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The Fundamentals of Blast Design

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Disclaimer

The information contained in this document is meant to be informative and educational to those individuals involved with the use of explosives. As such, it is believed that the information presented herein is both reliable and accurate; however, because the author and the Golden West Chapter of the ISEE have no control over the conditions under which the information might be used, any and all risks associated with the use of the information contained herein lies with the reader.

It must be understood by all concerned that blasting is not an exact science and that safe blasting incorporates experience as well as the study and proper application of the fundamentals involved. If the reader is not adequately experienced in the type of blast-related operation that he intends to undertake, he is advised to obtain assistance from a qualified, experienced person before commencing the work and under no circumstances should he attempt to design blasts or conduct blasting operations based solely upon use of the information contained herein.

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Introduction

Very simply put, blasting involves the controlled release of explosive energy in order to accomplish a particular task. Obviously, there could be many different tasks, but the one on which we will concentrate will be the production blasting of rock.

The process of designing a blast entails the thoughtful combining of basic physical parameters such as hole diameter, hole depth, distance between holes, type and volume of explosives to be loaded, initiation scheme to be utilized, etc., all as they apply to the volume, density and structure of the material to be blasted and the size gradation desired. Many factors must be considered before a particular design is produced. These factors, and how they can be adjusted to achieve the desired results, will be covered in this document.

Definitions

The following terms are commonly used in blasting and should be understood by anyone conversant in the subject:

Burden – There are actually two burdens, the drilled burden and the shot burden. Drilled burden is the distance between a row of holes and the nearest free face and is measured perpendicular to the row of holes. It is also the distance between two rows of holes. Shot burden can change slightly from drilled burden because it represents the distance between a hole that is detonating and the nearest free face that has developed in the blast. In either case, the burden represents that volume of material that a detonating hole or holes are expected to fragment and shift. When laying out the shot, the term usually refers to the drilled burden.

Spacing – Represents the distance between holes in a row. A drill pattern is always described in terms of burden and spacing, in that order. (i.e., a 6 x 8 pattern has a burden of 6 feet and a spacing of 8 feet.)

Overburden – Don't confuse this term with Burden above. Overburden is soil and other materials overlaying the rock to be blasted. Usually removed before drilling but occasionally left in place to confine the blast and allow loading explosives higher in the hole (nearer the top of the rock).

Sub-drilling – The amount of hole that is drilled below the intended floor of the excavation. Except in those situations where the rock is in horizontal bedding planes, the detonating charge will usually leave a crater at the bottom of the hole rather than shearing the rock on a horizontal plane. Because of this, it is not uncommon to sub-drill an amount approaching half of the burden distance in order to be able to excavate to the intended depth.

Stemming – In order to confine the energy from the explosive, the top portion of the hole is stemmed or back-filled with inert material. Because of their proximity to the hole, drill cuttings are usually used, although other material, including stemming plugs, can be used.

Decks or decking – This is a means of separating two or more charges within a hole. This is usually done to (1) reduce the amount of explosive detonating in a given instant by having the decks fired on different delays, or (2) to avoid loading explosives in weak zones, voids or mud seams in the rock. The decks are separated by inert stemming material and each deck requires some means of initiation.

Production or Primary Blasting – A blast that is intended to adequately fragment a given volume of rock. The rock may be removed in one or more production (or primary) blasts. If an excavation is of sufficient depth to require removal in more than one lift, each lift would be removed utilizing one or more production blasts.

Secondary Blasting – Blasts that may be required to remove or reduce material that wasn't adequately fractured in production blasts (i.e. trimming blasts or removing high bottom.) Also the blasting of boulders or other specialized blasts whether or not production blasting was conducted.

Powder Factor – The ratio between the weight of explosives detonated and volume of material blasted, usually defined in pounds per cubic yard for construction blasts. In mining, powder factor is usually expressed in pounds per ton (or tons per pound of explosive). When discussing powder factors, it is important to know whether one is using "shot powder factor" or "pay (or yield) powder factor". Shot powder factor includes the material in the sub-drilling zone in the calculations, while pay powder factor does not. Most blasters use shot powder factor because it more accurately describes the amount of work that the explosive is supposed to accomplish. Accountants tend to use pay powder factor because it more closely describes the amount of yield or saleable material generated by the explosives in the shot.

Detonator – The devices, either electric or non-electric, that are inserted in the explosives and used to detonate them.

Delay – The time interval between detonators (and their corresponding explosive charges) exploding. Because modern initiation systems provide for further subdividing of the delay times in conventional detonators, the delay times can be tuned for specific blasting needs.

Initiation System – The entire system for initiating the blast, including the blasting machine or starter, detonators, delay devices and their interconnecting parts.

Booster or Primer – A fairly sensitive charge that is used to initiate less sensitive explosive charges. Boosters are in a cast form with a detonator well and/or detonating cord tunnel, but Primers can also be a cartridge product.

Detonating Cord – A cord consisting of a core charge of pentaerythritol tetranitrate (PETN) over-wrapped with layers of plastics and textiles. It is available in various core loadings and detonates at approximately 23,000 feet per second. Originally developed as an initiation system, it has also been used in specialized blasting situations as the primary charge. Detonating cord is sometimes referred to as Primacord, which is the brand name of one specific product. (PETN is also the base charge in most detonators and is an ingredient in most cast boosters.)

Pre-splitting – A cautious blasting procedure where a row of lightly-loaded perimeter holes is detonated ahead of the main production blast and propagates a crack along the row of holes. This crack is intended to protect the final perimeter wall by allowing expanding gasses to vent and by intercepting cracking (back-break) from subsequent detonating production holes. NOTE: A pre-split crack has little or no effect in reducing vibration from subsequent blasts and, in fact, the pre-splitting blast creates more vibration per unit of explosive weight than many other forms of blasting.

Smooth-blasting – A cautious blasting technique similar to pre-splitting, except that the holes are detonated after the production holes in the main blast. The intent is not to form a crack, however, but to blast loose the remaining burden with the lighter charges without causing excessive damage to the perimeter wall. Often the charge weights in the nearest production holes are reduced to assist in preserving the perimeter.

Sinking Cut – a blast where no free vertical (or sloped) face exists and it is necessary to ramp down into a horizontal surface. In this type of blast a portion of the blasted material must be expelled upward to make room for the expanding material from subsequent holes detonating. Some flyrock will necessarily occur and must be taken into account in designing such blasts.

Throw or Heave - Movement or shifting of the blasted material an intended distance and direction by the force of the blast.

Flyrock - Material that is expelled from the blast and travels farther than expected or intended.

Blasting Mats – Mats used to cover a blast in an urban situation where flyrock cannot be tolerated and the situation dictates that explosives are loaded fairly high up in the holes. Note that it is not practical to cover large blast areas, and prevention of flyrock is best addressed in blast design for those situations. Blasting mats are usually fabricated from sections of rubber tires, manila rope, used conveyor belting or other similar materials. Many contractors opt to cover the blast with soil, sand or other fine material. This can be successful, but it is necessary that a sufficient amount be used and that it contain no rocks or other projectiles. Covering the blast with any of these materials or devices must be accomplished carefully so that the initiation system is not damaged in the process.

Swell – Because blasted rock occupies a greater volume than before it was blasted, the resulting increase in volume is referred to as "swell". Swell can be accounted for by vertical mounding or by displacement along a free face.

Scaled Distance (square root or cube root) – In order to compare the adverse effects of blasts of various sizes, a means of combining distance and charge weight to a common base is necessary. This combined number, which has no units, is **Scaled Distance** and is derived by dividing the distance to the blast by either the square root or the cube root of the maximum charge weight per delay. For conventional blasts using linear charges, it is common to use square root. For spherical charges (the length of the charge is less than four times the diameter), it is common to use cube root scaling. It is also common to use cube root scaling for comparing air pressures or water pressures from blasting. Either scaling method can be correct for its application, but the two methods should not be mixed.

Critical Diameter – That diameter below which an explosive may fail to propagate. For example, AN/FO, depending upon formulation accuracy, density and grain size, may fail to maintain an explosive reaction if the charge diameter is somewhere near 7/8" or less. Manufacturers package products in cartridge sizes and containers that should be well above critical diameter. Pumped or poured products, however, could have problems if loaded in holes that are too small. For most explosives, decreasing the diameter also reduces the detonation velocity, or the speed at which the explosive reaction proceeds through the explosive. Detonation velocity plays an important part in maintaining this reaction, so it must be maintained at a level high enough so that the reaction does not fail. .

Critical Density – Similarly, most explosives have a specific density above which detonation can possibly fail. Again, properly manufactured products would be safely below critical density for normal conditions. A problem could exist, however, if conditions cause an explosive product to be compressed to a point above its critical density. This could occur with an extremely deep hole where the weight of the explosive column compresses itself at the bottom of the hole. The density could also be increased by some outside agency. An example of this might be AN/FO in a small diameter hole (say less than 3"), with detonating cord extending down the length of the hole to a booster. Under normal conditions, the cord would not initiate the AN/FO, but would cause a tunnel to be formed in the AN/FO. The AN/FO immediately surrounding the tunnel would probably be compressed above its critical density and would have little, if any, contribution to the shot.

It should be noted that, in explosives that are affected by them, Critical Diameter and Critical Density are interactive and each can have an adverse impact on the other. In other words, as the density of such an explosive approaches the critical point, its critical diameter may also be increased, resulting in a failure to detonate.

The Physics of Energy Release and Rock Breakage.

Before addressing the specifics of blast design, it would be helpful to better understand just what happens when explosives are detonated in a borehole and how that process affects the surrounding material. When an explosive charge is detonated, a chemical reaction occurs that rapidly changes the solid or liquid explosive material into a hot gas. This reaction starts at the point of initiation and forms a convex shock wave on its leading edge that acts on the borehole wall and propagates through the explosive column. The reaction zone where this transformation takes place in the explosive can vary in thickness from about .04" in high velocity explosives to over 1.0" in products such as AN/FO. (In explosive products containing a large amount of aluminum particles, some of the reaction may still be taking place in the hot gasses behind the actual reaction zone.) Ahead of the reaction zone are undetonated explosive products and behind the reaction zone are expanding hot gasses (see Figure 1 on page 6). The faster the detonation process, the quicker the energy, in the form of a shockwave followed by gas pressure, is applied to the borehole wall. It takes rather sophisticated and expensive equipment to measure detonation pressure directly, but it can be roughly approximated using the following equation:

$$P = \frac{(2.16 \times 10^{-4})\ (0.45)\ (pc^2)}{1 + 0.0128\ (p)}$$

where,

P = detonation pressure (lbs / in^2)

p = explosive density (lbs / ft^3)

c = explosive detonation velocity (ft / sec)

(Note that the above simplified formula does not take into consideration factors such as pressure decay, the density or sonic velocity of the rock, explosive/rock coupling or other factors.)

Values for detonation pressures can range from roughly 2,000,000 psi for cast boosters in strong rock to around 100,000 psi for some lower-rate permissible explosives in weaker material.

Analysis of the above equation discloses that, of the two parameters, detonation velocity has more effect on detonation pressure than does the explosive density. While the pressure varies directly with the density, it varies with the square of the velocity. In other words, for explosives with similar densities, the detonation pressure will increase by a factor of four when the detonation velocity is doubled. (The reader is cautioned not to assume that higher detonation velocity explosives are always better. The opposite is often the case, as shall be seen later.)

The faster the detonation velocity of the explosive, the quicker the energy is applied to the borehole wall, and usually for a shorter time period. Conversely, with a slower detonation velocity, the energy is applied more slowly, and for a longer time period.

The degree of coupling between the explosive and the borehole wall will have an effect on how efficiently the shockwave is transmitted into the rock. Pumped or poured explosives will result in better transmission of energy than would cartridge products with an annular space between the cartridge and the borehole wall.

The pressure that builds up in the borehole depends not only upon explosive composition, but also the physical characteristics of the rock. Strong competent rock will result in higher pressures than weak, compressible rock.

When the shock wave reaches the borehole wall the fragmentation process begins. This shock wave, which starts out at the velocity of the explosive, decreases quite rapidly once it enters the rock and in a short distance is reduced to the sonic velocity of that particular rock.

Most rock has a compressive strength that is approximately 7 times higher than its tensile strength, i.e. it takes 7 times the amount of energy to crush it as it does to pull it apart. When the shockwave first encounters the borehole wall, the compressive strength of the rock is exceeded by the shockwave and the zone immediately surrounding the borehole is crushed. As the shockwave radiates outward at declining velocity, its intensity drops below the compressive strength of the rock and compressive crushing stops. The radius of this crushed zone varies with the compressive strength of the rock and the intensity of the shock wave, but seldom exceeds twice the diameter of the borehole. However, beyond this crushed zone, the intensity is still above the tensile strength of the rock and it causes the surrounding rock mass to expand and fail in tension, resulting in radial cracking. The hot gas following the shockwave expands into the radial cracks and extends them further. This is the zone where most of the fragmentation process takes place. (See Figure 2.)

Additionally, if the compressive shockwave pulse radiating outward from the hole encounters a fracture plane, discontinuity or a free face, it is reflected and becomes a tension wave with approximately the same energy as the compressive wave. This tension wave can possibly “spall” off a slab of rock (see figure 3). This reflection rock breakage mechanism depends heavily upon three important requirements: (1) the compressive wave (and resulting reflected tensile wave) must still be of sufficient intensity to exceed the tensile strength of the rock, (2) the material on opposite sides of the fracture plane or discontinuity must have different impedances, (3) the compressive pulse must arrive parallel to, or nearly parallel to, the fracture plane or free face.

If carried to extreme, when this reflective breakage or “spalling” process occurs at a free face, it can result in violent throw, a situation that is not desirable. This can be overcome by designing blasts with burden and spacing dimensions that are within reasonable limits.

Once the compressive and tensile stresses caused by the shockwave drop below the tensile strength of the rock, the shock wave becomes a seismic wave that radiates outward at the sonic velocity of the material through which it passes. At this point, it is no longer contributing to the fragmentation process.

Several key points have been learned through the years from studying the physics of energy release and how it applies to the fragmentation process:

1. Within the range of conventional blasting, the physical characteristics of the rock are more important than the characteristics of the explosive and can have a greater impact on the success or failure of a blast.

2. Final-size fragmentation is usually obtained before any appreciable rock movement or throw occurs.

3. Rock can absorb only so much energy and only at a certain maximum rate before it will fail.

4. The final displacement of the bulk of the rock is more a function of the duration of the gas pressure than its intensity.

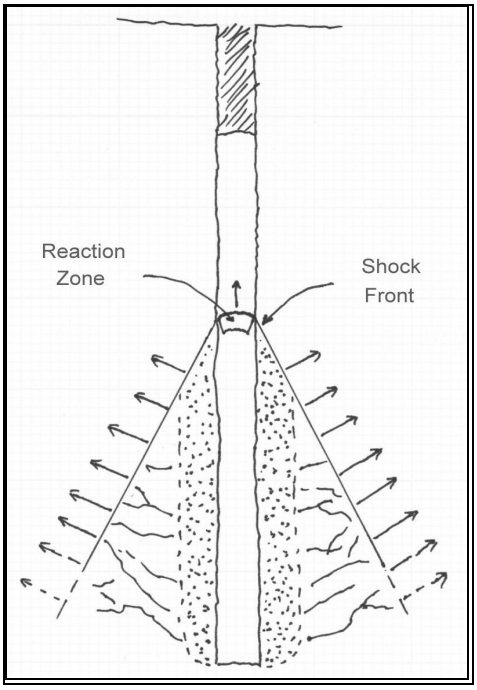


Figure 1

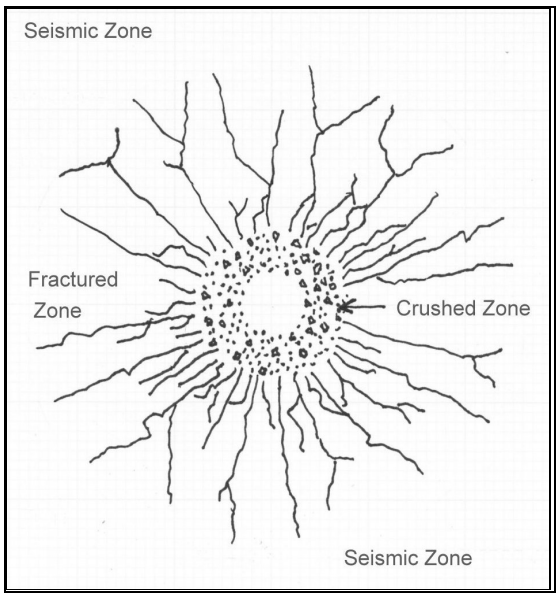


Figure 2

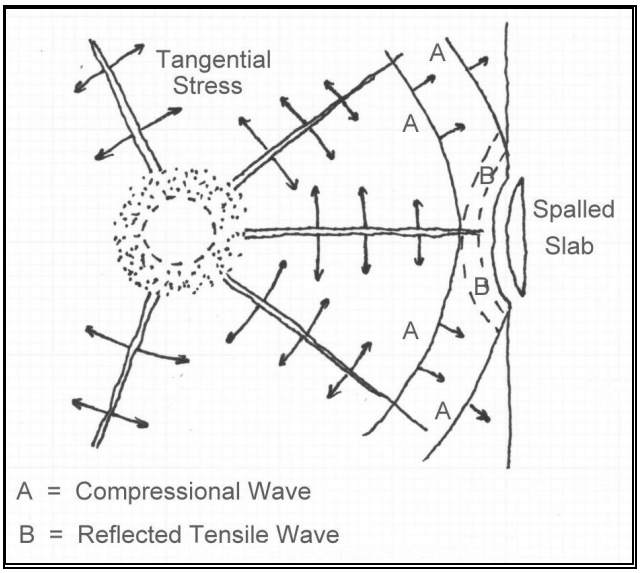


Figure 3

BLAST DESIGN CONSIDERATIONS

Blast design is not a precise science. Most successful blasters use some basic fundamentals in designing their blasts, but usually only achieve specific goals after gaining experience in blasting the particular type of rock at hand. The powder factors and other guidelines listed in this and following sections have been derived after reviewing the results of many blasting operations encompassing many variable factors. It is not intended that they be considered as recommendations for any particular blasting situation. The final responsibility for blast design will always rest with the blaster in charge.

A valuable tool for the blaster is the file of blast reports that he builds as he gains experience. Not only do these provide evidence of the quality of his work, but they also provide a wealth of information upon which he can draw as future blasting situations develop.

SHOT PLANNING

Before a blaster can design a blast, there are a number of site specific things that he must take into consideration that will have an impact on his design. If this is a site where he has been blasting for some time, for example a quarrying operation, he may only have to verify that nothing has changed since the last blast. For construction blasting and other blasting operations where conditions can change rapidly, or when blasting for the first time at a specific site, it is imperative that he exercise extreme care in identifying those conditions that would have an impact his designs.

Some considerations involve safety issues. Others pertain to such practical matters as fragmentation or explosive product availability. The experienced blaster may use a mental checklist, or quite possibly has developed a written checklist. Whatever the method, he should define at least the following items before he undertakes to design a blast:

A. Fragmentation desired:

Size of digging/handling equipment.

Size of crushing equipment (if required).

Rip-rap or dimensional stone desired?

Size limitations in project specifications?

B. Rock quality/character:

Hard? Soft? Porous?

Holes wet? Dry? Variable?

Joints and slip planes? Bedding planes?

Voids or other incompetent zones?

C. Site limitations:

Structures or other property to protect? At what distance?

Utilities nearby (underground or above ground)?

Vibration and airblast considerations?

Integrity of rock to be left in place.

On-site or off-site vehicle traffic?

Any other project specification limitations?

D. Safety limitations:

Adequate protection from flyrock?

Weather – is lightning a possibility?

Any nearby electrical hazards?

Any nearby RF (radio) hazards?

Impact hazards from rock fall?

Ventilation needed?

Traffic control required?

The impact of potential misfires. (How isolated is the site?
Is double-priming advisable to minimize misfires?)

E. Equipment / materials limitations:

Drilling equipment - size, condition.

Steel lengths available - depth of blast.

Explosives (including detonators) - Type, size, quantity available.

Adequate magazine site nearby?

Blasting mats available if needed?

Other blasting accessories?

Bear in mind that the above listing probably would not include ALL considerations for a specific site. Use your imagination. Investigate the area thoroughly and identify those items that will affect your blast or be affected by your blast and design accordingly.

BASIC BLAST DESIGN CALCULATIONS

In designing a blast, three principles should be kept in mind:

1. Explosive force functions best when the rock being blasted has a free face toward which it can break.

2. There must be an adequate void or open space into which the broken rock can move and expand (or swell).

3. To properly utilize the energy available, the explosive product should be well-confined within the rock.

If a blast is lacking in one or more of these three principles, the results will generally be less than desired.

Some years ago, the late Richard Ash gathered data from a large number of blasts and developed empirical formulas from that data to show the average relationships between hole diameter, burden, spacing, hole length, sub-drilling and stemming height. These relationships were later published in 1972 by the Bureau of Mines in information circular IC8550. With some modifications based on more recent information and on the author's experience, these relationships are presented here as the starting point for the initial design for a blast.

To facilitate understanding the relationships, some symbols and definitions are helpful (see also figure 4 on page 14):

D = Diameter (in inches) of the explosive in the borehole.

B = Burden, the distance (in feet) from a charge to the nearest free face in the direction that displacement will most likely occur.

S = Spacing, the distance (in feet) between two holes, measured perpendicular to the corresponding burden.

H = Hole length or depth (in feet).

J = Sub-drilling length (in feet), the depth that the hole extends below the anticipated grade or floor.

T = Stemming height or collar distance (in feet). The top portion of the hole containing inert materials intended to prevent premature ejection of gasses.

L = Bench or face height (in feet).

In these relationships, note that the Burden and Spacing dimensions are the "shot" burden and spacing which may or may not be the "drilled" burden and spacing. Changes in the initiation timing scheme will determine the difference. (See figure 7 on page 15.)

In the discussion that follows, it is important to understand that the blast parameters listed are inter-related and that changing one parameter will have an impact on others. For this reason, when any parameter is changed, it is important to re-analyze the blast design to determine that the remaining parameters are within acceptable ranges for the job at hand.

The burden that can be successfully blasted depends largely upon the strength of the rock and the amount of energy that is placed behind it. The amount of energy that can be loaded is dependent upon the hole volume, or diameter; hence, the hole diameter and rock strength largely determine the burden distance.

Often, the hole diameter has already been established by the drilling equipment on hand. If it hasn't, the optimum hole diameter should be selected based upon considerations such as fragmentation desired, bench height, rock quality, etc.

In selecting hole size, smaller hole diameters and tighter patterns will result in better fragmentation, but will increase drilling, loading and product costs. Taller bench heights will allow larger hole diameters and larger burdens and less drilling and blasting cost. (See Figure 5 on page 14.) Also, if the material to be blasted is blocky, it is quite likely that some blocks may emerge intact unless smaller hole diameters and tighter patterns place explosives within them. (See figure 6.)

Once the hole diameter has been established, burden distance can be selected. The following ratios can be used as first approximations in designing blasts. Bear in mind that the ratios will usually have to be adjusted as one learns more about how the particular rock reacts when blasted:

1. Burden = roughly 24 to 36 times the explosive diameter.

   Using AN/FO at a specific gravity of 0.82 g/cc:

light rock (2.2 g/cc density)	=	28 x diameter
medium rock (2.7 g/cc density)	=	25 x diameter
dense rock (3.2 g/cc density)	=	23 x diameter

   Using Slurries, Emulsions, etc at a specific gravity of 1.20 g/cc:

light rock (2.2 g/cc density)	=	33 x diameter
medium rock (2.7 g/cc density)	=	30 x diameter
dense rock (3.2 g/cc density)	=	27 x diameter

2. Spacing = 1.0 to 2.0 times the burden

holes shot instantly by row	=	1.8 - 2.0 x burden
large diameter holes shot sequentially	=	1.2 - 1.5 x burden
small diameter holes shot sequentially	=	1.5 - 1.8 x burden

3. Bench height = 1.5 to 4 times the burden, or possibly higher

   Bench height is usually limited on the low end by the height of the stemming column required and its limiting effect on the amount of explosive that can be loaded, and limited on the high end by the height of the digging equipment (for safety reasons).

4. Sub-drilling = 0.1 to 0.5 times the burden

flat bedding plane at toe	=	0.0 - 0.1 x burden
relatively easy toe	=	0.1 - 0.2 x burden
medium toe	=	0.2 - 0.4 x burden
difficult toe (vertical bedding)	=	0.5 x burden

5. Stemming column length = 0.5 to 1.3 times the burden

Increased multiplier if drill cuttings are used for stemming and/or holes are wet.
Decreased multiplier if stone chips are used for stemming and/or holes are dry.

For very cautious blasting (no throw or flyrock allowed):

Stemming = up to 36 times the hole diameter, possibly more

Stemming length between decks to be fired on separate delays:

dry hole = 6 times the hole diameter

wet hole = 12 times the hole diameter

In the foregoing parameters and relationships, a certain amount of caution must be exercised when selecting values. For example, too small a burden would result in excessive forward throw, while too large a burden would probably yield inadequate fragmentation with possible excessive upward throw. In a similar manner, too wide a spacing would result in loss of interaction between detonating charges, while too little spacing could cause partial cancellation of explosive forces and could contribute to excessive vibration.

The type of stemming material plays an important part in confining the gas generated from explosives detonating in the hole. Angular crushed stone chips are preferred. Round pebbles, dirt and water are not and should be avoided. Most of the time, drill cuttings are used, but they can be marginal.

If sub-drilling is not sufficiently deep, the result will be high bottom. Excessive sub-drilling, however, is wasteful of drilling labor and explosive energy.

All of these factors represent a judgment call on the part of the blaster designing the shot. Other factors such as initiation timing and direction have an impact and will have to be considered, as we shall see later.

POWDER FACTOR. In construction blasting, powder factor (PF) is expressed as pounds of explosive per cubic yard of material blasted. For mining, it is usually expressed as pounds of explosive per ton of material (or sometimes tons of material per pound of explosive).

To calculate the powder factor based upon an individual hole, use the formula:

$$\text{PF (lbs per cu yd)} = \frac{\text{lbs of explosive in the hole}}{B \times S \times H / 27}$$

(where Burden, Spacing and Height are measured in feet)

To calculate the powder factor for the entire blast:

$$\text{PF (lbs per cu yd)} = \frac{\text{total weight of explosive in blast}}{\text{total cubic yards of material blasted}}$$

To calculate the PF as pounds per ton of material blasted, you must first know the specific gravity of the rock that you are blasting and then convert the cubic yards to tons as follows:

Tons of rock = cubic yards x specific gravity x .8428

Some typical shot powder factors (expressed in lbs/cu yd):

Rock Class	MOH	Well shot	Medium shot	Poorly shot
Magnetite	9.0	1.90	1.45	1.05
Andesite	7.0	1.48	1.12	.91
Basalt	7.0	1.48	1.12	.91
Granite	6.5	1.38	1.05	.85
Serpentine	4.0	.86	.70	.52
Conglomerate	3.5	1.05	.85	.65
Sandstone	4.0	1.00	.80	.60
Shale	3.0	.90	.75	.54

Column designations in the preceding chart -

Well shot: roughly 90% 24 inch minus with max size 0.5 cu.yd.

Medium shot: roughly 90% 36 inch minus with max size 2.0 cu.yds.

Poorly shot: roughly 90% 60 inch minus with max size 5.0 cu.yds.

MOH: Scale of hardness (1 - 10)

Caution - Variations can be found in each of the rock classes above that could require a higher or lower powder factor to achieve similar results. The powder factors listed were from conventional production blasts. The geological characteristics of the rock play a very important role in fragmentation. For basalt in particular and for some granites, it should be noted that the rock structure (joints and seams) can greatly affect the powder factor required. Some columnar or diced basalt can be adequately blasted with only 75% of the powder factor indicated above. Additionally, other types of blasting may require powder factors that vary considerably from those listed. In tunnel driving it is not unusual to encounter powder factors as high as 4 to 5 lbs per cubic yard. At the other extreme, the blasting of boulders may only require a powder factor of 0.4 or 0.5 lbs (or less) per cubic yard. It is interesting to note that the overall average powder factor from all rock blasting combined is approximately 1 lb/cu yd.

Calculating Charge Weights.

Various means can be used to determine explosive charge weights. The most accurate when the specific gravity of the explosive is known, is to use the formula:

Pounds of explosive loading per foot of borehole = $\mathbf{D^2 \times S.G. \times 0.34}$

Where: **D** = explosive diameter (or hole diameter if using bulk loaded explosives)

S.G. = specific gravity (g/cc) of explosive

The factor **0.34** is used to convert g/cc to lbs/cu ft and to calculate the volume of the explosive charge. It is the result of the equation,

$$\frac{0.7854 \times 12 \times 62.4}{1728}$$

where: 0.7854 = ¼ of pi
12 = the number of inches in a foot
62.4 = the weight of 1 cubic foot of water in lbs (at approximately 45°F)
1728 = the number of cubic inches in a cubic foot

If you are using packaged explosives, you can divide the case or carton weight by the total number of cartridges. When using bulk loading equipment, there are usually gauges to indicate the amount of explosive that has been loaded.

Know what the loading is supposed to be for a given hole and then check the quantity closely. In situations involving limestone or other rock types with possible solution cavities, make sure you are not inadvertently filling a large void.

In critical blasting situations it is best to load cartridges and count them carefully. In most production blasting situations using poured or bulk explosives, it is more common to use some measuring method to determine where the top of the column is as loading proceeds. Bear in mind that variations in hole diameter will result in variations in the weight of explosive loaded. Be careful not to overload holes.

Test blasts are advisable for many blasting applications, especially when blasting for the first time at a specific site, or if geological conditions change. On many projects where there will be some critical blasts, it is preferable to start blasting at a point farther away from the critical area and then fine tune your blasts as you approach it. If in doubt as to the expected results, conduct a test blast.

The blaster's experience with various rock types and joint systems, coupled with his file of blast results, are two of his best tools for achieving good fragmentation.

Blast Hole Definitions

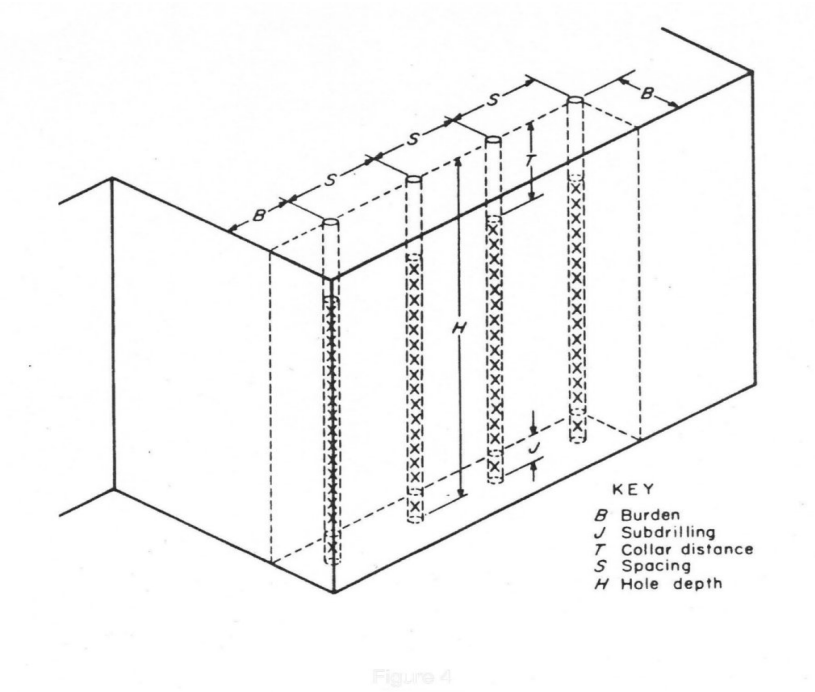


Figure 4

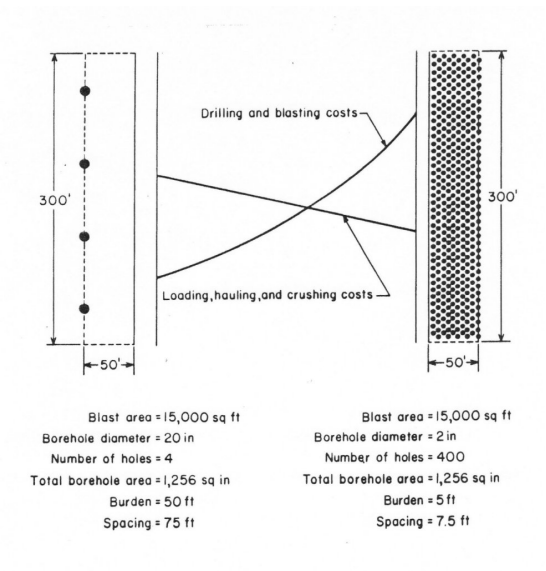


Figure 5

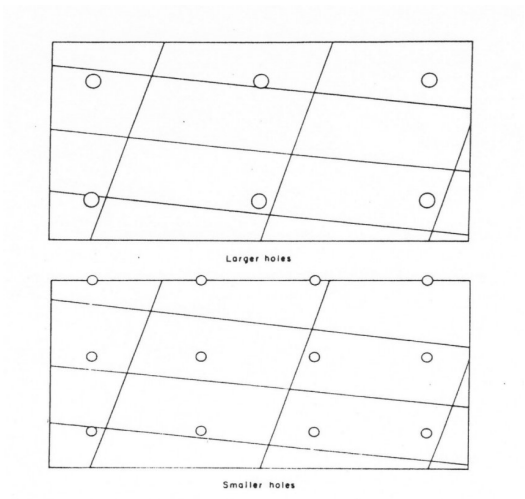
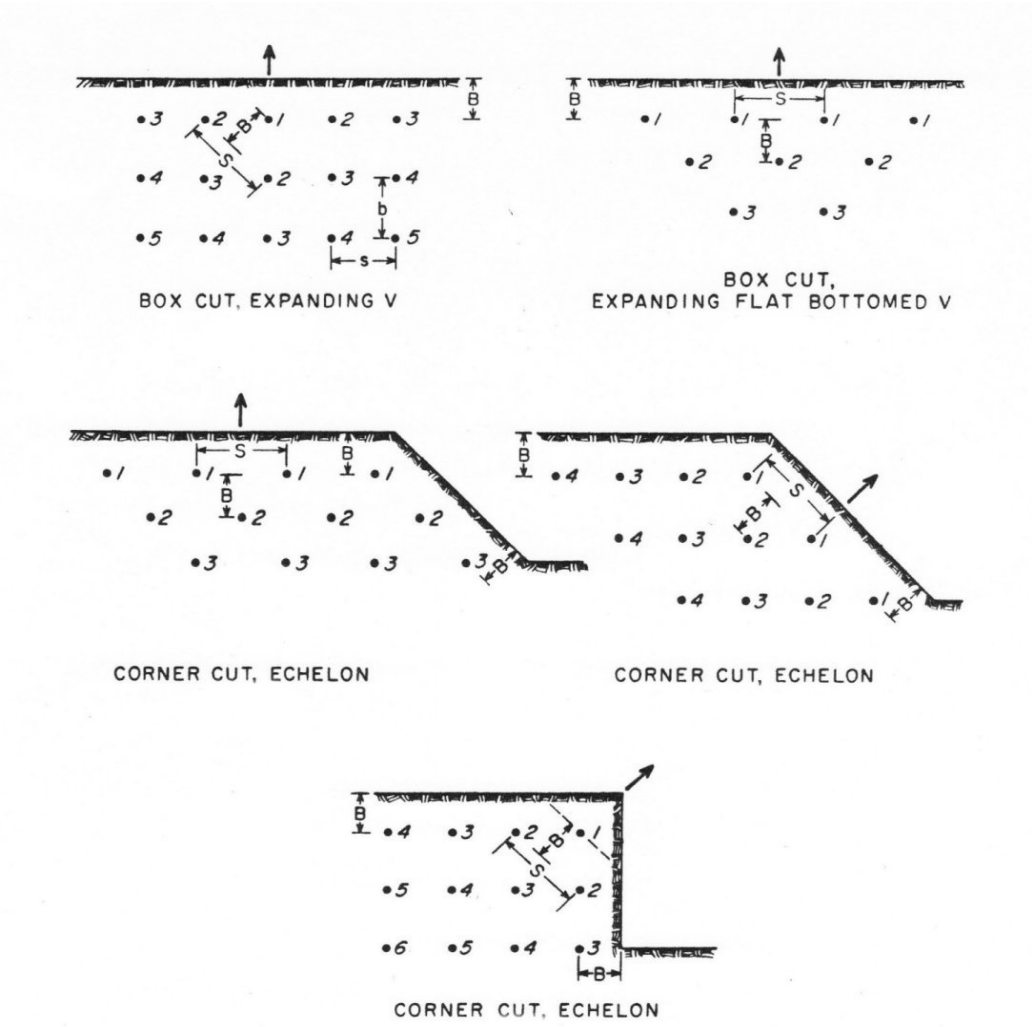


Figure 6

Blast Pattern Examples



B = Burden (shot) b = Burden (as drilled)

S = Spacing (shot) s = Spacing (as drilled)

1, 2, 3, etc = Initiation Sequence

Figure 7

Initiation Timing

Very seldom is a conventional blast set off where all charges are detonated in the same instant. Usually there is a specific time interval and direction or directions for delaying the charges.

For tunnels, drifts and shafts where there is no free face parallel to the axis of the holes, longer delay periods are utilized. These are intended to provide sufficient time delay for the fractured rock from the initial holes to be expelled so that there is room for the rock blasted by the following holes to expand. This type of blasting is somewhat specialized and will not be covered in this basic program.

In construction and in surface mining, millisecond delays are used between charges in a blast. There are several basic reasons for doing so:

1. To assure that one or more free faces progress through the shot, providing a consistent burden.

2. To enhance fragmentation between adjacent holes.

3. To reduce ground vibration and airblast.

4. To provide a means of directing the heave or displacement of the blasted material.

Theoretically, it is possible to "fine tune" the timing of a blast to achieve ideal results. Although rather sophisticated electronic detonators are available, standard millisecond (ms) delay systems can be obtained that will generally provide enough flexibility and a sufficient range of timing for most applications. There may be specific applications where extremely accurate delay detonators are necessary, but for most conventional blasting situations, the standard units are satisfactory. In many cases, a small amount of scatter in the times can actually be beneficial in reducing vibration, as long as the accuracy is adequate to prevent overlap, or near overlap, of detonation times.

As demand has grown for better fragmentation and reduced vibration, considerable research has been done with timing ratios and they way in which they relate to burden and spacing. The basic guidelines that have been developed in this area are:

1. The delay time between individual holes in a row:

The delay time between holes in a row should be between 1 ms and 5 ms per foot of burden, with 3 ms yielding good results in most instances.

Where airblast is a problem or potential problem, the delay time between holes in a row should be at least 2 ms per foot of spacing. This will result in a blast progression along the face or along a row of holes that is approximately half the speed of sound (or less) and reduces the low frequency airblast generated by face area movement or by surface area mounding.

(From a practical standpoint, the nominal time between delay numbers or some conventional surface delay device is often what is used for delay times between holes in a row and can be satisfactory for most situations.)

Where possible, corner holes at the end of rows should be given extra delay time because of the greater degree of fixation of the rock in those locations requires more time for the rock blasted by previously fired adjacent holes to move away.

2. Delay interval between rows:

The delay interval between rows should be from two to three times longer than the delay interval between holes in a row.

The last row in the shot should often be delayed slightly more than preceding rows. This serves to allow rock in previously fired rows time to move out and tends to reduce back-break in the rock behind the blast.

Cautionary notes: (1) Regardless of the delay times selected for holes in the same row or for the delay time between rows, it is absolutely essential that the delay intervals be sufficiently short that there is a buffer zone between a detonating hole and detonators that have yet to see their initiating signal. This is usually accomplished by using longer down-hole delays. Usually a buffer zone of three or four holes between a detonating hole and any holes that have yet to see an initiation signal is sufficient, but each case should be analyzed carefully in the blast design stages. Some people advocate that all detonators be activated before the first hole detonates. Such a requirement, when used with large diameter holes and wide patterns, usually results in detonation progression that is excessively fast, with rather violent results. Having all detonators activated before the first hole detonates shouldn't be necessary as long as a proper buffer distance to prevent cutoffs in the initiation system is maintained.

(2) An additional hazard can exist where delay times (compared to burden and spacing) are excessively long, causing cutoffs of the initiation system or powder columns due to ground shift. The actual interval where this can happen will vary depending upon geological conditions. Again, this needs to be analyzed on a case by case basis and accounted for during blast design.

9 millisecond delay criteria between detonating charges in a blast. The 9 ms criteria has been specified often enough that many persons accept it as the standard criteria for reducing vibration. Unfortunately, this is usually not the case. When the concept of Blast Scaling was developed, there was a need to define the term 'delay' in order to determine how much explosive was being detonated on any individual delay. Various delay detonators were tested as to their impact on vibration and other aspects of blasting. The shortest time interval available at the time was Dupont's 9 millisecond detonating cord connector. Following analysis, 9 ms was selected as the shortest interval to be used to define the number of holes detonating on one delay. Using the 9 ms criteria, all holes detonating within any 8 millisecond time frame are considered to have been detonated on the same delay.

9 ms is not the optimum delay timing for reducing vibration in most blasts. For blasts with large burdens and spacing, the optimum delay time may approach 19 ms to 25 ms, or possibly longer. On the other hand, for smaller close-in blasts, a delay time of 5 ms might prove to be the optimum, regardless of the fact that it doesn't meet a 9 ms criteria. A certain amount of common sense, coupled with good blast design concepts, has to be applied when determining the initiation timing to be used.

It should also be pointed out that there are extenuating circumstances where other timing criteria need to be applied. An example would be a 'sinking cut', where there is no free face toward which the rock can break. A sinking cut is used to open a new lower level in a quarry floor. A greater delay time must be used between the opening holes detonating and the next succeeding holes. This is usually accomplished by skipping one or two delays. This time is required to allow the rock broken from the opening holes to be expelled upward, providing relief for the following holes. (Because of the closely confined conditions in such a shot, it is common practice to double-prime several rows of holes surrounding the opening holes to counteract the tendency for cut-offs in the explosive columns. Additionally, one must plan for some flyrock in this type of shot.)

In blasting for tunnels, shafts and drifts the longer delays that are used are measured in seconds or half seconds rather than milliseconds. Again, the use of such long delays it to allow time for broken rock to be expelled from the face.

Direction of heave or throw. It is generally possible to control the direction of heave of the material from a blast through application of the initiation system timing sequence. In figure 7 on page 15, an arrow shows the directions of most logical heave when the various delay sequences shown are used. The numbers in the various figures represent the initiation sequence. Shooting row by row will generally lay the muck out in front of the shot. Shooting with a V-cut timing pattern will usually result in a muck pile that tends to mound up in the center in front of the shot. The method of digging out the shot will usually determine which is preferable.

The direction of maximum vibration (all other things being equal) will theoretically be in the direction opposite from the direction of heave. To notice a difference in vibration levels in this manner however, the geology would have to be similar in all directions from the blast.

Obviously there are many different delay patterns that have proven to be successful and to include them all in this basic treatise would not be possible. Blasters are usually willing to share their knowledge of successful blast designs and patterns and it would be wise for all blasters to document what works and what doesn't for various blasting situations.

Location and Orientation of Primer

In most instances the priming charge will be located at the bottom of the hole. If the priming charge was located at the top of the powder column, the energy would break through the surface earlier in the explosion process, gasses would vent sooner and much of their contribution to the fragmentation process would be lost. An exception might be the shooting of post holes or other similar situations where the priming charge is located at the top of the explosives column. In shooting such holes it is desirable to break and expel the rock upward and side breakage is usually not desired. If in doubt, for conventional blasts it is usually preferable to bottom-prime.

The orientation of the detonator in the priming charge should be such that the detonator is pointing in the direction of the explosives column. I.e. the detonator would be pointing upward in a bottom-priming charge and downward in a top-priming charge. The reason for this is that, due to their construction, most detonators are directional. The energy starts at the ignition point and then progresses through the various stages in the detonator until the base charge is fired. Detonators will initiate explosives from the side, but are more reliable in the direction of the base charge end of the detonator.

This orientation of the detonator also applies to the initiation of detonating cord. The detonator would normally be taped to the cord facing in the direction that the cord is expected to detonate.

Added note

In the author's investigative experience, the two leading causes of blasting problems are:

(1) blasts loaded too heavily

(2) initiation timing that is too fast for conditions

These causes can be readily eliminated by using powder factors that are correct for the site conditions and by using timing schemes that fit the above criteria.

Explosive Selection

In developing guidelines for selecting the proper explosive**,** it would prove beneficial to understand some of the qualities and characteristics of various types of explosive, and to relate their importance to the task at hand. Some characteristics will be very important, while others may not.

Size - Obviously, it has to fit in the drilled hole. Size is usually selected based upon the size of drill to be used, rather than the other way around. This is a moot point if pumped or poured products are used.

Water Resistance - This can be quite critical. If holes contain water, many variants of AN/FO cannot be used, along with some cartridge products. Water resistance can be gained from packaging or from formulation of the product. Water will affect a detonating hole in two ways: First, it can dissolve some of the ingredients, causing the product to partially or completely fail. Second, presence of water has a cooling or quenching effect on the detonation process. Some manufacturers rate their products' water resistance as poor, fair or good. One manufacturer used to indicate how many hours it could be expected to withstand water before being affected. If in doubt as to the adequacy of the product, seek the advice of your explosive supplier.

Sensitivity - Normally one would expect that an explosive product should be as sensitive to initiation as possible in order to assure detonation. This can be largely true, but there are exceptions. In those instances where holes are fairly close together and contain water, a less sensitive explosive may be required to keep adjacent holes from being prematurely fired by water hammer from a detonating hole. Some refer to this effect as "flash-over" or "sympathetic detonation"; others use the term "propagation". (Propagation normally refers to detonation proceeding through an explosive column, hence it is <u>not</u> the preferred term.) When a group of wet holes in this situation are all expected to detonate on separate delays, but instead detonate sympathetically, the results can be extremely violent.

Strength - Formerly, the strength (energy output) of a particular dynamite was rated at a percentage in comparison to a cartridge of nitroglycerin. For example, a 65% strength cartridge of semi-gelatin dynamite had the same energy content as a similar sized cartridge that was 65% nitroglycerin, even though the semi-gelatin only used a small amount of nitroglycerin as a sensitizer. To compound the matter, there was weight strength (energy per unit weight) and bulk strength (energy per unit volume). Unfortunately, some blasters tended to think that the higher the percentage, the better. This was not necessarily the case. In most situations requiring careful blast designs, how the explosive goes about doing its work is usually much more important than how much energy it has.

Today, explosives are usually rated for energy in calories per gram and/or calories per cubic centimeter. Often they are rated against an equivalent volume (bulk strength) or weight (weight strength) of AN/FO. Again, energy may not be as important as the detonation velocity and/or density of an explosive, depending upon the particular application.

Detonation Pressure - Detonation Pressure varies more with changes in Detonation Velocity than other factors, hence some discussion of it will also occur under that term. One situation where high detonation pressure does become very desirable is in the selection of a booster to initiate a column loaded with a relatively insensitive product. A booster with a high detonation velocity (hence, high detonation pressure) is desirable in order to properly initiate the product. Most explosives have a fairly consistent detonation velocity based upon diameter, confinement, density and formulation. If a booster has a higher detonation velocity than the column load, the detonating column will start to detonate close to the higher rate, but will drop back to its steady state velocity within a couple of borehole diameters.

Conversely, if a booster with a lower detonation velocity than the column load is used, the explosive column will start at the reduced rate (if it starts to detonate at all) and will eventually achieve its steady state velocity, but it will take a considerably longer distance to do so. Bear in mind that for bottom initiation, this would occur in the zone where you need maximum energy and, with reduced explosive performance there, you probably won't get adequate breakage.

Density – Usually is not a problem unless you are trying to load a very light product in wet holes. For example, if you package AN/FO at an average density of .82 g/cc into waterproof sleeves and try to load it in wet holes, it will float. Virtually all explosives with good water resistance also have densities that are higher than water, i.e. greater than 1.00 g/cc.

Detonation Velocity – Sometimes misunderstood by some blasters who may think that a higher velocity product will always be superior. Granted, it can be in those situations where a hard, brittle rock is being blasted. A higher velocity product is always preferred for seismic exploration work where a good sharp signal is the goal. For most average blasting situations, however, something less than the highest velocities available could be the preferred product. To investigate this further we need to look at the term "impedance matching".

Impedance Matching or, more accurately, Impedance Approximating -

The concept of impedance matching has been a fundamental principle in electronics for many years. While it differs in some respects from electronics, the concept of impedance matching can be applied to the process of transferring energy from the explosive into the rock.

The impedance of an explosive is represented by its shock energy production rate. The impedance of rock is represented by the rate at which it can accept the transfer of shock energy.

Very roughly (for conceptual purposes),

> Explosive impedance would equate to the density of the explosive multiplied by the detonation velocity of the explosion.
>
> Rock impedance would equate to the density of the rock multiplied by the velocity of sound in rock.

Since the impedance of a given piece of rock is fixed, any attempt at impedance matching would obviously have to entail the selection of an explosive that would more closely match the impedance of the rock. Because calculated impedance values in rock are usually far higher and have a much greater range than those calculated for conventional explosives, a better name for the concept of impedance matching might be **impedance approximating**.

To take advantage of the concept, the blaster would select an explosive with a lower impedance value (lower density, lower velocity) when attempting to blast rock with a lower impedance value (lower density, lower velocity) and, conversely select a higher impedance explosive to blast rock with a higher impedance value.

While impedance approximating will assist you in achieving better blasting results, the structure of the rock (joint systems, etc.) will play a very important part and will usually have a greater effect on blast results. Study the rock structure carefully. Consider it in your blast designs and then select your explosives to match the rock.

Cautious Blasting

When blasting to a critical final slope or to preserve a competent rock face, cautious blasting techniques may be required. In some instances the remaining rock face will be expected to stand on its own for the foreseeable future. In other instances, excessive back-break can result in having to place more concrete than might otherwise be necessary.

The two most common cautious blasting techniques are **pre-splitting** and **smooth blasting**. The loading of these two methods can be similar, but the order in which they are initiated is what sets them apart. Both methods use lightly loaded holes on relatively close spacing for the perimeter of the excavation.

In **pre-splitting**, holes along the perimeter line are detonated prior to the main blast (sometimes as a separate blast) and the resulting fracture forms a pre-split line that presents a barrier to fractures and allows expanding gasses from the production blast to vent rather than penetrate the preserved wall.

In **smooth blasting**, the perimeter holes are detonated last. The smooth blast holes are also lightly loaded and use a reduced burden to prevent back break into the remaining rock. In either of the above methods, it is sometimes necessary for the nearest production holes to be more lightly loaded than the rest of the production holes in the blast.

The choice between the two methods is usually dictated by whether or not the bulk of the rock to be blasted is needed to support the remaining rock while the shearing action is taking place. If the rock is of marginal quality it can be advantageous (although not uniformly successful) to leave the bulk in place and pre-split the face. In good quality rock, either method can usually be used successfully.

When using the pre-splitting method, it is very important to have sufficient burden against which to shoot. Not only will the results be better, but if too little burden exists, pre-splitting can result in violent flyrock.

In cautious blasting, the level of success is greatly dependent upon the quality of the rock, and no amount of blasting expertise can make up for extremely poor quality rock. Assuming the loading is not excessive, the geology will be the largest variable and will have the most impact on the final results. If the rock is competent, a good face will result. If the rock lacks sufficient strength or the main joint system runs somewhat perpendicular to the desired face, the results will probably not be as good as desired. Blast-generated cracks will quite naturally break to the nearest joint(s) or voids.

In both pre-splitting and smooth-blasting, loading can be accomplished with (a) special long, slender cartridges made for the purpose, (b) cartridges of conventional explosives taped at intervals to detonating cord, or (c) just detonating cord alone. The author prefers the use of detonating cord alone (in one or more strands of various core loading sizes), mainly for the closer control of the effects that it provides.

It is usually beneficial to stem the tops of pre-split holes. In some instances using detonating cord as the main charge, the pressure rise time in the hole is fast enough that a crack is propagated from hole to hole before the pressure can vent from an un-stemmed hole. If stemming is used, some form of hole plug should be placed on top of the explosive charge to prevent stemming from packing around the explosive.

Unfortunately, some specification writers and inexperienced personnel are of the opinion that, regardless of the quality of the rock, decreasing the spacing between holes and loading them progressively lighter will eventually achieve the desired result. This is usually not the case.

Powder Factors for Cautious Blasting

As defined earlier, cautious blasting of perimeter holes involves using reduced charges, normally on tighter centers, to shear the rock along the perimeter. The actual charge density to obtain the desired result will vary depending upon the quality of the rock and the jointing system involved.

Instead of the usual pounds per cubic yard or per ton of material, pre-splitting and/or smooth-blasting powder factors are usually expressed in weight of explosive in pounds of explosive (or grains of PETN in the case of detonating cord) per square foot of sheared area or wall area to be preserved. For example, if pre-split holes are drilled on 24" centers and each hole is loaded to the collar with 200 grain/foot detonating cord, the powder factor would be 100 grains per square foot. (Each foot of detonating cord would be expected to shear an area 12" by 24", or 2 square feet. 200 grains divided by 2 square feet equals 100 grains per square foot.)

Bearing in mind that test blasts will usually be required to fine tune the actual charge density, the following loadings will provide a starting point for testing.

When using detonating cord, it is usually preferable to start with a low figure such as 75 grains per square foot. If this is insufficient to crack the rock, the holes usually will still be available to be re-loaded after they have cooled. (50 grains per square foot will usually only crack the weakest rock. 200 grains per square foot will almost always result in severe flyrock, especially in cases with reduced burdens.)

When using explosive cartridges, either string-loaded or with spacers, the correct charge density should fall somewhere in the range of 0.07 lbs per square foot to 0.15 lbs per square foot. Test blasts using 0.10 lbs per square foot of perimeter area would be a reasonable starting point.

If a slightly heavier charge is loaded only to help pull the bottom, it can be added to the calculations, although many blasters do not, feeling that the bulk of the shearing is done by the string-loaded charge. Obviously, any bottom charge should be shown in the blast diagram and would be included in the total explosive weight for the shot.

Mechanism Involved in Pre-Splitting or Smooth Blasting

The actual mechanism for **pre-split** shearing is the pressurizing of all the holes simultaneously in a given line segment of the perimeter. When the pressure causes strain that exceeds the tensile strength of the rock, a crack will form, trending generally along the row of holes. Bear in mind, that the rock joint system will cause the sheared surface to range anywhere from flat to saw-toothed, depending on the orientation of the joints. Usually a competent surface is what is desired. If the surface is flat or reasonably smooth, so much the better. (Examples of pre-splitting and smooth-blasting can be found in road cuts in many states. In many of these, one can see half casts of the perimeter holes. The author tends to look upon these as evidence of blasting skill, although current environmental thinking is leaning toward making the remaining rock surface look "more natural".)

When using the smooth blast method of blasting to preserve a final rock slope, the reduced burden, coupled with the lighter charge weight assures that the final rock can be blasted away without adversely impacting the final slope or wall.

Charge densities for **smooth blasting** will normally have to be slightly higher than that used for **pre-splitting**. One must be careful, however, because the reduced burden in **smooth blasting** can result in violent throw unless adequate precautions are taken to prevent it.

For best results when using cartridge explosives, hole diameters should normally be at least twice the cartridge diameter.

Note: In all cases, test blasts are advisable. In addition, all safety requirements must be considered, even though these may not be "production" blasts. Violent flyrock may occur. Higher levels of vibration may be experienced. If in doubt, obtain the assistance of a qualified, experienced blaster.

Other Cautious Blasting Techniques

Several other methods for achieving cautious blasting results have been tried in the past. Cushion blasting was tried with mixed results in the 1960's. In cushion blasting a de-coupled charge is loaded and the void surrounding the charge is filled with inert stemming material. Results using this method were so unpredictable that one major powder company would not allow its representatives to use it or to recommend it. Another method that was tried was axial-loading, which also incorporated de-coupled charges held to the center of the hole with spiders or axial spacers, but without the inert stemming material. Decent results were observed using this method, but it was somewhat labor-intensive. Still another form of pre-splitting was the use of a gas-generating explosive charge loaded only in the bottoms of the holes. The remainder of each hole would be left empty except for some form of plug at the collar to prevent the gasses from venting. The two methods that have survived through the years however have been conventional pre-splitting and smooth-blasting.

Some research has been done in trying to "steer" the pre-split line by cutting grooves in opposite sides of the borehole wall in line with the desired pre-split. This was accomplished using special bits. While this appeared to be beneficial in some of the tests, the rock joint system remained the main controlling factor and the additional work and equipment required to cut the grooves pretty much offset any possible benefit.

Geology and its Effect on Blasting

To address how geology can have an impact on a blast, we need to look at some of the properties of the rock to be blasted. Quite a few years ago, the late Bill Burkle, who was then head of Hercules' Technical Services Division, put together a presentation, *Geology from the Blaster's Viewpoint*. Bill's observations were contained in various Hercules workshops and publications and were also the basis of a paper that he presented in the early years of the SEE. Much of what follows was taken from notes of the author's conversations with Bill and from his presentations and illustrations, although a portion is from the author's observations and experience in the field.

The technical background and terminology involved in the description of rock masses that are of importance to the geologist need not be so important to the blaster. In other words, you don't need to know exactly how the rock got to be the way it is or its official name, as long as you recognize, in your own terms, what it is and how it is put together. A good blaster should be able to recognize similar rock when he encounters it again and his archive of field notes and blast reports should indicate how he blasted it previously and what results were obtained.

There are five geological properties of rock that greatly influence rock 'blastability'. In order of importance, they are: (1) **Structure**, (2) **Resilience**, (3) **Strength**, (4) **Density** and, (5) **Velocity of Energy Transmission**. While some may feel that drillability is important, ease in drilling does not necessarily relate to ease in blasting. In fact, very often rock that is hard to drill (because of strength or density) may blast quite easily while soft rock that drills very easily may not blast well because it is too resilient.

1. Structure: The structure of rock is described in terms of strike, dip, jointing systems, faults, mud seams and grain size and orientation. The strike and dip are described in figure 8. Strike is always measured as a horizontal direction. Dip is the vertical angle measured on a plane that is perpendicular to the strike. If possible, it is usually preferable to align your blast face so that you are shooting with the dip (figure 9). This will usually result in a smoother bottom and will pull the toe better. Aligning the face to shoot against the dip (figure 10) has the opposite effect, although somewhat less backbreak will be experienced. Shooting against the strike (figure 11) usually results in irregular bottom with areas that will backbreak and may present you with an irregular face for the next shot. If some backbreak can be tolerated, shooting with the dip will usually provide the best results.

Joint systems can be found in many rock masses that can be put to good use. These can be described as major and minor joint systems that usually intersect at other than, but sometimes close to, right angles.

If you have a fairly well defined joint system, the face should be oriented so that it is parallel to the major joint system (figure 12). You will find that rock is more easily blasted when you are attempting to pull it out of a corner where the joint systems meet at greater than 90 degrees. If you try to shoot the rock from a joint corner that is less than 90 degrees, you can usually expect more back-break and possible violent throw.

If you are lucky enough to be blasting columnar or diced basalt in one of the many "table" mountains that exist in the west, the joint systems will generally make you look good. Good examples can be seen in California near Sonora and near Mammoth. The rock is already sized conveniently and all the blaster needs to do is "liberate" it with his blast.

Microstructures such as grain size and grain orientation and the matrix that holds the grains together will have an impact on your blasting, but there isn't much you can do specifically to adjust to these properties other than adjusting your powder factor.

When cemented cobbles are encountered, blast results may vary from good to quite difficult. In some instances the matrix turns to sand and the cobbles run free. At other times, the shock waves from the detonation process are reflected and/or refracted by changes in density and the mass becomes quite difficult to blast. A test blast would be advisable in most of these situations.

Where soft rock is interspersed with harder material or where you encounter mud seams, you should adjust your loading schemes to put the energy where it does the most good. Energy that is loaded in weak zones or in mud seams does not contribute much to the fragmentation process, but can contribute to airblast and flyrock. Drilling logs are a valuable source of information when working in these conditions. A similar condition exists when you are blasting rock that overlays softer material. An occasional deeper test hole should be drilled to determine the rock depth. Explosives should not be loaded too near the bottom of the rock/soil interface or they may prematurely break through the bottom and vent gasses downward, losing much of the energy.

2. Resilience. This refers to the elasticity of the rock and represents its ability to resist a shock wave and recover its original shape without fracturing. If tapping a rock with a hammer or banging two like pieces together results in a dull thud, the rock may be somewhat harder to blast than rock that yields a ringing sound after a similar blow.

3. Strength. By strength, we mean tensile strength rather than compressive strength. As discussed in the physics of rock breakage earlier, you will recall that we are wedging rock apart with hot expanding gasses rather than smashing it. The compressive strength of rock is usually 7 to 10 times greater than its tensile strength. This is why you need 7 to 10 times more explosive when you attempt to mudcap (blast with external charges) a boulder as opposed to drilling a hole in it and blasting it conventionally. Most blasters don't know what the tensile strength of their rock is and it isn't easy to measure. While strength affects blastability, it is not as great a factor as the rock's structure. Adjustments might include powder factor, explosive velocity (recall the previous comments under **Impedence Matching** and see 5 below) and to a lesser extent, timing.

4. Density. When explosives detonate within rock, the useful work is apportioned between fracturing the rock and displacing it. The denser the rock, the more energy is required to displace it. A cubic foot of material to be blasted may weigh from 85 pounds in the case of bituminous coal to 315 pounds for magnetite, but probably averages around 160 to 180 pounds per cu. ft. for most rock. You will probably need to increase the powder factor if you are trying to get good heave with dense rock.

5. Velocity of Energy Transmission. Rock velocities can range from about 5000 ft/sec to over 20,000 ft/sec. Usually the higher velocity rock is fine-grained and responds well to higher velocity explosives. Conversely, lower velocity rock (talc, shale, some sandstones, etc.) can usually be blasted more successfully with lower velocity explosives.

Additional Notes on Geology.

The blaster's best friend should be his driller. Usually the driller knows more about the rock (from drilling it) than anyone else on the project. In difficult blasting situations, the driller should log his holes, indicating where voids, weak zones and obvious changes in rock quality occur. The blaster can then use this information to design his blast for improved results.

Strive to learn more about the rock that you blast. Make note of the jointing systems and how they may have affected the outcome of your blasts. Keep this information with your archive of blast reports. Include information on the blast report pertaining to how the rock reacted to your blast designs. It can prove to be quite valuable to you later in your career.

For those interested in pursuing the subject of geology and its impact on blasting, *Explosives Engineering, Construction Vibrations and Geotechnology,* by Lewis L. Oriard (published by the International Society of Explosives Engineers) is an excellent source for additional information. Chapters 1 and 2 are particularly good in their treatment of the subject.

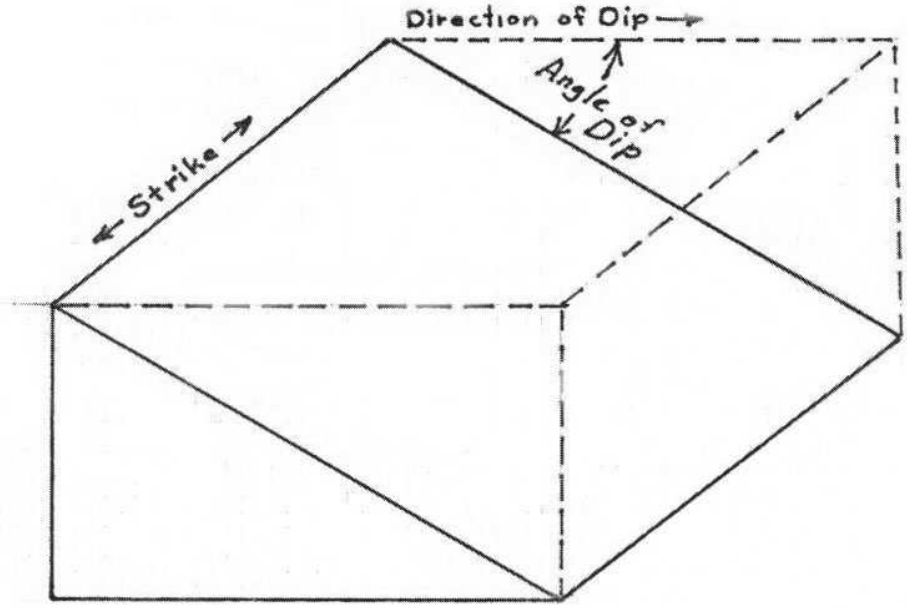


Figure 8

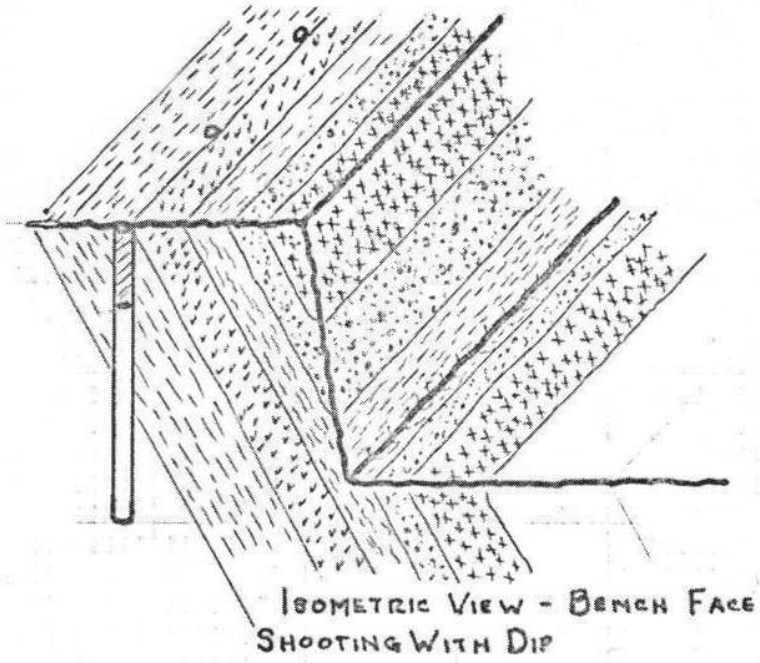
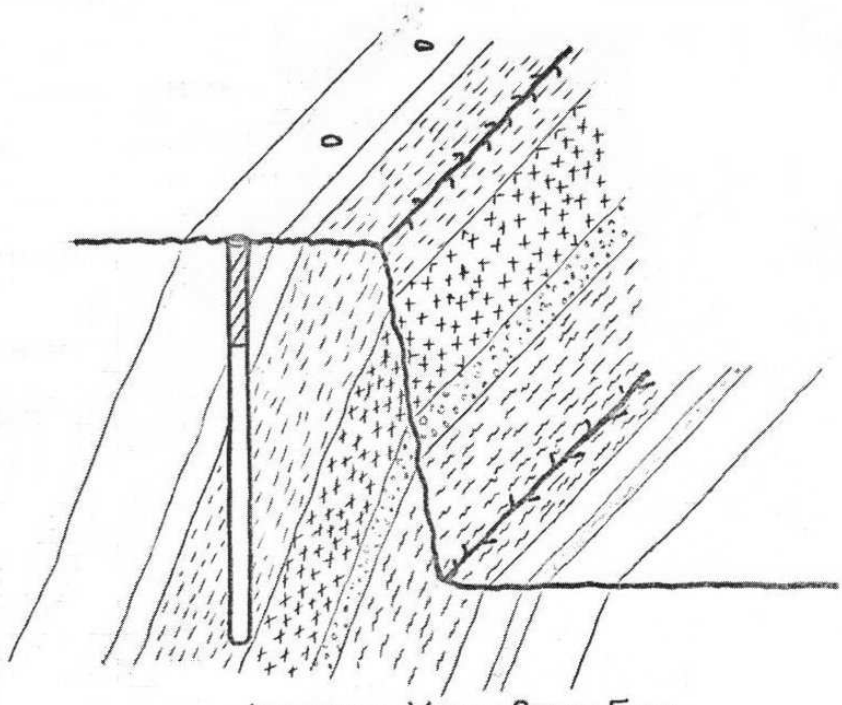
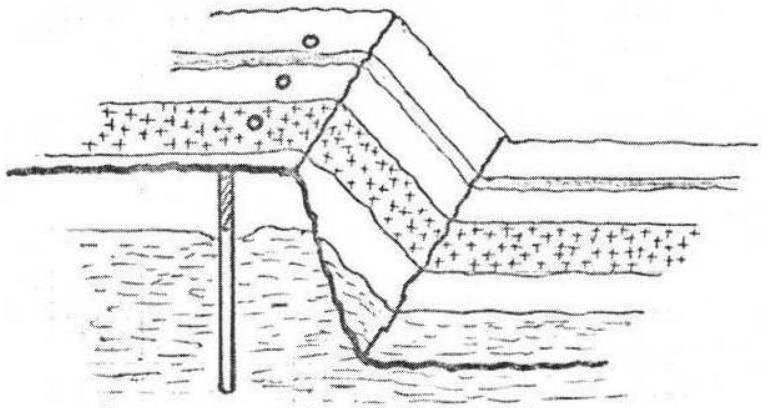


Figure 9



ISOMETRIC VIEW - BENCH FACE
SHOOTING AGAINST DIP

Figure 10



ISOMETRIC VIEW - BENCH FACE
SHOOTING AGAINST STRIKE

Figure 11

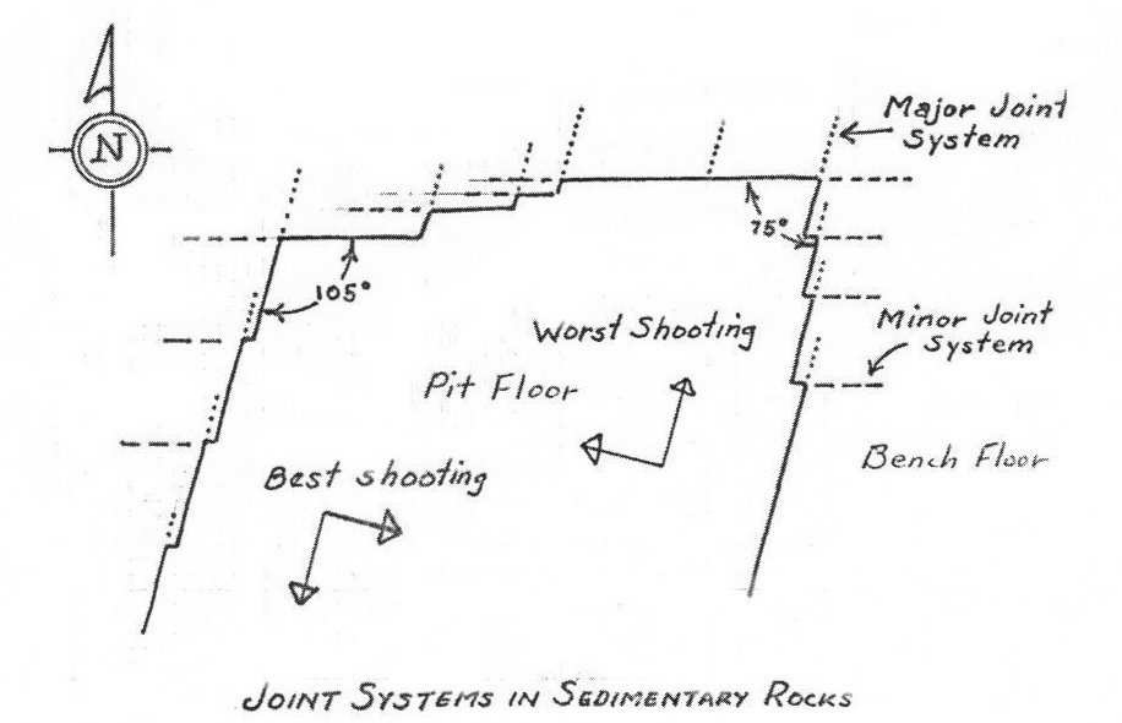


Figure 12

(Author's Note: Figures 8 through 12 are Bill Burkle's original drawings and have been retained in this document to honor Bill's contributions to the author's knowledge and to the knowledge of many of his other friends, associates and blasters.)

Blast Documentation

As a blaster starts his blast design process, he should also initiate his blast documentation, or Blast Report. This document serves to record, in detail, the exact layout, loading, timing, etc. of the blast.

As the blast is drilled and loaded, any variations from the initial blast design should be noted on the Blast Report.

If seismograph equipment is used to record vibration and airblast, the location of the seismograph should be shown and some reference made so that the seismic report and the blast report can later be correlated. Using the date and time that is recorded on the seismic records is one good method.

Following the blast, notations should be made as to the results obtained and also for unusual occurrences, if any.

Even though similar blast designs may be used for several or even many blasts, the blaster should refrain from just copying previous reports and changing the date. Such documents would be very suspect in legal situations. A signed Blast Report, in the blaster's own handwriting (especially with finger smudges), would be far better than any computer-generated report.

The specific form of the Blast Report may vary from blaster to blaster, but the data included should be the same. In all instances, sufficient data should be recorded on the report such that another blaster could duplicate the blast in every aspect, without needing any additional information. If he needs additional data to do so, that information should have been included in the original report.

Completed Blast Reports should be kept in a permanent file in the project office or mine/quarry office. These may be needed at some future date should there be any claims of damage from the blasting.

It is also a good idea for the blaster to make a copy for his or her own personal records. These will provide a wealth of information for future use. He will have good information to fall back on should he encounter similar blasting conditions in the future.

Remember, the job isn't complete until all the paperwork is done. As in most things related to blasting, don't take any shortcuts.