September 8, 2005

To: Montana Blaster Manual Recipients

All blasters must be certified to conduct blasting operations at coal mines in Montana. Prior to conducting a blast at a coal mine, the Blaster must complete a training course covering the topics outlined in A.R.M. 17.24.1262, Appendix B and using this manual, or using this manual as a self-teaching guide, and successfully pass the Blaster's Certification Exam. Anyone seeking information regarding courses should contact the Department of Environmental Quality, Industrial and Energy Minerals Bureau, PO Box 200901, Helena, MT 59620-0901, phone 406-444-4975 or 406-444-4970. If there are any questions as to the qualifications of a particular course, please contact the Department or submit a course outline and the instructor's name, address, and phone number to the Department prior to taking the course.

Once a training course and this manual have been completed, the Blaster must submit an application and affidavit (see Appendix F) to the Department certifying that a training course and manual has been successfully studied and completed, prior to taking the blaster exam. Upon receipt of an application, the Department will verify that the course completed satisfies the requirements of ARM 17.24.1262 and will inform the applicant of the date, time and place of the exam. Additional application and affidavit forms are available from the Department at the above address.

The Montana Blaster's Certification is valid for a period of 3 years. A minimum of 16 hours of refresher training is required during this period for recertification. Application and Affidavit for recertification can be obtained at the above address and must be filled out and submitted to the Department at least 60 days prior to expiration. If a recertification exam is needed, the applicant will be notified of the time, date, and place.

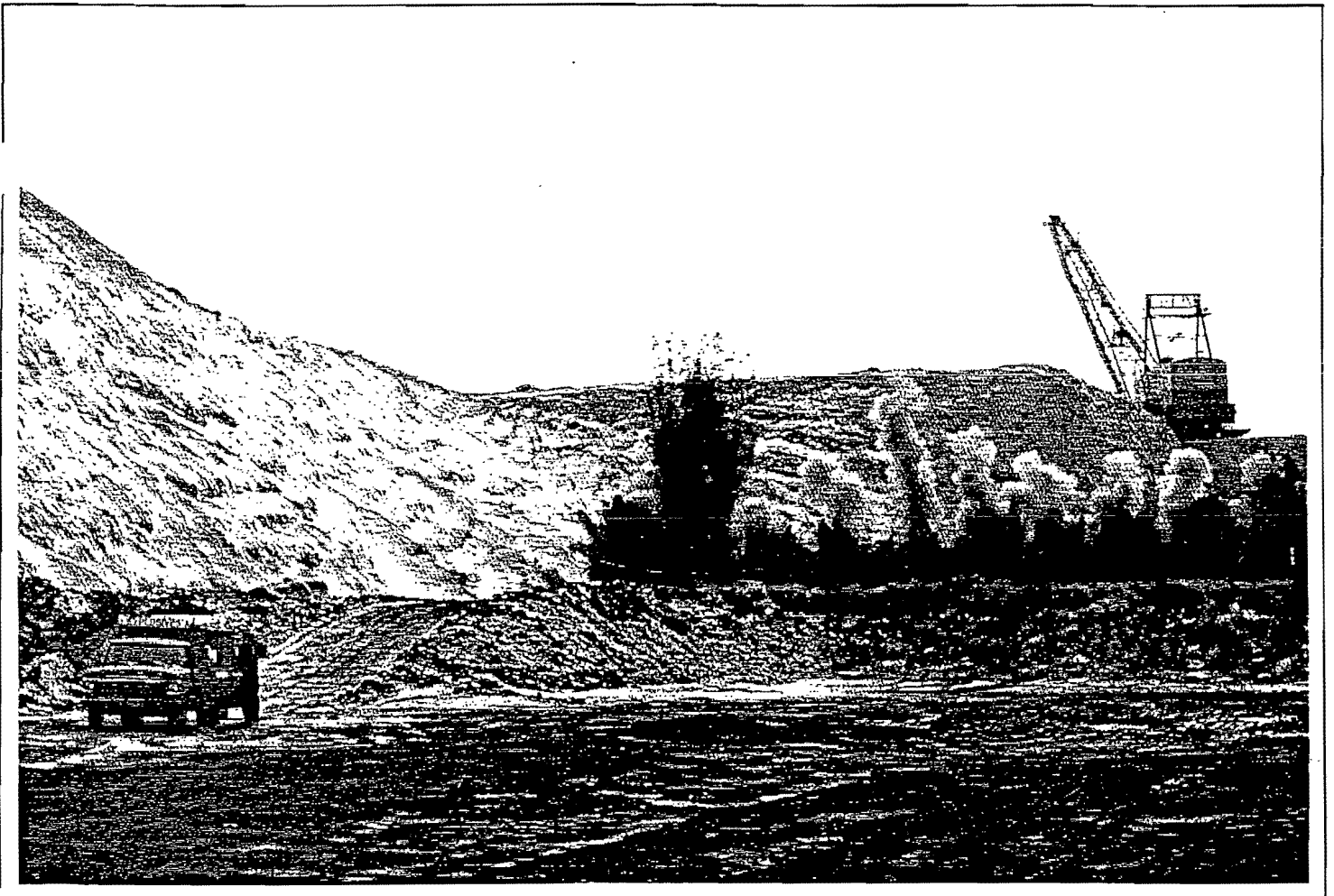The 2004 Coal Mine Blasting Rules have been added to this manual

MONTANA
BLASTER CERTIFICATION
TRAINING
MANUAL

Montana Department of State Lands

Second Edition

Montana

Department of State Lands

Blaster Certification

Training Manual

1989

MONTANA'S BLASTER CERTIFICATION
TRAINING MANUAL

ACKNOWLEDGEMENT

Special thanks go to Richard A. Dick, Larry B. Fletcher, and Dennis V. D'Andrea of the Twin Cities Research Center, Minneapolis, Minnesota, who developed a major share of this training manual. Also, thanks to the Department of Mines in the State of West Virginia who contributed with review questions and answers, and to Lynn Woomer for the cover photo.

The Montana Department of State Lands acknowledges the work of Dave Paszkiet, Sharleen Pendergrass, Claudia Furois, and Jim Dushin in revising this manual.

DISCLAIMER

None of the material in this manual is intended to replace manufacturers' recommendations on the use of the products involved. We strongly recommend that the individual manufacturer be consulted on the proper use of specific products. Also, reference to specific trade names or manufacturers does not imply endorsement by the Bureau of Mines or the State of Montana.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

mp.....ampere	°F.....degree Fahrenheit	lb.....pound
cm.....centimeter	fps....foot per second	mi.....mile
cu ft..cubic foot	g......gram	ppm....parts per million
cu yd..cubic yard	Hz.....hertz	psi....pound per square inch
dB.....decibel	Kb.....kilobar	sq ft..square foot
°......degree	kcal...kilocalorie	yd.....yard

Introduction

This training manual covers the latest technology in explosives and blasting procedures. It includes information and procedures developed by Bureau of Mines research, explosives manufacturers, and the mining industry. It is intended for use as supplemental training and also to provide experienced blasters a yearly update on the latest state of technology in the broad field of explosives and blasting.

The need for better and more widely available blasters' training has long been recognized in the blasting community. The Mine Safety and Health Administration (MSHA) of the Department of Labor requires health and safety training for blasters. In 1980, the Office of Surface Mining Reclamation and Enforcement (OSM), Department of the Interior, promulgated regulations of the certification of blasters in the area of environmental protection. As part of Montana's Permanent Program, similar regulations have recently been adopted. These regulations require Montana to make available, upon request, a training manual. Although numerous handbooks and textbooks are available (9, 24, 27, 29-30, 32, 46)* none are geared for use in training the broad spectrum of people involved in practical blasting. This manual is designed to fulfill that need.

Types of explosives and blasting agents and their key explosive and physical properties are discussed. Explosives selection criteria are described. The features of the traditional initiation systems—electrical, detonating cord, and cap and fuse—are pointed out, and the new nonelectric initiation systems are discussed. This manual describes various blasthole priming techniques. It also covers blasthole loading of various explosive types. Blast design, including geologic considerations, for both surface and underground blasting is detailed. Environmental effects of blasting, such as flyrock and air and ground vibrations, are discussed along with techniques of measuring and alleviating these undesirable side effects. Blasting safety procedures are detailed in the chronological order of the blasting process.

Where methods of accomplishing specific tasks are recommended, these should not be considered the only satisfactory methods. In many instances, there is more than one safe, effective way to accomplish a specific blasting task.

None of the material in this manual is intended to replace manufacturers' recommendations on the use of the products involved. We strongly recommend

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*Numbers in parentheses throughout the text refer to items in the bibliography preceding the appendixes.

that the individual manufacturer be consulted on the proper use of specific products.

Readers are encouraged to complete the review questions given at the end of each chapter and check their work with the answers given in Appendix E. This manual includes a glossary of terms used in explosives; Montana's Strip and Underground Mine Reclamation Rules and Regulations; Montana Safety and Health Standards for Metal and Nonmetal Mining and Related Industries; Federal Blasting Regulations; and references to articles providing more detailed information on the aforementioned items.

Persons seeking to become certified must complete the Affidavit found in Appendix F and forward it to the Bureau Chief, Coal and Uranium Bureau, Reclamation Division, Department of State Lands, Capitol Station, Helena, MT 59620.

Chapter 1:   Explosives Products

CHEMISTRY AND PHYSICS OF EXPLOSIVES

It is not essential that a blaster have a strong knowledge of chemistry and physics. However, a brief discussion of the reactions of explosives will be helpful in understanding how the energy required to break rock is developed.

An explosive is a chemical compound or mixture of compounds that undergoes a very rapid decomposition when initiated by energy in the form of heat, impact, friction, or shock (4). This decomposition produces more stable substances, mostly gases, and a large amount of heat. The very hot gases produce extremely high pressures within the borehole, and it is these pressures that cause the rock to be fragmented. If the speed of reaction of the explosive is faster than the speed of sound in the explosive (detonation), the product is called a high explosive. If the reaction of the explosive is slower than the speed of sound in the explosive (deflagration), the product is called a low explosive.

The principal reacting ingredients in an explosive are fuels and oxidizers. Common fuels in commercial products include fuel oil, carbon, aluminum, TNT, smokeless powder, monomethylamine nitrate, and monoethanol amine nitrate. Fuels often perform a sensitizing function. Common explosive sensitizers are nitroglycerin, nitrostarch, aluminum, TNT, smokeless powder, monomethylamine nitrate, and monoethalamine nitrate. Microballoons and aerating agents are sometimes added to enhance sensitivity. The most common oxidizer is ammonium nitrate, although sodium nitrate and calcium nitrate may also be used. Other ingredients of explosives include water, gums, thickeners, and cross-linking agents used in slurries (11), gelatinizers, densifiers, antacids, stabilizers, absorbents, and flame retardants. In molecular explosives such as nitroglycerin, TNT, and PETN, the fuel and oxidizer are combined in the same compound.

Most ingredients of explosives are composed of the elements oxygen, nitrogen, hydrogen, and carbon. In addition, metallic elements such as aluminum are sometimes used. For explosive mixtures, energy release is optimized at zero oxygen balance (5). Zero oxygen balance is defined as the point at which a mixture has sufficient oxygen to completely oxidize all the fuels it contains, but there is no excess oxygen to react with the nitrogen in the mixture to form nitrogen oxides.

Theoretically, at zero oxygen balance the gaseous products of detonation are H_2O, CO_2, and N_2, although in reality small amounts of NO, CO, NH_2, CH_4, and other gases are generated. Figure 1 shows the energy released by some of the common products of detonation. Partial oxidation of carbon to carbon mon-

STANDARD HEATS OF FORMATION, kcal/mole

Figure 1 (bar chart: Heat Release, kcal/atom of O, vertical axis from -40 to 160):

Heats of formation listed:
- MgO -144
- Al_2O_3 -399
- H_2O -58
- CO_2 -94
- CO -26
- N_2 0
- NO_2 +8
- N_2O +19
- NO +22

* Poisonous gas

Bars (left to right): NO *, N_2O, NO_2, N_2, CO *, CO_2, H_2O, Al_2O_3 (Solid), MgO (Solid)
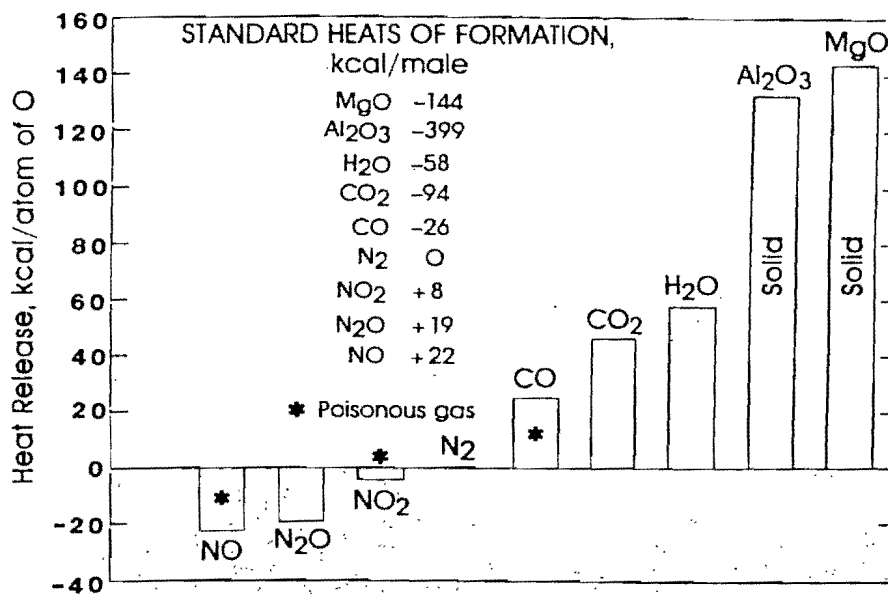
Figure 1: Energy relased by common products of detonation.

oxide, which results from an oxygen deficiency, releases less heat than complete oxidation to carbon dioxide. The oxides of nitrogen, which are produced when there is excess oxygen, are "heat robbers"; that is, they absorb heat when generated. Free nitrogen, being an element, neither absorbs nor releases heat upon liberation.

The gases resulting from improper oxygen balance are not only inefficient in terms of heat energy released but are also poisonous. Although the oxidation of aluminum yields a solid, rather than a gaseous product, the large amount of heat released adds significantly to the explosive's energy. Magnesium is even better from the standpoint of heat release, but is too sensitive to use in commercial explosives.

The principle of oxygen balance is best illustrated by the reaction of ammonium nitrate-fuel oil [$(NH_4NO_3)-(CH_2)_n$] mixtures. Commonly called AN-FO, these mixtures are the most widely used blasting agents. From the reaction equations for AN-FO, one can readily see the relationship between oxygen balance, detonation products, and heat release. The equations assume an ideal detonation reaction, which in turn assumes thorough mixing of ingredients, proper particle sizing, adequate confinement, charge diameter and priming, and protection from water. Fuel oil is actually a variable mixture of hydrocarbons and is not precisely CH_2, but this identification simplifies the equations and is accurate enough for the purposes of this manual. In reviewing these equations, keep in mind that the amount of heat produced is a measure of the energy released.

Equation 1   (94.5 percent AN and 5.5 percent FO):

$$3NH_4NO_3 + CH_2 \longrightarrow 7H_2O + CO_2 + 3N_2 + 0.93 \text{ kcal/g.}$$

Equation 2   (92.0 percent AN and 8.0 percent FO):

$$2NH_4NO_3 + CH_2 \longrightarrow 5H_2O + CO + 2N_2 + 0.81 \text{ kcal/g}.$$

Equation 3   (96.6 percent AN and 3.4 percent FO):

$$5NH_4NO_3 + CH_2 \longrightarrow 11H_2O + CO_2 + 4N_2 + 2NO + 0.60 \text{ kcal/g}.$$
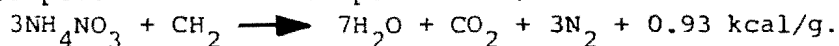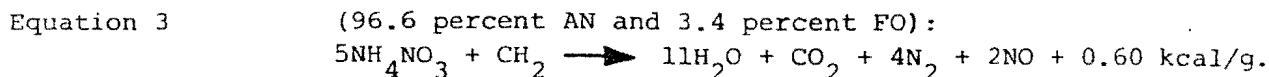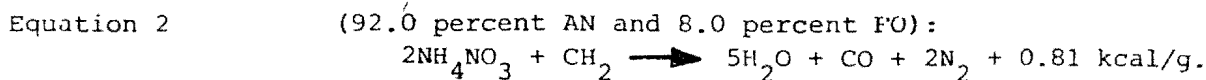
Equation 1 represents the reaction of an oxygen-balanced mixture containing 94.5 percent AN and 5.5 percent FO. None of the detonation gases are poisonous and 0.93 kilocalories (or 1,000 calories) of heat is released for each gram of AN-FO detonated. In equation 2, which represents a mixture of 92.0 percent AN and 8.0 percent FO, the excess fuel creates an oxygen deficiency. As a result, the carbon in the fuel oil is oxidized only to CO, a poisonous gas, rather than relatively harmless CO_2. Because of the lower heat of formation of CO, only 0.81 kilocalories (kcal) of heat is released for each gram of AN-FO detonated. In equation 3, the mixture of 96.6 percent AN and 3.4 percent FO has a fuel shortage that creates an excess oxygen condition. Some of the nitrogen from the ammonium nitrate combines with this excess oxygen to form extremely toxic NO_2. The heat absorbed by the formation of NO reduces the heat of reaction to only 0.60 kcal, which is considerably lower than that of an overfueled mixture. Also, the CO produced by an overfueled mixture is less toxic than NO and NO_2. For these reasons, a slight oxygen deficiency is preferable and the common AN-FO mixture for field use is 94 percent AN and 6 percent FO.

Although the simple AN-FO mixture is optimum for highest energy release per unit cost of ingredients, products with higher energies and densities are often desired. The common high-energy-producing additives, which may be used in both dry blasting agents and slurries, fall into two basic categories: explosives, such as TNT; and metals, such as aluminum. Equations 4 and 5 illustrate the reaction of TNT and aluminum as fuel-sensitizers with ammonium nitrate. The reaction products, again, assume ideal detonation, which is never actually attained in the field. In practice, aluminum is never the only fuel in the mixture, some carbonaceous fuel is always used.

Equation 4 (78.7 percent AN) and (21.3 percent TNT):

$$21NH_4NO_3 + 2C_6H_2CH_3(NO_2)_3 \longrightarrow 47H_2O + 14CO_2 + 24N_2 + 1.01 \text{kcal/g}.$$

Equation 5 (81.6 percent AN) and (18.4 percent Al):

$$3NH_4NO_3 \text{ and } 2Al \longrightarrow 6H_2O + Al_2O_3 + 3N_2 + 1.62 \text{kcal/g}.$$
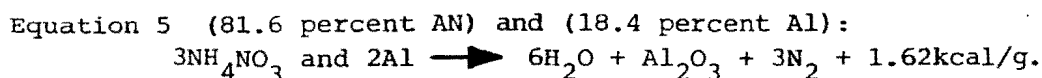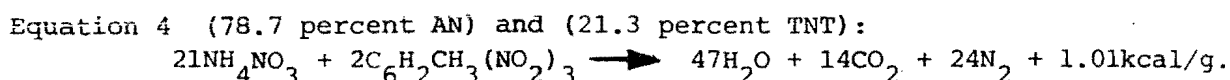
Both of these mixtures release more energy, based on weight, than ammonium nitrate-carbonaceous fuel mixtures and have the added benefit of higher densities. These advantages must be weighed against the higher cost of such high-energy additives. The energy of aluminized products continues to increase with larger percentages of metal, even though this "overfueling" causes an oxygen deficiency. Increasing energy by overfueling with metals, however, is uneconomical except for such specialty products as high-energy boosters.

The chemical reaction of an explosive creates extremely high pressures. It is these pressures which cause rock to be broken and displaced. To illustrate the pressures created in the borehole, a brief look will be taken at the detonation process as pictured by Dr. Richard Ash of the University of Mis - souri at Rolla. Figure 2, adapted from Ash's work shows (top) a column of
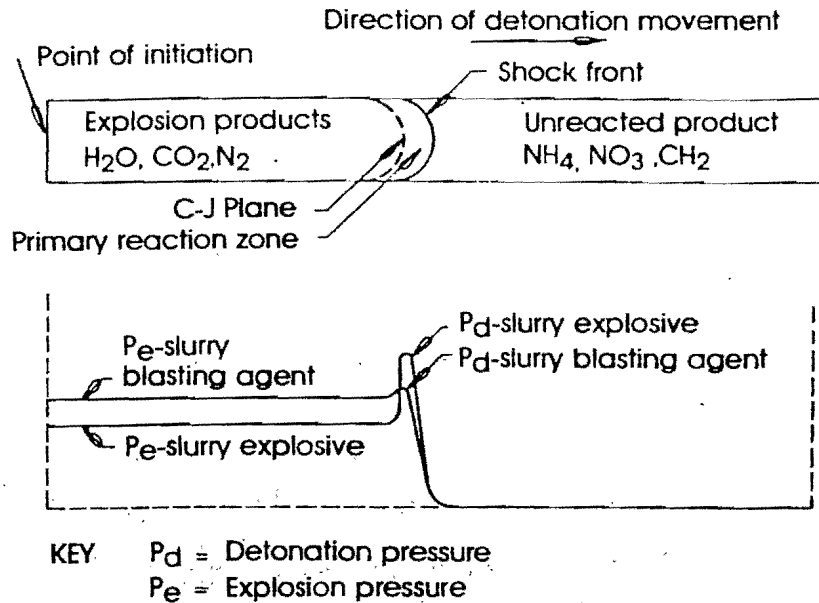
Figure 2: *Pressure profiles created by detonation in a borehole.*

explosive or blasting agent that has been initiated. Detonation has proceededto the center of the column. The primary reaction occurs between a shock front at the leading edge and a rear boundary known as the Chapman-Jouguet (C-J) plane. Part of the reaction may occur behind the C-J plane, particularly if some of the explosive's ingredients are coarse. The length of the reaction zone, which depends on the explosive's ingredients, particle size, density, and confinement, determines the minimum diameter at which the explosive will function dependably (critical diameter). High explosives, which have short reaction zones, have smaller critical diameters than blasting agents.

The pressure profiles in figure 2 (bottom) show the explosive forces applied to the rock being blasted. A general comparison is given between an explosive and a blasting agent, although it should be understood that each explosive or blasting agent has its own particular pressure profile depending on its ingredients, particle size, density, and confinement.

The initial pressure, called the detonation pressure (P_d) is created by the supersonic shock front moving out from the detonation zone. The detonation pressure gives the explosive its shattering action in the vicinity of the borehole. If the explosive reacts slower than the speed of sound, which is normally the case with black powder, there is no detonation pressure.

The detonation pressure is followed by a sustained pressure called explosion pressure (Pe), or borehole pressure. Borehole pressure is created by the rapid expansion of the hot gases within the borehole. The detonation pressure of high explosives is often several times that of blasting agents, but the borehole pressures of the two types of products are of the same general magnitude. The relative importance of detonation pressure and borehole pressure in

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breaking rock will be discussed in the "Properties of Explosives" section of this chapter.

TYPES OF EXPLOSIVES AND BLASTING AGENTS

This section will cover all explosive products that are used for industrial rock blasting, with the exception of initiators. Products used as the main borehole charge can be divided into three categories: nitroglycerin- (or nitrostarch-) based high explosives, dry blasting agents, and slurries, which may also be referred to as water gels or emulsions. These products can also be broadly categorized as explosives and blasting agents. For ease of expression, the term explosives will often be used in this manual to collectively cover both explosives and blasting agents. The difference between an explosive and a blasting agent is as follows:

A high explosive is any product used in blasting that is sensitive to a No. 8 cap and that reacts at a speed faster than the speed of sound in the explosive medium. A low explosive is a product in which the reaction is slower than the speed of sound. Low explosives are seldom used in blasting today. A blasting agent is any material or mixture consisting of a fuel and an oxidizer, intended for blasting, not otherwise classified as an explosive, provided that the finished product, as mixed and packaged for shipment, cannot be detonated by a No. 8 blasting cap in a specific test prescribed by the Bureau of Mines. Slurries containing TNT, smokeless powder, or other explosive ingredients, are classed as blasting agents if they are insensitive to a No. 8 blasting cap.

AN-FO, which in normal form is a blasting agent, can be made cap sensitive by pulverizing it to a fine particle size, and a slurry can be made cap sensitive by including a sufficient amount of finely flaked paint-grade aluminum. Although neither of these products contains an explosive ingredient, their cap sensitivity requires that they be classified as explosives. The term nitrocarbonitrate, or NCN, was once used synonymously with blasting agent under U.S. Department of Transportation (DOT) regulations for packaging and shipping blasting agents. DOT no longer uses this term.

Nitroglycerin-Based High Explosives

Nitroglycerin-based explosives can be categorized according to nitroglycerin content (4). Figure 3 shows this breakdown along with some relative properties and ingredients of the products. Table 1 shows some properties of nitroglycerin-based explosives. Property values are averages of manufacturers' published figures. As a group, nitroglycerin-based explosives are the most sensitive commercial products used today (excluding detonators). Because of this sensitivity, they offer an extra margin of dependability in the blasthole but are somewhat more susceptible to accidental detonation. This is a tradeoff that many operators who use small-diameter boreholes must make. Nitroglycerin dynamites account for less than 5 percent, by weight, of the explosives market (12), and almost all of that is in small-diameter work. Dynamite is available in cartridges of various sizes and shapes, as shown in figure 4.

Nitroglycerin, the first high explosive, is the sensitizer in dynamites and is seldom used alone, although it has been used in a somewhat desensitized

Table 1 - Properties of Nitroglycerin-based Explosives

Weight Strength, Percent	Bulk strength Percent	Specific Gravity	Detonation velocity, Feet per Second	Water Resistance	Fume Class
Straight dynamite					
50	50	1.4	17,000	Good	Poor
High-density ammonia dynamite					
60	50	1.3	12,500	Fair	Good
40	35	1.3	10,500	Fair	Good
20	15	1.3	8,000	Fair	Good
Low-density ammonia dynamite, high velocity					
65	50	1.2	11,000	Fair	Fair
65	40	1.0	10,000	Fair	Fair
65	30	0.9	9,500	Poor	Fair
65	20	0.8	8,500	Poor	Fair
Low-density ammonia dynamite, low velocity					
65	40	1.2	8,000	Fair	Fair
65	40	1.0	7,500	Poor	Fair
65	30	0.9	7,000	Poor	Fair
65	20	0.8	6,500	Poor	Fair
Blasting Gelatin					
100	90	1.3	25,000	Excellent	Poor
Straight gelatin					
90	80	1.3	23,000	Excellent	Poor
60	60	1.4	20,000	Excellent	Good
40	45	1.5	16,500	Excellent	Good
20	30	1.7	11,000	Excellent	Good
Ammonia gelatin					
80	70	1.3	20,000	Very good	Good
60	60	1.4	17,500	Very good	Very good
40	45	1.5	16,000	Very good	Very good
Semigelatin					
65	60	1.3	12,000	Very good	Very good
65	50	1.2	12,000	Very good	Very good
65	40	1.1	11,500	Good	Very good
65	30	0.9	10,500	Fair	Very good

| Ingredients | Nongelatinous | Gelatinous | Properties |

Figure labels (left, vertical): Decreasing cost | Increasing ammonium nitrate content | Decreasing nitroglycerin content

Figure labels (right, vertical): Decreasing density | Decreasing detonation velocity | Decreasing water resistance

Nongelatinous	Gelatinous
Nitroglycerin	Blasting gelatin
Straight dynamite	Straight gelatin
High density ammonia dynamite	Ammonia gelatin
Low-density ammonia dynamite	Semigelatin
Dry blasting agents	Slurries

Increasing water resistance ⟶

Figure 3: Relative ingredients and properties of nitroglycerin-based high explosives.

form from shooting oil wells. It has a specific gravity of 1.6 and a detonation velocity slightly over 25,000 feet per second (fps). Its extreme sensitivity to shock, friction, and heat make it hazardous to use.

Straight (nitroglycerin) dynamite consists of nitroglycerin, sodium nitrate, an antacid, a carbonaceous fuel, and sometimes sulfur. The term "straight" means that a dynamite contains no ammonium nitrate. The weight strength, usually 50 percent, indicates the approximate percentage of nitroglycerin or other explosive oil. The use of straight dynamite is limited because of its high cost and sensitivity to shock and friction. Fifty percent straight dynamite, by far the most common straight dynamite, is referred to as ditching dynamite and is used in propagation blasting.

High-density ammonia dynamite, also called extra dynamite, is the most widely used dynamite. It is like straight dynamite, except that ammonium nitrate replaces part of the nitroglycerin and sodium nitrate. Ammonia dynamite is manufactured in grades of 20 to 60 percent weight strength, although these grades are not truly equivalent to straight dynamites of the same weight strength (see properties in table 1). Ammonia dynamite is less sensitive to shock and friction than straight dynamite. It is most commonly used in small quarries, in underground mines, in construction, and as an agricultural explosive.

Low-density ammonia dynamite is manufactured in a weight strength of about 65 percent. The cartridge (bulk) strength ranges from 20 to 50 percent, depending on the bulk density of the ingredients. A high-velocity series and a low-velocity series are manufactured. Low-density ammonia dynamite is useful in very soft or prefractured rock or where coarse rock such as riprap is required.

Blasting gelatin is a tough, rubber-textured explosive made by adding nitrocellulose, also called guncotton, to nitroglycerin. An antacid is added to provide storage stability and wood meal is added to improve sensitivity. Blasting gelatin emits large volumes of noxious fumes upon detonation and is expensive. It is seldom used today. Sometimes called oil well explosive, it has been used in deep wells where high heads of water are encountered. Blasting gelatin is the most powerful nitroglycerin-based explosive.
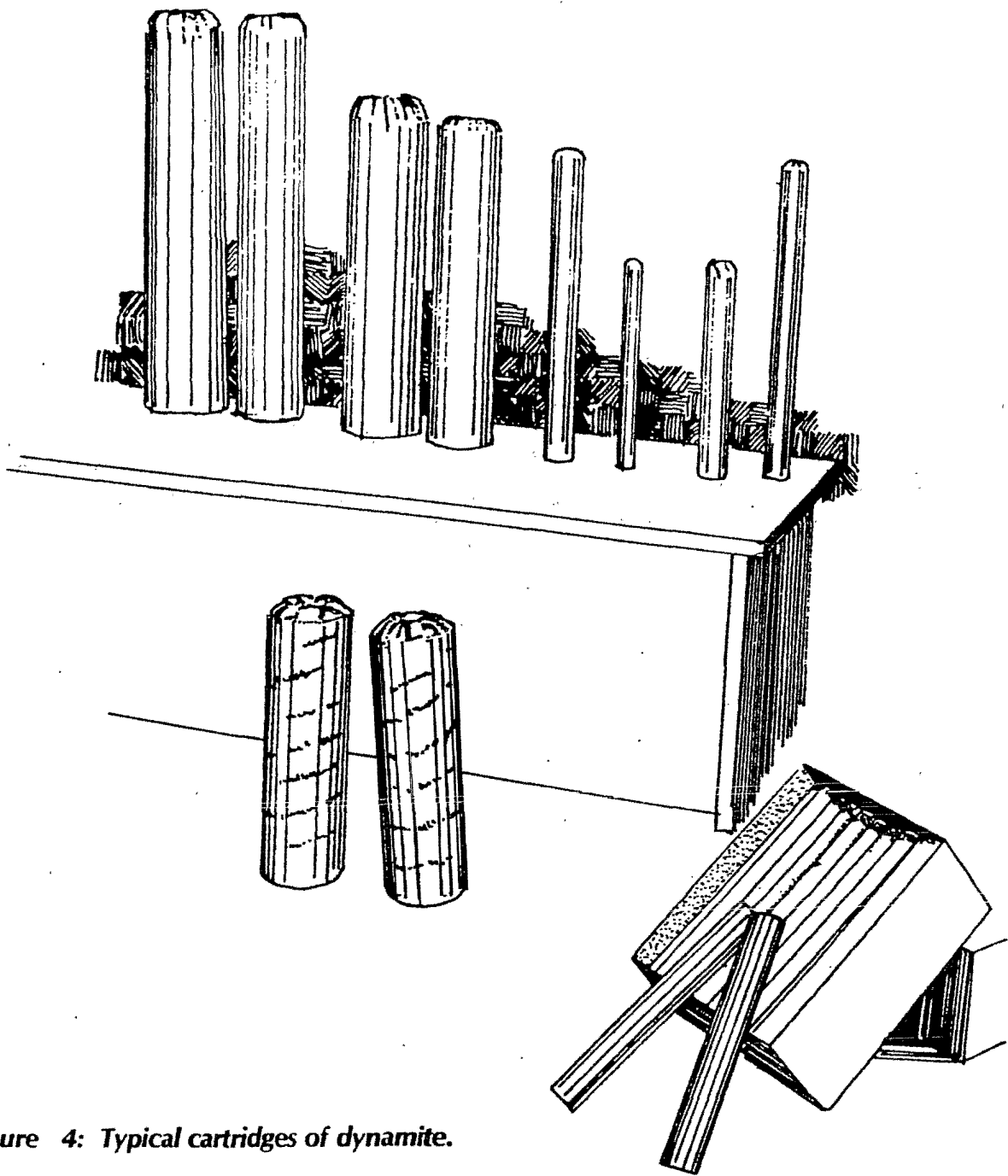
Figure 4: Typical cartridges of dynamite.

Straight gelatin is basically a blasting gelatin with sodium nitrate, carbonaceous fuel, and sometimes sulfur added. It is manufactured in grades ranging from 20 to 90 percent weight strength and is the gelatinous equivalent of straight dynamite. Straight gelatin has been used mainly in specialty areas such as seismic or deep well work, where a lack of confinement or a high head of water may affect its velocity. To overcome these conditions, a high-velocity gelatin is available which is likely straight gelatin except that it detonates near its rated velocity despite high heads of water.

Ammonia gelatin, also called special gelatin or extra gelatin, is a straight gelatin in which ammonium nitrate has replaced part of the nitrogly-

cerin and sodium nitrate. Manufactured in weight strengths ranging from 40 to 80 percent, it is the gelatinous equivalent of ammonia dynamite. Ammonia gelatin is suitable for underground work, in wet conditions, and as a toe load, primarily in small-diameter boreholes. The higher grades (70 percent or higher) are useful as primers for blasting agents.

Semigelatin has a weight strength near 65 percent. The cartridge (bulk) strength ranges from 30 to 60 percent with variations in the bulk density of the ingredients. Semigelatin is versatile and is used in small-diameter work where some water resistance is required. It is useful underground, where its soft, plastic consistency makes it ideal for loading into holes drilled upward.
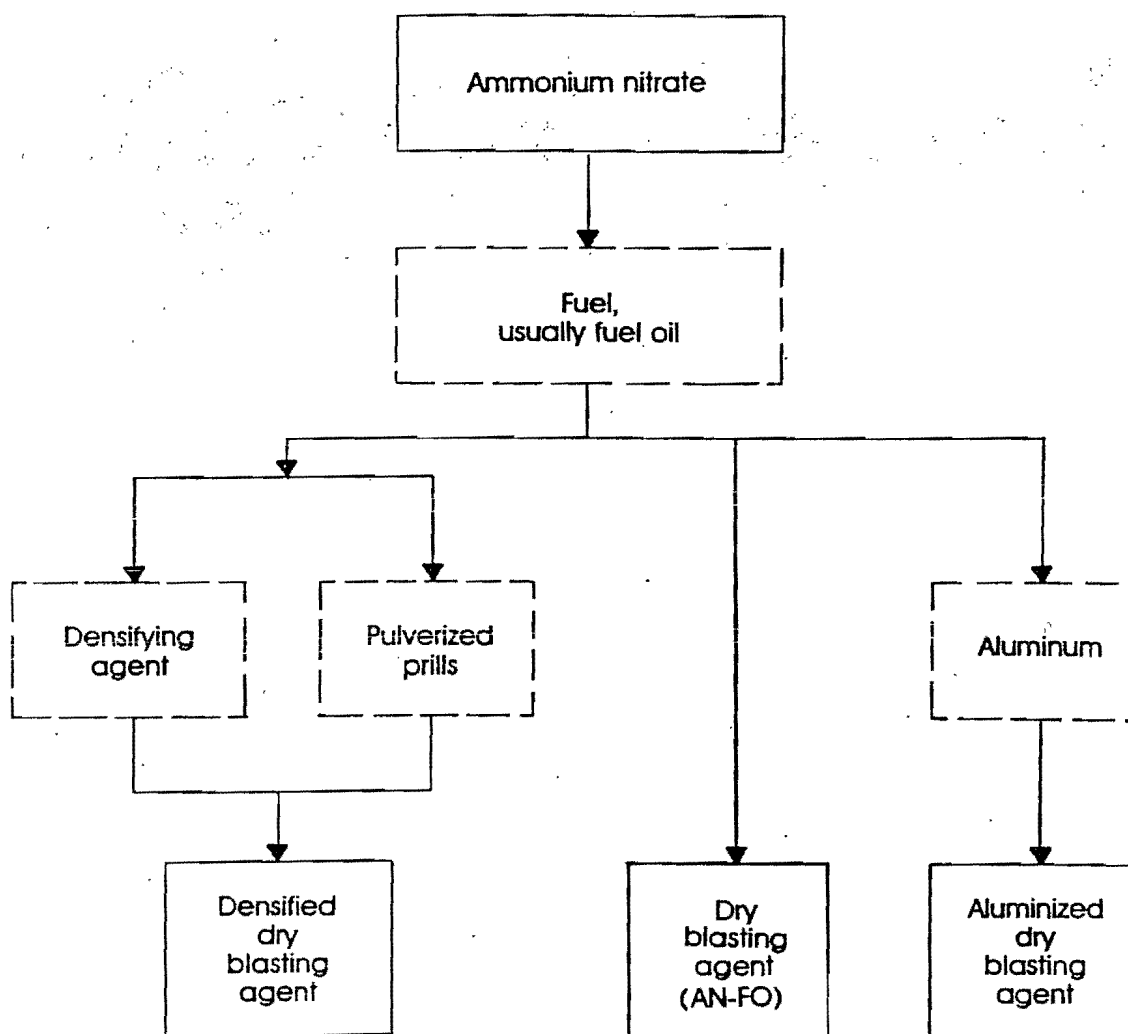
Figure 5: Types of dry blasting agents and their ingredients.

Nitrostarch explosives are sensitized with nitrostarch, a solid molecular explosive, rather than an explosive oil. They are manufactured in various

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grades, strengths, densities, and degrees of water resistance to compete with
most grades of nitroglycerin-based dynamites. They are similar to dynamites
in many ways with their most significant differences being somewhat higher
impact resistance and their "headache-free" nature.

Dry Blasting Agents

     In this manual, the term dry blasting agent describes any material used
for blasting which is not cap sensitive and in which water is not used in the
formulation. Figure 5 describes the dry blasting agents in use today.

     Early dry blasting agents employed solid carbon fuels combined with ammo-
nium nitrate in various forms. Through experimentation it was found that
diesel fuel oil mixed with porous ammonium nitrate prills (figure 6) gave the
best blasting results. Hence, the term AN-FO (ammonium nitrate-fuel oil) has
been synonymous with dry blasting agent. An oxygen-balanced AN-FO is the
cheapest source of explosive energy available today. Adding finely divided or
flaked aluminum to dry blasting agents increases the energy output but at an
increase in cost. Aluminumized dry mixes are sometimes used in combination
with cast primers as primers for AN-FO. Aluminumized mixes may also be used
as a high-energy toe load and as the main column charge where blasting is
difficult.
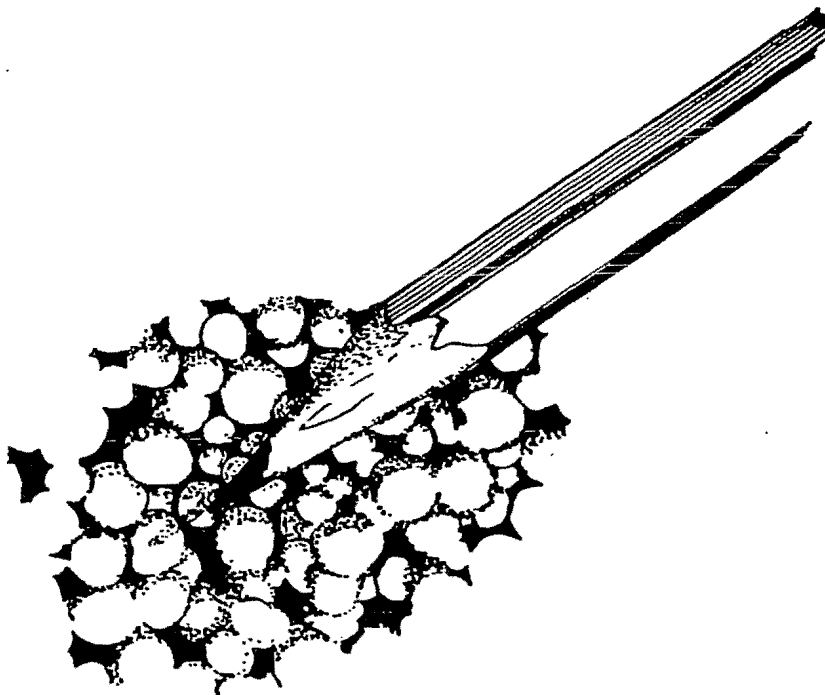
Figure 6: Porous ammonium nitrate prills.

It is difficult to give precise numerical values for the properties of
dry blasting agents because the properties vary with ingredient particle size,

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density, confinement, charge diameter, water conditions, and coupling ratio
(5). Yancik has prepared an excellent manual on explosive properties of AN-FO
(9).

Coupling ratio is the percentage of the borehole diameter filled with
explosive. Poured bulk products are completely coupled, which increases their
efficiency. Cartridged products are partially decoupled, and thus lose some
efficiency.

AN-FO's theoretical energy is optimized at oxygen balance (approximately
94.5 percent AN and 5.5 percent FO), where the detonation velocity approaches
15,000 fps in large charge diameters. Excess fuel oil (8 percent or more) can
seriously reduce sensitivity to initiation. Inadequate fuel oil causes an
excess of harmful nitrogen oxide fumes in the detonation gases. Specific gra-
vities of AN-FO range from 0.5 to 1.15; 0.80 to 0.85 is the most common range.
The lighter products are useful in easily fragmented rock or to eliminate the
need for alternate decks of explosive and stemming where a low powder factor
is desirable. The densified dry mixes are packaged in water-proof containers
for use in wet blastholes (figure 7).

Densification is necessary to enable the cartridges to sink in water. To
obtain a higher specific gravity, part of the prills are pulverized and then
the mixture of whole and pulverized prills is vibrated or otherwise compressed
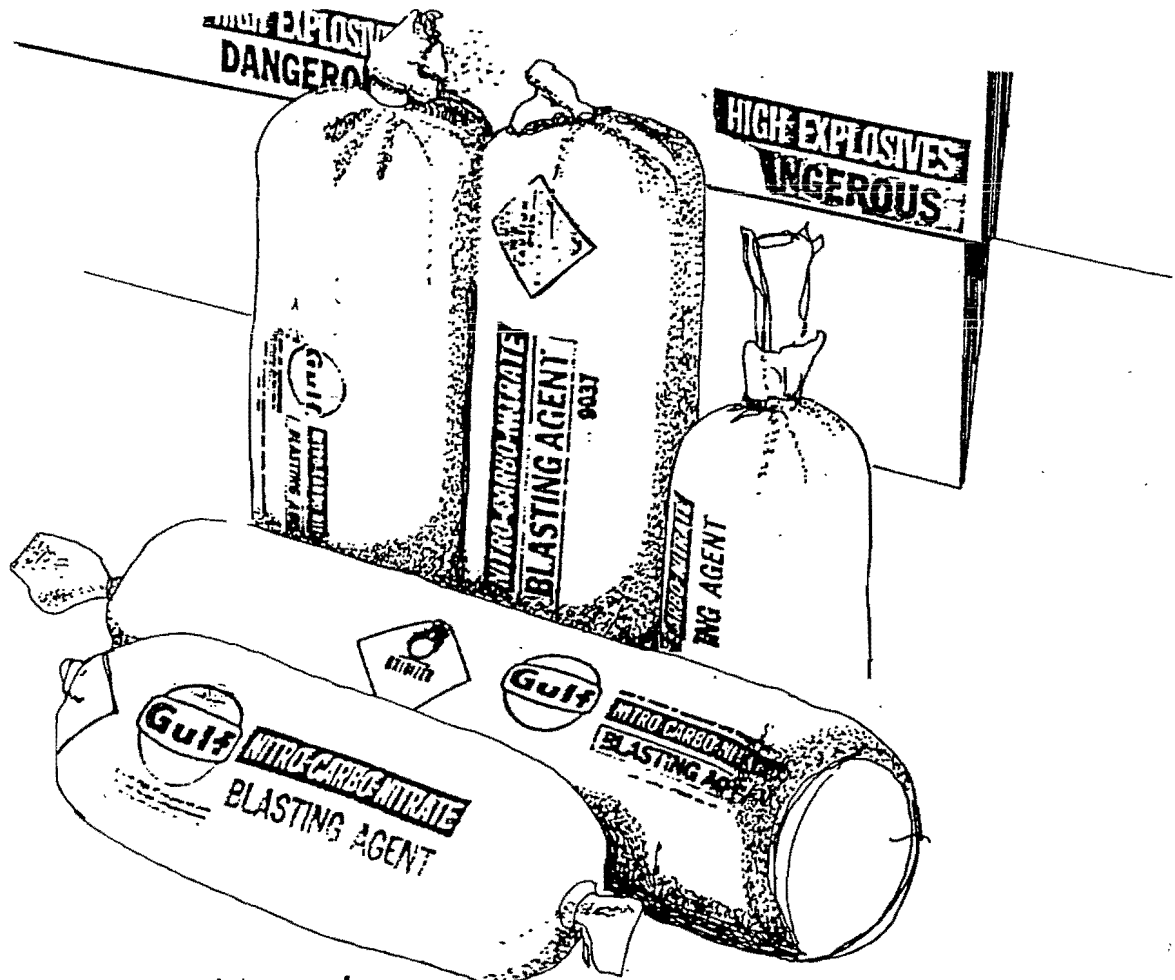
Figure 7: Water-resistant packages
of AN-FO for use in wet boreholes.   11

into rigid cartridges or polyburlap bags. Densifying ingredients, such as ferrosilicon, are seldom used today because they add little or nothing to the explosive's energy. The sensitivity of AN-FO decreases with increased density. The "dead press" limit, above which detonation is undependable, is about 1.25 grams per cubic centimeter (g/cm^3).

The detonation velocity of AN-FO is strongly affected by charge diameter. The critical diameter is near 1 inch with a normal prill and oil mixture. The velocity increases with diameter and levels off near a 15-inch diameter at a velocity of nearly 15,000 fps. The minimum primer required to AN-FO increases as charge diameter increases. There is a tendency to underprime in large-diameter boreholes. A good rule of thumb is, when in doubt, overprime. Many operators claim improved results when they use primers that fill, or nearly fill, the blasthole diameter.

The undesirable effect of water on dry blasting agents has often been seen in poor blasts where AN-FO was used in wet boreholes with insufficient external protection. Excess water adversely affects the velocity, sensitivity, fume class, and energy output of a dry blasting agent. The extreme result is a misfire. It is essential when using AN-FO in wet conditions that positive protection in the form of a waterproof package or a borehole liner be used (3).

Dry blasting agents can be purchased in four forms. In increasing order of cost they are as follows:

1. As separate ingredients in bulk form for onsite mixing,
2. premixed in bulk form for onsite storage or direct borehole delivery (a premixed product may cost about the same as separate ingredients),
3. in paper or polyethylene packages for pouring into the borehole,
4. in waterproof cartridges or polyburlap containers.

Waterproof containers are the most expensive forms and eliminate the advantage of direct borehole coupling. They should be used only where borehole conditions dictate. Because improper ingredient proportions or an insufficiently mixed product cause inefficient detonation and poor fume qualities, thorough mixing and close quality control should be exercised in an onsite mixing operation. The use of a colored dye in the fuel gives a visual check on mixing and also makes the blasting agent more easily visible in case of misfire.

Recent trials in taconite mines have employed a dense dry blasting agent composed of 87 percent crushed ammonium nitrate prills and 13 percent of a 50-50 mixture of a nitropropane and methanol. This product has slightly more energy per unit weight than AN-FO and can be loaded at a density of approximately 1.2 g/cm^3, giving it a high-energy density. Because of the experimental nature of this product, Mine Safety and Health Administration (MSHA) should be consulted before putting it to use.

Slurries

A slurry (figure 8) is a mixture of nitrates such as ammonium nitrate and sodium nitrate, a fuel sensitizer, either explosive or nonexplosive, and varying amounts of water (1). A water gel is essentially the same as a slurry and

the two terms are frequently used interchangeably. An emulsion is somewhat different from a water gel or slurry in physical character but similar in many functional respects. The principal differences are an emulsion's generally higher detonation velocity and a tendency to wet or adhere to the blasthole, which in some cases may affect its bulk loading characteristics. In this discussion, slurries, water gels, and emulsions will be treated as a family of products.

Although they contain large amounts of ammonium nitrate, slurries are made water resistant through the use of gums, waxes, and cross-linking agents. The variety of possible slurry formulations is almost infinite. Frequently, a slurry is specially formulated for a specific job. The list of possible fuel
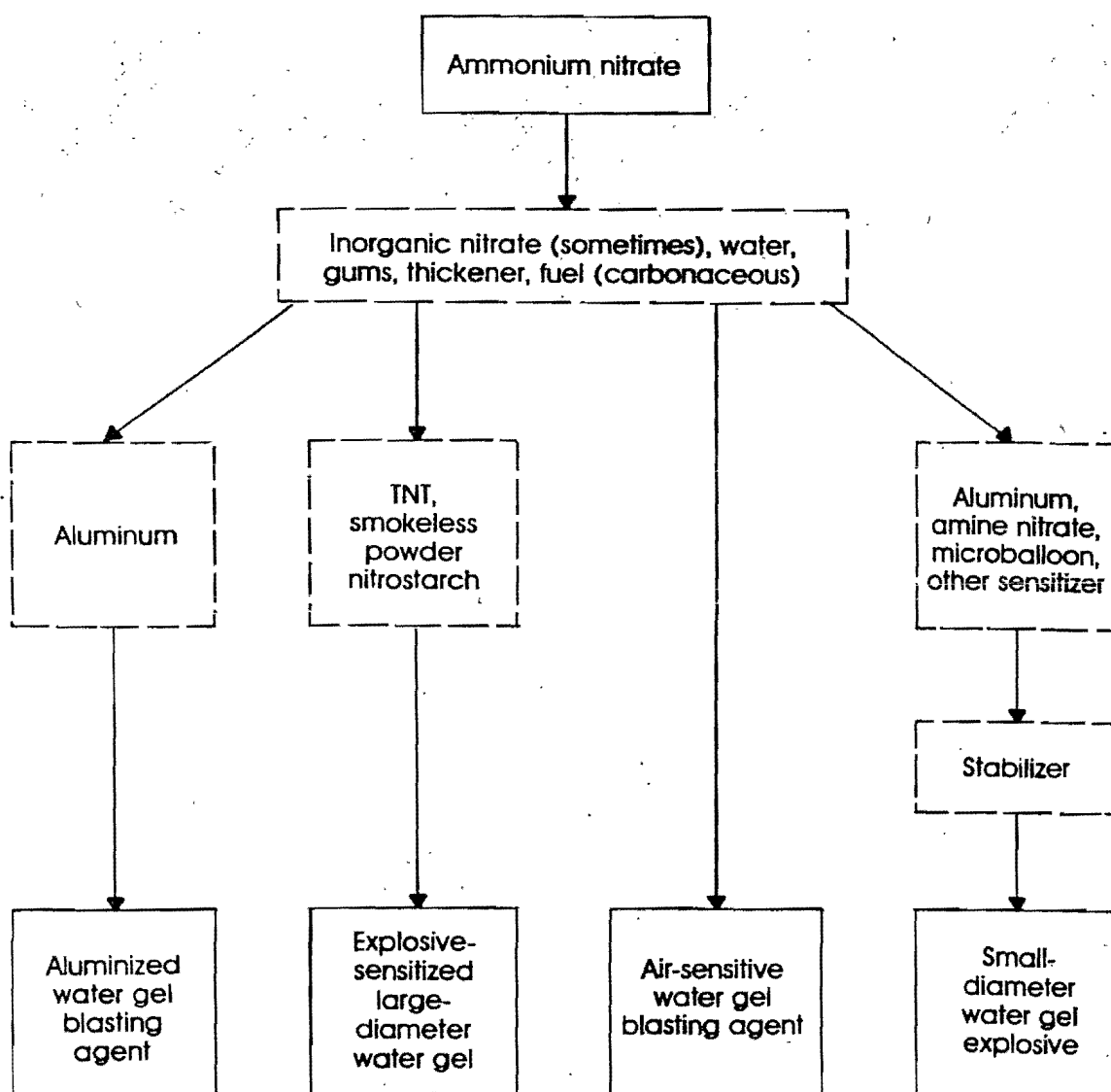
Figure 8: Formulations of water-based products.

sensitizers is especially long (11), although carbonaceous fuels, aluminum, and amine nitrates are the most common.

Slurries may be classified as either explosives or blasting agents. Those that are sensitive to a No. 8 cap are classified as explosives, even though they are less sensitive than dynamites. It is important that slurries be stored in magazines appropriate to their classification.

Except for their excellent water resistance and higher density and bulk strength, slurries are similar in many ways to dry blasting agents. Good oxygen balance, decreased particle size and increased density, increased charge diameter, good confinement and coupling, and adequate priming all increase their efficiency. Although slurry blasting agents tend to lose sensitivity as their density increases, some explosive-based slurries function well at densities up to 1.6. The effect of charge diameter on the detonation velocity of slurries is not as pronounced as it is on AN-FO.

Most non-cap-sensitive slurries depend on entrapped air for sensitivity and most cap-sensitive varieties are also dependent, to a lesser degree, on this entrapped air. If this air is removed from a slurry through pressure from an adjacent blast, prolonged periods of time in the borehole, or prolonged storage, the slurry may become densensitized.

Slurries can be delivered as separate ingredients for onsite mixing, premixed for bulk loading (figure 9) in polyethylene bags for bulk loading or loading in the bag (figure 10), or they may be cartridged. Their consistency may be anywhere from a liquid to a cohesive gel.
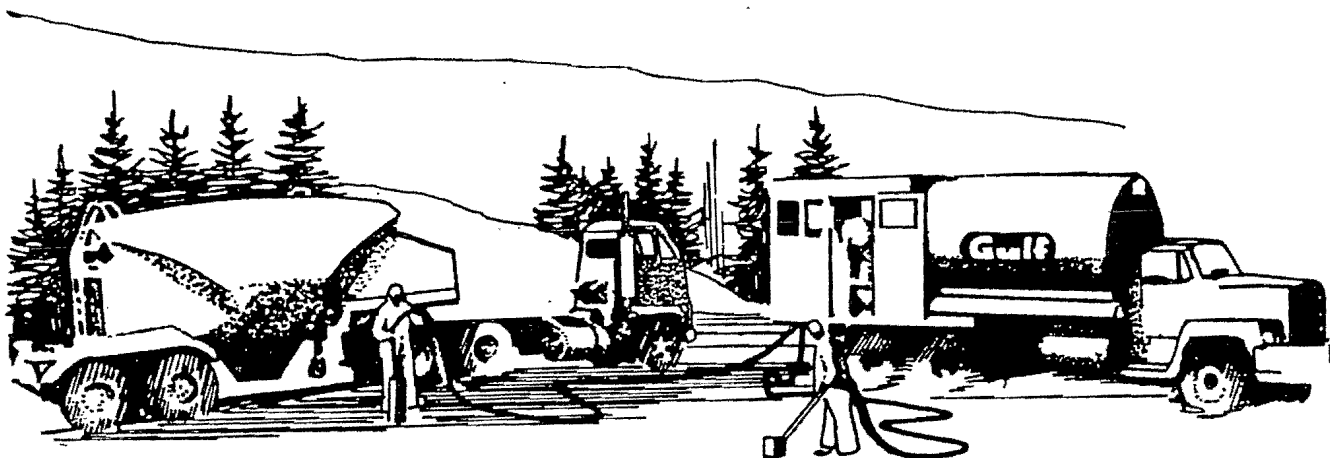
Figure 9: Slurry bulk loading trucks.

Cartridged slurries for use in small-diameter blastholes (2-inch diameter or less) are normally made cap sensitive so they can be substituted for dynamites. However, their lower sensitivity as compared with dynamite should be

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kept in mind. The sensitivity and performance of some grades of slurries are adversely affected by low temperatures. Slurries designed for use in medium-diameter blastholes (2- to 5-inch diameters) may be cap sensitive but they often are not. Those that are not cap sensitive must be primed with a cap-sensitive explosive. Slurries for use in large diameters (greater than 5 inches) are the least sensitive slurries.

Figure 10: Loading slurry-filled polyethylene bags.

Slurries containing neither aluminum nor explosive sensitizers are the cheapest, but they are also the least dense and powerful. In wet conditions where dewatering is not practical, and the rock is not extremely difficult to fragment, these low-cost slurries offer competition to AN-FO.

Aluminumized slurries or those containing significant amounts of other high-energy sensitizers develop sufficient energy for blasting in hard, dense rock. However, the economics of using total column charges of highly aluminumized slurry are doubtful because of the significantly higher cost of these product. High-energy slurries have improved blasting efficiency when used in combination with the primer at the toe or in another zone of difficult breakage.

Detonating cord downlines can have a harmful effect on the efficiency of blasting agent slurries, depending on the size of the blasthole and the strength of the cord. When using detonating cord downlines, the slurry manufacturer should be consulted concerning the effect of the cord on the slurry.

The technology of slurries is very dynamic. New products are continually being developed. Blasters should check the technical literature to be aware of developments that affect their blasting programs.

Two-Component Explosives

Individually, the components of two-component explosives, also called binary explosives, are not classified as explosives. When shipped and stored separately they are not normally regulated as explosives, but they should be protected from theft. However, some organizations, such as the U.S. Forest Service and some state and local government agencies, may treat these components as explosives for storage purposes.

The most common two-component explosive is a mixture of pulverized ammonium nitrate and nitromethane, although other fuel sensitizers such as rocket fuel have been used. The components are carried in separate containers to the jobsite, where the container of liquid fuels is poured into the ammonium nitrate container. After the prescribed waiting time, the mixture becomes cap sensitive and is ready for use.

Two-component explosives are sometimes used where only small amounts of explosives are required, such as in power line installation and light construction. Where large amounts of explosives are needed, the higher cost per pound and the inconvenience of onsite mixing negate the savings and convenience realized through less stringent storage and distribution requirements. In some states, Pennsylvania for example, the user of two-component explosives is considered a manufacturer and must obtain a manufacturer's license.

Permissible Explosives

Permissible explosives are designed for use in underground coal mines, where the presence of explosive gases or dust presents an abnormal blasting hazard. Both nitroglycerin-based permissibles and slurry, water gel, and emulsion permissibles are available. Briefly stated, the specifications of a permissible explosive are as follows:

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1. The chemical composition furnished by the applicant must agree, within tolerance, with that determined by MSHA.
2. The explosive must pass a series of propagation tests.
3. The airgap sensitivity of 1-1/4-inch cartridges must be at least 3 inches.
4. The explosive must pass nonignition tests when fired unstemmed into a mixture of natural gas, air, and bituminous coal dust.
5. The explosive must pass tests for nonignition when fired stemmed in a gallery of air and natural gas.
6. The volume of poisonous gases produced by a pound of explosive must not exceed 2.5 cubic feet.
7. The explosive must exhibit insensitivity in the pendulum friction test.

Permissible explosives must be used in a permissible manner, as described briefly in the "Underground Coal Mine Blasting" section of Chapter 4. MSHA must also approve explosives used in gassy noncoal mines. For gassy noncoal mines, MSHA sometimes approves products such as AN-FO, and specifies the manner in which they are to be used.

Sodium chloride or other flame depressants are used in permissible explosives to minimize the chance of igniting the mine atmosphere. As a result, permissible explosives are less energetic than other explosives and have a

CAST PRIMER FOR
DETONATING CORD

CAST PRIMER FOR
BLASTING CAP
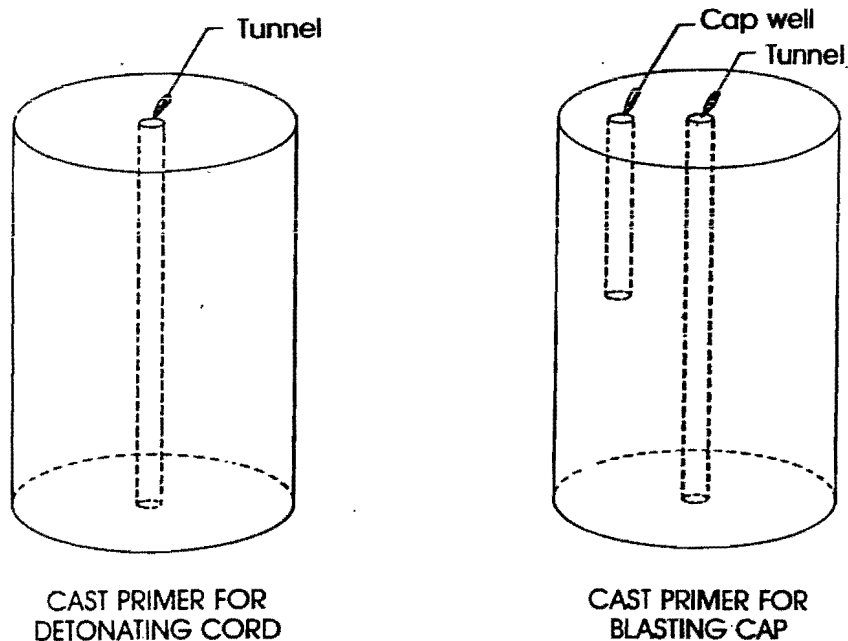
Figure 11: Cast primers for blasting caps and detonating cord.

lower rock-breaking capability. They should be used only where required by a gassy atmosphere. Permissible explosives are allowed to generate more fumes than other explosives, but most do not. MSHA periodically publishes an updated list of brand names and properties of permissible explosives (14).

Primers and Boosters

The terms "primer" and "booster" are often confused. According to MSHA, a primer--sometimes called a capped primer--is a unit of cap-sensitive explosive used to initiate other explosives or blasting agents. A primer contains a detonator. A booster is often, but not always, cap sensitive, but does not contain a detonator. A booster is used to perpetuate or intensify an explosive reaction.

Although various products have been used as primers and boosters, an explosive with a high detonation pressure such as a high-strength ammonia gelatin or a cast military explosive (composition B or pentolite) (figure 11) is recommended. Cast primers have a sensitive inner core that will accept detonation from a detonator or detonating cord, but are quite insensitive to external shock or friction. Cast primers are available which have built-in millisecond delay units (figure 12). These primers, when strung on a single detonating cord downline, enable the blaster to place as many delayed decks in the blasthole as the blast design requires.
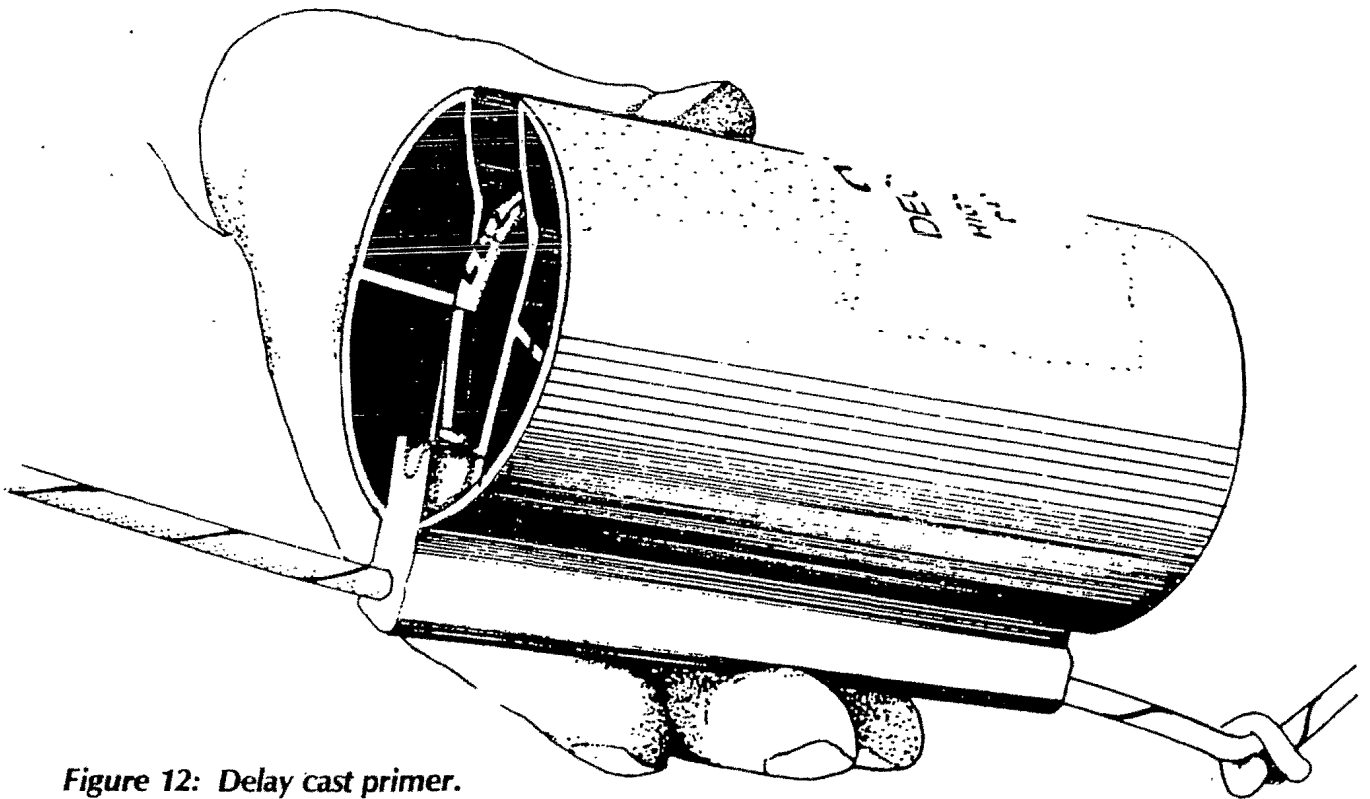
Figure 12: Delay cast primer.

Although small one-pound cast primers are popular, even in large bore-holes, a primer functions best when its diameter is near that of the borehole. A two-stage primer, with a charge of high-energy dry blasting agent or slurry poured around a cast primer or ammonia gelatin, is frequently used in large-diameter blastholes. In Sweden, in small-diameter work, excellent results have been reported with a high-strength blasting cap used to initiate AN-FO, thus eliminating the need for a primer. In the United States, a more common practice in small-diameter work is to use a small primer design to fit directly over a blasting cap, or a small cartridge of ammonia gelatin. More detailed priming recommendations are given in Chapter 2.

Liquid Oxygen Explosive and Black Powder

Liquid oxygen explosive (LOX) and black powder merit a brief mention because of their past importance. LOX consists of a cartridge of lampblack, carbon black, or charcoal, dipped into liquid oxygen just before loading. It derives its energy from the reaction of the carbon and oxygen to form carbon dioxide. LOX is fired with an ordinary detonator and attains velocities of 12,000 to 19,000 fps. LOX, primarily used in U.S. strip coal mining, has been replaced by blasting agents, although it is still used in other countries.

Black powder, a mixture of potassium or sodium nitrate, charcoal, and sulfur, dates from ancient times. Once the principal commercial explosive, black powder is extremely prone to accidental initiation by flame or spark. When initiated, it undergoes burning at a very rapid rate. This rapid burning, called deflagration, is much slower than typical detonation velocities. Black powder has a specific gravity of 1.6 or less, depending or granulation, has poor water resistance, and emits large volumes of noxious gases upon deflagration. Black powder finds limited use in blasting dimension stone where a minimum of shattering effect is desired. It is not an efficient explosive for fragmenting rock.

PROPERTIES OF EXPLOSIVES

Explosives and blasting agents are characterized by various properties that determine how they will function under field conditions. Properties of explosives which are particularly important to the blaster include strength, detonation velocity, density, water resistance, fume class, detonation pressure, borehole pressure, and sensitivity and sensitiveness. Numerous other properties can be specified for explosives but have not been included here because of their lack of importance to the field blaster.

Strength

The strength of explosives has been expressed in various terms since the invention of dynamite. The terms "weight strength" and "cartridge strength," which originally indicated the percentage of nitroglycerin in an explosive, were useful when nitroglycerin was the principal energy-producing ingredient in explosives. However, with the development of products with decreasing proportions of nitroglycerin, these strength ratings have become misleading and inaccurate (4) and do not realistically compare the effectiveness of various explosives.

More recently, calculated energy values have been used to compare the strengths of explosives with AN-FO being used as a base of 1.0. Although this system has not been universally adopted, it is an improvement over weight strength and cartridge strength in estimating the work an explosive will do. Other strength rating systems such as seismic execution value, strain pulse measurement, cratering, and the ballistic mortar have been used, but do not give a satisfactory prediction of the field performance of an explosive.

Underwater tests have been used to determine the shock energy and expanding gas energy of an explosive. These two energy values have been used quite successfully by explosives manufacturers in predicting the capability of an explosive to break rock.
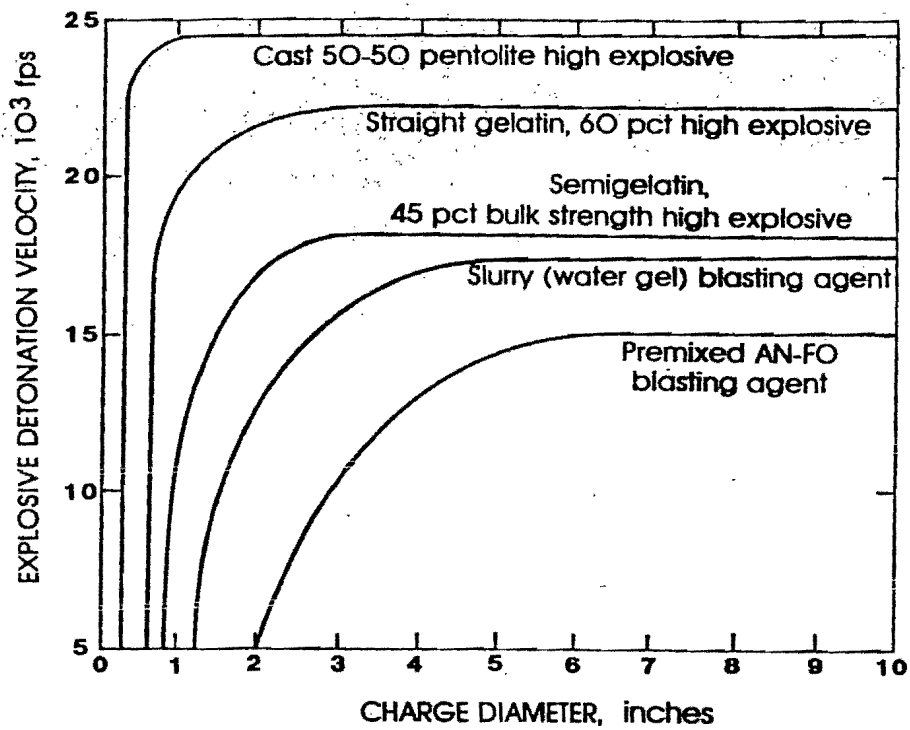
Figure 13: Effect of charge diameter on detonation velocity.

Detonation Velocity

Detonation velocity is the speed at which the detonation front moves through a column of explosives. It ranges from about 5,500 to 25,000 fps for products used commercially today. A high detonation velocity gives the shattering action that many experts feel is necessary for difficult blasting conditions, whereas low-velocity products are normally adequate for the less demanding requirements typical of most blasting jobs. Detonation velocity, particularly in modern dry blasting agents and slurries, may vary considerably depending on field conditions. Detonation velocity can often be increased by the following (5):

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1. Using a larger charge diameter (see figure 13).
2. Increasing density (although excessively high densities in blasting agents may seriously reduce sensitivity).
3. Decreasing particle size (pneumatic injection of AN-FO in small diameter boreholes accomplishes this).
4. Providing good confinement in the borehole.
5. Providing a high coupling ratio (coupling ratio is the percentage of the borehole diameter filled with explosive).
6. Using a larger initiator or primer (this will increase the velocity near the primer but will not alter the steady state velocity).

There is a difference of opinion among experts as to how important detonation velocity is in the fragmentation process. It probably is of some benefit in propagating the initial cracks in hard, massive rock. In the softer, prefractured rocks typical of most operations, it is of little importance.

Density

Density is normally expressed in terms of specific gravity, which is the ratio of the density of the explosive to that of water.

Explosives with a density of less than 1 will float in water. A useful expression of density is loading density, which is the weight of explosive per unit length of charge at a specified diameter, commonly expressed in pounds per foot. Figure 14 shows a nomograph for finding loading density. Cartridge count (number of 1-1/4- by 8-inch cartridges per 50-pound box) is useful when dealing with cartridged high explosives and is approximately equal to 141 divided by the specific gravity. The specific gravity of commercial products ranges from 0.5 to 1.7.

The density of an explosive determines the weight that can be loaded into a given column of borehole. Where drilling is expensive, a more expensive, dense product is frequently justified. The energy-per-unit volume of explosive is actually a more important consideration, although it is not a commonly reported explosive property.

Water Resistance

Water resistance is the ability of an explosive product to withstand exposure to water without losing sensitivity or efficiency. Gelled products such as gelatin dynamites and water gels have good water resistance. Nongelatinized high explosives have poor-to-good water resistance. Ammonium nitrate prills have no water resistance and should not be used in the water-filled portions of a borehole. The mission of brown nitrogen oxide fumes from a blast often indicates inefficient detonation frequently caused by water deterioration, and signifies the need for a more water-resistant explosive or external protection from water in the form of a plastic sleeve or a waterproof cartridge.

Fume Class

Fume class is a measure of the amount of toxic gases, primarily carbon monoxide and oxides of nitrogen, produced by the detonation of an explosive.

Charge diameter, inches

Loading density, Pounds/foot

Specific gravity
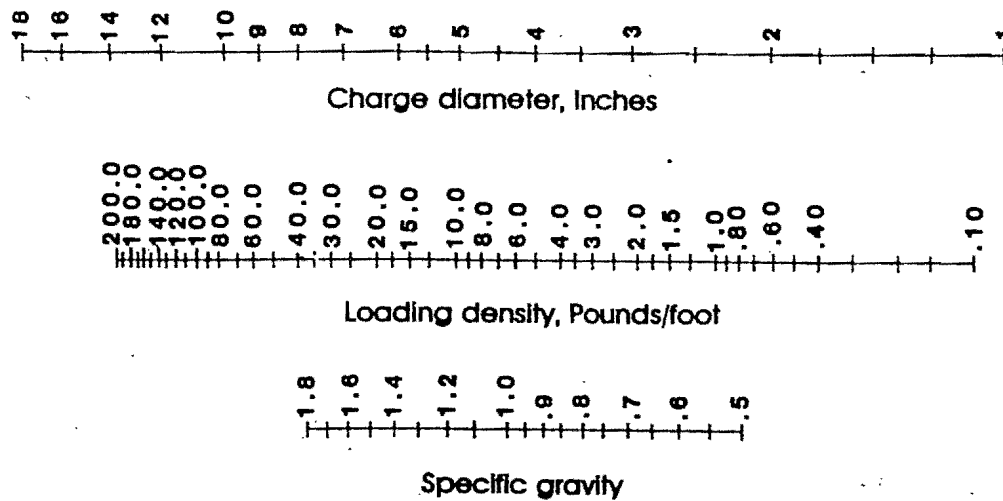
Figure 14: Nomograph for finding loading density.

Most commercial blasting products are oxygen balanced both to minimize fumes and to optimize energy release per unit cost of ingredients. Fumes are an important consideration in tunnels, shafts, and other confined spaces. Certain blasting conditions may produce toxic fumes even with oxygen-balanced explosives. Insufficient charge diameter, inadequate priming or initiation, water deterioration, removal of wrappers, or the use of plastic borehole liners all increase the likelihood of generating toxic gases. Table 2 shows fume classes adopted by the Institute of Makers of Explosives (7). MSHA standards limit the volume of poisonous gases produced by a permissible explosive to 2.5 cubic feet/pound of explosive.

Table 2—Fume Classes Designated by the Institute of Makers of Explosives
(Bichel gage method)

Fume class	Cubic foot of poisonous gas per 200 g of explosive
1	0.16
2	0.16-0.33
3	0.33-0.67

Detonation Pressure

The detonation pressure of an explosive is primarily a function of the detonation velocity squared times the density. It is the head-on pressure of the detonation wave propagating through the explosive column, measured at the C-J plane (figure 2). Although the relationship of detonation velocity and density to detonation pressure is somewhat complex, and depends on the ingredients of an explosive, the following approximation is one of several that can be made (4):

$$P = 4.18 \times 10^{-7} DC^2 / (1 + 0.8\,D),$$

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where P = detonation pressure, in kilobars, (1 kb = 14,504 psi),
      D = specific gravity,
and   C = detonation velocity, in fps.

The nomograph in figure 15, based on this formula, can be used to approx-
imate the detonation pressure of an explosive when the detonation velocity and
specific gravity are known. Some authorities feel that a high detonation
pressure resulting in a strong shock wave is of major importance in breaking
very dense, competent rock. Others, including Swedish experts (8), feel that
it is of little or no importance. As a general recommendation, in hard, mas-
sive rock, if the explosive being used is not giving adequate breakage, a
higher velocity explosive (hence, a higher detonation pressure explosive) may
alleviate the problem. Detonation pressures for commercial products range
from about 5 to over 150 kb.

Detonation velocity 10³feet per second

Detonation pressure, kb

Specific gravity
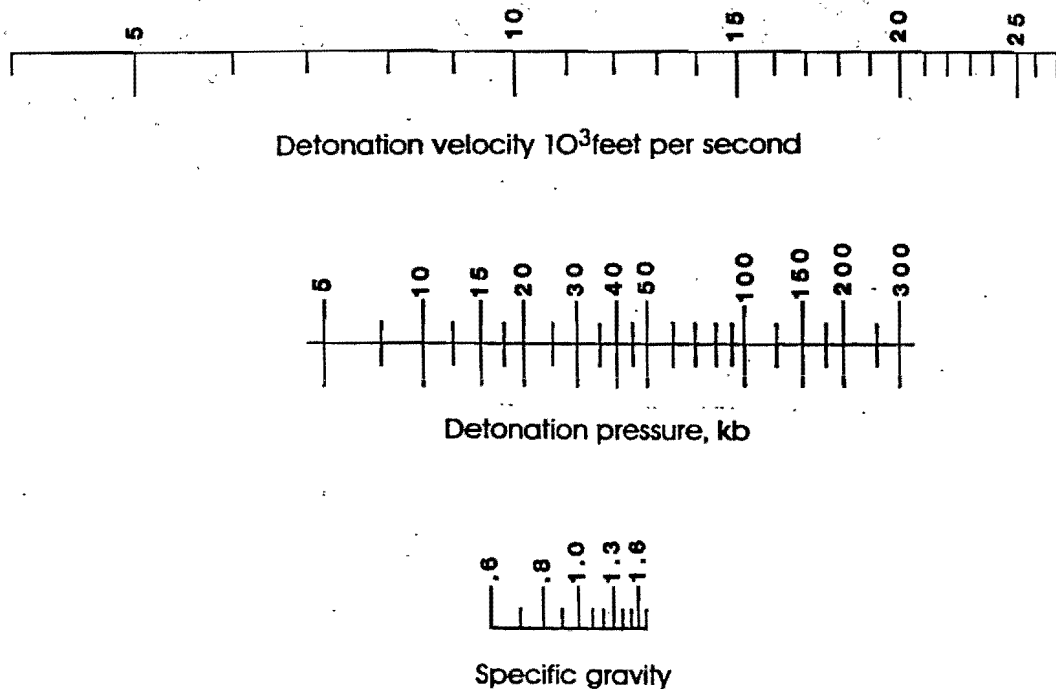
Figure 15: Nomograph for finding detonation pressure.

Borehole Pressure

Borehole pressure, sometimes called explosion pressure, is the pressure
exerted on the borehole walls by the expanding gases of detonation after the
chemical reaction has been completed. Borehole pressure is a function of con-
finement and the quantity and temperature of the gases of detonation. Borehole
pressure is generally considered to play the dominant role in breaking most
rocks and in displacing all types of rocks encountered in blasting. This
accounts for the success of AN-FO and aluminumized products which yield low
detonation pressures but relatively high borehole pressures. The 100 percent
coupling obtained with these products also contributes to their success. Bore-
hole pressures for commercial products range from less than 10 to 60 kb or
more. Borehole pressures are calculated from hydrodynamic computer codes or

approximated from underwater test results, since borehole pressure cannot be measured directly. Many AN-FO mixtures have borehole pressures larger than their detonation pressures. In most high explosives the detonation pressure is the greater.

A Swedish formula (8) for comparing the relative rock-breaking capability of explosives is

$$S = 1/6 \ (V_x/V_o) + 5/6 \ (Q_x/Q_o),$$

where S is the strength of the explosive, V is the reaction product gas volume, Q is the heat energy, the subscript x denotes the explosive being rated, and the subscript o denotes a standard explosive. This corresponds closely to the borehole pressure of an explosive. Although the complexity of the fragmentation process precludes the use of a single property for rating explosives, more and more explosives engineers are relying on borehole pressure as the single most important descriptor in evaluating an explosive's rock-breaking capability.

Sensitivity and Sensitiveness

There are two closely related properties that have become increasingly important with the advent of dry blasting agents and slurries, which are less sensitive than dynamites. Sensitivity to a No. 8 test blasting cap, under certain test conditions, means that a product is classified as an explosive. Lack of cap sensitivity results in a classification as a blasting agent. Sensitivity among different types of blasting agents varies considerably and depends on ingredients, particle size, density, charge diameter, confinement, the presence of water, and often, particularly with slurries, temperature (2). Manufacturers often specify a minimum recommended primer for their products, based on field data. In general, products that require larger primers are less susceptible to accidental initiation and are safer to handle.

Sensitiveness is the capability of an explosive to propagate a detonation once it has been initiated. Extremely sensitive explosives, under some conditions, may propagate from hole to hole. An insensitive explosive may fail to propagate throughout its charge length if its diameter is too small. Sensitiveness is closely related to critical diameter, which is the smallest diameter at which an explosive will propagate a stable detonation. Manufacturers' technical data sheets give recommended minimum diameters for individual explosives.

EXPLOSIVE SELECTION CRITERIA

Proper selection of the explosive is an important part of blast design needed to assure a successful blasting program (6). Explosive selection is dictated by economic considerations and field conditions. The blaster should select a product that will give the lowest cost per unit of rock broken, while assuring that fragmentation and displacement of the rock are adequate for the job at hand. Factors which should be taken into consideration in the selection of an explosive include explosive cost, charge diameter, cost of drilling, fragmentation difficulties, water conditions, adequacy of ventilation, atmospheric temperature, propagating ground, storage consideration, sensitivity considerations, and explosive atmospheres.

Explosive Cost

No other explosive product can compete with AN-FO on the basis of cost per unit of energy. Both of the ingredients--ammonium nitrate and fuel oil--are relatively inexpensive; both participate fully in the detonation reaction, and the manufacturing process consists of simply mixing a solid and a liquid ingredient (figure 16). The safety and ease of storage, handling, and bulk loading add to the attractive economics of AN-FO. It is because of these economics that AN-FO now accounts for approximately 80 percent, by weight, of all the explosives used in the United States. By the pound, slurry costs range from slightly more than AN-FO to about four times the cost of AN-FO. The cheaper slurries are designed for use in large-diameter blastholes and contain no high-cost, high-energy ingredients. They are relatively low in energy per pound. The more expensive slurries are those designed to be used in small diameters, and high-energy products containing large amounts of aluminum or other high-energy ingredients. Dynamite cost ranges from four to six times that of AN-FO, depending largely on the proportion of nitroglycerin or other explosive oil.

Figure 16: Field mixing of AN-FO

Despite its excellent economics, AN-FO is not always the best product for the job because it has several shortcomings. AN-FO has no water resistance, it has a low specific gravity and, under adverse field conditions, it tends to detonate inefficiently. Following are additional factors that should be taken into account when selecting an explosive.

Charge Diameter

The dependability and efficiency of AN-FO are sometimes reduced at smaller charge diameters, especially in damp conditions or with inadequate confinement. In diameters under 2 inches, AN-FO functions best when pneumatically loaded into a dry blasthole. When using charge diameters smaller than 2 inches, many blasters prefer the greater dependability of a cartridged slurry or dynamite despite the higher cost. The cost saving that AN-FO offers can be lost through one bad blast.

At intermediate charge diameters, between 2 and 4 inches, the use of dynamite is seldom justified because AN-FO and slurries function quite well at these diameters. Slurries designed for use in intermediate charge diameters are somewhat cheaper than small-diameter slurries and are more economical than dynamite. The performance of AN-FO in a 4-inch diameter blasthole is substantially better than 2 inches. Where practical, bulk loading in intermediate charge diameters offers attractive economics.

In blasthole diameters larger than 4 inches, a bulk-loaded AN-FO or slurry should be used unless there is some compelling reason to use a cartridged product. AN-FO's efficiency and dependability increase as the charge diameter increases. Where the use of a slurry is indicated, low-cost varieties function well in large charge diameters.

Cost of Drilling

Under normal drilling conditions, the blaster should select the lowest cost explosive that will give adequate, dependable fragmentation. However, when drilling costs increase, typically in hard, dense rock, the cost of explosive and the cost of drilling should be optimized through controlled, in-the-mine experimentation with careful cost analysis. Where drilling is expensive, the blaster will want to increase the energy density of the explosive, even though explosives with high-energy densities tend to be more expensive. Where dynamites are used, gelatin dynamites will give higher energy densities than granular dynamites. The energy density of a slurry depends on its density and the proportion of high-energy ingredients, such as aluminum, used in its formulation. Because of the diverse varieties of slurries on the market, the individual manufacturer should be consulted for a recommendation on a high-energy slurry.

In small-diameter blastholes, the density of AN-FO may be increased by up to 20 percent by high-velocity pneumatic loading. The loading density (weight per foot of borehole) of densified AN-FO cartridges is about the same as that of bulk AN-FO because of the void space between the cartridge and the borehole wall. The energy density of AN-FO can be increased by the addition of finely divided aluminum. The economics of aluminized AN-FO improve where the rock is more difficult to drill and blast.

Fragmentation Difficulties

Expensive drilling and fragmentation difficulties frequently go hand-in-hand because hard, dense rock may cause both. Despite the controversy as to the importance of detonation velocity in rock fragmentation, there is evidence that a high velocity does help in fragmenting hard, massive rock (10). with

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cartridged dynamites, the detonation velocity increases as the nitroglycerin content increases, with gelatin dynamites having higher velocities than their granular counterparts. Several varieties of slurry, and particularly emulsions, have high velocities. The individual manufacturer should be consulted for a recommendation on a high-velocity product. In general, emulsions exhibit higher velocities than water gels.

The detonation velocity of AN-FO is highly dependent on its charge diameter and particle size. In diameters of 9 inches or greater, AN-FO's detonation velocity will normally exceed 13,000 fps, peaking near 15,000 fps in a 15-inch diameter. These velocities compare favorably with velocities of most other explosive products. In smaller diameters, the detonation velocity falls off, until at diameters below 2 inches the velocity is less than half the 15,000-fps maximum. In these small diameters, the velocity may be increased to nearly 10,000 fps by high-velocity pneumatic loading, which pulverizes the AN-FO and gives it a higher loading density. As a cautionary note, pressures higher than 30 pounds per square inch (psi) should never be used with a pressure vessel pneumatic loader. Full line pressures of 90 to 110 psi are satisfactory for ejectors. In many operations with expensive drilling and difficult fragmentation, it may be advantageous for the blaster to compromise and use a dense, high-velocity explosive in the lower position of the borehole and AN-FO as a top load.

Water Conditions

AN-FO has no water resistance. It may, however, be used in blastholes containing water if one of two techniques is followed. First, the AN-FO may be packaged in a water-resistant, polyburlap container. To enable the AN-FO cartridge to sink in water, part of the prills are pulverized and the mixture is vibrated to a density of about 1.1 g/cm^3. Of course, if a cartridge is ruptured during the loading process, the AN-FO will quickly become desensitized. In the second technique, the blasthole is dewatered by using a down-the-hole submersible pump (3). A waterproof liner is then placed into the blasthole and AN-FO is loaded inside the liner before the water reenters the hole. Again, the AN-FO will quickly become desensitized if the borehole liner is ruptured. The appearance of orange-brown nitrogen oxide fumes upon detonation is a sign of water deterioration, and an indication that a more water-resistant product or better external protection should be used.

Slurries are gelled and cross-linked to provide a barrier against water intrusion, and as a result, exhibit excellent water resistance. The manufacturer will usually specify the degree of water resistance of a specific product. When dynamites are used in wet holes, gelatinous varieties are preferred. Although some granular dynamites have fair water resistance, the slightly higher cost of gelatins is more than justified by their increased reliability in wet blastholes.

Adequacy of Ventilation

Although most explosives are oxygen-balanced to maximize energy and minimize toxic detonation gases, some are inherently "dirty" from the standpoint of fumes. Even with oxygen-balanced products, unfavorable field conditions may increase the generation of toxic fumes, particularly when explosives without water resistance get wet. The use of plastic borehole liners, inade-

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quate charge diameters, removal of a cartridged explosive from its wrapper, inadequate priming, or an improper explosive ingredient mix may cause excessive fumes.

In areas where efficient evacuation of detonation gases cannot be assured (normally underground), AN-FO should be used only in absolutely dry conditions. Most small-diameter slurries have very good fume qualities. Large-diameter slurries have variable fume qualities. The manufacturer should be consulted for a recommendation where fume control is important. Of the cartridged dynamites, ammonia gelatins and semigelatins have the best fume qualities. High-density ammonia dynamites are rated good, low-density ammonia dynamites are fair, and straight dynamites are poor, as shown in table 1. In permissible blasting, where fumes are a concern, care should be exercised in selecting the explosives because many permissibles have poor fume ratings. Permissibles with good fume ratings are available.

Atmospheric Temperature

Until the development of slurries, atmospheric temperatures were not an important factor in selecting an explosive. For many years, dynamites have employed low-freezing explosive oils which permit their use in the lowest temperatures encountered in the United States. AN-FO and slurries are not seriously affected by low temperatures if priming is adequate. A potential problem exists with slurries that are designed to be cap sensitive. At low temperatures, many of these products may lose their cap sensitivity, although they will still function well if adequately primed. If a slurry is to be used in cold weather, the manufacturer should be asked about the temperature limitation on the product.

The effect of temperature is alleviated if explosives are stored in a heated magazine or if they are in the borehole long enough to achieve the ambient borehole temperature. Except in permafrost or in extremely cold weather, borehole temperatures are seldom low enough to render slurries insensitive.

Propagating Ground

Propagation is the transfer or movement of a detonation from one point to another. Although propagation normally occurs within an explosive column, it may occur between adjacent blastholes through the ground. In ditch blasting, a very sensitive straight nitroglycerin dynamite is sometimes used to purposely accomplish propagation through the ground. This saves the cost of putting a detonator into each blasthole. Propagation ditch blasting works best in soft, water-saturated ground.

In all other types of blasting, propagation between holes is undesirable because it negates the effect of delays. Propagation between holes will result in poor fragmentation, failure of a round to pull properly, and excessive ground vibrations, airblast, and flyrock. In underground blasting, the entire round may fail to pull. The problem is most serious when using small blastholes loaded with dynamite. Small blastholes require small burdens and spacings, increasing the chance of hole-to-hole propagation, particularly when sensitive explosives are used. Water-saturated material and blasthole deviation compound the problem. When propagation is suspected, owing to poor frag-

mentation, violent shots, or high levels of ground vibrations, the use of a less sensitive product usually solves the problem. Straight nitroglycerin dynamite is the most sensitive commercial explosive available, followed by other granual dynamites, gelatin dynamites, cap-sensitive slurries, and blasting agents, in decreasing order of sensitivity.

A different problem can occur when AN-FO or slurry blasting agents are used at close spacings in soft ground. The shock from an adjacent charge may dead press a blasting agent column and cause it to misfire.

Storage Considerations

Federal requirements for magazine construction are less stringent for blasting agents than for high explosives (13). Magazines for the storage of high explosives must be well ventilated and must be resistant to bullets, fire, weather, and theft, whereas a blasting agent magazine need only be theft resistant. Although this is not an overriding reason for selecting a blasting agent rather than an explosive, it is an additional point in favor of blasting agents.

Some activities such as power line installation and light construction require the periodic use of very small amounts of explosives. In this type of work, the operator can use two-component explosives advantageously. Two-component explosives are sold as separate ingredients, neither of which is explosive. The two components are mixed at the jobsite as needed, and the mixture is considered a high explosive. Persons who mix two-component explosives are often required to have a manufacturer's license.

Federal regulations do not require ingredients of two-component explosives to be stored in magazines nor is there a minimum distance requirement for separation of the ingredients from each other or from explosive products. Even though there is no federal regulation requiring magazine storage, two-component explosives should be protected from theft.

The use of two-component explosives eliminates the need for frequent trips to a magazine. However, when large amounts of explosives are used, the higher cost and the time-consuming process of explosive mixing begin to outweigh the savings in travel time.

Sensitivity Considerations

Sensitivity considerations address questions of the safety and the dependability of an explosive. More sensitive explosives, such as dynamites are somewhat more vulnerable to accidental initiation by impact or spark than blasting agents. Slurries and nitrostarch-based explosives are generally less sensitive to impact than nitroglycerin-based dynamites. However, more sensitive explosives, when all conditions are equal, are less likely to misfire in the blasthole. For example, upon accidental impact from a drill bit, a blasting agent is less likely to detonate than a dynamite. This does not mean that the blasting agent will not detonate when accidentally impacted. Conversely, under adverse situations, such as charge separation in the blasthole, very small charge diameters, or low temperatures, dynamites are less likely to misfire than blasting agents. This tradeoff must be considered primarily when selecting an explosive for small-diameter work. Other selection criteria

usually dictate the use of blasting agents when the blasthole diameter is large.

It can be concluded from 1981 explosive consumption figures (12) and field observations that most of the dynamite still used in this country is used in construction, small quarries, and underground mines, where many operators consider a more sensitive explosive beneficial in their small-diameter blasting. When safely handled and properly loaded, dynamites, dry blasting agents, and slurries all have a place in small-diameter blasting.

Explosive Atmospheres

Blasting in a gassy atmosphere can be catastrophic if the atmosphere is ignited by the flame from the explosive. All underground coal mines are classified as gassy; some metal-nonmetal mines may contain methane or other explosive gases; and many construction projects encounter methane. Where gassy conditions are suspected, MSHA or OSHA should be consulted for guidance.

Permissible explosives (14) offer protection against gas explosions. Most permissible explosives are relatively weak explosives, and will not do an adequate job in most rock, although some relatively powerful permissible gelatins, emulsions, and slurries are available.

All underground coal mines are classified as gassy by MSHA, and permissible explosives are the only type of explosives that can be used in these mines without a variance from MSHA. Salt, limestone, uranium, potash, copper, trona, and oil shale mines may contain methane or other explosive gases and may be classified gassy on an individual basis by MSHA. In these gassy metal-nonmetal mines, MSHA may permit the use of nonpermissible products such as AN-FO, detonating cord, and certain other high explosives and blasting agents. These mines are required to operate under modified permissible rules developed by MSHA on a mine-by-mine basis.

REFERENCES

1. Cook, M.A. Explosives--A Survey of Technical Advances. Ind. and Eng. Chem., v. 60, No. 7, July 1968, pp. 44-55.
2. Damon, G.H., C.M. Mason, N.E. Hanna, and D.R. Forshey. Safety Recommendations for Ammonium Nitrate-Based Blasting Agents. BuMines IC 8746, 1977, 31 pp.
3. Dannenberg, J. Blasthole Dewatering Cuts Costs. Rock Products, v. 76, No. 12, December 1973, pp. 66-68.
4. Dick, R.A. Factors in Selecting and Applying Commercial Explosives and Blasting Agents. BuMines IC 8405, 1968, 30 pp.
5. _____. The Impact of Blasting Agents and Slurries on Explosives Technology. BuMines IC 8560, 1972, 44 pp.
6. Drury, F., and D.G. Westmaas. Considerations Affecting the Selection and Use of Modern Chemical Explosives. Proc. 4th Conf. on Explosives and Blasting Technique, New Orleans, LA, Feb. 1-3, 1978. Society of Explosives Engineers, Montville, OH, pp. 12-153.
7. E.I. du Pont de Nemours & Co., Inc. (Wilmington, DE). Blaster's Handbook. 16th ed., 1978, 494 pp.
8. Johansson, C.H. and U. Langefors. Methods of Physical Characterization of Explosives. Proc. 36th Internat. Cong. of Ind. Chem., Brussels, v. 3,

1966, p. 610; available for consultation at Bureau of Mines Twin Cities
Research Center, Minneapolis, MN.

9. Monsanto Co. (St. Louis, MO). Monsanto Blasting Products AN-FO Manual.
    Its explosive Properties and Field Performance Characteristics.
    September 1972, 37 pp.

10. Porter, D.D. Use of Fragmentation to Evaluate Explosives for Blasting.
    Min. Cong. J., v. 60, No. 1, January 1974, pp. 41-43.

11. Robinson, R.V. Water Gel Explosives--Three Generations. Canadian Min.
    and Met. Bull., v. 62, No. 692, December 1969, pp. 1317-1325.

12. U.S. Bureau of Mines. Apparent Consumption of Industrial Explosives and
    Blasting Agents in the United States, 1981. Mineral Industry Survey,
    June 23, 1982, 12 pp.

13. U.S. Department of the Treasury; Bureau of Alcohol, Tobacco, and
    Firearms. Explosive Materials Regulations. Federal Register, v. 42, No.
    149, Aug. 3, 1977, pp. 39316-39327; Federal Register, v. 45, No. 224,
    Nov. 18, 1980, pp. 76191-76209.

14. U.S. Mine Enforcement and Safety Administration. Active List of
    Permissible Explosives and Blasting Devices Approved Before Dec. 31,
    1975, MESA Inf. Rep. 1046, 1976, 10 pp.

CHAPTER 1
REVIEW QUESTIONS

1. Explosives with a density of less than 1 will _____ in water.

2. Water resistance of an explosive may be improved by _____

    _____.

3. A blasting agent is _____.

4. Bulk AN-FO (specific gravities between 0.80 to 0.85) should not be loaded

    in wet holes because _____

    _____.

5. When using AN-FO under wet conditions, it should be _____

    _____.

6. Seventy percent dynamite has approximately _____ percent nitroglycerin.

7. A primer is _____ and contains a(an)

    _____.

8. The letters AN-FO stand for _____.

9. List 8 criteria used in selecting an explosive.

    a. _____

    b. _____

    c. _____

    d. _____

    e. _____

    f. _____

    g. _____

    h. _____

10. What is the effect of using less than 94.5 percent ammonium nitrate in an

    AN-FO mixture? _____

    _____.

11. What are 2 major classes of explosives? _____ and _____ .

12. Permissible explosives are most often used for _____ mining.

13. Flaked aluminum is often added to dry blasting agents to _____
_____.

14. Propagation is _____.

15. Cap sensitivity may _____ with

low temperatures and the blaster must be sure the explosive is adequately

_____.

16. All _____ mines are classified as gassy and only

_____ explosives may be used without a variance from

MSHA.

Chapter 2:   Initiation and Priming

INITIATION SYSTEMS

A considerable amount of energy is required to initiate a high explosive such as a dynamite or cap-sensitive slurry. In blasting, high explosives are initiated by a detonator, which is a capsule containing a series of relatively sensitive explosives that can be readily initiated by an outside energy source. Blasting agents, which are the most common products used as the main column charge in the blasthole, are even less sensitive to initiation than high explosives. To assure dependable initiation of these products, the initiator is usually placed into a container of high explosives, which in turn is placed into the column of blasting agent.

An initiation system consists of three basic parts.

1.   An initial energy source.
2.   An energy distribution network that conveys energy into the individual blastholes.
3.   An in-the-hole component that uses energy from the distribution network to initiate a cap-sensitive explosive.

The initial energy source may be electrical, such as a generator or condenser-discharge blasting machine, a power line used to energize an electric blasting cap, or a heat source such as a spark generator or a match. The energy conveyed to and into the individual blastholes may be electricity, a burning fuse, a high-energy explosive detonation, or a low-energy dust or gas detonation. Figure 17 shows a typical detonator or "business end" of the initiation system. This detonator, when inserted into a cap-sensitive explosive and activated, will initiate the detonation of the explosive column. Commercial detonators vary in strength from No. 7 to No. 12. Although No. 6 and No. 8 detonators are the most common, there is a trend toward higher strength detonators, particularly when blasting with cap-sensitive products which are less sensitive than dynamites.

The primer is the unit of cap-sensitive explosive containing the detonator. Where the main blasthole charge is high explosive, the detonator may be inserted into the column at any point. However, most of the products used for blasting today (blasting agents) are insensitive to a No. 8 detonator. To detonate these products, the detonator must be inserted into a unit of cap-sensitive explosive, which in turn is inserted into the blasting agent column at the desired point of initiation.

The discussions of the various initiation and priming systems will concentrate primarily on common practice. With each system there are optional techniques and "tricks of the trade" that increase system versatility. It is
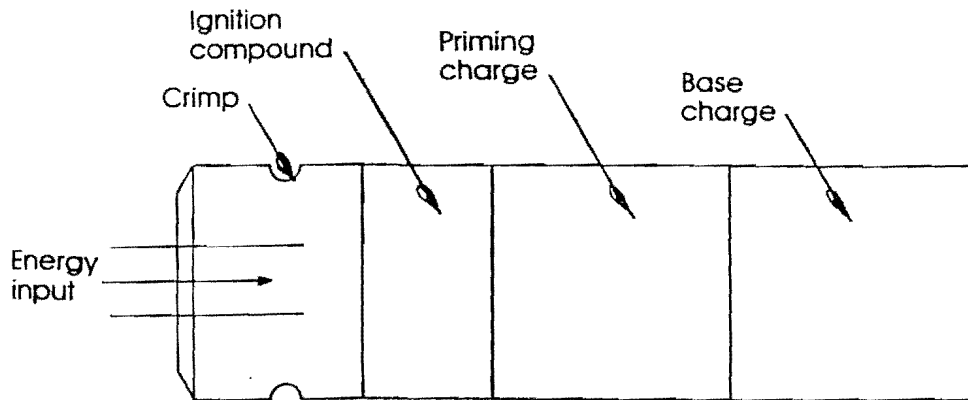
Figure 17: Instantaneous detonator.

a good idea to confer with the manufacturer before finalizing your initiation and priming program, so you fully understand how to best use a specific system.

DELAY SERIES

Figure 17 shows an instantaneous detonator. In this type of detonator, the base charge detonates within a millisecond or two after the external energy enters the detonator. However, in most types of blasting, time intervals are required between the detonation of various blastholes or even between decks within a blasthole. To accomplish this, a delay element containing a burning powder is placed immediately before the priming charge in the detonator. Figure 18 shows a delay detonator.
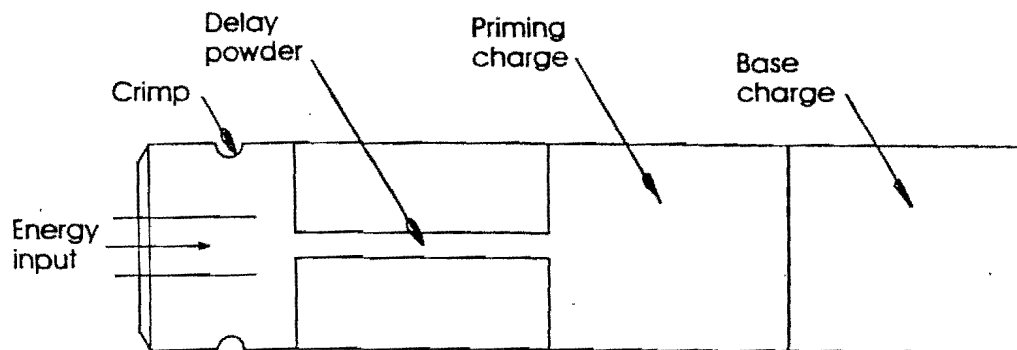
Figure 18: Delay detonator.

There are three basic delay series: slow or tunnel delays, fast or millisecond delays, and coal mine delays for use in underground coal mines. For all commercial delay detonators, the delay time is determined by the length and burning rate of the delay powder column. As a result, slow delay

caps may be quite long in dimension, whereas lower period millisecond delays are shorter. Although the timing of delay detonators is sufficiently accurate for most blasting needs, these delays are not precise, as indicated by recent research. Recently, however, manufacturers' tolerances for some delay caps have been tightened. It is important to use the manufacturers' recommended current level to initiate electric blasting caps. Current levels above or below the recommended firing level can further increase the scatter in delay cap firing times. Extremely high currents can speed up delay firing times. Near the minimum firing current, delays can become extremely erratic.

Slow delays are useful underground under tight shooting conditions where it is essential that the burden on one hole moves before a subsequent hole fires. This situation may occur in tunnels, shafts, underground metal-non-metal mines, and in trenching. Slow delays are available with an initiation systems except surface detonating cord and delay cast primers. Delay intervals are typically 0.5 to 1 seconds.

Millisecond delays are the most commonly used delays and are useful wherever the tight conditions previously mentioned are not present. Millisecond delays provide improved fragmentation, controlled throw, and reduced ground vibration and airblast, as compared with simultaneous firing. They are available with all initiation systems. In millisecond (ms) detonators, delay intervals are 25 to 50 ms in the lower periods and are longer in the higher periods. In detonating cord delay connectors, the delay may be as short as 5 ms.

Coal mine delays are a special series of millisecond delays. Because only electric initiation systems are permissible in underground coal mines, coal mine delays are available only with electric initiators. Delay intervals are from 50 to 100 ms; instantaneous caps are prohibited. Coal mine delay caps always use copper-alloy shells and iron leg wires. Iron leg wires are also available optionally with ordinary electric detonators and are used primarily to facilitate magnetic removal of the wires from the muck pile, such as in trona and salt mines.

ELECTRIC INITIATION

Electric initiation has been used for many years in both surface and underground blasting. An electric blasting cap (figure 19) consists of two insulated leg wires that pass through a waterproof seal and into a metal capsule containing a series of explosive powders (figure 20). Leg wires of various lengths are available to accommodate various borehole depths. Inside the capsule the two leg wires are connected by a fine filament bridge wire embedded in a highly heat-sensitive explosive. Upon application of electric current, the bridge wire heats sufficiently to initiate the ignition mixture, which in turn initiates a series of less sensitive, more powerful explosives. Detonators are available in strengths ranging from about No. 6 to No. 12, with No. 6 and No. 8 being most common. Trends recently are toward higher strength detonators.

Most electric blasting caps have copper leg wires. Iron leg wires are available for use where magnetic separation is used to remove the leg wires at the preparation plant. Atlas Powder Co. has prepared an excellent handbook that describes electric blasting procedures in detail (2). (Note: Reference

to specific trade names or manufacturers does not imply endorsement by the Bureau of Mines or the State of Montana).
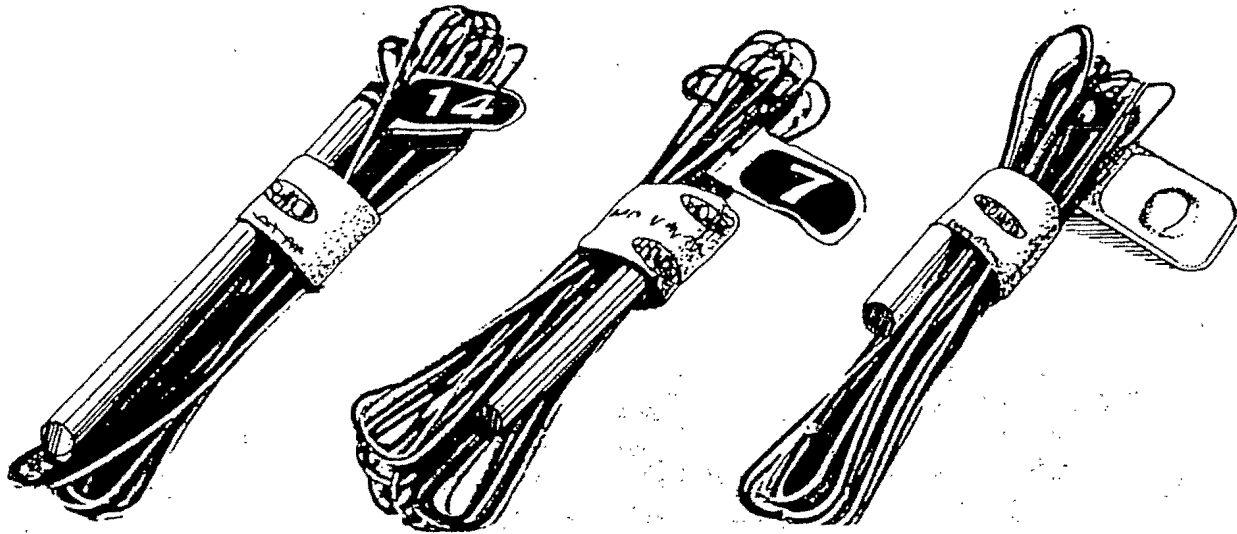
Figure 19: Electric blasting caps.
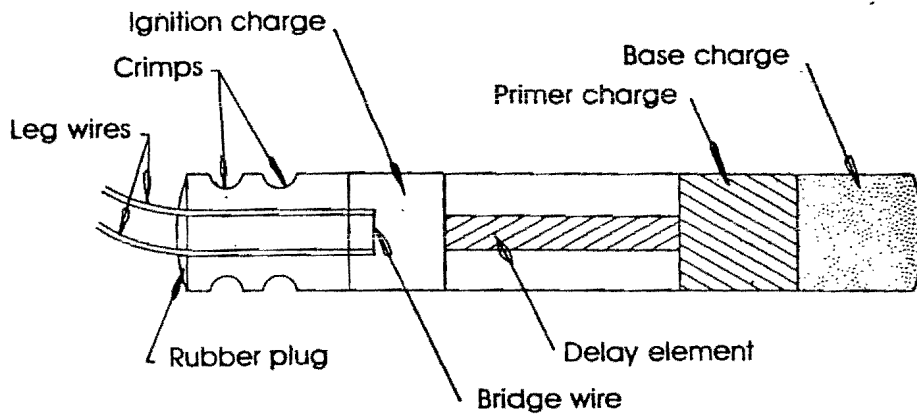
Figure 20: Delay electric blasting cap.

The Saf-T-Det and Magnadet electric blasting caps are two recent developments. The Saf-T-Det resembles a standard electric blasting cap but has no base charge. A length of 100-grain or less detonating cord is inserted into a well to act as a base charge just before the primer is made up. The device is similar to an electric blasting cap in regard to required firing currents and extraneous electricity hazards. The Saf-T-Det is manufactured in India and is available in the United States.

The Magnadet is also similar to a standard electric blasting cap, except that the end of each cap lead contains a plastic-covered ferrite toroidal

ring. The system is hooked up by passing a single wire through each ring. A special blasting machine is used to fire these detonators. The manufacturer, ICI of Scotland, claims ease of hookup and protection against extraneous electricity as advantages of this system.

Types of Circuits

In order to fire electric blasting caps, the caps must be connected into circuits and energized by a power source. There are three types of electric blasting circuits (figure 21). In order of preference they are series, parallel series, and parallel. In series circuits, all the caps are connected consecutively so that the current from the power source has only one path to follow. The series circuit is recommended because of its simplicity. Also, all the caps receive the same amount of current.

Figure 22 shows recommended wire splices for blasting circuits. To splice two small wires, the wires are looped and twisted together. To connect a small wire to a large wire, the small wire is wrapped around the large wire.

The electrical resistance of a series of caps is equal to the sum of the resistances of the individual caps. For most blasting machines, it is recommended that the number of caps in a single series be limited to 40 to 50, depending on the leg wire length. Longer leg wires require smaller series. The limit for most small twist-type blasting machines is 10 caps with 20-foot leg wires.

Many blasters minimize excess wire between holes to keep the blast site from being cluttered. The ends of the cap series are extended to a point of safety by connecting wire, which is usually 20 gage, but should be heavier where circuit resistance is a problem or when using parallel circuits. This connecting wire is considered expendable and should be used only once. The connecting wire is connected in turn to the firing line, which is connected to the power source.

The firing line contains two single conducting wires of 12 gage or heavier, and is reused from shot to shot. It may be on a reel mechanism for portability, or may be installed along the wall of a tunnel in an underground operation. Installed firing lines should not be grounded, should be made of copper rather than aluminum, and should have a 15-foot lightning gap near the power source to guard against premature blasts. The firing line should be inspected frequently and replaced when necessary.

When the number of caps in a round exceeds 40 to 50, the parallel series circuit is recommended. In a parallel series circuit, the caps are divided into a number of individual series. Each series should contain the same number of caps or the same resistance to assure even current distribution. The leg wires of the caps in each series are connected consecutively. Next, two bus wires, as shown in figure 21, are placed in such a position that each end of each series can be connected as shown in the figure. The bus wire is usually about 14 gage or heavier and may be either bare or insulated. Where bare wires are used, care must be exercised to prevent excessive current leakage to the ground. It is recommended that insulated bus wires be used and that the insulation be cut away at point of connection with the blasting cap series. To assure equal current distribution to each series, one bus wire

38
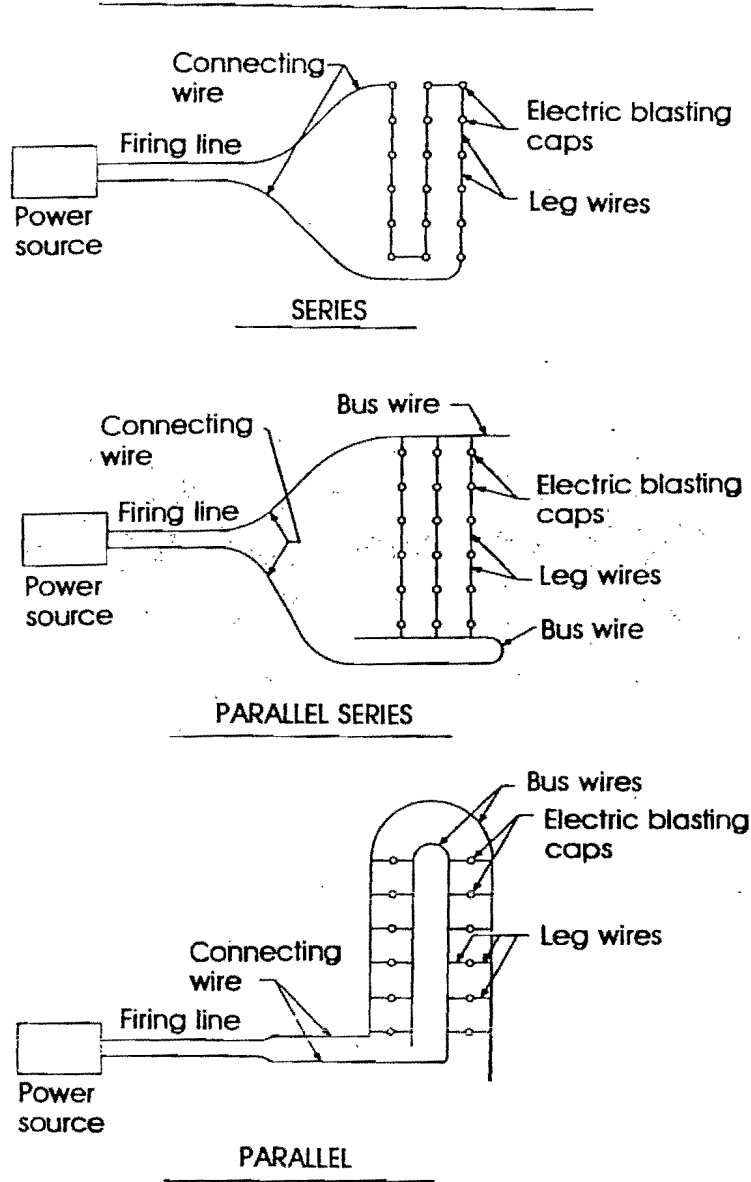
Figure 21: Types of electric blasting circuits.

should be reversed as shown in figure 21. With parallel series circuits, 14 gage or heavier gage connecting wire is used to reduce the total circuit resistance.

The third type of blasting circuit is the straight parallel circuit. The straight parallel circuit is less desirable to use than the series or series parallel circuits for two reasons. First, its nature is such that it cannot be checked. Broken leg wires or faulty connections cannot be detected once the circuit has been hooked up. Second, because the available current is divided by the number of caps in the circuit, power line firing must often be used to provide adequate current for large parallel circuits. The problems associated with power line firing will be discussed later.

Leg Wire   Leg wire or connecting wire

Connecting wire   Firing line

Leg wire   Bus wire

LIGHT GAGE TO LIGHT GAGE (TWISTED LOOP)

LIGHT GAGE TO HEAVY GAGE
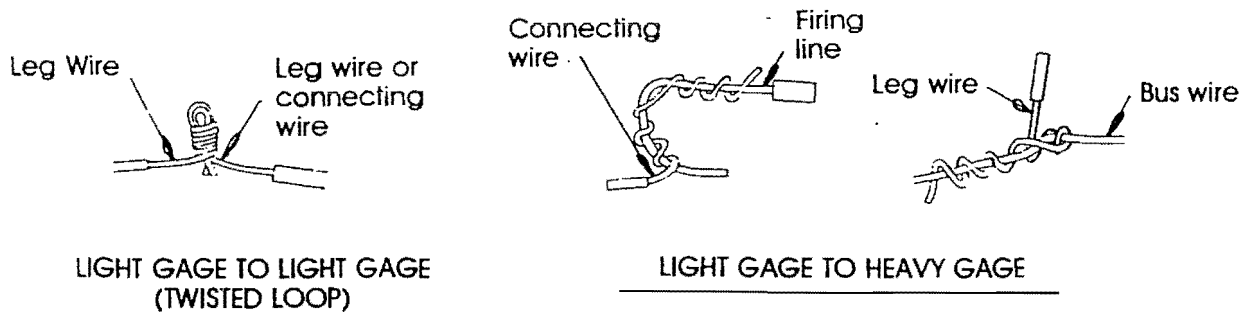
Figure 22: Recommended wire splices.

Parallel circuits are not appropriate for surface blasting but they are used to some extent for tunnel blasting. Parallel circuits are similar to parallel series except that instead of each end of a series circuit being connected to alternate bus wires, each leg wire of each cap is connected directly to the bus wires, as shown in figure 21. In underground blasts using parallel circuits, bare bus wire is usually strung on wooden pegs driven into the face to avoid grounding. As with parallel series circuits, the bus wires are reversed as shown in figure 21.

In a parallel circuit the lead wire (firing line) represents the largest resistance in the circuit. Keeping the lead wire as short as possible, consistent with safety, is the key to firing large numbers of caps with parallel circuits. Doubling the length of the lead wire reduces the number of caps that can be fired by almost half. Heavy (12- to 14-gage) bus wires are used to reduce the resistance.

Circuit Calculations

Only the very basics of circuit calculations are covered here. For more detail on circuit calculations or other of the many intricacies of electrical blasting, the reader should refer to a detailed electric blasting handbook such as reference 2. Figure 23 shows the resistance calculations for cap circuits for series, parallel series, and straight parallel circuits.

The resistance of a series circuit is the easiest to calculate. First, the resistance of a single cap, as specified by the manufacturer, is multiplied by the number of caps to determine the resistance of the cap circuit. To this is added the resistance of the connecting wire and that of the firing line to determine the resistance of the total circuit. The firing line contains two wires; therefore, there will be 2 feet of wire for every foot of firing line. Where bus wire is used (parallel or parallel series circuits) the resistance of one-half of the length of the bus wire is added to find the total circuit resistance. When firing from a power line, the voltage of the line divided by the resistance of the circuit will give the current flow. In

a single series circuit, all of this current flows through each cap. The min-
imum recommended firing current per cap is 1.5 amp dc or 2.0 amp ac. The cur-
rent output of condenser (capacitor) discharge blasting machines may vary with
the circuit resistance, but not linearly. Manufacturer's specifications must
be consulted to determine the amperage of a specific machine across a given
resistance. For a generator blasting machine, the manufacturer rates the
machine in terms of the number of caps it can fire.

The resistance calculation for a parallel series circuit is as follows.
First, the resistance of each cap series is calculated as previously des-
cribed. Remember, in a good parallel series circuit the resistance of each
series should be equal. The resistance of a single series is then divided by

$$R_T = N R_C$$
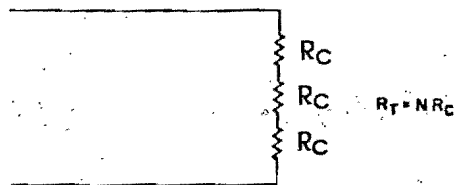
KEY
R_T Total resistance
R_C Resistance of 1 cap
N   Number of caps
N_S Number of series
$N_{1,2,3}$ Number of caps in a series

SIMPLE SERIES

$$\frac{1}{R_T} = \frac{1}{N_1 R_C} + \frac{1}{N_2 R_C} + \frac{1}{N_3 R_C} + \cdots$$

If $N_1 = N_2 = N_3$,

then $R_T = R_C \dfrac{N_1}{N_S}$.

$$R_T = \frac{R_C}{N}$$
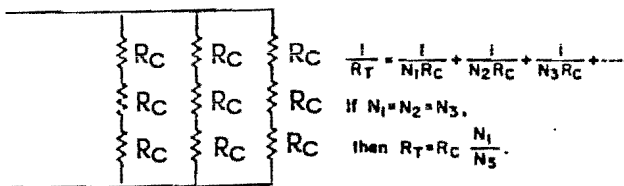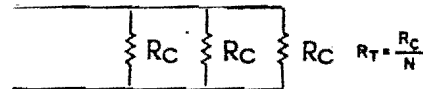
PARALLEL SERIES   PARALLEL

Figure 23: Calculation of cap circuit resistance.

the number of series to find the resistance of the cap circuit. To this are
added the resistance of half the length of bus wire used, the resistance of
the connecting wire, and the resistance of the firing line, to obtain the
total circuit resistance. The locations of the bus wire, connecting wire, and
firing line are shown in figure 21. The current flow is determined either by
dividing the power line voltage by the circuit resistance or in the case of a
condenser discharge machine, by checking the manufacturer's specifications.
The current flow is divided by the number of series to determine the current
flow through each series.

For straight parallel circuits, the resistance of the cap circuit is
equal to the resistance of a single cap divided by the number of caps. As can
readily be seen, this is usually a very small value. For 20 short-leg wire
caps, the resistance is less than 0.1 ohm. The resistances of the connecting
wire, the firing line, and one-half the bus wire are added to find the total

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resistance. The current flow is determined in the same manner as with series and parallel series circuits. The current flow is divided by the number of caps to determine the current flow through each cap.

Power Sources

Electric blasting circuits can be energized by generator-type blasting machines, condenser-discharge blasting machines, and power lines. Storage and dry-cell batteries are definitely not recommended for blasting because they cannot be depended on for a consistent output.

Generator blasting machines may be of the rack-bar (push down) or the key-twist type. The capacity of rack-bar machines ranges from 30 to 50 caps in a single series, while key-twist machines will normally initiate 10 or 20 caps in a single series. The actual current put out by these machines depends on the condition of the machine and the effort exerted by the shot-firer. When using a rack-bar machine, the terminals should be on the opposite side of the machine from the operator. Both the rack-bar and twist machines should be operated vigorously to the end of the stroke because the current flows only at the end of the stroke. Because the condition of a generator blasting machine deteriorates with time, it is important that the machine be checked periodically with a rheostat designed for that purpose. The directions for testing with a rheostat are contained on the rheostat case or on the rheostat itself. Although the generator machine has been a dependable blasting tool, its limited capacity and variable output have caused it to be replaced, for most applications, by the condenser (capacitor) discharge machine (figure 24).

As the name implies, the capacitor discharge (CD) machine employs dry-cell batteries to charge a series of capacitors. The energy stored in the capacitor is then discharged into the blasting circuit. CD machines are available in a variety of designs and capacities, with some capable of firing over 1,000 caps in a parallel series circuit.

All CD machines operate basically in the same manner. One button or switch is activated to charge the capacitors and a second button or switch is activated to fire the blast. An indicator light or dial indicates when the capacitor is charged to its rated capacity. Ideally, the overall condition of a CD blasting machine should be checked with an oscilloscope. However, the current output can be checked by using a specially designed setup combining a rheostat and a resistor (2) or by using a capacitor discharge checking machine (7). The powder supplier should be consulted about the availability of machines for checking CD machines.

A sequential blasting machine (figure 25) is a unit containing 10 CD machines that will fire up to 10 separate circuits with preselected time interval between the individual circuits. When used in conjunction with millisecond-delay electric blasting caps, the sequential machine provides a very large number of separate delay intervals (3), which can be useful in improving fragmentation and in controlling ground vibrations and airblast. Because blast pattern design and hookup can be quite complex, the sequential blasting machine should be used only by well-trained persons or under the guidance of a consultant or a powder company representative. A poorly planned sequential timing pattern will result in poor fragmentation and excessive overbreak, flyrock, ground vibrations, and noise.

42
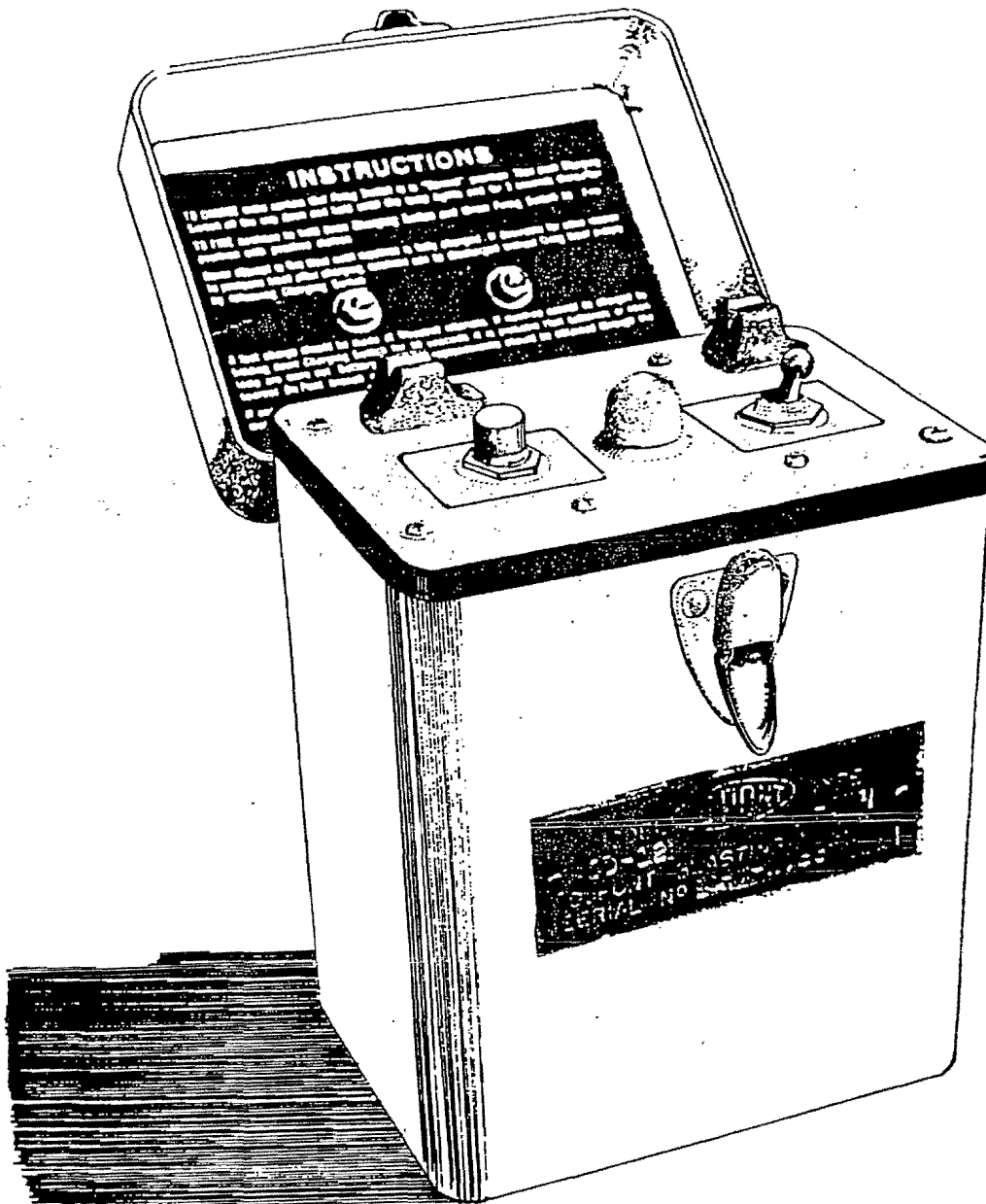
Figure 24: Capacitor discharge blasting machine.

The third alternative for energizing electric blasting circuits is the power line. Power line blasting is often done with parallel circuits where the capacity of available blasting machines is inadequate. When firing off a power line, the line should be dedicated to blasting alone, should contain at least a 15-foot lightning gap, and should be visually checked for damage and for resistance on a regular basis. Power line shooting should not be done unless precautions are taken to prevent arcing. Arcing can result in erratic timing, a hangfire, or a misfire.
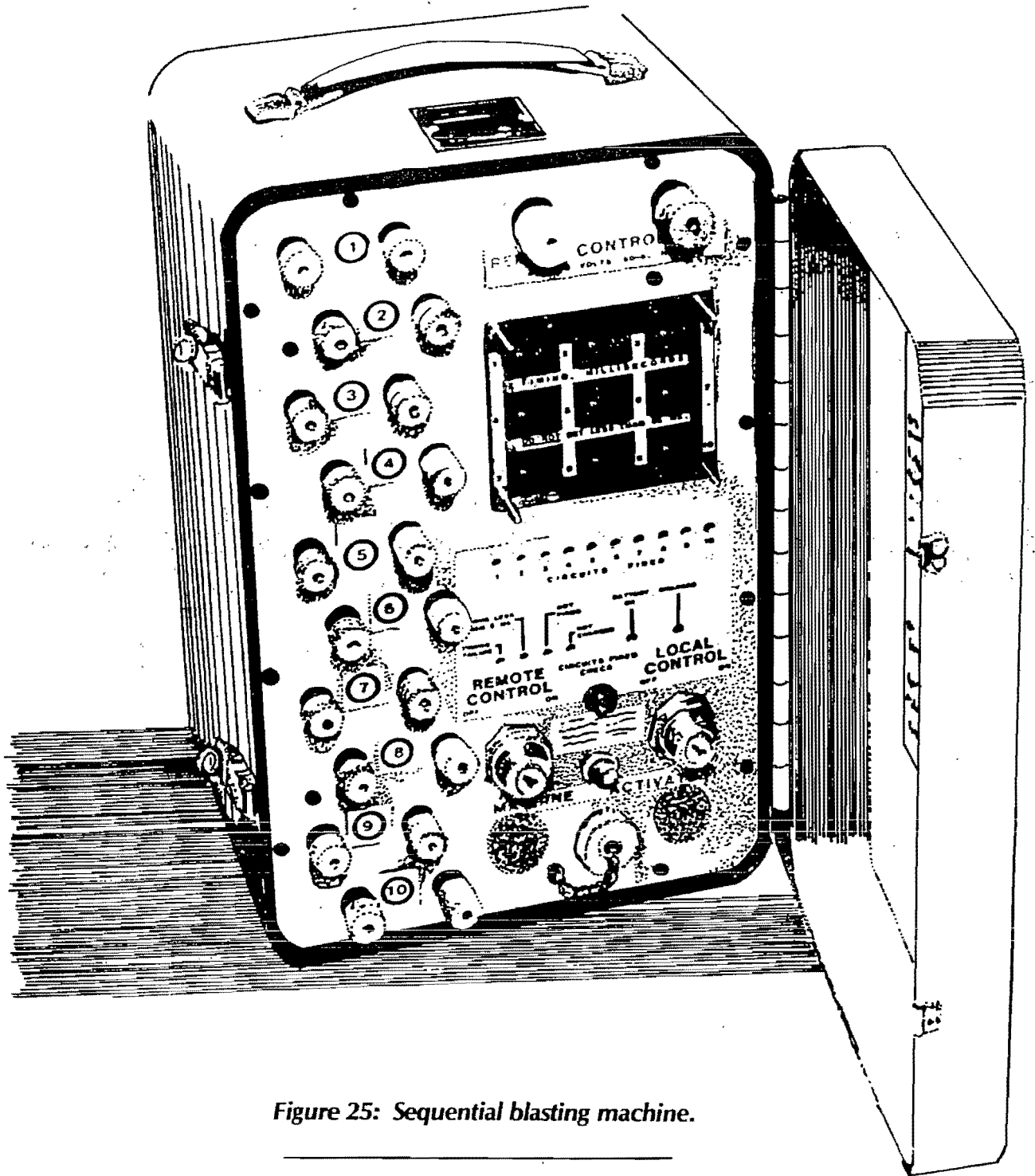
Figure 25: Sequential blasting machine.

Arcing in a cap results from excessive heat buildup, which is caused by too much current applied for too long a period of time. A current of 10 amp or more continuously applied for a second or more can cause arcing. To guard against arcing, the blasting may either use a blasting switch in conjunction with the power line, or add a No. 1 period millisecond-delay cap, placed in a quarter stick of explosive, to the circuit and tape the explosive to one of the connecting wires leading to the cap circuit. A better solution, if possible, is to use a high-output CD machine to fire the shot, using a parallel series circuit if necessary.

It is important to check the resistance of the blasting circuit to make sure that there are no broken wires or short circuits and that the resistance of the circuit is compatible with the capacity of the power source. There are two types of blasting circuit testers; a blasting galvanometer (actually an ohmmeter) shown in figure 26 and a blasting multimeter, shown in figure 27. The blasting galvanometer is used only to check the circuit resistance, whereas a blasting multimeter can be used to check resistance, ac and dc voltage, stray currents, and current leakage (2). Only a meter specifically designed for blasting should be used to check blasting circuits. The output of such meters is limited to 0.05 amp, which will not detonate an electric blasting cap, by the use of a silver chloride battery and/or internal current-limiting circuitry.

Figure 26: Blasting galvanometer.
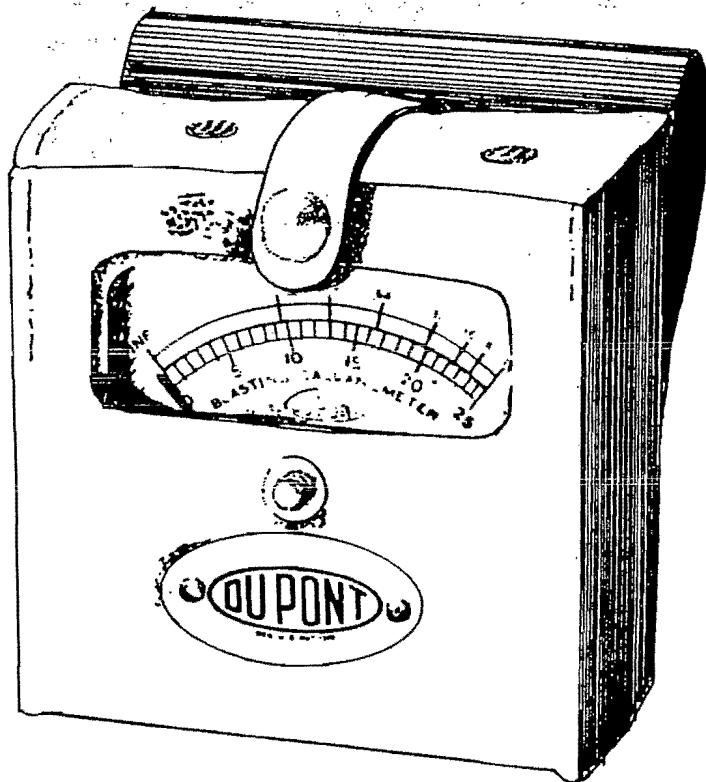
Other equipment, such as a "throw-away" go/no go device for testing circuits and a continuous ground current monitor, is available. The explosives supplier should be consulted to determine what specific electrical blasting accessory equipment is available and what equipment is needed for a given job.

It is generally recommended that each component of the circuit be checked as hookup progresses. After each component is tested, it should be shunted. Each cap should be checked after the hole has been loaded and before stemming. In this way, a new primer can be inserted if a broken leg wire is detected. A

total deflection of the circuit tester needle (no resistance) indicates a short circuit. Zero deflection of the needle (infinite resistance) indicates a broken wire. Either condition will prevent a blasting cap, and possibly the whole circuit, from firing.

Before testing the blasting circuit, its resistance should be calculated. After the caps have been connected into a circuit, the resistance of the circuit is checked and compared with the calculated value. A zero deflection at this time indicates a broken wire or a missed connection and an excessive deflection indicates a short circuit between two wires.

After the circuit resistance has been checked and compared, the connecting wire is then added and the circuit is checked again. If a parallel series circuit is used, the change in resistance should be checked as each series is added to the bus wire. In a straight parallel circuit, a break in the bus wire can sometimes be detected. However, a broken or a shorted cap wire cannot be detected in a straight parallel circuit because it will not affect the resistance significantly.

A final check of the circuit is made at the shotfirer's location after the firing line has been connected. If a problem is found in a completed circuit, the circuit should be broken up into separate parts and checked to isolate the problem. The firing line should be checked for a break or a short after each blast, or at the end of each shift, as a minimum.

To check for a break in the firing line, the two wires at one end of the line are shunted and the other end is checked with a blasting meter. A large deflection indicates that the firing line is not broken; a zero deflection indicates a broken wire. To test for a short, the wires of one end of the lead line are separated and the other end is checked with the meter. A zero deflection should result. If there is a deflection, the lead line has a short circuit and hazardous, costly, embarrassing misfires can be avoided through proper use of the blasting galvanometer or blasting multimeter.

Certain conditions, such as damaged insulation, damp ground, a conductive ore body, water in a borehole, bare wires touching the ground, or bulk slurry in the borehole, may cause current to leak from a charged circuit. Although this is not a common occurrence, you may want to check for it if you are experiencing unexplained misfires. To check properly for current leakage check with a consultant or an electric blasting handbook (2). Measures for combating current leakage include using fewer caps per circuit, using heavier gage lead lines and connecting wires, keeping bare wire connections from touching the ground, or using a nonelectric initiation system.

Extraneous Electricity

The principal hazard associated with electric blasting systems is lightning. Extraneous electricity in the form of stray currents, static electricity, and radio frequency energy, and from high-voltage power lines can also be a hazard. Electric blasting caps should not be used in the presence of stray currents of 0.05 amp or more. Stray currents usually come from heavy
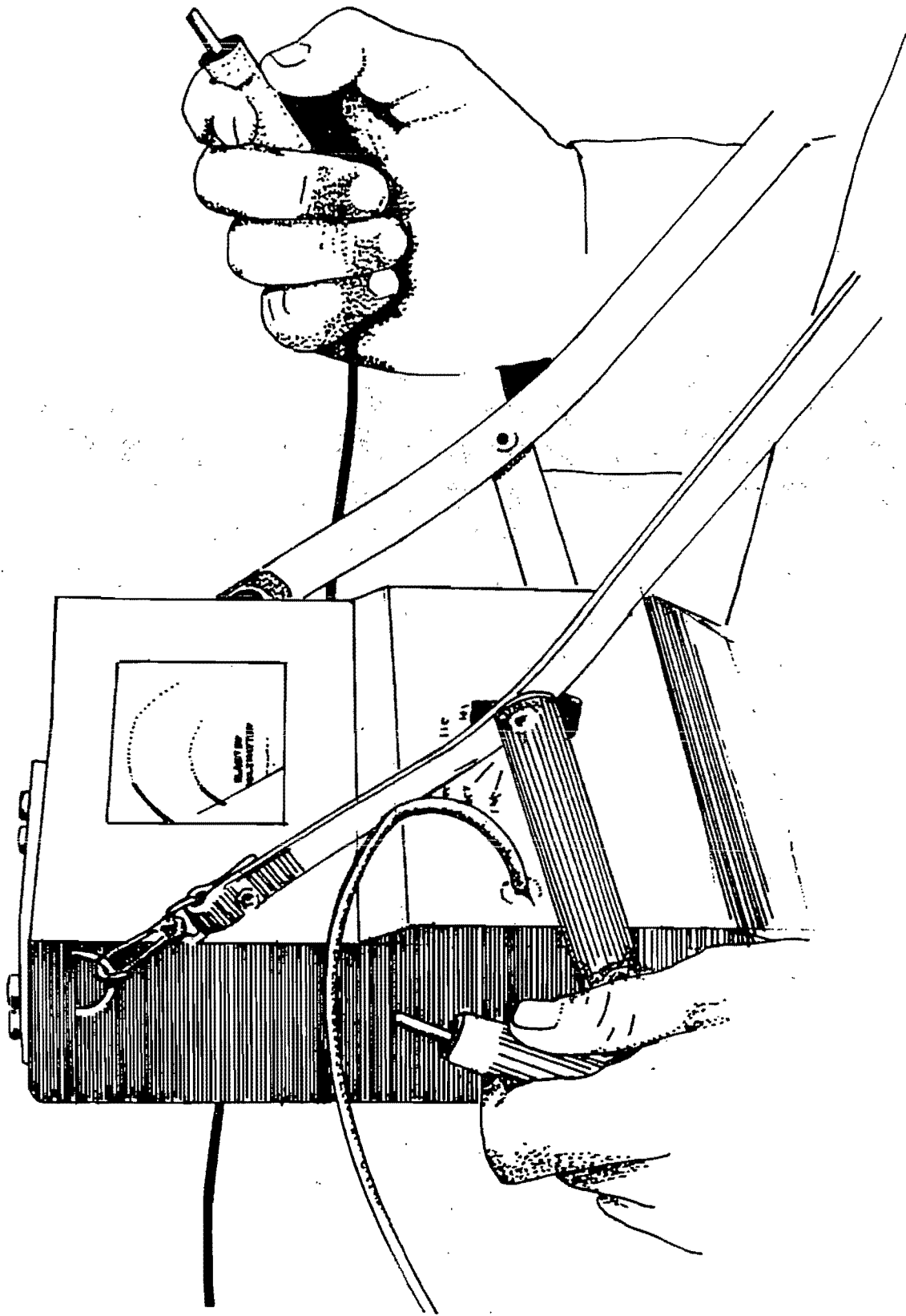
Figure 27: Blasting multimeter.

equipment or power systems in the area, and are often carried by metal conductors or high-voltage power lines. Atlas (2) outlines techniques for checking for stray currents. Instruments have recently been developed which monitor ground currents continuously and sound an alarm when an excess current is detected. The supplier should be consulted about the availability of these units.

Static electricity may be generated by pneumatic loading, particles carried by high winds (particularly in a dry atmosphere) and rubbing a person's clothes. Most electric blasting caps are static resistant. When pneumatically loading blasting agents with pressure pots or venturi loaders, a semiconductive loading hose must be used; a plastic borehole liner should not be used; and the loading vessel should be grounded.

Electrical storms are a hazard regardless of the type of initiation system being used. Even underground mines are susceptible to lightning hazards. Upon the approach of an electrical storm, loading operations must cease and all personnel must retreat to a safe location. The powder manufacturer should be consulted on the availability of commercial storm warning devices. Some operators use static on an AM radio as a crude detector of approaching storms. Weather reports are also helpful.

Broadcasting stations, mobile radio transmitters, and radar installations present the hazard of radio frequency energy. The IMI (11) has prepared charts giving transmission specifications and potentially hazardous distances.

High-voltage power lines present the hazards of capacitive and inductive coupling, stray current, and conduction of lightning. Atlas (2) details precautions to be taken when blasting near high-voltage power lines. A specific hazard with power lines is the danger of throwing part of the blasting wire onto the power line. This shorts the power line to the ground and has been responsible for several deaths. Care should be exercised in laying out the circuit so that the wires cannot be thrown on a power line. Other alternatives are to weight the wires so they cannot be thrown or attach a charge that cuts the blasting wire.

Additional Considerations

Electric blasting is a safe, dependable system when used properly under the proper conditions. Advantages of the system are its reasonably accurate delays, ease of circuit testing, control of blast initiation time, and lack of airblast or disruptive effect on the explosive charge. In addition to extraneous electricity, the blaster should guard against kinks in the cap leg wires, which can cause broken wires, especially in deep holes. Different brands of caps may vary in electric properties, so only one brand per blast should be used. It is recommended that the blaster physically carry the key or handle to the power source so the shot cannot be inadvertently fired while checking out the shot.

A device called an exploding bridge wire is available for use where a single cap is used to initiate a nonelectric circuit. This device has the safety advantages of a lack of primary explosive in the cap and a high voltage required for firing. A special firing box is required for the system. The

high power required and high cost of the exploding bridge wire device make it unsuitable for use in multicap circuits.

DETONATING CORD INITIATION

Detonating cord initiation has been used for many years as an alternative to electric blasting where the operator prefers not to have an electric initiator in the blasthole. Detonating cord (figure 28) consists of a core of high explosive (usually PETN) contained in a waterproof plastic sheath, enclosed in a reinforcing covering of various combinations of textile, plastic, and water-proofing. Detonating cord is available with PETN cord loadings ranging from 1 to 400 gr/ft.
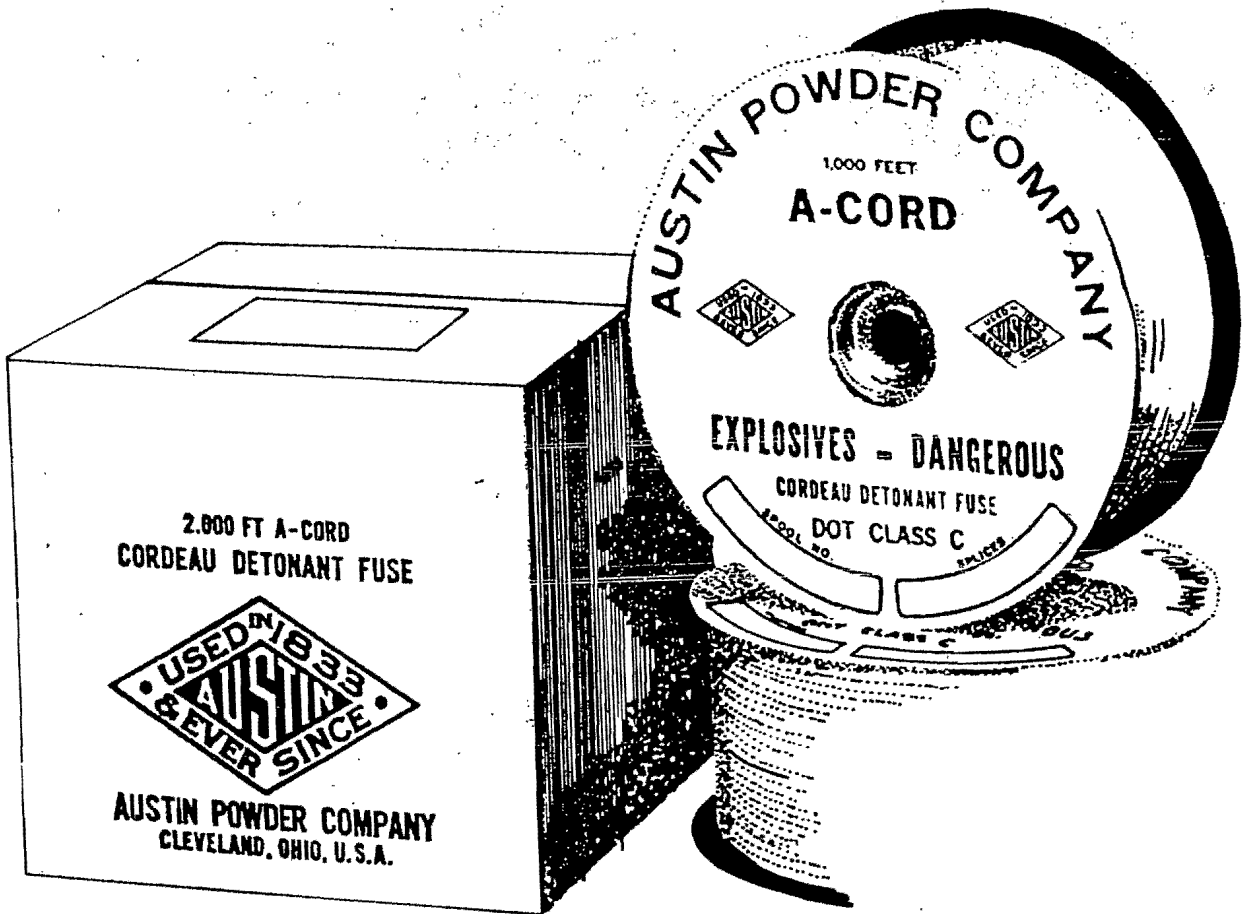
Figure 28: Detonating cord.

All cords can be detonated with a blasting cap and have a detonation velocity of approximately 21,000 fps. Detonating cord is adaptable to most surface blasting situations. When used in a wet environment the ends of the cord should be protected from water. PETN will slowly absorb water and, as a result, will become insensitive to initiation by a blasting cap. Even when

wet, however, detonating cord will propagate if initiated on a dry end.
Understanding the function of a detonating cord initiation system requires a
knowledge of the products available. The Ensign Bickford Company has pub-
lished a manual (8) that describes detonating cord products in detail. Tech-
nical data sheets are available from Austin Powder Company and Apache Powder
Company.

Detonating Cord Products

The most common strengths of detonating cord are from 25 to 60 grains/
foot. These strengths are used for trunklines, which connect the individual
blastholes into pattern, and for downlines, which transmit the energy from the
trunkline to the primer cartridge. The lower strength cords are cheaper, but
some have less tensile strength and may be somewhat less dependable under
harsh field conditions. Some cast primers are not dependably initiated by 25-
grain cord or lighter cord. However, under normal conditions, the lighter
cord loads offer economy and their greater flexibility makes field procedures,
such as primer preparation and knot tying, easier.

Detonating cord strengths of 100 to 200 grains/foot are occasionally used
where continuous column initiation of a blasting agent is desired. Cords with
200 to 400 gr of PETN per foot are occasionally used as a substitute for ex-
plosive cartridges in very sensitive or small, controlled blasting jobs. Con-
trolled blasting is described in the "Blast Design" chapter.

Detonating cord strengths lower than 25 gr/ft are sometimes used. Fif-
teen- to twenty-grain products may be used for small-diameter holes, for sec-
ondary blasting, and in the Nonel system described later. A 7.5-gr cord is
also used in the Nonel system. A 4-gr/ft product is used as part of an
assembly called a Primaline Primadet. A Primaline Primadet consists of a
length of 4-gr cord crimped to a standard instantaneous or delay blasting cap.
The cap is inserted into the primer and the 4-gr cord serves as a downline.
Various cord lengths are available to suit specific borehole depths. These
Primadets are primarily used in underground mines, such as salt, where Nonel
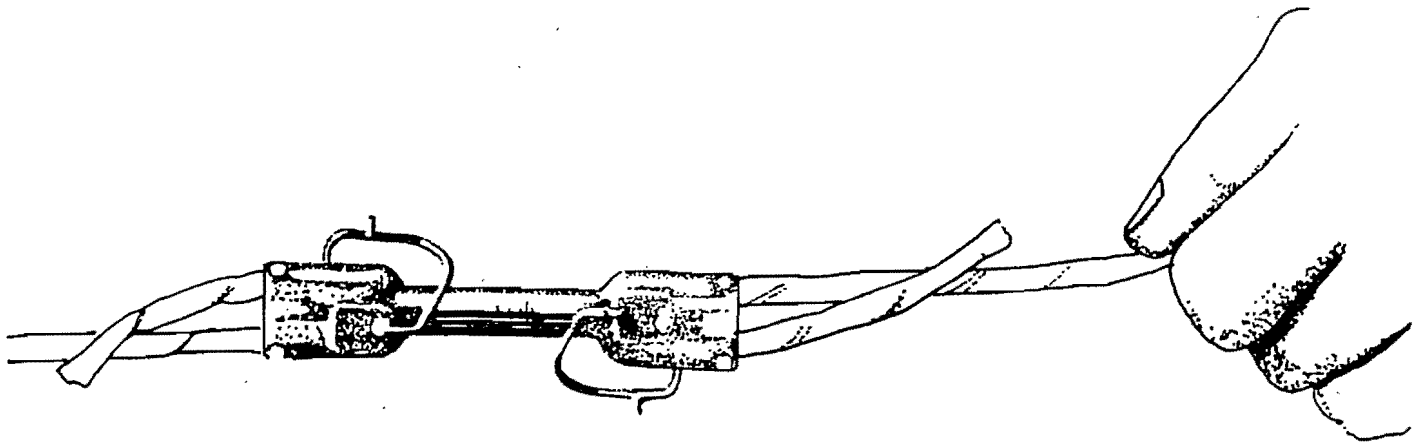
Figure 29: Clip-on surface detonating cord delay connector.

tubes would be a product contaminant. DuPont's new Detaline System uses a 2.4-gr cord.

Millisecond-delay surface connectors are used for delaying detonating cord blasts. To place a delay between two holes, the trunkline between the holes is cut and the ends are joined with a delay connector. One type of delay connector is a plastic assembly containing a delay element (figure 29). At each end of the element is an opening into which a loop of the severed trunkline can be inserted. A tapered pin is used to lock the trunkline cord into place. A Nonel delay connector has also been developed for detonating cord blasting (figure 30). This connector consists of two plastic blocks, each containing a delay initiator, connected by a short length of Nonel tubing. Each end of the severed trunkline is wrapped around the notch in one of the plastic blocks. Both types of delay connector are bidirectional.

Field Application

After the primer has been lowered to its proper location in the blast-hole, the detonating cord is cut from the spool. About 2 or 3 feet of cord should extend from the hole to allow for charge settlement and tying into the trunkline. When the entire shot has been loaded and stemmed, the trunkline is laid out along the path of desired initiation progression. Trunkline-to-trunkline connections are usually made with a square knot. A tight knot, usually a clove hitch, a half hitch, or a double-wrap half hitch, is used to connect the downline to the trunkline (figure 31). Any excess cord from the downline should be cut off and disposed of. If Primadets or other in-hole delay assemblies are used, a plastic connector often serves as the connection to the trunkline. The cord lines should be slack, but not excessively so. If too much slack is present, the cord may cross itself and possibly cause a cutoff (figure 32). Also, if the lines are too tight and form an acute angle, the downline may be cut off without detonating.
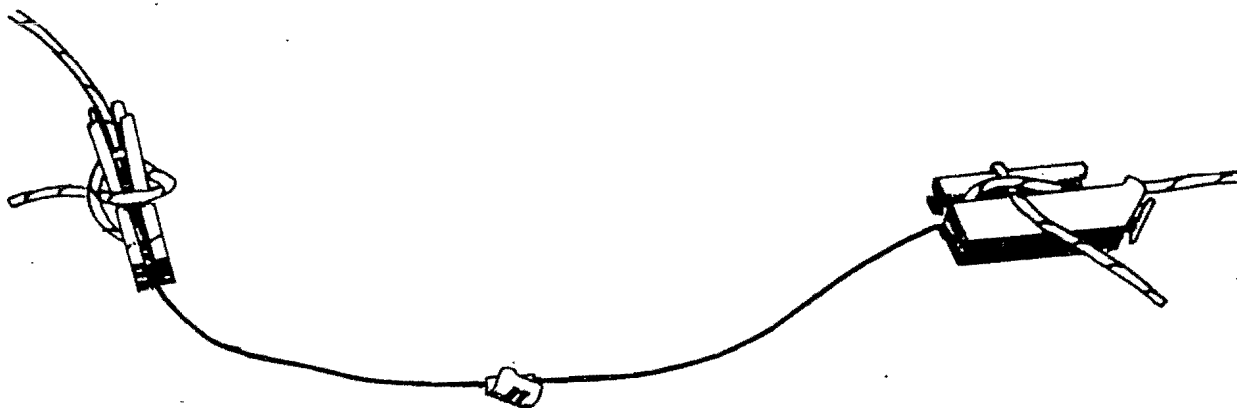
Figure 30: Nonel surface detonating cord delay connector.

Downlines of detonating cord can adversely affect the column charge of explosive in the blasthole. With cap-sensitive explosives, continuous, axial initiation will occur with any cord containing 18 or more grains of PETN per foot of cord. Lower strength cords may also cause axial initiation. Four-grain cord will not initiate most cap-sensitive explosives. With blasting

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agents, the effect of detonating cord is less predictable. The blasting agent may be desensitized or it may be marginally initiated. Hagan (10) has studied this problem. The effect depends on the cord strength, blasting agent sensitivity, blasthole diameter, and position of the cord within the blasthole. As a general rule, 50-gr cords are compatible with blasthole diameters of 8 inches or more. In charge diameters of 5 to 8 inches, 25-gr or lighter downlines should be used. In diameters below 5 inches, low-energy (4- to 10-gr) downlines or alternative, nondisruptive initiation systems are recommended. The manufacturer should be consulted for recommendations on the use of detonating cord with various explosive products. A low-energy initiation system called Detacord, developed by DuPont, is described later in this chapter.

Delay Systems

Surface delay connectors offer an unlimited number of delays. For example, a row of 100 holes could be delayed individually by placing a delay between each hole and initiating the row from one end. Typical delay intervals for surface connectors are 5, 9, 17, 25, 35, 45, and 65 ms. Because these connectors are normally used for surface blasting, half-second delay ms periods are not available.

Cutoffs may be a problem with surface delay connectors. When the powder column in one hole detonates, the connections between holes to be fired later may be broken by cratering or other movement of the rock mass. This may cause a subsequent hole to misfire. To correct this situation, MSHA requires that the pattern of trunklines and delay connectors be designed so that each blasthole can be reached by two paths from the point of initiation of the blast round. The patterns can become somewhat complex and should be laid out and carefully checked on paper before attempting to lay them out in the field. Where possible, the pattern should be designed so that the delay sequence in which the holes fire is the same no matter which path is taken from the point of blasting initiation. The "Blast Design" chapter gives suggestions for selecting the actual delay intervals between blastholes.

Figure 33 shows a typical blast laid out with delay connectors. Note that each hole can be reached by two paths from the point of initiation. A time of 1 ms is required to 21 feet of detonating cord to detonate. This time is not sufficient to significantly alter the delay interval between holes.

When detonating cord downlines are used, detonation of the cord in the blasthole proceeds from the top down. This presents two disadvantages. First, the detonation of the cord may have an undesirable effect on the column charge as it proceeds downward and the stemming may be loosened. Second, if the hole is cut off by burden movement caused by detonation of an earlier hole (figure 34) the powder in the lower portion of the hole will not detonate. The use of a Primaline Primadet delay unit in the hole will correct both of these problems.

The Primaline Primadet is a delay cap attached to a length of 4-gr/ft detonating cord. It is available in both millisecond and long-delay periods. The Primaline Primadet is connected to the trunkline with a plastic connector or a double-wrap half hitch. If the delay pattern of the blast is such that the number of available Primadet delay periods is adequate, an undelayed

Trunkline

Downline

Half hitch

Trunkline

Downline

Double-wrap half hitch

Trunkline

Downline

**Stiff cord downline
Half hitch**

**Cord-to-cord connection
Square knot**

Downline to Trunkline Connections
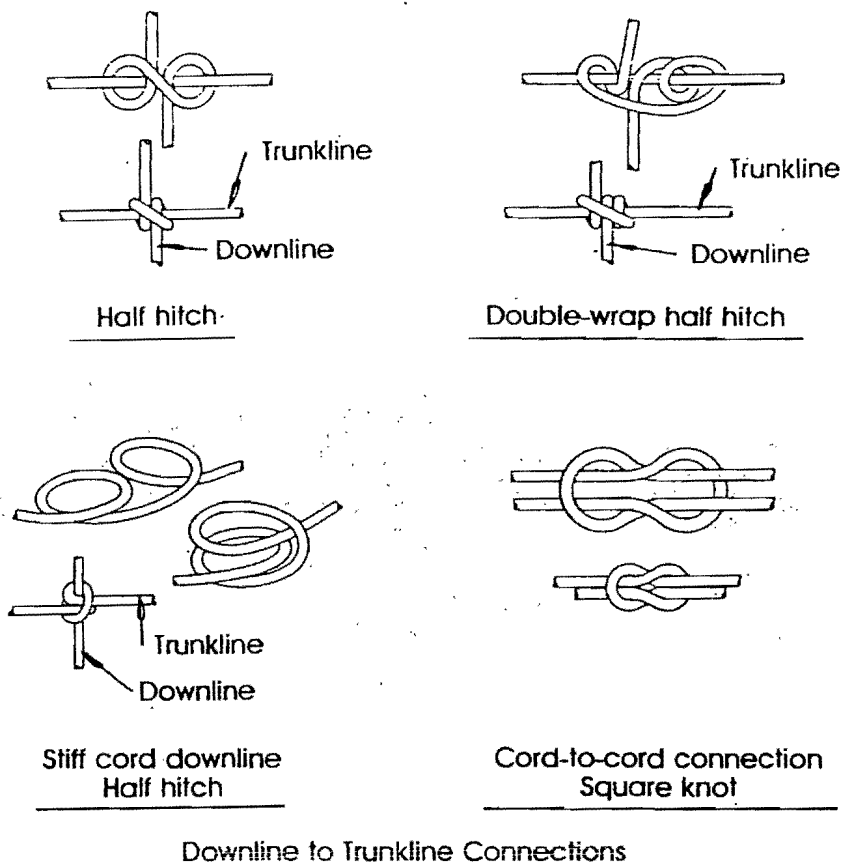
Figure 31: Recommended knots for detonating cord.

trunkline may be used. The delay period of the cap would then be the delay
period of the hole. As an example, to attain the delay pattern in figure 33,
cap delay periods one through nine would be placed in the appropriate holes
and trunklines would contain no delays. In this situation, the delay in every
cap would be actuated before the first hole detonates. This would reduce the
chance of a cutoff. The 4-gr Primaline Primadet is steadily being replaced by
other nonelectric systems, described later in this chapter.

Another alternative to obtain the delay pattern in figure 33 and avoid
the cutoff problem, would be to use the array of surface delays shown in the
figure and an in-hole delay of an identical period in each blasthole. For
example, if a 75-millisecond delay is used in each hole, and the trunkline
delays are each 9 millisecond, the delays in all of the holes except the two
rear corner holes will be actuated before the first hole in the pattern fires,
thus alleviating the cutoff problem. More complex patterns involving both
surface and in-hole delays can be designed where desirable. An alternative
method of obtaining in-hole delays with detonating cord is to use delay cast
primers (figure 12). These are cast primers with built-in, nonelectric,
millisecond delays. They can be strung on detonating cord downlines of 25 gr

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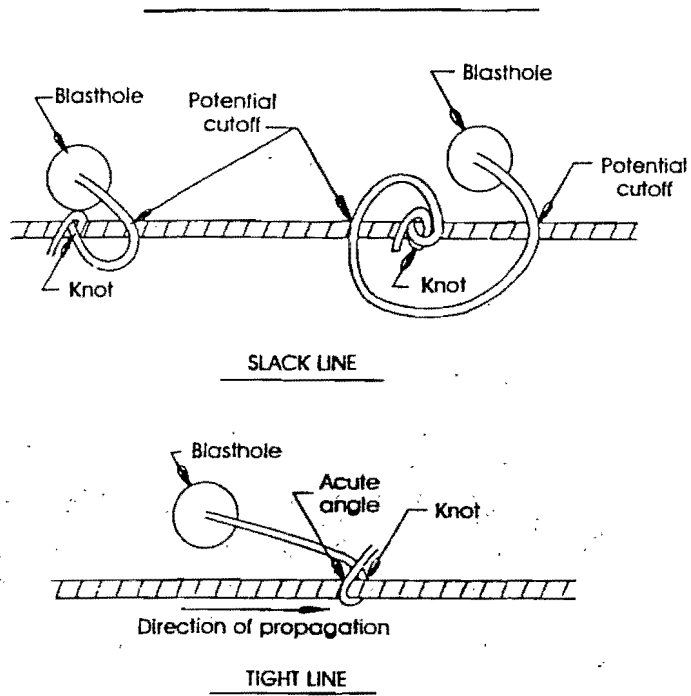
Figure 32: Potential cutoffs from slack and tight detonating cord lines.
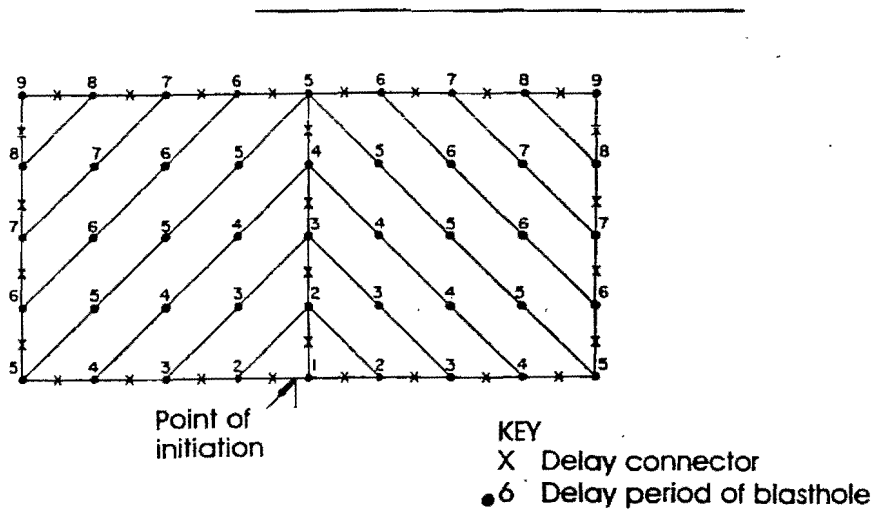
Figure 33: Typical blast pattern with surface delay connectors.
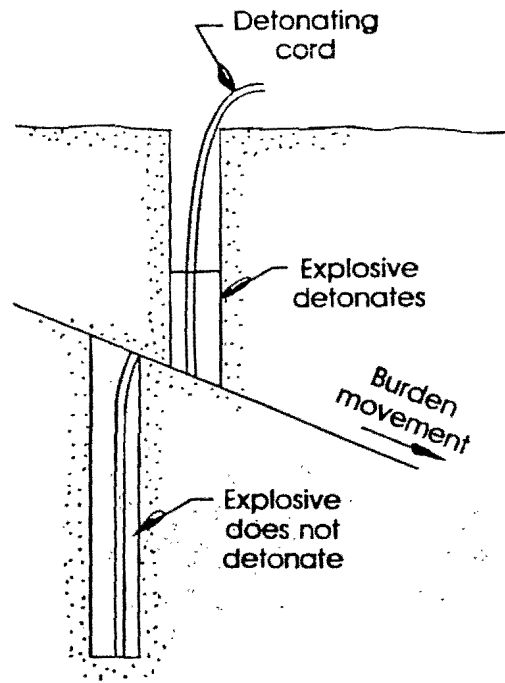
Figure 34: Misfire caused by cutoff from burden movement.

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or more and are particularly useful in obtaining multiple delayed deck charges with a single downline. It bears repeating that delay patterns involving both surface and in-hole delays can be somewhat complex and should be carefully laid out on paper before attempting to install them in the field.

General Considerations

Two of the primary advantages of detonating cord initiation systems are their ruggedness and their insensitivity. They function well under severe conditions such as in hard, abrasive rock, in wet holes, and in deep, large-diameter holes. They are not susceptible to electrical hazards, although lightning is always a hazard while loading any blast. Detonating cord is quite safe from accidental initiation until the initiating cap or delay connectors are attached. Available delay systems are extremely flexible and reasonably accurate.

There are several disadvantages that may be significant in certain situations. Systems employing only surface connectors for delays present the potential for cutoffs. Surface connectors also present the hazard of accidental initiation by impact. Detonating cord trunklines create a considerable amount of irritating, high-frequency airblast (noise). In populated areas, the cord should be covered with 15 to 20 inches of fine material or alternative, noiseless systems should be used. Detonating cord downlines presents the problem of charge or stemming disruption. As discussed previously, this depends on the borehole diameter, the type of explosive, and core load of

explosive in the cord. The means of checking the system is visual examination.

Vehicles should never pass over a loaded hole because the detonating cord lines may be damaged, resulting in a misfire or premature ignition. A premature ignition could also result from driving over a surface delay connector.

Detaline System

DuPont's Detaline System is a recently developed initiation system which is based on low-energy detonating cord. It functions similarly to conventional detonating cord systems except that the trunkline is low in noise, downlines will not disrupt the column of explosive, it will not initiate blasthole products, except dynamites, and all connections are made with connectors, rather than knots. The four components of the system are Detaline Cord, Detaline Starters, Detaline MS Surface Delays, and Detaline MS In-Hole Delays.

The Detaline Cord (Detacord) is a 2.4-gr/ft detonating cord with an appearance similar to standard detonating cord. The cord is cut to lengths required for the blast pattern. This low-energy cord, while low in noise, has sufficient energy to disintegrate the cord upon detonation, which is advantageous where contamination of the blasted product must be avoided. Detacord will not propagate through a knot, which is why connectors are required. To splice a line or to make a nondelayed connection, a Detaline Starter is required. The body of the starter is shaped much like a clip-on detonating cord millisecond delay connector, except that the starter is shaped like an arrow to show the direction of detonation. To make a splice, the starter is connected to the two ends of the Detacord using the attached sawtooth pin, making sure that the arrow points in the direction of detonation. To make a connection, the donor trunkline is hooked into the tail of the starter and the acceptor trunkline or downline is hooked into the pointed end of the connector.

The Detaline System has provisions for both surface and in-hole delays. The surface delays, which come in periods of 9, 17, 30, 42, 60, and 100 millisecond, are shaped like the starter but are colored according to the delay. The surface delays are also unidirectional, with the arrow showing the direction of detonation. The surface delays can be hooked into a trunkline in which case their function is similar to that of a standard millisecond delay connector. They can also be used as starters, connected between the trunkline and downline at the collar of the blasthole. In this situation, the delay affects only the downline and not the trunkline.

A Detaline MS In-Hole Delay resembles a standard blasting cap except for a special top closure for insertion of the Detacord. It functions similarly to a surface delay. Nineteen delay periods, ranging from 25 to 1,000 ms, are available. The delay is connected to the downline and is inserted into the primer.

Hookup of the Detaline System is similar to conventional detonating cord except that connectors are used rather than knots; right-angle connections are not necessary. When it is time to hook up, the Detaline trunkline is reeled out over the length of each row. Each downline is connected to the arrow end of a starter or a surface delay. The tail of each starter or surface delay is then connected into the trunkline. The open sides of the pattern are then

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connected in a manner similar to conventional detonating cord systems using Detacord and appropriate starters and surface delays. It is essential that all Detacord-to-Detacord connections be made with starters or surface delays rather than knots.

The Detaline System bears many similarities to conventional detonating cord systems. The system is checked out visually before firing. Combining surface and in-hole delays gives an almost unlimited number of delay combinations. It is convenient to build redundancy into the system. At firing time, the end of the trunkline tail extending from the shot is placed into the arrow end of a starter, and an electric or fuse blasting cap is inserted into the tail end of the starter and initiated.

The detonation energy of the 2.4-gr Detacord is adequate to disintegrate the trunkline. However, the resulting trunkline noise is quite low; typically about 13 decibels lower than 25-gr detonating cord in field trials. A downline of Detacord will detonate most dynamites but will not detonate most water gels. A major advantage of a Detacord downline is that it will not disrupt a column of blasting agent. Detacord can be used as a total system or in conjunction with some standard detonating cord components. As with most newer systems, evolutionary changes may occur in the coming years. It is important that the manufacturer be consulted for recommended procedures for using Detacord. The manufacturer will also be able to recommend which variation of the system best suits a particular field situation.

CAP-AND-FUSE INITIATION

Cap and fuse is the oldest explosive initiation system; however, its use has dwindled steadily. Its primary remaining use is in small underground mines, although a few large mines still use it. Surface applications are limited to secondary blasting and the initiation of detonating cord rounds with a single cap.
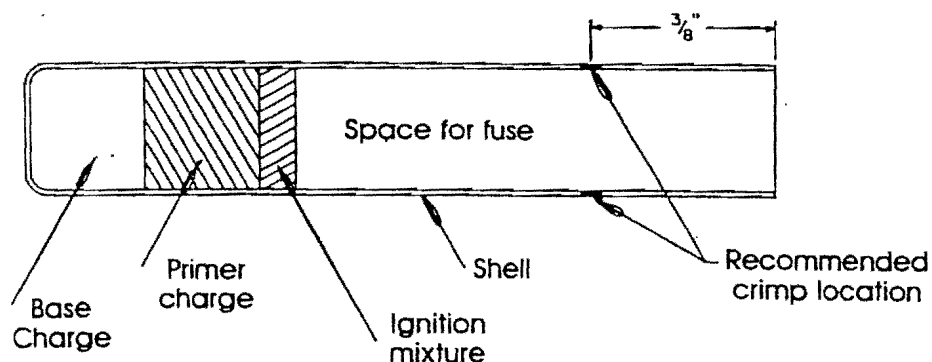
Figure 35: Blasting cap for use with safety fuse.

Components

The detonator used in a cap-and-fuse system is a small capsule that is open at one end (figure 35). The capsule contains a base charge and a heat-sensitive primer charge of explosive. The powder charge in the cap is initiated by a core of flammable powder in the safety fuse. Safety fuse has an appearance somewhat similar to detonating cord except that the surface of safety fuse is smoother and more waxy and the core load is black. The core load of detonating cord is white.

To assemble a cap and fuse, the fuse is carefully cut squarely and inserted into the cap until it abuts against the explosive charge in the cap. The fuse should never be twisted against the explosive charge in the cap. The cap is then crimped near the open end with an approved bench or hand crimper. The crimp should be no more than three-eighths of an inch from the open end of the cap.

Field Applications

Currently, all safety fuse burns at the nominal rate of 40 sec/ft. Both dampness and high altitude will cause the fuse to burn more slowly. Fuse should be test burned periodically so that the blaster can keep a record of its actual burning rate. "Fast fuse" has been blamed for blasting accidents but the fact is that this rarely if ever occurs. However, pressure on the fuse may increase its burning rate.

One of the most important considerations in the use of cap-and-fuse systems is the use of a positive, approved lighting mechanism. Matches, cigarette lighters, carbide lamps, or other open flames are not approved for lighting fuse. MSHA regulations specify hotwire lighters, lead spitters, and Ignitacord as approved ignitiation systems. The safest, most controllable lighting method is Ignitacord. In South Africa, where safety fuse is most often sold as an assembly with an Ignitacord connector attached, the safety record with cap and fuse is much better than it is in the United States.

The Ignitacord connector fits over the end of the fuse and is crimped in a manner similar to the cap. Figure 36 shows a typical cap, fuse, and Ignitacord assembly. The cap is attached to the fuse with a bench or hand crimper, and never with the teeth or pliers. When crimping the cap, care should be taken not to crimp the zone containing the powder. The Ignitacord connector is crimped to the other end of the fuse with a hand crimper. The Ignitacord is inserted into the notch near the end of the connector and the notch is closed using the thumb.

To guard against water deterioration, it is a good idea to cut off a short length of fuse immediately before making cap-and-fuse assemblies. In deciding the length of fuse to cut for each primer, the lighting procedure must be considered. Ignitacord is strongly recommended because of its safety record.

When Ignitacord is used, each fuse must have a burning time of at least 2 minutes. To make sure of this time, the fuse must be calibrated periodically by test burning. The Ignitacord is attached to the Ignitacord connectors in

the desired order of firing. If all the fuses are cut accurately to the same length, the desired order of firing will be achieved.

COMPONENTS
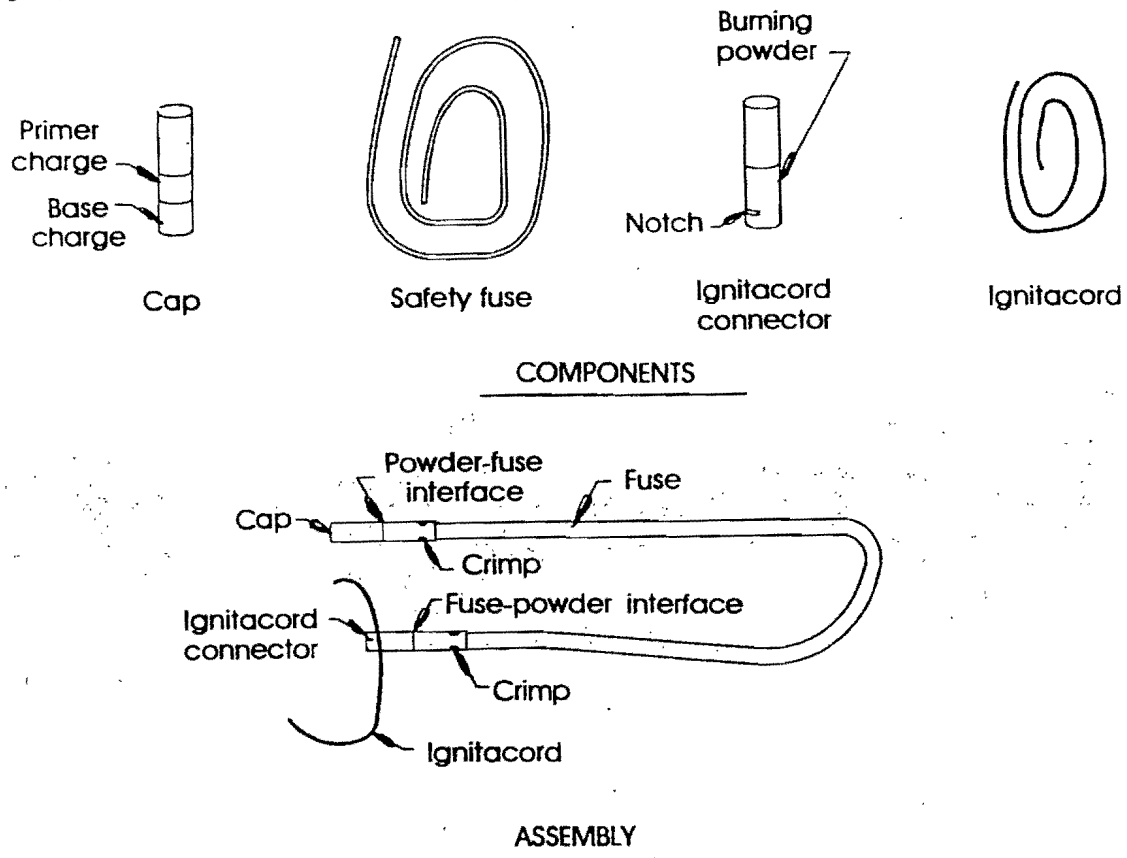
ASSEMBLY

Figure 36: Cap, fuse, and Ignitacord assembly.

With Ignitacord, only one lighting is required before the shotfirer returns to a safe location. Hotwire lighters and lead spitters require that each fuse be lit individually. The primary hazard of using safety fuse is the tendency of blasters to linger too long at the face, making sure that all the fuses are lit. To guard against this, MSHA regulations specify minimum burning times for fuses, depending on how many fuses one person lights. Keep in mind that two people are required to be at the face while lighting fuse rounds.

If a person lights only one fuse, the minimum burning time is 2 minutes; for 2 to 5 fuses the minimum is 2-2/3 minutes; for 6 to 10 fuses the minimum is 3-1/3 minutes; and for 11 to 15 fuses the minimum is 5 minutes. One person may not light more than 15 fuses in a round. Although individual fuse lengths may be varied for delay purposes, it is more dependable to cut all the fuses to the same length and use the sequence of lighting to determine the firing sequence.

To avoid misfires due to cutoff fuses, MSHA requires that the fuse in the last hole to fire is burning within the hole before the first hole fires. Kinks and sharp bends in the fuse should be avoided because they may cut off

the powder train and cause a misfire. Many people who use cap and fuse do so because they feel that it is simpler to use than other initiation systems. However, proper use of cap and fuse requires as much or more skill and care as other systems.

Delays

Cap and fuse is the only initiation system that offers neither flexibility nor accuracy in delays. Because of variations in lengths of fuse, burning rates, and time of lighting, the individual holes will fire at erratic intervals at best, and out of sequence at worst. It is impossible to take advantage of the fragmentation benefits of millisecond delays when using the cap-and-fuse system.

General Considerations

There is no situation in which cap and fuse can be recommended as the best system to use. The system has two overpowering flaws--inaccurate timing and a poor safety record. The former results in generally poor fragmentation, a higher incidence of cutoffs, and less efficient pull of the round. All of these factors nullify the small cost advantage derived from the slightly lower cost of the system components. The poor safety record attained by cap and fuse is an even more serious drawback. It is the only system that requires the blaster to activate the blast from a hazardous location and then retreat to safety. The use of Ignitacord rather than individual fuse lighting alleviates this problem. A Bureau of Mines' study (14) determined that the accident rate with cap and fuse is 17 times that of electric blasting, based on the number of units used. Too often the person lighting the fuse is still at the face when the round detonates. The time lag between lighting the fuse and the detonation of the round makes security more difficult than with other systems.

Cap and fuse does have the advantages of lack of airblast, no charge disruption, somewhat lower component costs, and protection from electrical hazards. If an operator decides to use the cap-and-fuse system, incorporation of Ignitacord for lighting multiple holes is strongly recommended because of its safety record.

OTHER NONELECTRIC INITIATION SYSTEMS

Beginning in about 1970, efforts were devoted toward developing new initiation systems that combined the advantages of electric and detonating cord systems. Basically, these systems consist of a cap similar to an electric blasting cap, with one or two small tubes extending from the cap in a manner similar to leg wires. Inside these tubes is an explosive material that propagates a mild detonation which activates the cap. Delay periods similar to those of electric blasting caps are available, except that there are no coal mine delays because these devices are not approved for use in underground coal mines. These initiation systems are not susceptible to extraneous electricity, create little or no airblast, do not disrupt the charge in the blasthole, and have delay accuracies similar to those of electric or detonating cord systems (5).

At the time this manual was written, two relatively new nonelectric initiation systems--Hercudet and Nonel--were on the market. Other nonelectric

systems are either under development or in the conception stage. Both Hercu-
det and Nonel were introduced to the U.S. market in the mid-1970s. Because of
their relative recency, minor changes are still being made in these systems.
The following discussions are intended to give only general information on the
systems. Blasters planning to use the systems should contact the manufactur-
ers, Hercules Inc. and Ensign Bickford Co. for specific recommendations on
their use. A third system, du Pont's Detaline System, is discussed in the
"Detonating Cord Initiation" section of this chapter.

Hercudet

The hookup of the Hercudet (also called gas detonation) blasting system
resembles a plumbing system. The cap is higher strength than most electric
blasting caps. Both millisecond and long delays are available. Instead of
leg wires, two hollow tubes protrude from the cap. The cap may be used in a
primer in the hole or at the collar of the hole for initiating detonating cord
downlines. In addition to the Hercudet cap, system accessories include duplex
trunkline tubing, single trunkline tubing; various types of tees, connectors,
ells, and manifolds for hooking up the system; circuit testers; a gas supply
unit containing nitrogen, oxygen, and fuel cylinder; and blasting machine.
The system functions by means of the low-energy detonation of an explosive gas
mixture introduced into the hollow tubes. This low-energy detonation does not
burst or otherwise affect the tubing.

For surface blasting, a cap with 4-inch leads is used (figure 37). For
surface initiation of detonating cord downlines, this cap is connected
directly to the trunkline tubing by means of the reducing connector that is
factory-attached to the cap. The reducing connector is needed because the
trunkline tubing is larger than the capline tubing. A special plastic con-
nector is used to attach the cap to the detonating cord downline.
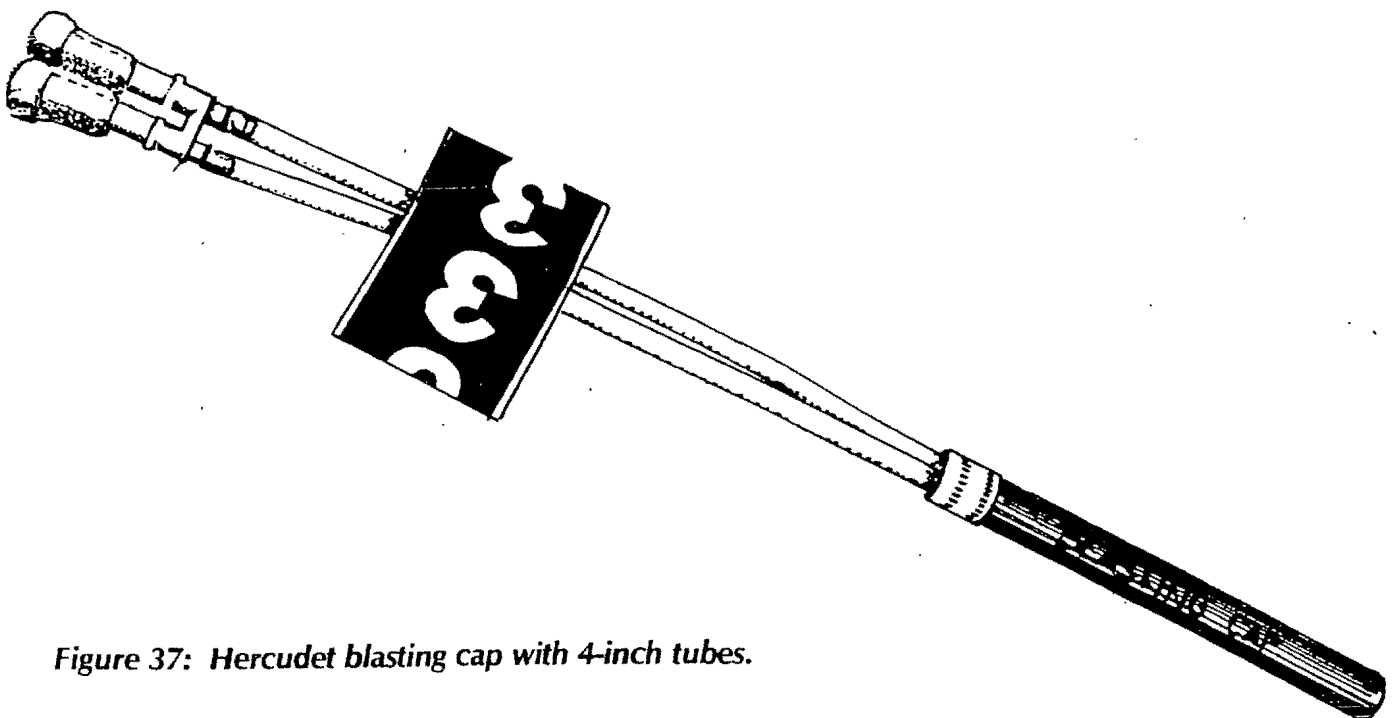
Figure 37: Hercudet blasting cap with 4-inch tubes.
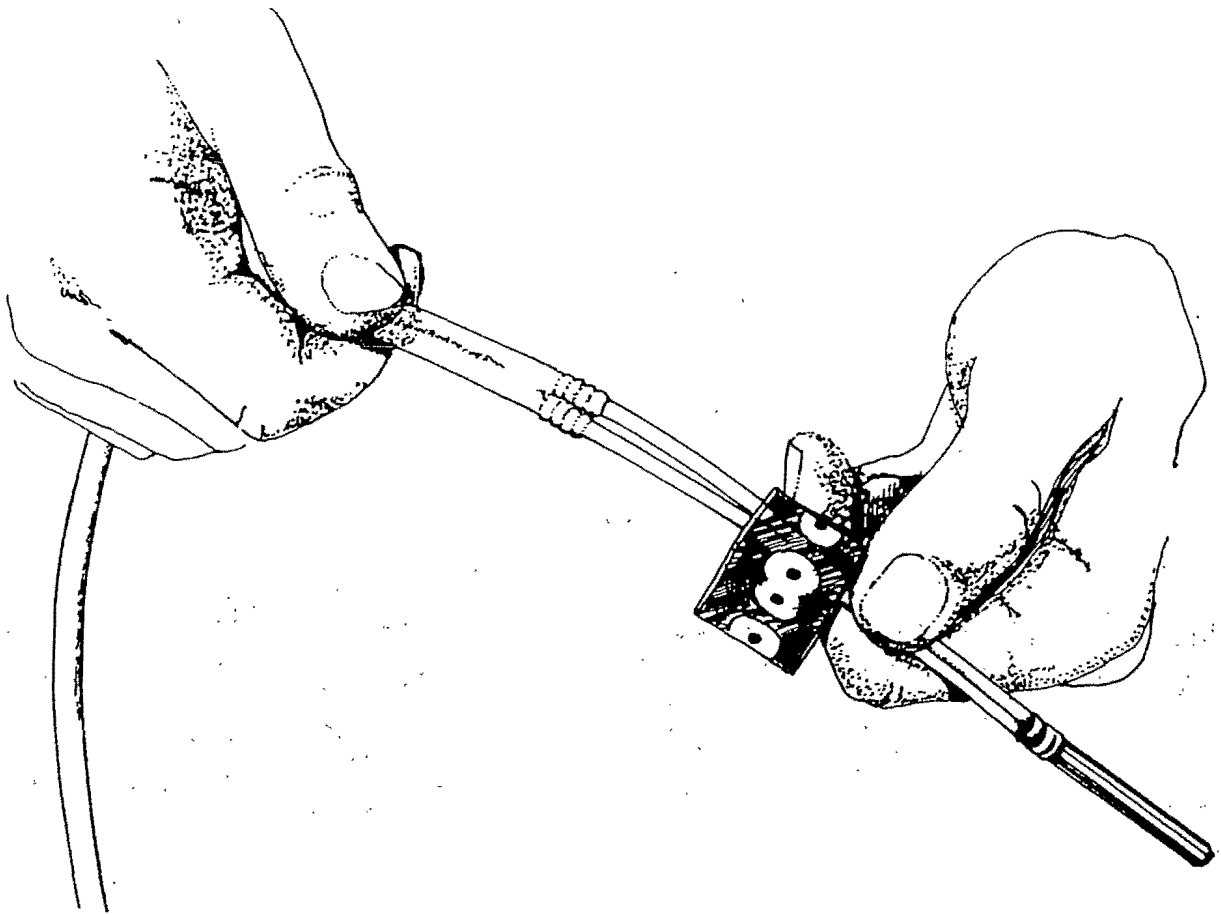
Figure 38: Extending Hercudet leads with duplex tubing.
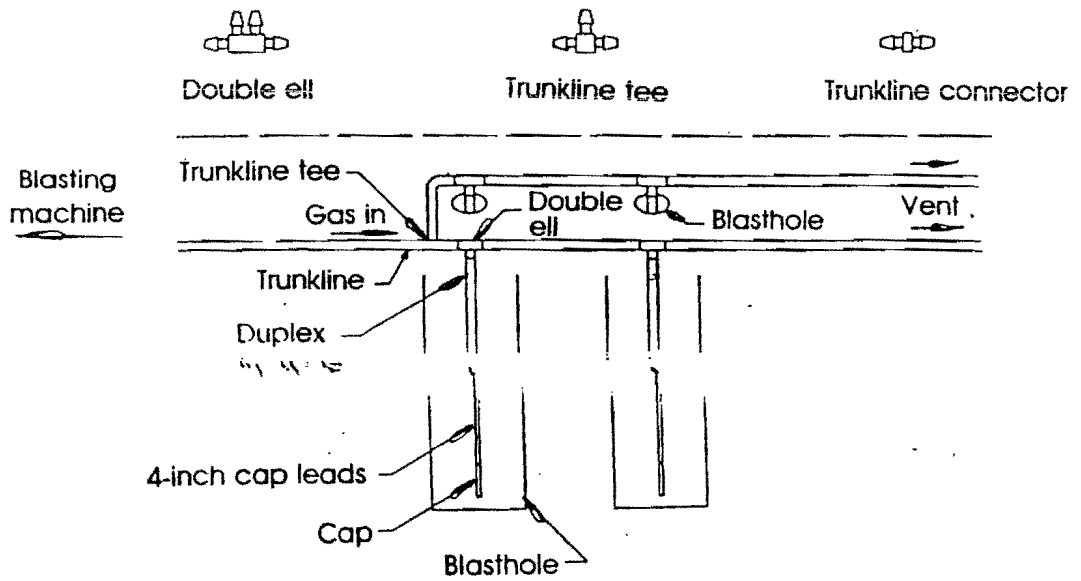
Figure 39: Hercudet connections for surface blasting.

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When in-hole initiation is desired, the 4-inch cap leads are extended by connecting them to an appropriate length of larger-diameter duplex trunkline tubing (figure 38). This trunkline tubing is cut squarely, leaving 2 to 3 feet of tubing extending from the borehole collar, and a plastic double ell fitting is inserted. Trunkline tubing is later connected hole to hole. Figure 39 shows typical Hercudet connections for surface blastings with in-hole delays.

Not all cast primers have tunnels large enough to accept the Hercudet duplex tubing. This should be checked before purchasing cast primers. When using cartridge primers with Hercudet, the tubing is taped to the primer, not half-hitched to it.

For underground blasting, millisecond and long-period delay caps are available with 16- to 24-foot lengths of tubing. The tubes are cut to the appropriate length by the blaster. The tubes are then connected in series or series-in-parallel, similar to electric cap circuits, using capline connectors and manifolds instead of the wire splices used in electric blasting. In all Hercudet blasting circuits, the tubing at the end of each series is vented to the atmosphere. The tubing network should be kept free of kinks.

When the circuit has been hooked up, a length of trunkline tubing is strung out to the firing position, similar to the firing line in an electric system. At this time, nitrogen from the gas supply unit is turned on and a pressure test module is used to check the integrity of the tube circuit (figure 40). The tester uses flow and/or pressure checks to locate blockages or leaks in the circuit. As with a galvanometer in electric blasting, each
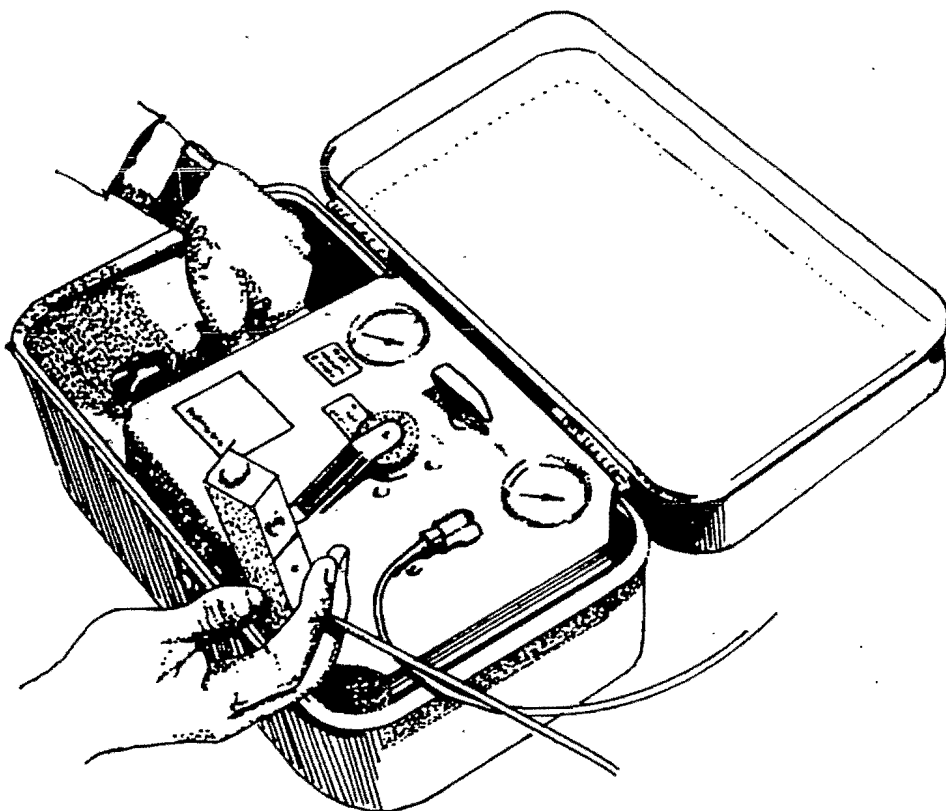
Figure 40: Hercudet pressure test module.
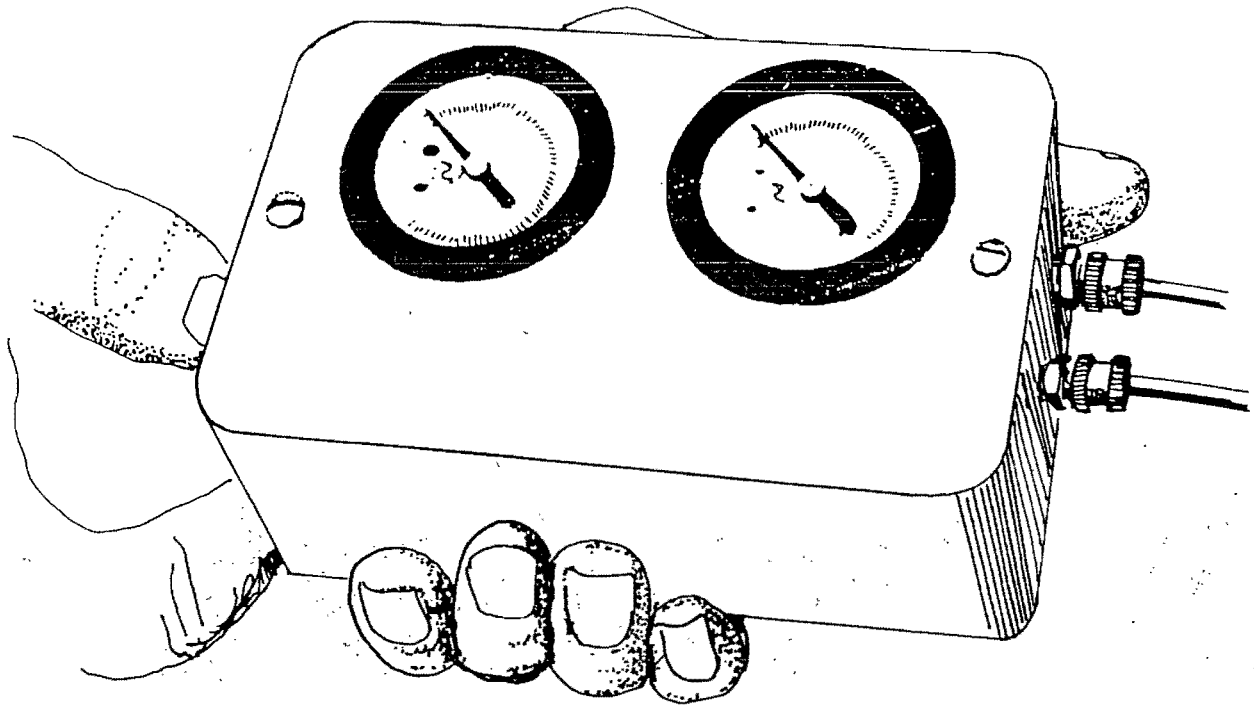
Figure 41: Hercudet tester for small hookups.

series should be checked individually, followed by a check of the entire system. The Hercudet tester is a smaller unit than the pressure test module and uses a hand air pump to test single boreholes or small hookups (figure 41). If a plug or a leak is detected when checking the completed circuit, the circuit is broken into segments and checked with the Hercudet tester or pressure test module.

Once the system has been checked and the blast is ready to fire, the blasting machine, connected to the bottle box (gas supply unit), is used to meter a fuel and oxidizer mixture into the firing circuit (figure 42). Until this detonatable gas mixture is put into the tubes, the connections between the caps are totally inert. The explosive gas must be fed into the system for an adequate period of time to assure that the system is entirely filled. Gas feeding continues until the blaster is ready to fire the shot. The time required to charge the circuit with gas depends on the size of the circuit.

When the system has been charged, the blasting machine control lever is moved to "arm" and the "fire" button is pushed, causing a spark to ignite the gas mixture. A low-energy gas detonation travels through the tubing circuit and through the air space inside the top of each individual cap at a speed of 8,000 fps and ignites the delay element in the cap.

The relatively slow (8,000 fps) detonation rate of the gas introduces an additional delay element into the system. For instance, assuming a gas detonation rate of 8 ft/ms, with caps at a depth of 30 feet in blastholes spaced 12 feet apart (a total travel path of 72 feet from cap to cap), a 9-ms delay
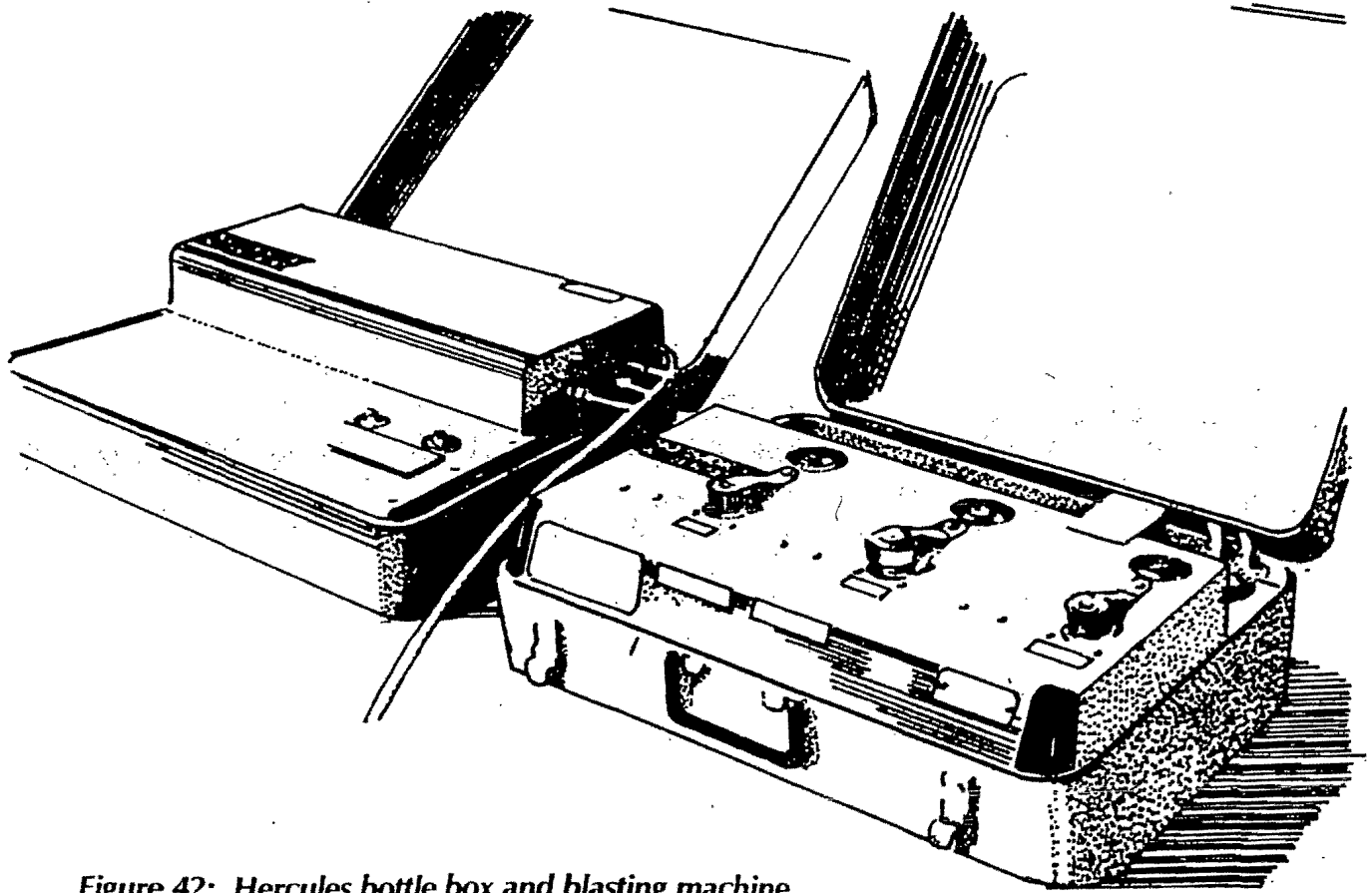
Figure 42: Hercules bottle box and blasting machine.

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between caps will be introduced by the tubing. It is essential that these tube delays be taken into account when calculating the actual firing times of the individual caps. Although the calculations are not complex, it is important that they are done carefully, before hooking up the blast, to avoid possible errors in the last minute rush to get the shot off. The delay time of the tubing can be used to advantage by coiling tubing in the trunkline at any location where an additional surface delay is desired.

The Hercudet system has the advantages of no airblast, no charge disruption, no electrical hazards, versatile delay capability, and system checkout capability. The inert nature of the system until the gas is introduced is a safety benefit. Specific crew training by a representative of the manufacturer is necessary because the system is somewhat different in principle than the older systems, such as detonating cord and electric blasting. Care must be taken not to get foreign material, such as dirt or water, inside the tubing or connector while hooking up the shot, and to avoid knots or kinks in the tubing.

The hookup of the Nonel (also called the shock tube) system is similar in some respects to the detonating cord system. The cap used in the system is higher strength than most electric blasting caps. Instead of leg wires, a single hollow tube protrudes from the cap (figure 43). The Nonel tube has a thin coating of reactive material on its inside surface, which detonates at a
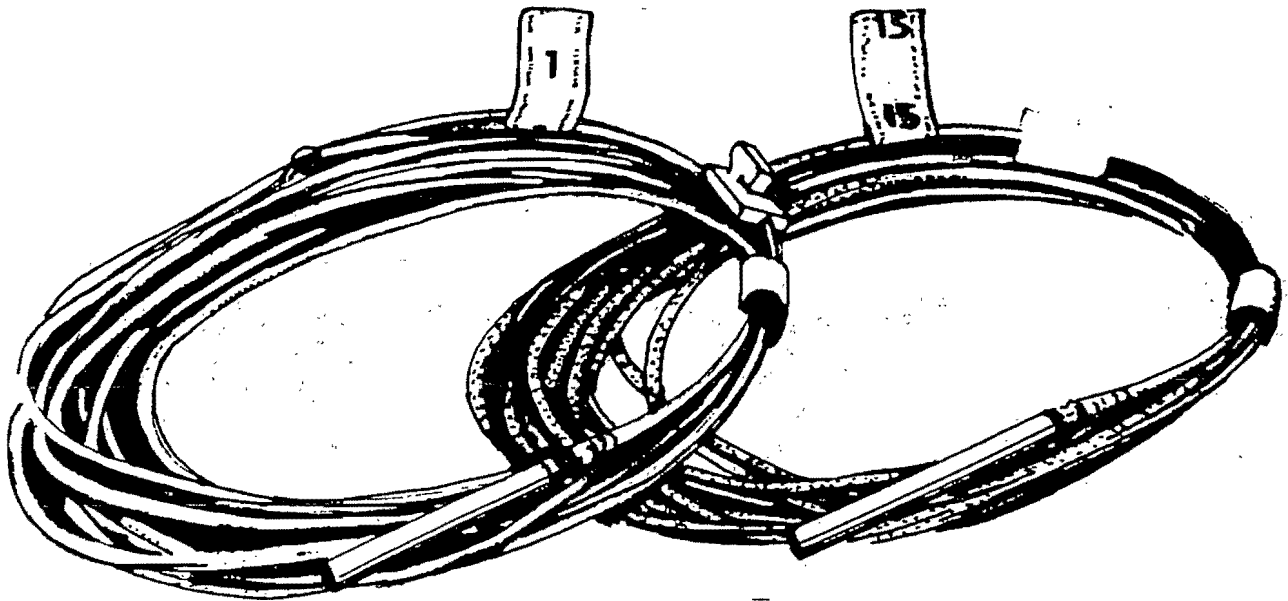
Figure 43: Nonel blasting cap.

speed of 6,000 fps. This is a very mild dust explosion that has insufficient energy to damage the tube. Several variations of the Nonel system can be used, depending on the blasting situation. In addition to the Nonel tube-cap assembly, system accessories include noiseless trunklines with built-in delays, noiseless lead-in lines, and millisecond delay connectors for detonating cord trunklines.

One Nonel system for surface blasting uses a Nonel Primadet in each blasthole with 25- to 60-gr/ft detonating cord as a trunkline. The Nonel cap used in this system is factory crimped to a 24-inch length of shock tube with a loop in the end (figure 44). The caps are available in a variety of millisecond delay periods. A 7.5-gr detonating cord downline is attached to the loop with a double-wrapped square knot. The 7.5-gr detonating cord extends out of the borehole. This downline will not disrupt a column charge of blasting agent but it may initiate dynamite and other cap-sensitive products. As a precaution, 7.5-gr to 7.5-gr connections should never be made, because propagation from one cord to the other is not dependable. Because the force of the shock tube detonation is not strong enough to disrupt the tube, it will not initiate high explosives. A 25- to 60-gr trunkline is used in this system with a double clove hitch used for downline-to-trunkline connections. The delay systems used with this method of initiation are the same as those discussed in the "Detonating Cord Initiation" section. They include in-hole cap delays and surface delay connectors.
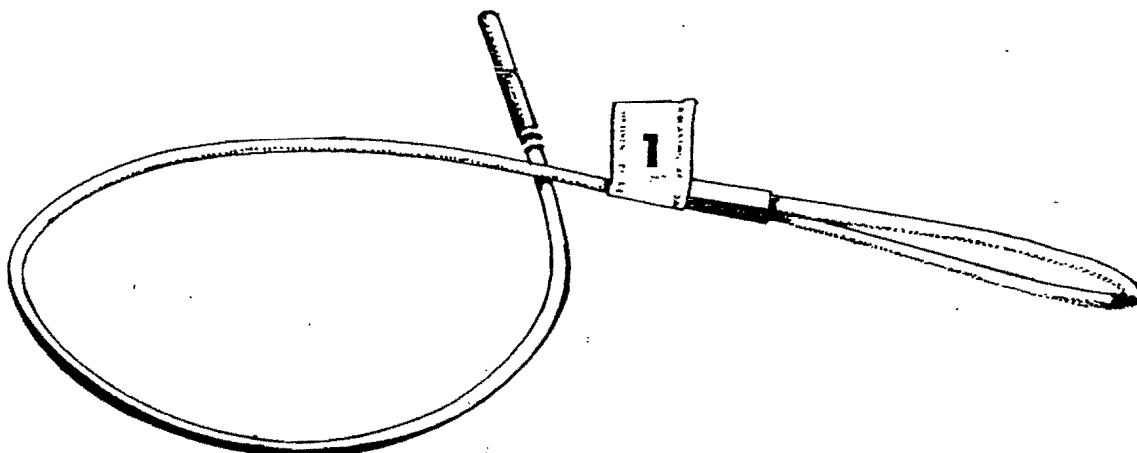
Figure 44: Nonel Primadet cap for surface blasting.

In some cases, this system creates an excessive amount of airblast and noise. To prevent this, the detonating cord trunkline can be replaced by an electric blasting cap circuit with a cap connected to each downline, or a noiseless Nonel trunkline can be used.

The noiseless Nonel trunkline is employed as follows. First, each hole is primed and loaded. The downline should be an 18-gr or larger detonating cord. A 7.5-gr downline can be used if a 25-gr pigtail is used at the top end, tied into the connector block. The noiseless trunkline delay unit consists of a length of shock tube, 20 to 60 feet in length, with a quick connecting sleeve on one end and a plastic block containing a millisecond delay blasting cap (delay assembly) on the other end, and a tag denoting the delay period (figure 45). The delay may be from 5 to 200 ms.

The sleeve is attached to the initial hole to be fired and the shock tube is extended to the next hole in sequence. The downline from this next hole is connected to the plastic block containing the delay cap, using about 6 inches of cord at the end of the downline. Another delay unit is selected and the sleeve is attached to the downline below the plastic block. The shock tube is extended to the next hole, where the delay assembly is connected to the downline. The process is repeated until all the holes are connected. Figure 46 shows a portion of a shot hooked up in this way.

The downlines and the plastic blocks containing the delay cap should be covered to reduce noise and flying shrapnel. When the blaster is ready to fire the shot, an initiating device is attached to the downline of the first hole. This device may be an electric blasting cap, a cap and fuse, or a Nonel noiseless lead-in (figure 47). A noiseless lead-in is a length of shock tube, up to 1,000 feet long, crimped to an instantaneous blasting cap. The shock tube is initiated by using an electric blasting cap, cap and fuse, or other initiating device recommended by the manufacturer.
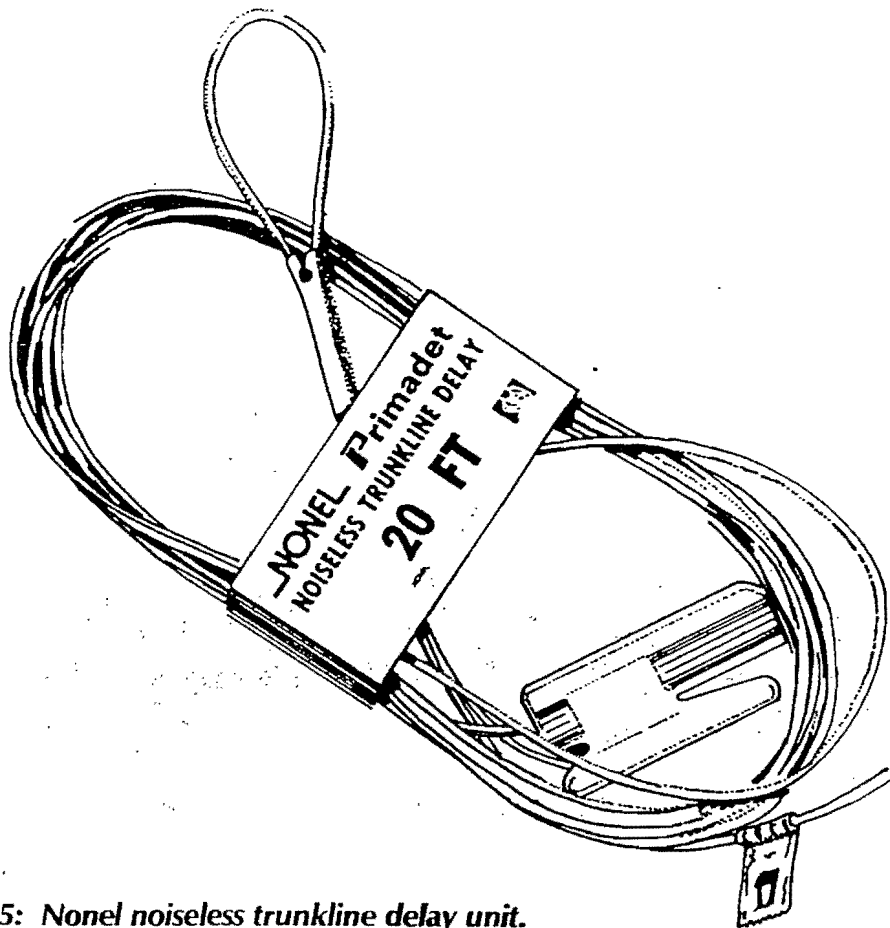
Figure 45: Nonel noiseless trunkline delay unit.

For underground blasting, millisecond and long-period delay caps are available with 10- to 20-foot lengths of shock tube attached. Common practice is to use a trunkline of 18- to 25-gr detonating cord. The Nonel tube from each blasthole is attached to the trunkline with a J-connector. A simpler method is to use the bunching system, where up to 30 tubes are tied together parallel, in a bunch, and detonating cord is clove-hitched around the bunch. The manufacturer should be consulted to demonstrate the bunching technique and to determine the number of wraps of detonating cord required for a given size bunch.

When pneumatic loading is used, a plastic cap holder can be utilized to center the cap in the hole and to reduce movement of the cap. It is important that the Nonel tube is in a straight line, fairly taut, and that crossovers or contact with the trunkline are avoided. This is true in all Nonel blasting but particularly in heading rounds, where the blast face is more crowded. Just before blasting, an electric cap or cap and fuse is connected to the trunkline.

The Nonel system has the advantages of no airblast (when a noiseless trunkline is used), no charge disruption (when Nonel tube or a 7.5-gr cord in conjunctionwith a Nonel Primadet is used as a downline), no electrical hazards, and a versatile delay capability. Keep in mind that electrical storms

3-inch sleeve connector

Shock tube

Plastic delay assembly

DELAY ASSEMBLY

Downline **Cap** **Noiseless lead-in**

Plastic delay assembly **Nonel shock tube**

TYPICAL DELAY PATTERN
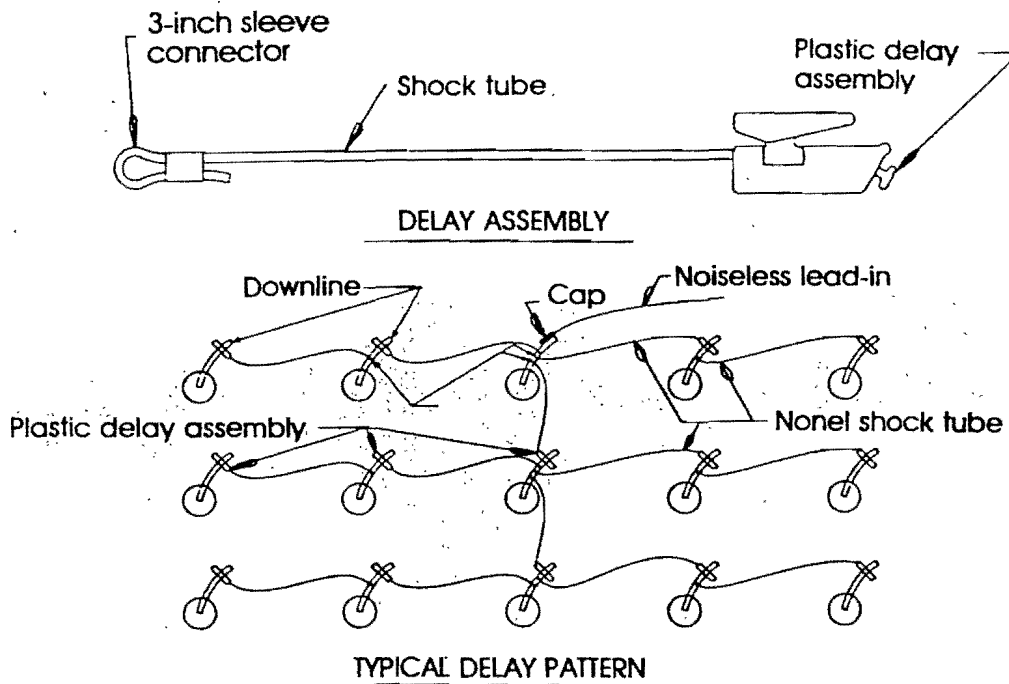
Figure 46: Noiseless trunkline using Nonel delay assemblies.

are a hazard with an initiation system. System checkout is done through visual inspection. Nonel shock tube assemblies should never be cut or trimmed; this may cause the system to malfunction. The shock tube will initiate nothing but the cap crimped onto it. Because of the variations available and new concepts involved, specific crew training by a manufacturer's representative is highly recommended before using the Nonel system.

PRIMING

Essentially, the term "primer" is used to describe a unit of cap-sensitive explosive that contains a detonator, while the term "booster" describes a unit of explosive that may or may not be cap sensitive and is used to inten sify an explosive reaction but which does not contain a detonator. Although a primer is generally thought of as containing a blasting cap, the primer cartridge may also be detonated by a downline of detonating cord.

The possible undesirable effect of the cord on blasting agents, described in the "Detonating Cord Initiation" section, must be considered. If the column charge is cap sensitive, detonating cord will cause initiation to proceed from the top down. The manufacturer should be consulted to determine the minimum strength detonating cord that will reliably initiate a specific type of

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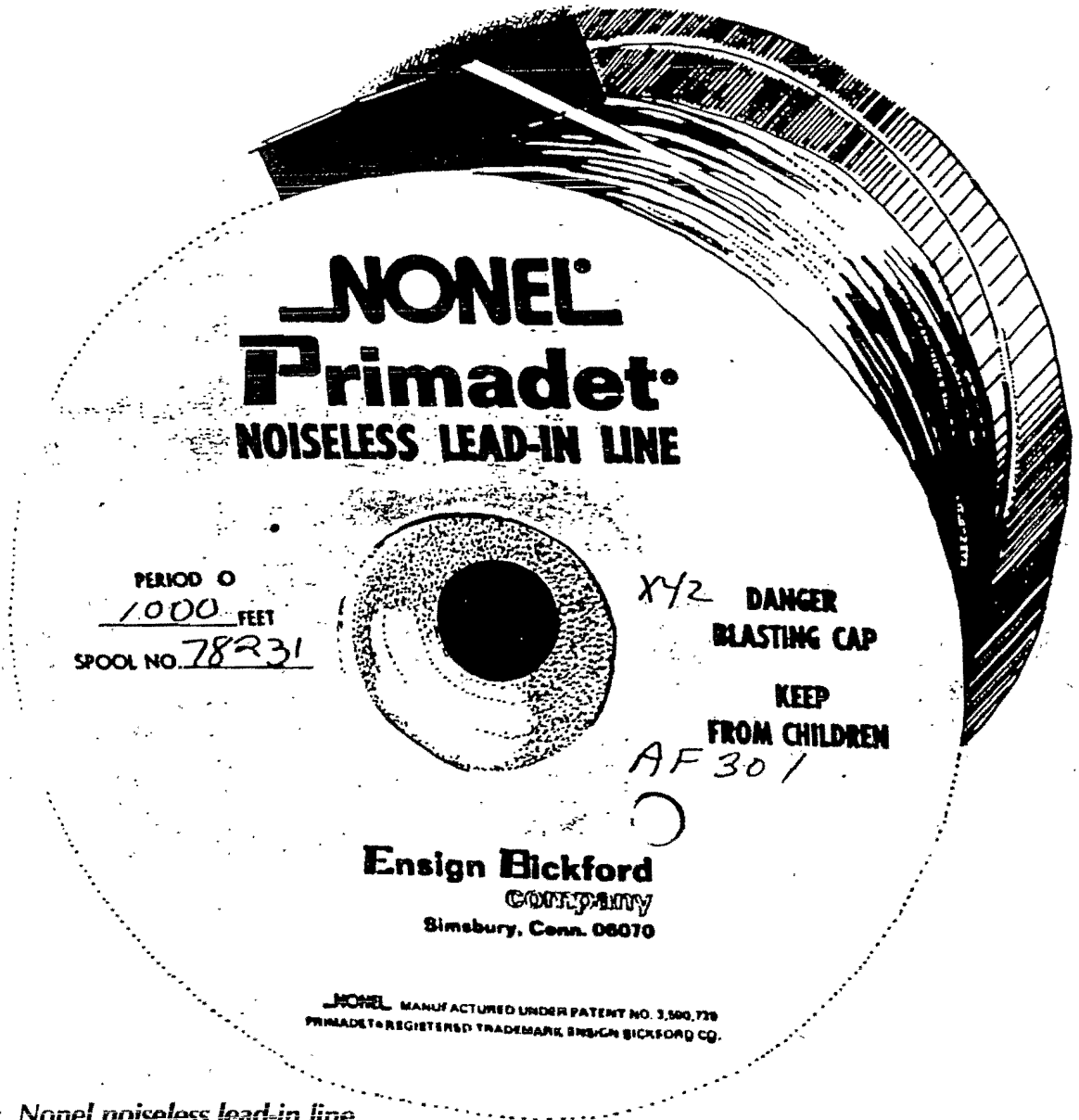
Figure 47: Nonel noiseless lead-in line.

primer. Most cast primers require a detonating cord strength greater than 25 gr/ft for reliable initiation.

Types of Explosive Used

The primer should have a higher detonation velocity than that of the column charge being primed. Some experts feel that priming efficiency continues to increase as the primer's detonation velocity increases. In blastholes of 3-inch diameter and less, cartridged dynamites and cap-sensitive cartridged slurries are commonly used as primers. For maximum efficiency, the diameter of the cartridge of explosive should be as near to the blasthole diameter as can be conveniently loaded. Gelatin dynamites are preferred over granular types because of their higher density, velocity, and water resistance. Some granular dynamites may be desensitized when subjected to prolonged exposure to water or to the fuel oil in AN-FO. Cast primers (figure 11) may be used if the borehole is large enough to accommodate them. Small units of explosive that fit directly over the shell of a blasting cap can be used for priming bulk blasting agents in small-diameter holes. In some

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situations, where boreholes are dependably dry, a high-strength cap alone has been used to prime a bulk-loaded AN-FO in a small-diameter hole. However, it is strongly recommended that a small booster fitting directly over the shell of the cap be used rather than a high-strength blasting cap alone. The cap manufacturer should be consulted for a recommendation if you are in doubt.

In larger diameter blastholes, cast primers are used predominantly, although some operators prefer to use cartridged high explosives. Ideally, the primer should fill the diameter of the blasthole as nearly as possible. However, primers are relatively expensive compared to the blasting agents used in larger boreholes, so economics are a factor in primer choice.

All blasting agents are subject to transient detonation velocities (4,6). That is, they may begin detonating at a relatively low velocity at the point

Figure 48: Highly aluminized AN-FO booster.

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of initiation with the velocity rapidly building up until the blasting agent reaches its stable velocity, called the steady state velocity. This buildup occurs within about three charge diameters. A low initial velocity probably causes some loss of energy at the primer location. Low initial velocities can result when the primer is too small or of inadequate strength, or when the blasting agent is poorly mixed or partially desensitized by water.

In large-diameter slurry columns, a 1-lb. cast primer or a cartridge of gelatin dynamite is often an adequate primer. In AN-FO columns where conditions are dependably dry, a 1-lb. primer is sometimes adequate. However, where dampness exists, or where low transient velocities are a particular concern, it is recommended that a 25- or 50-lb. charge of high-energy slurry or aluminized AN-FO be poured around the primer. This is called combination priming. High detonation pressure slurries (12, 13) and highly aluminized products (9) have been recommended as combination primers (figure 48). Bureau of Mines research (4) indicates that each type of product does a good job of raising the velocity in the transient zone. An added benefit of combination priming is the margin of safety in damp boreholes that may partially desensitize AN-FO.

Cast primers have been developed which incorporate an internal millisecond delay. The cast primers and the delay devices are supplied separately, with directions for assembly (figure 12). These delay primers are slipped onto a detonating cord downline and are especially useful in providing multiple delays in the blasthole on a single downline.

Primer Makeup

Proper care and technique in making primers is very important because this is the time in the blasting process at which the sensitive initiator and the powerful explosive cartridge are first combined. Because of the additional hazard involved, primers should be made up as close to the blast site as practical and immediately before loading.

In large tunnel projects, it is generally agreed that an outside primer makeup facility is best, assuming that transportation from the facility to the working face is safeguarded. Primers should be dismantled before removal from the blasting site. An adequate hole must be punched into the cartridge to insure the detonator can be fully imbedded. Care must be taken to assure that the detonator does not come out of the primer cartridge during loading. The primer cartridge should never be tamped or dropped down the borehole. One or more cartridges or a few feet of AN-FO should be placed above the primer cartridge before dropping or tamping begins.

In small-diameter holes, it is especially important that the end of the cap points in the direction of the main charge. It is also strongly recommended that in small-diameter holes the primer cartridge be the first cartridge placed into the blasthole. When priming small-diameter cartridges, the hole for the detonator is usually punched in the end of the cartridge. With electric caps, the wires are usually half hitched around the cartridge (figure 49). Two half hitches are commonly used. The tubes or fuse from nonelectric detonators are not half hitched. It is recommended that the tubes or fuse be taped to the cartridge to assure that the cap is not pulled out during loading.

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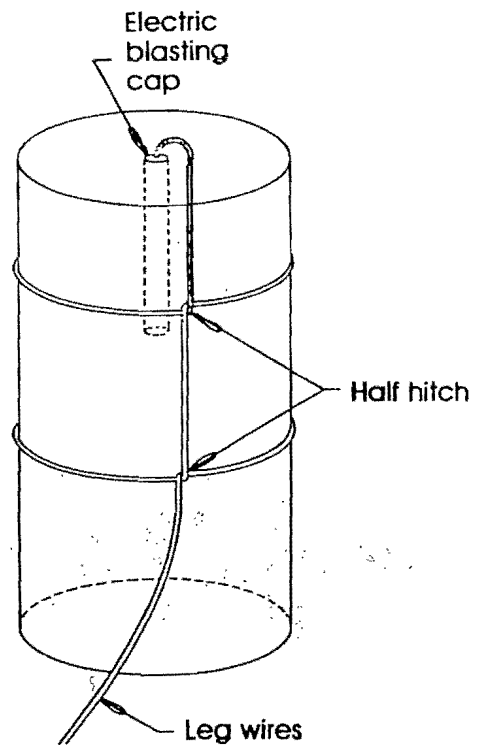
Figure 49: Cartridge primed with electric blasting cap.

Some safety fuses will not stand the sharp bend required for end priming. In this case, a diagonal hole is punched all the way through the cartridge and a second diagonal hole is punched partially through. The cap and fuse is strung through the first hole, placed into the second hole, and pulled secure. Here again, taping of the fuse to the cartridge will assure that the cap is not pulled out during loading.

.When attaching detonating cord directly to the small-diameter primer cartridge, the detonating cord is usually inserted into a deep axial hole in the end of the cartridge. The cord is then either taped to the cartridge, passed through a diagonal hole in the cartridge, or secured with a half hitch to assure that the cord will not pull out.

When priming large-diameter cartridges with electric blasting caps, a diagonal hole is punched from the top center of the cartridge and out the side about 8 inches from the top. The cap wires are doubled over, threaded through the hole, and wrapped around the cartridge. The cap is placed into a hole punched into the top of the cartridge and the assembly is pulled tight. Tape may be used for extra security.

Detonating cord is secured to large-diameter cartridges by punching a diametrical hole through the cartridge, passing the cord through the cartridge, and tying the cord at the top of the cartridge with a secure knot.

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This should not be done when using non-water-resistant explosive products in wet boreholes because the cartridge may become desensitized by water entering the punched hole. Cap and fuse is not commonly used with large cartridges. With other nonelectric initiators, it is recommended that cast primers rather than large-diameter explosive cartridges be used.

Cast primers (figure 11) are most commonly used to prime large-diameter blastholes. For use with detonating cord, a cast primer with a single axial hole is used. The cord is passed through the cord "tunnel" and tightly knotted at the bottom of the primer. It is not necessary to tie the cord around the primer because this knot will not pull back through the tunnel. Subsequent primers can be added wherever desired by passing the downline at the blasthole collar through the primer tunnel and sliding the primer down the downline. Placement of delay cast primers on the downline is done in a similar fashion, except that the tunnel for the cord is connected to the perimeter of the primer rather than passing through the center of the primer itself.

Cast primers for use with detonators have a cap well in addition to a tunnel. The cap is inserted through the tunnel and back up into the well, making sure that the cap is seated in the bottom of the well (figure 50). Although the cap will usually stay securely in the primer using this type of configuration, it is a good idea to use a wrap of tape around the end containing the cap well for security. Remember that not all cast primers have tunnels large enough to accept the Hercudet duplex tubing.

Primer Location

Proper location of the primer is important from the standpoint of both safety and efficiency (1, 6). When using cartridged products in small-diameter blastholes, the primer should be the first cartridge placed into the hole, with the cap pointing toward the collar. This assures maximum confinement and the most efficient use of the explosive's energy. Placing the primer in the bottom minimizes bootlegs and also protects against leaving undetonated explosives in the bottom of the hole if the cartridges become separated. The primer cartridge must not be cut, deformed, or tamped. If bulk products are being loaded, the primer may be raised slightly from the bottom of the hole.

In bench blasting with a bulk loaded product, where subdrilling is used, the primer should be placed at toe level, rather than in the bottom of the hole, to reduce ground vibrations. If there is some compelling reason to place the primer at the collar of the hole, the detonator should be pointed toward the bottom of the hole.

In large-diameter blastholes, the location of the primer is more a matter of choice, although bottom initiation is recommended to maximize confinement of the charge. To help reduce vibrations, the primer should be at the toe level rather than in the bottom of the hole, where subdrilling is used. Bottom-initiated holes tend to produce less flyrock and airblast than top-initiated holes, assuming that all other blast dimensions are equal. If pulling the toe is not a significant problem, some operators prefer to place the primer near the center of the charge. This gives the quickest total reaction of the explosive column and may yield improved fragmentation. Top priming is seldom recommended, except where the only fragmentation difficulty is a hard
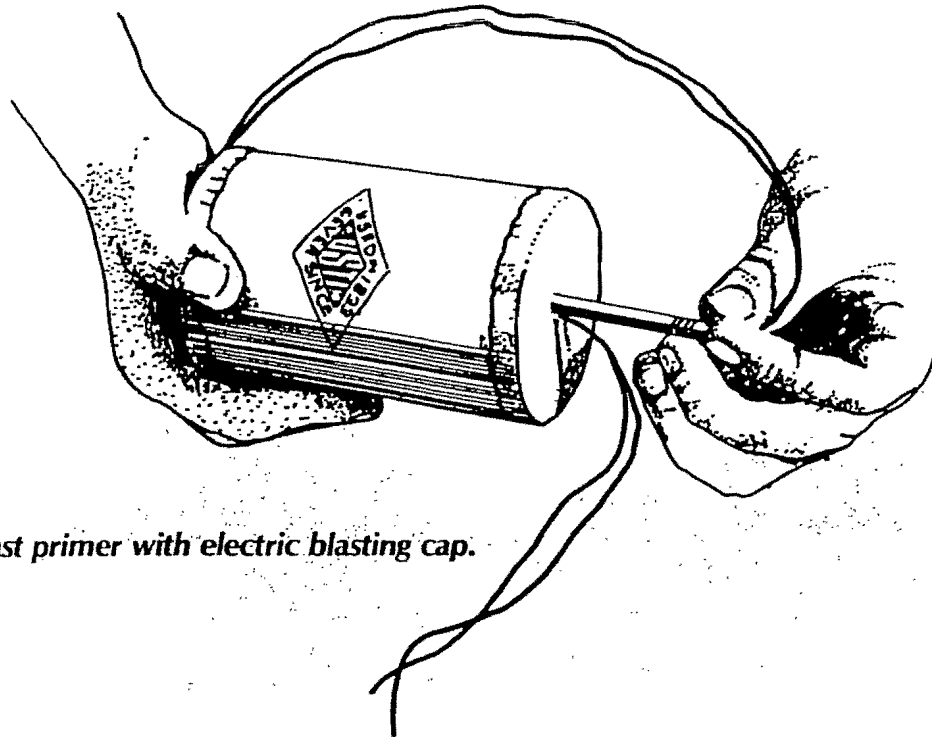
Figure 50: Priming cast primer with electric blasting cap.

band of rock in the upper portion of the bench. A rule of thumb, when using a single primer in a large-diameter blasthole, is to place the primer in the zone of most difficult breakage. This will normally be the toe area. Figure 51 summarizes some desirable and undesirable locations for primers in large-diameter blastholes.

Multiple Priming

In many blasting situations, single-point priming may be adequate. However, there are some situations in which multiple primers in a single borehole may be needed. The first is where deck charges are used. Deck charges are used (a) to reduce the powder factor in a blast while maintaining satisfactory powder distribution, (b) to break up boulder-prone caprock in the stemming area of the blast, or (c) to reduce the charge weight per delay to reduce vibrations. In situation c, each deck in the hole is on a different delay period. In a and b, the decks within a single hole may be on the same or on different delays. In any case, each deck charge requires a separate primer. Some states require at least two primers per blasthole.

The second reason for multiple priming is as a safety factor to assure total column detonation. With modern explosives and blasting agents, once detonation has been established, it will proceed efficiently through the entire powder column. However, an offset in the powder column (figure 34) may occur before detonation and cause part of the column not to propagate. This is most likely to occur with very long, thin charges or where slip planes are present in the burden area. In these cases, two or more primers should be spaced throughout the powder column. Frequently, these primers will be on the same

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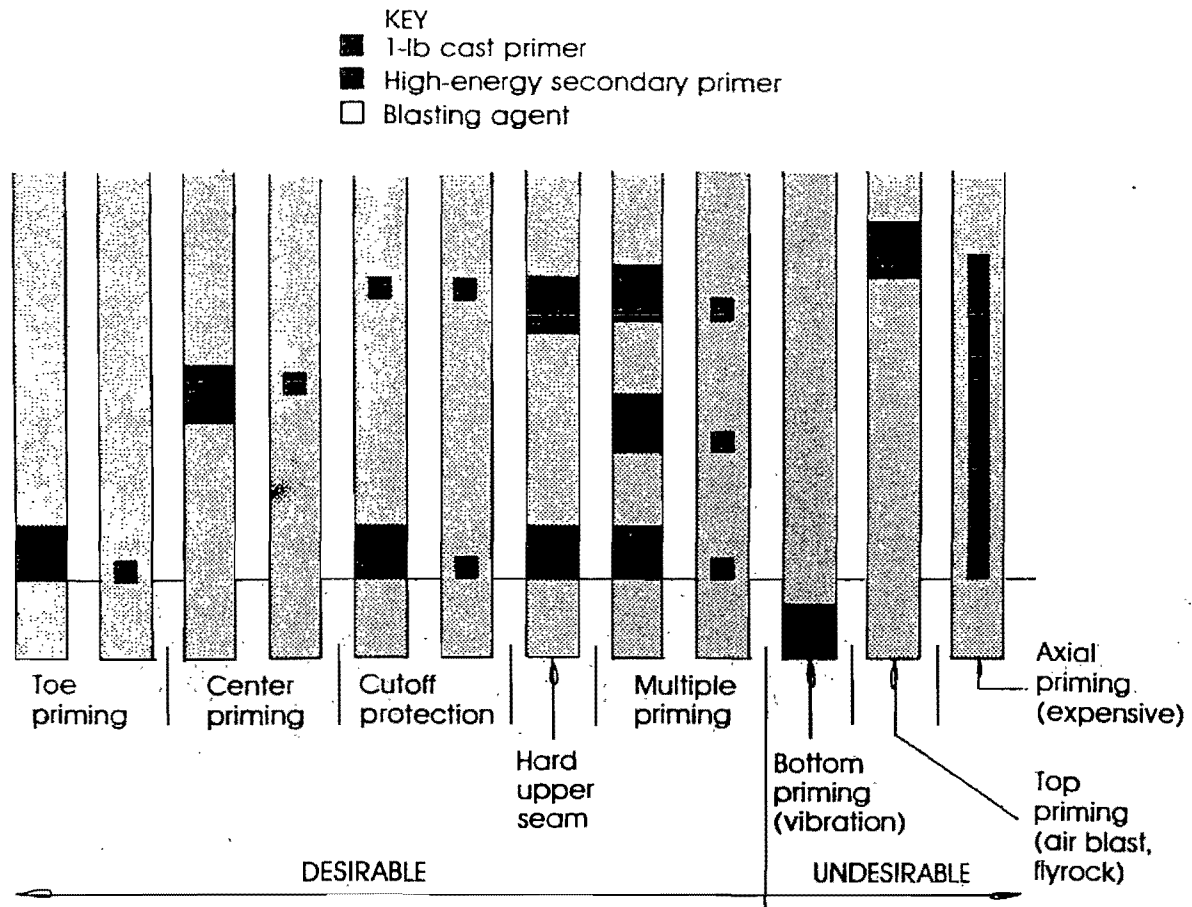
Figure 51: Priming blasting agents in large-diameter blastholes.

delay. Where single-point priming is preferred, but one or more additional primers are needed to assure total column propagation, the additional primers are put on a later delay period.

With multiple delayed decks in a blasthole, detonation should proceed from the bottom up where a good free face exists. Where the shot is tight, such as in area coal mining, detonation from the top down will give some relief to the lower decks.

Axial priming, which employs a central core of primer throughout an AN-FO column, has been used successfully but appears to have no particular advantage over single point or multiple point priming. Axial priming is more expensive than conventional priming.

REFERENCES

1. Ash, R.L. The Mechanics of Rock Breakage, Parts I, II, III, and IV. Pit and Quarry, v. 56, No. 2, August 1963, pp. 98-112; No. 3, September 1963, pp. 118-123; No. 4, October 1963, pp. 126-131: No. 5, November 1963, pp. 109-111, 114-118.
2. Atlas Powder Co. (Dallas, TX). Handbook of Electric Blasting. Rev. 1976, 93 pp.

3. Chironis, N.P. New Blasting Machine Permits Custom-Programmed Blast Patterns, Coal Age, v. 79, No. 3, March 1974, pp. 78-82.

4. Condon, J.L., and J.J. Snodgrass. Effects of Primer Type and Borehole Diameter on AN-FO Detonation Velocities. Min. Cong. J., v. 60, No. 6, June 1974, pp. 46-47, 50-52.

5. Dick, R.A. New Nonelectric Explosive Initiation Systems. Pit & Quarry, v. 68, No. 9, March 1976, pp. 104-106.

6. _____. Puzzled About Primers for Large Diameter AN-FO Charges? Here's Some Help to End the Mystery. Coal Age, v. 81, No. 8, August 1976, pp. 102-107.

7. E. I. du Pont de Numours & Co., Inc. (Wilmington, DE). Blaster's Handbook. 16th ed., 1978, 494 pp.

8. Ensign Bickford Co. (Simsbury, CN). Primacord Detonating Cord. 9th printing, copyright 1963, 68 pp.

9. Grant, C.H. Metallized Slurry Boosting: What It Is and How It Works. Coal Age, v. 71, No. 4, April 1966, pp. 90-91.

10. Hagan, T.N. Optimum Priming for Ammonium Nitrate Fuel-Oil Type Explosives. Proc. Southern and Central Queensland Conf. of the Australasian Inst. of Min. and Met., Parkville, Australia, July 1974, pp. 283-297; available for consultation at Bureau of Mines Twin Cities Research Center, Minneapolis, MN.

11. Institute of Makers of Explosives Safety Library (Washington, DC). Safety Guide for the Prevention of Radio Frequency Radiation Hazards in the Use of Electric Blasting Caps. Pub. No. 20, September 1981, 23 pp.

12. Junk, N.M. Overburden Blasting Takes on New Dimensions. Coal Age, v. 77, No. 1, January 1972, pp. 92-96.

13. _____. Research on Primers for Blasting Agents. Min. Cong. J., v. 50, No. 4, April 1964, pp. 98-101.

14. Sengupta, D., G. Granch, M. Heydari, and K. Hanna. The Impact of Eliminating Safety Fuse From Metal/Nonmetal Mines (Contract J0295010, Sci. Applications, Inc.). BuMines OFR 61-81, August 1980, 21 pp.; NTIS PB 81-214386.

CHAPTER 2
REVIEW QUESTIONS

1. What instrument is used to check for stray current at the blast sites?

    _____.

2. A typical CD blasting machine has _____ buttons which must be

    pressed in order to properly fire a shot.

3. The light on the CD blasting machine comes on to indicate--

    a. the shot has gone off

    b. the machine's batteries are ok

    c. the capacitor is charged to its rated capacity

4. Explain why a crimping tool is used.

5. A sequential timer can be used to set off as many as _____ different

    circuits.

6. Lightning, stray currents, static electricity and radio frequency energy

    are forms of _____.

7. Is it all right to use an electrician's meter to measure resistance in a

    blasting circuit? _____

8. When is it all right to use a 12-volt automobile battery to fire a blast?

    _____.

9. Detonating cord fires at a rate of _____.

10. Millisecond connectors are used to create delays when a blast is initi-

    ated with _____.

11. Many nonelectric delay systems provide two places where delays may occur.

    These two types of delays are _____ and _____.

12. Delays are achieved with detonating cord using _____.

13. Delays with the Ensign-Bickford Nonel are achieved by using _____.

14. The Hercudet system is charged after _____

    is introduced into a plastic tubing system.

15. What is the burning rate of a safety fuse? _____

16. Resistance is measured in _____.

17. Which copper wire has a higher electrical resistance: a 10-gauge or an

    18-gauge wire? _____.

18. Look at the wiring of the electric blasting caps in the three shots

    below. Identify each as either (a) series, (b) parallel, or (c) series-

    parallel.

1)   **2)**   **3)**

to power   to power   to power
source   source   source
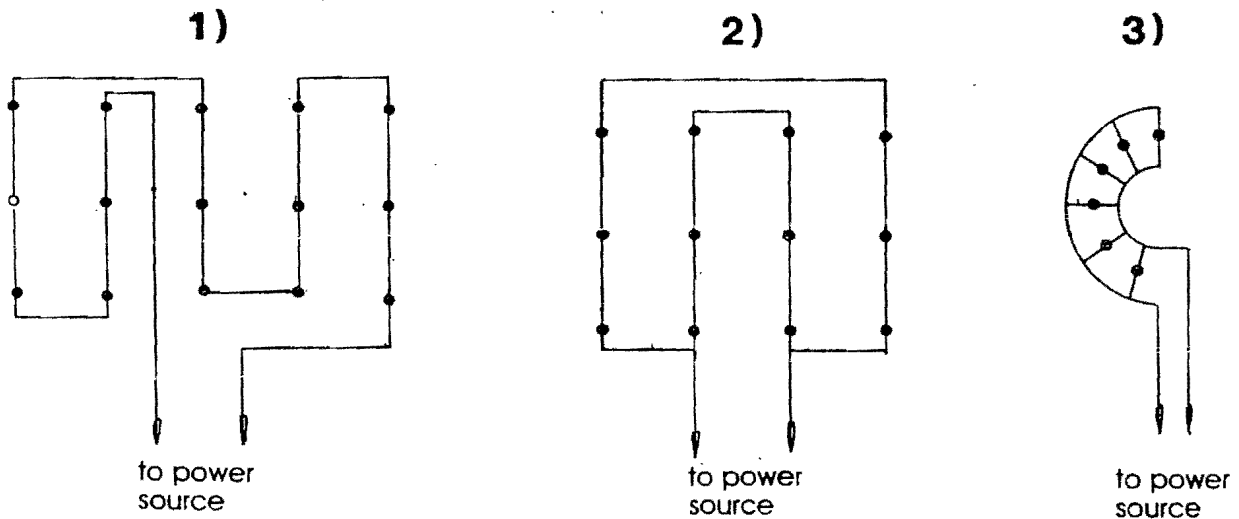
19. A shot consists of 4 holes wired in a series (see diagram below). Using

    delay caps with a resistance of 2.21, calculate the resistance of the

    circuit.

    Resistance = _____   to power
                                                             source

20. A shot consists of 4 holes wired in parallel. Each hole contains a delay cap with a resistance of 1.81. Calculate the resistance of the circuit with holes wired as shown in the diagram below.

to power source
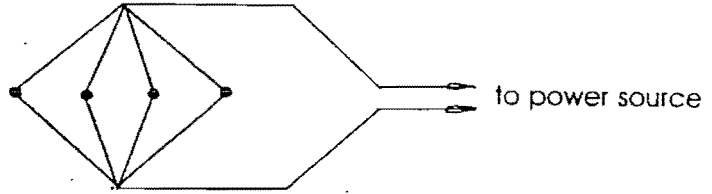
Resistance = _____

21. You are to fire a shot with 16 holes wired in series-parallel as shown in the diagram below. Using delay caps with a resistance of 2.59 ohms, calculate the resistance of the circuit.

to power source

Four Series of (4) Caps Each
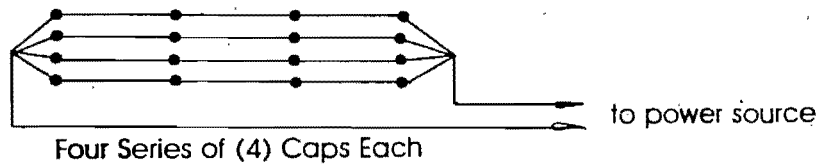
Resistance = _____

22. Two important factors in determining proper primer location are?

_____ and _____.

23. Top priming can result in _____ and

_____, and bottom priming can cause _____.

24. The three parts of an initiation system are:

a. _____

b. _____

c. _____

25. Three advantages of millisecond delays over simultaneous firing are:

a. _____

b. _____

c. _____

26. Four types of initiation systems are:

a. _____   c. _____

b. _____   d. _____

27. When is multiple priming used? _____

28. In small diameter holes, which direction should the end of the cap point?

_____

29. What are the two parts of a primer?

a. _____

b. _____

Chapter 3: Blasthole Loading

Blasthole loading involves placing all of the necessary ingredients into the blasthole, including the main explosive charge, deck charges, initiation systems, primers, and stemming. Blasthole loading techniques vary depending on borehole diameter, type of explosive, and size of the blast. For the purpose of this discussion, boreholes have been arbitrarily classified as small diameter (less than 4 inches) and large diameter (greater than 4 inches). Small-diameter boreholes may be drilled at practically any inclination from vertically down to vertically up. Large-diameter blastholes are usually drilled vertically down, but in some cases are angled or horizontal.

As a specific precaution, blastholes should never be loaded during the approach or progress of an electrical storm. General descriptions of blasthole loading procedures are in the literature (2-5).

CHECKING THE BLASTHOLE

Before loading begins, the blastholes should be checked. Depending on the designed depth, either a weighted measure or a tamping pole should be used to check that the boreholes are at the proper depth. If a hole is deeper than the plan calls for, drill cuttings or other stemming material should be used to bring the bottom of the hole up to the proper level. Loading an excessively deep blasthole is a waste of explosive and usually increases ground vibrations. Boreholes that are less than the planned depth should either be cleaned out with the drill or compressed air, or redrilled. Sometimes economics or equipment limitations may dictate that a shot be fired with a few short holes. The blasting foreman should make this decision.

Occasionally, a borehole may become obstructed. On a sunny day, a mirror may be used to check for obstructions. Obstructions in small holes may sometimes be dislodged with a tamping pole. In large vertical holes, a heavy weight suspended on a rope and dropped repeatedly on the obstruction may clear the hole. It may be necessary to use the drill string to clear a difficult obstruction or, if the obstruction cannot be cleared, redrilling may be necessary.

If it is necessary to redrill a hole adjacent to a blocked hole, the blocked hole should be filled with stemming. If this is not done, the new hole may shoot into the blocked hole and vent, causing excessive flyrock, airblast, and poor fragmentation. A hole must not be redrilled where there is a danger of intersecting a loaded hole.

While checking the hole for proper depth, it is convenient to check for water in the borehole. With just a little experience, the blaster can closely estimate the level of water in a borehole by visually checking the tamping

pole or weighted tap for wetness after the borehole depth check has been made. To get a more accurate check, the weighted end of the tape can be jiggled up and down at the water level. A splashing sound will indicate when the weight is at the water level.

A blasthole may pass through or bottom into an opening. Where this opening is not unduly large, it may be filled with stemming material (figure 52). Where the opening is too large for this to be practical, the hole must either be left unloaded, redrilled in a nearby location, or plugged.
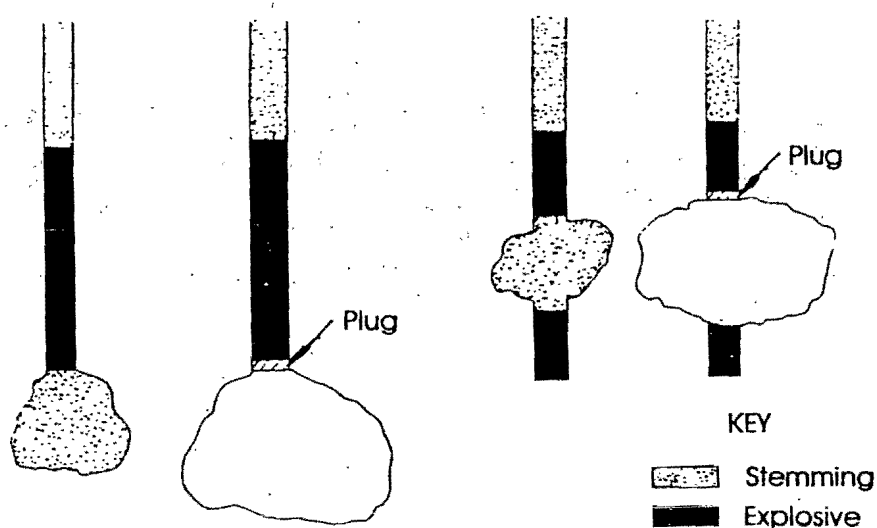
Figure 52: Corrective measures for voids.

A simple method for plugging a blasthole is as follows. A stick is tied to the end of a rope, lowered into the void, and pulled back up so it lodges crosswise across the hole. The rope is staked securely at the borehole collar. Bulky materials, such as empty powder bags or rags, are then dropped down the hole and dirt is shovelled down the hole to form a solid bottom, after which explosive loading can proceed. Where voids are commonplace, you may want to develop a tailormade borehole plugging device.

In some districts, hot holes may be encountered, although this is not very common. Hot holes may occur in anthracite mining or other areas of in situ coal seam fires. If there is reason to suspect a hot hole, the hole can be checked by suspending a thermometer in it for a few minutes. Explosive materials should not be loaded into holes hotter than 150° F.

GENERAL LOADING PROCEDURES

Blastholes may be loaded with bulk or packaged products. Bulk products are either poured into the hole, augered, pumped, or blown through a loading hose. Packaged products are either dropped into the hole, or pushed in with a

tamping pole, weighted tape, or loading hose. This will give warning of a cavity or oversized hole that is causing a serious overcharge of explosive to be loaded, and will also assure that sufficient room is left at the top of the hole for the proper amount of stemming. When the powder column has reached the proper location, the primer is loaded into the borehole. It is important that the wires, tubes, or detonating cord leading from the primer are properly secured at the borehole collar in vertical or nearly vertical holes, using a rock or stake.

In almost all situations it is recommended that the explosive charge be totally coupled. Total coupling means that the charge completely fills the borehole diameter. Bulk loading of explosives assures good coupling. When cartridged products are used, coupling is improved by slitting the cartridges and tamping them firmly into place. There are four situations where cartridges or packages of explosives should not be tamped.

1. Permissible coal mine blasting, where deforming the cartridge is against regulations.
2. In controlled blasting, where string loads or even gaps between cartridges are used to reduce the charge load in the perimeter holes to prevent shattering.
3. In water, where the package serves as protection for a non-water-resistant explosive product.
4. When using a primed cartridge.

It is recommended that all blastholes be stemmed to improve the efficiency of the explosive and to reduce airblast and flyrock. As a rule of thumb, the length of stemming should be from 14 to 28 times the borehole diameter. Sized crushed stone makes the most efficient stemming. However, for reasons of economy and convenience, drill cuttings are most commonly used. Large rocks should never be used as stemming because they could become a dangerous source of flyrock and may also damage the wires, cord, or tubes of the initiation system. Because it is inconvenient to stem horizontal holes, horizontal rounds are sometimes left unstemmed, although it is recommended that all blastholes be stemmed to improve blasting efficiency. By regulation, underground coal mine rounds must be stemmed with noncombustible stemming such as waterfilled cartridges or clay "dummies."

Care must be exercised in using detonating cord downlines in relatively small blastholes. See "Field Application" in the "Detonating Cord Initiation" section of chapter 2 for recommended grain loads of detonating cord as a function of blasthole diameter.

One solution to blasting in wet boreholes is to use a water-resistant explosive. However, economics often favor dewatering the blasthole and loading it with AN-FO inside a protective plastic borehole liner. Although dewatering has been used mostly in large-diameter holes, it can be used in diameters below 4 inches. To dewater, a pump is lowered to the bottom of the hole. When the water has been removed, the hole is lined with a plastic sleeve as follows. A roll of hollow plastic tubing is brought to the collar of the hole. A rock is placed inside the end of the tubing and a knot is tied in the end of the tubing to hold the rock in place. The tubing is reeled into the borehole, and care is taken not to tear it. The tubing is cut off at the collar, allowing 4 to 6 feet extra for charge settlement. The AN-FO and

primer are loaded inside the tubing and the hole is stemmed. Where water is seeping into the borehole, it is important that the tubing and AN-FO be loaded quickly to prevent the hole from refilling with water.

SMALL-DIAMETER BLASTHOLES

When small-diameter blastholes are loaded, the primer cartridge is normally loaded at the bottom of the hole. This gives maximum confinement at the point of initiation and also guards against leaving undetonated explosive in the bottom of the borehole if it should become plugged during loading or cut off during the blasting process. Some experts condone, or even recommend, a cushion stick or two, but the general recommendation is not to use a cushion stick. To avoid having the detonator fall out of the primer cartridge, the cartridge should never be slit, rolled, or otherwise deformed. The primer cartridge should never be tamped.

Cartridged Products

Cartridge dynamites and slurries (water gels) are commonly used in small-diameter blastholes. These cartridges are usually slit, loaded by hand, and tamped to provide maximum coupling and loading density. One or two cartridges should be loaded after the primer before tamping begins. Tamping should be done firmly, but not excessively. Using the largest diameter cartridge compatible with the borehole diameter will increase coupling and loading density.

Pneumatic systems for loading water gel cartridges are available. The cartridges are propelled through a loading hose at high velocity at a rate of up to one cartridge per second. The cartridges are automatically slit as they enter the blasthole and each cartridge splits upon impact. Because of the high impact imparted to the cartridges, loading dynamites with this type of loading system is not permitted. Pneumatic cartridge loaders are especially useful in loading holes that have been drilled upward.

Bulk Dry Blasting Agents

Bulk dry blasting agents, usually AN-FO, may be loaded into small-diameter blastholes by pouring from a bag or by pneumatic loading through a loading hose (figure 53). Poured charges in diameters less than 4 inches lose some efficiency because of AN-FO's low density and its reduced detonation velocity at small diameters. As with all bulk loading, good coupling is achieved. Caution should be exercised in using poured AN-FO charges in diameters less than 2 inches. This should be done only under bone-dry conditions because AN-FO's efficiency begins to drop significantly at this point, and water will compound the problem.

Pneumatic loading of AN-FO in small holes is recommended because of ease of handling, faster loading rates, and the improved performance of the AN-FO caused by partial pulverizing of the prills, which gives a higher loading density and greater sensitivity (1, 4). The two basic types of pneumatic loading systems are the pressure vessel and the ejector or venturi-type loader.

A pressure-vessel-type AN-FO loader should have a pressure regulator so that the tank pressure does not exceed the manufacturer's recommendation,

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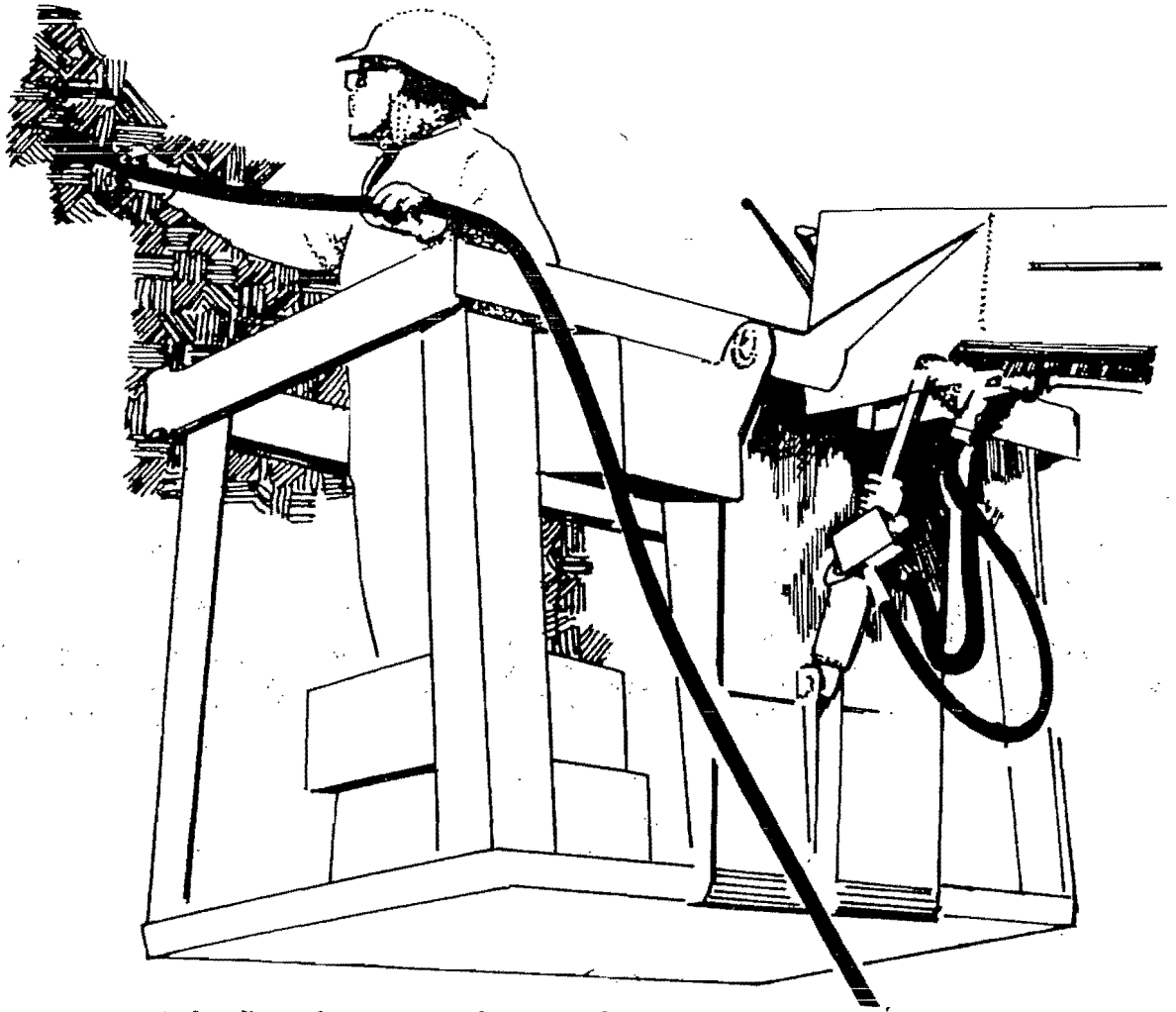
Figure 53: Pneumatic loading of AN-FO underground.

usually 30 psi. This low-pressure type loader propels the prills into the borehole at a low velocity and high volume rate, loading the AN-FO at a density slightly above its poured density with a minimum amount of prill breakage. In a pressure vessel, the compartment containing the AN-FO is under pressure during loading. Loading rates of over 100 lb/min can be achieved with some equipment and pressure vessels with AN-FO capacities of 1,000 lb are available. The smaller and more portable pressure vessel loaders have loading rates of 15 to 50 lb/min and AN-FO capacities of 75 to 200 lb. Pressure vessels larger than 1 cubic foot in volume should meet ASME specifications for construction.

The ejector-type system (figure 54) uses the venturi principle to draw AN-FO from the bottom of an open vessel and propel it at a high velocity but low volume rate into the borehole, pulverizing the prills and giving bulk loading densities near 1.00. Ejector systems operate from line pressures of 40 to 80 psi and load at rates of 7 to 10 lb/min. Combination loaders are available that force feed a venturi from a pressurized pot. This system gives the same high loading density and prill breakage as the straight venturi loader with an increase in loading rate. Specifications of pneumatic loading systems are given in table 3. The detonation velocity of AN-FO as a function

of charge diameter for poured and pneumatically loaded charges is shown in figure 55. The benefits of high-velocity pneumatic loading are significant at small borehole diameters.
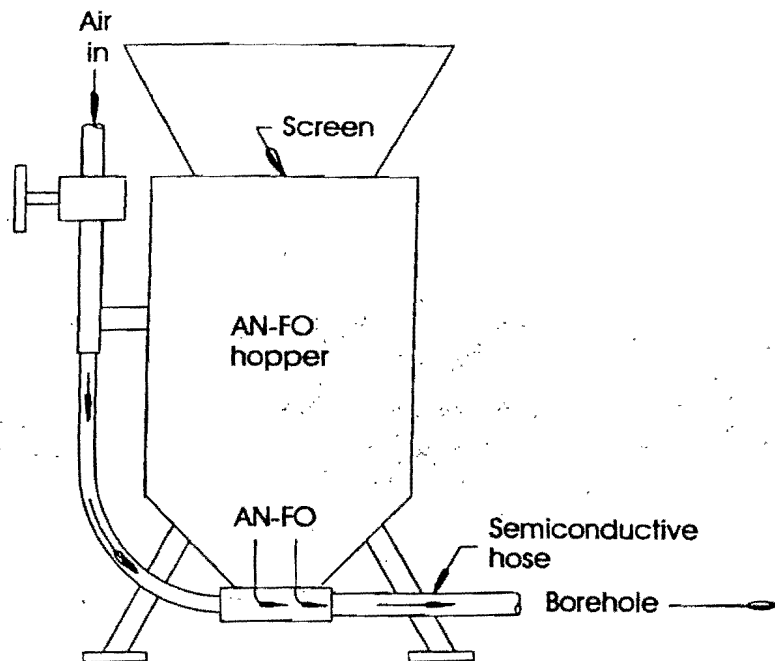
Figure 54: Ejector-type pneumatic AN-FO loader.

Table 3--Characteristics of Pneumatically Loaded AN-FO in Small-Diameter Blastholes

Loading device	Tank pressure psi	Jet pressure psi	Loading rate[1] lb/min	Loading density g/cm^3
Pressure vessel	10-30	NAp	15-70	0.80-0.85
Ejector loader (jet)	NAp	40-80	7-10	0.90-1.00
Combination loader	20	20-80	15-25	0.90-1.00

NAp Not Applicable
[1] Varies with hose diameter.

A problem may arise where a high-pressure ejector loader is used to load AN-FO in small holes in soft formations such as uranium ore. The pulverized prills may be dead pressed by the compression from adjacent charges fired on earlier delays. This can cause the AN-FO not to fire.

Static electricity can be a hazard when loading AN-FO pneumatically into small-diameter boreholes. Static electricity hazards can be reduced by using

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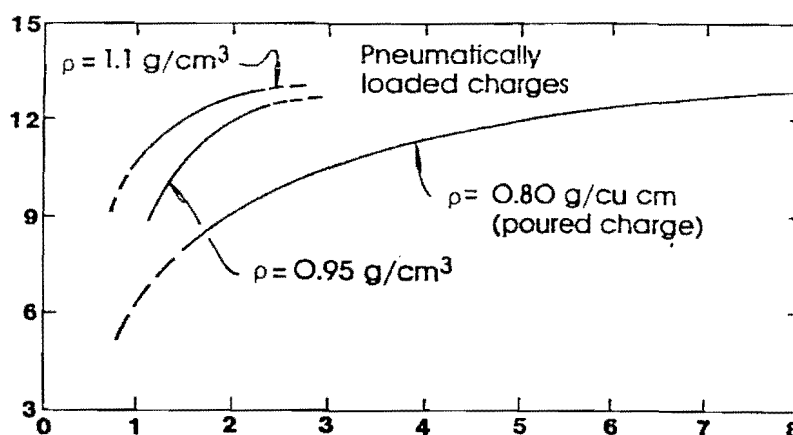
Figure 55: AN-FO detonation velocity as a function of charge diameter and density.

antistatic caps or nonelectric initiators such as Hercudet, Nonel, or Detaline. A semiconductive hose with a minimum resistance of 1,000 ohms/ft and 10,000 ohms total resistance, and a maximum total resistance of 2,000,000 ohms for the entire system, should be used. The pneumatic loader should be properly grounded.

Homemade loading equipment should not be used. All equipment should be operated at the proper pressure. Gaps in the powder column can be avoided by keeping the hopper full and maintaining a constant standoff distance between the end of the loading hose and the column of AN-FO. Loading proficiency improves through operator experience.

The pneumatic loading tube is useful for blowing standing drill water from a horizontal borehole. However, if the borehole is "making water," external protection for the AN-FO by means of a plastic sleeve is required. Loading inside a plastic borehole sleeve is not recommended for underground work because of the static electricity hazard during loading and toxic fumes generated during blasting. If plastic-sleeve protection with pneumatic loading in well-ventilated locations is required, a nonelectric detonating system should be used because the insulating effect of the sleeve is likely to cause a buildup of static electricity.

Bulk Slurries

Slurries may be bulk loaded into blasthole diameters as small as 2 inches. These products are frequently poured from bags (figure 56), but occasionally bulk pumping units are used (figure 57). The sensitivity of slurries, and hence the diameter at which they may be used effectively, depends largely on their formulation. The use of bulk slurries in diameters below those intended for the product can result in substandard blasts or misfires. The manufacturer should be consulted when loading bulk slurries into small-diameter blastholes.
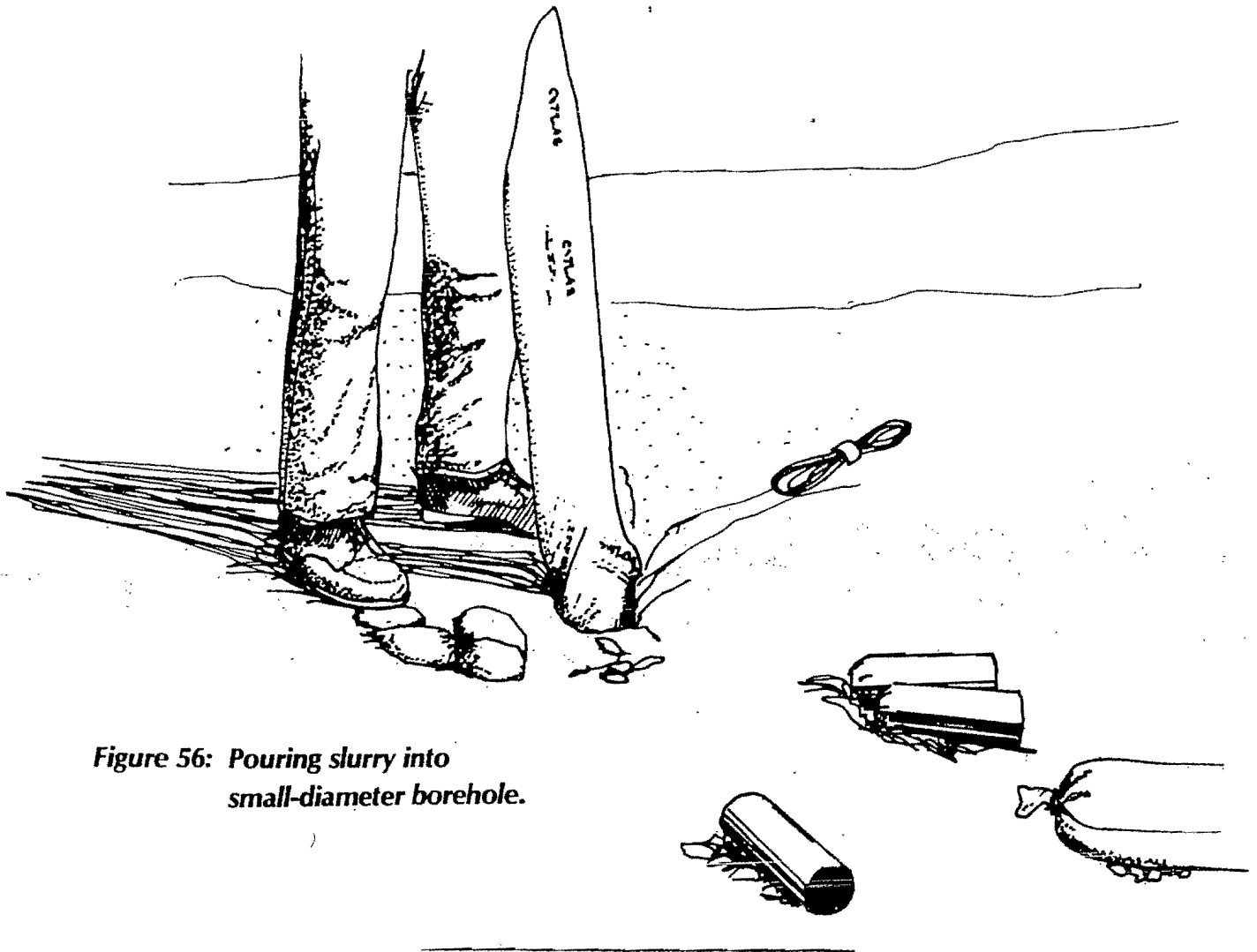
Figure 56: Pouring slurry into small-diameter borehole.

Permissible Blasting

Loading blastholes in underground coal mines is strictly regulated by MSHA in order to prevent ignition of explosive atmospheres. Only permissible explosives may be used in underground coal mines. Certain nitroglycerin-based explosives, emulsions, slurries, and water gels have been certified as permissible by MSHA (6).

The primer plus the remaining cartridges are stringloaded and pushed back into the hole as a single unit to avoid getting coal dust between the cartridges. Charge weights may not exceed 3 pounds per borehole. Black powder, detonating cord, and AN-FO are not permissible. Blastholes are initiated with copper alloy shell electric blasting caps. All holes must be stemmed with noncombustible material such as water bags or clay dummies. The stemming length must be at least 24 inches or one-half the depth of the borehole, whichever is less. Additional rules of permissible blasting are given in the "Blast Design" chapter. Permissible blasting procedures are also required for gassy noncoal mines, but are frequently less stringent than for coal mines.

Figure 57: Pumping slurry into small-diameter borehole.

LARGE-DIAMETER BLASTHOLES

With few exceptions, economics and efficiency favor the use of bulk loading in blasthole diameters larger than 4 inches. The products are cheaper, loading is faster, and the well-coupled bulk charge gives better blasting

efficiency. As described in the "Priming" section of chapter 2, large-diameter blastholes may be top, center, or toe primed, or multiple primers may be used.

Packaged Products

Large-diameter dynamite cartridges are seldom used today except for occasional use as primers. AN-FO and slurries give better economy in large-diameter blastholes. When wet boreholes are encountered, and the operator wants to use AN-FO, water-resistant polyburlap packages of partially pulverized, densified AN-FO are used (figure 7). Densification is necessary so that the packages will sink in water. AN-FO packages should be carefully lowered into water-filled holes rather than dropped, because a broken bag will result in desensitized AN-FO, an interruption in the powder column and, most likely, some unfired AN-FO. A disadvantage of waterproof AN-FO packages is that some borehole coupling is lost. Also, the heat lost to the water will reduce the energy released. Where it is desired to use AN-FO in wet boreholes, the option of borehole dewatering should be investigated.

· Slurries are available in polyethylene packaging in diameters up to 8 inches (figure 10). Some of these products are semi-rigid and others are in dimensionless bags that will slump to fit the borehole diameter. With the semi-rigid cartridges, the advantage of borehole coupling is lost.

Bulk Dry Blasting Agents

Bulk loading offers significant advantages over loading of packaged products in large-diameter blastholes, including cheaper products, faster loading, and better use of the available space in the borehole.

The bulk AN-FO or prills are stored in overhead storage bins, from which they are loaded into the bulk trucks. The AN-FO may be trucked to the blast site in premixed form or the oil may be metered into the prills as they are placed into the blasthole. Bulk loading systems for dry blasting agents (AN-FO) may be of the auger or pneumatic type.

Auger loading gives the fastest loading rates. A side-boom auger is satisfactory for loading one row of holes at a time. Where it is desired to reach more holes from one setup, an overhead-boom auger with a 350° radius of swing can be used. With this type of equipment, flexible tubing usually extends from the end of the auger boom to ground level. The amount of blasting agent delivered into the blasthole is sometimes indicated on a meter in the truck. In other situations, a hopper with a given volume of capacity is hung at the end of the auger boom to measure the AN-FO as it is loaded. Bulk loading trucks have capacities of from 2,000 to 30,000 pounds of AN-FO, and with auger systems can deliver up to 600 pounds of AN-FO per minute into a blasthole.

Pneumatic loading is also used in large-diameter boreholes. Pneumatic units are especially useful in rough terrain, where a long loading hose is used to load numerous blastholes from a single setup.

Hand pouring AN-FO from 50-pound bags is still practiced at operations where the capital expense of a bulk system cannot be justified. This, of course, gives the same complete coupling as bulk loading.

Bulk Slurries

Bulk slurry pumping is commonplace in large-diameter, vertical-hole blasting. Some slurry trucks have capacities of up to 30,000 pounds of slurry and have typical pumping rates of 200 to 400 pounds/minute. A bulk slurry truck may bring a plant-mixed slurry to the borehole or it may carry separate ingredients for onsite mixing.

Onsite slurry mixing is more complex than AN-FO mixing and is usually done by a competent explosive distributor rather than the consumer. Plant mixing permits closer quality control in the blending of ingredients, whereas onsite mixing permits different energy densities to be loaded from hole to hole or in different locations within a single hole.

The slurry is pumped as a liquid (figure 58) and a cross-linking ingredient is added just as the slurry enters the loading hose. Cross linking to a gelatinous consistency begins in the hose and is completed in the borehole. A meter on the bulk truck indicates the amount of slurry that has been loaded.

Hand pouring of slurry from polyethylene packages (figure 56) is still practiced at operations where the volume of slurry used does not justify a bulk-loading truck. Pouring, rather than loading the entire package, gives complete borehole coupling.

REFERENCES

1. Atlas Powder Co. (Dallas, TX). Pneumatic Loading of Nitro-Carbo-Nitrates; Static Electricity, Fumes, and Safe Handling. Undated, 17 pp.
2. Dannenberg, J. How to Solve Blasting Materials Handling Problems. Rock Products, v. 74, no. 9, September 1971, pp. 63-65.
3. Dick, R.A. Explosives and Borehole Loading. Subsection 11.7, SME Mining Engineering Handbook, ed. by A.B. Cummins and I.A. Given. Society of Mining Engineers of the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, v. 1, 1973, pp. 11-78--11-99.
4. E.I. du Pont de Nemours & Co., Inc. (Wilmington, DE). Blaster's Handbook. 16th ed., 1978, 494 pp.
5. Langefors, U., and B.A. Kihlstrom. The Modern Technique of Rock Blasting. John Wiley & Sons, Inc., New York, 1963, 405 pp.
6. U.S. Mine Enforcement and Safety Administration. Active List of Permissible Explosives and Blasting Devices Approved Before December 31, 1975. MESA Inf. Rep. 1046, 1976, 10 pp.

Figure 58: Slurry leaving end of loading hose.

CHAPTER 3
REVIEW QUESTIONS

1. Unstemmed and blocked boreholes may cause _____, _____, and

    _____ when an adjacent new hole is blasted.

2. When should all boreholes be rechecked? _____

3. When wet holes exist, what type of explosive should be used? _____

    _____

4. What are the four situations where cartridges or packages of explosives

    should not be tamped?

    a. _____

    _____

    b. _____

    _____

    c. _____

    _____

    d. _____

    _____

5. Blastholes should never be loaded when _____.

6. When should a borehole not be redrilled? _____

7. What are four methods of loading bulk products?

    a. _____

    b. _____

    c. _____

    d. _____

8. How are package products loaded? _____

9. Stemming length used for permissible blasting is _____

    and for other blasting is _____.

10. What is the advantage of onsite mixing of a bulk slurry and what is the disadvantage? _____

_____

11. What is a disadvantage of using waterproof AN-FO packages?

_____

12. What is a hazard when loading AN-FO pneumatically into small diameter boreholes?

_____

13. Why should blastholes be stemmed?

_____

Chapter 4: Blast Design

Blast design is not a precise science. Because of the widely varying nature of rock, geologic structure, and explosives, it is impossible to set down a series of equations which will enable the blaster to design the ideal blast without some field testing. Tradeoffs must frequently be made in designing the best blast for a given situation. This chapter will describe the fundamental concepts of blast design. These concepts are useful as a first approximation for blast design and also in troubleshooting the cause of a bad blast. Field testing is necessary to refine the individual blast dimensions.

Throughout the blast design process, two overriding principles must be kept in mind: (a) Explosives function best when there is a free face approximately parallel to the explosive column at the time of detonation, and (b) there must be adequate space into which the broken rock can move and expand. Excessive confinement of explosives is the leading cause of poor blasting results such as backbreak, ground vibrations, airblast, unbroken toe, flyrock, and poor fragmentation.

Many of the principles discussed in this section were first presented by Ash (2) and later reported by Pugleise (7) during a study of quarry in practices in the U.S.

PROPERTIES AND GEOLOGY OF THE ROCK MASS

The character of the rock mass is a critical variable affecting the design and results of a blast. The nature of rock is very qualitative and cannot be quantified numerically. Rock character often varies greatly from one part of a mine to another or from one end of a construction job to another. Decisions on explosive selection, blast design, and delay pattern must take into account firsthand knowledge of the rock mass. For this reason, the onsite blaster usually has a significant advantage over an outside consultant in designing a blast. Although the number of variations in the character of rock is practically infinite, a general discussion of the subject will be helpful. The Bureau of Mines has published a report (7) that discusses the effects of geology on blast design.

Characterizing the Rock Mass

The keys to characterizing the rock mass are a good geologist and a good driller. The geologist concentrates on obtaining data from the rock surface. Jointing is probably the most significant geologic feature of the rock. The geologist should document the direction, severity, and spacing between the joint sets. In most sedimentary rocks, there are at least three joint sets--

one dominant and two less severe. The strike and dip of bedding planes are also documented by the geologist. The presence of major zones of weakness

such as faults, open beds, solution cavities, or zones of incompetent rock or unconsolidated material are also determined. Samples of freshly broken rock can be used to determine the hardness and density of the rock.

An observant driller can be of great help in assessing rock variations that are not apparent from the surface. Slow penetration and excessive drill noise and vibration indicate a hard rock that will be difficult to break. Fast penetration and a quiet drill indicate a softer, more easily broken zone of rock. Total lack of resistance to penetration, accompanied by a lack of cuttings or return water or air, means that the drill has hit a void zone. Lack of cuttings or return water may also indicate the presence of an open bedding plane or other crack. A detailed drill log indicating the depth at which these various conditions exist can be very helpful to the person design-ing the blast. The driller should also document changes in the color or nature of the drill cuttings, which will tell the blaster the location of various beds in the formation.

Rock Density and Hardness

Some amount of displacement is required to prepare a muckpile for effi-cient excavation. The density of the rock is a major factor in determining how much explosive is needed to displace a given volume of rock (powder fac-tor). The burden-to-charge diameter ratio, which will be discussed in the next section, "Surface Blasting," varies with rock density, causing the change in powder factor. The average burden-to-charge-diameter ratio of 25 to 30 is for average density rocks such as limestone (2.5 to 2.8 g/cm^3), schist (2.6 to 2.8 g/cm^3), or porphyry (2.5 to 2.6 g/cm^3). Denser rocks, such as basalt (2.9 g/cm^3) and magnetite (4.9 to 5.2 g/cm^3), require small ratios (higher powder factors). Lighter materials, such as some sandstones (2.0 to 2.6 g/cm^3) or bituminous coal (1.2 to 1.5 g/cm^3), can be blasted with higher ratios (lower powder factors).

The hardness or brittleness of rock can have a strong effect on blasting results. Soft rock is much more "forgiving" than hard rock. If soft rock is slightly underblasted, it will probably still be diggable. If soft rock is slightly overblasted, excessive violence will not usually occur. On the other hand, slight underblasting of hard rock will often result in a tight muckpile that is difficult to dig. Overblasting of hard rock is likely to cause exces-sive flyrock and airblast. Therefore, blast designs for hard rock require closer control and tighter tolerances than those for soft rock.

Voids and Incompetent Zones

Unforeseen voids and zones of weakness, such as solution cavities, under-ground workings, mud seams, and faults, are serious problems in blasting. Explosive energy always seeks the path of least resistance (figure 59). Where the rock burden is composed of alternate zones of hard material and incompe-tent material or voids, the explosive energy will be vented through the incom-petent zones, resulting in poor fragmentation. Depending on the orientation of the zones of weakness with respect to free faces, excessive violence in the form of airblast and flyrock may occur. A particular problem occurs when the
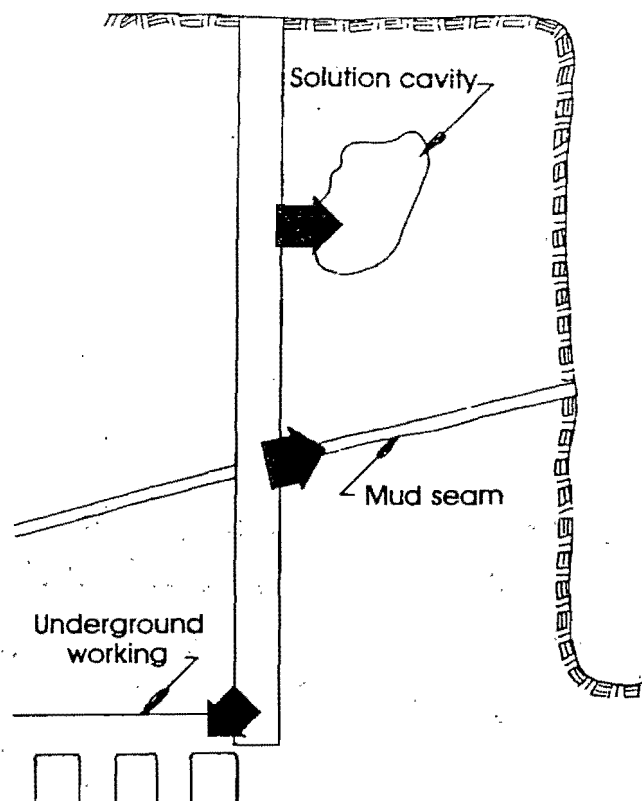
Figure 59: Loss of explosive energy through zones of weakness.

blasthole intersects a void zone. In this situation, unless particular care is taken in loading the charge, the void will be loaded with a heavy concentration of explosive, resulting in excessive airblast and flyrock.

If these voids and zones of weakness can be identified, steps can be taken during borehole loading to improve fragmentation and avoid violence. The best tool for this is a good drill log. The depths of voids and incompetent zones encountered by the drill should be documented. The geologist can help by plotting the trends of mud seams and faults. When charging the blasthole, inert stemming material rather than explosives, should be loaded through these weak zones. Voids should be filled with stemming (figure 52). Where this is impractical because of the size of the void, it may be necessary to block the hole just above the void before continuing the explosive column, as described in the "Checking the Blasthole" section of chapter 3.

Where the condition of the borehole is in doubt, the rise of the powder column should be checked frequently as loading proceeds. If the column fails to rise as expected, there is probably a void. At this point, a deck of inert stemming material should be loaded before powder loading continues. If the column rises more rapidly than expected, frequent checking will assure that adequate space is left for stemming.

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Alternate zones of competent and incompetent rock usually result in unacceptably blocky fragmentation. A higher powder factor will seldom correct this problem; it will merely cause the blocks to be displaced farther. Usually, the best way to alleviate this situation is to use smaller blastholes with

smaller blast pattern dimensions to get a better powder distribution. The explosive charges should be concentrated in the competent rock, with the incompetent zones being stemmed through wherever possible.

Jointing

Jointing can have a pronounced effect on both fragmentation and the stability of the perimeter of the excavation. Close jointing usually results in good fragmentation. However, widely spaced jointing, especially where the jointing is pronounced, often results in a very blocky muckpile because the joint planes tend to isolate large blocks in place. Where the fragmentation is unacceptable, the best solution is to use smaller blastholes with smaller blast pattern dimensions. This extra drilling and blasting expense will be more than justified by the savings in loading, hauling, and crushing costs, and the savings in secondary blasting.

Where possible, the perimeter holes of a blast should be aligned with the principal joint sets. This will tend to produce a more stable excavation, whereas rows of holes perpendicular to a primary joint set will tend to produce a more ragged, unstable perimeter (figure 60). Jointing will often determine how the corners at the back of the blast will break out. To minimize backbreak and violence, tight corners, shown in figure 61, should be avoided. The open corner at the left of the figure is preferable. Given the predominant jointing in figure 61, more stable conditions will result if the first blast is opened at the far right and is designed so that the hole in the rear inside corner contains the highest numbered delay.
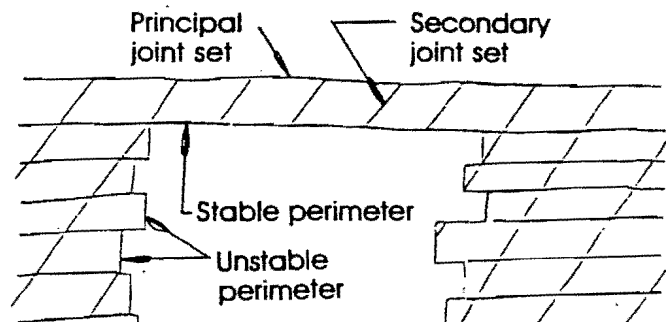
Figure 60: Effect of jointing on the stability of an excavation.

Bedding

Bedding can also have an effect on both the fragmentation and the stability of the excavation perimeter. Open bedding planes or beds of weak material should be treated as zones of weakness. Stemming, rather than explosive, should be loaded into the borehole at the location of these zones as shown in figure 62. In a bed of hard material, it is often beneficial to load an explosive of higher density than is used in the remainder of the borehole. To break an isolated bed of hard material near the collar of the blasthole, a
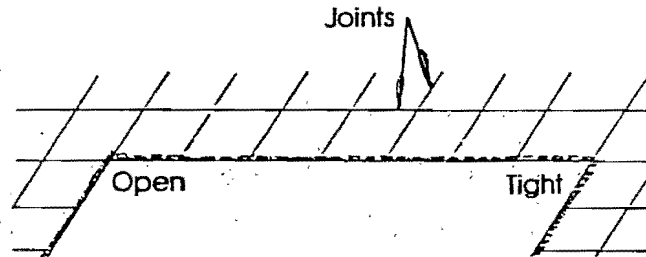
Figure 61: Tight and open corners caused by jointing.
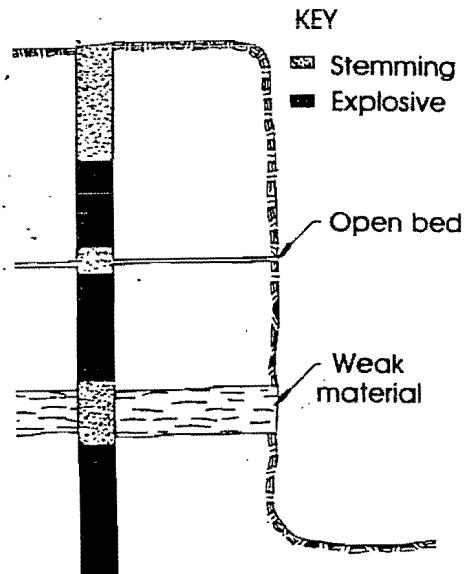
Figure 62: Stemming through weak material and open beds.

deck charge is recommended, as shown in figure 63, with the deck being fired on the same delay as the main charge or one delay later. Occasionally, satellite holes are used to help break a hard zone in the upper part of the burden. Satellite holes are short holes, usually smaller in diameter than the main blastholes, which are drilled between the main blastholes.

A pronounced bedding plane is frequently a convenient location for the floor of the bench. It not only gives a smoother floor, but also may reduce subdrilling requirements.

KEY

▨ Stemming

■ Explosive

DECK CHARGE

SATELLITE HOLE
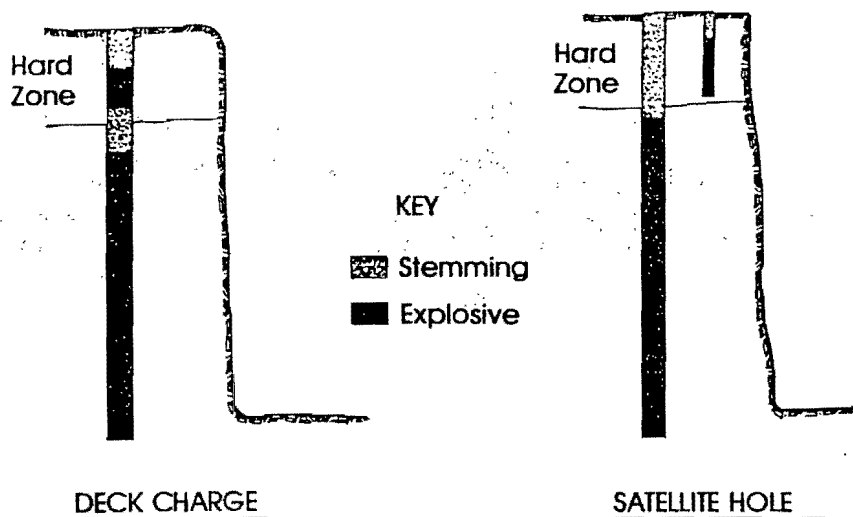
Figure 63: Two methods of breaking a hard collar zone.

Dipping beds frequently cause stability problems and difficulty in breaking the tow of the burden. When the beds dip into the excavation wall, the stability of the slope is enhanced (figure 64). However, when beds dip outward from the wall, they form slip planes that increase the likelihood of slope deterioration. Blasthole cutoffs caused by differential bed movement are also more likely. Beds dipping outward from the final slope should be avoided wherever possible.

Although beds dipping into the face improve slope stability, they do create toe problems (figure 64), because the toe material tends to break out along the bedding planes. Dipping beds such as these require a tradeoff. Which is the more serious problem in the job at hand, a somewhat unstable slope or an uneven toe? In some cases advancing the opening perpendicular to the dipping beds may be a good compromise.

Many blasting jobs encounter site-specific, geologic conditions not covered in this general discussion. A good explosives engineer is constantly studying the geology of the rock mass and making every effort to use the geology to advantage, or at least to minimize its unfavorable effects.

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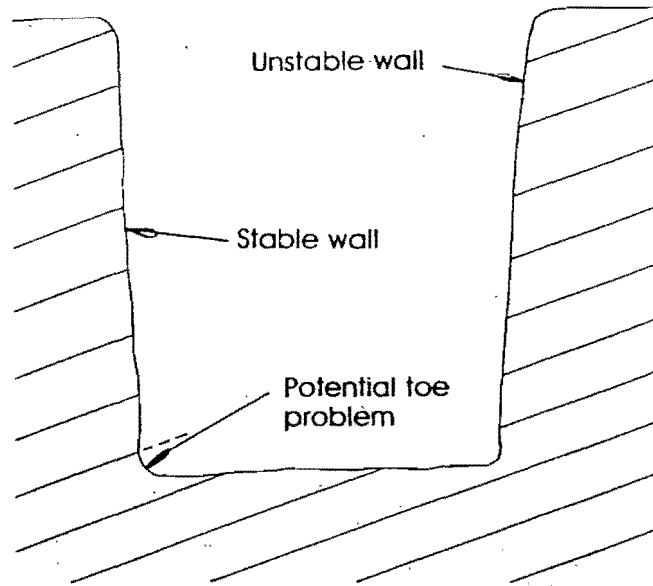
Figure 64: Effect of dipping beds on slope stability and potential toe problems.

SURFACE BLASTING

Blasthole Diameter

The size of the blasthole is the first consideration of any blast design. The blasthole diameter, along with the type of explosive being used and the type of rock being blasted, will determine the burden. All other blast dimensions are a function of the burden. This discussion assumes that the blaster has the freedom to select the borehole size. In many operations, one is limited to a specific size borehole based on available drilling equipment.

Practical blasthole diameters for surface mining range from 2 to 17 inches. As a general rule, large blasthole diameters yield low drilling and blasting costs because large holes are cheaper to drill per unit volume, and less sensitive, cheaper blasting agents can be used in larger diameters. However, larger-diameter blastholes also result in large burdens and spacings and collar distances; hence, they tend to give coarser fragmentation. Figure 65 (3) illustrates this comparison using 2- and 20-inch-diameter blastholes as an example. Pattern A contains four 20-inch blastholes and pattern B contains 400 2-inch blastholes. In all bench blasting operations, some compromise between these two extremes is chosen. Each pattern represents the same area of excavation, 15,000 sq ft; each involves approximately the same volume of blastholes; and each can be loaded with about the same weight of explosive.

In a given rock formation, the four-hole pattern will give relatively low drilling and blasting costs. Drilling costs for the large blastholes will be low, a low-cost blasting agent will be used, and the cost of detonators will be minimal. However, in a difficult blasting situation, the broken material will be blocky and nonuniform in size, resulting in higher loading, hauling,

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and crushing costs as well as requiring more secondary breakage. Insufficient breakage at the toe may also result.

Blast area - 15,000 sq ft
Borehole diameter - 20 in
Number of holes - 4
Total borehole area - 1,256 sq in
Burden - 50 ft
Spacing - 75 ft

Drilling and blasting costs

Loading, hauling, and crushing costs

Blast area - 15,000 sq ft
Borehole diameter - 2 in
Number of holes - 400
Total borehole area - 1,256 sq in
Burden - 5 ft
Spacing - 7.5 ft
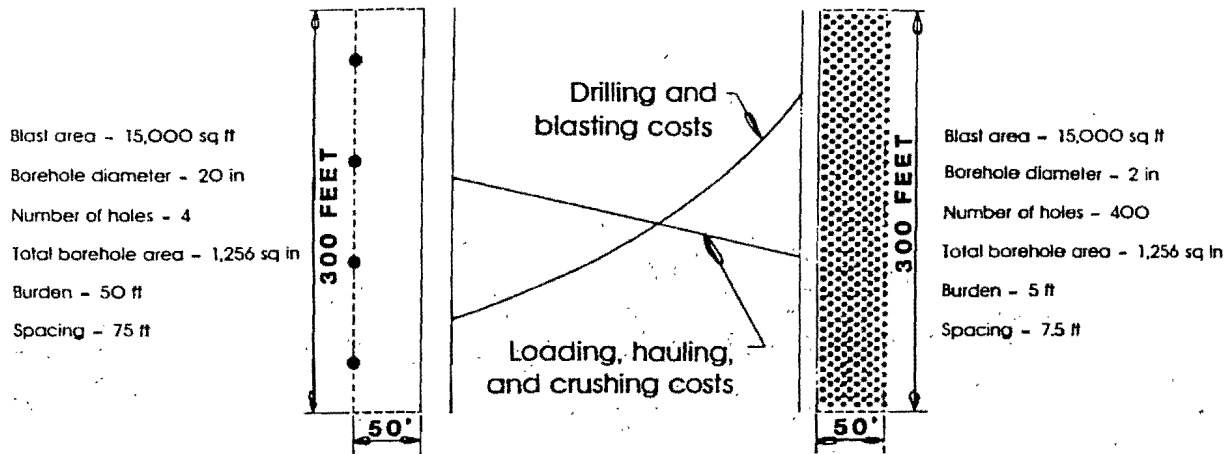
Figure 65: Effect of large and small blastholes on unit costs.

On the other hand, the 400-hole pattern will yield high drilling and blasting costs. Small holes cost more to drill per unit volume, powder for small-diameter blastholes is usually more expensive, and the cost of detonators will be higher. However, the fragmentation will be finer and more uniform, resulting in lower loading, hauling, and crushing costs. Secondary blasting and tow problems will be minimized. Size of equipment, subsequent processing required for the blasted material, and economics will dictate the type of fragmentation needed, and hence the size of blasthole to be used.

Geologic structure is a major factor in determining blasthole diameter. Planes of weakness such as joints and beds, or zones of soft, incompetent rock tend to isolate large blocks of rock in the burden. The larger the blast pattern, the more likely these blocks are to be thrown unbroken into the muck-pile. Note that in the pattern on the left in figure 66, some of the blocks are not penetrated by a blasthole, whereas in the smaller pattern on the right, all of the blocks contain at least one blasthole. Owing to the better explosives distribution, the bottom pattern will give better fragmentation.

As more blasting operations are carried out near populated areas, environmental problems such as airblast and flyrock often occur because of an insufficient collar distance above the explosive charge. As the blasthole diameter increases, the collar distance required to prevent violence increases. The ratio of collar distance to blasthole diameter required to prevent violence varies from 14:1 to 28:1, depending on the relative densities and velocities of the explosive and rock, the physical condition of the rock, the type of stemming used, and the point of initiation. A larger collar distance is required where the sonic velocity of the rock exceeds the detonation velocity of the explosive or where the rock is heavily fractured or low in density. A

Larger holes

Smaller holes
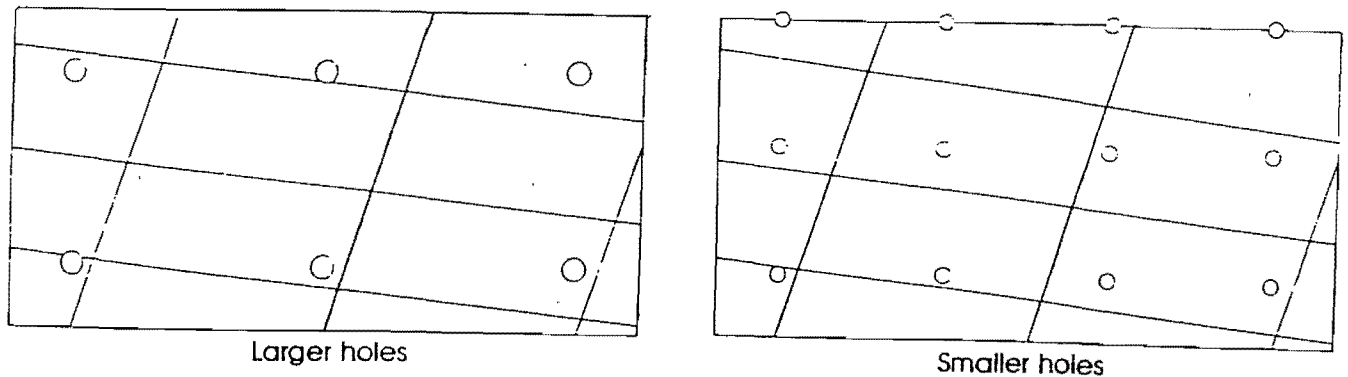
Figure 66: Effect of joining on selection of blasthole size.

top-initiated charge requires a large collar distance than a bottom-initiated charge. As the collar distance increases, the powder distribution becomes poorer, resulting in poorer fragmentation of the rock in the upper part of the bench.

Ground vibrations are controlled by reducing the weight of explosive fired per delay interval. This is more easily done with small blastholes than with large blastholes. In many situations where an operator used large-diameter blastholes near populated areas, several delayed decks must be used within each hole to control vibrations.

Large holes with large blast patterns are ideally suited to an operation with the following characteristics: a large volume of material to be moved; large loading, hauling, and crushing equipment; no requirement for fine, uniform fragmentation; an easily broken toe; few ground vibration or airblast problems (few nearby neighbors); and a relatively homogeneous, easily frag-mented rock without excessive, widely spaced planes of weakness or voids. Many blasting jobs, however, present constraints that require smaller blast-holes.

In the final analysis, the selection of blasthole size is based on econ-omics. It is important to consider the economics of the overall excavation or mining system. Savings realized through indiscrimate cost cutting in the drilling and blasting program may well be lost through increased loading, hauling, and crushing costs and increased litigation costs owing to disgrun-tled neighbors.

Types of Blast Patterns

There are three commonly used drill patterns; square, rectangular, and staggered. The square drill pattern (figure 67) has equal burdens and spacings, while the rectangular pattern has a larger spacing than burden. In both the square and rectangular patterns, the holes of each row are lined up

directly behind the holes in the preceding row. In the staggered pattern
(figure 67), the holes in each row are positioned in the middle of the
spacings of the holes in the preceding row. In the staggered pattern, the
spacing should be larger than the burden.
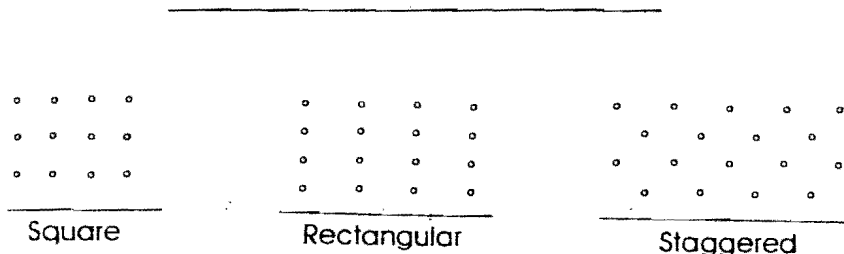
Figure 67: Three basic types of drill pattern.

The staggered drilling pattern is used for row-on-row firing; that is,
where the holes of one row are fired before the holes in the row immediately
behind them, as shown in figure 68. The square or rectangular drilling pat-
terns are used for firing V-cut (figure 69) or echelon rounds. Either side of
the blast round in figure 69, by itself, would be called an echelon blast
round. In V-cut or echelon blast rounds, the burdens and subsequent rock
displacement are at an angle to the original free face. Looking at figure 69,
with the burdens developed at a 45° angle with the original free face, you can
see that the originally square drilling pattern has been transformed to a
staggered blasting pattern with a spacing twice the burden. The simple
patterns discussed here account for the vast majority of the surface blasts
fired.
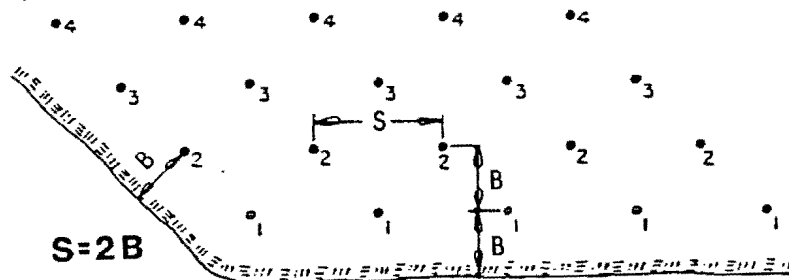
Figure 68: Corner cut staggered blast pattern—simultaneous initiation within rows
(blasthole spacing, S, is twice the burden, B).

Burden

Figure 70 is an isometric view showing the relationship of the various
dimensions of a bench blast. The burden is defined as the distance from a
blasthole to the nearest free face at the instant of detonation. In multiple
row blasts, the burden for a blasthole is not necessarily measured in the dir-
ection of the original free face. One must take into account the free faces
developed by blastholes fired on lower delay periods. As an example, in fig-
ure 68, where one entire row is blasted before the next row begins, the burden

is measured in a perpendicular direction between rows. However, in figure 69, the blast progresses in a V-shape. In this situation, the true burden on most of the holes is measured at an angle of 45° from the original free face, as shown in the figure.
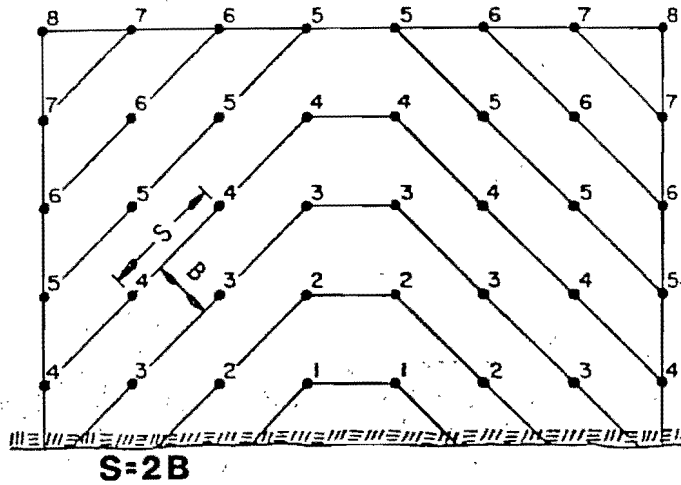
Figure 69: V-echelon blast round (true spacing, S, is twice the true burden, B).

KEY
B   Burden
J   Subdrilling
T   Collar distance
S   Spacing
H   Hole depth
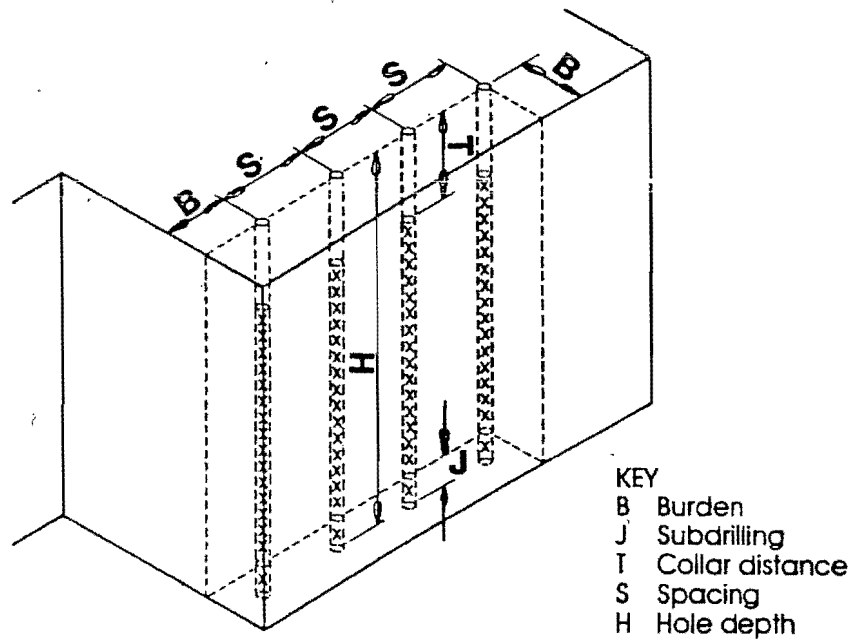
Figure 70: Isometric view of a bench blast.

It is very important that the proper burden be calculated, taking into account the blasthole diameter, the relative density of the rock and the explosive, and to some degree, the length of the blasthole. An insufficient burden will cause excessive airblast and flyrock. Too large a burden will give inadequate fragmentation, toe problems, and excessive ground vibrations. Where it will be necessary to drill a round before the previous round has been excavated, it is important to stake out the first row of the second round before the first round is fired. This will assure a proper burden on the first row of blastholes in the second blast round.

The burden dimension is a function of the charge diameter. For bulk-loaded charges, the charge diameter is equal to the blasthole diameter. For tamped cartridges, the charge diameter will be between the cartridge diameter and the blasthole diameter, depending on the degree of tamping. For untamped cartridges, the charge diameter is equal to the cartridge diameter. When blasting with AN-FO or other low-density blasting agents with densities near 0.85 g/cm^3, in typical rock with a density near 2.7 g/cm^3, the normal burden is approximately 25 times the charge diameter. When using denser products such as slurries or dynamites, with densities near 1.2 g/cm^3, the normal burden is approximately 30 times the charge diameter. It should be stressed again that these are first approximations, and field testing often results in minor adjustments to these values. The burden-to-charge-diameter ratio is seldom less than 20 or seldom more than 40, even in extreme cases. For instance, when blasting with a low-density blasting agent, such as AN-FO, in a dense formation such as iron ore, the desired burden may be about 20 times the charge diameter. When blasting with denser slurries or dynamites in low-density formations, such as some sandstones or marbles, the burden may approach 40 times the charge diameter. Table 4 summarizes these approximations.

Table 4--Approximate B/D Ratios for Bench Blasting

AN-FO (density--0.85 g/cm^3):	
light rock (density--2.2g/cm^3)	28
Average rock (density--2.7 g/cm^3)	25
Dense rock (density--3.2g/cm^3)	23
Slurry, dynamite (density--1.2 g/cm^3)	
light rock (density--2.2g/cm^3)	33
Average rock (density--2.7 g/cm^3)	30
Dense rock (density--3.2g/cm^3)	27

B = Burden   D = Charge diameter

High-speed photographs of blasts have shown that flexing of the burden plays an important role in rock fragmentation. A relatively long, slender burden flexes, and thus breaks more easily than a short, stiffer burden. Figure 71 shows the difference between using a 6-inch blasthole and a 12-1/4-inch blasthole in one 40-foot bench, with a burden-to-charge-diameter ratio of 30 and appropriate subdrilling and stemming dimensions. Note the inherent stiffness of the burden with the 12-1/4-inch blasthole as compared with the 6-inch blasthole. Based on this consideration, lower burden-to-charge-diameter

ratios should be used as a first approximation when the blasthole diameter is large in comparison to the bench height. Care must be taken that the burden ratio is not so small that it would create violence. Once the burden has been determined, it becomes the basis for calculating subdrilling, collar distance (stemming), and spacing.
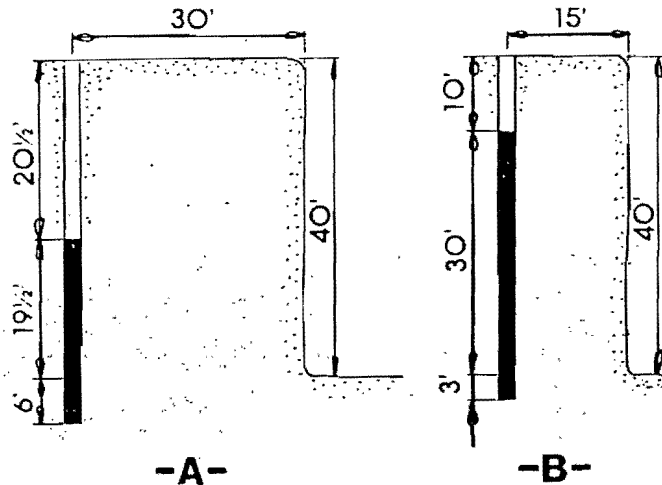
Figure 71: Comparison of a 12¼-inch diameter (A) blasthole (stiff burden) with a 6-inch diameter (B) blasthole (flexible burden) in a 40-foot bench.

Subdrilling

Subdrilling is the distance drilled below the floor level to assure that the full face of rock is removed. Where there is a pronounced parting at a floor level, to which the explosive charge can conveniently break, subdrilling may not be required. In coal stripping, it is common practice to drill down to the coal and then backfill a foot or two before loading explosives, resulting in a negative subdrill. In most surface blasting jobs, however, it is necessary to do some subdrilling to make sure the shot pulls to grade. A good first approximation for subdrilling under average conditions is 30 percent of the burden. Where the toe breaks very easily, the subdrill can sometimes be reduced to 10 to 20 percent of the burden. Even under the most difficult conditions, the subdrill should not exceed 50 percent of the burden. If the toe cannot be pulled with a subdrill-to-burden ratio of 0.5, the fault probably lies in too large a burden.

Priming the explosive column at the toe level gives maximum confinement and normally gives the best breakage. Other factors being equal, toe priming usually requires less subdrilling than collar priming.

Excessive subdrilling is a waste of drilling and blasting expense and may also cause excessive ground vibrations owing to the high degree of confinement of the explosive in the bottom of the blasthole, particularly when the primer is placed in the bottom of the hole. In multiple-bench operations, excessive subdrilling may cause undue fracturing in the upper portion of the bench below, creating difficulties in collaring holes in the lower bench. Insuffi-

cient subdrilling will cause high bottom, resulting in increased wear and tear on equipment and expensive secondary blasting. Table 5 summarizes the recommended subdrilling approximations.

Table 5—Approximate J/B Ratios for Bench Blasting

	Ratio
Open bedding plane at toe	0
Easy toe	0.1-0.2
Normal toe	0.3
Difficult toe	0.4-0.5

B = Burden   J = Subdrilling

Collar Distance (Stemming)

Collar distance is the distance from the top of the explosive charge to the collar of the blasthole. This zone is usually filled with an inert material called stemming to give some confinement to the explosive gases and to reduce airblast. Research has shown that crushed, sized rock works best as stemming but it is common practice to use drill cuttings because of economics. Too small a collar distance results in excessive violence in the form of airblast and flyrock and may cause backbreak. Too large a collar distance creates boulders in the upper part of the bench. The selection of a collar distance is often a tradeoff between fragmentation and the amount of airblast and flyrock that can be tolerated. This is especially true where the upper part of the bench contains rock that is difficult to break. In this situation, the difference between a violent shot and one that fails to fragment the upper zone properly may be a matter of only a few feet of stemming. Collar priming of blastholes normally causes more violence than center or toe priming, and requires the use of a longer collar distance.

Field experience has shown that a collar distance equal to 70 percent of the burden is a good first approximation, except where collar priming is used. Careful observation of airblast, flyrock, and fragmentation will enable the blaster to further refine this dimension. Where adequate fragmentation in the collar zone cannot be attained while still controlling airblast and flyrock, deck charges or satellite holes may be required (figure 63).

A deck charge is an explosive charge near the top of the blasthole, separated from the main charge by inert stemming. If boulders are being created in the collar zone but the operator fears that less stemming would cause violence, the main charge should be reduced slightly and a deck charge added. The deck charge is usually shot on the same delay as the main charge or one delay later. Care must be exercised not to place the deck charge too near the top of the blasthole, or excessive flyrock may result. As an alternative, short satellite holes between the main blastholes can be used. These satellite holes are usually smaller in diameter than the main blastholes and are loaded with a light charge of explosives.

From the standpoint of public relations, collar distance is a very important blast design variable. One violent blast can permanently alienate neighbors. In a delicate situation, it may be best to start with a collar distance equal to the burden and gradually reduce this if conditions permit. Collar distances greater than the burden are seldom necessary.

Spacing

Spacing is defined as the distance between adjacent blastholes, measured perpendicularly to the burden. Where the rows are blasted one after the other as in figure 68, the spacing is measured between holes in a row. However, in figure 69, where the blast progresses on an angle to the original free face, the spacing is measured at an angle from the original free face.

Spacing is calculated as a function of the burden and also depends on the timing between holes. Too close a spacing causes crushing and cratering between holes, boulders in the burden, and toe problems. Too wide a spacing causes inadequate fracturing between holes, accompanied by humps on the face and toe problems between holes (figure 72).
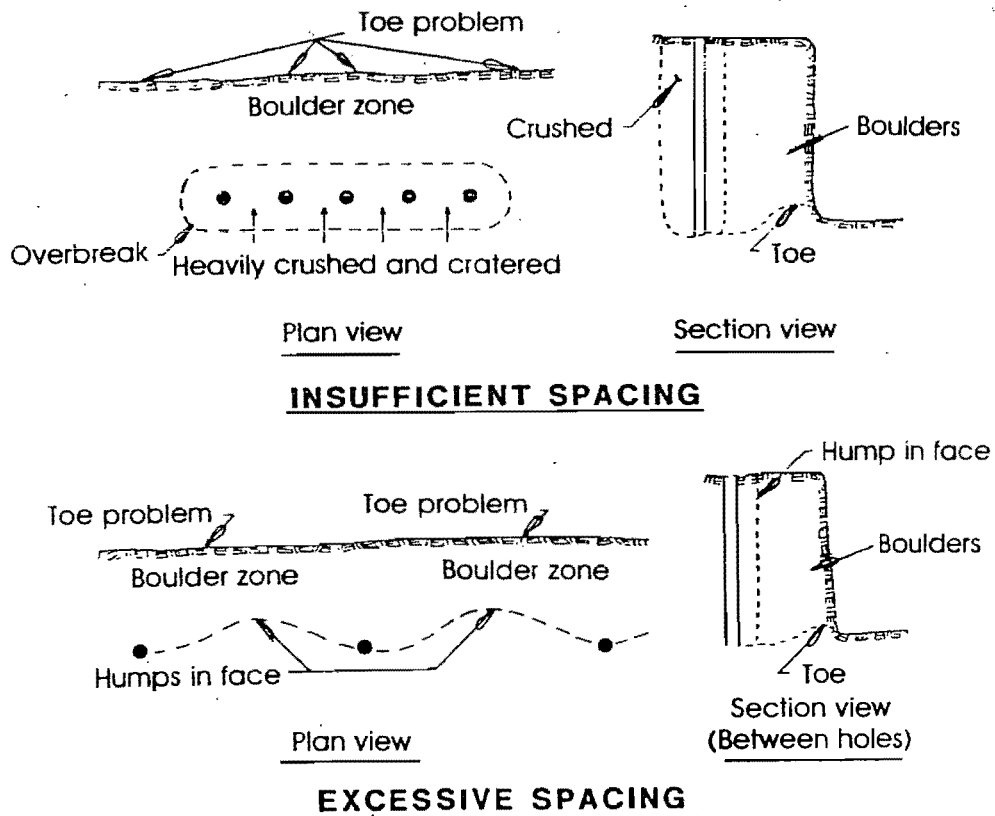
Figure 72: Effects of insufficient and excessive spacing.

When the holes in a row are initiated on the same delay period, a spacing equal to twice the burden will usually pull the round satisfactorily. Actually, the V-cut round in figure 69 also illustrates simultaneous initiation within a row, with the rows being the angled lines of holes fired on the same delay. The true spacing is twice the true burden even though the holes were originally drilled on a square pattern.

Field experience has shown that the use of millisecond delays between holes in a row results in better fragmentation and also reduces the ground vibrations produced by the blast. When millisecond delays are used between holes in a row, the spacing-to-burden ratio must be reduced to somewhere between 1.2 and 1.8 with 1.5 being a good first approximation. Various delay patterns may be used within the rows, including alternate delays (figure 73) and progressive delays (figure 74). Generally, large-diameter blastholes require lower spacing-to-burden ratios (usually 1.2 to 1.5 with millisecond delays) than small-diameter blastholes (usually 1.5 to 1.8). Because of the complexities of geology, the interaction of delays, differences in explosive and rock strengths, and other variables, the proper spacing-to-burden ratio must be determined through onsite experimentation, using the preceding values as first approximations.

Except when using controlled blasting techniques such as smooth blasting and cushion blasting, which will be described later in this chapter, the spacing should never be less than the burden.
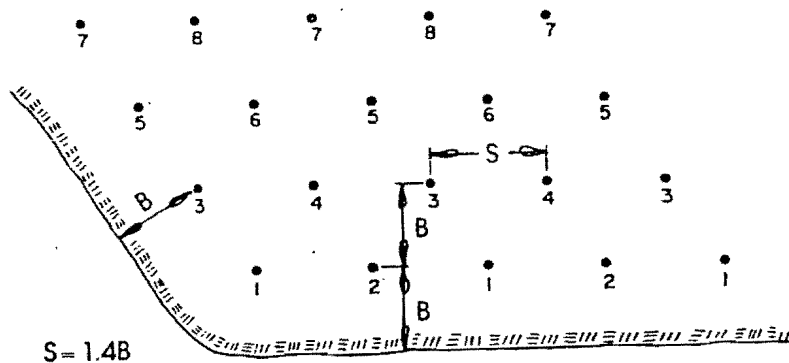
Figure 73: Staggered blast pattern with alternate delays (spacing, S, is 1.4 times the burden, B).

Hole Depth

In any blast design, it is important that the burden and the blasthole depth (or bench height) be reasonably compatible. As a rule of thumb for bench blasting, the hole depth-to-burden ratio should be between 1.5 and 4.0. Hole depths less than 1.5 times the burden cause excessive airblast and fly-rock and, because of the short, thick shape of the burden, give coarse, uneven
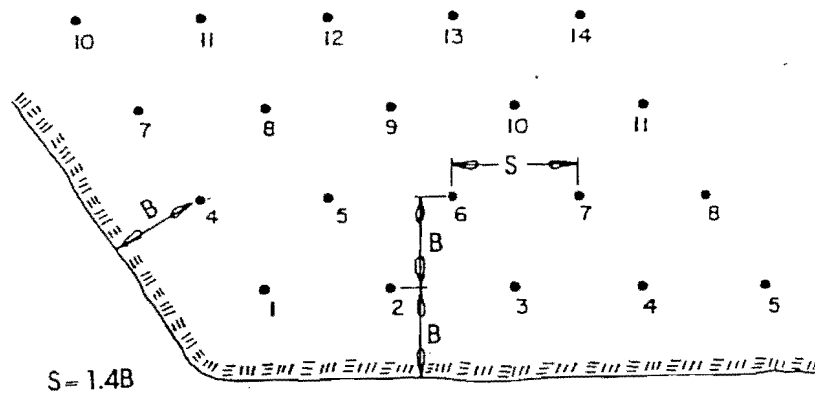
Figure 74: Staggered blast pattern with progressive delays (spacing, S, is 1.4 times the burden, B).

fragmentation. Where operational conditions require a ratio of less than 1.5, the primer should be placed at the toe of the bench to assure maximum confinement. Keep in mind that placing the primer in the subdrill can cause increased ground vibrations. If an operator continually finds use of a hole depth-to-burden ratio of less than 1.5 necessary, consideration should be given to increasing the bench height or using a smaller drill.

Hole depths greater than four times the burden are also undesirable. The longer a hole is, in respect to its diameter, the more error there will be in its location at toe level, which is the most critical portion of the blast. A poorly controlled blast will result. Extremely long, slender holes have even been known to intersect.

High benches with short burdens also create hazards, such as a small drill having to pull in the front row of holes near the edge of a high ledge or a small shovel having to dig at the toe of a precariously high face. The obvious solution to this problem is to use a lower bench height. There is no real advantage to a high bench height. Lower benches give more efficient blasting results, lower drilling cost and chances for cutoffs, and are safer from an equipment operation standpoint. If it is impractical to reduce the bench height, larger drilling and rock handling equipment should be used, which will effectively reduce the blasthole depth-to-burden ratio.

A major problem with long slender charges is the greater potential for cutoffs in the explosive column. Where it is necessary to use blast designs with large hole depth-to-burden ratios, multiple priming should be used as insurance against cutoffs.

Delays

Millisecond delays are used between charges in a blast round for three reasons:

1. To assure that a proper free face is developed to enable the explosive charge to efficiently fragment and displace its burden.

2.   To enhance fragmentation between adjacent holes.
3.   To reduce the ground vibrations created by the blast.

There are numerous possible delay patterns, several of which were covered in figure 68, 69, 73, and 74.

Andrews (1), of DuPont, conducted numerous field investigations to determine optimum delay intervals for bench blasting and reached the following conclusions:

1.   The delay time between holes in a row should be between 1 and 5 milliseconds per foot of burden. Delay times less than 1 millisecond per foot of burden cause premature shearing between holes, resulting in coarse fragmentation. If an excessive delay time is used between holes, rock movement from the first hole prevents the adjacent hole from creating additional fractures between the two holes. A delay of 3 milliseconds per foot of burden gives good results in many kinds of rock.

2.   The delay time between rows should be two to three times the delay time between holes in a row. This is longer than most previous recommendations. However, in order to obtain good fragmentation and control flyrock, a sufficient delay is needed so that the burden from previously fired holes has enough time to move forward to accommodate broken rock from subsequent rows. If the delay between rows is too short, movement in the back rows will be upward rather than outward (figure 75).

3.   Where airblast is a problem, the delay between holes in a row should be at least 2 milliseconds per foot of spacing. This will prevent airblast from one charge from adding to that of subsequent charges as the blast proceeds down the row.

4.   For the purpose of controlling ground vibrations, most regulatory authorities consider two charges to be separate events if they are separated by a delay of 8 milliseconds or more.

Following these recommendations should yield good blasting results. However, when using surface delay systems such as detonating cord connectors and

INADEQUATE DELAYS   ADEQUATE DELAYS
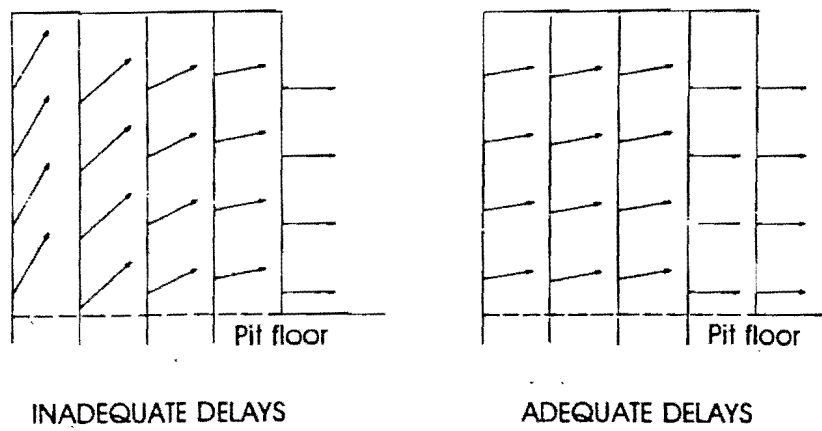
Figure 75: The effect of inadequate delays between rows.

sequential timing blasting machines, the chances for cutoffs will be increased. To solve this problem, in-hole delays should be used in addition to the surface delays. For example, when using surface detonating cord connectors, one might use a 100-ms delay in each hole. This causes ignition of the in-hole delays well in advance of rock movement, thus minimizing cutoffs. With a sequential timer, the same effect can be accomplished by avoiding the use of electric caps with delays shorter than 75 to 100 milliseconds.

From the standpoint of simplicity in blast design, it is best if all the explosive in a blasthole is fired as a single column charge. However, it is sometimes necessary, where firing large blastholes in populated areas, to use two or more delayed decks within a blasthole to reduce ground vibrations. Blast rounds of this type can become quite complex, and should be designed under the guidance of a competent person.

All currently used delay detonators employ pyrotechnic delay elements. That is, they depend on a burning powder train for their delay. Although these delays are reasonably accurate, overlaps have been known to occur (9). Therefore, when it is essential that one charge fires before an adjacent charge, such as in a tight corner of a blast, it is a good idea to skip a delay period. Development of blasting caps with electronic delays is a good future possibility.

Powder Factor

Powder factor, in the opinion of the authors, is not the best tool for designing blasts.

Blast designs should be based on the dimensions discussed earlier in this chapter. However, powder factor is a necessary calculation for cost accounting purposes. In blasting operations, such as coal stripping or construction work, where the excavated material has little or no inherent value, powder factor is usually expressed in terms of pounds of explosive per cubic yard of material broken. Powder factors for surface blasting can vary from 0.25 to 2.5 lb/yd^3, with 0.5 to 1.0 lb/yd^3 being most typical.

Powder factor for a single blasthole is calculated by the following formula:

$$P.F. = \frac{L(0.3405d)(D^2)}{(B)(S)(H)/(27)}$$

where P.F. = powder factor, pounds of explosive per cubic yard of rock,
     L = length of the explosive charge, feet,
     d = density of the explosive, grams per cubic centimeter,
     D = charge diameter, inches,
     B = burden dimension, feet,
     S = spacing dimension, feet,
and   H = bench height, feet.

Many explosives companies publish tables that give loading densities in pounds per foot of blasthole for different combinations of d and D. The nomograph in figure 14 also calculates the density in pounds per foot of borehole.

Powder factor is a function of type of explosive, rock density, and geology. Table 6 gives typical powder factors for surface blasting.

Table 6--Typical Powder Factors for Surface Blasting

Degree of difficulty in rock breakage	Powder Factor lb/cu yd
Low	0.25-0.40
Medium	0.40-0.75
High	0.75-1.25
Very high	1.25-2.50

Higher energy explosives, such as those containing large amounts of aluminum, can break more rock per pound than lower energy explosives. However, most of the commonly used explosive products have fairly similar energy values and thus have similar rock-breaking capabilities. Soft, light rock requires less explosive per yard than hard, dense rock. Large-hole patterns require less explosive per yard of rock blasted because a larger proportion of stemming is used. Of course, larger blastholes frequently result in coarser fragmentation because of poorer powder distribution. Massive rock with few existing cracks or planes of weakness requires a higher powder factor than a formation that has numerous, closely spaced geologic flaws. Finally, the more free faces a blast has to break to, the lower will be the powder factor. For example, a corner cut, with two vertical free faces, will require less powder than a box cut with only one vertical free face; and a box cut will require less powder than a sinking cut, which has only the ground surface as a free face. In a sinking cut, it is desirable, where possible, to open a second free face by using a V-cut somewhere near the center of the round. V-cuts are discussed in more detail in the "Underground Blasting" section of this chapter.

When blasting materials that have an inherent value per ton, such as limestone or metallic ores, powder factors are sometimes expressed as pounds of explosive per ton of rock or tons of rock per pound of explosive.

Secondary Blasting

Some primary blasts, no matter how well designed, will leave boulders that are too large to be handled efficiently by the loading equipment or large enough to cause plugups in crushers or preparation plants. Secondary fragmentation techniques must be used to break these boulders.

In the case of boulders too large to be handled, the loader operator will set the boulders aside for treatment. Identifying material large enough to cause plugups is not always quite so apparent. The operator must be instructed to watch for material that is small enough for convenient loading but which is large enough to cause a bottleneck later in the processing cycle.

Secondary fragmentation can be accomplished in four ways:

1. A heavy ball suspended from a crane may be dropped repeatedly on the boulder until the boulder breaks. This is a relatively inefficient method, and breaking a large or tough (nonbrittle) rock may take a considerable period of time. This method is adequate where the number of boulders produced is not excessive.

2. A hole may be drilled into the boulder and a wedging device inserted to split the boulder. This is also a slow method but may be satisfactory where only a limited amount of secondary fragmentation is necessary. An advantage of this method is that it does not create the flyrock associated with explosive techniques or, to some degree, with drop balls.

3. Loose explosive may be packed into a crack or depression in the boulder, covered with damp earthen material, and fired. This type of charge is called a mudcap, plaster, or adobe charge. This method is inefficient because of a lack of explosive confinement, and relatively large amounts of explosive are required. The result is considerable noise and flyrock, and, often, an inadequately broken boulder. The system is hazardous because the primed charge, lying on the surface, is prone to accidental initiation by external impacts from falling rocks or equipment. External charges should be used to break boulders only where drilling a hole is impractical, and when used, extreme caution concerning noise, flyrock, and accidental initiation through impact must be exercised. If it is found necessary to shoot a multiple mudcap blast, long delays or cap and fuse are not recommended.

4. The most efficient method of secondary fragmentation is through the use of small (1- to 3-inch) boreholes loaded with explosives. The borehole is normally collared at the most convenient location, such as a crack or a depression in the rock, and is directed toward the center of mass of the rock. The hole is drilled two-thirds to three-fourths of the way through the rock. Because the powder charge is surrounded by free faces, less explosive is required to break a given amount of rock than in primary blasting. One-quarter pound per cubic yard will usually do the job. Careful location of the charge is more important than its precise size. When in doubt, it is best to estimate on the low side and underload the boulder. With large boulders, it is best to drill several holes to distribute the explosive charge, rather than to place the entire charge in a single hole. All secondary blastholes should be stemmed. As a cautionary note, secondary blasts are usually more violent than primary blasts.

Any type of initiation system may be used to initiate a secondary blast. For connecting large numbers of boulders, where noise is not a problem, detonating cord is often used. The "Detonating Cord Initiation" section in chapter 2 describes precautions to be taken against cord cutoffs. Electric blasting is also frequently used.

Although secondary blasting employs relatively small charges, its potential hazards must not be underestimated. Flyrock is often more severe and more difficult to predict than with primary blasting. Secondary blasts require at least as much care in guarding as do primary blasts. Secondary blasting can truly be called an art, with experience being an important key to success.

UNDERGROUND BLASTING

Underground blast rounds can be divided into two basic categories: (a) heading, drift, or tunnel rounds, in which the only free face is the surface from which the holes are drilled, and (b) bench or stope rounds in which there are one or more free faces in addition to the face on which the blastholes are drilled. Blasts falling under the second category are designed in the same way as surface blast rounds. This discussion will cover blasts falling under the first category of only one initial free face.

Opening Cuts

The initial and most critical part of a heading round is the opening cut. The essential function of this cut is to provide additional free faces to which the rock can be broken. The DuPont Blaster's Handbook (4) discusses opening cuts. Although there are many specific types of opening cuts, and the terminology can be quite confusing, all opening cuts fall into one of two classifications: angled cuts and parallel hole cuts (figure 76).

An angled cut, which may be referred to as a V-cut, draw cut, slab cut, or pyramid cut, breaks out a wedge of rock to create an opening to which the remaining holes can displace their burdens. Angled cuts are difficult to drill accurately. The bottoms of each pair of cut holes should be as close as possible. If they cross, the depth of round pulled will be less than designed. If bottoms are more than a foot or so apart, the round may not pull to its proper depth. The angle between the cut holes should be 60 degrees or more, to minimize bootlegging. Some mines that drill a standard angled cut supply their drillers with a template to assure proper spacing and angles of the angled holes. The selection of the specific type of angled cut is a function of the rock, the type of drilling equipment, the philosophy of mine management, and the individual driller. A considerable amount of trial and error is usually involved in determining the best angled cut for a specific mine. In small openings, it is often impossible to position the drill properly to drill an angled cut. In this case a parallel hole cut must be used.

Parallel hole cuts, which may also be called Michigan cuts, Cornish cuts, shatter cuts, burn cuts, or Coromant cuts, are a series of closely spaced holes, some loaded and some not loaded (figure 77) which, when fired, pulverize and eject a cylinder of rock to create an opening to which the burdens on the remaining holes can be broken. Because they require higher powder factors and more drilling per volume of rock blasted, the use of parallel hole cuts is usually restricted to narrow headings, where there is not enough room to drill an angled cut.

Parallel hole cuts involve more drilling than angled cuts because the closely spaced holes break relatively small volumes of rock. However, they are relatively easy to drill because the holes are parallel. Like angled cuts, accurately drilled parallel hole cuts are essential if the blast round is to pull properly. Some drill jumbos have automatic controls to assure that holes are drilled parallel. Units of this type are a good investment for mines that routinely drill parallel hole cuts. A template may also be used in drilling a parallel hole cut (figure 78).
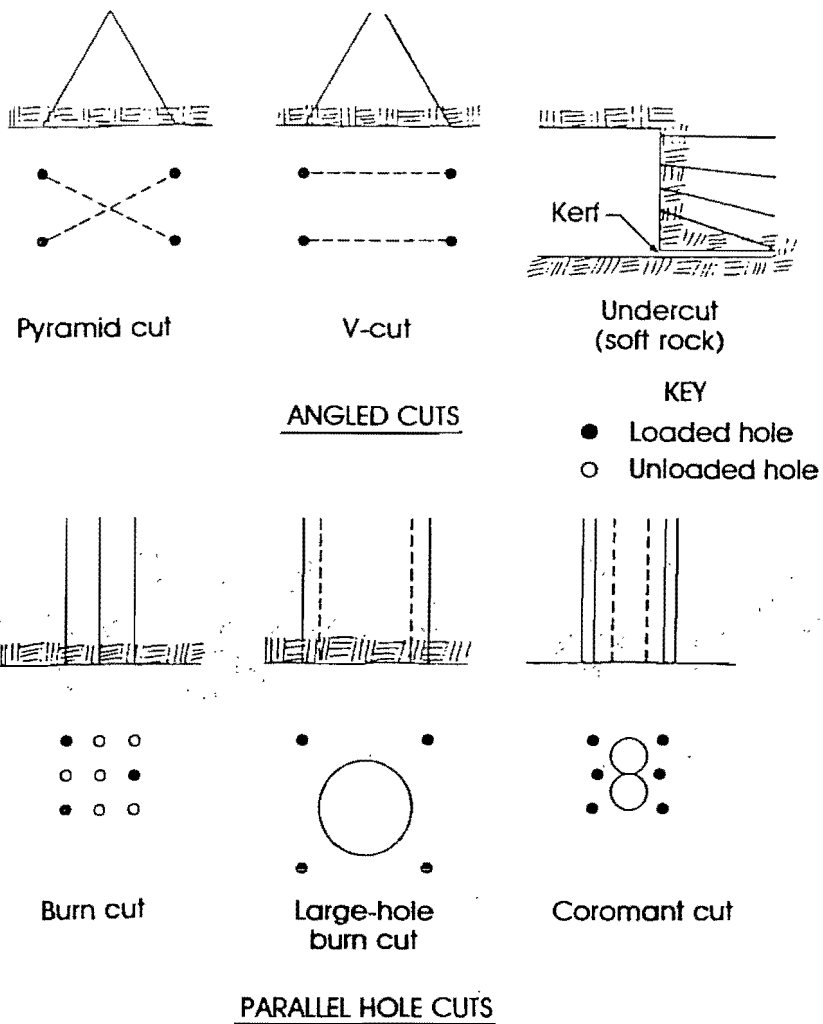
Figure 76: Types of opening cuts.

The selection of the type of parallel hole cut depends on the rock, the type of drilling equipment, the philosophy of mine management, and the individual driller. As with angled cuts, trial and error is usually involved in determining the best parallel hole cut for a specific mine.

For all types of opening cuts, it is important that the cut pulls to its planned depth, because the remainder of the round will not pull more deeply than the cut. In blasting with burn cuts, care must be exercised not to overload the burn holes; this could cause the cut to "freeze" or not pull properly. Proper loading of the cut depends on the design of the cut and the type of rock being blasted, and often must be determined by trial and error.

Some research has been done involving burn cuts with one or more large central holes (8), and a few mines have adopted this practice. The advantage of the large central hole is that it gives a dependable void to which succeeding holes can break, which is not always obtained with standard burn cuts.

KEY ——— ● Loaded hole   o Unloaded hole
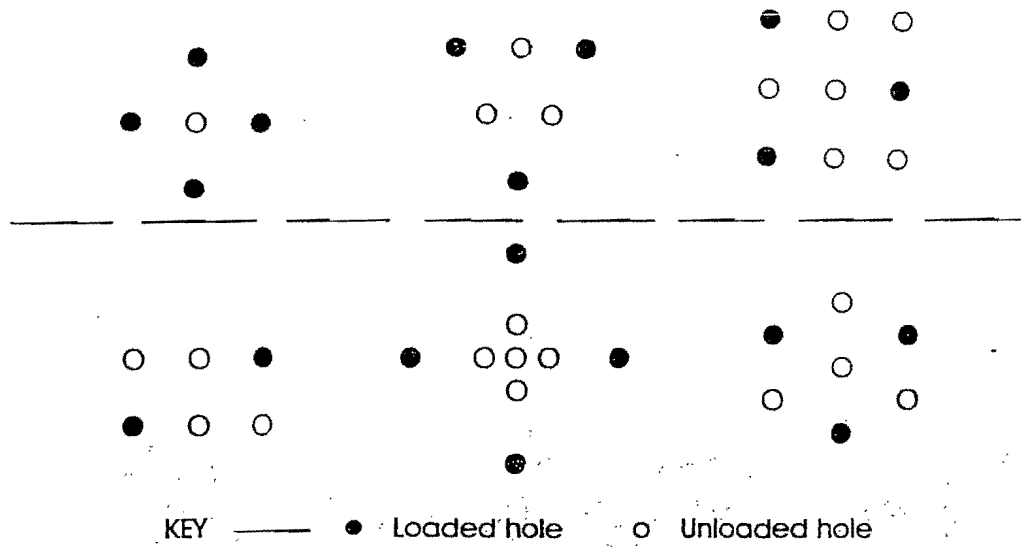
Figure 77: Six designs for parallel hole cuts.
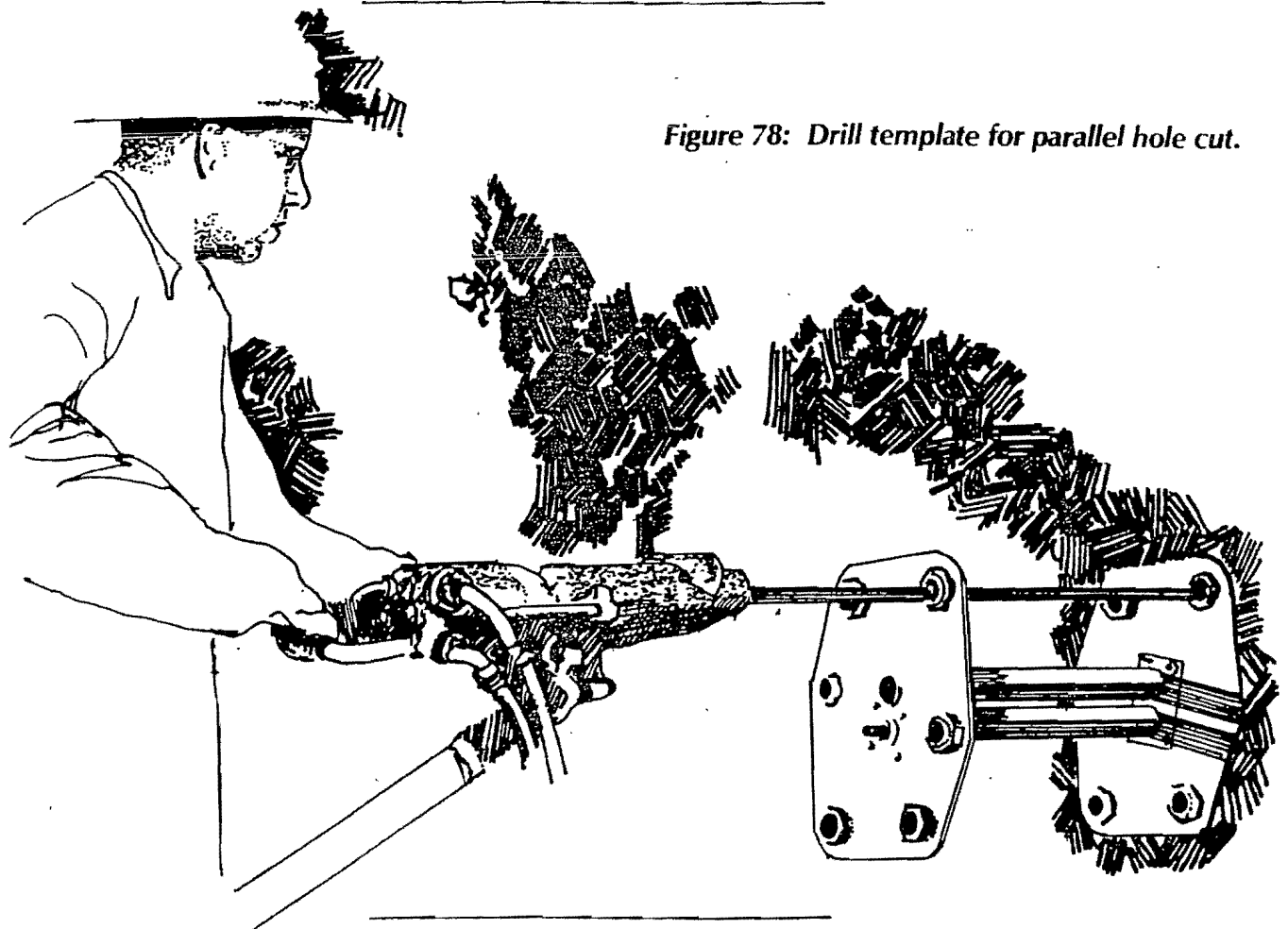
Figure 78: Drill template for parallel hole cut.

This assures a more dependable and deeper pull of the blast round. The disadvantages of the large central hole are the requirement for an extra piece of equipment to drill the large hole and the extra time involved. Sometimes a compromise is used where intermediate-sized holes, such as 4- or 5-inch diameter, are drilled using the same equipment used to drill the standard blastholes.

In some soft materials, particularly coal, the blasted cut is replaced by a sawed kerf, usually at floor level (figure 79). In addition to giving the material a dependable void to which to break, the sawed cut assures that the floor of the opening will be smooth.

Front view   Side view

UNDERCUT

Front view   Side view

OVERCUT

KEY

● Blasthole delay period
2
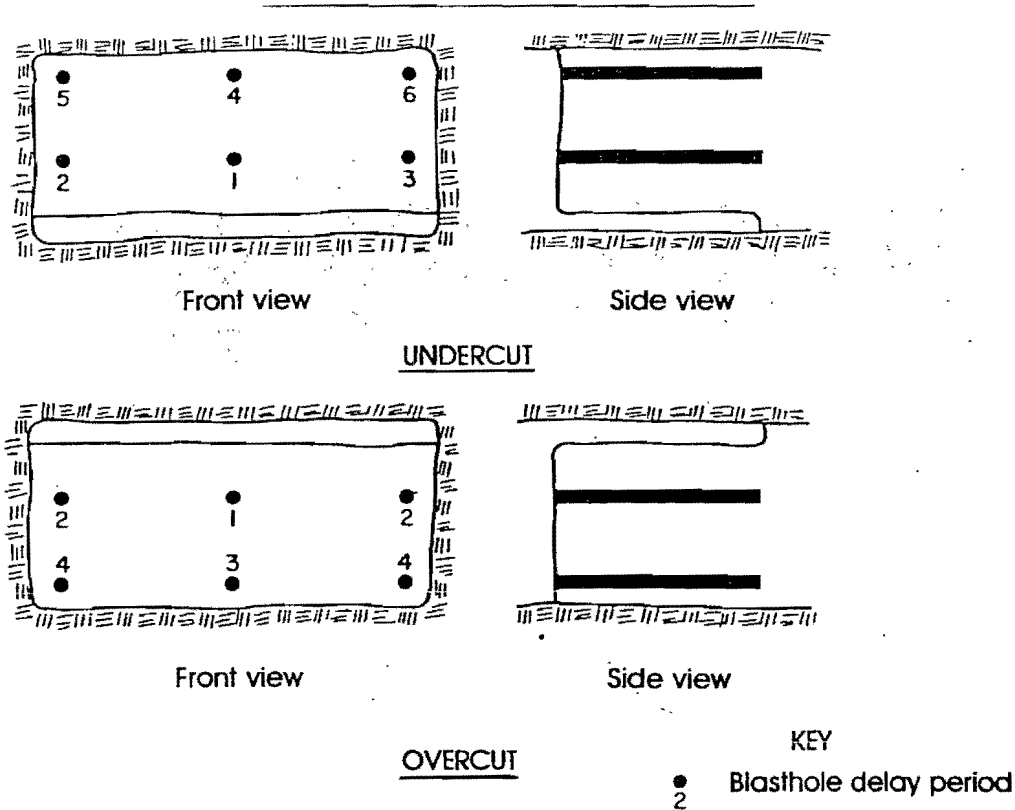
Figure 79: Blast round for soft material using a sawed kerf.

Blasting Rounds

Once the opening cut has established the necessary free face, the remainder of the blastholes must be positioned so that they successively break their burdens into the void space. It is important to visualize the progression of the blast round so that each hole, at its time of initiation, has a proper free face parallel or nearly parallel to it. Figure 80 gives the typical nomenclature for blastholes in a heading round.

The holes fired immediately after the cut holes are called the relievers. The burdens on these holes must be carefully planned. If the burdens are too small, the charges will not pull their share of the round. If the burdens are too large, the round may freeze because of insufficient space for the rock to

120
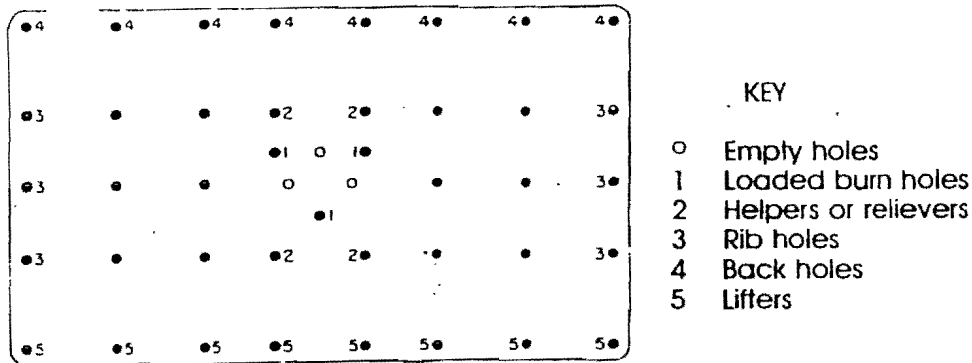
Figure 80: Nomenclature for blastholes in a heading round.

KEY

O	Empty holes
1	Loaded burn holes
2	Helpers or relievers
3	Rib holes
4	Back holes
5	Lifters

expand into. After several relievers have been fired, the opening is usually large enough to permit the design of the remainder of the blasting in accordance with the principles discussed under the "Surface Blasting" section. In large heading rounds, the burden and spacing ratios are usually slightly less than those for surface blasts. In small headings, where space is limited, the burden and spacing ratios will be smaller still. This is another area where trial and error plays a part in blast design.

The last holes to be fired in an underground round are the back holes at the top, the rib holes at the sides, and the lifters at the bottom of the heading. Unless a controlled blasting technique is used (discussed later in this chapter), the spacing between these perimeter holes is about 20 to 25 blasthole diameters. Figure 81 shows two typical angled-cut blast rounds. After the initial wedge of rock is extracted by the cut, the angles of the subsequent blastholes are progressively reduced until the perimeter holes are parallel to the heading or looking slightly outward. In designing burden and spacing dimensions for angled cut blast rounds, the location of the bottom of the hole, rather than the collar, is considered.

Figure 82 shows two typical parallel-hole-cut blast rounds. These rounds are much simpler to drill than angled-cut rounds. Once the central opening has been established, the round resembles a bench round turned on its side. Figure 83 shows a comparison of typical muckpiles obtained from V-cut and burn-cut blast rounds. Burn cuts give more uniform fragmentation and a more compact muckpile than V-cuts, where the muckpile is more spread out and variable in fragmentation. Powder factors and the amount of drilling required are higher for burn cuts.

Delays

Two series of delays are available for underground blasting: (a) millisecond delays, which are the same as those used in surface blasting, and (b) slow or tunnel delays. The choice of delay depends on the size of the heading being blasted and on the fragmentation and type of muckpile desired. Slow delays give coarser fragmentation and usually give a more compact muckpile, whereas millisecond delays give finer fragmentation and a more spread

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Top view   Top view

Front view   Front view

V-CUT ROUND   SLABBING ROUND
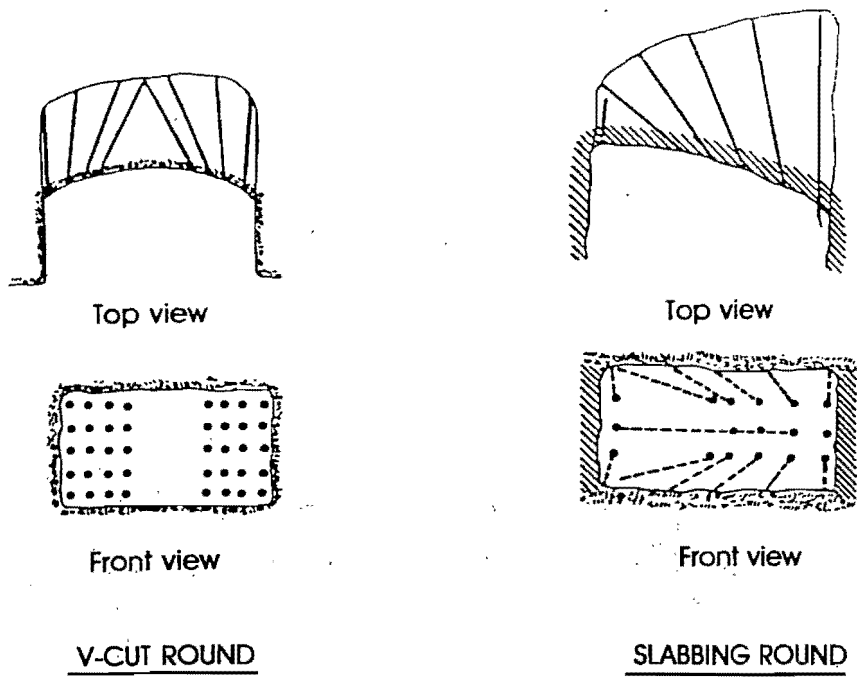
Figure 81: Angled cut blast rounds.

Top view   Top view

Front view   Front view

REGULAR   LARGE-HOLE
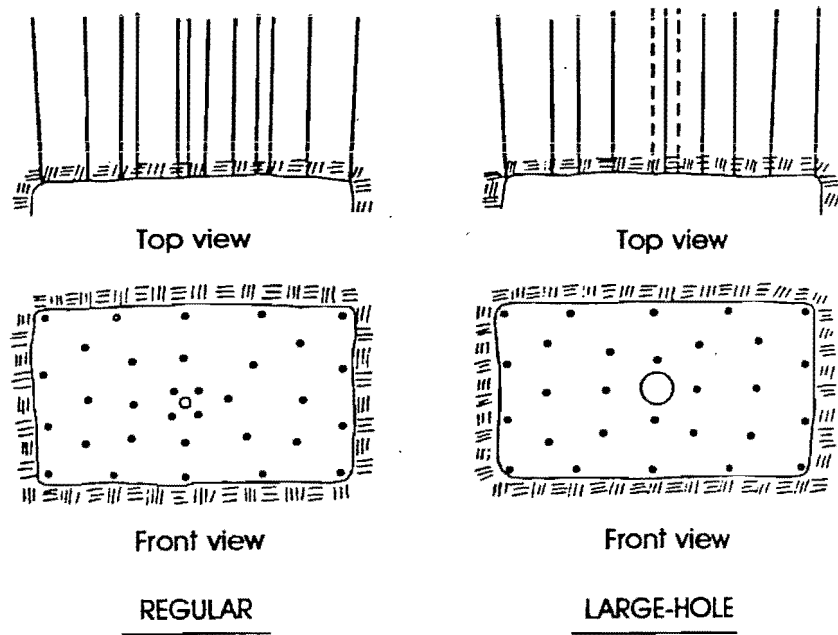
Figure 82: Parallel hole cut blast rounds.
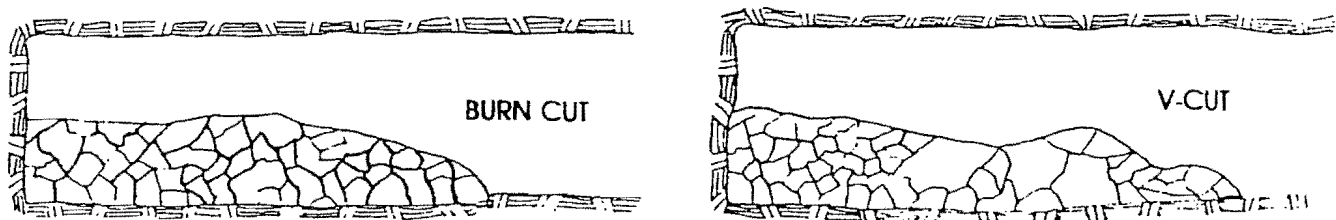
Figure 83: Fragmentation and shape of muckpile as a function of type of cut.

out muckpile (figure 84). In small headings where space is limited, particularly when using parallel hole cut rounds, slow delays are necessary to assure that the rock from each blasthole has time to be ejected before the next hole fires. Where a compromise between the results of millisecond delays and slow delays is desired, some operators use millisecond delays and skip delay periods.
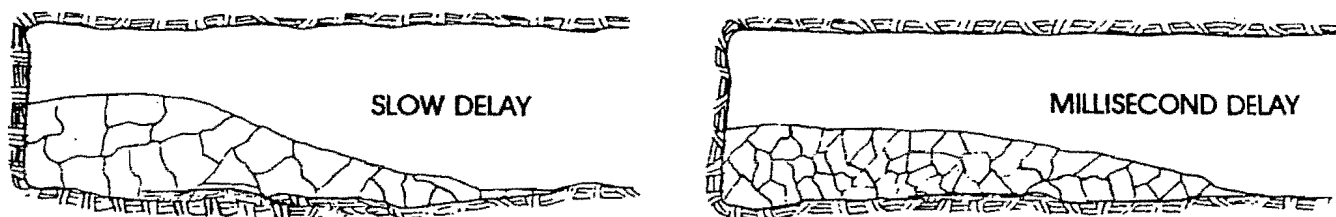
Figure 84: Fragmentation and shape of muckpile as a function of delay

In an underground blast round, it is essential that the delay pattern be designed so that each hole, at its time of firing, has a good free face to which it can displace its burden. Figure 85 shows a typical delay pattern for a burn cut blast round in a heading in hard rock. Figure 86 shows a delay pattern for a V-cut blast round.

The shape of the muckpile is affected by the order in which the delays are fired (figure 87). If the blast is designed so that the back holes at the roof are fired last, a cascading effect is obtained, resulting in a compact muckpile. If the lifters are fired last, the muckpile will be displaced away from the face.

Powder Factor

As with surface blasting, factors for underground blasting vary depending on several factors. Powder factors for underground blasting may vary from 1.5 to 12 lb/yd^3. Soft, light rock headings with large cross sections, large blastholes, and angle-cut rounds all tend to give lower powder factors than hard, dense rock, small headings, small blastholes, and parallel hole cuts.

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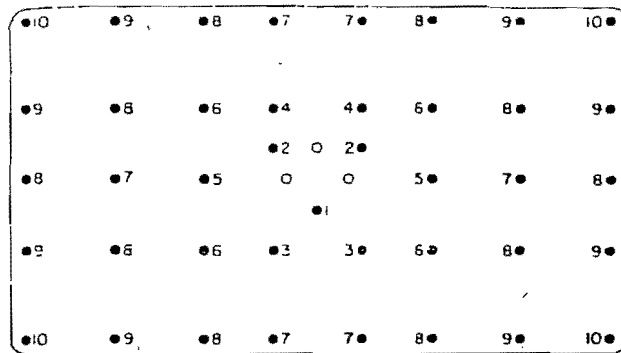
KEY

●9 Loaded hole with delay period

○ Unloaded hole

Figure 85: Typical burn cut blast round delay pattern.
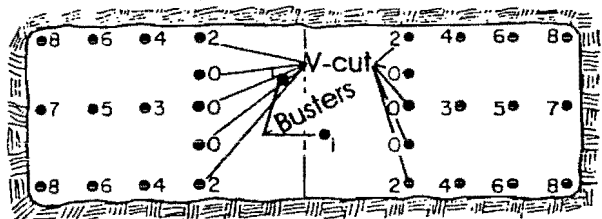
Front view

Top view

KEY

●4 Loaded hole with delay period
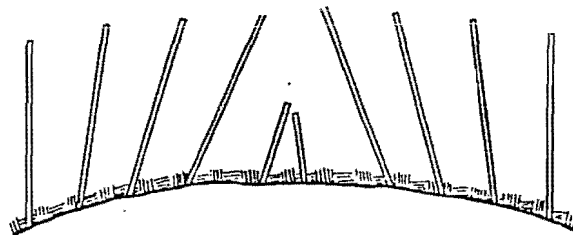
Figure 86: Typical V-cut blast round delay pattern.

UNDERGROUND COAL MINE BLASTING

Underground coal mine blasting is different from most rock blasting in two important respects. Operations take place in a potentially explosive atmosphere containing methane and coal dust, and the coal is much easier to break than rock. The loading and firing methods, as well as the explosive type, must be permissible, as specified by the Mine Safety and Health Admin-

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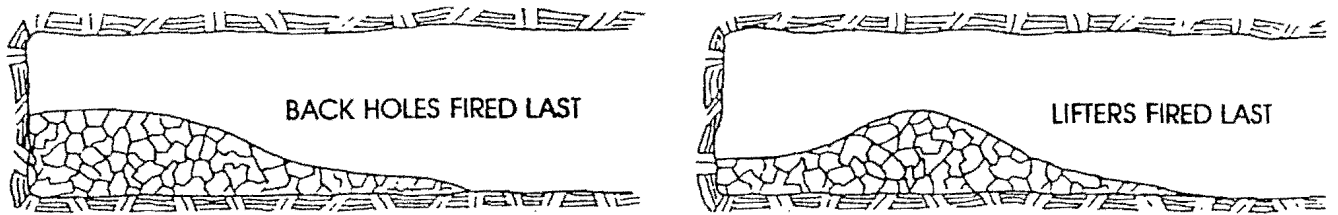
Figure 87: Shape of muckpile as a function of order of firing.

istration (MSHA). In addition, underground coal mine blasting is closely regulated by state regulatory agencies. This discussion is intended to point out some of the main differences between coal blasting and rock blasting and should not be considered as a guide to regulatory compliance. People involved in underground coal mine blasting need to become thoroughly familiar with the MSHA regulations dealing with permissible blasting, which are identified in Appendix D, and those of the State in which they blast. Hercules (6) has published a shotfirer's guide for underground coal mine blasting.

Black powder or other nonpermissible explosives, including detonating cord, may not be stored or used in underground coal mines. Unconfined shots—that is, those not contained by boreholes—may not be fired, although a permissible, external charge is currently under development. In most states, the coal must be undercut (figure 79) before blasting. The boreholes should not be deeper than the cut to assure that the coal is not fired off the solid. The minimum depth of cut should be 3-1/2 feet.

Charge weights should not exceed 3 pounds per borehole. Boreholes should have a minimum 18-inch burden in all directions. If this specification cannot be met, the charge weight should be reduced to prevent underburdened shots. Blast rounds should be limited to 20 holes. All holes should be bottom primed with the cap at the back of the hole, although this is not always required by regulation. Aluminum-cased detonators should not be used and leg wires should not be more than 16 feet long, or of equivalent resistance. Permissible blasting machines are designed to provide sufficient energy to a circuit using the rated number of electric blasting caps with 16-foot iron leg wires. Should these machines be used with copper wire detonators, their apparent capacity is increased. Zero-delay detonators should not be used in a circuit with millisecond-delay detonators.

Permissible explosives must remain in the original cartridge wrapper throughout storage and use, without admixture with other substances. Cartridges must be loaded in a continuous train, in contact with each other, and should not be deliberately crushed, deformed, or rolled. Permissible explosives must conform with original specifications, within limits of tolerance prescribed by MSHA. The cartridge must be of a diameter which has been approved. All blastholes must be stemmed with incombustible material. Holes deeper than 4 feet should contain at least 24 inches of stemming and holes less than 4 feet deep should be stemmed for at least half their length. Water stemming bags, when used, should be at least 15 inches long and should have a diameter within 1/4 inch of the borehole diameter. Shots must be fired with a permissible blasting unit of adequate capacity.

"Controlled blasting" describes several techniques for improving the competence of the rock at the perimeter of an excavation. Du Pont, among other companies, has published an excellent pamphlet describing and giving general specifications for the four primary methods of controlled blasting (5). Much of this discussion is adapted from that publication. The recommended dimensions have been determined through years of on-the-job testing and evaluation. These recommended dimensions are given as ranges of values. The best value for a given blasting job is a function of the geology, specifically the number and severity of planes of weakness in the rock, and the quality of rock surface that is required. Normal blasting activities propagate cracks into the excavation walls. These cracks reduce the stability of the opening. The purpose of controlled blasting is to reduce this perimeter caretaking and thus increase the stability of the opening. Figure 88 shows a stable slope produced by controlled blasting.
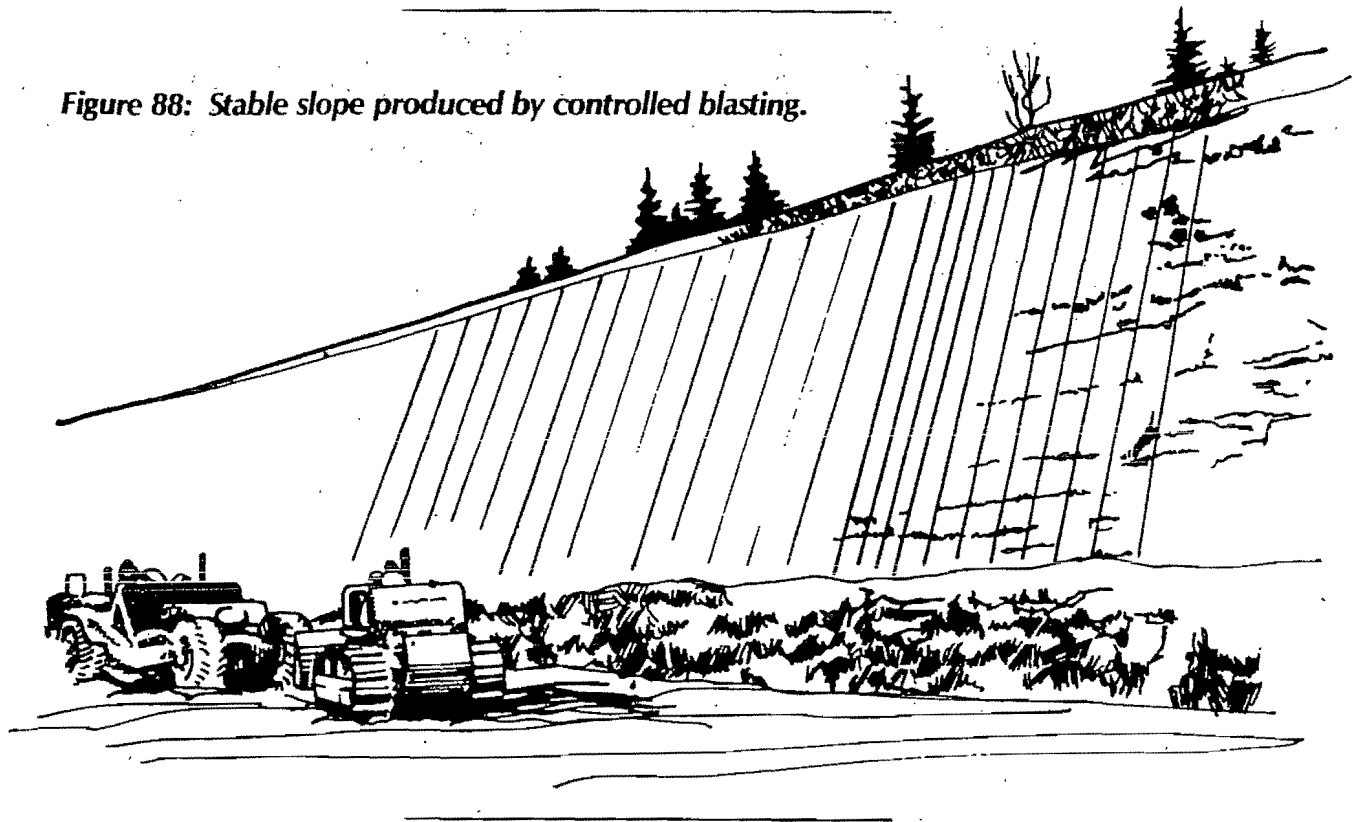
Figure 88: Stable slope produced by controlled blasting.

Line Drilling

Line drilling involves the drilling of a row of closely spaced holes along the final excavation line. It is not really a blasting technique because the line-drilled holes are not loaded with explosive. The line-drilled holes provide a plane of weakness to which the final row of blastholes can break and also reflect a portion of the blast's stress wave. Line drilling is used mostly in small blasting jobs and involves small holes in the range of 2- to 3-inch diameter. Line drilling holes are spaced (center to center) two to four diameters apart. The maximum practical depth to which

line drilling can be done is governed by how accurately the alignment of the holes can be held at depth, and is seldom more than 30 feet.

To further protect the final perimeter, the blastholes adjacent to the line drill are often more closely spaced and are loaded more lightly than the rest of the blast, using deck charges and detonating cord downlines if necessary. Best results are obtained in a homogeneous rock with little jointing or bedding, or when the holes are aligned with a major joint plane.

The use of line drilling is limited to jobs where even a light load of explosives in the perimeter holes would cause unacceptable damage. The results of line drilling are unpredictable, the cost of drilling is high, and the results are heavily dependent on the accuracy of drilling. Table 7 gives average specifications for line drilling.

Table 7--Average Specifications for Line Drilling

Hole Diameter inches	Spacing feet
2.00	0.33-0.67
3.00	0.50-1.00

Presplitting

Presplitting, sometimes called preshearing, is similar to line drilling except that the holes are drilled farther apart and a very light load of explosive is used in the holes. Presplit holes are fired before any of the main blastholes adjacent to the presplit are fired. Although the specific theory of presplitting is in dispute, the light explosive charges propagate a sheared zone, preferably a single crack, between the holes, as shown in figure 89. In badly fractured rock, unloaded guide holes may be drilled between the loaded holes. The light powder load may be obtained by using specially designed, slender cartridges; partial or whole cartridges taped to a detonating cord downline; explosive cut from a continuous reel; or even heavy-grain detonating cord. A heavier charge of tamped cartridges is used in the bottom few feet of the hole. Figure 90 shows three types of blasthole loads used for presplitting (also see table 8). Many operators now use 3/4- or 7/8-inch by 2-foot cartridges connected with couplers.

It is desirable to completely stem around and between the cartridges in the borehole. It is also desirable, although not essential, to fire all the presplit holes on the same delay period. The maximum depth for a single presplit is limited by the accuracy of the drillholes, and is usually about 50 feet. A deviation of greater than 6 inches from the desired plane of shear will give inferior results. Presplitting far ahead of the production blast should be avoided. It is preferable to presplit a shorter section and dig that section out so that the quality of the presplit can be checked. If the presplit is unsatisfactory, adjustments can be made in subsequent blasts.

Figure 89: Crack generated by a presplit blast.
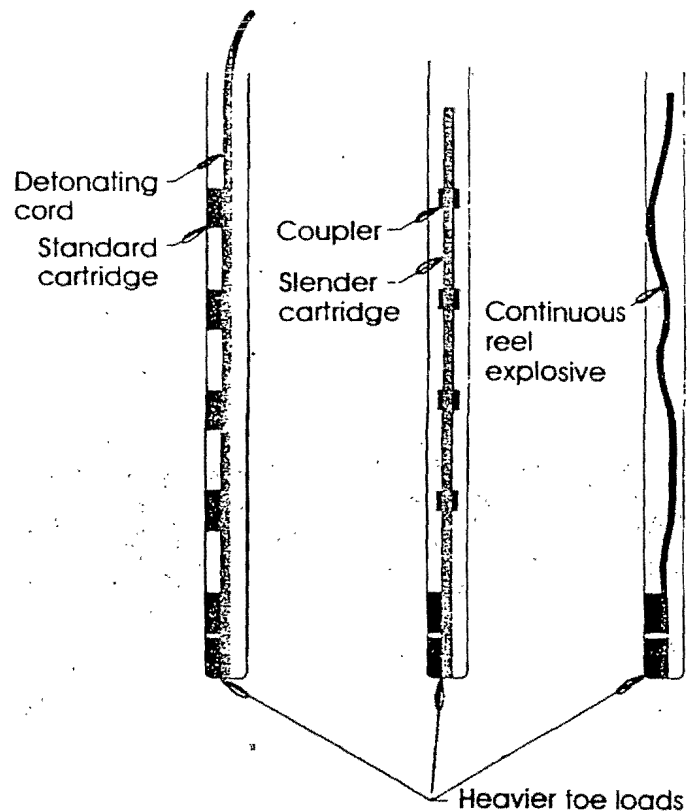
Figure 90: Three typical blasthole loads for presplitting.

Some operators prefer to fire the presplit with the main blast by firing the presplit holes on the first delay period. Although presplitting is usually thought of as a surface blasting technique, it is occasionally used underground. The increased hole spacing reduces drilling costs as compared with line drilling. Table 9 gives average specifications for presplitting.

Smooth Blasting

Smooth blasting, also called contour blasting, perimeter blasting, or sculpture blasting, is the most widely used method of controlling overbreak in underground openings such as drifts and stopes. It is similar to presplitting in that it involves a row of holes at the perimeter of the excavation that are more lightly loaded and more closely spaced than the other holes in the blast round. The light powder load is usually accomplished by "string loading" slender cartridges. Unlike presplitting, the smooth blast holes are fired after the main blast. This is usually done by loading and connecting the entire round and firing the perimeter holes one delay later than the last hole in the main round. As a first approximation, the burden on the perimeter holes should be approximately 1.5 times the spacing, as shown in figure 91. Table 10 gives average specifications for smooth blasting.

Table 8 -- Loading Densities

Pounds of Explosives per Foot of Borehole for Given Densities
(grams/cc)

Densities

Inches	0.80	0.85	0.92	0.98	1.00	1.05	1.10	1.15	1.20	1.25	1.30	1.35	1.40	1.45	1.50
3	2.45	2.60	2.82	3.00	3.06	3.22	3.37	3.52	3.68	3.83	3.98	4.14	4.29	4.44	4.60
3½	3.34	3.55	3.84	4.09	4.17	4.38	4.59	4.80	5.01	5.21	5.42	5.63	5.84	6.05	6.26
4	4.36	4.63	5.01	5.34	5.45	5.72	5.99	6.27	6.54	6.81	7.08	7.36	7.63	7.90	8.17
4½	5.52	5.86	6.34	6.76	6.90	7.24	7.58	7.93	8.27	8.62	8.96	9.31	9.65	10.00	10.34
5	6.81	7.24	7.83	8.34	8.51	8.94	9.36	9.79	10.22	10.64	11.07	11.49	11.92	12.34	12.77
5½	8.24	8.76	9.48	10.09	10.30	10.82	11.33	11.85	12.36	12.88	13.39	13.91	14.42	14.94	15.45
6	9.81	10.42	11.28	12.01	12.26	12.87	13.48	14.10	14.71	15.32	15.94	16.55	17.16	17.77	18.39
6¼	10.64	11.31	12.24	13.04	13.30	13.97	14.63	15.30	15.96	16.63	17.29	17.96	18.62	19.29	19.95
6½	11.51	12.23	13.24	14.10	14.39	15.11	15.83	16.54	17.26	17.98	18.70	19.42	20.14	20.86	21.58
7	13.35	14.28	15.35	16.35	16.69	17.52	18.35	19.19	20.02	20.86	21.69	22.52	23.36	24.19	25.03
7½	15.32	16.28	17.62	18.77	19.15	20.11	21.07	22.03	22.98	23.94	24.90	25.86	26.82	27.77	28.73
8	17.43	18.52	20.05	21.36	21.79	22.82	23.97	25.06	26.15	27.24	28.33	29.42	30.51	31.60	32.69
8½	19.68	20.91	22.63	24.11	24.60	25.83	27.06	28.29	29.52	30.75	31.98	33.21	34.44	35.67	36.90
9	22.07	23.44	25.37	26.03	27.58	28.96	30.34	31.72	33.10	34.48	35.86	37.23	38.61	39.99	41.37
10	27.24	28.94	31.33	33.37	34.05	35.75	37.46	39.16	40.86	42.56	44.27	45.97	47.67	49.37	51.08
10-5/8	30.75	32.67	35.37	37.67	38.44	40.36	42.28	44.21	46.13	48.05	49.97	51.89	53.82	55.74	57.66
12	39.23	41.68	45.11	48.05	49.03	51.49	53.94	56.39	48.84	61.29	63.74	66.20	68.85	71.10	73.55
15	61.29	65.12	70.49	75.08	76.62	80.45	84.28	88.11	91.94	95.77	99.60	103.43	107.26	111.09	114.92

(row labels, top to bottom: B O R E H O L E D I A M E T E R S)

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As a compromise between standard blasting and smooth blasting, some operators slightly reduce the spacing of their perimeter holes (see table 11), as compared with standard design, and string load regular cartridges of explosive. It is recommended procedure to seal the explosive column with a tamping plug, clay dummy, or other object to prevent the string-loaded charges from being extracted from the hole by charges on earlier delays.

Table 9--Average Specifications for Presplitting

Hole diameter inches	Spacing feet	Explosive charge lb/ft
1.50-1.75	1.00-1.50	0.08-0.25
2.00-2.50	1.50-2.00	0.08-0.25
3.00-3.50	1.50-3.00	0.13-0.50
4.00	2.00-4.00	0.25-0.75

KEY
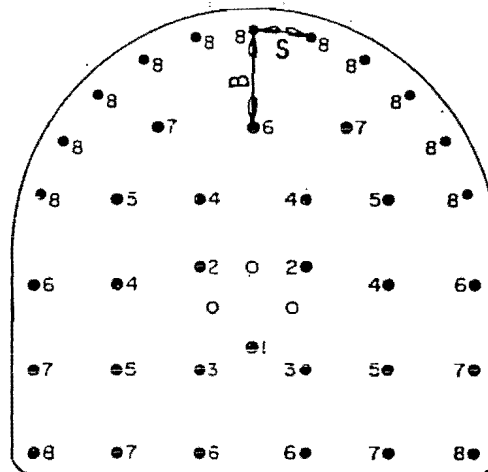
o Unloaded hole

•7 Loaded hole with delay period

Figure 91: Typical smooth blasting pattern (burden, B, is larger than spacing, S).

Table 10--Average Specifications for Smooth Blasting

Hole diameter inches	Spacing feet	Burden feet	Explosive charge lb/ft
1.50-1.75	2.00	3.00	0.12-0.25
2.00	2.50	3.50	0.12-0.25

Table 11 -- Cubic Yards of Rock per foot of Borehole

BURDEN

SPACING	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
4	1.19	1.33	1.48	1.63	1.78	1.93	2.07	2.22	2.37	2.52	2.67	2.81	2.96	3.11	3.26	3.41	3.56
5	1.48	1.67	1.85	2.04	2.22	2.41	2.59	2.78	2.96	3.15	3.33	3.52	3.70	3.89	4.07	4.26	4.44
6	1.78	2.00	2.22	2.44	2.67	2.89	3.11	3.33	3.56	3.78	4.00	4.22	4.44	4.67	4.89	5.11	5.33
7	2.07	2.33	2.59	2.85	3.11	3.37	3.63	3.89	4.15	4.41	4.67	4.93	5.19	5.44	5.70	5.96	6.22
8	2.37	2.67	2.96	3.26	3.56	3.85	4.15	4.44	4.74	5.04	5.33	5.63	5.93	6.22	6.52	6.81	7.11
9	2.67	3.00	3.33	3.67	4.00	4.33	4.67	5.00	5.33	5.67	6.00	6.33	6.67	7.00	7.33	7.67	8.00
10	2.96	3.33	3.70	4.07	4.44	4.81	5.19	5.56	5.93	6.30	6.67	7.04	7.41	7.78	8.15	8.52	8.89
11	3.26	3.67	4.07	4.48	4.89	5.30	5.70	6.11	6.52	6.93	7.33	7.74	8.15	8.56	8.96	9.37	9.78
12	3.56	4.00	4.44	4.89	5.33	5.78	6.22	6.67	7.11	7.56	8.00	8.44	8.89	9.33	9.78	10.22	10.67
13	3.85	4.33	4.81	5.30	5.78	6.26	6.74	7.22	7.70	8.19	8.67	9.15	9.63	10.11	10.59	11.07	11.56
14	4.15	5.67	5.19	5.70	6.22	6.74	7.26	7.78	8.30	8.81	9.33	9.85	10.37	10.89	11.41	11.29	12.44
15	4.44	5.00	5.56	6.11	6.67	7.22	7.78	8.33	8.89	9.44	10.00	10.56	11.11	11.67	12.22	12.78	13.33
16	4.74	5.33	5.93	6.52	7.11	7.70	8.30	8.89	9.48	10.07	10.67	11.26	11.85	12.44	13.04	13.63	14.22
17	5.04	5.67	6.30	6.93	7.56	8.19	8.81	9.44	10.07	10.70	11.33	11.96	12.50	13.22	13.85	14.48	15.11
18	5.33	6.00	6.67	7.33	8.00	8.67	9.33	10.00	10.67	11.33	12.00	12.67	13.33	14.00	14.67	15.33	16.00
19	5.63	6.33	7.04	7.74	8.44	9.15	9.85	10.56	11.26	11.96	12.67	13.37	14.07	14.78	15.48	16.19	16.89
20	5.93	6.67	7.41	8.15	8.89	9.63	10.37	11.11	11.85	12.59	13.33	14.07	14.81	15.56	16.30	17.04	17.78
21	6.22	7.00	7.78	8.56	9.33	10.11	10.89	11.67	12.44	13.22	14.00	14.78	15.56	16.33	17.11	17.89	18.67
22	6.52	7.33	8.15	8.96	9.78	10.59	11.41	12.22	13.04	13.85	14.67	15.48	16.30	17.11	17.93	18.74	19.56
23	6.81	7.67	8.52	9.37	10.22	11.07	11.93	12.78	13.63	14.48	15.33	16.19	17.04	17.89	18.74	19.59	20.44
24	7.11	8.00	8.89	9.78	10.67	11.56	12.44	13.33	14.22	15.11	16.00	16.89	17.78	18.67	19.56	20.44	21.33

Smooth blasting reduces overbreak and reduces the need for ground support. This usually outweighs the cost of the additional perimeter holes.

Cushion Blasting

Cushion blasting is surface blasting's equivalent to smooth blasting. Like other controlled blasting techniques, it involves a row of closely spaced, lightly loaded holes at the perimeter of the excavation. Holes up to 6-1/2 inches in diameter have been used in cushion blasting. Drilling accuracy with this size borehole permits depths of up to 90 feet for cushion blasting. After the explosive has been loaded, stemming is usually placed in the void space around the charges. The holes are fired after the main excavation is removed. A minimum delay between the holes is desirable. The same loading techniques that apply to presplitting are used with cushion blasting, except that the latter often involves larger holes. The burden on the cushion holes should always be larger than the spacing between holes.

The larger holes associated with cushion blasting result in larger spacings as compared with presplitting, thus reducing drilling costs. Better results can be obtained in unconsolidated formations than with presplitting, and the larger holes permit better alignment at depth. Table 12 gives average specifications for cushion blasting.

Table 12--Average Specifications for Cushion Blasting

Hole diameter inches	Spacing feet	Burden feet	Explosive charge lb/ft
2.00-2.50	3.00	4.00	0.08-0.25
3.00-3.50	4.00	5.00	0.13-0.50
4.00-4.50	5.00	6.00	0.25-0.75
5.00-5.50	6.00	7.00	0.75-1.00
6.00-6.50	7.00	9.00	1.00-1.50

FORMULAS

Volume of rock per borehole = $\dfrac{\text{B (burden)} \times \text{S (spacing)} \times \text{D (depth)}}{27}$ = cubic yd

Volume of rock per shot = (volume of rock per borehole) x (number of holes)

Loading density = weight of explosives loaded per foot of borehole

Pounds of explosives per borehole = (loading density) x (column charge length)

Column charge length = (depth of borehole) - (stemming)

Powder factor = $\dfrac{\text{pounds of explosives per borehole}}{\text{cubic yards of rock per borehole}}$

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$$W = (D/D_s)^2$$

    W = maximum charge weight of explosives
    D = distance from blast to nearest structure
    D_s = scaled distance currently used in Montana's Permanent Program
        (See A.R.M. 26.4.624)

Steps for Calculating Pounds of Explosive per Delay:

1. Determine the firing time for each hole or deck in a blast.
2. Compare the firing time of each hole or deck to every other firing time in the blast. If two firing times are within 8 milliseconds they must be considered the same delay.
3. Determine the number of holes or decks per delay.
4. Determine the largest amount of explosives per hole or deck. Be sure to include all the explosives. (Add the weight of the primers and the blasting agent or explosive.)
5. Multiply the maximum pounds per hole or deck by the number of holes or decks per delay.

REFERENCES

1. Andrews, A.B. Design of Blasts. Emphasis on Blasting. Ensign Bickford Co. (Simsbury, CN), Spring 1980, pp. 1,4.
2. Ash, R.L. The Mechanics of Rock Breakage, Parts I, II, III, and IV. Pit and Quarry, v. 56, No. 2, August 1963, pp. 98-112; No. 3, September 1963, pp. 118-123; No. 4, October 1963, pp. 126-131; No. 5, November 1963, pp. 109-111, 114-118.
3. Dick, R.A., and J.J. Olson. Choosing the Proper Borehole Size for Bench Blasting. Min. Eng., v. 24, No. 3, March 1972, pp. 41-45.
4. E.I. du Pont de Nemours & Co., Inc. (Wilmington, DE) Blaster's Handbook. 16th ed., 1978, 494 pp.
5. _____. Four Major Methods of Controlled Blasting. 1964, 16 pp.
6. Hercules, Inc. (Wilmington, DE). Shotfirer's Guide. 1978, 12 pp.
7. Pugleise, J.M. Designing Blast Patterns Using Empirical Formulas. BuMines IC 8550, 1972, 33 pp.
8. Schmidt, R.L., R.J. Morrell, D.H. Irby, and R.A. Dick. Application of Large-Hole Burn Cut in Room-and-Pillar Mining. BuMines RI 7994, 1975, 25 pp.
9. Winzer, S.R. The Firing Times of MS Delay Blasting Caps and Their Effect on Blasting Performance. Prepared for National Science Foundation (NSF APR 77-05171). Martin Marietta Laboratories (Baltimore, MD). June 1978, 36 pp.; available for consultation at Bureau of Mines Twin Cities Research Center, Minneapolis, MN.

CHAPTER 4
REVIEW QUESTIONS

1. Define burden. _____

   _____

2. Define spacing. _____

   _____

3. What is subdrilling? _____

   _____

4. How do you figure column charge length? _____

   _____

5. Study the drawing below.
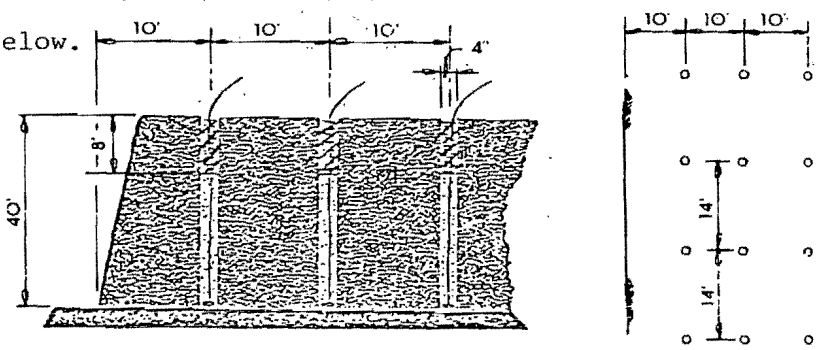
   Using the number given above, fill in the following dimensions:

   a. Burden distance _____

   b. Spacing distance _____

   c. Depth of drill hole _____

   d. Depth of stemming _____

   e. Column charge length _____

   f. Diameter _____

6. What is the formula for volume of rock per borehole? _____

   _____

7. List the factors that are important for determining burden and spacing.

   _____

   _____

8. Using the table on page 129, Table 11, Cubic Yards of Rock per Foot of Borehole, determine the volume per borehole for a blast with 15-foot burden and 20-foot spacing and 30-foot-deep hole.

9. Using the formula, determine the volume per borehole for a blast with a 10-foot burden and 15-foot spacing and a 40-foot depth.

10. What is loading density? _____

_____

11. A 6-inch borehole is loaded with a water gel that has a density of 1.20 g/cc. Stemming height is 8 feet; hole depth is 40 feet.

   a. Find the pounds of explosive per foot of borehole (the loading density).

   b. Find the total pounds of explosive per borehole.

12. Each hole in a blast contains 215 pounds of AN-FO. The burden is 12 feet; the spacing is 18 feet; the hole depth is 30 feet. Find the powder factor.

13. You are working on a shot with the following dimensions:

burden	= 13'
spacing	= 16'
depth	= 30'
diameter	= 6"
stemming	= 8'

number of holes = 24

You are loading with AN-FO which has a density of .80. Figure out the following:

a. volume of rock per borehole _____

b. volume of rock per shot _____

c. loading density _____

d. pounds of explosive per hole _____

e. powder factor _____

f. Is this shot practical? _____ Why or why not? _____

_____

14. You are working on a shot with the following dimensions:

    burden   = 10'

    spacing   = 15'

    depth   = 20'

    diameter   = 4-3/4"

    stemming   = 7'

    number of holes = 20

You are loading with wet hole tubes of AN-FO-HD, tube diameter = 4".

Density is 1.05 g/cc. Figure out the following:

a. volume of rock per borehole _____

b. volume of rock per shot _____

c. loading density _____

d. pounds of explosive per hole _____

e. powder factor _____

f. Is this shot practical? _____ Why or why not? _____

_____

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15. Although secondary blasting employs small charges, _____ is

often more severe and the most hazardous.

16. List two types of delay mechanisms for detonating cord shots.

_____

_____

17. A shot is loaded so that four holes detonate per delay. Each hole con-

tains four 50-pound bags of AN-FO and one 1-pound cast primer. Calculate

the maximum pound of explosive per delay.

_____

18. Study the electric shot design below.

350 300 250 300 350

250 200 175 200 250

175 150 75 150 175

75 50 25 50 75

FACE
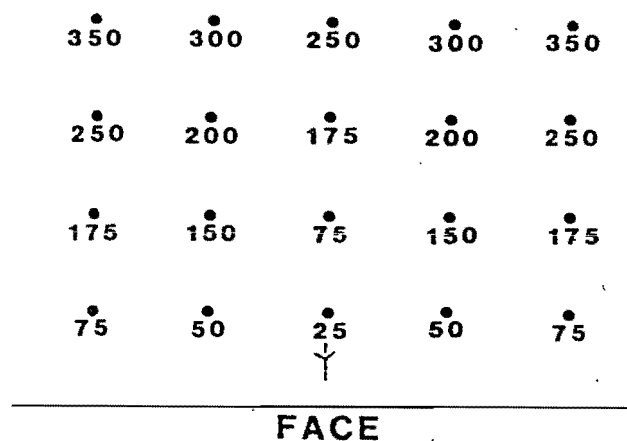
Each hole contains 402 pounds of explosives. What is the maximum amount

of explosives per delay period? _____

19. Study the nonelectric shot design below.

17 17 17 17 17 17
42
17 17 17 17 17 17
42
17 17 17 17 17 17

point of initiation

138
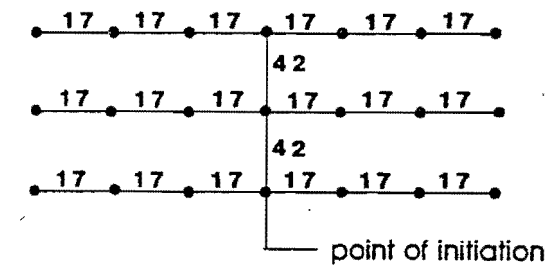
Each hole contains 304 pounds of explosives. Figure the maximum pounds per delay. _____

20. Explosives function best when:

   a. _____

   b. _____

21. What is the most significant geologic feature of rock to pay attention to when blasting? _____

22. How can voids be identified? _____

23. What is the best way to blast where there are alternating zones of competent and incompetent rock? _____

24. What is a deck charge used for? _____

25. Insufficient collar distance above the explosive charge can result in:

   a. _____   b. _____

26. Reducing the weight of explosive fired per delay interval controls _____

   _____.

27. Excessive ground vibration may be caused by:

   a. _____   b. _____

   c. _____   d. _____

28. Hole depths should not be more than _____ the burden.

29. For good fragmentation and flyrock control, the delay time between rows should be (a) _____ because

   (b) _____.

30. When is the only time powder factor is useful? _____

31. What secondary blasting technique may have the most environmental impact?

   _____

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32. What two potentially explosive materials are common in an underground mine?

    a. _____   b. _____

33. What is the purpose of controlled blasting? _____

34. What are four methods of controlling blasting? What does it do?

    a. _____

    b. _____

    c. _____

    d. _____

Chapter 5: Environmental Effects of Blasting

There are four environmental effects of blasting.

1. Flyrock
2. Ground vibrations
3. Airblast
4. Dust and gases

Flyrock is a potential cause of death, serious injury, and property damage. Ground vibrations and airblast are potential causes of property damage and human annoyance, but are very unlikely to cause personal injury. Flyrock, ground vibrations, and airblast all represent wasted explosive energy. Excessive amounts of these undesirable side effects are caused by improper blast design or lack of attention to geology. When excessive side effects occur, part of the explosive energy that was intended to give the proper amount of rock fragmentation and displacement is lost to the surrounding rock and atmosphere. Serious dust or gas problems are seldom caused by blasting. A larger-than-normal amount of dust may be caused by a violent shot. Noxious gases, normally oxides of nitrogen or carbon monoxide, are the result of an inefficient explosive reaction. Because of its sporadic nature, blasting is not a significant source of air pollution.

When blasting in the vicinity of structures (figure 92) such as homes, hospitals, schools, and churches, a preblast survey that documents the condition of the structures is often beneficial. A preblast survey has a twofold purpose. First, it increases communications between the community and the mine operator. It has long been recognized that good public relations are the operator's best means of reducing blasting complaints. A preblast survey helps the operator maintain good community relations. Many companies have been conducting preblasting surveys for years and have found them to be an excellent investment.

The second purpose of a preblast survey is to provide a baseline record of the condition of a structure against which the effects of blasting can be assessed. When combined with a postblast survey, this will help assure equitable resolution of blast damage claims. Montana's Permanent Program regulations require that a preblast survey be conducted, at the homeowners' request, on all homes within 0.5 mile of blasting at surface coal mines.

Good blast recordkeeping is essential to any successful blasting operation. A blasting record is useful in troubleshooting the cause of undesirable blasting results such as flyrock, airblast, ground vibrations, and poor fragmentation. The blasting record may also provide excellent evidence in litigation on blast damage or nuisance. Figure 93 gives an example of a blasting record. Depending on the blasting situation, some of the information con-
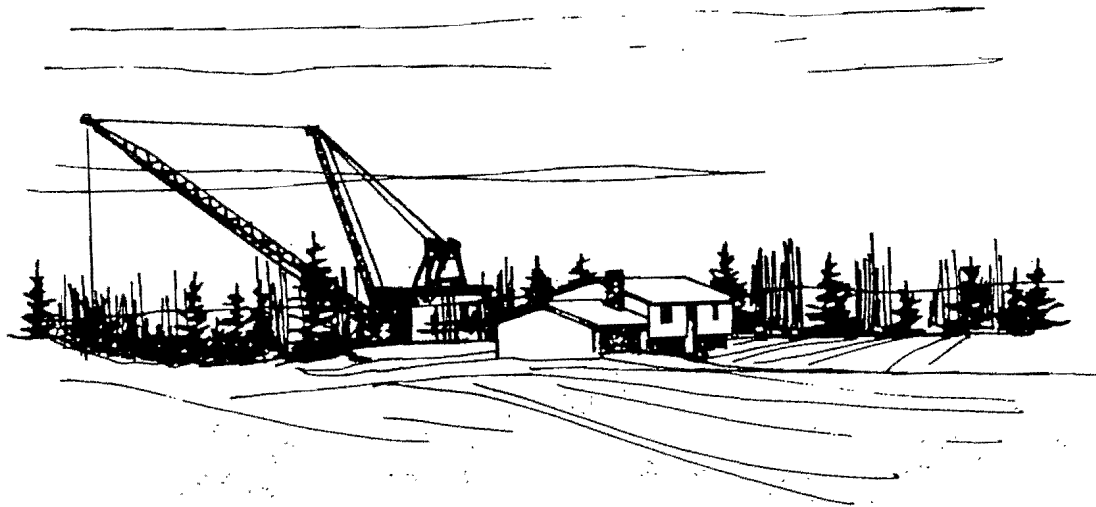
Figure 92: Mining near a residential structure.

tained in figure 93 may not be required. On the reverse side of the blasting record, a sketch of the blast pattern, including delays, and a sketch of a typical loaded hole should be drawn.

FLYROCK

Flyrock, primarily associated with surface mining, is the most hazardous effect of blasting. It is a leading cause of onsite fatalities and equipment damage from blasting. Occasionally, flyrock will leave the mine site and cause serious injury and damage to persons and property beyond the mine limits. Flyrock distances can range from zero, for a well-controlled coal strip-mine blast, to nearly a mile for a poorly confined, large, hard-rock mine blast. The term flyrock can be defined as an undesirable throw of material. Muckpile displacements on the order of 100 feet are often desirable for certain types of loading equipment, such as front-end loaders. Even larger displacements may be desirable for explosive casting of waste material.

Causes and Alleviation

Excessive flyrock is most often caused by an improperly designed or improperly loaded blast (5). A burden dimension less than 25 times the charge diameter often gives a powder factor too high for the rock being blasted. The excess explosive energy results in long flyrock distances. On the other hand, an excessively large burden may cause violence in the collar zone, especially where an inadequate amount or an ineffective type of stemming is used. This situation is compounded when top priming is used, as opposed to center or toe priming.

To prevent or correct flyrock problems, the blaster should make sure that the burden is proper and that enough collar distance is used. One-fourth-

```
                          BLASTING RECORD
                                              Date: _____
                              Company
Location of blast:      _____

Time of blast:       _____

Date of blast:       _____

Name of blaster: _____    License No.:  _____

Number of persons in blasting crew:  _____

Direction:  _____  Distance:  _____ feet from blast
to nearest dwelling, school, church, commercial, or institutional building.

Weather data
     Temperature:  _____  Wind direction and speed: _____
     Cloud cover:  _____

Type of material blasted:  _____

No. of holes:  _____  Burden: _____ Spacing: _____ Depth: _____
                         Diameter: _____

Type of explosive used:  _____

Maximum weight of explosive detonated within any 8-ms period: _____ lb.

Maximum number of holes detonated within any 8-ms period: _____

Total weight of explosives, including primers, this blast: _____ lb.

Method of firing and type of circuit:  _____

Type and length of stemming:  _____

Were mats or other protection used?  _____

Type of delay detonator used:  _____  Delay periods used: _____
                                                     Calibration
Seismic data:  T _____, V _____, L _____, dB _____ Signal _____

Location of seismograph:  _____  Distance from blast _____ ft.

Name of person taking seismograph reading:  _____

Name of person and firm analyzing the seismograph record:  _____

_____

                         Signed: _____, Blaster
```

Figure 93: Example of a blasting record

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inch-size material makes better stemming than fines, particularly where there is water in the boreholes. In come cases, it may be necessary to lengthen the stemming zone above the main charge and use a small deck charge to reduce flyrock and still assure that the caprock is broken. Top initiation is a particularly poor practice where flyrock is a problem. In multiple-row shots, long delays between later rows, on the order of 10 milliseconds per foot of burden, may reduce flyrock. Precautions should be taken against cutoffs when using delays of this length.

Zones of weakness and voids are often causes of flyrock. These potential problems can sometimes be foreseen through consultation with the drill operator and through past experience in the area being blasted. An abnormal lack of resistance to drill penetration usually indicates a mud seam, a zone of incompetent rock, or even a void. The driller should note the depth and the severity of this zone of weakness on the drill log. Any explosive loaded in this zone will follow the line of least resistance and "blow out," causing flyrock (figure 59). Placing a few feet of stemming, rather than explosive, in this area will reduce the likelihood of flyrock (figure 62). Void zones such as mine openings or solution cavities cause violent explosions when packed with explosives. It is always a good idea to measure the buildup of the column as explosive loading proceeds. If buildup is abnormally slow, the zone should be stemmed and the powder column continued above it. Measuring the column buildup will also assure that adequate room is left for stemming above the charge.

Protective Measures

Despite careful planning and good blast design, flyrock may occasionally occur and must always be protected against. Some margin for error must always be maintained. Abnormally long flyrock distances should be measured and recorded for future reference. The size of the guarded perimeter should take these cases into account. An adequate number of guards must be posted at safe distances. Any people within this perimeter must have safe cover and must be adequately warned. Remember that warning signs, prearranged blasting times, or warning sirens, in themselves, are seldom adequate for blast guarding. It is particularly good if the blaster has a commanding field of view of the blast area so the shot can be aborted at the last minute if necessary.

Montana Permanent Program regulations prohibit throwing flyrock more than one-half the distance to the nearest dwelling or occupied structure, and beyond the operator's property line. Local flyrock regulations may also exist. In small, close-in construction blasts, special protective mats may be used to contain flyrock. However, this is impractical in mine blasts or other large blasts.

GROUND VIBRATIONS

All blasts create ground vibrations. When an explosive is detonated in a borehole, it creates a shock wave that crushes the material around the borehole and creates many of the initial cracks needed for fragmentation. As this wave travels outward, it becomes a seismic, or vibration wave. As the wave passes a given piece of ground, it causes that ground to vibrate. The situation is similar to the circular ripples produced on the surface of a pool of calm water when it is struck by a rock (6). Ground vibrations are measured
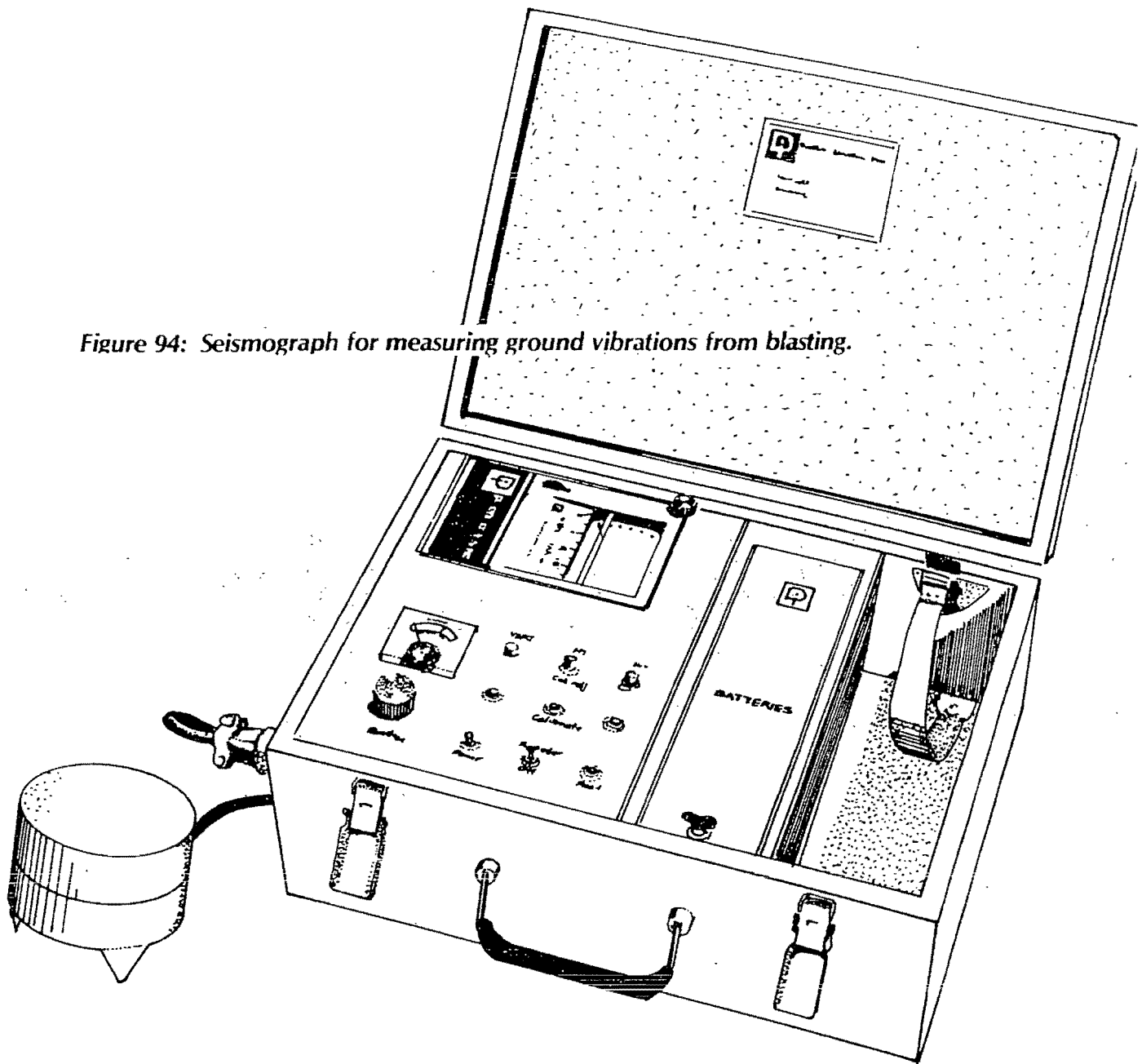
Figure 94: Seismograph for measuring ground vibrations from blasting.

with seismographs (12) (figure 94); they are measured in terms of amplitude (size of the vibrations) and frequency (number of times the ground moves back and forth in a given time period). In blasting, amplitude is usually measured in terms of velocity (inches per second) and frequency is measured in hertz, or cycles per second. Excessively high ground-vibration levels can damage structures. Even moderate to low levels of ground vibration can be irritating to neighbors and can cause legal claims of damage or nuisance. One of the best protections against claims is good public relations (1). While striving to minimize ground vibrations, the blaster or the company should inform local residents of the need for and the importance of the blasting, and the relative harmlessness of properly controlled blasting vibrations when compared to the day-to-day stresses to which a structure is subjected. Prompt and sincere response to complaints is important.

Causes

Excessive ground vibrations are caused either by putting too much explosive energy into the ground or by not properly designing the shot. Part of the energy that is not used in fragmenting and displacing the rock will go into ground vibrations. The vibration level at a specific location is primarily determined by the maximum weight of explosives that is used in any single delay period in the blast and the distance of that location from the blast (9).

The delays in a blast break it up into a series of smaller, very closely spaced individual blasts. The longer the intervals between delays, the better the separation will be between the individual blasts. Most predictive schemes and regulatory agencies use a guide of 8 or 9 milliseconds as the minimum delay that can be used between charges if they are to be considered as separate charges for vibrations purposes. However, there is nothing magical about the 8- or 9-milliseconds interval. For small, close-in blasts, a smaller delay may give adequate separation.

With large blasts at large distances from structures, longer delays are required to obtain true separation of vibrations because the vibration from each individual charge lasts for a longer period of time. In general, vibration amplitudes at structures sitting on the formation of rock being blasted will be greater than at structures sitting on other formations. However, the vibrations may be higher in frequency, which reduces the response of structures and the likelihood of damage.

In addition to charge weight per delay, distance, and delay interval, two factors may affect the level of ground vibrations at a given location. The first is overconfinement. A charge with a properly designed burden will produce less vibration per pound of explosive than a charge with too much burden

NORMAL VIBRATIONS   EXCESSIVE VIBRATIONS
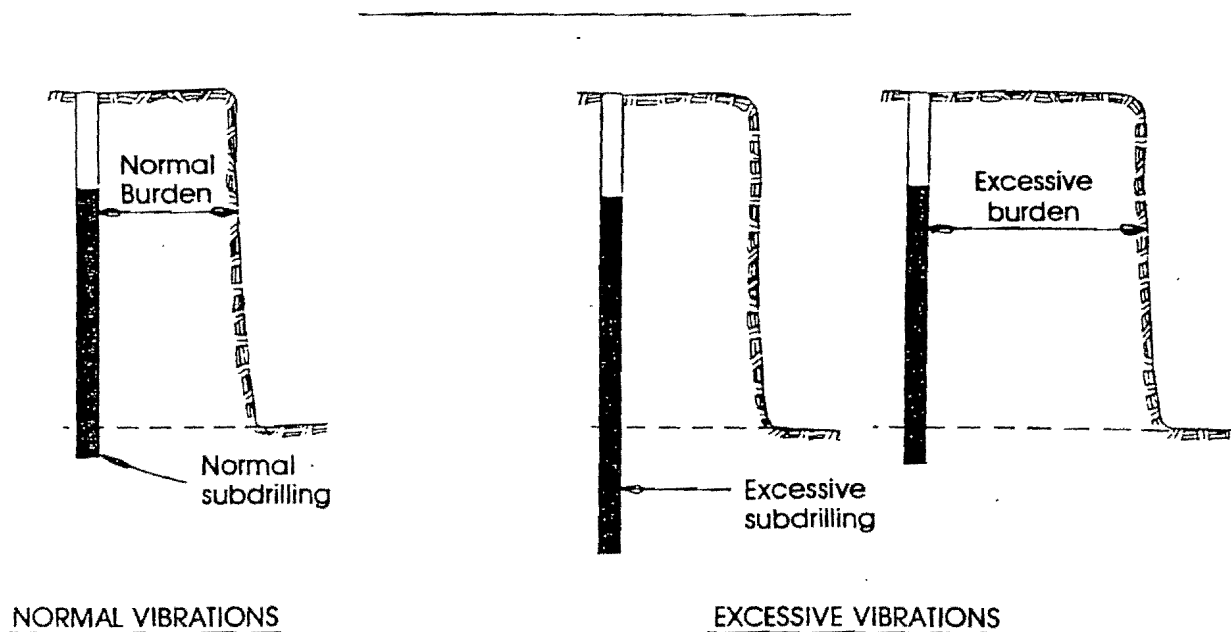
Figure 95: Effects of confinement on vibration levels.

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KEY
•ⱼ Blasthole and delay period

Intermediate vibrations

Lower vibrations

Higher vibrations

•₁ •₂ •₃ •₄ •₅
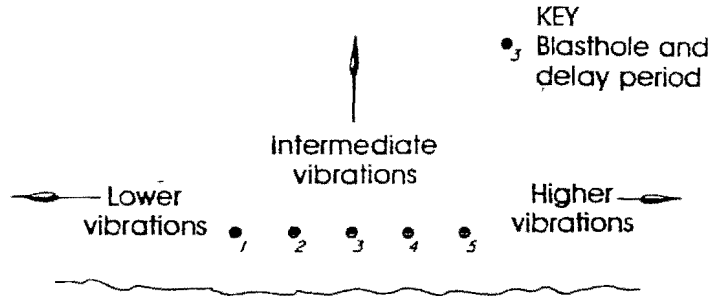
Figure 96: Effect of delay sequence on particle velocity.

(figure 95). An excessive amount of subdrilling, which results in an extreme-
ly heavy confinement of the explosive charge, will also cause higher levels of
ground vibration, particularly if the primer is placed in the subdrilling. In
multiple row blasts, there is a tendency for the later rows to become overcon-
fined (figure 75). To avoid this, it is often advisable to use longer delay
periods between these later rows to give better relief. In some types of
ground, these longer delays may increase the chance of cutoffs, so some trade-
offs must be made. Also, if delays proceed in sequence down a row, the vibra-
tions in the direction that the sequence is proceeding will be highest (figure
96) because of a snowballing effect.

Recent studies (13) have shown that millisecond delays in commercial de-
tonators are less accurate than was previously believed. This may result in
extremely close timing between adjacent delay periods or, very rarely, an
overlap. Where it is critical that one hole detonates before an adjacent hole
to provide relief, it may be a good idea to skip a delay period between the
two holes.

Most underground mines shoot relatively small blasts and do not have
vibration problems. However, where vibrations are a problem, the discussions
in this chapter apply to underground blasting as well as surface blasting.

Prescribed Vibration Levels and Measurement Techniques

Two vibration limits are important—the level above which damage is
likely to occur and the level above which neighbors are likely to complain.
There is no precise level at which damage begins to occur. The damage level
depends on the type, condition, and age of the structure, the type of ground
on which the structure is built, and the frequency of the vibration, in hertz
(Hz). Research completed by the Bureau of Mines in the late 1970s (9) recom-
mends that for very close-in construction blasting, where the frequency is
above 40 Hz, vibration levels be kept below 2 inches/second to minimize
damage. However, all mine and quarry blast vibrations, and those from large
construction jobs, have frequencies below 40 Hz. For these blasts, the vibra-
tion level should be kept below 0.75 inches/second for older homes with
plaster-on-lath walls. These values could change as more research is done.

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People tend to complain about vibrations far below the damage level. The threshold of complaint for an individual depends on health, fear of damage (often greater when the owner occupies the home), attitude toward the mining operation, diplomacy of the mine operator, how often and when blasts are fired, and the duration of the vibrations. The tolerance level could be below 0.1 inches/second where the local attitude is hostile toward mining, where the operator's public relations stance is poor, or where numerous older people own their homes. On the other hand, where the majority of people depend on the mine for their livelihood, and where the mine does a good job of public relations, levels above 0.5 inches/second might be tolerated. By using careful blast design and good public relations, it is usually possible for an operator to live in harmony with neighbors without resorting to expensive technology.

Several options are available for measuring ground vibrations (12). Many operators prefer to hire consultants to run their monitoring programs. Either peak reading seismographs or seismographs that record the entire vibration event on a paper record may be used. Peak reading instruments are cheaper, easier to use, and are adequate for assuring regulatory compliance in most cases. However, seismographs that record the entire time history are more useful for understanding and troubleshooting ground vibration problems. Instruments that measure three mutually perpendicular components (radial, transverse, vertical) are most common, and most regulations specify this type of measurement. Vector sum instruments will always give a higher reading (usually 10 to 25 percent higher) than the highest single component of a three-component instrument. Because vector sum instruments always give a higher reading, they should be satisfactory for regulatory compliance even where the regulation specifies three components.

Some seismographs require an operator to be present while others operate remotely, usually for a period of a month between battery charges. Operator-attended instruments are cheaper but require the expense of the operator. They can be moved from place to place to gather specific data on specific blasts. Remotely installed instruments are useful in that they record each blast without sending an operator out each time. These instruments should be installed in places that are protected from weather and tampering. When recording remotely, it is easier to detect tampering with seismographs that record the entire time history than with peak reading instruments.

When accelerations larger than 0.3 g are expected, the seismograph should be secured to the ground surface. Many instruments are equipped with stakes for this purpose. Epoxy or bolting may be used on hard surfaces. When possible, when the expected acceleration level is high, the gage should be buried in the ground.

Seismograph records provide excellent evidence in case of later complaints or lawsuits on damage or nuisance from blasting. A complete blast record, as shown in figure 93, describing the layout, loading, initiation, and other pertinent aspects of the blast is also essential.

Scale Distance Equation

Where vibrations are not a serious problem, regulations will often permit the blaster to use the scaled distance equation rather than measure vibrations with a seismograph. The scaled distance equation is:

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$$S.D. = D/W^{1/2} \text{ or } W = (D/S.D.)^2$$

where S.D. is the scaled distance, D is the distance from the blast to the structure of concern, in feet, and W is the maximum charge weight of explosives, in pounds, per delay of 8 milliseconds or more. The scaled distance permitted depends on the allowable vibration level. For instance, Bulletin 656 (7) says that a scaled distance of 50 or greater will protect against vibrations greater than 2 inches/second. Therefore, at a distance of 500 feet, 100 pounds of explosive could be fired; at 1,000 feet, 400 pounds; at 1,500 feet, 900 pounds; etc. The original OSM regulations (2-3) specified a scaled distance of 60 or greater to protect against 1 inch/second, giving distance-weight combinations of 600 feet and 100 pounds; 1,200 feet and 400 pounds, 1,800 feet and 900 pounds, etc. Montana's Permanent Program is currently using several scaled distances. Appendix B contains additional information with regard to the scaled distance equation and the regulations.

The scaled distance approach works well when the mine is an adequate distance from structures, vibrations are not a problem, and the operator wants to save the expense of measuring vibrations. At close distance, however, the scaled distance becomes quite restrictive in terms of allowable charge weights per delay and monitoring is often a more economical option.

Reducing Ground Vibrations

A properly designed blast using the principles described in chapter 4 will give lower vibrations per pound of explosive than a poorly designed blast. Proper blast design includes using a spacing-to-burden ratio equal to or greater than 1.0, using proper delay patterns, and using a proper powder factor. Blasthole locations should be carefully laid out and drilling should be controlled as closely as possible. Bench marks should be established for use in setting out hole locations for the next blast before the current blast is made to avoid possible errors due to backbreak (4).

The following techniques can be used to reduce ground vibrations.

1. Reduce the charge weight of explosives per delay. This is most easily done by reducing the number of blastholes fired on each delay. If there are not enough delay periods available, this can be alleviated by using a sequential timer blasting machine or a combination of surface and in-hole nonelectric delays. The manufacturer should be consulted for advice when using the sequential timer or complex delay systems. If the blast already employs only one blasthole per delay, smaller diameter blastholes, a lower bench height, or several delayed decks in each blasthole can be used. Delays are often required when presplitting.
2. Overly confined charges, such as those having too much burden or too much subdrilling should be avoided. The primer should not be placed in the subdrilling. Where it appears that a later row of blastholes will have inadequate relief, a delay period should be skipped between rows.
3. The length of delay between charges can be increased. This is especially helpful when firing large charge weights per delay at large blast-to-structure distances. However, this will increase the duration of the blast and may cause more adverse reactions from neighbors.

4. If delays in a row are arranged in sequence, the lowest delay should be placed in the hole nearest the structure of concern. In other words, the shot should be propagated in a direction away from the structure.
5. The public's perception of ground vibrations can be reduced by blasting during periods of high local activity such as the noon hour or shortly after school has been dismissed. Blasting during typically quiet periods should be avoided, if possible.

AIRBLAST

Airblast is a transient impulse that travels through the atmosphere. Much of the airblast produced by blasting has a frequency below 20 Hz and cannot be heard effectively by the human ear. However, all airblast, both audible and inaudible, can cause a structure to vibrate in much the same way as ground vibrations (8,10). Airblast is measured with special gages, pressure transducers, or wide-response microphones (11). These instruments are often an integral part of blasting seismographs (figure 97). As with ground vibrations, both amplitude and frequency are measured. Amplitude is usually measured in decibels, sometimes in pounds per square inch, and frequency is measured in hertz. Research has shown that airblast from a typical blast has less potential than ground vibrations to cause damage to structures. It is, however, frequently the cause of complaints. When a person senses vibrations
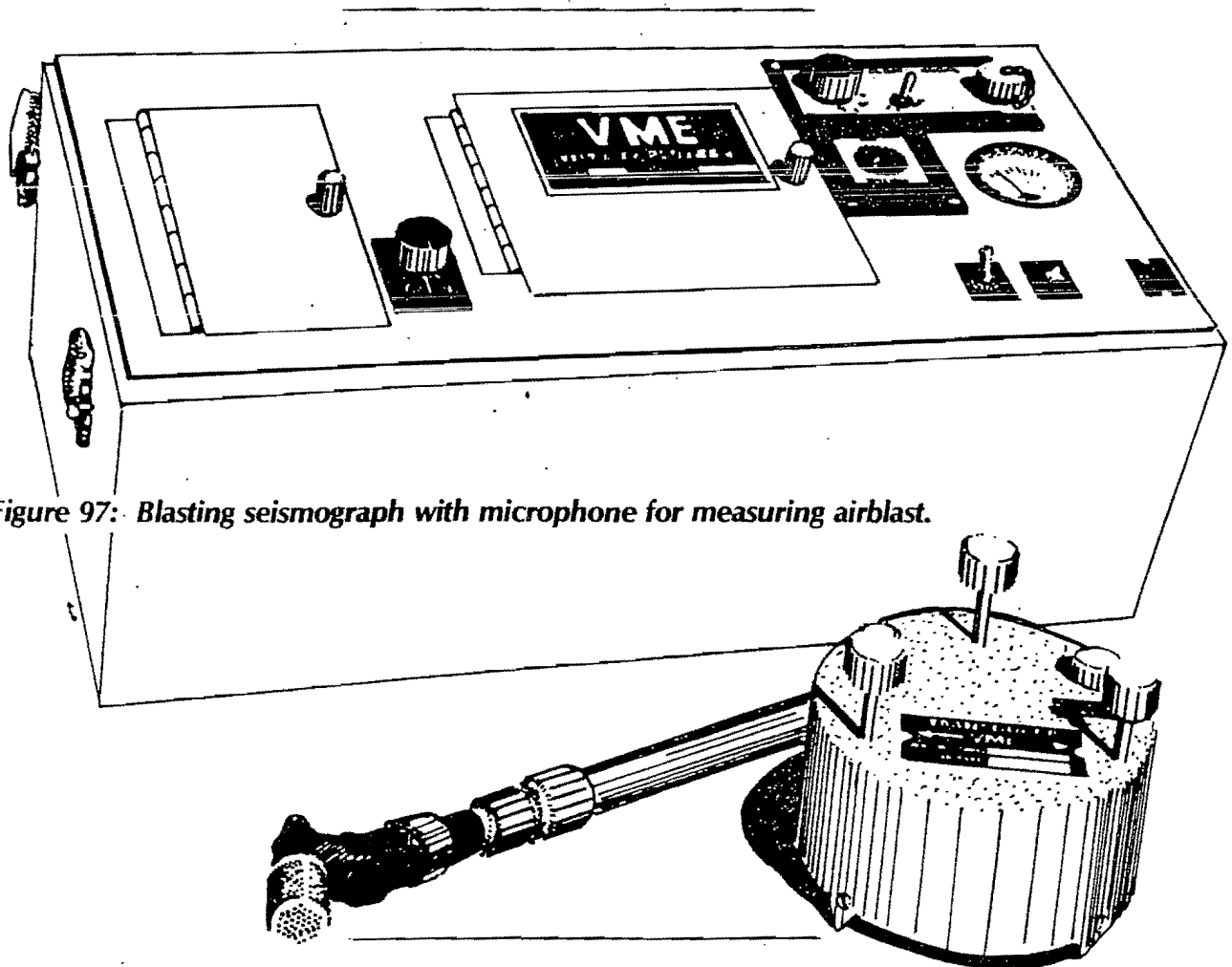
Figure 97: Blasting seismograph with microphone for measuring airblast.

from a blast, or experiences house rattling, it is usually impossible to tell whether ground vibrations or airblast is being sensed. A discussion of airblast should be part of any mine public relations program.

Causes

Airblast is caused by one of three mechanisms (6) as shown in figure 98. The first cause is energy released from unconfined explosives, such as uncovered detonating cord trunklines or mudcaps, used for secondary blasting. The second cause is the release of explosive energy from inadequately confined borehole charges. Some examples are inadequate stemming, inadequate burden, or mud seams. The third cause is movement of the burden and the ground surface. Most blasts are designed to displace the burden. When the face moves out, it acts as a piston to form an air compression wave (airblast). For this reason, locations in front of the free face receive higher airblast levels that those behind the free face.
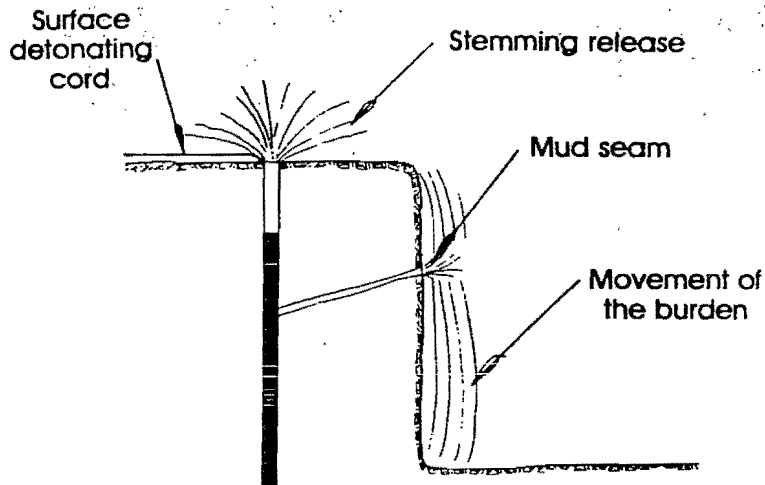
Figure 98: Causes of airblast.

Prescribed Airblast Levels and Measurement Techniques

Siskind (8) has studied the problem of damage from airblast. Table 13 shows the airblast levels prescribed for preventing damage to structures.

Table 13--Maximum Recommended Airblast Levels

Frequency range of instrumentation	Maximum level, dB
0.1 to 200 Hz, flat response	134 peak
2 to 200 Hz, flat response	133 peak
6 to 200 Hz, flat response	129 peak
C-weighted, slow response	105 C

As indicated in the table, different instruments have different lower frequency limits. Because much of the airblast is contained in these lower frequency levels, some of the instruments measure more of the airblast than others. That is the reason for the different maximum levels in the table. It is necessary to meet only one of these values, depending on the specifications of the instrument used.

Because airblast is a major cause of blasting complaints, merely meeting the levels given in the table is sometimes not sufficient. Airblast levels should be kept as low as possible by using the techniques described later in this section. This will go a long way toward reducing complaints and conflicts with neighbors.

Any instrument with a frequency range listed in table 13 can be used to measure airblast. Many operators prefer to hire consultants to monitor airblast. Most of the discussion under ground vibration measurement techniques also applies to airblast measurement. Both peak reading instruments and those that record the entire airblast time history are available. The peak reading devices are satisfactory for regulatory compliance, but those that record the entire airblast time history are much better for troubleshooting purposes. A single airblast reading is taken at a given location. The gage should be 3 to 5 feet above the ground and should be at least 5 feet to one side of any structure to prevent distortion to the record due to airblast reflections.

Airblast can be measured by an operator-attended instrument or by a remotely installed instrument. Operator-attended instruments are cheaper but require the expense of the operator. These instruments are more flexible in that data can be recorded at different locations for different blasts. Remotely installed instruments are useful in that they record each blast fired without requiring an operator each time. One disadvantage of remote monitoring is that a high reading can be induced by a loud noise near the instrument. For this reason, instruments that record the entire airblast event are recommended for remote monitoring, so that a nonblasting event can be identified by its noncharacteristic wave trace.

All airblast monitors should be equipped with wind screens to cut down the background noise level and protect the microphone. Remotely installed instruments should be protected from the weather.

Airblast recordings provide good evidence in case of complaints or lawsuits. Airblast readings taken in conjunction with ground vibration readings are especially helpful in determining which of the two are the primary cause of complaints.

Reducing Airblast

Properly executed blasts, where surface explosives are adequately covered and borehole charges are adequately confined, are not likely to produce harmful levels of airblast. Close attention must be paid to geology to prevent the occasional "one that gets away from you."

The following techniques can be used to reduce airblast.

1. Unconfined explosives should not be used. Where surface detonating cord is used it should be buried. Cords with lighter core loads require less depth of burial.
2. Sufficient burden and stemming on the blastholes are essential. Where the length of stemming is marginal, coarse stemming material will give better charge confinement than fines, particularly where there is water in the stemming zone. One-fourth-inch material makes excellent stemming. A longer stemming dimension should be used where part of the burden at the crest has been robbed from the front row of holes, which usually creates more airblast than subsequent rows.
3. Geologic conditions that cause blowouts should be compensated for. These include mud seams, voids, or open bedding (should be stemmed through) and solution cavities or other openings (a check of column rise will avoid overloading).
4. Holes must be drilled accurately to maintain the designed burden. This is especially important with inclined holes.
5. If there is a high free face in the direction of nearby built-up areas, the face should be reorientated, if possible, or reduced in height.
6. Collar priming should be avoided where airblast is a problem. (Collar priming is seldom desirable.)
7. Early morning, late afternoon, or night firing, when temperature inversions are most likely, should be avoided. Blasting when a significant wind is blowing toward nearby built-up areas will increase airblast.
8. Use of longer delays between rows than between holes in a row will promote forward rather than upward movement of the burden. Five milliseconds per foot of burden between rows is a good rule of thumb, but this should be increased in later rows for shots with many rows.
9. Exclusively long delays that may cause a hole to become unburdened before it fires should be avoided.

Public reaction to airblast can be reduced by blasting during periods of high activity such as the noon hour or shortly after school has been dismissed. Blasting during quiet periods should be avoided.

DUST AND GASES

Every blast generates some amount of dust and gases. The amounts of dust generated by blasting do not present a serious problem. Other phases of the mining operation such as loading, hauling, crushing, and processing produce considerably more dust than blasting. Even though a violent blast may produce more than the normal amount of dust, blasting is a relatively infrequent operation and, as a result, the total amount of dust produced in a day is insignificant when compared to other sources. Well-controlled blasts create little or no dust. Because dust in the muckpile can be a problem to mine personnel, it is common practice to thoroughly wet the muckpile before and during mucking operations. In underground operations, an appropriate amount of time is allowed for the dust to settle or to be expelled from the area by the ventilation system before miners enter the blast area.

The most common toxic gases produced by blasting are carbon monoxide and nitrogen oxides. While these gases are considered toxic at levels of 50 ppm and 5 ppm, respectively, blast fumes are quickly diluted to below these levels by the ventilation systems in underground mines and by natural air movement in surface mines. In underground operations, it is important to allow time for

toxic gases to be expelled by the ventilation system before miners enter the area. In surface mining, it is a good idea to wait for a short period of time before entering the immediate blast area, particularly if orange-brown fumes (nitrogen oxides) are present. It is extremely rare for significant concentrations of toxic gases to leave the mine property. If large amounts of orange-brown fumes are consistently present after blasts, the source of the problem should be determined and corrected. The primary causes of excessive nitrogen oxides are poor blasting agent mixtures, degradation of blasting agents during storage, use of non-water-resistant products in wet blastholes, and inefficient detonation of the blasting agent due to loss of confinement.

REFERENCES

1.   Bauer, A., and J.W. Sanders. Good Blasting Techniques and Public Relations. Min. Cong. J., v. 54, No. 11, November 1968, pp. 81-85.
2.   Dick, R.A. A Review of the Federal Surface Coal Mine Blasting Regulations. Proc. 5th Conf. on Explosives and Blasting Technique, St. Louis, MO, Feb. 7-9, 1979. Society of Explosives Engineers, Montville, OH, pp. 1-7.
3.   Dick, R.A., and D.E. Siskind. Ground Vibration Technology Pertaining to OSM Regulations. Proc. Symp. on Surface Coal Mining and Reclamation Coal Conf. & Expo., Louisville, KY, Oct. 23-25, 1979. McGraw-Hill, New York, pp. 13-18.
4.   E.I. du Pont de Nemours & Co., Inc. (Wilmington, DE). Blaster's Handbook, 16th ed., 1978, 494 pp.
5.   Lundborg, N., P.A. Persson, A. Ladegaard-Pederson, and R. Holmberg. Keeping the Lid on Flyrock From Open Pit Blasting. Eng. and Min. J., v. 176, No. 5, May 1975, pp. 95-100.
6.   Miller, P. Blasting Vibration and Airblast. Atlas Powder Co. (Dallas, TX), 16 pp.
7.   Nicholls, H.R., C.F. Johnson, and W.I. Duvall. Blasting Vibrations and Their Effects on Structures. BuMines B 656, 1971, 105 pp.
8.   Siskind, D.E., V.J. Stachura, M.S. Stagg, and J.W. Kopp. Structure Response and Damage Produced by Airblast From Surface Mining. BuMines RI 8485, 1980, 111 pp.
9.   Siskind, D.E., M.S. Stagg, J.W. Kopp, and C.H. Dowding. Structure Response and Damage Produced by Ground Vibration From Surface Mine Blasting. BuMines RI 8507, 1980, 74 pp.
10. Siskind, D.E., and C.R. Summers. Blast Noise Standards and Instrumentation. BuMines TPR 78, May 1974, 18 pp.
11. Stachura, V.J., D.E. Siskind, and A.J. Engler. Airblast Instrumentation and Measurement Techniques for Surface Mine Blasting. BuMines RI 8508, 1981, 53 pp.
12. Stagg, M.S., and A.J. Engler. Measurement of Blast-Induced Ground Vibrations and Seismograph Calibration. BuMines RI 8506, 1980, 62 pp.
13. Winzer, S.R. The Firing Times of MS Delay Blasting Caps and Their Effect on Blasting Performance. Prepared for National Science Foundation (NSF APR 77-05171). Martin Marietta Laboratories (Baltimore, MD), June 1978, 36 pp.; available for consultation at Bureau of Mines Twin Cities Research Center, Minneapolis, MN.

CHAPTER 5
REVIEW QUESTIONS

1. According to Montana's Permanent Program, the peak particle velocity may
not exceed _____ inch(es) per second at a man-made building located
6,000 feet away from the blast site.

2. A scale distance (DS) factor of _____ in the
equation $W = (D/SD)^2$ is used to make sure particle velocity will be less
than 1.0 inch per second.

3. Any homeowner within _____ of a surface mine permit area
can request a preblast survey.

4. Using the scale distance formula, determine the maximum weight of
explosives that may be detonated if the nearest structure is 1,000 feet
away. (Use Appendix B)

_____

5. Find the maximum amount of explosives that can be detonated if the
nearest home is 5,250 feet from the blast. (Use Appendix B)

_____

6. A shot design consisting of 50 holes is loaded with 252 pounds of explo-
sives per hole. The nearest structure is 2,000 feet away. What is the
maximum number of holes that can be detonated in any one delay period?
(Use Appendix B)

_____

7. What three mechanisms cause airblasts?

a. _____

b. _____

c. _____

8. List four factors that a blaster should take into account when trying to
control airblast.

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a. _____   c. _____

b. _____   d. _____

9. What are two causes of excessive undesirable side effects?

    a. _____

    b. _____

10. What are two purposes of the preblast survey?

    a. _____

    b. _____

11. What blasting problem is a leading cause of onsite fatalities? _____

    _____

12. What are zones of weakness and what happens to explosives loaded in a

    zone of weakness? _____

13. Montana regulations prohibit flyrock:

    a. _____

    b. _____

14. How are ground vibrations measured? _____

15. Factors which affect the level of ground vibrations include:

    a. _____

    b. _____

    c. _____

    d. _____

    e. _____

16. What vibration limits are important? _____

17. Airblast levels can be measured using: _____.

18. How do you determine whether an airblast or a ground vibration is the

    primary cause of a complaint? _____

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Chapter 6:   Blasting Safety

The following is a discussion of good, safe blasting procedures, moving chronologically from initial explosive storage through postshot safety measures. In addition to these procedures, the blaster must become familiar with all the safety regulations which govern the operation. These safety regulations contain additional advice on safe operating procedures for all phases of the blasting operation. The safety procedures discussed here are not meant to be, nor should be considered to be, a substitute for adherence to safety regulations. All general workplace safety recommendations also apply to blasting activities.

The Institute of Makers of Explosives (IME) has published an excellent series of safety publications (5-13). The National Fire Protection Association (NFPA) has published recommendations on the storage and handling of ammonium nitrate and blasting agents (14-16).

EXPLOSIVES STORAGE

The Bureau of Alcohol, Tobacco and Firearms (BATF) regulates explosive importation, manufacture, distribution and storage, including proper record-keeping to protect the public from misuse. Safe storage of explosives in the mining industry, including BATF regulations, is enforced by the Mine Safety and Health Administration (MSHA). In all other industries, safe explosive storage is regulated by BATF and the Occupational Safety and Health Administration (OSHA). In addition, most states, and many county and local government agencies, enforce their own explosive safety regulations.

Magazines for explosive storage must conform to specifications laid down by BATF and MSHA or OSHA. IME Pamphlet No. 1 (13) gives recommended standards for magazine construction. Magazines must be separated from each other, surrounding buildings, and rights-of-way according to the American Table of Distances (IME Pamphlet No. 2 [5]). Separation distance requirements between ammonium nitrate and blasting agent storage facilities are less than for high explosives. However, the distance requirements for separation of blasting agents and ammonium nitrate from occupied structures and rights-of-way are the same as those for high explosives. Detonators may not be stored with other explosive materials. High explosives must be stored in a type 1 (BATF) or type 2 magazine. Blasting agents may be stored in a type 1 magazine with high explosives. When explosives and blasting agents are stored together, all of the material in the magazine is considered to be high explosives for separation distance purposes. Blasting agents may be stored in any approved magazine.

Except when explosives are being deposited or withdrawn, magazines must be kept locked. Only authorized personnel should deposit or withdraw explo-

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sives. The number of authorized persons should be kept to a minimum for both safety and security purposes. In this way, accountability problems can be minimized. Explosive stocks should be piled nearly (figure 99) to facilitate safe handling, and the oldest explosives should be used first to assure freshness. This is important for all explosive materials but especially for AN-FO, to prevent fuel segregation or evaporation. Segregation and evaporation of fuel from AN-FO is a particular problem in bulk storage (figure 100).
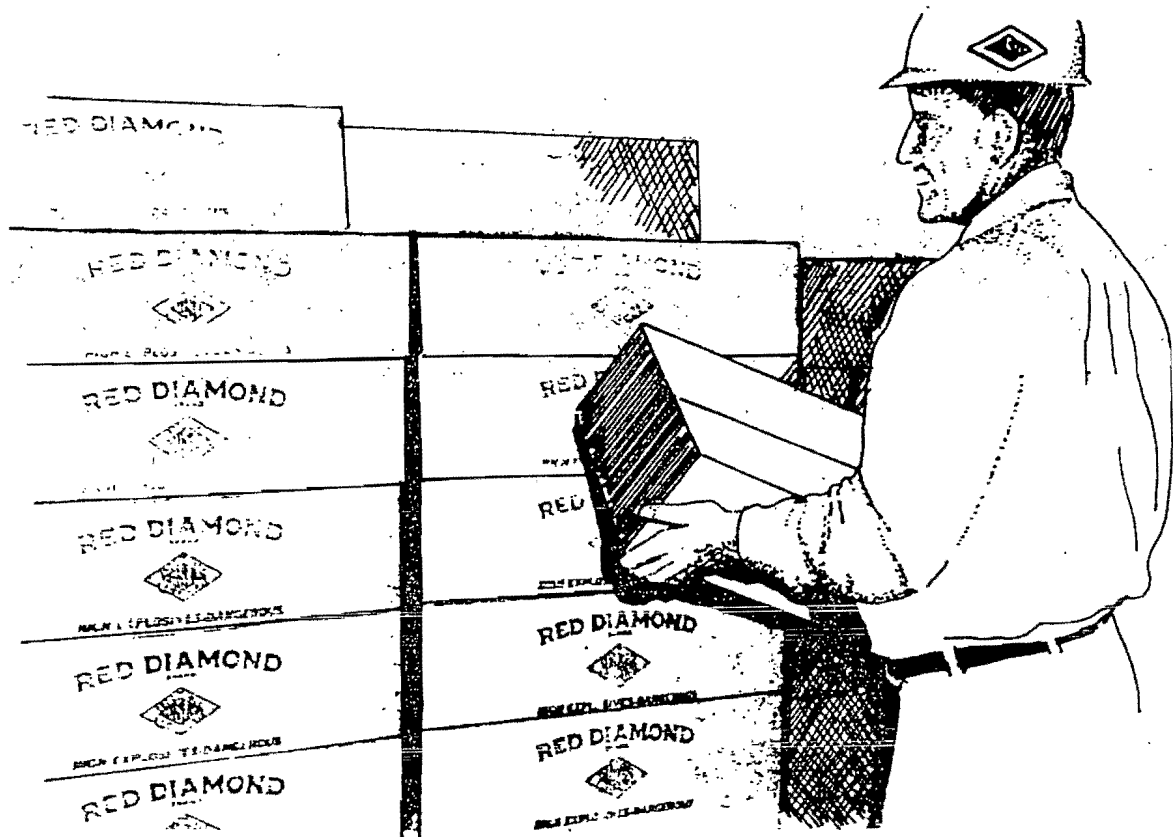
Figure 99: Proper stacking of explosives.

Prolonged storage should be avoided. Good housekeeping standards should be maintained both inside and outside the magazine. To minimize the fire hazard, vegetation outside the magazine, except live trees over 10 feet high, should be cleared for a distance of at least 25 feet and rubbish should be cleared for at least 50 feet. Smoking or flames are not permitted in or within 50 feet of an outdoor storage magazine. Magazines should be clearly marked. The IME recommends a sign stating "Explosives--Keep Off" in 3-inch-high letters with a 1/2-inch brush stroke. The explosives sign should be placed so that a bullet passing through the sign will not strike the magazine.

Primed explosives must never be stored in magazines. Misfired explosives should be disposed of immediately or stored in a separate magazine while awaiting disposal assistance. Assistance in disposing of deteriorated or unwanted explosives will be provided by the explosives distributor upon request.
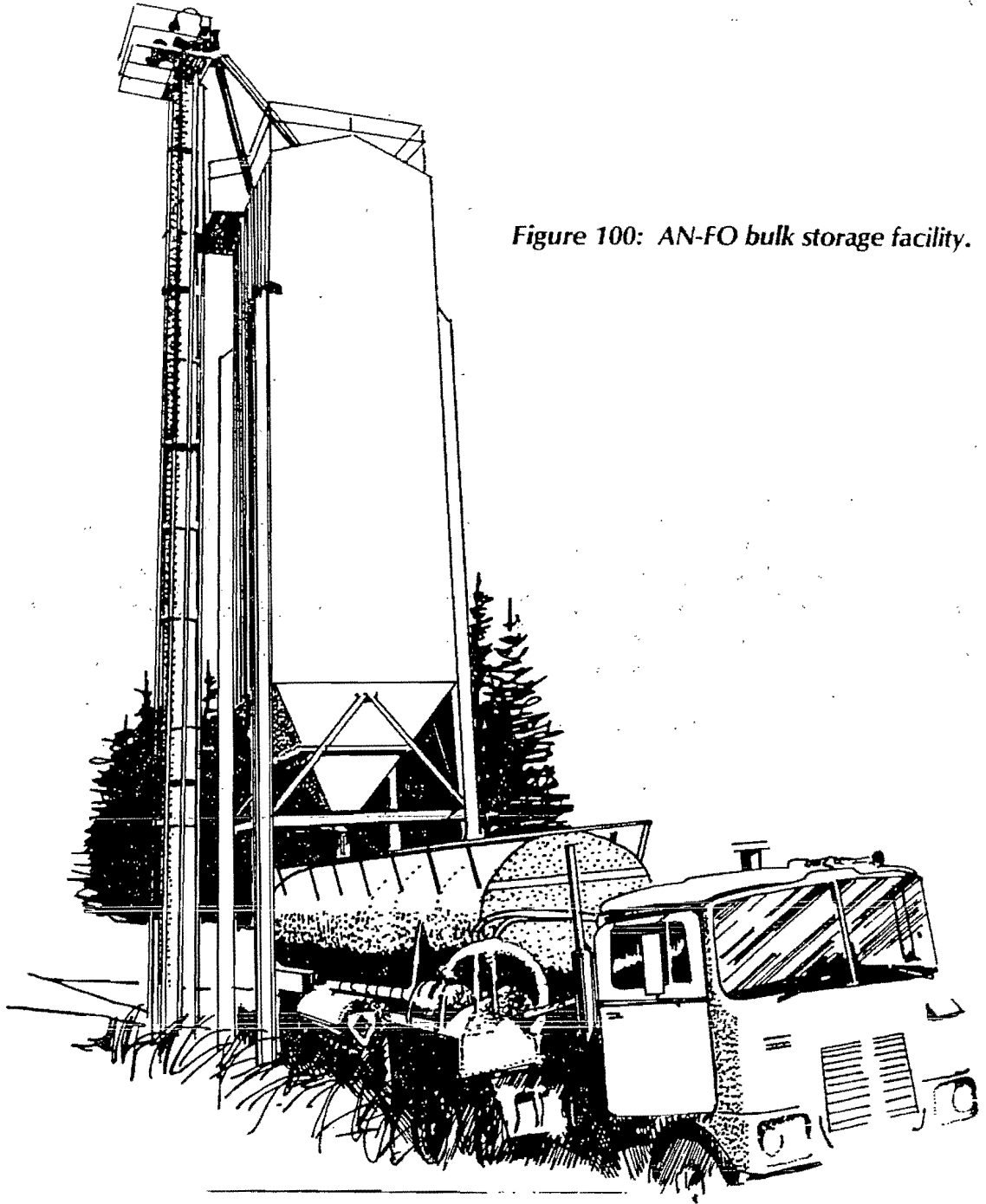
Figure 100: AN-FO bulk storage facility.

TRANSPORTATION FROM MAGAZINE TO JOBSITE

If the route from the magazine to the jobsite leaves company property, the transporter is subject to all state and local transportation regulations regarding vehicle specifications, placarding, and other operational procedures.

Explosives transportation should be done only in an approved vehicle that is in good repair and especially outfitted for the job. The practice of using the most conveniently available vehicle for explosive transportation should be avoided. The interior of the explosives compartment must be constructed of nonsparking material. If detonators are to be hauled on the same vehicle as explosives, they must be properly separated. MSHA regulations require a mini-

mum separation by 4-inch hardwood or the equivalent. Detonators should be protected from electrical contact. Adequate fire fighting equipment should be kept on the vehicle at all times. Small fires that are clearly isolated from the explosive cargo should be fought. However, if fire reaches the explosive cargo, the vehicle should be abandoned and guarded at a safe distance because it may detonate.

The operator of the explosives vehicle should be well trained in both driving and explosives handling. Before moving out with the explosives load, the driver should make sure that the explosives cannot fall from the vehicle because frictional impact will readily initiate explosives. Explosives transport by rail and track equipment is particularly susceptible to the frictional impact hazard.

At the jobsite, the explosives should be stored in a safe location, away from traffic if possible. The blast area should be delineated with cones or cordoned off, and unauthorized persons should not be permitted within this area. Where appropriate, the explosives should be stored in an approved day magazine. Explosives should not be stored where they can be hit by falling rock or working equipment. Explosives should be under constant surveillance whenever they are not in a magazine.

PRECAUTIONS BEFORE LOADING

Before any loading activities are started, the blast area must be clearly marked with flags, cones, or other readily identifiable markers. All unnecessary equipment must be removed from this area. All persons not essential to the powder loading operation should leave. Observers should be under the control of a responsible person who will assure that they do not create a hazard by wandering about the area. Any electrical power that might create a hazard should be disconnected. Where electric blasting is being used and the presence of extraneous electricity is suspected, appropriate checks should be made with a blasters' multimeter (1) or a continuous ground current monitor should be used. Where extraneous electricity problems persist, a nonelectric initiation system should be used. Two-way radios in the near vicinity should be turned off when electric blasting is being used. IME Pamphlet No. 20 (10) gives safe transmitter distances as a function of the type and power of the transmitter.

PRIMER PREPARATION

It is a cardinal rule that primers be made up at the working face or as close to it as possible. The detonators and primer cartridges or cast primers should be brought in as separate components. The preparation of primers at a remote location and their transportation to the jobsite presents an undue hazard on the transportation route and should be permitted only where required by extenuating circumstances. In large tunnel projects, use of an outside primer makeup facility is often considered safer than making the primers at the face. All unused primers should be dismantled before removing them from the jobsite. Assembled primers containing detonators should never be stored.

A nonsparking tool should be used to punch the hole in the cartridge for cap placement. To assure control, the number of persons making up primers should be as few as practical. Electric hazards should be checked for if

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electric caps are being used. It is extremely important that the cap be fully imbedded into the cartridge and attached in such a way that it will not be dislodged when tension is put on the wires or tubes. A hard cartridge should not be rolled for softening. This will destroy the integrity of the cartridge and the cap may not stay fully imbedded. A good nonsparking powder punch should make an adequate hole in any cartridge without rolling it. The dangers of the cap falling out of the cartridge are twofold: (a) the cap may be struck during loading or tamping operations and cause a premature detonation, or (b) the cap may fail to initiate the primer when it is activated. When using electric caps with small-diameter explosive cartridges, the cartridge should be punched at the end for cap insertion and the leg wires should be fastened to the cartridge by a half hitch to remove the possibility of tension on the cap (figure 49).

The structure of larger cartridges may require punching the cap hole in the side. With cast primers, the cap is passed through the channel and into the cap well (figure 50). The leg wires may be taped to the cast primer for extra security. Primer preparation for other types of blasting caps, such as Nonel, Primadet, and Hercudet, is similar to that for electric blasting caps. However, because propagation through the tubing of some of these products may be hampered by sharp bends, taping the tubing to the cartridge is recommended rather than half hitching. The manufacturer should be consulted for recommendations.

Where detonating cord is connected directly to the primer cartridge, it should be secured with a tight knot, supplemented by half hitches. With a cast primer, detonating cord is passed through the channel and a knot is tied at the end of the cord to keep the primer from slipping off. Subsequent primers can be slid down the detonating cord. When using cap and fuse, a diagonal hole is made through the cartridge. The cap and fuse are passed through this hole and into a second hole made for cap emplacement. Sometimes the cap is placed into a single, diagonally placed side hole and the fuse is tied to the cartridge with string. With fuse that will withstand a 180° bend, end priming, similar to that used with electric blasting caps, may be used. Cast primers are not normally used with cap and fuse.

BOREHOLE LOADING

Before loading begins, the area should be doublechecked for unnecessary personnel and equipment. If electric caps are being used, possible electrical hazards should be doublechecked. If an electrical storm approaches at any time when explosives are present, the area must be vacated, regardless of whether electric detonators are being used. Weather reports, lightning detectors, or even static from AM radio receivers may serve as warning of approaching electrical storms. Before any detonators or explosives are brought into the blast area, all circuits in the immediate vicinity should be de-energized.

Before loading begins, each borehole should be checked for proper depth. This will help prevent excessive column buildup, which could result in inadequate stemming and excessive flyrock. In most situations, holes that are too deep should be partially backfilled. Short holes may require cleaning or redrilling.

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Using a weighted tape, the column buildup should be checked frequently during loading. With relatively short, small-diameter holes, a tamping pole can be used to check the depth and also to check for blockages. If the build-up is less than anticipated, this may result in a cavity packed full of explosive which may blow out violently when detonated. If the column builds up more quickly than expected, frequent checking will prevent overloading. Proper stemming length is described in the "Blast Design" chapter. As a general rule of thumb, the stemming should be 14 to 28 borehole diameters.

When loading small-diameter cartridges, a nonsparking tamping pole should be used. Although there are differences of opinion, there is a consensus that a cushion stick should not be used in small-diameter holes; therefore, the primer should be the first cartridge placed into the hole. The base of the cap should point toward the collar. The primer cartridge should never be slit and should be pushed into place firmly. It should never be tamped vigorously. Two or three cartridges may then be slit, placed as a column, and tamped firmly. The remaining cartridges may be slit and tamped firmly. Excessive tamping should never be done. Care should be taken not to damage the detonator's leg wires or tubes.

Cartridges are often loaded in large-diameter blastholes by dropping them down the hole. However, the primer cartridge and a cartridge or two above the primer should be lowered to prevent damage to the primer. Leg wires or tubes from detonators may also be prone to damage from dropped cartridges. "Wet bags" of AN-FO should not be dropped; they depend on the cartridge material for water resistance, and dropping them may break the package and cause water leakage and subsequent desensitization of the AN-FO. A potential problem in bulk loading of large-diameter blastholes is overloading. Here it is especially important to check the column rise frequently as loading progresses (figure 101).

When pneumatically loading blastholes with pressure pots or venturi loaders, over electric blasting cap leg wires, it is essential that the loader be properly grounded to prevent buildup of static electricity. This ground should not be to pipes, air lines, rails, or other fixtures that are good conductors of stray current. Extraneous electricity is also a potential hazard with nonelectric detonators. Plastic liners should not be used when pneumatically loading small blastholes; this increases the chance for static buildup. This is particularly hazardous with electric detonators. A semi-conductive loading hose with a minimum resistance of 1,000 ohms/foot and 10,000 ohms total resistance, and a maximum total resistance of 2,000,000 ohms should be used. Such a hose will permit a static charge to bleed off but will not allow stray currents to enter the borehole. Where extraneous electricity is a problem, or where it is illegal to load pneumatically over leg wires, a nonelectric initiation system should be used. This does not entirely eliminate the hazard, so the safeguards mentioned previously should still be followed.

HOOKING UP THE SHOT

The size of crew used to hook up the shot should be kept to an absolute minimum. A single person should be in charge of final checkout to assure that the hookup plan has been properly followed and that the blast is ready to fire.
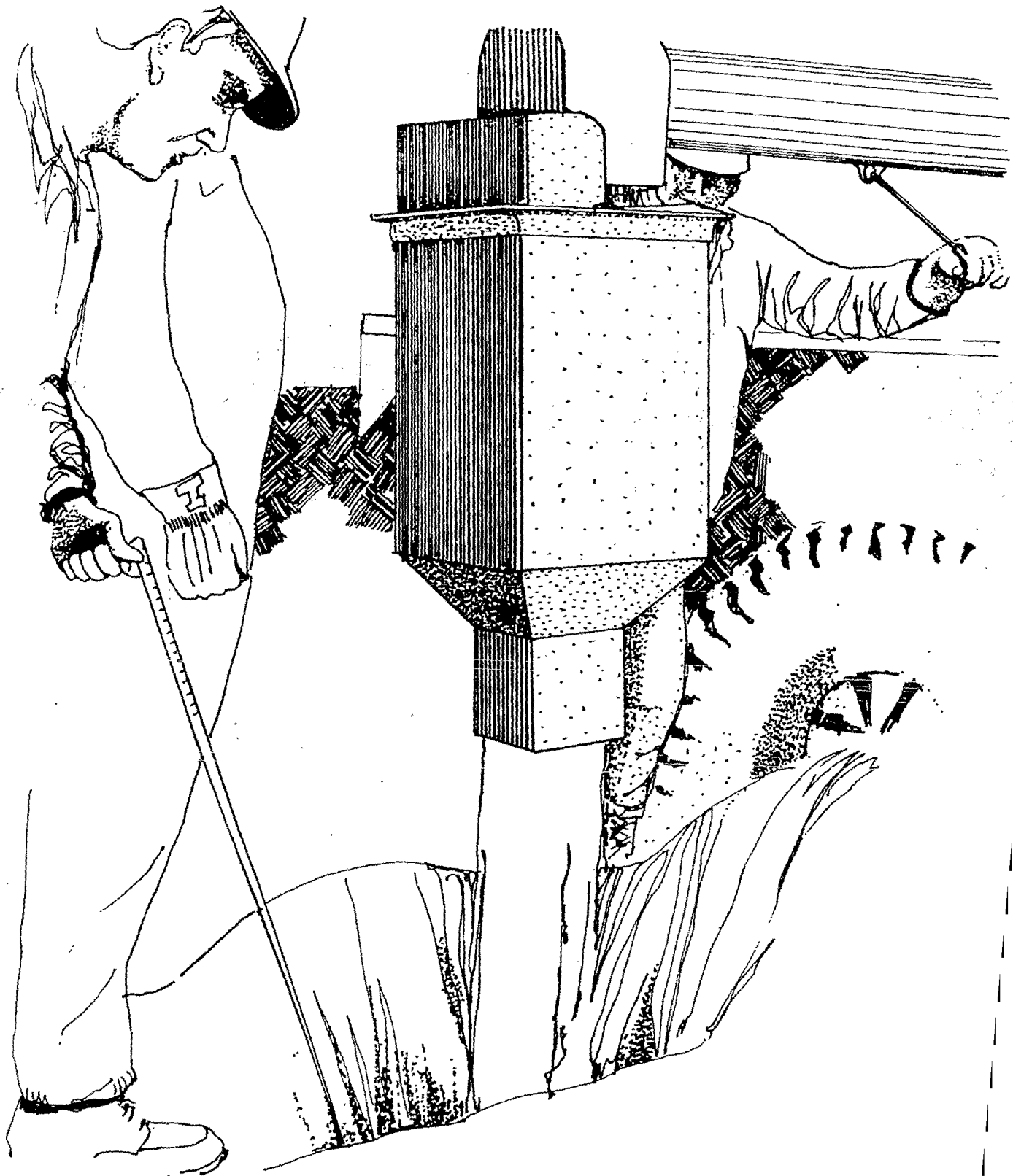
Figure 101: Checking the rise of the AN-FO column with a weighted tape.

When blasting electrically, the series circuit is the easiest, safest, and surest. If several shots are to be fired together, or if there is an excessive number of caps in one shot, a parallel series circuit should be used. Make sure that each series has the same resistance. A twisted loop is the best connection for two relatively light gage wires. Splices for connecting light gage wire to heavy gage wires are shown in figure 22. Excessive wire between holes may be coiled or removed for neatness and to facilitate visual inspection of the circuit. Make sure that bare connections do not touch each other or the ground in order to avoid short circuits, current leakage, or picking up extraneous currents.

After each portion of the circuit has been hooked up, check for continuity and proper resistance with a blasting multimeter or galvanometer. The circuit should then be shunted until ready for the final hookup before blasting. It is especially important that the lead wire be kept shunted at the shotfirer's location until the blast is ready to be fired.

A blasting machine is recommended for firing all shots. If a power line is used, it should be one that is specifically dedicated to blasting and is equipped with a safeguard against overenergizing the caps and against the resulting arcing. Batteries should never be used for firing electrical blasting circuits because their output is unpredictable and may cause only a portion of the round to be fired. Parallel circuits are less desirable because they require high current and cannot be checked for shorts or broken wires. If power line firing or straight parallel circuits are necessary, the cap manufacturer should be consulted for procedures for minimizing problems.

When firing with detonating cord systems, make sure the knots are tight and secure. Tight lines and severe angles between lines should be avoided (figure 32). The cord should not be permitted to cross itself. The cord circuit should be laid out so that each hole can be initiated by at least two paths from the detonator used to initiate the circuit. After the hookup has been completed, the circuit should be carefully checked visually by the person in charge of the blast. The initiating cap should not be connected to the detonating cord until it is time to blast.

When blasting with fuse, the use of Ignitacord is recommended for multiple hole blasts. A principal cause of fuse accidents is trying to light too many fuses at one time. Secondary causes are wet or deteriorated fuse and insufficient or improper lighting equipment. When using Ignitacord, all fuses should be the same length. The path of the Ignitacord will determine the delay sequence. The Ignitacord should not cross itself, because crosslighting is a possibility. At least two people must be present when lighting fuses. If fuses are being lit individually, no person may light more than 15 fuses. MSHA regulations specify burning times for fuses, depending on the number of fuses a person lights. The burning speed of fuse should be tested frequently. All fuse burns nominally at about 40 sec/foot. All fuses must be burning inside the hole before the first hole detonates.

Accident rates show that fuse blasting is inherently more hazardous than other initiation methods. Many of these incidents occur with highly experienced miners. It is recommended that, wherever practical, fuse blasting be replaced by an alternative initiation system. When using the more recently developed initiation systems such as Hercudet, Detaline, and Nonel, the

blaster should seek advice on the proper hookup procedures from the manufacturer or distributor. Certain aspects of these systems are still evolving and recommended procedures change from time to time.

SHOT FIRING

More people are injured and killed during the shot firing operation than any other phase of blasting. This is usually due to inadequate guarding, improper signaling, or some other unsafe practice that permits a person to be too close to the blast when it is detonated. It is essential that the blaster take positive steps to assure that no one, including the blaster and the crew, is in the area of potential flyrock at the time of detonation.

The blaster should allow adequate time immediately before blasting to inspect the blast area for any last minute problems. A fail-safe system should be in effect to assure that the blast is not inadvertently fired. This can be done by safeguarding the key or handle to the blasting machine or switch. While proceeding from the loaded shot toward the shotfiring location, the blaster should make sure that all connections between the blasting circuit and the firing mechanism are intact.

The blaster must make sure that there are enough guards to seal off the area and protect people from inadvertently proceeding into the blast area. It is common procedure to block access to the blast zone 5 to 10 minutes before the blast. The guards should proceed outward from the blast area, clearing all personnel from the area as they proceed. They should take up guard positions beyond the range of flyrock, concussion, and toxic gases. Once the area has been sealed off, the guards must permit no one to pass unless they first inform the shotfirer and receive assurance from the shotfirer that the blast will be postponed.

A warning siren with an audible range of about 0.5 mile should be sounded before the blast. However, signs or audible warnings alone are not dependable for keeping people out of the blast area. These types of warning may not be understood by all people in the area and they do not clearly delineate the hazardous area. Many underground mines have check-in and check-out procedures that are used to assure that no one will stray into the blast area. These systems reduce the number of guards required. The guards must be told if more than one blast is to be fired. Even after all blasts have been fired, it is important that the guards receive an audible or visual all-clear signal before allowing people to pass. If the guard is in doubt, the area should be kept secure until the doubt is removed.

The shotfirer should choose a safe firing location with adequate distance or cover (figure 102) for protection from flyrock, concussion, and toxic gases. Ideally, the shotfirer should have two-way visual or audible contact with the guards. On a surface blast, the shotfiring location should command a good view of the area surrounding the blast. Just before the shotfiring mechanism is prepared for activation, the blaster should alert the guards to seal off the area and should receive a positive response from each guard. Immediately before firing, the shot guards are again alerted and if their response is positive, the shot is fired. If the shot fails to fire, security must be maintained while the blaster attempts to correct the problem. Once security

is removed, the entire guarding procedure should be repeated before the shot is fired.
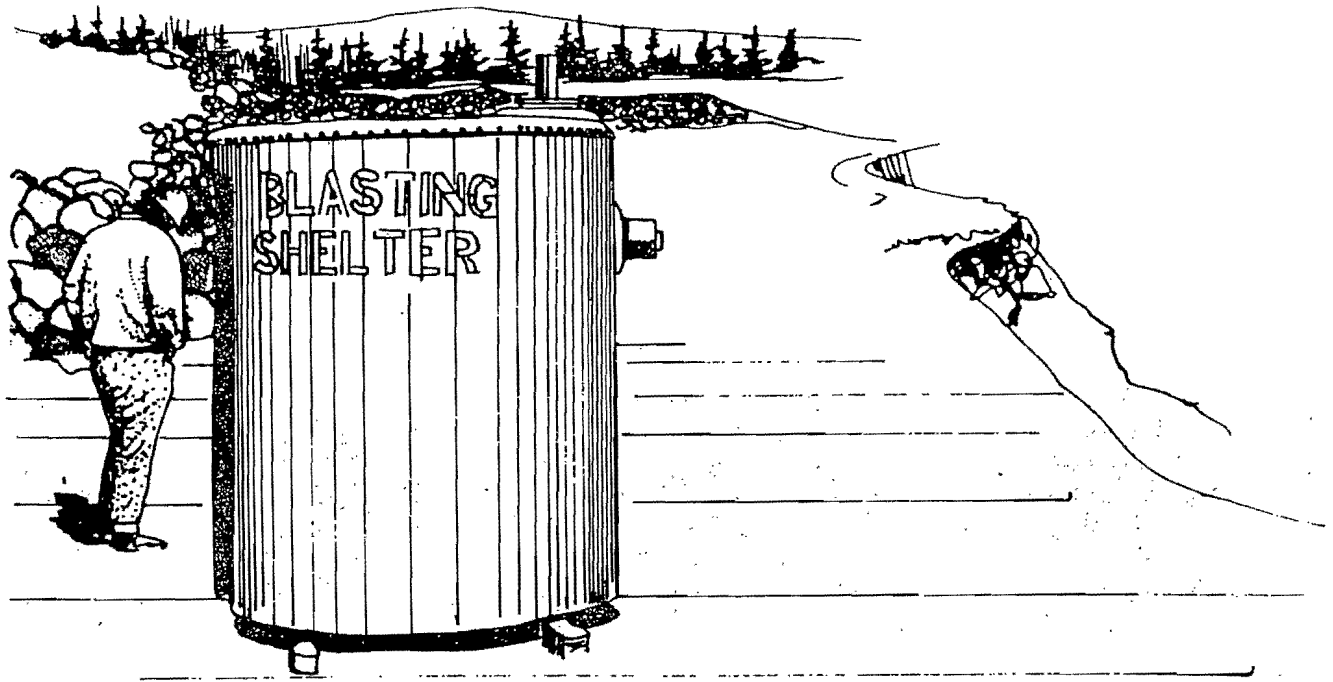
Figure 102: Blasting shelter.

In some situations, particularly underground, contact between the shot-firer and the guards may be impractical. In this case, the guards must clear and secure the area and maintain security until all shots are fired or until they are relieved of the responsibility by the blaster. This may mean guarding the area for an extended period of time.

Some situations will exist which will not fit the preceding discussion. The principles, however, will remain the same--(a) the blast area must be cleared and guarded and (b) security must be maintained until it is certain that the blasting activities in the area have ceased for the time in question.

Blasting at surface mines at night is especially hazardous because of the lack of visibility and should be done only in an emergency.

POSTSHOT SAFETY

At least 15 seconds should be allowed for all flyrock to drop. Even after all flyrock has subsided, the hazards of toxic gases and loose rock in the blast area exist. The area should not be reentered until the toxic gases have been dispersed. The time required for this may range from a minute for surface blasting to an hour or more for a poorly ventilated, underground open-ing. In case of a known or suspected misfire, a waiting period of at least 30 minutes with cap-and-fuse blasting or at least 15 minutes with electric initi-ation systems must be observed. If explosives are suspected to be burning in a blasthole, a 1-hour minimum waiting period must be observed. The practice of blasting between shifts is recommended because it avoids or minimizes guarding problems and allows gases to clear before reentry.

The first person reentering the blast area should inspect the area for loose rock that poses a hazard to personnel. The area should be dangered off until any loose rock has been barred down or otherwise taken care of. The blast area should be checked for misfires. Loose explosives or detonating cord in the muckpile often indicate a misfire. Leg wires, detonating cord, or tubes extending from a borehole may indicate a misfire. Another indicator is an area of the blast that has not broken or pulled properly, or an unusual shape of the muckpile. In many cases, this takes the form of an unusually long bootleg. Because a misfire is not always obvious, a trained eye is often required to detect one. Other people must not be permitted into the blast area until it is certain that no hazards exist.

DISPOSING OF MISFIRES

A good method of misfire disposal is to remove the undetonated charge by water flushing or air pressure. Horizontal or shallow holes are most amenable to this technique. It is important to visually inspect the hole using a light source to assure that all of the charge has been removed.

Where removal of the misfired charge is too difficult, an alternative is to detonate the charge. If leg wires, tubes, or detonating cord are protruding from the holes, and they are intact, they may be reconnected and fired. If this cannot be done, the stemming may be removed, a new primer inserted at the top of the powder column, and the hole refired. Caution must be exercised in refiring misfired holes from which much of the burden has been removed. Excessive flyrock is likely to result and the area must be guarded accordingly. If neither of these alternatives are feasible, the charge will have to be dug out. First, the hole should be flooded with water to desensitize any non-water-resistant explosive present. Next, the rock surrounding the misfire is dug out carefully, with an observer present to guide the excavator. Extreme discretion must be exercised in this operation.

The practice of drilling and shooting a hole adjacent to the misfire has been used, but can be extremely hazardous. People have been killed using this technique because the new hole intersected the misfired hole and detonated it. All of the previously described techniques are preferable to drilling an adjacent hole. MSHA metal-nonmetal regulations prohibit drilling a hole where there is a danger of intersecting a charged or misfired hole.

DISPOSAL OF EXPLOSIVE MATERIALS

For years, the method recommended by the IME for destroying explosives was burning. However, the recent proliferation of nonflammable explosive products has caused the IME to withdraw this recommendation and its pamphlet that described proper burning techniques. The recommendation now made by the IME is to contact the nearest explosive distributor, whether or not it handles the brand of explosive in question. The distributor should dispose of the unwanted explosive.

PRINCIPAL CAUSES OF BLASTING ACCIDENTS

Although there is a potential for serious accidents at every stage of explosive use, certain aspects of blasting have more accident potential than

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others. Case history articles describing typical blasting accidents have been written (2-3). Avoiding the following four types of accidents, listed in approximate order of frequency, would significantly improve the safety record of the blasting industry.

Improper Guarding. This includes improper guarding of the blast area or blasting crew members taking inadequate cover. Many people underestimate the potential range of flyrock.

Impacting Explosives. Most often, this involves drilling into holes containing explosives, frequently bootlegs. However, it may involve striking explosives with excavator buckets, tracked equipment, or rail equipment, or excessive beating on explosives with a tamping pole.

Unsafe Cap-and-Fuse Practices. For various reasons, all involving unsafe acts or carelessness, the blaster is still in the vicinity of the blast when it detonates.

Extraneous Electricity. Exposure of electric blasting caps to stray ground current, static buildup, radio frequency energy, inductive coupling, or improper test instruments can cause unscheduled detonation. Lightning is a hazard with all types of explosive materials.

Other causes of accidents include explosive fires that detonate (hang-fires), poor warning systems, loading hot holes, and exposure to blast fumes.

UNDERGROUND COAL MINE BLASTING

All underground coal mine blasting is done electrically. The foregoing discussion applies to underground coal mine blasting. There are additional hazards caused by the potentially explosive atmosphere present in underground coal mines. Both methane and coal dust present an explosion hazard. As a result, underground coal mine blasters must undergo rigorous, specialized training before they can become qualified shotfirers. Because of its specificity, a discussion of underground coal mine blasting safety is beyond the scope of this manual. An excellent pocket-size pamphlet (4) is available from Hercules, Inc., which gives recommended procedures for underground coal mines. Additional information concerning underground blasting safety can be obtained through the Montana Department of Labor and Industry, Workers' Compensation Division, Bureau of Safety and Health.

REFERENCES

1. Atlas Powder Co. (Dallas, TX). Handbook of Electric Blasting. Rev. 1976, 93 pp.
2. Dick, R.A., and J.G. Gill. Metal and Nonmetal Mine Blasting Accidents During 1975-1976. Min. Eng., v. 29, No. 11, November 1977, pp. 36-39.
3. _____ . Recent Blasting Fatalities in Metal-Nonmetal Mining. Pit and Quarry, v. 67, No. 12, June 1975, pp. 85-87.
4. Hercules, Inc. (Wilmington, DE). Shotfirer's Guide. 1978. 12 pp.
5. Institute of Makers of Explosives Safety Library (Washington, DC). The American Table of Distances. Pub. No. 2, May 1983, 17 pp.
6. _____ . Do's and Don'ts Instructions and Warnings. Pub. No. 4, December 1983, 19 pp.

7. _____ . Glossary of Commercial Explosives Industry Terms. Pub. No. 12, January 1985, 31 pp.

8. _____ . Recommendations for the Safe Transportation of Detonators in a Vehicle with Certain Other Explosive Materials. Pub. No. 22, January 1985, 8 pp.

9. _____ . Transportation and Distribution Handbook. Pub. No. 14, January 1985, 56 pp.

10. _____ . Safety Guide for the Prevention of Radio-Frequency Radiation Hazards in the Use of Electric Blasting Caps. Pub. No. 20, September 1981, 23 pp.

11. _____ . Safety in the Transportation, Storage, Handling, and Use of Explosives. Pub. No. 17, January 1984, 38 pp.

12. _____ . Suggested Code of Regulations for the Manufacture, Transportation, Storage, Sale, Possession and Use of Explosive Materials. Pub. No. 3, January 1985, 41 pp.

13. _____ . Construction Guide for Storage Magazines. Pub. No. 1, January 1983, 22 pp.

14. National Fire Protection Association (Boston, MA). Manufacture, Storage, Transportation and Use of Explosives and Blasting Agents--1973. Pamphlet 495, 1973, 69 pp.

15. _____ . Separation of Ammonium Nitrate Blasting Agents From Explosives--1976. Pamphlet 492, 1976, 10 pp.

16. _____ . Storage of Ammonium Nitrate--1975. Pamphlet 490, 1975, 22 pp.

CHAPTER 6
REVIEW QUESTIONS

1. An area of at least _____ feet around magazines must be kept clean of dry leaves, grass, undergrowth, trash and other debris.

2. Except when explosives are being deposited or withdrawn, magazines must be kept _____.

3. How many explosives and detonators may be transported in the same vehicle? _____

4. What type of material must the interior of an explosive compartment be made of? _____

5. How must the blast area be marked? _____

6. After placing a cap in a primer, what standard method is used to insure that it does not fall out? _____

7. A warning signal should have an audible range of _____ mile(s).

8. If a misfire is too difficult to remove, what would be the safest way to dispose of it? _____

9. Dynamite punches must be made out of _____

10. Detonators may/may not be stored with other explosive material.

11. How does one assure the freshness of explosives? _____

12. Do any regulations apply when the route to the magazine from the jobsite leaves permitted property? _____ If so, what? _____

13. Under what conditions are observers allowed on a blastsite? _____

14. What is the effect of rolling a hard cartridge? _____

15. An unused primer should be dismantled when? _____

16. Can primers with detonators be stored? _____

17. Why should borehole depth be checked before loading? _____

_____

18. Why can't batteries be used for firing circuits? _____

_____

19. Flyrock is a leading cause of fatalities, but more people are killed and

    injured during what phase of blasting than any other? _____

_____

20. What is crosslighting? _____

21. What is the function of guards? _____

22. Why is blasting at night hazardous? _____

23. What is the minimum amount of time it takes for all flyrock to drop?

_____

_____

24. What are three clues to a misfire?

    a. _____

    b. _____

    c. _____

25. What is the most lethal activity adjacent to a misfire?

_____

26. What are four principal causes of blasting accidents?

    a. _____

    b. _____

    c. _____

    d. _____

BIBLIOGRAPHY

1.   Andrews, A.B. Design of Blasts. Emphasis on Blasting. Ensign Bickford Co. (Simsbury, CN), Spring 1980, pp. 1,4.

2.   Ash, R.L. The Mechanics of Rock Breakage, Parts I, II, III, and IV. Pit and Quarry, v. 56, No. 2, August 1963, pp. 98-112; No. 3, September 1963, pp. 118-123, No. 4, October 1963, pp. 126-131; No. 5, November 1963, pp. 109-111, 114-118.

3.   Atlas Powder Co. (Dallas, TX). Handbook of Electric Blasting. Rev. 1976, 93 pp.

4.   _____. Pneumatic Loading of Nitro-Carbo-Nitrates; Static Electricity, Fumes, and Safe Handling. Undated, 17 pp.

5.   Bauer, A., and J.W. Sanders. Good Blasting Techniques and Public Relations. Min. Cong. J., v. 54, No. 11, November 1968, pp. 81-85.

6.   Chironis, N.P. New Blasting Machine Permits Custom-Programmed Blast Patterns. Coal Age, v. 79, No. 3, March 1974, pp. 78-82.

7.   Condon, J.L. and J.J. Snodgrass. Effects of Primer Type and Borehole Diameter on AN-FO Detonation Velocities. Min. Cong. J., v. 60, No. 6, June 1974, pp. 46-47, 50-52.

8.   Cook, M.A. Explosives--A Survey of Technical Advances. Ind. and Eng. Chem., v. 60, No. 7, July 1968, pp. 44-55.

9.   _____. The Science of Industrial Explosives. Ireco Chemicals, Salt Lake City, UT, 1974, 449 pp.

10. Damon, G.H., C.M. Mason, N.E. Hanna, and D.R. Forshey. Safety Recommendations for Ammonium Nitrate-Based Blasting Agents. BuMines IC 8746, 1977, 31 pp.

11. Dannenberg, J. Blasthole Dewatering Cuts Costs. Rock Products, v. 76, No. 12, December 1973, pp. 66-68.

12. _____. How to Solve Blasting Materials Handling Problems. Rock Products, v. 74, No. 9, September 1971, pp. 63-65.

13. Dick, R.A. Explosives and Borehole Loading Subsection 11.7, SME Mining Engineering Handbook, ed. by A.B. Cummins and I.A. Given, Society of Mining Engineers of the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, v. 1, 1973, pp. 11-78--11-99.

14. _____. Factors in Selecting and Applying Commercial Explosives and Blasting Agents. BuMines IC 8405, 1968, 30 pp.

15. _____. The Impact of Blasting Agents and Slurries on Explosives Technology. BuMines IC 8560, 1972, 44 pp.

16. _____. New Nonelectric Explosive Initiation Systems. Pit and Quarry, v. 68, No. 9, March 1976, pp. 104-106.

17. _____. Puzzled About Primers for Large Diameter AN-FO Charges? Here's Some Help to End the Mystery. Coal Age, v. 81, No. 8, August 1976, pp. 102-107.

18. _____. A Review of the Federal Surface Coal Mine Blasting Regulations. Proc. 5th Conf. on Explosives and Blasting Technique, St. Louis, MO, February 7-9, 1979. Society of Explosives Engineers, Montville, OH, pp. 1-7.

19. Dick, R.A., and J.G. Gill. Metal and Nonmetal Mine Blasting Accidents During 1975-1976. Min. Eng., v. 29, No. 11, November 1977, pp. 36-39.

20. _____. Recent Blasting Fatalities in Metal-Nonmetal Mining. Pit and Quarry v. 67, No. 12, June 1975, pp. 85-87.

21. Dick, R.A., and J.J. Olson. Choosing the Proper Borehole Size for Bench Blasting. Min. Eng., v. 24, No. 3, March 1972, pp. 41-45.

43. Johansson, C.H., and U. Langefors. Methods of Physical Characterization of Explosives. Proc. 36th Internat. Cong. of Ind. Chem., Brussels, v. 3, 1966, p. 610; available for consultation at Bureau of Mines Twin Cities Research Center, Minneapolis, MN.

44. Junk, N.M. Overburden Blasting Takes on New Dimensions. Coal Age, v. 77, No. 1, January 1972, pp. 92-96.

45. _____. Research on Primers for Blasting Agents. Min. Cong. J., v. 50, No. 4, April 1964, pp. 98-101.

46. Langefors, U., and B.A. Kihlstrom. The Modern Technique of Rock Blasting. John Wiley & Sons, Inc., New York, 1963, 405 pp.

47. Lundborg, N., P.A. Persson, A. Ladegaard-Pederson, and R. Holmberg. Keeping the Lid on Flyrock from Open Pit Blasting. Eng. and Min. J., v. 176, No. 5, May 1975, pp. 95-100.

48. Miller, P. Blasting Vibration and Airblast. Atlas Powder Co. (Dallas, TX), 16 pp.

49. Monsanto Co. (St. Louis, MO). Monsanto Blasting Products AN-FO Manual. Its Explosive Properties and Field Performance Characteristics. September 1972, 37 pp.

50. National Fire Protection Association (Boston, MA). Manufacture, Storage, Transportation and Use of Explosives and Blasting Agents--1973. Pamphlet 495, 1973, 69 pp.

51. _____. Separation of Ammonium Nitrate Blasting Agents From Explosives--1976. Pamphlet 492, 1976, 10 pp.

52. _____. Storage of Ammonium Nitrate--1975. Pamphlet 490, 1975, 22 pp.

53. Nicholls, H.R., C.F. Johnson, and W.I. Duvall. Blasting Vibrations and Their Effects on Structures. BuMines B 656, 1971, 105 pp.

54. Porter, D.D. Use of Fragmentation to Evaluate Explosives for Blasting. Min. Cong. J., v. 60, No. 1, January 1974, pp. 41-43.

55. Pugliese, J.M. Designing Blast Patterns Using Empirical Formulas. BuMines IC 8550, 1972, 33 pp.

56. Robinson, R.V. Water Gel Explosives--Three Generations. Canadian Min. and Met. Bull., v. 62, No. 692, December 1969, pp. 1317-1325.

57. Schmidt, R.L., R.J. Morrell, D.H. Irby, and R.A. Dick. Application of Large-Hole Burn Cut in Room-and-Pillar Mining, BuMines RI 7994, 1975, 25 pp.

58. Sengupta, D., G. French, M. Heydari, and K. Hanna. The Impact of Eliminating Safety Fuse from Metal/Nonmetal Mines (Contract J0295010, Sci. Applications, Inc.). BuMines OFR 61-81, August 1980, 11 pp.; NTIS PB 81-214386.

59. Siskind, D.E., V.J. Stachura, M.S. Stagg, and J.W. Kopp. Structure Response and Damage Produced by Airblast from Surface Mining. BuMines RI 8485, 1980, 111 pp.

60. Siskind, D.E., M.S. Stagg, J.W. Kopp, and C.H. Dowding. Structure Response and Damage Produced by Ground Vibration from Surface Mine Blasting. BuMines RI 8507, 1980, 74 pp.

61. Siskind, D.E., and C.R. Summers. Blast Noise Standards and Instrumentation. BuMines TPR 78, May 1974, 18 pp.

62. Stachura, V.J., D.E. Siskind, and A.J. Engler. Airblast Instrumentation and Measurement Techniques for Surface Mine Blasting. BuMines RI 8508, 1981, 53 pp.

63. Stagg, M.S., and A.J. Engler. Measurement of Blast-Induced Ground Vibrations and Seismograph Calibration. BuMines RI 8506, 1980, 62 pp.

APPENDIX A—Glossary of Terms Used in Explosives and Blasting

Acoustical impedance--The mathematical expression characterizing a material as to its energy transfer properties. The product of its unit density and its sonic velocity.

Adobe charge--See mud cap.

Airblast--An airborn shock wave resulting from the detonation of explosives. May be caused by burden movement or the release of expanding gas into the air. Airblast may or may not be audible.

Airdox--A system that uses 10,000 psi compressed air to break undercut coal. Airdox will not ignite a gassy or dusty atmosphere.

Aluminum--A metal commonly used as a fuel or sensitizing agent in explosives and blasting agents. Normally used in finely divided particle or flake form.

American Table of Distances--A quantity-distance table published by IME as pamphlet No. 2, which specifies safe explosive storage distances from inhabited buildings, public highways, passenger railways and other stored explosive materials.

Ammonium nitrate (AN)--The most commonly used oxidizer in explosives and blasting agents. Its formula is NH_4NO_3.

AN-FO--An explosive material consisting of ammonium nitrate and fuel oil. The most commonly used blasting agent.

Axial priming--A system for priming blasting agents in which a core of priming material extends through most or all of the blasting agent charge length.

Back break--Rock broken beyond the limits of the last row of holes.

Back holes--The top holes in a tunnel of drift round.

Base charge--The main explosive charge in a detonator.

BATF--Bureau of Alcohol, Tobacco and Firearms, U.S. Department of the Treasury, which enforces explosives control and security regulations.

Beds or bedding--Layers of sedimentary rock, usually separated by a surface of discontinuity. As a rule, the rock can be readily separated along these planes.

Bench--The horizontal ledge in a quarry face along which holes are drilled vertically. Benching is the process of excavating whereby terraces or ledges are worked in a stepped sequence.

Binary explosive--An explosive based on two nonexplosive ingredients, such as nitromethane and ammonium nitrate, which are shipped and stored separately and mixed at the jobsite to form a high explosive.

Black powder--A low explosive consisting of sodium or potassium nitrate, carbon, and sulfur. Black powder is seldom used today because of its low energy, poor fume quality, and extreme sensitivity to sparks.

Blast--The detonation of explosives to break rock.

Blast area--The area near a blast within the influence of flying rock missiles, or concussion.

Blaster--A qualified person in charge of a blast. Also, a person (blaster-in-charge) who has passed a test, approved by OSM, which certifies his or her qualifications to supervise blasting activities.

Blasters' galvanometer; blasters' multimeter--See galvanometer, multimeter.

Blasthole--A hole drilled in rock or other material for the placement of explosives.

Blasting agent--An explosive that meets prescribed criteria for insensitivity to initiation. For storage, any material or mixture consisting of a fuel and oxidizer, intended for blasting, not otherwise defined as an explo-

Cap--See detonator.

Capped fuse--A length of safety fuse to which a blasting cap has been attached.

Capped primer--A package or cartridge of cap-sensitive explosive which is specifically designed to transmit detonation to other explosives and which contains a detonator (MSHA).

Cap sensitivity--The sensitivity of an explosive to initiation, expressed in terms of an IME No. 8 test detonator or a fraction thereof.

Carbon monoxide--A poisonous gas created by detonating explosive materials. Excessive carbon monoxide is caused by an inadequate amount of oxygen in the explosive mixture (excessive fuel).

Cardox--A system that uses a cartridge filled with liquid carbon dioxide, which, when initiated by a mixture of potassium perchlorate and charcoal, creates a pressure adequate to break undercut coal.

Cartridge--A rigid or semirigid container of explosive or blasting agent of a specified length or diameter.

Cartridge count--The number of 1-1/4- by 8-inch cartridges of explosives per 50-lb case.

Cartridge strength--A rating that compares a given volume of explosive with an equivalent volume of straight nitroglycerin dynamite, expressed as a percentage.

Cast primer--A cast unit of explosive, usually pentolite or composition B, commonly used to initiate detonation in a blasting agent.

Chambering--The process of enlarging a portion of blasthole (usually the bottom) by firing a series of small explosive charges. Chambering can also be done by mechanical or thermal methods.

Chapman-Jouguet (C-J) plane--In a detonating explosive column, the plane that defines the rear boundary of the primary reaction zone.

Circuit tester--See galvanometer; multimeter.

Class A explosive--Defined by the U.S. Department of Transportation (DOT) as an explosive that possesses detonating or otherwise maximum hazard; such as, but not limited to, dynamite, nitroglycerin, lead azide, black powder, blasting caps, and detonating primers.

Class B explosive--Defined by DOT as an explosive that possesses flammable hazard; such as, but not limited to, propellant explosives, photographic flash powders, and some special fireworks.

Class C explosive--Defined by DOT as an explosive that contains Class A or Class B explosives, or both, as components but in restricted quantities. For example, blasting caps or electric blasting caps in lots of less than 1,000.

Collar--The mouth or opening of a borehole or shaft. To collar in drilling means the act of starting a borehole.

Collar distance--The distance from the top of the powder column to the collar of the blasthole, usually filled with stemming.

Column charge--A long, continuous charge of explosive or blasting agent in a borehole.

Commercial explosives--Explosives designed and used for commercial or industrial, rather than military, applications.

Composition B--A mixture of RDX or TNT which, when cast, has a density of 1.65 g/cm^3 and a velocity of 25,000 fps. It is useful as a primer for blasting agents.

Condenser-discharge blasting machine--A blasting machine that uses batteries or

Deflagration--A subsonic but extremely rapid explosive reaction accompanied by
     gas formation and borehole pressure, but without shock.
Delay blasting--The use of delay detonators or connectors that cause separate
     charges to detonate at different times, rather than simultaneously.
Delay connector--A nonelectric, short-interval delay device for use in delaying
     blasts that are initiated by detonating cord.
Delay detonator--A detonator, either electric or nonelectric, with a built-in
     element that creates a delay between the input of energy and the explosion
     of the detonator.
Delay electric blasting cap--An electric blasting cap with a built-in delay
     that delays cap detonation in predetermined time intervals, from milli-
     seconds up to a second or more, between successive delays.
Delay element--That portion of a blasting cap which causes a delay between the
     instant of application of energy to the cap and the time of detonation of
     the base charge of the cap.
Density--The weight per unit volume of explosive, expressed as cartridge count
     or grams per cubic centimeter. See loading density.
Department of Transportation (DOT)--A Federal agency that regulates safety in
     interstate shipping of explosives and other hazardous materials.
Detaline System--A nonelectric system for initiating blasting caps in which the
     energy is transmitted through the circuit by means of a low-energy
     detonating cord.
Detonating cord--A plastic-covered core of high-velocity explosive, usually
     PETN, used to detonate charges of explosives. The plastic covering, in
     turn, is covered with various combinations of textiles and waterproofing.
Detonation--A supersonic explosive reaction that propagates a shock wave
     through the explosive accompanied by a chemical reaction that furnishes
     energy to sustain the shock wave propagation in a stable manner. Detona-
     tion creates both a detonation pressure and a borehole pressure.
Detonation pressure--The head-on pressure created by the detonation proceeding
     down the explosive column. Detonation pressure is a function of the
     explosive's density and the square of its velocity.
Detonation velocity--See velocity.
Detonator--Any device containing a detonating charge that is used to initiate
     an explosive. Includes, but is not limited to, blasting caps, electric
     blasting caps, and nonelectric instantaneous or delay blasting caps.
Ditch blasting--See propagation blasting.
DOT--See Department of Transportation.
Downline--The line of detonating cord in the borehole which transmits energy
     from the trunkline down the hole to the primer.
Drilling pattern--See pattern.
Drop ball--Known also as a headache ball. An iron or steel weight held on a
     wire rope which is dropped from a height onto large boulders for the pur-
     pose of breaking them into smaller fragments.
Dynamite--The high explosive invented by Alfred Nobel. Any high explosive in
     which the sensitizer is nitroglycerin or a similar explosive oil.

Echelon pattern--A delay pattern that causes the true burden, at the time of
     detonation, to be at an oblique angle from the original free face.
Electric blasting cap--A blasting cap designed to be initiated by an electric
     current.
Electric storm--An atmospheric disturbance of intense electrical activity pre-
     senting a hazard in all blasting activities.

Fuse lighter--A pyrotechnic device for rapid and dependable lighting of safety fuse.

Galvanometer--(More properly called blasters' galvanometer.) A measuring instrument containing a silver chloride cell and/or a current limiting device which is used to measure resistance in an electric blasting circuit. Only a device specifically identified as a blasting galvanometer or blasting multimeter should be used for this purpose.

Gap sensitivity--A measure of the distance across which an explosive can propagate a detonation. The gap may be air or a defined solid material. Gap sensitivity is a measure of the likelihood of sympathetic propagation.

Gas detonation system--A system for initiating caps in which the energy is transmitted through the circuit by means of a gas detonation inside a hollow plastic tube.

Gelatin--An explosive or blasting agent that has a gelatinous consistency. The term is usually applied to a gelatin dynamite but may also be a water gel.

Gelatin dynamite--A highly water-resistant dynamite with a gelatinous consistency.

Generator blasting machine--A blasting machine operated by vigorously pushing down a rack bar or twisting a handle. Now largely replaced by condenser discharge blasting machines.

Grains--A system of weight measurement in which 7,000 grains equal 1 lb.

Ground vibration--A shaking of the ground caused by the elastic wave emanating from a blast. Excessive vibrations may cause damage to structures.

Hangfire--The detonation of an explosive charge at a time after its designed firing time. A source of serious accidents.

Heading--A horizontal excavation driven in an underground mine.

Hercudet--See gas detonation system.

Hertz--A term used to express the frequency of ground vibrations and airblast. One hertz is one cycle per second.

High explosive--Any product used in blasting which is sensitive to a No. 8 test blasting cap and reacts at a speed faster than that of sound in the explosive medium. A classification used by BATF for explosive storage.

Highwall--The bench, bluff, or ledge on the edge of a surface excavation. This term is most commonly used in coal strip mining.

Ignitacord--A cordlike fuse that burns progressively along its length with an external flame at the zone of burning and is used for lighting a series of safety fuses in sequence. Burns with a spitting flame similar to a Fourth-of-July sparkler.

IME--The Institute of Makers of Explosives. A trade organization dealing with the use of explosives, concerned with safety in manufacture, transportation, storage, handling, and use. The IME publishes a series of blasting safety pamphlets.

Initiation--The act of detonating a high explosive by means of a cap, a mechanical device, or other means. Also the act of detonating the initiator.

Instantaneous detonator--A detonator that contains no delay element.

Jet loader--A system for loading AN-FO into small blastholes in which the AN-FO is drawn from a container by the venturi principle and blown into the hole at high velocity through a semiconductive loading hose.

Minimum firing current--The lower current (amperage) that will initiate an electric blasting cap within a specified short interval of time.

Misfire--A charge, or part of a charge, which for any reason has failed to fire as planned. All misfires are dangerous.

Monomethylaminenitrate--A compound used to sensitize some water gels.

MS connector--A device used as a delay in a detonating cord circuit connecting one hole in the circuit with another or one row of holes to other rows of holes.

MSHA--The Mine Safety and Health Administration. An agency under the Department of Labor which enforces health and safety regulations in the mining industry.

Muckpile--A pile of broken rock or dirt that is to be loaded for removal.

Mud cap--Referred to also as adobe, bulldoze, blistering, or plaster shot. A charge of explosive fired in contact with the surface of a rock, usually covered with a quantity of mud, wet earth, or similar substance. No borehole is used.

Multimeter--(More properly called blasters' multimeter.) A multipurpose test instrument used to check line voltages, firing circuits, current leakage, stray currents, and other measurements pertinent to electric blasting. Only a meter specifically designated as a blasters' multimeter or blasters' galvanometer should be used to test electric blasting circuits. National Fire Protection Association (NFPA)--An industry-government association that publishes standards for explosive material and ammonium nitrate.

Nitrocarbonitrate--A classification once given to a blasting agent by DOT for shipping purposes. This term is now obsolete.

Nitrogen oxides--Poisonous gases created by detonating explosive materials. Excessive nitrogen oxides may be caused by an excessive amount of oxygen in the explosive mixture (excessive oxidizer), or by inefficient detonation.

Nitroglycerin (NG)--The explosive oil originally used as the sensitizer in dynamites represented by the formula $C_3H_5(ONO_2)_3$.

Nitromethane--A liquid compound used as a fuel in two-component (binary) explosives and as rocket fuel.

Nitropropane--A liquid fuel that can be combined with pulverized ammonium nitrate prills to make a dense blasting mixture.

Nitrostarch-A solid explosive, similar to nitroglycerin in function, used as the base of "nonheadache" powders.

Nonel--See shock tube system.

Nonelectric delay blasting cap---A detonator with a delay element, capable of being initiated nonelectrically. See shock tube system; gas detonation system; Detaline System.

No. 8 test blasting cap--See test blasting cap No. 8.

OSHA--The Occupational Safety and Health Administration. An agency under the Department of Labor which enforces health and safety regulations in the construction industry, including blasting.

OSM--The Office of Surface Mining Reclamation and Enforcement. An agency under the Department of Interior which enforces surface environmental regulations in the coal mining industry.

Overbreak--Excessive breakage of rock beyond the desired excavation limit.

Overburden--Worthless material lying on top of a deposit of useful materials.

Powder chest--A substantial, nonconductive, portable container equipped with a lid and used at blasting sites for temporary storage of explosives.

Powder factor--A ratio between the amount of powder loaded and the amount of rock broken, usually expressed as pounds per ton or pounds per cubic yard. In some cases, the reciprocals of these terms are used.

Preblast survey--A documentation of the existing condition of a structure. The survey is used to determine whether subsequent blasting causes damage to the structure.

Premature--A charge that detonates before it is intended. Prematures can be hazardous.

Preshearing--See presplitting.

Presplitting--A form of controlled blasting in which decoupled charges are fired in closely spaced holes at the perimeter of the excavation. A presplit blast is fired before the main blast. Also called preshearing.

Pressure vessel--A system for loading AN-FO into small-diameter blastholes. The AN-FO is contained in a sealed vessel, to which air pressure is applied, forcing the AN-FO through a semiconductive hose and into the blasthole. Also known as pressure pot.

Prill--In blasting, a small porous sphere of ammonium nitrate capable of absorbing more than 6 percent by weight of fuel oil. Blasting prills have a bulk density of 0.80 to 0.85 g/cm^3.

Primary blast--The main blast executed to sustain production.

Primary explosive--An explosive or explosive mixture, sensitive to spark, flame, impact or friction, used in a detonator to initiate the explosion.

Primer--A unit, package, or cartridge or cap-sensitive explosive used to initiate other explosives or blasting agents and which contains a detonator (MSHA).

Propagation--The detonation of explosive charges by an impulse from a nearby explosive charge.

Propagation blasting--The use of closely spaced, sensitive charges. The shock from the first charge propagates through the ground, setting off the adjacent charge, and so on. Only one detonator is required. Primarily used for ditching in damp ground.

Propellant explosive--An explosive that normally deflagrates and is used for propulsion.

Pull--The quantity of rock or length of advance excavated by a blast round.

Radiofrequency energy--Electrical energy traveling through the air as radio or electromagnetic waves. Under ideal conditions, this energy can fire an electric blasting cap. IME Pamphlet No. 20 recommends safe distances from transmitters to electric blasting caps.

Radiofrequency transmitter--An electric device, such as a stationary or mobile radio transmitting station, which transmits a radiofrequency wave.

RDX--Cyclotrimethylenetrinitramine, an explosive substance used in the manufacture of compositions B, C-3, and C-4. Composition B is useful as a cast primer.

Relievers--In a heading round, holes adjacent to the cut holes, used to expand the opening made by the cut holes.

Rib holes--The holes at the sides of a tunnel or drift round, which determine the width of the opening.

Rip rap--Coarse rocks used for river bank or dam stabilization to reduce erosion by water flow.

Rotational firing--A delay blasting system in which each charge successively

agent containing substantial portions of water (MSHA). See emulsion; water gel.

Smooth blasting--A method of controlled blasting, used underground, in which a series of closely spaced holes is drilled at the perimeter, loaded with decoupled charges, and fired on the highest delay period of the blast round.

Snake hole--A borehole drilled slightly downward from horizontal into the floor of a quarry face. Also, a hole drilled under a boulder.

Sodium nitrate--An oxidizer used in dynamites and sometimes in blasting agents.

Spacing--The distance between boreholes or charges in a row, measured perpendicular to the burden and parallel to the free face of expected rock movement.

Specific gravity--The ratio of the weight of a given volume of any substance to the weight of an equal volume of water.

Splitter cord--See Ignitacord.

Springing--See chambering.

Square pattern--A pattern of blastholes in which the holes in succeeding rows are drilled directly behind the holes in the front row. In a truly square pattern, the burden and spacing are equal.

Squib--A firing device that burns with a flash. Used to ignite black powder or pellet powder.

Stability--The ability of an explosive material to maintain its physical and chemical properties over a period of time in storage.

Staggered pattern--A pattern of blastholes in which holes in each row are drilled between the holes in the preceding row.

Static electricity--Electrical energy stored on a person or object in a manner similar to that of a capacitor. Static electricity may be discharged into electrical initiators, thereby detonating them.

Steady state velocity--The characteristic velocity at which a specific explosive, under specific conditions, in a given charge diameter, will detonate.

Stemming--The inert material, such as drill cuttings, used in the collar portion (or elsewhere) of a blasthole to confine the gaseous products of detonation. Also, the length of blasthole left uncharged.

Stick count--See cartridge count.

Stray current--Current flowing outside its normal conductor. A result of defective insulation, it may come from electrical equipment, electrified fences, electric railways, or similar items. Flow is facilitated by conductive paths such as pipelines and wet ground or other wet materials. Galvanic action of two dissimilar metals, in contact or connected by a conductor, may cause stray current.

Strength--A property of an explosive described in various terms such as cartridge or weight strength, seismic strength, shock or bubble energy, crater strength, ballistic mortar strength, etc. Not a well-defined property. Used to express an explosive's capacity to do work.

String loading--The procedure of loading cartridges end to end in a borehole without deforming them. Used mainly in controlled blasting and permissible blasting.

Subdrill--To drill blastholes beyond the planned grade lines or below floor level to insure breakage to the planned grade or floor level.

Subsonic--Slower than the speed of sound.

Supersonic--Faster than the speed of sound.

APPENDIX B: Montana Coal Blasting Rules and Regulations
(**NOTE:** The Administrative Rules of Montata for coal mining were updated in 2004.)

- Visit the following website for current rules related to coal mine blasting in Montana - *http://www.deq.state.mt.us/CoalUranium/BlastingInfo.asp*

- Visit the following website for current rules related to hard rock mine blasting in Montana - *http://www.deq.state.mt.us/hardrock/LawsRules.asp*

APPENDIX C: Montana Department of Labor and Industries

- Visit the following website for mine safety and health information from the Montana Mine Safety Bureau - *http://erd.dli.mt.gov/safetyhealth/minesh.asp*

APPENDIX D: Federal Blasting Programs

- Visit the following website for federal mine safety and health information from MSHA.GOV - *http://www.msha.gov/regsinfo.htm*

- Visit the following website for explosive safety and regulatory information from ATF - *http://www.atf.gov/explarson/index.htm*

- Visit the following website for mine permitting and reclamation information from OSMR.GOV - *http://www.osmre.gov/osmreg.htm*

APPENDIX B: Montana Rules and Regulations Pertaining to Coal Blasting

ARM 17.24.310

17.24.310 BLASTING PLAN (1) Each application must contain a blasting plan for the proposed permit area. The plan must explain how the applicant intends to comply with the requirements of ARM 17.24.621 through 17.24.626 and 17.24.1260 through 17.24.1263, and must include the following:

(a) types and approximate amounts of explosives to be used for each type of blasting operation to be conducted;

(b) description of procedures and plans generally used for:

(i) drilling patterns, including size, number, depths, and spacing of holes;

(ii) charge and packing of holes;

(iii) types of fuses and detonation controls;

(iv) sequence and timing of firing holes;

(v) a description of procedures and plans for recording of (i) through (iv) above and retention of those records;

(c) description of blasting warning and site access control equipment and procedures;

(d) description of types, capabilities, sensitivities, and locations of use of any blast monitoring equipment and procedures proposed to be used;

(e) description of plans for recording and reporting to the department the results of preblasting surveys, if required;

(f) description of unavoidable hazardous conditions for which deviations from the blasting schedule will be needed; and

(g) a general description of structures to be protected and a discussion of design factors to be used to protect the public and to meet the applicable airblast, flyrock, and ground vibration standards in ARM 17.24.624.

(2) For underground mines the department may, on a case-by-case basis, waive any of the requirements in (1) of this rule that do not apply to underground blasting operations. (History: 82-4-204, 82-4-205, MCA; IMP, 82-4-222, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 3042.)

ARM 17.24.621-626

17.24.621 GENERAL REQUIREMENTS FOR USE OF EXPLOSIVES

(1) Each operator shall comply with all applicable state and federal laws in the use of explosives.

(2) Blasts that use more than 5 pounds of explosive or blasting agent must be conducted according to the schedule required by ARM 17.24.623.

(3) All blasting operations must be conducted by experienced, trained, and competent persons who understand the hazards involved. Each person responsible for blasting operations must possess a valid certification. See ARM 17.24.1260 through 17.24.1263. (History: 82-4-204, 82-4-205, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 2852.)

17.24.622 PREBLASTING SURVEY (1)(a) At least 30 days before initiation of blasting, the operator shall advise, in writing, all residents or owners of dwellings or other structures within 1/2 mile of the permit area how to request a preblasting survey.

(b) Any survey requested more than 10 days before the planned initiation of blasting must be completed by the operator before the initiation of blasting.

(c) On the request to the department by a resident or owner of a dwelling or structure that is located within 1/2 mile of any part of the permit area, the operator must promptly conduct a preblasting survey of the dwelling or structure and promptly submit a report of the survey to the department and to the person requesting the survey. If a structure is renovated or added to subsequent to a preblasting survey, then upon request to the department a survey of such additions and renovations must be performed in accordance with this section.

(2) The survey must determine the condition of the dwelling or structure and document any preblasting damage and other physical factors that could reasonably be affected by the blasting. Assessments of structures such as pipes, cables, transmission lines, and wells and other water systems must be limited to surface condition and readily available data. Special attention must be given to the preblasting condition of wells and other water systems used for human, animal, or agricultural purposes and to the quantity and quality of the water.

(3) A written report of the survey must be prepared and signed by the person who conducted the survey. The report may include recommendations of any special conditions or proposed adjustments to the blasting procedure that should be incorporated into the blasting plan to prevent damage. Copies of the report must be provided to the person requesting the survey and to the department. If the person requesting the survey disagrees with the results of the survey, he or she may notify, in writing, both the permittee and the department of the specific areas of disagreement. (History: 82-4-204, 82-4-205, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 2852.)

17.24.623 BLASTING SCHEDULE (1) The operator shall publish a blasting schedule at least 10 days, but not more than 20 days, before beginning a blasting program in which blasts that use more than five pounds of explosive or blasting agent are detonated. The blasting schedule must be published once in a newspaper of general circulation in the locality of the blasting site.

(2) Copies of the schedule must be distributed by mail to local governments and public utilities and by mail or delivered to each residence within 1/2 mile of the permit area described in the schedule. For the purposes of this section, the permit area does not include haul or access roads, coal preparation and loading facilities, and transportation facilities between coal excavation areas and coal preparation or loading facilities, if blasting is not conducted in these areas. Copies sent to residences must be accompanied by information advising the owner or resident how to request a preblasting survey.

(3) The operator shall republish and redistribute the schedule by mail at least every 12 months.

(4) A blasting schedule must not be so general as to cover the entire permit area or all working hours, but it must identify as accurately as possible the location of the blasting sites and the time periods when blasting will occur.

(5) The blasting schedule must contain at a minimum:

(a) name, address and telephone number of the operator;

(b) identification of the township, range and section for specific areas in which blasting will take place;

(c) days and time periods when explosives are to be detonated;

(d) methods to be used to control access to the blasting area;

(e) types of audible warnings and all-clear signals to be used before and after blasting; and

(f) a description of unavoidable hazardous situations referred to in ARM 17.24.310(1)(f) that have been approved by the department for blasting at times other than those described in the schedule.

(6) Before blasting in areas or at times not in a previous schedule, the operator shall prepare a revised blasting schedule according to the procedures of (1). Whenever a schedule has previously been provided to the owner or residents under (1) with information on requesting a preblasting survey, the notice of change need not include information regarding preblast surveys.

(7) If there is a substantial pattern of non-adherence to the published blasting schedule as evidenced by the absence of blasting during scheduled periods, the department may require the operator to prepare a revised blasting schedule according to the procedures in (6). (History: 82-4-204, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; AMD, 1994 MAR p. 2957, Eff. 11/11/94; TRANS, from DSL, 1996 MAR p. 2852; AMD, 1999 MAR p. 811, Eff. 4/23/99; AMD, 1999 MAR p. 2168, Eff. 12/3/99; AMD, 2004 MAR p. 2548, Eff. 10/22/04.)

17.24.624 SURFACE BLASTING REQUIREMENTS (1) The department may limit the area covered, timing, and sequence of blasting, if such limitations are necessary and reasonable in order to protect the public health and safety or welfare.

(2) All blasting must be conducted between sunrise and sunset except that:

(a) The department may specify more restrictive time periods, based on public requests or other relevant information, according to the need to adequately protect the public from adverse noise or seismic disturbances.

(b) Blasting may, however, be conducted between sunset and sunrise if:

(i) a blast that has been prepared during the afternoon must be delayed due to the occurrence of an unavoidable hazardous condition and cannot be delayed until the next day because a potential safety hazard could result that cannot be adequately mitigated;

(ii) in addition to the required warning signals, oral notices are provided to persons within 1/2 mile of the blasting site; and

(iii) a complete written report of blasting at night is filed by the operator with the department not later than three days after the night blasting. The report must include a description in detail of the reasons for the delay in blasting including why the blast could not be held over to the next day, when the blast was actually conducted, the warning notices given, and a copy of the blast record required by ARM 17.24.626.

(3) Blasting must be conducted at times announced in the blasting schedule, except in those unavoidable hazardous situations, previously approved by the department in the permit application, whenever operator or public safety require unscheduled detonation. Any deviation from the times announced must be reported to the department not later than three days after the unavoidable blast. A complete description of the unavoidable hazardous situation must accompany the report.

(4) Warning and all-clear signals of different character that are audible within a range of 1/2 mile from the point of the blast must be given. Each person within the permit area and each person who resides or regularly works within 1/2 mile of the permit area must be notified of the meaning of the signals through appropriate instructions. These instructions must be periodically delivered or otherwise communicated in a manner that can be reasonably expected to inform such persons of the meaning of the signals. The operator shall maintain signs in accordance with ARM 17.24.524.

(5) Access to an area possibly subject to flyrock from blasting must be regulated to protect the public and livestock. Blasting must not eject flyrock onto property outside the permit area. Access to the area must be controlled to prevent the presence of livestock or unauthorized personnel during blasting and until an authorized representative of the operator has reasonably determined:

(a) that no unusual circumstances, such as imminent slides or undetonated charges, exist; and

(b) that access to and travel in or through the area can be safely resumed.

(6)(a) Airblast must be controlled so that it does not exceed the values specified below at any dwelling, or public, commercial, community or institutional building, unless the structure is owned by the operator and is not leased to any other person. If a building owned by the operator is leased to another person, the lessee may sign a waiver relieving the operator from meeting the airblast limitations of this section.

Lower Frequency limit of measuring   Maximum level in
system, Hertz (Hz) (+3dB)   decibels (dB)
0.1 Hz or lower - flat response . . . . . .134 peak.
2 Hz or lower - flat response . . . . . . .133 peak.
6 Hz or lower - flat response . . . . . . .129 peak.
C-weighted, slow response . . . . .105 peak dBC.

If necessary to prevent damage, the department shall specify lower maximum allowable airblast levels than those above.

(b) In all cases, except the C-weighted, slow-response system, the measuring systems used must have a flat frequency response of at least 200 Hz at the upper end. The C-weighted system must be measured with a Type 1 sound level meter that meets the standard American national standards institute (ANSI) S1.4-1971 specifications. The ANSI S1.4-1971 is hereby incorporated by reference as it exists on April 1, 1980. Copies
of this publication are on file with the Department of Environmental Quality, P.O. Box 200901, Helena, MT 59620-0901.

(c) The operator may satisfy the provisions of this section by meeting any of the four specifications in the chart in (a).

(d) The operator shall conduct periodic monitoring to ensure compliance with the airblast standards. The department may require an airblast measurement of any or all blasts, and may specify the location of such measurements, except as noted in (a).

(7) Except where lesser distances are approved by the department, based upon a preblasting survey, seismic investigation, or other appropriate investigation, blasting must not be conducted within:

(a) 1,000 feet of any dwelling, or public, commercial, community or institutional building;

(b) 500 feet of facilities including, but not limited to, disposal wells, petroleum or gas storage facilities, municipal water storage facilities, fluid transmission pipelines, gas or oil collection lines, or water and sewage lines or any active or abandoned underground mine.

(8) If otherwise approved, a blast design, including measures to protect the above facilities, must be submitted which contains the information required in ARM 17.24.310 and signed by a certified blaster.

(9) Flyrock, including blasted material traveling along the ground, must not be cast from the blasting vicinity more than half the distance to the nearest dwelling or other occupied structure and in no case beyond the line of property owned or leased by the permittee, or beyond the area of regulated access required under (5).

(10) Blasting must be conducted to prevent injury to persons, damage to public or private property outside the permit area, adverse impacts on any underground mine, and change in the course, channel, or availability of ground or surface waters outside the permit area.

(11) In all blasting operations, except as otherwise authorized in this section, the maximum peak particle velocity must not exceed the following limits at the location of any dwelling, or public, commercial, community or institutional building:

Distance (D) from the blasting site, in feet	Maximum allowable peak particle velocity (V max) for ground vibration, in inches/second	Scaled-distance factor to be applied without seismic monitoring Ds
0 to 300	1.25	50
301 to 5,000	1.00	55
5,001 and beyond	0.75	65

(a) Peak particle velocities must be recorded in three mutually perpendicular directions. The maximum peak particle velocity is the largest of any of the three measurements.

(b) The department shall reduce the maximum peak velocity allowed, if it determines that a lower standard is required because of density of population or land use, age or type of structure, geology or hydrology of the area, frequency of blasts, or other factors.

(12) If blasting is conducted in such a manner as to avoid adverse impacts on any underground mine and changes in the course, channel, or availability of ground or surface water outside the permit area, then the maximum peak particle velocity limitation of (11) does not apply at the following locations:

(a) at structures owned by the operator and not leased to another party; and

(b) at structures owned by the operator and leased to another party, if a written waiver by the lessee is submitted to the department prior to blasting.

(13) An equation for determining the maximum weight of explosives that can be detonated within any eight-millisecond period is in (14). If the blasting is conducted in accordance with this equation, the peak particle velocity is deemed to be within the limits specified in (11).

(14) The maximum weight of explosives to be detonated within any eight-millisecond period may be determined by the formula $W = (D/Ds)^2$ where W = the maximum weight of explosives, in pounds, that can be detonated in any eight-millisecond period; D = the distance, in feet, from the blast hole nearest to a dwelling, or public, commercial, community or institutional building, except as noted in (12); and Ds = the scaled distance factor, using the values identified in (11). (History: 82-4-204, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; AMD, 1990 MAR p. 936, Eff. 5/18/90; TRANS, from DSL, 1996 MAR p. 2852; AMD, 2004 MAR p. 2548, Eff. 10/22/04.)

17.24.625 SEISMOGRAPH MEASUREMENTS (1) Whenever a seismograph is used to monitor the velocity of ground motion and the peak particle velocity limits of ARM 17.24.624(11) are not exceeded, the equation in ARM 17.24.624(14) need not be used. If that equation is not used by the operator, a seismograph record must be obtained for each shot.

(2) The use of a modified equation to determine maximum weight of explosives per delay for blasting operations at a particular site may be approved by the department, on receipt of a petition accompanied by reports including seismograph records of test blasting on the site. The department may not approve the use of a modified equation if the peak particle velocity for the limits specified in ARM 17.24.624(11) are exceeded, meeting a 95% statistical confidence level.

(3) The operator may use the ground vibration limits described in the blasting-level chart referenced in 30 CFR 816.67(d)(4) as an alternative to (1) and (2), upon approval by the department.

(4) The department may require a seismograph record of any or all blasts and may specify the location at which the measurements are to be taken. (History: 82-4-204, 82-4-205, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 2852; AMD, 1999 MAR p. 811, Eff. 4/23/99.)

17.24.626 RECORDS OF BLASTING OPERATIONS (1) A record of each blast, including seismograph records, must be retained for at least three years and must be available for inspection by the department and the public on request. Blasting records must be complete and accurate at the time of inspection. The record must contain the following data:

(a) name of the operator conducting the blast;
(b) location, date, and time of the blast;
(c) name, signature, and license number of blaster-in-charge;
(d) direction and distance, in feet, from the blast hole nearest to a dwelling, or commercial, public, community, or institutional building either:
(i) not located in the permit area; or
(ii) not owned nor leased by the person who conducts the mining activities.
(e) weather conditions, including temperature, wind direction, and approximate velocity;
(f) type of material blasted;
(g) number of holes, burden, and spacing;
(h) diameter and depth of holes;
(i) types of explosives used;
(j) total weight of explosives used and total weight of explosives used in each hole;
(k) maximum weight of explosives detonated within any eight-millisecond period;
(l) maximum number of holes detonated within any eight-millisecond period;
(m) initiation system;
(n) type and length of stemming;
(o) mats or other protections used;
(p) type of delay detonator and delay periods used;
(q) sketch of the delay pattern;
(r) number of persons in the blasting crew;
(s) seismographic and airblast records, where required, including:
(i) the calibration signal of the gain setting or certification of annual calibration;
(ii) seismographic reading, including exact location of seismograph and its distance from the blast, airblast reading, dates and times of readings;
(iii) name of the person taking the seismograph reading; and
(iv) name of the person and firm analyzing the seismographic record; and
(t) reasons and conditions for each blast occurring outside the time frames published in the blasting schedule. (History: 82-4-204, MCA; IMP, 82-4-231, MCA; NEW, 1980 MAR p. 725, Eff. 4/1/80; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 2852; AMD, 2004 MAR p. 2548, Eff. 10/22/04.)

ARM 17.24.1260

17.24.1260 REQUIREMENTS FOR THE CONDUCT OF BLASTING OPERATIONS (1) Each operator shall conduct each blasting operation under direction of an individual who has been certified by the department pursuant to ARM 17.24.1261 and who is familiar with the operation's

blasting plan and site-specific blasting performance standards. The certified blaster's responsibilities include, but are not limited to, determining blasting pattern, hole pattern, type and quantity of explosives, maintenance of blasting records, and safety of employees involved in the storage, transportation, and use of explosives.

(2) A certified blaster may not delegate the direction of blasting operations to any individual who is not a certified blaster.

(3) A certified blaster and at least 1 other person must be present during the detonation of each blast.

(4) A certified blaster shall immediately exhibit on-site or at the mine office his certificate to any authorized representative of the department or the federal coal regulatory authority upon request.

(5) An operator shall require that persons who are not certified blasters receive direction and on-the-job training from a certified blaster before those persons assist in the storage, transportation, and use of explosives. (History: 82-4-204(4), 82-4-205(7), 82-4-231(10)(e), MCA; IMP, 82-4-231 (10)(e), MCA; NEW, 1984 MAR p. 1373, Eff. 9/14/84; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 3042.)

7.24.1261 CERTIFICATION OF BLASTERS (1)(a) A person seeking certification as a blaster shall submit to the department an application on a form provided by the department. The applicant shall include a verifiable statement that he has successfully completed a training course, provided by the department, the operator, or other person, meeting the requirements of ARM 17.24.1262(1) and incorporating the training manual prepared by the department.

(b) The department shall make available to the public, upon request and payment of a reasonable fee, a copy of the training manual. The training manual must be updated as necessary.

(2) The department shall issue a blaster certification to each applicant who:

(a) has 2 years field experience in blasting;

(b) has successfully completed a 24-hour blaster training course meeting the requirements of ARM 17.24.1262; and

(c) achieves a grade of 80% or higher on an examination administered by the department. The examination must, at a minimum, reflect the training manual prepared by the department and examine in the topics set forth in ARM 17.24.1262. The examination must also incorporate an equally weighted section that covers practical field experience on blasting procedures and occurrences. An applicant who fails may retake the examination. If the applicant fails the examination a second time, he shall successfully complete a blaster training course again and reapply for certification before retaking the examination.

(3) Blaster certifications are non-transferable.

(4) A certification shall expire 3 years after issuance. The department shall recertify if the blaster:

(a) submits to the department, at least 60 days prior to the expiration of his certification, an application for recertification on a form provided by the department;

(b) has documented successful completion of 16 hours of refresher training meeting the requirements of ARM 17.24.1262 during the certification period; and

(c) has conducted or directed blasting operations within the 12 months immediately preceding the date of application for recertification or receives a grade of 80% or better on a recertification examination. The only new developments that the department may include in the recertification examination are those that have been included in the updates to the training manual. The applicant for recertification may take the examination twice.

(5) The department shall certify any person who has a current state or federal blaster certificate under any program approved by the federal coal regulatory authority under 30 CFR Part

850 and can demonstrate that he or she has met requirements equivalent to those in (1) and (2) above. The period of the department's certification must be coextensive with the period of certification under the other program but may not exceed 3 years. (History: 82-4-204(4), 82-4-205(7), 82-4-231(10)(e), MCA; IMP, 82-4-231(10)(e), MCA; NEW, 1984 MAR p. 1373, Eff. 9/14/84; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 3042; AMD, 1999 MAR p. 811, Eff. 4/23/99; AMD, 1999 MAR p. 2768, Eff. 12/3/99.)

17.24.1262 BLASTER TRAINING COURSES (1) A blaster training course must provide appropriate training in and discuss practical applications of:
(a) use of explosives, including:
(i) selection of the type of explosive to be used;
(ii) determination of the properties of explosives which will produce desired results at an acceptable level of risk;
(iii) handling, transportation and storage;
(b) design of blasts, including:
(i) geologic and topographic considerations;
(ii) blast hole design;
(iii) pattern design, field layout, and timing of blast holes;
(iv) field applications;
(c) loading of blast holes, including priming and boostering;
(d) use of initiation systems and blasting machines;
(e) effects of blasting vibrations, airblast, and flyrock, including;
(i) monitoring techniques;
(ii) methods to control adverse effects;
(f) use of secondary blasting;
(g) discussion of current federal and state rules applicable to the use of explosives;
(h) maintenance of blast records;
(i) determination of blasting schedules;
(j) design and use of preblasting surveys including availability, coverage, and use of in-blast design;
(k) requirements of blast plans;
(l) signs, warning signals, and site control;
(m) identification of unpredictable hazards including:
(i) lightning;
(ii) stray currents;
(iii) radio waves;
(iv) misfires; and
(n) updates to the department's training manual. (History: 82-4-204(4), 82-4-205(7), 82-4-231(10)(e), MCA; IMP, 82-4-231(10)(e), MCA; NEW, 1984 MAR p. 1373, Eff. 9/14/84; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 3042; AMD, 1999 MAR p. 811, Eff. 4/23/99.)

17.24.1263 SUSPENSION OR REVOCATION OF BLASTER CERTIFICATION (1) The following are grounds for suspension or revocation of blaster certification:
(a) failure to comply with any order of the department;
(b) conviction of criminal possession or sale of dangerous drugs;
(c) unlawful use in the work place of, or current addiction to, alcohol, narcotics, or other dangerous drugs;
(d) violation of any state or federal explosives laws or regulations;

(e) providing of false information or a misrepresentation to obtain certification;

(f) failure to present blaster certification upon request of the department or federal coal regulatory authority personnel;

(g) delegating responsibility to any individual who is not a certified blaster;

(h) storage, transportation, or use of explosives in a manner that could threaten life or limb or cause environmental harm.

(2) If the department finds that a certified blaster has committed one or more of the acts prohibited in (1), the department may, and upon a finding of willful conduct shall, suspend or revoke the certification of the blaster. The department shall determine whether to suspend or revoke and the length of suspension on the basis of determination of reasonable necessity to protect human life or limb and to prevent environmental degradation.

(3) If the department has probable cause to believe that a certified blaster has committed any of the acts prohibited in (1) and that the blaster's certification should or must be suspended or revoked, the department shall notify the blaster and his employer in writing by certified mail at the address contained in the blaster's application for certification or at a subsequent address of which the blaster has notified the department in writing. The blaster does not defeat service by refusing to accept or failing to pick up the notice. The notice must advise the blaster of the department's proposed action, the alleged facts upon which the proposed action is based, and the blaster's right to request a contested case hearing before the board of environmental review. If the department determines that suspension of the blaster's certification is reasonably necessary in order to protect human life or limb or the environment, it may suspend the certification until the hearing is held; provided, however, that no such suspension may be in effect for longer than 45 days. At the close of the hearing, the hearing officer may, based on a finding that the department will probably prevail and that continued suspension is reasonably necessary, continue the suspension until a final decision is made. (History: 82-4-204, 82-4-231, MCA; IMP, 82-4-231, MCA; NEW, 1984 MAR p. 1373, Eff. 9/14/84; AMD, 1989 MAR p. 30, Eff. 1/13/89; TRANS, from DSL, 1996 MAR p. 3042; AMD, 2004 MAR p. 2548, Eff. 10/22/04.)

APPENDIX E—Answers to Review Questions

1. float

2. placing explosives in water-resistant packaging

3. an oxidizer and fuel mixture that is insensitive to a No. 8 blasting cap

4. it has no water resistance and a density of less than 1

5. densified and placed in water resistant packaging

6. 70

7. unit of cap-sensitive explosive and contains a detonator

8. Ammonium Nitrate and Fuel Oil

9. a. cost of explosive
    b. charge diameter
    c. cost of drilling
    d. fragmentation difficulties
    e. water conditions
    f. adequacy of ventilation
    g. atmospheric temperature
    h. propagating ground
    i. storage consideration
    j. sensitivity considerations
    k. explosive atmospheres

10. reduce the sensitivity

11. explosives and blasting agents

12. underground

13. increase explosive energy

14. the transfer or movement of a detonation from one point to another

15. decrease, primed

16. underground, permissible

b. energy distribution network
c. an in-hole component that uses energy to initiate the explosive

25. a. initial energy sources
    b. controlled throw
    c. reduced ground vibration and airblast

26. a. electric   c. Hercudet (gas detonation)
    b. cap and fuse   d. Nonel (shock tube)

27. a. when deck charges are used
    b. as a safety factor, to assure total column detonation

28. in the direction of the main charge

29. a. cap sensitive explosive
    b. detonator

1. the distance from the explosive charge to the nearest free face--also usually the distance between rows of holes

2. the distance between boreholes in a row

3. the distance drilled below the floor level to assure that the full face of rock is removed

4. subtract stemming height from hole depth

5. a. 10'
   b. 14'
   c. 40'
   d. 8'
   e. 32'
   f. 4"

6. $$\frac{\text{Burden x Spacing x Depth}}{27}$$

7. geology, breakage, diameter, explosive type, depth

8. 333 cubic yards

9. 222 cubic yards

10. the weight of explosives loaded per foot of borehole

11. a. 14.71 lbs/ft
    b. 470.72 lbs

12. .9 lbs/cu yard

13. a. 231.1 cu. yds.
    b. 5,546.4 cu. yds.
    c. 9.81 lbs./ft.
    d. 215.82 lbs.
    e. 0.933
    f. yes, the powder is close to one which is typical

14. a. 111 cu. yds.
    b. 2,222 cu. yds.
    c. 5.72 lbs/ft.
    d. 74.36 lbs.
    e. .669
    f. probably, the powder factor is almost .7 which is fairly close to one which is typical

15. flyrock

b. the burden needs time to move forward in order to accommodate broken rock from subsequent holes

30. to calculate costs

31. a mudcap, plaster, or adobe charge

32. coal and methane

33. to reduce perimeter cracking and increase the stability of the opening

34. a. line drilling   c. smooth blasting
    b. presplitting   d. cushion blasting
    e. improves the competence of the rock at the perimeter of the excavation

17. a. peak recorders or
    b. recorders that measure the entire blast

18. by comparing the recorded vibrations of each

24. a. loose explosives or detonating cord in muckpile
    b. legwires, detonating cord or tubes extending from a borehole
    c. a misshapen muckpile

25. drilling and shooting an adjacent hole prior to disposing of the misfire

26. a. improper guarding   c. unsafe cap and fuse practices
    b. impacting explosives   d. extraneous electricity

APPENDIX F

AFFIDAVIT

	New Application	Renewal Application
		(Check One)

.fication Number	
Certification Date	
Expiration Date	

Please Type or Print Legibly

BLASTER CERTIFICATION PROGRAM

Montana Department of
ENVIRONMENTAL **Q**UALITY

APPLICATION & RENEWAL FORM

HEIGHT	WEIGHT
COLOR OF EYES	COLOR OF HAIR
BIRTH DATE (MM/DD/YY)	SOCIAL SECURITY NO.

NAME OF APPLICANT (Print or type)

MAILING ADDRESS

CITY	STATE	ZIP CODE

ARE YOU CURRENTLY CERTIFIED UNDER ANOTHER STATE OR FEDERAL PROGAM	HAS YOUR BLASTER CERTIFICATION EVER BEEN REVOKED?
(Circle One)　　YES　　NO	(Circle One)　　YES　　NO
LIST PROGRAM AND CERTIFICATION NUMBERS	IF YES, WHY?
1.) ATF -	
2.)	

APPLICANT'S EXPERIENCE RECORD (List Most Recent Experience First.)

FROM YR	TO MO/YR	COMPANY (City, State)　　FOREMAN	TYPE OF BLASTING EXPERIENCE *

*NOTE: Attach additional pages as necessary.

TRAINING COURSES COMPLETED WITHIN THE LAST THREE YEARS (Attach Verification) *

LENGTH OF COURSE (HRS)	NAME OF COURSE	DATE COURSE COMPLETED	COURSE DESCRIPTION

PRESENT EMPLOYER OR NAME OF BUSINESS	HOME PHONE
BUSINESS ADDRESS	BUSINESS PHONE
CITY	STATE,　ZIP CODE

APPLICANT SWEARS THAT ALL OF THE FOLLOWING ARE TRUE:
(a)　I am physically and mentally fit to handle explosives safely;
(b)　I am experienced in the use of explosives
(c)　I have not been convicted of a felony or misdemeanor involving the use of explosives;
(d)　I am of good moral character;
(e)　I am not addicted to narcotic drugs or intemperate in the use of alcohol;
(f)　That I have read the Montana Blaster Certification Manual and am familiar with the contents therein;
　　The statements made in this application **are true**.

NOTE:
Effective May 24, 2003, each applicant must list their ATF licence or permit as applicable (Safety Explosives Act - November 25, 2002).

Blaster certification expires every **three years**.

You must submit this form **60 days** prior to your expiration date.

The Department will notify you concerning exam information, if applicable.

APPLICANT'S SIGNATURE	DATE

NEW BLASTER CERTIFICATION AFFIDAVIT

As part of the Montana Blaster Certification Program for new blasters, pursuant to ARM 17.24.1261(2) and (5), please complete the affidavit and application and return them to: Montana Department of Environmental Quality, Industrial & Energy Minerals Bureau, P.O. Box 200901, Helena, MT. 59620-0901. You will then be notified of the time, place, and date of examination.

I, the undersigned, hereby certify that I have a minimum of 24 months of blasting experience and have read, studied, and completed the review questions and understand all the material in the document titled, Montana Blaster Certification Training Manual. In addition, I have completed a 24-hour training course meeting the requirements set forth by ARM 17.24.1262 and hereby do attach a verifiable statement indicating such completion.

_____   _____
(Signature of applicant)   (Date)

_____
(Company)

SUBSCRIBED AND SWORN to before me this _____ day of _____, 20_____.

_____
Notary Public for the State of _____

Residing at _____

My Commission Expires _____