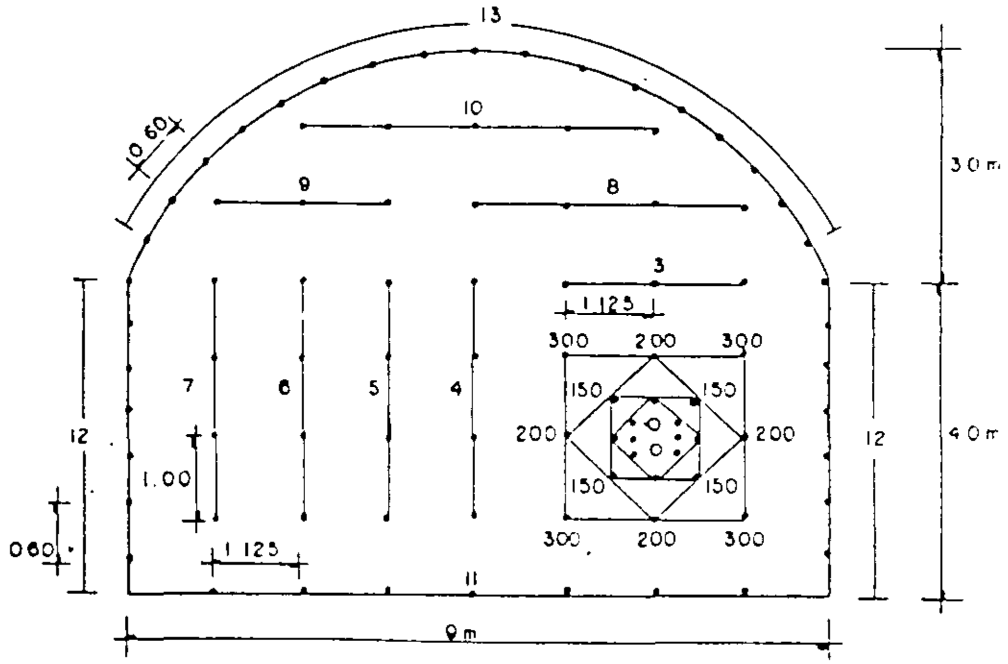
Estopines milisegundos

DETALLE DE CUÑA

FIG. 8

TUNEL AUXILIAR A TUBERIAS A PRESION
BIFURCACION

PLANTILLA DE BARRENACION Y CARGA



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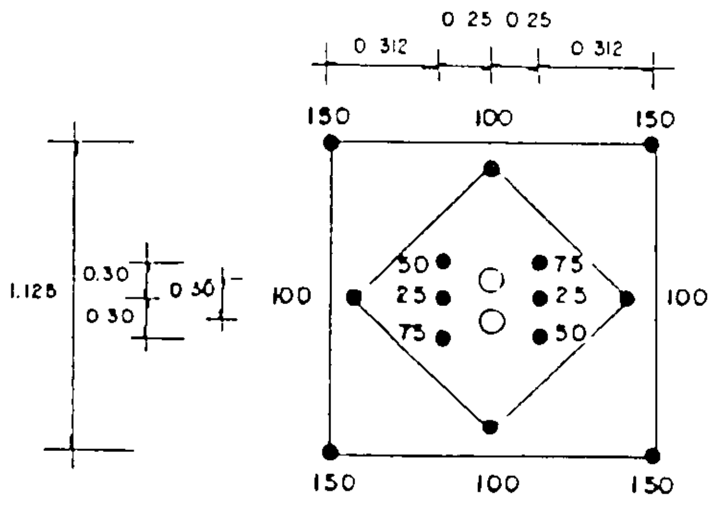


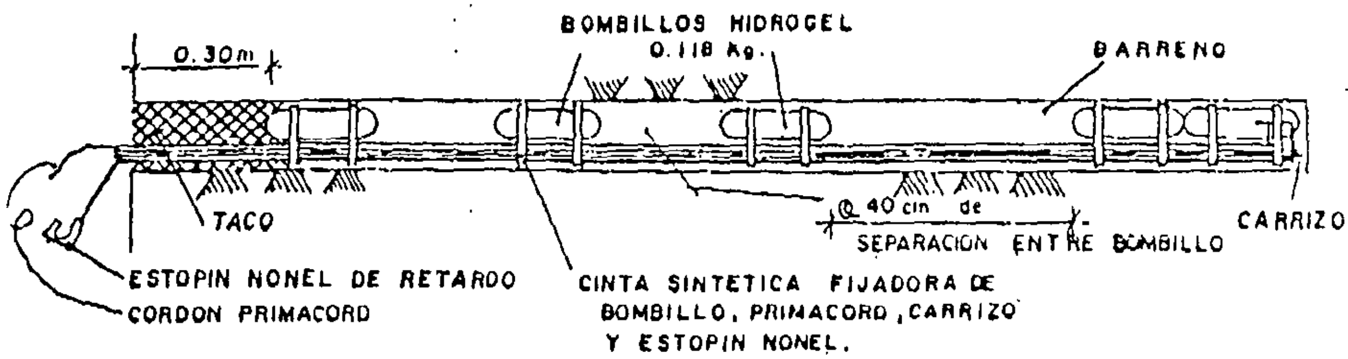
Estopines milisegundos

DETALLE DE CUÑA

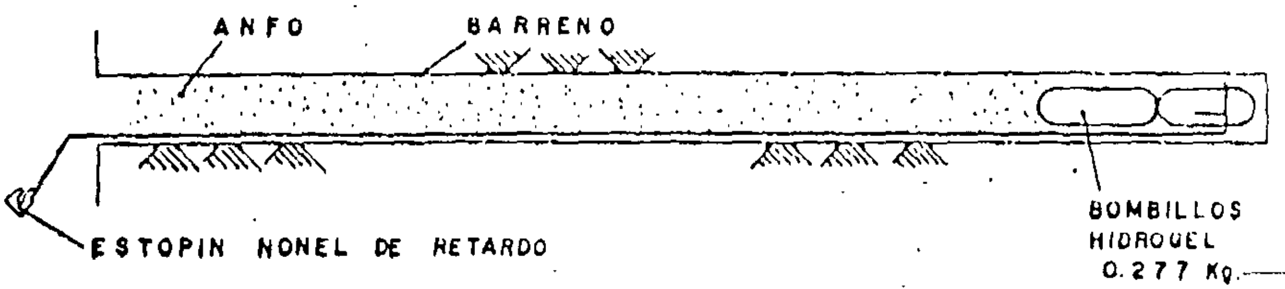
FIG. 9

- DETALLE BARRENO DE POSTCORTE

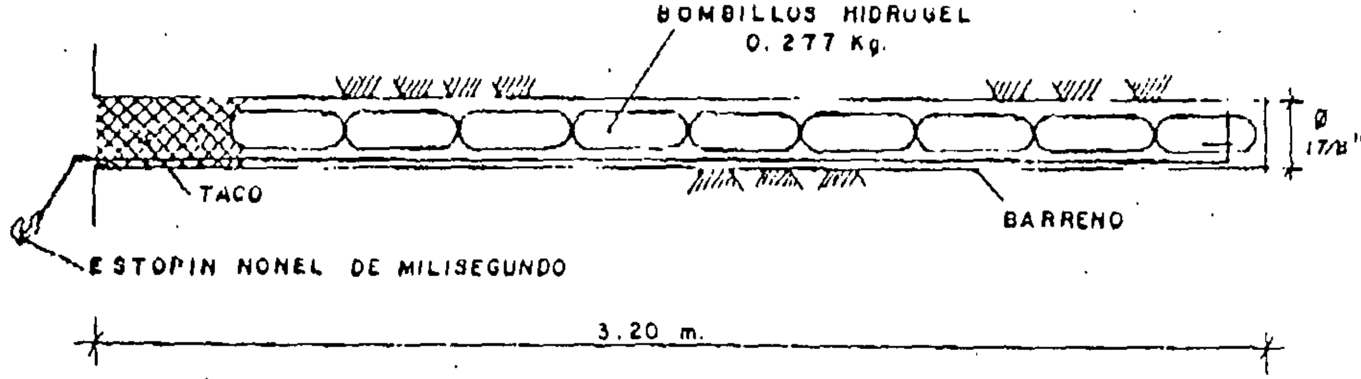
Para: Ing. Cuellar
De: Fidencio



- DETALLE BARRENO DE CUÑA Y TODOS LOS AYTES



- DETALLE BARRENO DE PISO



FACTOR DE CARGA = 1.1 Kg/m^3

FIG. 4

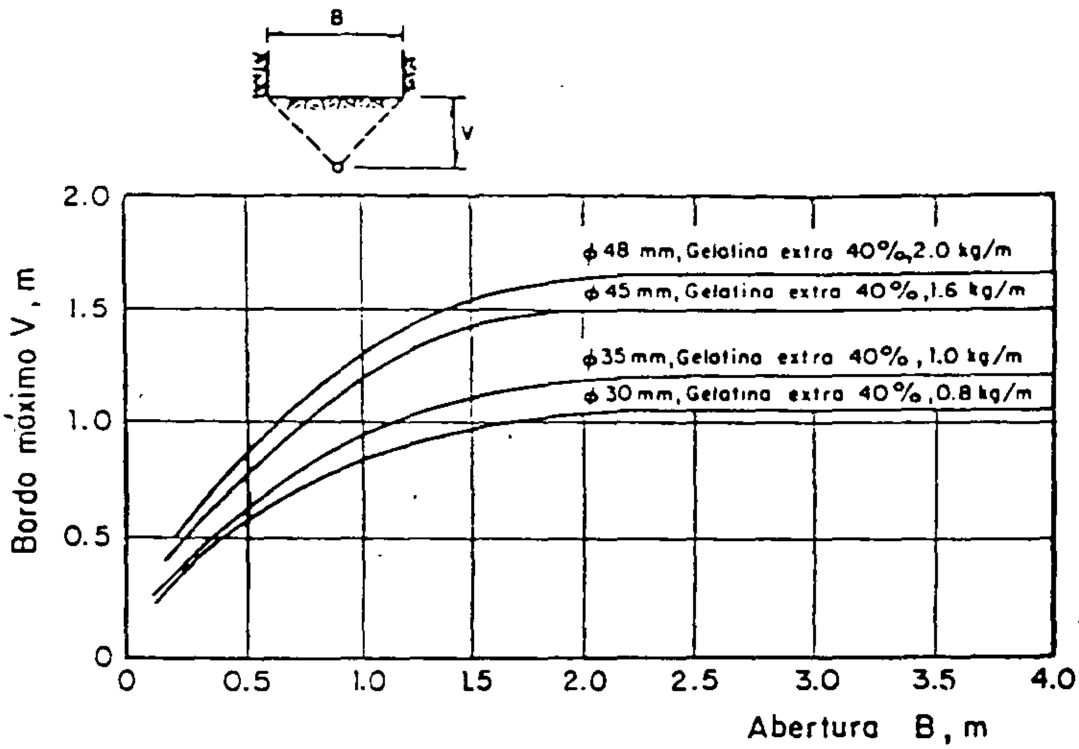


FIG I.34 Relación entre abertura, B, concentración de carga y bordo máximo, V

ciamientos de los barrenos de cada una de las zonas del túnel que se señalan en la fig I.38.

-Barrenos ayudantes con proyección horizontal o hacia arriba

El bordo o distancia entre los barrenos y la cavidad central no debe ser mayor que la mitad de la profundidad del barreno menos veinte centímetros. No deberá tomarse esta condición como base para el cálculo.

El espaciamiento de los barrenos debe ser igual a 1.1 veces el bordo.

tad de la concentración de la carga de fondo. La zona de retaque debe ser igual a la mitad del bordo.

TABLA I.12 Carga específica de fondo

Ayts Horiz o arriba

Diámetro de los barrenos, en mm	Carga específica, en kg/m^3
30	1.1
40	1.3
50	1.5

En la tabla I.13 se muestran los espaciamientos calculados de acuerdo con las cargas específicas de fondo necesarias, considerando explosivos de peso volumétrico de 1.3 g/cm^3 y el diámetro de barrenos de la tabla I.12.

Ayts Horiz

TABLA I.13 Espaciamientos y bordos en función de los diámetros de los barrenos

Amit (ABIERTO

Diámetro de barreno, en mm	Area por barreno, en m^2	Bordo, en m	Espaciamiento, en m
32	0.91	0.90	1.00
35	1.00	0.95	1.05
38	1.15	1.00	1.15
45	1.44	1.15	1.25
48	1.57	1.20	1.30*
51	1.71	1.25	1.35*

* Estos espaciamientos son sólo para túneles de gran diámetro; en el caso de áreas menores su magnitud es menor como se muestra en las gráficas de la fig I.34.

Las concentraciones y cargas de fondo y de columna de la tabla I.14 han sido calculadas a partir de las recomendaciones anteriores, en función del diámetro de los barrenos. Estos datos han sido obtenidos de la práctica e incluyen los errores normales de perforación.

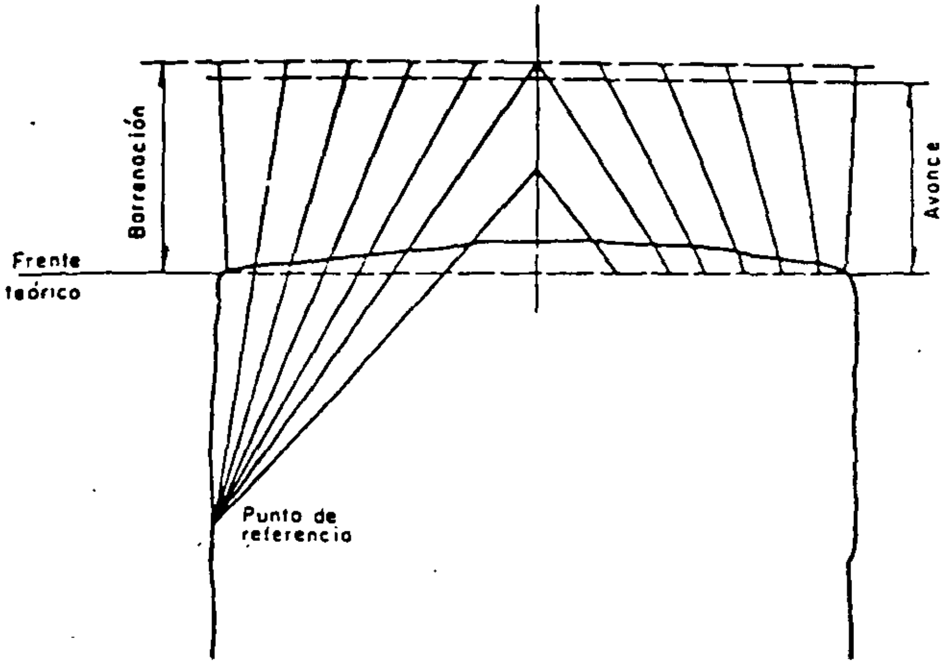


FIG I.35 Distribución en planta de los barrenos de la cuña y los de fuera de la cuña

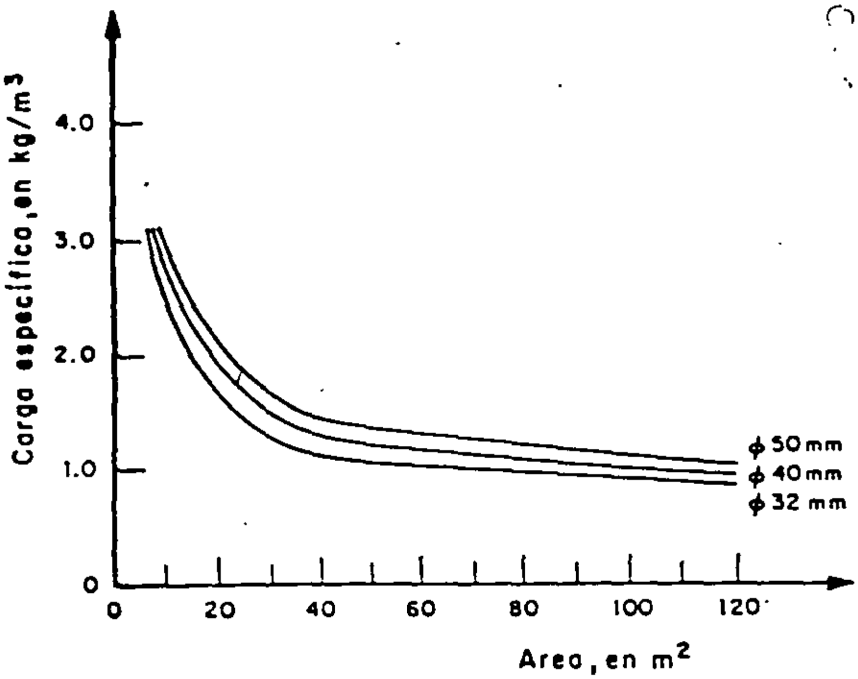


FIG I.36 Cargas específicas utilizadas normalmente en túneles

29

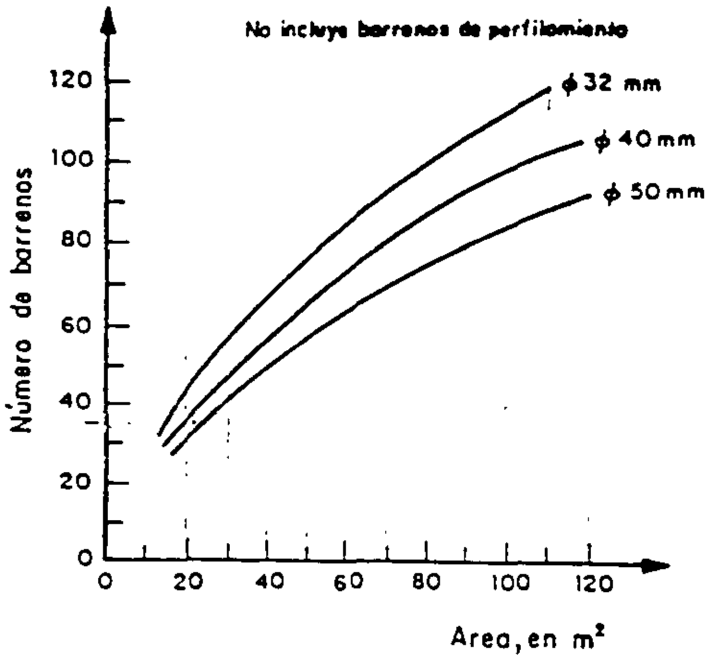


FIG I.37 Número de barrenos en función del área del frente

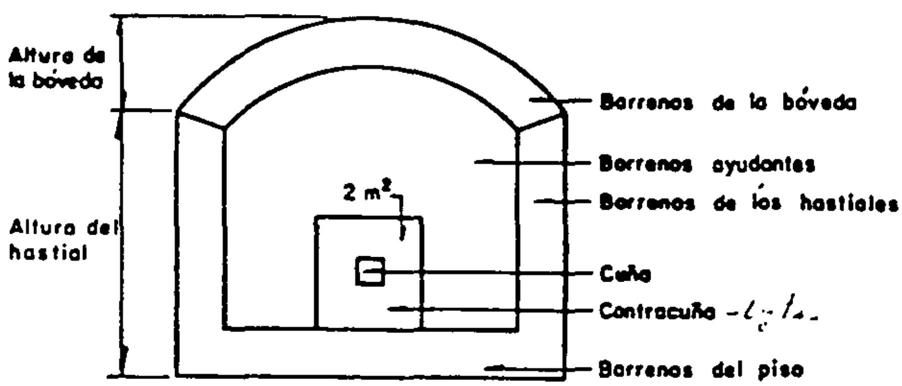


FIG I.38 Zonas de distribución de los barrenos

La carga de fondo ocupa el tercio inferior del barreno con la carga específica de la tabla I.12.

La concentración de la carga de columna en kg/m puede tomarse igual a la mi

30

The relationship can also be used for the "cut spreader" holes above the holes in the cut, the width of the free surface corresponding to the diameter of the large hole as follows

$V = 0.7 \times B$

The loosening section should be so wide that the stoping holes have the possibility to break out at the right angle which implies $2 \times V_{\text{stoping holes}}$.

The burden for the holes in the cut must not be confused with the centre-to-centre distance normally used. The table below can serve as a guide:

CUÑA

Large hole diameter	Small hole diameter	Burden	Centre-to-centre distance
mm	mm	mm	mm
57	32	40	85
76	32	53	107
76	45	53	113
2 × 57	32	80	125
2 × 57	45	80	131
2 × 76	32	106	160
2 × 76	45	106	167
100	45	70	143
100	51	70	146
125	51	88	176

In the case of easily blasted rock, the centre-to-centre distance may need to be increased.

CUÑA

Experience shows that the nearest holes in the cut can be charged as follows:

Drill hole diameter	Charge concentration	Suitable large hole diameter
mm	kg/m	mm
32	0.25[1]	57 – 2 × 76
35	0.30[1]	76 – 2 × 76
38	0.36[1]	76 – 2 × 76
45	0.45	2 × 76 – 125
48	0.55	2 × 76 – 125
51	0.55	2 × 76 – 125

[1] 25 mm Donarit I can normally be used in spite of the fact that it corresponds to a Gelatine Donarit I charge of 0.46 kg/m.

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A similar form of cut with double large holes. diameter 76 mm. has been used a great deal:

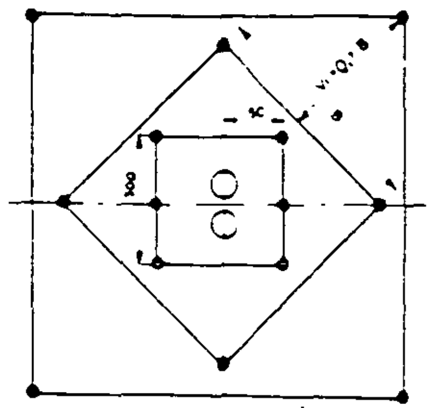

Fig. 9.2.3

The figure also shows the make-up of the "cut spreader" holes outside the cut. The charge in the "cut spreader" holes is large because of the great constriction.

AYTES CUÑA

Burden	Bottom charge	Column charge. kg m with diam., mm			
		32	38	45	48
m	kg				
0 20	0.25	0.30	0.45	0.60	0.75
0.30	0.40	0.30	0.45	0.60	0.75
0 40	0.50	0.35	0.50	0.70	0.80
0.50	0.65	0.50	0.70	1.00	1.15
0 60	0.80	0.50	0.70	1.00	1.15
0.70	0.90	0.50	0.70	1.00	1.15

Uncharged section = 0.5 × V.

Loosening holes with burden greater than 0.70 m are charged in the same way as stoping holes with horizontal breakage (see section 9.1 entitled Charge calculations).

Fig. 9 2.4 shows a cut made up of large gauge holes and the cut spreader section for various drill hole diameters. The outermost holes in the cut spreader section can be described as stoping holes but they have been adapted to some extent geometrically so that they fit into the picture more easily.

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SALIDA HACIA ABAJO

Drill hole diam.	Depth of hole	Burden	Hole spacing	Bottom charge		Column charge		Uncharged section
mm	m	m	m	kg	kg/m	kg	kg/m	m
33	1.6	0 60	0.70	0 60	1.10	0 30	0 40	0.30
32	2.4	0.90	1.10	0 80	1.00	0.55	0.50	0.45
31	3 2	0.85[1]	1 10	1.00	0.95	0 85	0.50	0.45
38	2.4	0.90[1]	1.20	1 15	1.44	0.80	0.70	0.50
37	3.2	1.00[1]	1.20	1.50	1.36	1.15	0 70	0.50
45	3.2	1.15[1]	1.40	2.25	2.03	1.50	1.00	0.55
48	3.2	1.20[1]	1.45	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20[1]	1.45	3 00	2.30	2.45	1.15	0 60
51	3.2	1.25[1]	1.50	2.70	2 60	1.95	1 30	0 60
51	4.0	1.25[1]	1.50	3.40	2 60	2.70	1.30	0.60

[1] In tunnels with a cross-sectional area of more than 70 m^2, the burden and hole spacing can often be increased considerably since the drill holes break much more easily. Blasting then becomes similar to bench blasting.

In most cases the burden can be increased by 10% so that the hole spacing is also considerably greater.

The spacing of the stoping holes can be increased to larger areas with respect to the cross-sectional area of the tunnel. It can also be said that in many cases where rock is easy to blast, the hole spacing shown in the table may be too close. In practice a lower charge concentration in the bottom section is often attained than that shown in the table. This implies that in the case of easily blasted rock, the hole spacing shown in the table can be used even if the charge concentration is lower.

Calculating the charge in wall holes

Normally the walls and roof of the tunnel are smooth-blasted (see section 9.5 entitled Smooth blasting). This calculation concerns cases where no smooth blasting is carried out.

The burden including "look-in" or "look-out" is selected as being 0.9 × the burden for the stoping holes.

Hole spacing = 1.2 × V.

The height of the bottom charge is reduced to 1/6 × depth of hole.

Uncharged section = 0.5 × burden. The concentration of the column charge is reduced to 0.40 × the concentration of the bottom charge.

DE PARED

Example

Drill hole diam	Depth of hole	Burden	Hole spacing	Bottom charge		Column charge		Uncharged section
mm	m	m	m	kg	kg m	kg	kg m	m
33	1.6	0.55	0.65	0.30	1.10	0.45	0.45	0.30
32	2.4	0.80	0.95	0.40	1.00	0.65	0.40	0.40
31	3.2	0.80	0.95	0.50	0.95	0.90	0.40	0.40
38	2.4	0.90	1.10	0.60	1.44	0.85	0.60	0.45
37	3.2	0.90	1.10	0.75	1.36	1.20	0.55	0.45
45	3.2	1.00	1.20	1.10	2.03	1.80	0.80	0.50
48	3.2	1.10	1.30	1.20	2.30	2.00	0.90	0.55
48	4.0	1.10	1.30	1.50	2.30	2.50	0.90	0.55
51	3.2	1.15	1.40	1.40	2.60	2.10	1.00	0.60
51	4.0	1.15	1.40	1.70	2.60	2.70	1.00	0.60

TECHO

Calculating the charge in roof holes

The hole spacing is carried out as for the wall holes. The column charge is reduced to 0.30 × the concentration of the bottom charge.

Drill hole diam.	Depth of hole	Burden	Hole spacing	Bottom charge		Column charge		Uncharged section
mm	m	m	m	kg	kg/m	kg	kg/m	m
31	1.6	0.55	0.65	0.30	1.10	0.35	0.35	0.30
32	2.4	0.80	0.95	0.40	1.00	0.50	0.30	0.40
31	3.2	0.80	0.95	0.50	0.95	0.70	0.30	0.40
38	2.4	0.90	1.10	0.60	1.44	0.70	0.45	0.45
37	3.2	0.90	1.10	0.75	1.36	0.90	0.40	0.45
45	3.2	1.00	1.20	1.10	2.03	1.30	0.60	0.50
48	3.2	1.10	1.30	1.20	2.30	1.45	0.70	0.55
48	4.0	1.10	1.30	1.50	2.30	1.95	0.90	0.55
51	3.2	1.15	1.40	1.40	2.60	1.70	0.80	0.60
51	4.0	1.15	1.40	1.70	2.60	2.25	0.80	0.60

DE PISO

Example.

Drill diam.	Drilling depth	Burden	Hole spacing	Bottom charge		Column charge		Uncharged section
mm	m	m	m	kg	kg/m	kg	kg/m	m
33	1.6	0 60	0.70	0 60	1.10	0.70	0.75	0.10
32	2 4	0.90	1.00	0.80	1.00	1.00	0.70	0.20
31	3.2	0.90	0.95	1 00	0.95	1.30	0.65	0.20
38	2.4	1.00	1.10	1.15	1.44	1.40	1.00	0.20
37	3.2	1.00	1.10	1.50	1.36	1.80	0.95	0.20
45	3.2	1.15	1.25	2.25	2 03	2.60	1 40	0.25
48	3 2	1.20	1.30	2.50	2.30	3.00	1.60	0.25
48	4.0	1 20	1.30	3 00	2 30	4.25	1.60	0.25
51	3 2	1.25	1.35	2.70	2.60	3.20	1.80	0.25
51	4.0	1.25	1.35	3.40	2.60	4.75	1.80	0.25

Calculating the charge in stoping holes with breakage downwards

Since these drill holes require less force for swelling and furthermore are helped by the force of gravity, the specific charge in the bottom section can be reduced to.

Drill hole diameter	Specific charge
mm	kg/m^3
30	1.0
40	1.2
50	1.4

Hole spacing can be increased to 1.2 × burden. Otherwise the calculation is made up in the same way as for the stoping holes described earlier. In the case of tunnels with small cross-section areas, burden and hole spacing are decreased according to the geometrical conditions.

The hole spacing indicated in the table applies on condition that the charge concentration in the bottom section attains the value in the table. If the charging method used results in a lower concentration, then the holes must be more closely spaced so that the required specific charge is attained.

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SALIDA HORIZ. Y HACIA ARRIBA

The concentration and strength of the bottom charge and the column charge can be calculated from the relationship mentioned earlier:

Drill hole diam.	Depth of hole	Burden	Hole spacing	Bottom charge		Column charge		Uncharged section
mm	m	m	m	kg	kg/m	kg	kg/m	m
33	1.6	0.60	0.70	0.60	1.10	0.30	0.40	0.20
32	2.4	0.90	1.00	0.80	1.00	0.55	0.50	0.45
31	3.2	0.90	0.95	1.00	0.95	0.85	0.50	0.45
38	2.4	1.00	1.10	1.15	1.44	0.80	0.70	0.50
37	3.2	1.00	1.10	1.50	1.36	1.15	0.70	0.50
45	3.2	1.15	1.25	2.25	2.03	1.50	1.00	0.55
48	3.2	1.20	1.30	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20	1.30	3.00	2.30	2.45	1.15	0.60
51	3.2	1.25	1.35	2.70	2.60	1.95	1.30	0.60
51	4.0	1.25	1.35	3.40	2.60	2.70	1.30	0.60

33—38 mm covers the range for both drill series 11 and 12 and also full-length drill rods with 33 and 38 mm bits respectively.

Burden and hole spacing are those used in practice – faulty drilling is included in the basic calculation for tunnel blasting.

The table shows that faulty drilling and swelling requirements are compensated for by larger bottom charges as hole depth is increased. Full utilization of the largest diameter holes implies large charges per hole which, from the viewpoint of rock technology, is unfavourable.

Calculating the charge in the floor holes

The burden and spacing for the floor holes can be calculated in the same way as for the stoping holes mentioned above. However, it is important for the "look-in" or "look-out" to be included in the burden dimensions. Since "look-out" is included in the burden, the drill holes close to the floor must be collared for burden and "look-out". For example with a burden of 1.00 m and "look-out" of 0.20 m, the holes in the round must be collared 1.00 – 0.20 m = 0.80 m above the floor hole collaring point. The uncharged section is taken to be 0.2 × burden. The column charge concentration is increased to 70 % of the bottom charge.

TABLA I.14 Cargas, espaciamientos y bordos en barrenos ayudantes con proyección horizontal o hacia arriba

Taco

Diámetro barreno. mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.30	0.40	0.30
32	2.4	0.90	1.00	0.80	1.00	0.55	0.50	0.45
31	3.2	0.90	0.95	1.00	0.95	0.85	0.50	0.45
38	2.4	1.00	1.10	1.15	1.44	0.80	0.70	0.50
37	3.2	1.00	1.10	1.50	1.36	1.15	0.70	0.50
45	3.2	1.15	1.25	2.25	2.03	1.50	1.00	0.55
48	3.2	1.20	1.30	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20	1.30	3.00	2.30	2.45	1.15	0.60
51	3.2	1.25	1.35	2.50	2.60	1.95	1.30	0.60
51	4.0	1.25	1.35	3.40	2.60	2.70	1.30	0.60

-Barrenos de piso

El bordo y el espaciamiento de estos barrenos debe calcularse del mismo modo que los barrenos ayudantes. Sin embargo, debe considerarse en el bordo una corrección debido al emboquille de preparación para la voladura siguiente. Por ejemplo, con un bordo de 1.00 m y un margen para emboquille de 0.20 m, la segunda fila de barrenos del piso debe estar 0.80 m arriba de la entrada de los barrenos de la primera fila. La zona de retaque debe ser de 0.20 veces el bordo, es decir, mucho menor que en los barrenos ayudantes y la concentración de la carga de columna se fija hasta de un 70 por ciento de la concentración de la carga de fondo.

En la tabla I.15 se presentan las concentraciones de carga de fondo y de columna, el espaciamiento, el bordo y la zona de retaque para distintos diámetros de barrenos.

-Barrenos ayudantes con proyección hacia abajo

Debido a la ayuda de la gravedad, estos barrenos requieren una menor carga específica que los anteriores. La carga específica de fondo puede ser la de la tabla I.16.

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TABLA I.15 Cargas, espaciamientos y bordos en barrenos de piso

Diámetro barreno mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.70	0.75	0.10
32	2.4	0.90	1.00	0.80	1.00	1.00	0.70	0.20
31	3.2	0.90	0.95	1.00	0.95	1.30	0.65	0.20
38	2.4	1.00	1.10	1.15	1.44	1.40	1.00	0.20
37	3.2	1.00	1.10	1.50	1.36	1.80	0.95	0.20
45	3.2	1.15	1.25	2.25	2.03	2.60	1.40	0.25
48	3.2	1.20	1.30	2.50	2.30	3.00	1.60	0.25
48	4.0	1.20	1.30	3.00	2.30	4.25	1.60	0.25
51	3.2	1.25	1.35	2.70	2.60	3.20	1.80	0.25
51	4.0	1.25	1.35	3.40	2.60	4.75	1.80	0.25

TABLA I.16 Carga específica de fondo

Diámetro de los barrenos, en mm	Carga específica, en kg/m³
30	1.0
40	1.2
50	1.4

El espaciamiento de estos barrenos puede ser de 1.2 veces el bordo. Las demás características son las señaladas para los otros barrenos ayudantes.

En túneles de sección trasversal pequeña las cargas deberán aumentarse y el bordo y el espaciamiento disminuirse de acuerdo con las funciones de las gráficas que se presentan en las figs I.34, I.36 y I.37.

En la tabla I.17 se presentan las cargas, bordos y espaciamientos de estos barrenos. Los espaciamientos indicados son aplicables siempre que la con-

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centración de carga en el fondo alcance, asimismo, el valor señalado. Si la concentración de carga resulta menor, el espaciamiento deberá reducirse para obtener la carga específica requerida.

Los valores de espaciamientos y bordos indicados en la tabla I.17 pueden aumentarse, particularmente cuando la roca es fácil de excavar y cuando los túneles tienen un área de más de 70 m^2. También es frecuente en estos casos utilizar los espaciamientos señalados pero con menores concentraciones de carga.

TABLA I.17 Cargas, espaciamientos y bordos en barrenos ayudantes con proyección hacia abajo

Diámetro barreno, mm	Profundidad barreno, m	Bordo, m	Espaciamiento, m	Carga de fondo		Carga de columna		Zona de retaque, m
				kg	kg/m	kg	kg/m	
33	1.6	0.60	0.70	0.60	1.10	0.30	0.40	0.30
32	2.4	0.90	1.10	0.80	1.00	0.55	0.50	0.45
31	3.2	0.85	1.10	1.00	0.95	0.85	0.50	0.45
38	2.4	1.00	1.20	1.15	1.44	0.80	0.70	0.50
37	3.2	1.00	1.20	1.50	1.36	1.15	0.70	0.50
45	3.2	1.15	1.40	2.25	2.03	1.50	1.25	0.55
48	3.2	1.20	1.45	2.50	2.30	1.70	1.15	0.60
48	4.0	1.20	1.45	3.00	2.30	2.45	1.15	0.60
51	3.2	1.25	1.50	2.70	2.60	1.95	1.30	0.60
51	4.0	1.25	1.50	3.40	2.60	2.70	1.30	0.60

-Barrenos de los hastiales

Las voladuras de los hastiales y de la bóveda corresponden por lo común al tipo de voladuras denominado recorte o poscorte perimetral (inciso 7.2.1.5). En esta sección se tratan los casos que no son voladuras de recorte.

El bordo, considerando el emboquille de preparación para la voladura siguiente, se toma igual a 0.90 veces el bordo de los barrenos ayudantes.

El espaciamiento que mejores resultados ha aportado en la práctica es 1.2 veces el bordo; la longitud de la carga de fondo un sexto de la profundidad del barreno; la zona de retaque la mitad del bordo; y la concentración de la carga de columna de 0.40 veces la carga de fondo. La tabla I.18 está elaborada con las especificaciones anteriores.

TABLA I.18 Cargas, espaciamientos y bordos en barrenos de los hastiales

Diámetro barreno mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.55	0.65	0.30	1.10	0.45	0.45	0.30
32	2.4	0.80	0.95	0.40	1.00	0.65	0.40	0.40
31	3.2	0.80	0.95	0.50	0.95	0.90	0.40	0.40
38	2.4	0.90	1.10	0.60	1.44	0.85	0.60	0.45
37	3.2	0.90	1.10	0.75	1.36	1.20	0.55	0.45
45	3.2	1.00	1.20	1.10	2.03	1.80	0.80	0.50
48	3.2	1.10	1.30	1.20	2.30	2.00	0.90	0.55
48	4.0	1.10	1.30	1.50	2.30	2.50	0.90	0.55
51	3.2	1.15	1.40	1.40	2.60	2.10	1.00	0.60
51	4.0	1.15	1.40	1.70	2.60	2.70	1.00	0.60

-Barrenos de la bóveda (tabla I.19)

En estos barrenos la carga de columna se reduce a 0.30 veces la concentración de la carga de fondo. Las demás características son iguales a las de los barrenos de los hastiales.

b) Resumen de las características de los barrenos que no pertenecen a la cuña

Nomenclatura:

V bordo o separación de la cavidad previamente abierta, en m

V_1 bordo práctico, en m

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H	**profundidad del barreno, en m**
q	**carga específica, en kg/m³**
d	**diámetro del barreno, en mm**
Q_{bk}	concentración de la carga de fondo, en kg/m
Q_{pk}	concentración de la carga de columna, en kg/m
h_b	altura de la carga de fondo, en m
h_o	longitud del retaque, en m
E	Distancia entre barrenos, en m

TABLA I.19 Cargas, espaciamientos y bordos en barrenos de la bóveda

Diámetro barreno mm	Profundidad barreno, m	Bordo m	Espaciamiento m	Carga de fondo		Carga de columna		Zona de retaque m
				kg	kg/m	kg	kg/m	
33	1.6	0.55	0.65	0.30	1.10	0.35	0.35	0.30
32	2.4	0.80	0.95	0.40	1.00	0.50	0.30	0.40
31	3.2	0.80	0.95	0.50	0.95	0.70	0.30	0.40
38	2.4	0.90	1.10	0.60	1.44	0.70	0.45	0.45
37	3.2	0.90	1.10	0.75	1.36	0.90	0.40	0.45
45	3.2	1.00	1.20	1.10	2.03	1.30	0.60	0.50
48	3.2	1.10	1.30	1.20	2.30	1.45	0.80	0.55
48	4.0	1.10	1.30	1.50	2.30	1.95	0.90	0.55
51	3.2	1.15	1.40	1.40	2.60	1.70	0.80	0.60
51	4.0	1.15	1.40	1.70	2.60	2.25	0.80	0.60

-Barrenos ayudantes con proyección horizontal o hacia arriba

d(mm)	q(kg/m³)
30	1.1
40	1.3
50	1.5
h_b	H/3

$$V_1 \leq \frac{H - 0.40\,m}{2}$$ **(ésta es una condición y no es una base de cálculo)** (I.4)

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c) Cuñas de barrenos paralelos

Debe calcularse la separación entre el barreno vacío central y los barrenos cargados de la cuña de manera que el área del barreno vacío sea de cuando menos un 15 por ciento del área de influencia de los barrenos de la cuña que disparan en primer término (inciso 7.2.1.3a, fig I.31). La separación así calculada no debe rebasar la que se muestra en la tabla I.20.

TABLA I.20 Separación entre los barrenos vacíos y cargados de la cuña de barrenos paralelos

Diámetro del barreno central, mm	Diámetro de los barrenos cargados, mm	Bordo o separación entre barrenos, mm	Distancia entre centros, mm
57	32	40	85
76	32	53	107
76	45	53	113
2 x 57*	32	80	125
2 x 57*	45	80	131
2 x 76*	32	106	160
2 x 76*	45	106	167
100	45	70	143
100	51	70	146
125	51	88	176

* Dos barrenos centrales.

Las cargas que se presentan en la tabla I.21 son, en general, adecuadas para los barrenos más próximos al barreno central.

Los barrenos denominados de contracuña, situados fuera de ésta, son adaptados al área de la sección trasversal del túnel.

La carga de los barrenos de la contracuña es muy elevada debido a su gran confinamiento. La fig I.39 muestra la disposición de la contracuña para una cuña de dos barrenos centrales.

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(cuña)

TABLA I.21 Cargas asignadas a los barrenos más próximos al central

Diámetro de los barrenos cargados, mm	Carga asignada (kg/m)	Diámetro del barreno central, mm
32	0.25	de 57 a 2 x 76
35	0.30	de 76 a 2 x 76
38	0.36	de 76 a 2 x 76
→ 45	→ 0.45	de 2 x 76 a 125
48	0.55	de 2 x 76 a 125
51	0.55	de 2 x 76 a 125

En la tabla I.22 se presentan valores de cargas que han dado buenos resultados en barrenos de contracuña.

TABLA I.22 Valores empíricos de carga en barrenos de contracuña (Ayudantes)

Bordo o separación entre barrenos m	Carga de fondo kg	Carga de columna en kg/m para diámetros de los barrenos cargados de:			
		32 mm	38 mm	45 mm	48 mm
0.20	0.25	0.30	0.45	0.60	0.75
0.30	0.40	0.30	0.45	0.60	0.75
0.40	0.50	0.35	0.50	0.70	0.80
→ 0.50	0.65	0.50	0.70	1.00	1.15
0.60	0.80	0.50	0.70	1.00	1.15
0.70	0.90	0.50	0.70	1.00	1.15

Longitud sin carga (taco) = 0.5 V.

d) Cuña en V

En esta sección se proporcionan reglas generales para el cálculo de cargas considerando una cuña de vértice interior de 60°. Si este ángulo es menor la carga debe incrementarse.

La dimensión V de la cuña (fig I.40) es función de la cantidad de explosivos que pueden cargarse en los barrenos con arreglo a su diámetro. En la

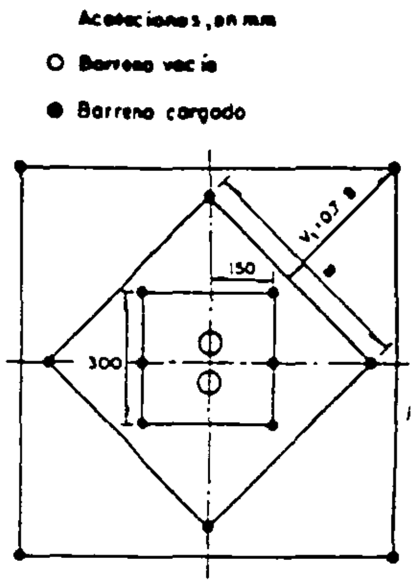

FIG I.39 Cuña de dos barrenos centrales y contracuña

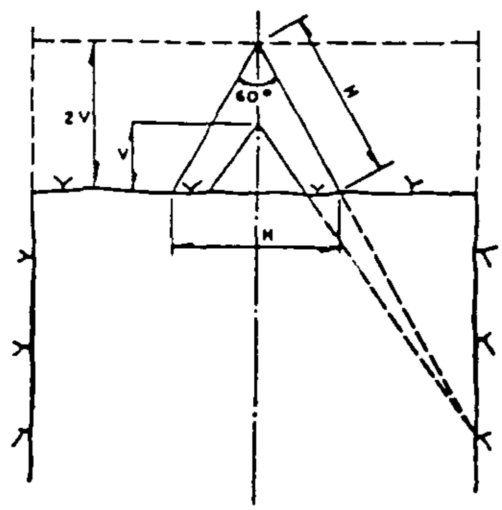

FIG I.40 Cuña en V

tabla I.23 se proporcionan valores que pueden servir de orientación en la determinación de la dimensión y carga de la cuña en V.

En cuñas en V la longitud de la carga de fondo debe ser de cuando menos un tercio de la profundidad del barreno. La carga de columna debe ser igual a la mitad de la carga de fondo. La zona de retaque debe ser un tercio de la dimensión V de la cuña, pero debe ser adaptada al espaciamiento de los barrenos de manera que no haya exceso de carga en la parte de la columna.

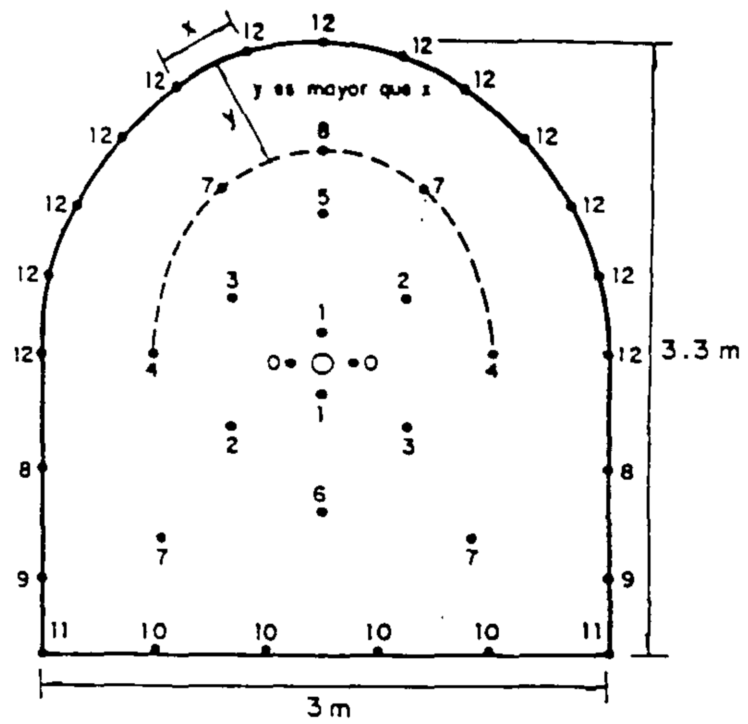

FIG I.41 Distribución típica de retardos en un túnel

que la roca fragmentada ya ha sido desplazada, ofreciéndoles un espacio de alivio suficiente. Este alivio permite una voladura del bordo final con un sacudimiento mínimo.

En la tabla I.24 se proporcionan valores prácticos recomendados de espaciamientos, bordos y concentraciones de carga promedio para dos diámetros de barreno, utilizando explosivos de 1.2 a 1.3 g/cm³ de peso volumétrico.

TABLA I.24 Poscorte perimetral

Diámetro barreno mm	Espaciamiento m	Bordo m	Concentración total de carga en el barreno kg/m
38 - 45	0.60	0.90	0.18 - 0.38
51	0.75	1.05	0.18 - 0.38

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Los cartuchos largos de diámetro pequeño de explosivos de baja densidad, permiten una distribución adecuada de la carga a lo largo del barreno. Los cartuchos de 20 cm de longitud se han empleado con éxito en voladuras de poscorte perimetral utilizando espaciadores entre cartuchos para reducir la carga total en kg/m; sin embargo, este procedimiento da como resultado concentraciones de carga relativamente altas en distintos puntos.

7.2.1.6 Precorte

En el precorte los barrenos de contorno se disparan antes de efectuar la voladura propiamente dicha. El precorte produce una grieta entre los barrenos de contorno. Esta grieta evita que las ondas de choque de la voladura principal se trasmitan en toda su intensidad hacia la pared terminada y minimiza la profundidad de la fragmentación en la roca. Como los barrenos están muy próximos entre sí, las grietas se forman siguiendo las líneas de barrenos, y los mismos barrenos constituyen el inicio del agrietamiento. Esto significa que la inclusión de barrenos vacíos entre los cargados, puede mejorar los resultados.

En la tabla I.25 se indican algunas cargas y espaciamientos en función del diámetro de los barrenos.

Si no existen limitaciones en las vibraciones del terreno se utiliza el encendido instantáneo; por lo contrario, si es necesario limitar la magnitud de las vibraciones del terreno se utilizan microretardos. La formación de grietas resulta menos eficiente que con la iniciación instantánea, a menos que se reduzca el espacio entre barrenos. Si el tiempo de retardo es muy grande no se logra el precorte.

TABLA I.25 Precorte

Diámetro del barreno mm	Espaciamiento m	Concentración de carga kg/m
25 - 32	0.20 - 0.30	0.08
25 - 32	0.35 - 0.60	0.18
40	0.35 - 0.50	0.18
51	0.40 - 0.50	0.36
64	0.60 - 0.80	0.38

TABLA 7.2

Concentración de la carga, en kg/m, para diversas piedras *(V) y extension (B) de la cara libre (tabla preliminar).*

Piedra max *V* m	B	Concentracion de la carga, kg/m *l*										Carga limite *l*, kg/m con *piedra* libre
		0,10 m *0,3*	0,15 *0,5*	0,20 *0,7*	0,25 *0,8*	0,30 *1*	0,35 *1,2*	0,40 *1,3*	0,50 *1,7*	0,60 *2*	0,80 *2,7*	1,4 m *4,7 ft*	
						kg/m							
0,10		0,12	0,08	0,06									
0,15		0,30	0,18	0,13	0,11	0,09							
0,20		0,60	0,35	0,24	0,20	0,16	0,14	0,12					
0,25		1,0	0,60	0,35	0,30	0,26	22	0,18					
0,30		1,3	0,9	0,60	0,50	0,35	..31	0,26	0,22	0,18			
0,35			1,2	0,9	0,65	0,45	0,40	0,35	0,30	0,25			
0,40			1,6	1,2	0,9	0,7	0,6	0,50	0,40	0,30	0,24		
0,50				2,0	1,6	1,3	1,0	0,7	0,60	0,50	0,36		0,13
0,60					2,2	1,9	1,6	1,3	1,0	0,7	0,52		0,17
0,70						2,5	2,2	1,8	1,3	0,9	0,7		0,25
0,80							3,2	2,4	1,8	1,4	1,0	0,6	0,32
1,00								4,0	3,0	2,4	1,4	0,9	0,5
1,20									4,4	3,8	2,5	1,2	0,7
1,40										5,0	3,6	1,6	0,9
1,60											4,8	2,4	1,1
2,00												4,0	1,9

Las cifras en lbs/pie son 2/3 de las cifras para kg/m.

en la que los subíndices *a* y *b* se refieren a las figuras 7.5 *a* y *b* respectivamente. La relación (7.4) se estudiará en el próximo capítulo; la relación (7.5) se da en la tabla 7.2.

Estas ecuaciones difieren solamente en el valor de la constante, que en las voladuras con una salida circular es un 60 % mayor que con una rectangular, debido a la mayor constricción en el primer caso.

Las anteriores relaciones abarcan los datos experimentales actuales, pero las voladuras con una salida rectangular deben estar sujetas a ulteriores consideraciones teóricas. Sin embargo, la parte experimental de la investigación ha alcanzado una etapa que suministra bases para la discusión de algunos de los principales puntos involucrados: la determinación del diámetro óptimo de los barrenos no cargados en las voladuras de cuele paralelo, la construcción de los esquemas de perforación y el cálculo de la carga con miras a la colocación de los taladros y a la dispersión real de la perforación.

Las cargas dadas en la tabla son suficientes para la rotura pero no comprenden necesariamente los valores límites; muchos de los valores

TABLA 8.1

Concentración de la carga (l) en kg/m para cueles cilíndricos y máxima distancia cuando se dispara hacia barrenos vacíos con diámetros comprendidos entre ø = 2 · 57 y 200 mm (d representa el diámetro del barreno cargado). La potencia relativa del explosivo es s = 1,0.

ø mm	50	2 · 57	75	83	100	2 · 75	110	125	150	200
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
à mm	90	150	130	145	175	200	190	220	250	330

Hay que señalar, especialmente con barrenos vacíos de pequeños diámetros, lo mucho que hay que aumentar la carga cuando se incrementa la distancia entre centros. Para ø = 30 mm se necesita una carga de 1,0 kg m para una distancia entre centros de 11 cm y menos de la mitad de la concentración de carga para una distancia de 8 cm. Este es

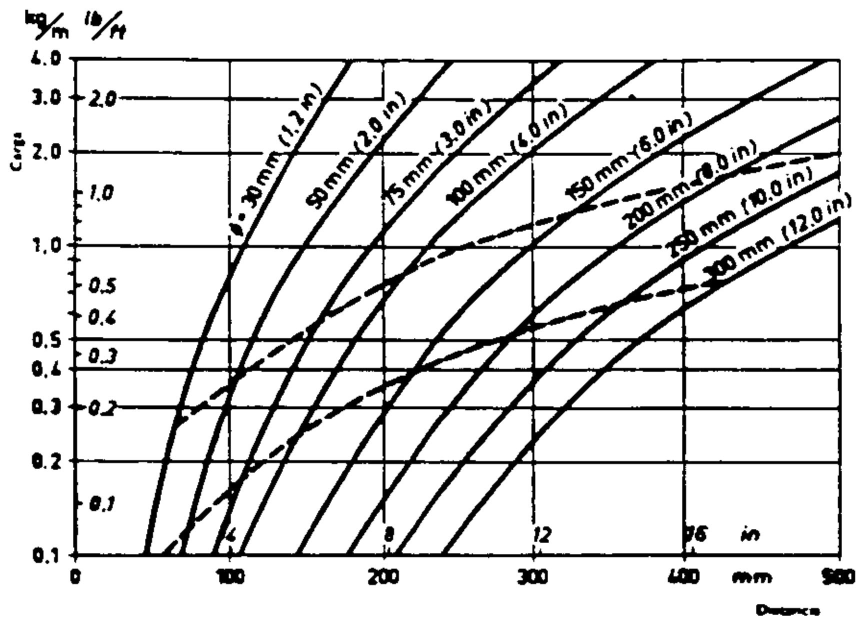


Fig 8.4

Relación entre la cantidad de carga y la distancia entre los barrenos cuando se dispara hacia un barreno vacío con un diámetro de 30-150 mm ---- corresponde a las líneas de puntos de la figura 8 5. Diámetro de los barrenos cargados = 32 mm.

TABLA 8.6

Cueles en doble espiral con diversos diámetros del barreno vacío (ϕ). Datos acordes con la figura 8.10. Las concentraciones de carga l_1 y l_2 se refieren a los barrenos marcados con - y ● respectivamente.

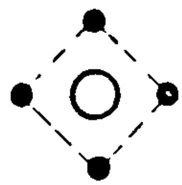
ϕ mm		75	85	100	110	125	150	200
a	mm	110	120	130	140	160	190	250
b	mm	130	140	160	170	190	230	310
c	mm	160	175	195	210	240	290	380
d	mm	270	290	325	350	400		
l_1	kg m	0,30	0,35	0,40	0,45	0,5	0,6	0,8
l_2	kg m	0,65	0,75	0,85	0,9	1,1	1,3	1,7

bajar apreciablemente el avance medio. El encendido de los diferentes barrenos del cuele debe efectuarse según la secuencia dada en las figuras 8.10 y 8.11. Respecto a los cueles de doble espiral, Táby y Coromant, el encendido de los barrenos num. 1 y 2 con detonadores instantáneos puede mejorar los resultados. Estos detonadores instantáneos deben situarse en la boca del barreno. Para evitar el riesgo de decapitación de las otras cargas, los demas detonadores deben colocarse en el fondo de los barrenos. No deberían llevar carga de fondo los 6-8 barrenos más próximos al barreno vacío.

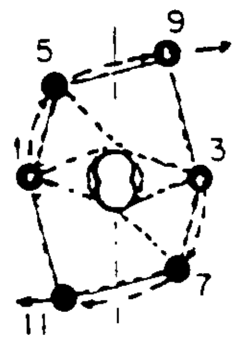
a) *Cueles cilíndricos*

Cueles en doble espiral (fig. 8.10 *a*)

El esquema de barrenos en espiral proporciona la abertura más amplia. Sin embargo, cuando se pretendan obtener grandes avances deberá usarse el doble espiral de la figura 8.10 *a*, adoptando la separación entre barrenos y la concentración de carga de la tabla 8.6. Con el esquema en doble espiral se tiene la ventaja de que pueden iniciarse sucesivamente los barrenos opuestos, con lo que se obtiene una mejor limpieza de la abertura. Además, la seguridad en el avance aumenta, ya que cada sección de la doble espiral puede romper con independencia. Según los datos de avances medios dados en la tabla 8.3, el cuele en doble espiral es definitivamente superior a los demás tipos de cueles, con un avance por lo menos un 20 % mayor que los demás de barrenos paralelos. Una desventaja en la práctica es el hecho de que los dos barrenos más próximos al vacío de 100-110 mm sólo distan 130-140 mm de su centro y en la perforación con estructuras estacionarias y con el equipo actual se precisa una distancia de 160 mm. Esto lleva consigo la necesidad de aumentar el diámetro del barreno vacío hasta 125 mm, con lo que la distancia al centro puede incrementarse hasta 160 mm y, si se desea, aumentar el

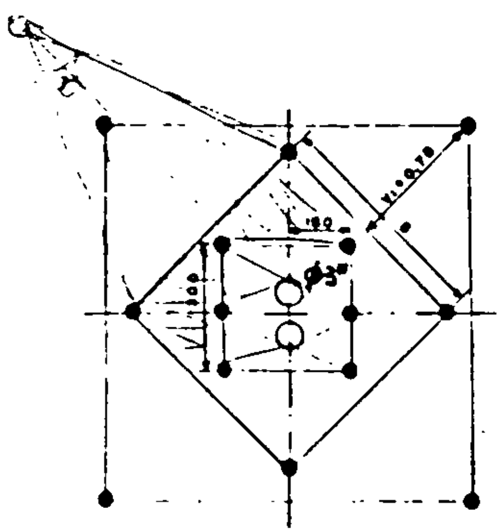


CUÑA "CINCO DE OROS"
CON UN BARRENO DE GRAN DIAMETRO



CUÑA COROMANT
(ADECUADA PARA GALERIA PEQUEÑA

El dispositivo guía se fija a la roca mediante un expansor



CUÑA DE EXPANSION CON DOS BARRENOS QUEMADOS DE GRAN DIAMETRO

CUÑA DE EXPANSION PARA UNO O DOS BARRENOS QUEMADOS

Avance de 3.9 a 4 m

- No se debe trabajar con diámetros grandes en todo el frente del túnel ing rock. These factors are not controlled by any single property of the explosive, but the total energy content is a very useful characteristic by which to rate explosives relative to one another. Table I shows this figure for some explosives. This table also has a column showing characteristic impedance. This is density times velocity of detonation, and its use is discussed later when we discuss the impedance of rocks.

X

Table I

MEASURED ENERGIES

Explosive	Specific Gravity	Shock Energy	Bubble Energy	Total Energy Weight Basis	Total Energy Volume Basis	Detonation Velocity	Characteristic Impedance
	g./cc.	$\frac{\text{Ft.-lb.}}{\text{Lb.}} \times 10^6$	$\frac{\text{Ft.-lb.}}{\text{Lb.}} \times 10^6$	$\frac{\text{Ft.-lb.}}{\text{Lb.}} \times 10^6$	$\frac{\text{Ft.-lb.}}{\text{Ft.}^3}$	Ft. Per Sec. 5-in. Diameter	$\frac{\text{Lb.-sec.}}{\text{In.}^3}$
Ammonium-Nitrate Fuel Oil (AN/FO)	0.80	0.55	0.53	1.08	54	12,500	11
Gelamite® 1	1.28	0.46	0.45	0.91	75	17,700	25
50% NG Dynamite	1.3	0.62	0.61	1.23	103	18,000	26
40% Extra Dynamite	1.3	0.44	0.48	0.92	75	15,800	23
60% Extra Dynamite	1.24	0.48	0.51	0.99	77	17,350	24
Pentolite	1.55	0.56	0.55	1.11	107	24,000	42

When the density of an explosive is relatively high, its grains are closely packed in contact with one another and the shock front of detonation is communicated from grain to grain more efficiently than if the grains are loosely packed to give lower density. The effect of this is shown in Table I, where velocity of detonation is directly related to density (expressed as specific gravity). High velocity of detonation is important in breaking many rocks.

In the blasting of rock, breakage is directly related to the amount of energy transferred from the explosive to the rock. U. S. Bureau of Mines investigators[17] found that within the range of their experiments the amount of energy transferred to a given rock was a linear function of the characteristic impedance of the explosive (see Table I). They concluded that "explosives that had the larger characteristic impedance, or impedance more nearly matching the characteristic impedance of the rock, transferred more energy to the rock."

In this connection, the method of packing explosives into boreholes becomes a factor, since the impedance of both rock and explosive is of the order of 10,000 times that of air and 1,000 times that of water. This very large contrast in impedance causes serious energy losses if there is air or water between the explosive and the rock surrounding the hole.

Delay Caps

Short-period delay caps have been used successfully to reduce vibrations from blasting. Delay detonation separates the pressure fronts and the bundles of energy which they deliver to the rock, so that breaking the rock is done as a series of events that are closely spaced but independent.

Practical result of this technique has been to improve fragmentation and to reduce appreciably the amount of leftover energy that is carried by vibrations to surrounding territory. The greatest amount of energy that reaches surrounding ground and buildings from a delay blast is related to that released by the most explosive on any one of the delay intervals.

Figures 8 and 9 show the effect of millisecond delay firing in reducing elastic waves recorded at a distance of 2,500 ft. from blasts of approximately the same size at one quarry.

Figure 8 Record, no delays

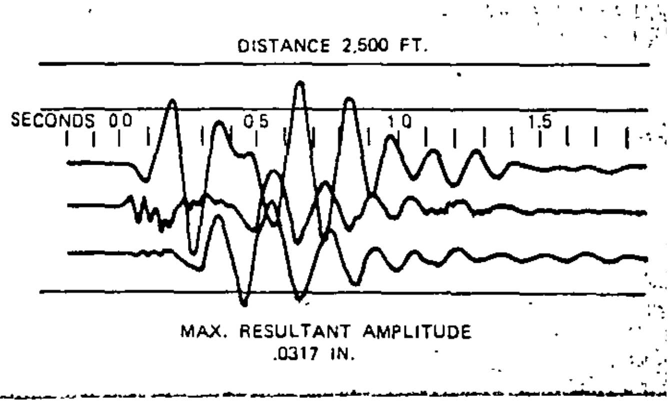
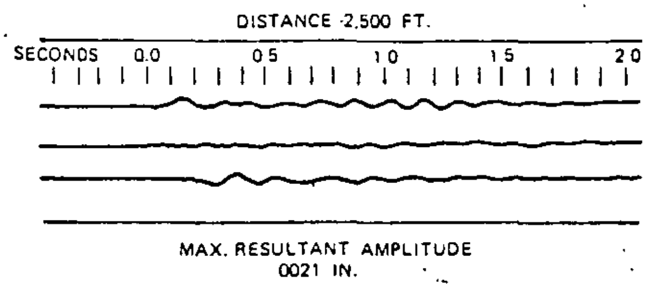


Figure 9 Record, with delays



Rock Characteristics

When explosives are used to break rock, joints often control the pattern of rupture. There have been places where, in spite of hole layout and explosive distribution, breakage has been poor due to its following the pre-blast joint planes. Also, if the dominant joints in a blasting face are steeply inclined, there is a hazard of slides of rock masses bounded by joints and loosened by blasting.

There are two other characteristics of rocks that are important in determining their response to an explosive. These are *elasticity* and *characteristic impedance.*

Elasticity is qualitatively indicated by hardness — the harder, the more elastic. It is measured by the speed, v,

Table II

COMPRESSIONAL WAVE SPEEDS AND CHARACTERISTIC IMPEDANCE FOR CERTAIN ROCKS

Rock	Velocity of Compressional Waves ft./sec.	Characteristic Impedance lb.-sec./in.^{3}
Granite	18,200	54
Marlstone*	11,500	27
Sandstone	10,600	26
Chalk†	9,100	22
Shale	6,400	15

*A hardened mixture of clay materials and calcium carbonate normally containing 25 to 75% clay. A type of limestone.
†A very soft limestone.

FACULTAD DE INGENIERIA U.N.A.M.
DIVISION DE EDUCACION CONTINUA

CURSOS ABIERTOS

TECNOLOGÍA PARA EL USOS DE EXPLOSIVOS

TEMA

NORMATIVIDAD

CONFERENCISTA
ING. RAÚL CUELLAR BORJA
PALACIO DE MINERÍA
MAYO 2000

Palacio de Minería Calle de Tacuba 5 Primer piso Deleg. Cuauhtémoc 06000 México, D.F. APDO. Postal M-2285
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NORMATIVIDAD

POR: JAVIER MARTINEZ LUNA

LEY FEDERAL DE ARMAS DE FUEGO
Y EXPLOSIVOS

La Ley Federal de Armas de Fuego y Explosivos (LEY), fue públicada en el Diario Oficial de la Federación de fecha 25 de enero de 1972. Las disposiciones de esta LEY se consideran de interés público.

La aplicación de la LEY corresponde a:

1. *El Presidente de la República;*
2. *La Secretaría de Gobernación;*
3. *La Secretaría de la Defensa Nacional; y*
4. *A las demás Autoridades Federales en los casos de su competencia.*

La LEY consigna que las autoridades de los Estados, del Distrito Federal y de los Municipios, tendrán la intervención que la LEY y su Reglamento señalen.

El control y vigilancia de las actividades y operaciones industriales y comerciales que se realicen con explosivos, artificios y sustancias químicas, será hecho por la Secretaría de la Defensa Nacional.

Por lo que se refiere a los explosivos, la LEY establece tres tipos de permisos a saber:

1. *Permisos Generales;*
2. *Permisos Ordinarios; y*
3. *Permisos Extraordinarios.*

Los tres tipos de permisos que señala la LEY son de naturaleza intransferible.

La Secretaría de la Defensa Nacional tiene la facultad discrecional de negar, suspender o cancelar los permisos mencionados, cuando a su juicio las actividades amparadas en los permisos puedan causar peligro a las personas, a las instalaciones o alterar la tranquilidad de la población.

Los Permisos Generales, se concederán a personas que se dediquen de manera permanente a las actividades reguladas por la LEY, tendrán vigencia durante el año en que se expidan y podrán ser revalidados a juicio de la Secretaría de la Defensa Nacional.

Los Permisos Ordinarios se otorgarán en cada caso para realizar operaciones mercantiles con personas que tengan permiso general vigente o con comerciantes de otros paises.

Los Permisos Extraordinarios se otorgarán a personas que eventualmente se dediquen a alguna de las actividades regulas por la LEY.

Las sociedades que pretendan dedicarse a la fabricación y comercialización de explosivos, podrán permitir en su capital una participación de hasta el 49% de inversión extranjera, en los términos que establece la Ley de Inversión Extranjera.

Este porcentaje de inversión extranjera no incluye a las sociedades que adquieran y utilicen explosivos para actividades industriales y extractivas.

La Secretaría de la Defensa Nacional, tiene la facultad de practicar visitas de inspección a las negociaciones que se dediquen a las actividades reguladas por la LEY y a solicitar los informes necesarios respecto de estas actividades.

Las negociaciones tienen la obligación de prestar todas la facilidades a las autoridades militares para la práctica de las visitas de inspección.

La LEY considera como sanciones la fabricación, almacenamiento, transporte, comercialización, entre otros, sin el permiso correspóndiente.

OCTUBRE 1996

Permisos de Explosivos

Información general

Si usted requiere el uso de explosivos para romper roca en cualquiera de sus obras, será necesaria la obtención del permiso correspondiente de acuerdo a los requerimientos de la Dirección de Armas de Fuego y Explosivos de la Secretaría de la Defensa Nacional.

En el anexo No. 1 se podrá observar copia del Oficio No. 17221, girado por C. General de Brigada D.E.M. Jaime Palacios Guerrero, el 19 de junio de 1991 a esta Cámara donde nos proporciona los tipos de permisos que existen y los requisitos a cumplir para la obtención de los mismos.

Detalles del procedimiento

Polvorines:

El constructor que requiera el uso de productos explosivos por necesidad de su operación deberá construir polvorines que reúnan las caracteristicas solicitadas por la Secretaria de la Defensa Nacional a través de la Dirección de Armas de Fuego y Explosivos, siendo éstas las siguientes:

Lugar:

Los polvorines deberán ser colocados de acuerdo a la tabla de Seguridad de Distancia-Cantidad que viene en el Reglamento de Armas de Fuego y Explosivos de la Secretaria de la Defensa Nacional, la cual se puede observar en el anexo No. 2.

Capacidad:

La capacidad de los polvorines deberá estar en función de las necesidades del usuario y a la autorización de la Secretaria de la Defensa Nacional

En este caso se recomienda que la capacidad deba ser calculada de acuerdo a los consumos diarios de explosivos, al tiempo que se requiera para la obtención de los permisos para compra. Se debe tomar en cuenta la ubicación de la Zona Militar a cuya jurisdicción corresponda la obra. Otro concepto que se debe tomar en cuenta es la ubicacion de los proveedores y el tiempo de entrega de los productos una vez que se cuente con el permiso para compra de los mismos.

Todo lo anteriormente mencionado es con el propósito de que el usuario tenga en sus polvorines la cantidad de inventarios que le permitan mantenerse en operación evitando paros por falta de productos explosivos. Estos inventarios pueden ser para la operación de una semana, dos semanas o en algunos casos para un mes normal de operación.

Almacenamiento:

Antes de iniciar el almacenamiento de explosivos en un polvorín, se debe obtener el permiso correspondiente de parte de la Secretaria de la Defensa Nacional.

El almacenamiento de productos explosivos deberá ser de acuerdo a la tabla de compatibilidad para materiales empacados o envasados que vienen en el Manual de Armas de Fuego y Explosivos de la Secretaria de la Defensa Nacional. Esta tabla se puede ver en el anexo No. 3.

Ejemplo:

Agente explosivo (anfos)	=	Alto explosivo (godyne, emulsión, etc.)
Estopín elec.	=	Fulminantes
Mecha clover	=	Cordones detonantes

Construcción:

La construcción de los polvorines, es recomendable hacerla de la siguiente manera:

Cimentación: De mamposteria (Piedra braza)

Muros: Tabicón cemento-arena ó tabique, reforzado lo anterior con castillos a cada tres metros de distancia, de 15 cms. x 15 cms. de concreto armado.

Puertas: Deberán ser de madera de 4" de grueso con bastidor de metal (tanto en la base soporte como todo el perimetro de la puerta). (En el anexo No. 5 se puede ver el detalle de una puerta)

Techo:

Altura máxima de 4 mts., altura minima a las orillas de 2 70 mts., dejando respiradero entre la pared y el techo de 20 cms , el cual deberá ser protegido con algún tipo de malla metálica, para evitar que animales pequeños se introduzcan al polvorin. El material utilizado deberá ser de lámina de asbesto.

La parte más baja entre el techo y el piso deberá ser de 2 7 mts. de altura como se muestra en el anexo No. 4 Los polvorines podrán tener un techo a una o dos aguas.

Farallón:

El polvorín deberá estar rodeado por la corteza de algún cerro o en su defecto deberá contar con un farallón de tres mts. de altura y 15 mts. de largo a terminar a flor de tierra y teniendo entre el frente del polvorín a farallón cinco o seis metros como mínimo.

Características que deben reunir los polvorines

En general éstos deberán cumplir con las especificaciones complementarias que se muestran en el anexo No. 4.

En el caso de la construcción de polvorines se sugiere hacer el diseño de los mismos de acuerdo a las necesidades de su operación, asesorándose con personal experimentado en este campo.

Una vez que se cuenta con los polvorines, construidos de acuerdo a las tablas de distancias de seguridad de la Secretaría de la Defensa Nacional, es necesario conseguir las autorizaciones por parte de las autoridades correspondientes como son:

1.— Certificado del lugar de consumo expedido por la primera autoridad administrativa (Presidente Municipal o Delegado Político en el Distrito Federal). Modelo No. 4 (anexo 6).

2.— Opinión favorable del Gobernador del Estado o del Jefe del Departamento del Distrito Federal firmada por el titular. (Anexo 7) Esta opinión se debe solicitar por escrito acompañada por el certificado del lugar de consumo expedido por la primera autoridad administrativa (Punto No. 1).

3.— Cuando se cuente con las autorizaciones antes mencionadas, los documentos originales se deben adjuntar a la siguiente documentación que deberá ser presentada en los módulos correspondientes en el edificio de la Secretaría de la Defensa Nacional, en Lomas de Sotelo, siendo estos los siguientes:

— Solicitud, modelo anexo que se proporciona gratuitamente. Modelo (anexo 8).

— Referencias del lugar de consumo, se proporcionan en el anexo 9.

— Para personas físicas, copia certificada del Registro Civil del Acta de Nacimiento del solicitante.

— Para personas morales, Acta Constitutiva de la empresa.

— Plano de conjunto a 1000 metros alrededor del lugar de consumo y a escala de 1:4000, en la que figuran en su caso instalaciones militares, vías de comunicación, líneas eléctricas, telefónicas, telegráficas, acueductos, gasoductos, construcciones para casa-habitación, obras de arte, zonas arqueológicas, históricas o instalaciones industriales, que pudieran ser afectadas, con los principales accidentes topográficos Ejemplos (anexo 10).

— Plano circunstanciado a escala adecuada para la localización de sus instalaciones con especificaciones

Si la solicitud incluye almacenamiento

— Certificado de seguridad y referencia de los polvorines, modelos anexos que se proporcionan gratuitamente. (Modelo No. 2 anexo 11)

Se recomienda adquirir el Manual de Armas de Fuego y Explosivos de la Secretaría de la Defensa Nacional.

SECRETARIA DE LA DEFENSA NACIONAL
DEPARTAMENTO DE REGISTRO Y CONTROL DE ARMAS DE FUEGO Y EXPLOSIVOS
Lomas de Sotelo, D.F.

Tabla (13-1) de Seguridad de Distancia-Cantidad
(Materiales debidamente empacados o envasados)

Descripción del material	DISTANCIAS EN METROS Kilos De	a	POLVORINES CON PROTECCION Edificios habitados	Vias férreas	Caminos carreteras	Lineas de alta tensión	Entre polvorines
1. Dinamita, explosivos al nitrato de amonio, pólvoras negra y sin humo.	000	500	126	100	100	100	11
	500	750	146	100	100	100	13
	750	1,000	160	100	100	100	14
	1,000	1,250	170	100	100	100	15
	1,250	1,500	180	100	100	100	17
	1,500	2,000	200	100	100	100	18
	2,000	3,000	230	100	100	100	20
	3,000	4,000	250	100	100	100	23
	4,000	5,000	260	110	100	100	25
	5,000	6,000	270	117	100	100	26
	6,000	7,000	275	122	100	100	27
2. Artificios (fulminantes, estopines, conectores MS, cordon detonante, etc.)	7,000	8,000	285	127	100	100	28
	8,000	9,000	295	132	100	100	30
	9,000	10,000	305	137	100	100	31
	10,000	12,000	330	148	100	100	33
	12,000	14,000	350	154	105	103	35
	14,000	16,000	370	160	110	105	36
	16,000	18,000	390	168	116	112	38
	18,000	20,000	405	173	121	118	39
	20,000	25,000	445	185	135	130	43
	25,000	30,000	480	200	145	140	46
3. Por lo que respecta a los "artificios", únicamente se autoriza el almacenamiento en cada polvorin lo equivalente a 4 toneladas.	30,000	35,000	510	208	155	150	49
	35,000	40,000	535	218	160	155	53
	40,000	45,000	550	226	166	162	56
	45,000	50,000	565	240	169	166	63
	50,000	60,000	575	250	171	168	66
	60,000	70,000	585	262	175	172	73
	70,000	80,000	605	274	182	178	80
	80,000	90,000	620	284	186	183	86
	90,000	100,000	635	294	191	188	93
	100,000	125,000	675	378	210	206	117
4. Nitrocelulosa (30-70) ó sea 30 partes en peso del solvente por 70 partes del producto, con una nitración de 12.2% como maximo. Cloratos, fósforos, etc	000	500	115	100	100	100	10
	500	750	135	100	100	100	12
	750	1,000	145	100	100	100	14
	1,000	5,000	235	100	100	100	23
	5,000	25,000	400	170	122	120	40
	25,000	50,000	500	215	156	150	50
	50,000	75,000	535	242	165	160	70
	75,000	100,000	570	275	170	166	85
	100,000	125,000	607	340	190	188	110

5 Trinitrotolueno, ciclonita, fulminatos, picratos, etc	000	500	152	125	125	125	15
	500	750	175	135	135	135	20
	750	1,000	192	150	150	145	25
	1,000	5,000	312	165	165	160	35
	5,000	25,000	530	222	180	175	50
	25,000	50,000	675	283	200	200	75
6. Artificios pirotécnicos.	000	500	100	100	100	50	35
	500	1,000	160	160	160	100	45
A. Fabricantes.	1,000	5,000	200	200	200	150	55
	5,000	10,000	250	250	250	200	65
7 Artificios pirotecnicos. A. Comercio	A. La cantidad de artificios pirotécnicos que puedan tener en existencia es de 50 gramos por cada metro cúbico de espacio libre en el depósito de almacenamiento, en la inteligencia de que en los 50 gramos mencionados están incluidos la mezcla explosiva y la inerte, la capacidad total de seguridad sera determinada según la ubicacion de los depósitos y las dimensiones de los mismos.						
8 Almacenamiento de municiones en pequeño calibre para armas de fuego y para usos industriales.	1. La cantidad de municiones que pueden tener en existencia las personas o negociaciones que se dediquen a esta actividad es de 500 gramos por cada metro cúbico de espacio libre en el almacen o depósito, en la inteligencia de que en los 500 gramos está incluida la materia explosiva y la inerte, asi como la capsula. 2. Cuando se almacenen cartuchos que solamente tengan colocada la cápsula, se tomarán 85 gramos del explosivo que contengan dichas cápsulas por cada metro cúbico de espacio libre. 3. Si las negociaciones están establecidas en calles de mucho transito, solo se permitirá almacenar como máximo 50 kilogramos contenida en cartucho						

NOTA: Las distáncias arriba indicadas, son para cuando los polvorines o depósitos se encuentren protegidos por obstaculos naturales o artificiales, en caso contrario las distancias aumentan en un "cien por ciento (100%)"
En el interior de las fábricas unicamente se autoriza el almacenamiento de nitrocelulosa en una cantidad maxima de 5,000 Kgs observando las distancias de la presente tabla, disminuidas en un ochenta por ciento (80%)

COMPATIBILIDAD DE MATERIALES EMPACADOS O ENVASADOS

LA "X" INDICA QUE EL MATERIAL DE LA LINEA HORIZONTAL PUEDE ALMACENARSE CON EL ARTICULO DE LA COLUMNA VERTICAL.	Pólvora	Acido pícrico	Dinitrotolueno	Nitroalmidones	Nitroglicerina	Nitrocelulosa	Nitroguanidina	Tetril	Fulminato de mercurio	Nitruros de plomo, plata y cobre	Estifanato de plomo	Cloratos, percloratos y peróxidos	Sodio metálico	Magnesio en polvo	Aluminio en polvo negro u opaco	Fósforo	P E T N	T N T	Dinamita y amatoles	Nitrocarbonitratos húmedos	Nitrocarbonitratos secos	Nitrocarbonitratos ácidos	Fosgeno	Ciclonita (rdx)	Iniciadores de alta presión detonantes	Detonantes (estopines, cápsulas)	Mechas de seguridad	Cordones detonantes	Cordones encendedores de mecha	Conectores detonantes	Conectores encendedores	Artificios pirotécnicos	Cargas industriales
Pólvora	X																																
Acido pícrico		X																															
Dinitrotolueno			X																														
Nitroalmidones				X																													
Nitroglicerina					X																												
Nitrocelulosa						X																											
Nitroguanidina							X																										
Tetril								X																									
Fulminato de mercurio									X																								
Nitruros de plomo, plata y cobre										X																							
Estifanato de plomo											X																						
Cloratos, percloratos y peróxidos												X																					
Sodio metálico													X																				
Magnesio en polvo														X																			
Aluminio en polvo negro u opaco															X																		
Fósforo																X																	
P.E.T.N.																	X		X	X	X		–	X	X		X	X					
T.N.T.																		X															
Dinamita y amatoles																	X		X	X	X				X		X	X					
Nitrocarbonitratos húmedos																	X		X	X	X				X		X	X					
Nitrocarbonitratos secos																	X		X	X	X				X		X	X					
Nitrocarbonitratos ácidos																						X											
Fosgeno																							X										
Ciclonita (rdx)																	X							X	X		X	X					
Iniciadores de alta presión detonantes																	X		X	X	X			X	X		X	X					
Detonantes (estopines, cápsulas)																										X			X	X	X		
Mechas de seguridad																	X		X	X	X			X	X		X	X	X	X	X		
Cordones detonantes																	X		X	X	X			X	X		X	X					
Cordones encendedores de mecha																										X	X		X	X	X		
Conectores detonantes																											X			X			
Conectores encendedores																											X			X	X		
Artificios pirotécnicos																																X	
Cargas industriales																																	

//

ANEXO No. 1

SECRETARIA
DE LA
DEFENSA NACIONAL
DIR. GRAL. REG. FED
ARMAS FGO Y EXP.

EMDNTA

DEPENDENCIA.. DIRECCION GENERAL DEL REGISTRO FEDERAL DE ARMAS DE FUEGO Y EXPLOSIVOS.
SECCION ..... TECNICA DE EXPLOSIVOS.
MESA ..... TRAMITE.
NUMERO DEL OFICIO 17221
EXPEDIENTE ....

ASUNTO:- Se le informan los requisitos para la obtención de Permisos para el uso de explosivos.

Lomas de Sotelo, D.F., a 19 de junio de 1991.

C. PRESIDENTE DE LA CAMARA NACIONAL
DE LA INDUSTRIA DE LA CONSTRUCCION.
ALBORADA NUMERO 100.
COL. PARQUES DEL PEDREGAL.
14017 - MEXICO, D.F.

POR ACUERDO DEL C. GENERAL SECRETARIO DE LA DEFENSA NACIONAL, se le manifiesta a usted, que debido a la desinformación que los diversos organismos tienen acerca de la Ley Federal de Armas de Fuego y Explosivos y los requisitos que deben cumplir las personas físicas y morales para el otorgamiento de los Permisos para el uso de explosivos, se le informan los requisitos que la legislación vigente solicita.

I. PERMISO GENERAL.- Para actividades permanentes.
II. PERMISO EXTRAORDINARIO.- Para Actividades Eventuales (Por única vez).
III. PERMISO ORDINARIO.- Para comercialización entre empresas o particulares con Permiso General Vigente (Incluyendo Importaciones y Exportaciones).

Los requisitos para la obtención de ellos son:

- Solicitud, modelo anexo que se proporciona gratuitamente.
- Opinión Favorable del Gobernador del Estado o del Jefe del Departamento del Distrito Federal firmada por el titular.
- Certificado del lugar de consumo expedido por la primera autoridad administrativa (Presidente Municipal o Delegado Político en el Distrito Federal).
- Referencias del lugar de consumo, anexos que también se proporcionan.
- Para Personas Físicas, copia certificada del registro civil del acta de nacimiento del solicitante.

AL CONTESTAR ESTE OFICIO CITENSE LOS DATOS CONTENIDOS EN EL CUADRO DEL ANGULO SUPERIOR DERECHO

A la hoja número dos.

ESTADOS UNIDOS MEXICANOS

SECRETARIA
DE LA
DEFENSA NACIONAL
DIR. GRAL. REG. FED.
ARMAS FGO. Y EXP.

DEPENDENCIA.. DIRECCION GENERAL DEL
REGISTRO FEDERAL DE ARMAS
DE FUEGO Y EXPLOSIVOS.
SECCION ... TECNICA DE EXPLOSIVOS.
MESA ... TRAMITE.
NUMERO DEL OFICIO ...
EXPEDIENTE ... 17221

ASUNTO: HOJA NUMERO DOS.

- Para Personas Morales, Acta Constitutiva de la empresa.
- Plano de conjunto a 1000 metros alrededor del lugar de consumo y a escala de 1:4000, en que figurarán en su caso: instalaciones militares, vías de comunicación, lineas eléctricas, telefónicas, telegráficas, acueductos, gasoductos, construcciones para casa-habitación, obras de arte, zonas arqueológicas, históricas o instalaciones industriales, que pudieran ser afectadas. con los principales accidentes topográficos.
- Plano circunstanciado a escala adecuada para la localización de sus instalaciones con especificaciones.

Si la solicitud incluye almacenamiento.

- Certificado de seguridad y Referencias de los polvorines, modelos anexos que se proporcionan gratuitamente.

Entregados los documentos debidamente requisitados y que la zona militar correspondiente haya inspeccionado que reunen las medidas de control, seguridad y vigilancia para el uso de explosivos, esta Secretaría, si estan completos y correctos los documentos, normalmente entrega los Permisos a quienes los hayan solicitado, en un plazo no mayor de 10 días hábiles.

Por lo anterior, se le agradecerá hacerlo del conocimiento de sus agremiados; enfatizando que la tardanza es la obtención de los documentos que son expedidos en otras dependencias.

SUFRAGIO EFECTIVO. NO REELECCION.
EL DIRECTOR.

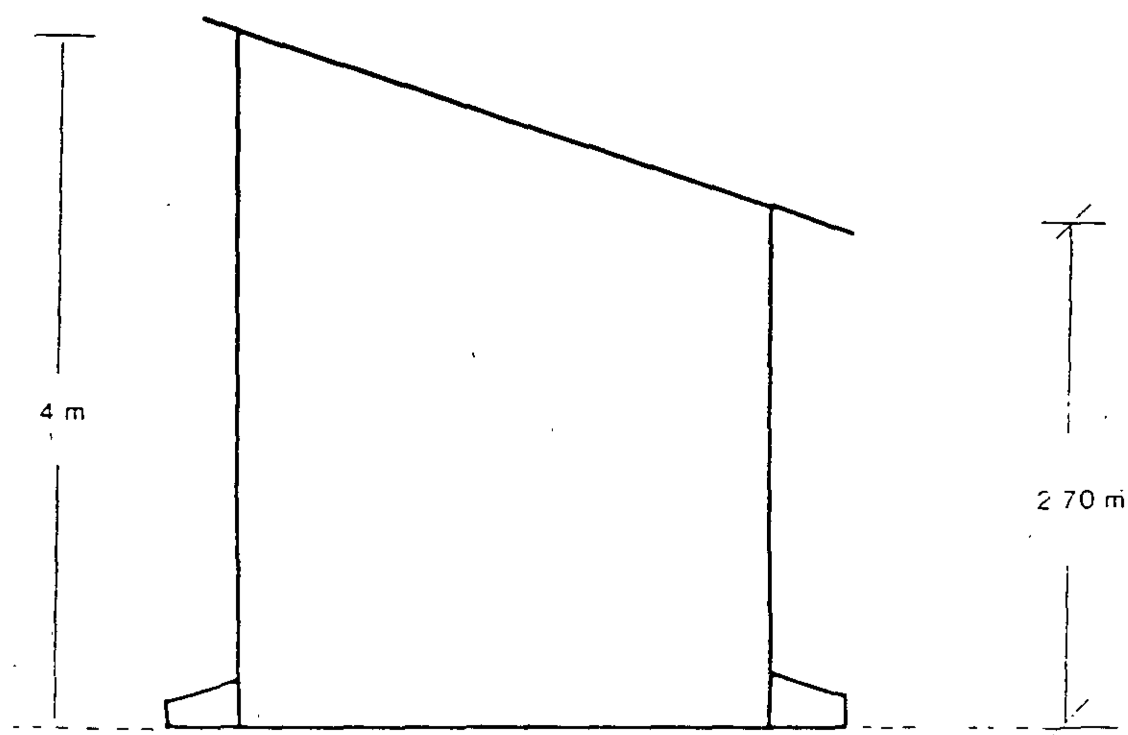
GENERAL [illegible] PALACIOS GUERRERO.

AL CONTESTAR ESTE OFICIO CITENSE LOS DATOS CONTENIDOS EN EL CUADRO DEL ANGULO SUPERIOR DERECHO

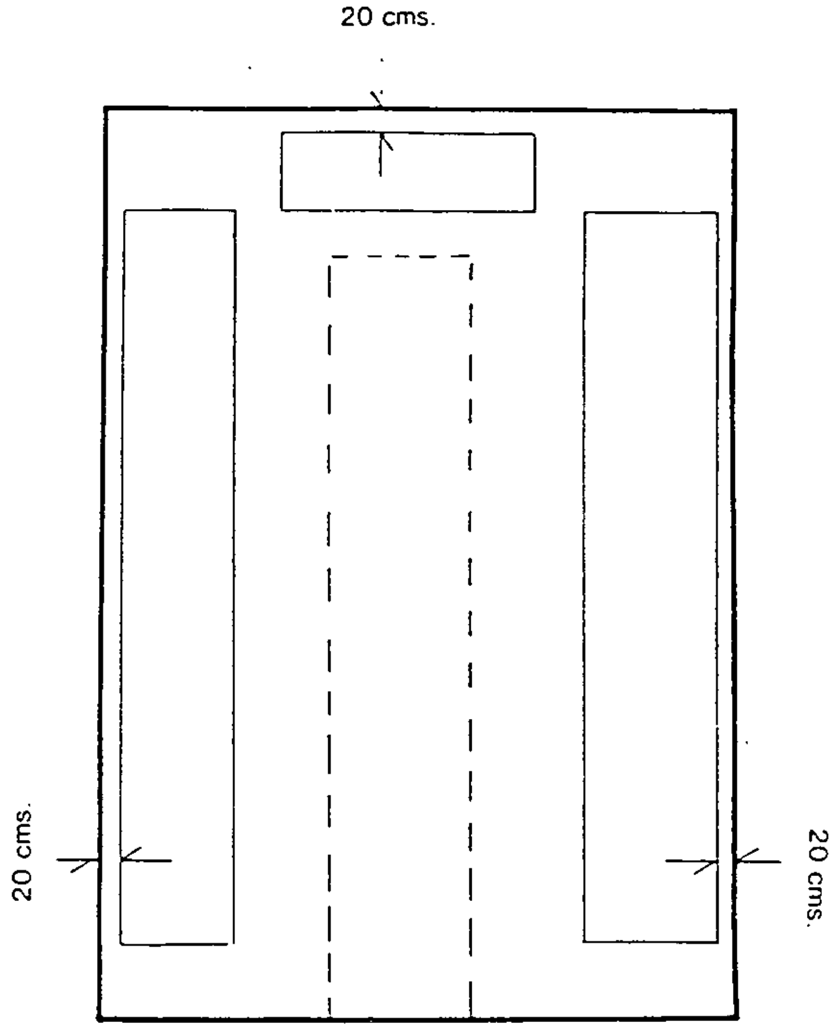
AAA-A'IV-ssa.

CARACTERISTICAS POLVORINES

1.— Pendiente en Banqueta
2.— Dren Perimetral
3.— Pala y Pico disponibles
4.— Bote de Arena
5.— Extinguidores (2)
6.— Puerta de Acero y Madera
con Chapa y Candado
7.— Tierra Fisica
8.— Rejilla de ventilación con
protección antirroedor
9.— Libre de Humedad
10.— Pisos pulidos y líneas de accesos



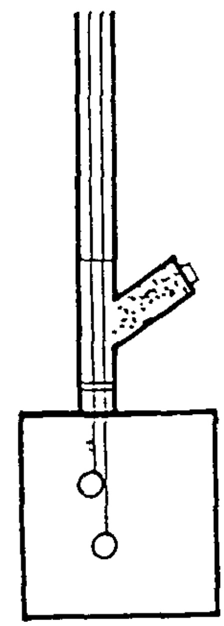
Corte Esquematico



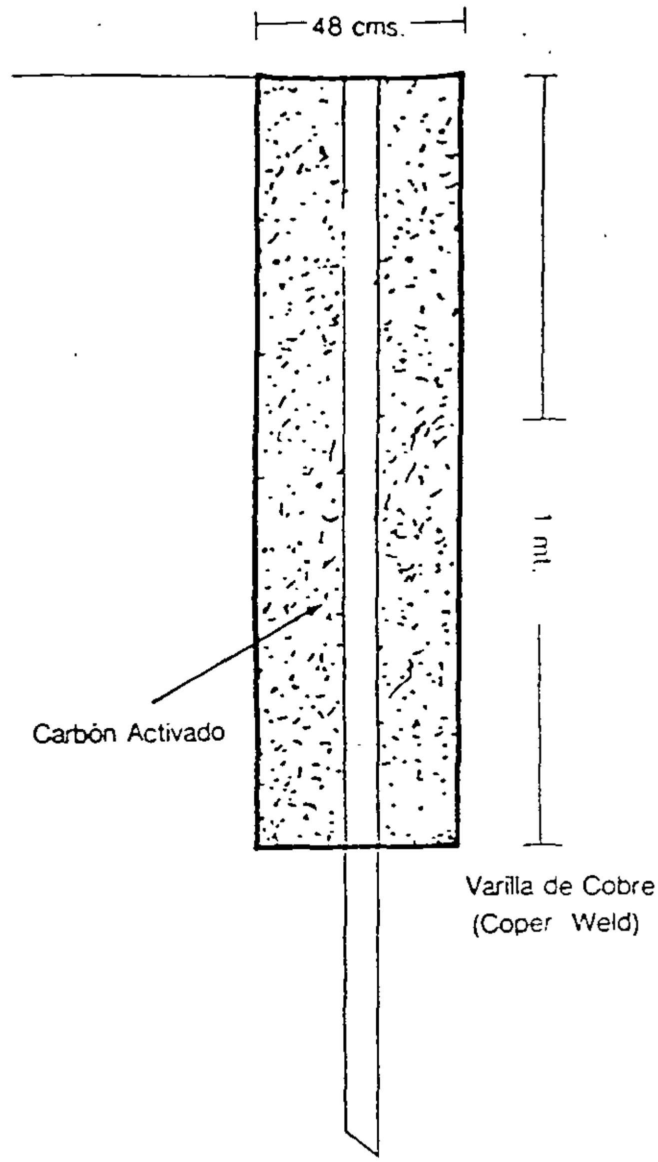
Planta Polvorin

11.— Separar estibas de paredes
12.— VIGILANCIA (24 Hrs.)
13.— Cercado
14.— Pararrayos
15.— Aplanado y Pintura
16.— Tarima de madera
17.— 20 mts. libre de mat. orgánica. alrededor
18.— Talud o protecc natural

19.— Iluminación APE Nema 9
Controles por fuera 110 volts
— Conduit de Pared Gruesa
— Sellos EYS

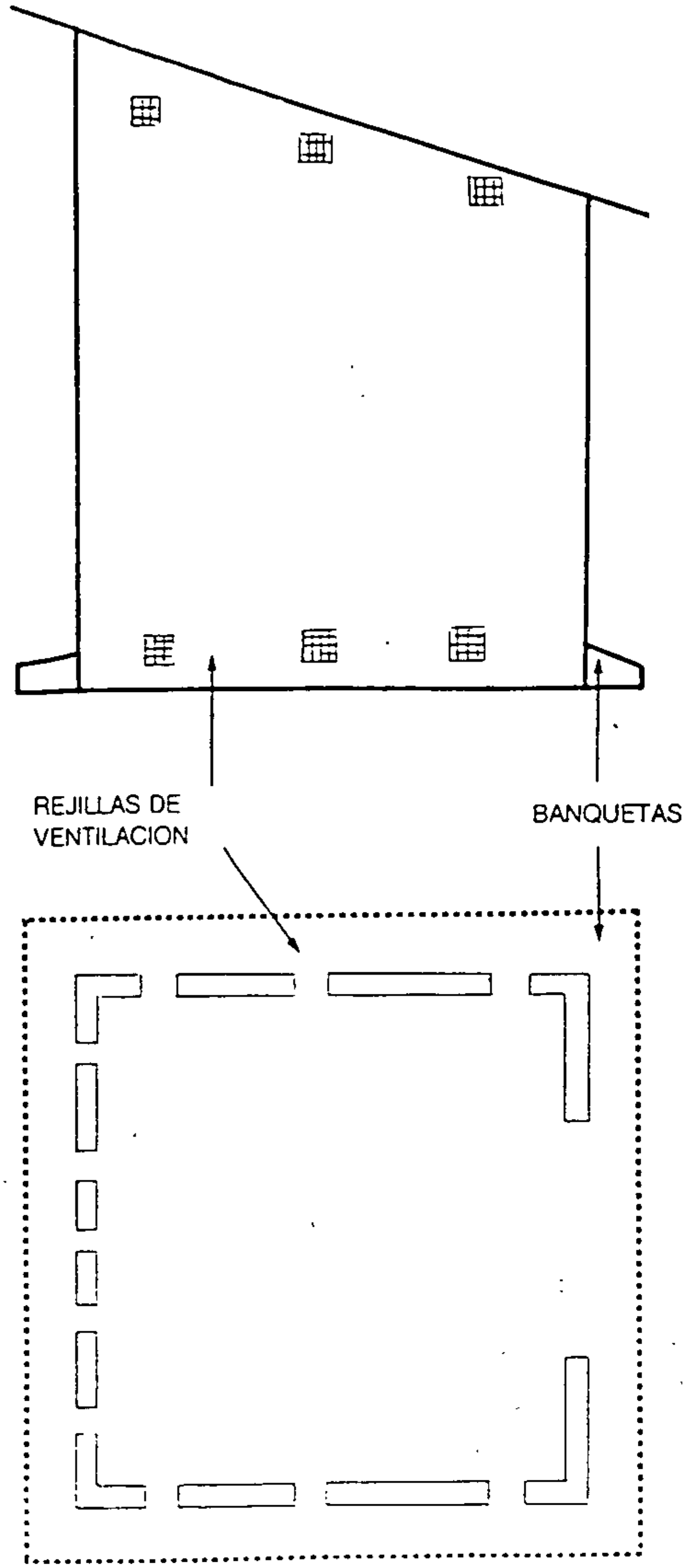


Lámparas Especiales



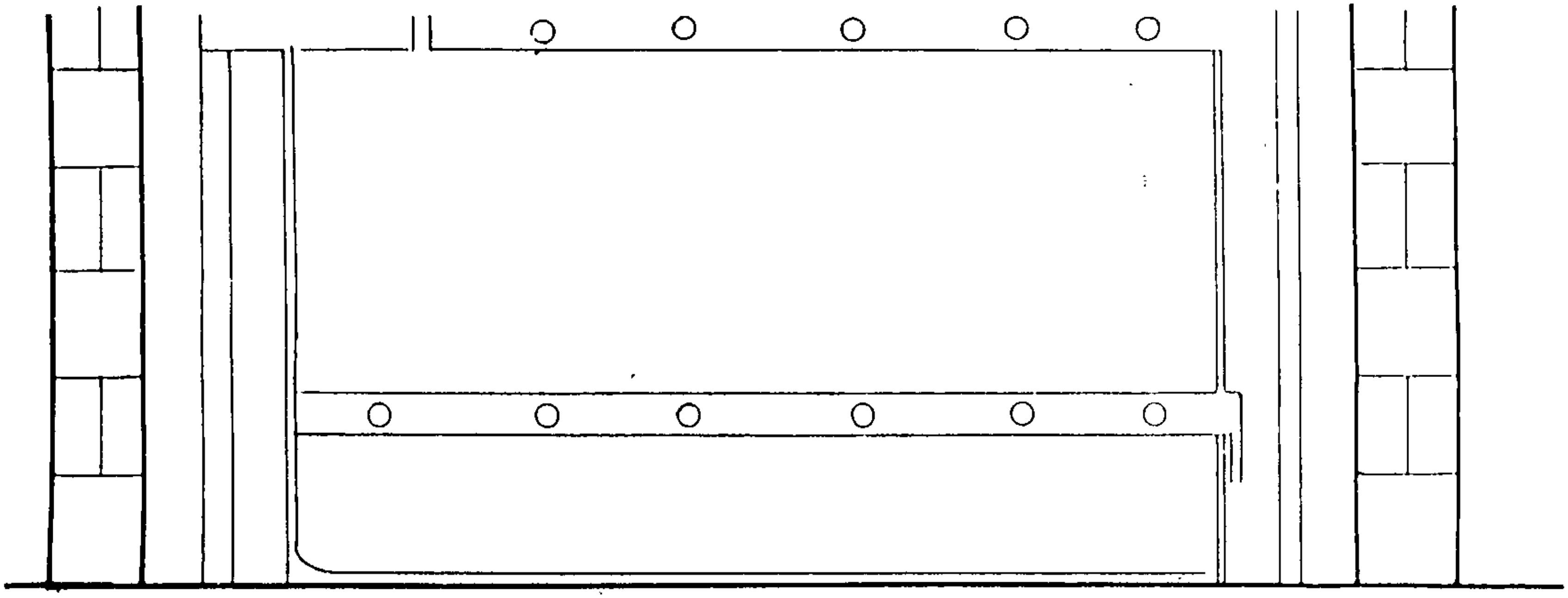
Detalles de tierra física

20.— Libro de registro de entradas y salidas
21.— Copia en cuadro del permiso
22.— Limite máximo de personas (letrero)
23.— Anuncios·
24.— Tambores de 200, con agua
Polvorin No ________________
Peligro Explosivos
Prohibido Fumar



CROQUIS SIN ESCALA DE LA DISPOSICION
DE LAS REJILLAS DE VENTILACION

ANEXO No. 5



PUERTA DE ACCESO
DETALLE

15.5

18

MODELO No. 4

ANEXO No. 7

SECRETARIA DE LA DEF. NAL.

DIR. GRAL. REG. FED. ARMAS DE FUEGO Y EXPLOSIVOS.

CERTIFICADO DE SEGURIDAD DEL POLVORIN O ALMACEN No. ______

EL SUSCRITO ________________________________ PRIMERA AUTORIDAD.
(Nombre y Apellido)

HACE CONSTAR Y CERTIFICA

QUE LOS POLVORINES UBICADOS EN: ________________________________
(Referidos a puntos conocidos del terreno para su fácil localización)

____

DESTINADOS PARA ALMACENAR: ________________________________
(Pólvora, dinamita, explosivos al nitrato de amonio,

____
artificios, clorato, nitrocelulosa, nitrato de amonio, etc.)

QUE SERA UTILIZADO POR: ________________________________
(Denominación o razón social)

CON DOMICILIO EN: ________________________________

______________ ______________ ______________
Localidad Municipio Estado

EN LA ACTIVIDAD DE: ________________________________
(Explotación de canteras, industria de la construcción, minerametalúrgica, cerillera, de

____
pinturas, etc.)

POR SUS CONDICIONES, SITUACION Y MEDIDAS DE SEGURIDAD, SON ADECUADOS: NO PRESENTAN PELIGRO PARA MANTENER EL ORDEN PUBLICO, ESTAN PROTEGIDOS CONTRA ROBOS Y GARANTIZAN LA TRANQUILIDAD DE LA POBLACION.

______________ a ______________ de ______________ de 19 ______

____
EL PRESIDENTE MUNICIPAL
(FIRMA Y SELLO)

ANEXO No. 6

SECRETARIA DE LA DEFENSA NACIONAL
DIRECCION GENERAL DEL REGISTRO FEDERAL DE ARMAS DE FUEGO Y EXPLOSIVOS
LOMAS DE SOTELO, D.F.

CERTIFICADO DE SEGURIDAD DEL LUGAR DE CONSUMO DE EXPLOSIVOS, ARTIFICIOS O SUBSTANCIAS QUIMICAS RELACIONADAS CON LOS MISMOS, EXPEDIDO POR LA PRIMERA AUTORIDAD ADMINISTRATIVA.

EL SUSCRITO ______________________________ PRIMERA AUTORIDAD

ADMINISTRATIVA DE. ______________________________

HACE CONSTAR Y CERTIFICA:

QUE ______________________________

(Denominación o razón social)

CON DOMICILIO EN.

CALLE ______________ NUMERO ______________ CIUDAD, POBLACION O LOCALIDAD ______________

MUNICIPIO O DELEGACION ______________ ESTADO, TERRITORIO O DISTRITO ______________ Z.P. ______ TELEFONO ______

EMPLEARA LOS MATERIALES SIGUIENTES: ______________________________

(pólvora, dinamita, explosivos al nitrato de

____

amonio, artificios, nitrocelulosa, clorato de potasio, etc.)

TRABAJOS QUE EFECTUARA PRECISAMENTE EN EL LUGAR DE CONSUMO UBICADO EN

____

(Referido a puntos conocidos del terreno para su fácil localización)

EL CUAL POR SU SITUACION, NO REPRESENTA PELIGRO PARA LA SEGURIDAD Y TRANQUILIDAD PUBLICA

______________________ a ________ de ______________________ de 19 ____

____

Sello y firma

ANEXO No. 8

SECRETARIA DE LA DEFENSA NACIONAL
DIRECCION GENERAL DEL REGISTRO FEDERAL DE ARMAS DE FUEGO Y EXPLS

SOLICITUD DE PERMISO GENERAL PARA DEDICARSE A LA COMPRA Y CONSUMO DE EXPLOSIVOS, ARTIFICIOS Y SUBSTANCIAS QUIMICAS RELACIONADAS CON EXPLOSIVOS (ARTICULO 42 FRACCION I DE LA L.F.A.F.Y.E.)

A. DATOS DEL SOLICITANTE:

Apellido Paterno ______ Apellido Materno ______ Nombre (s) ______

Fecha de Nacimiento ______ Sexo ______ Lee ______ Escribe ______ Profesion u Oficio ______ Nacionalidad ______

Calle ______ Número ______

Ciudad, Población o Localidad ______ Código Postal. ______

Municipio o Delegación ______ Estado, Distrito ______ Teléfono ______

Referencias del Domicilio cuando se requieran. ______

C. DATOS DE LA NEGOCIACION.

Denominación o Razón Social ______

Calle ______ Número ______

Ciudad, Población o Localización ______ Código Postal ______

Municipio o Delegación ______ Estado o Distrito ______

Actividad a la que se dedicará ______ Teléfono ______

EXPLOSIVOS SOLICITADOS MENSUALMENTE: (CANTIDADES) Y (TIPOS)

ALTO EXPLOSIVO ______

AGENTES EXPLOSIVOS ______

ARTIFICIOS ______

______

SUBST. QUIMICAS ______

OTROS ______

______ Lugar y fecha

______ Firma Autorizada.

ANEXO No. 8.1

SECRETARIA DE LA DEFENSA NACIONAL
DIRECCION GENERAL DEL REGISTRO FEDERAL DE ARMAS DE FUEGO Y EXPLOSIVOS.
LOMAS DE SOTELO, D.F.

SOLICITUD DE PERMISO EXTRAORDINARIO PARA LA COMPRA DE POLVORA DE EXPLOSIVOS DE ARTIFICIOS O DE SUBSTANCIAS QUIMICAS RELACIONADAS CON LOS MISMOS (ARTICULO 57 DEL REGLAMENTO DE LA LEY FEDERAL DE ARMAS DE FUEGO Y EXPLOSIVOS).

DATOS DEL SOLICITANTE:

PRIMER APELLIDO SEGUNDO APELLIDO PRIMER NOMBRE SEGUNDO NOMBRE

FECHA DE NACIMIENTO NACIONALIDAD SEXO LEE ESCRIBE PROFESION, OFICIO

OCUPACION CALLE NUMERO CIUDAD, POBLACION O LOCALIDAD

MUNICIPIO O DELEGACION ESTADO, TERRITORIO O DISTRITO Z.P. TELEFONO

REFERENCIAS DEL DOMICILIO CUANDO LAS REQUIERA

DATOS DE LA NEGOCIACION

DENOMINACION O RAZON SOCIAL

CALLE NUMERO CIUDAD, POBLACION O LOCALIDAD

MUNICIPIO O DELEGACION ESTADO, TERRITORIO O DISTRITO Z.P. TEL.

ACTIVIDAD A LA QUE SE DEDICARA

CANTIDADES Y CLASES DE MATERIALES EXPLOSIVOS POR COMPRAR

TIEMPO EN QUE SE CONSUMIRAN LOS MATERIALES SEÑALADOS EN EL PUNTO ANTERIOR

PROTESTO, QUE LOS DATOS ANOTADOS SON VERIDICOS, QUE LA FIRMA ES AUTENTICA Y LA UNICA QUE UTILIZARE EN LOS DOCUMENTOS QUE DIRIJA A LA SECRETARIA DE LA DEFENSA NACIONAL.

Lugar y Fecha Firma del solicitante

SECRETARIA DE LA DEFENSA NACIONAL
DIRECCION GENERAL DEL REGISTRO FEDERAL DE ARMAS DE FUEGO Y EXPLOSIVOS.
LOMAS DE SOTELO, D.F.

REFERENCIAS DEL LUGAR DONDE EL SOLICITANTE CONSUMIRA O USARA LOS EXPLOSIVOS ARTIFICIOS O SUSTANCIAS QUIMICAS RELACIONADAS CON LOS MISMOS, EN LAS OBRAS, OPERACIONES INDUSTRIALES O EXPLOTACION MINERA QUE SEÑALA EN SU GESTION PETITORIA.

____

____

(Denominación o Razón Social del peticionario)

SITUACION EXACTA DEL LUGAR DE CONSUMO: ______________________
(Referida a puntos conocidos del terreno

____

para facilitar su localización).

UBICADO EN: ____

Municipio Delegación Estado Distrito

DISTANCIAS MAS CORTAS, EN SUS ALREDEDORES A: ______________ MTS.

Casas habitación

MTS. MTS. MTS MTS.

Carreteras Vias Férreas Lineas eléctricas Polvorines

"EXISTE O NO" BARRERA DE PROTECCION A: ______________ ______________

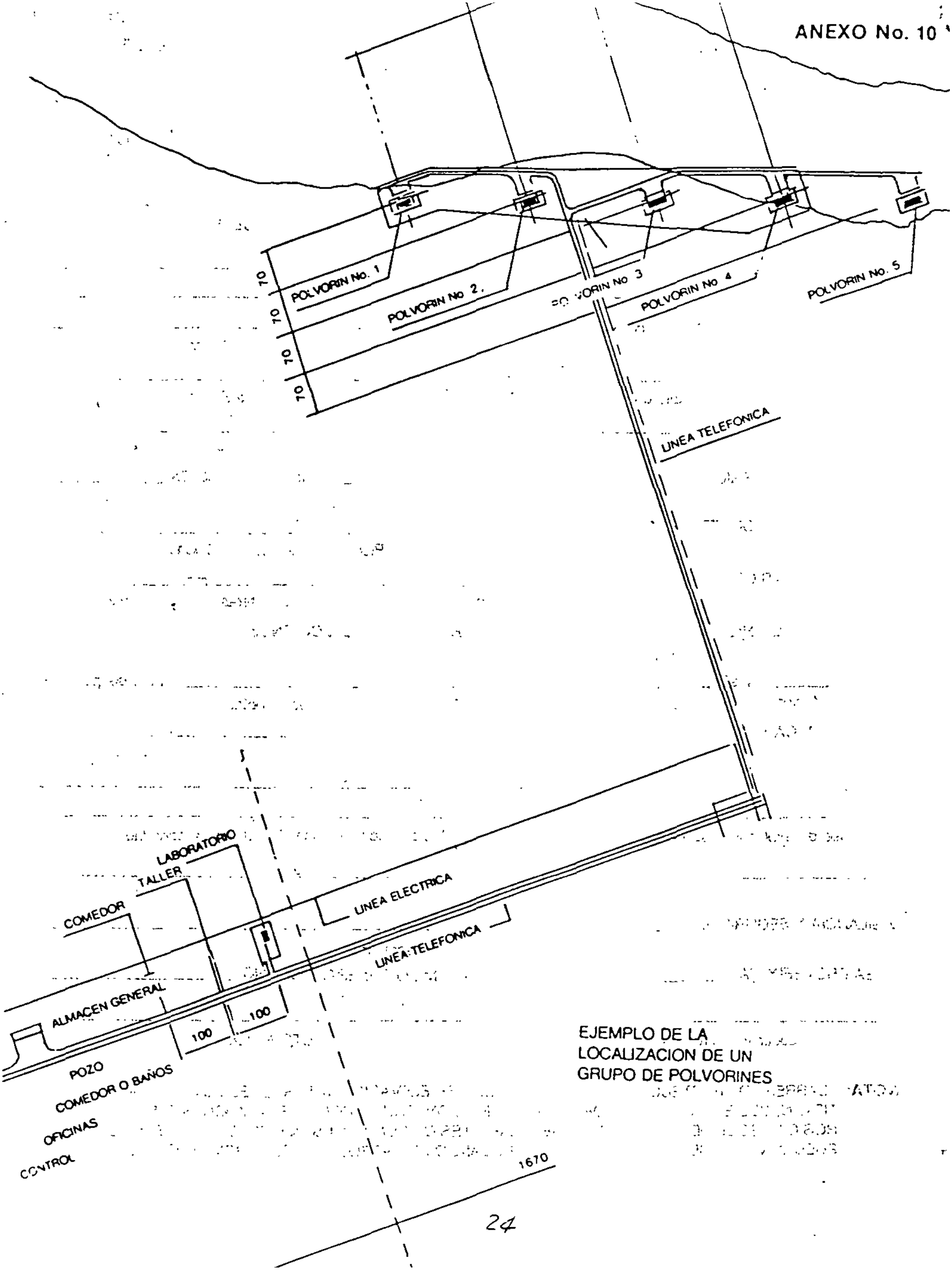
Casas habitación

____

Carreteras Vias férreas Lineas eléctricas Polvorines

______________________ ______________________
LUGAR Y FECHA FIRMA DEL INTERESADO

NOTA: "BARRERA DE PROTECCION", SIGNIFICA CUALQUIER ELEVACION NATURAL DEL TERRENO MURALLA ARTIFICIAL DE ESPESOR NO MENOR DE UN METRO CONSTRUIDA CON TIERRA, ADOBES O SACOS TERRENOS, O BOSQUE DE TAL DENSIDAD QUE LAS PARTES CIRCUNDANTES QUE REQUIERAN PROTECCION NO PUEDAN VERSE DESDE EL LUGAR DE CONSUMO DE EXPLOSIVOS AUN CUANDO LOS ARBOLES ESTEN DESPROVISTOS DE HOJAS.



EJEMPLO DE LA
LOCALIZACION DE UN
GRUPO DE POLVORINES

ANEXO No. 11

SECRETARIA DE LA DEF. NAL. **MODELO No. 2** **DIR. GRAL. REG. FED. ARMAS, DE FUEGO Y EXPLOSIVOS.**

REFERENCIAS DE POLVORINES

REFERENCIAS DE POLVORINES DONDE EL SOLICITANTE ALMACENARA EXPLOSIVOS, ARTIFICIOS Y/O SUBSTANCIAS QUE UTILIZARA EN OBRAS, OPERACIONES INDUSTRIALES, COMERCIALES O EN LA EXPLOTACION MINERA.

POLVORINES No. ________________ (o ALMACEN)

NOMBRE ____

RAZON SOCIAL ____

SITUACION EXACTA DEL POLVORIN ____

Referida a puntos conocidos del terreno para facilitar su colocación.

UBICADO EN ____________________ ó ____________________

Municipio o Delegación Estado Distrito Federal

TIPO ________ ________ ________ ________ ________

Superficial Semi-enterrado Enterrado Socavón de mina Móvil

DIMENSIONES INTERIORES ________ mts. ________ mts. ________ mts. VENTILACION ____________

Largo Ancho Alto

MATERIALES DE CONSTRUCCION DE ________ ________ ________ ________ ________

Cimientos Muros Piso Puertas Techo

DISTANCIAS MAS CORTAS DEL POLVORIN A: ____________ mts. ________ mts. ________

Casas habitación carreteras vias

________ mts. No. ________ mts. SI O NO EXISTE BARRA DE PROTECCION A:

férreas polvorín

________ mts. ________ mts. ________ mts. ____________ mts. del polvorín

casas habitación carreteras vias férreas líneas eléctricas

ARTICULO Y CANTIDAD POR ALMACENAR: ____

____

____

tratándose de explosivos, se tendrá en cuenta: capacidad y tablas de "compatibilidad" y distancia cantidad

____

VIGILANCIA Y SEGURIDAD: ____

(describirlas)

CASA PROVEEDORA ____________________ PERMISO GENERAL NUMERO ____________

____________________ ____________________

Lugar y fecha AUTORIZADO

NOTA: "BARRERA DE PROTECCION", SIGNIFICA CUALQUIER ELEVACION NATURAL DEL TERRENO MURALLA ARTIFICIAL DEL ESPESOR O MENOR DE UN METRO CONSTRUIDA CON TIERRA, ADOBES O SACOS TERREROS O BOSQUE DE TAL DENSIDAD QUE LAS PARTES CIRCUNDANTES QUE REQUIERAN PROTECCION NO PUEDAN VERSE DESDE EL POLVORIN, AUN CUANDO LOS ARBOLES ESTEN PROVISTOS DE HOJAS.