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MILITARY ENGINEERING APPLICATION OF COMMERICAL EXPLOSIVES

Joseph Briggs

Army Engineer Waterways Experiment Station
Livermore, California

October 1972

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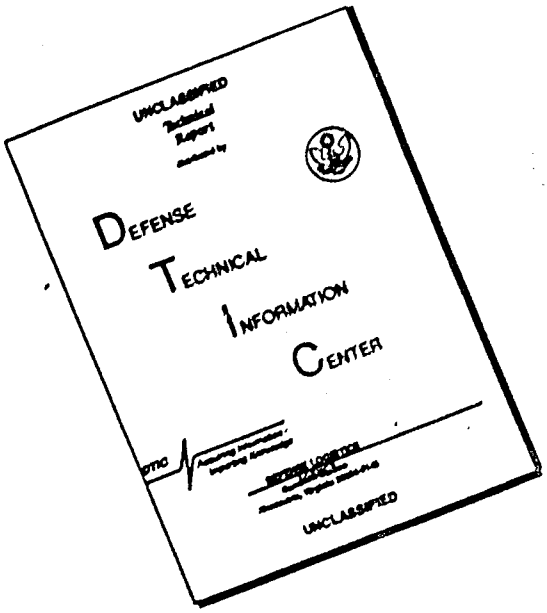

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TECHNICAL REPORT E-73-2

MILITARY ENGINEERING APPLICATIONS OF COMMERCIAL EXPLOSIVES: AN INTRODUCTION

MAJ. Joseph Briggs

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13. ABSTRACT Explosive excavation is a relatively new technology for creating barriers, destroying targets, and constructing military aids (e.g., bunker emplacements) in the theater of operations. This technology stems from research in explosive excavation performed since 1962. It is acknowledged that the military engineer is in need of a military engineering tool that will satisfy his earth-moving requirements and increase his ability to defeat the enemy in less time, with fewer men, and with less equipment.

The report compares the advantages of various bulk commercial explosives with the Army's arsenal of conventional explosives. Large-scale excavation with bulk explosives shows overall cost and time benefits. Two military related experiments utilizing bulk explosives are presented -- with an evaluation of the associated cost and time savings.

The report describes the mechanism of crater formation, the characteristics of explosively produced craters, and the types of projects in which craters would be useful in the theater of operations (e.g., tank traps, bridge destruction). The report also provides design procedures for the emplacement of single charges and multiple charges for row craters. It discusses geologic media and a variety of drilling techniques. There is research required to further develop commercial explosives for military applications.

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TECHNICAL REPORT E-73-2
MILITARY ENGINEERING APPLICATIONS OF COMMERCIAL EXPLOSIVES: AN INTRODUCTION

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Livermore, California

MS. date: October 1972

Preface

The U.S. Army Engineer Waterways Experiment Station (USAEWES) Explosive Excavation Research Laboratory (EERL) was the USAEWES Explosive Excavation Research Office (EERO) prior to 21 April 1972. Prior to 1 August 1971 the organization was known as the USAE Nuclear Cratering Group.

This report was prepared by the U.S. Army Waterways Experiment Station Explosive Excavation Research Laboratory (EERL) to introduce the combat and construction engineer planner to the potential use of large explosive charges in the theater of operations primarily for barriers, target destruction, and construction purposes. The study was sponsored by the Office of the Chief of Engineers, under the staff supervision of the Directorate of Military Engineering. The experience of EERL in modeling nuclear cratering explosions with chemical explosives over the past ten years and more recent experiments designed to develop more economical techniques of chemical explosive excavation are the primary sources of information for this report. The knowledge developed from the EERL research program has reached a level at which certain explosive excavation projects can be successfully and confidently designed. This report is written with the objective of documenting this technology and its potential for the Tactical Commander. This task is part of EERL's mission of developing chemical explosive excavation techniques for military purposes. Although optimum procedures have not yet been established by field testing for the full range of possible excavation or target destruction situations, improvements brought about by future research work will be documented by appropriate additions or revisions to this report.

The Directors of EERL during the preparation of this report were LTC Robert L. LaFrenz and LTC Robert R. Mills, Jr. The Deputy Director (Military) was MAJ Richard H. Gates.

Abstract

Explosive excavation is a relatively new technology for creating barriers, destroying targets, and constructing military aids (e.g., bunker emplacements) in the theater of operations. This technology stems from research in explosive excavation performed since 1962. It is acknowledged that the military engineer is in need of a military engineering tool that will satisfy his earth-moving requirements and increase his ability to defeat the enemy in less time, with fewer men, and with less equipment.

The report compares the advantages of various bulk commercial explosives with the Army's arsenal of conventional explosives. Large-scale excavation with bulk explosives shows overall cost and time benefits. Two military related experiments utilizing bulk explosives are presented—with an evaluation of the associated cost and time savings.

The report describes the mechanism of crater formation, the characteristics of explosively produced craters, and the types of projects in which craters would be useful in the theater of operations (e.g., tank traps, bridge destruction). The report also provides design procedures for the emplacement of single charges and multiple charges for row craters. It discusses geologic media and a variety of drilling techniques. There is research required to further develop commercial explosives for military applications.

Acknowledgments

This report was prepared in response to the requirement to expedite development of explosive excavation techniques for military purposes under the Military Engineering Design and Expedient Construction Criteria Program (MEDECC).

The guidance and encouragement provided during the preparation of this report by the previous and present Directors of EERL, LTC Robert L. LaFrenz and LTC Robert R. Mills, Jr., and MAJ Richard H. Gates, the Deputy Director, are greatly appreciated.

This report represents the combined effort of many persons in EERL. The work of the following authors is acknowledged and sincerely appreciated: Majors Richard H. Gates and Charles E. Gardner for Chapter 4; Captains Howard E. Reed and Robert F. Bourque, and Specialist John F. Dishon for Chapter 3; Captain Wade J. Wnuk, 1LT Joseph C. Struckel, and Specialist Jon S. Morishita for Chapter 5; Bruce B. Redpath for Chapter 6; and Specialist Larry Mays for Chapter 7.

Finally, special thanks are due to Mrs. Virginia Hutchison and her staff for the typing during the preparation of the report.

Conversion Factors

British units of measurements used in this report can be converted to metric units as follows:

Multiply	By	To obtain
inches	2.54	centimeters
feet	0.3048	meters
cubic feet	0.02832	cubic meters
cubic yards	0.764555	cubic meters
pounds	0.4535924	kilograms
pounds per square inch	0.00689476	meganewtons per square meter
pounds per cubic foot	16.02	kilograms per cubic meter
Fahrenheit degrees	—[a]	Celsius or Kelvin degrees
foot-pounds	0.138255	meter-kilograms

[a]To obtain Celsuis (C) temperature readings from Fahrenheit (F) readings, use the following formula: $C = (5/9)(F - 32)$. To obtain Kelvin (K) readings, use: $K = (5/9)(F - 32) + 273.15$.

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MILITARY ENGINEERING APPLICATIONS OF COMMERCIAL EXPLOSIVES: AN INTRODUCTION

Chapter 1 Introduction

SCOPE

Chemical explosive excavation offers the military engineer a practical substitute for Atomic Demolition Munitions (ADM) to implement barrier and denial plans, to destroy large targets, and to perform large-scale excavations in the theater of operations. The tremendous achievements by the explosives industry during the past ten years in developing reliable, safe, easy-to-handle bulk explosives have prompted the Explosive Excavation Research Laboratory (EERL) of the U.S. Army Waterways Experiment Station (WES) to initiate an extensive research and development program covering the use of commercial explosives in military engineering applications. The use of bulk explosives by the Corps of Engineers in civil works construction has been previously reported by the U.S. Army Engineer Nuclear Cratering Group (NCG).[1]

Since 1962, EERL (formerly the Nuclear Cratering Group) has executed extensive laboratory and field cratering tests and demonstrations with commercial explosives. Some of these experimental Corps of Engineers civil works projects, such as canals, waterways connections, harbors, and railway and highway cuts, vividly illustrated the potential of explosive excavation with large charges.

In 1971, under the Military Engineer Design and Expedient Construction Criteria Program (MEDECC), EERL received the task of evaluating and suggesting improvements in the Army's capabilities for rapidly creating obstacles and barriers to defeat the advances of the enemy's armor and bridging capabilities.

In late 1972 EERL's research into the military applications of this technology was formalized into the Military Engineering Applications of Commercial Explosives program (MEACE).

This initial report introduces the concept of using commerical explosives to satisfy military excavation requirements: in particular, barrier formation, target destruction, and large-scale excavations in the theater of operations.

DEFINITION OF TERMS

As used in this report, the term chemical explosives refers to all nonnuclear explosive systems, including those commonly used commercially and by the military. This term covers a wide range of products, from high explosives to blasting agents and oxidizers. The various types of chemical explosives

are discussed in detail in Chapter 3. Explosive excavation will be used in this text to refer to techniques of moving earth and rock with chemical explosives, although the principles are the same when nuclear explosives are employed.

The term military explosives refers to those explosives used primarily by the military only, such as Composition C-4 and TNT. The term commercial explosives, on the other hand, refers to products available on the open market that are not commonly found in the military inventory. In particular, EERL's research has been in the field of bulk explosives, such as ammonium nitrate-fuel oil (ANFO), nitromethane, and slurries or water gels. Bulk explosives are easy to use in large quantities (i.e., several hundred pounds or more), are relatively much safer to handle than high explosives, are composed mostly of cheap, widely available ingredients, and, in the case of slurries, can be mixed to provide varying explosive energy.

The applications and design procedures discussed in this report could be used, in principle, with any explosive. However, it is with bulk explosives that these applications and design procedures are most feasible. The "ideal" bulk explosive would be one composed of two or three inert, nonexplosive, easily stored ingredients that could be mixed together in variable proportions on-site to provide an explosive with properties best suited to the particular job. The resulting mixture could then be pumped in any quantity into the emplacement cavity and detonated with a common gap and booster system. Aluminized slurry systems are very close to this ideal already. It is to introduce the great potential of such a system that this report is presented.

BASICS OF EXPLOSIVE EXCAVATION

In most instances the tactical requirement to create a barrier or to deny the enemy access to an area, to destroy a large target, or to conduct some form of military construction employing explosive excavation techniques implies some form of cratering or the use of large explosive charges to produce an excavation by fracturing and ejecting large volumes of earth or rock.[1] An understanding of the cratering concept of explosive excavation and some nomenclature is essential to the appreciation of the capabilities and the potential of the techniques discussed in thie report.

A crater consists of three concentric zones known as the apparent crater, the true crater, and the rupture zone. These are illustrated in Fig. 1, a cross section of a typical crater in rock.

The apparent crater is the net excavated volume below the original ground surface. Its radius, depth, and volume are the basic criteria for the engineering design of an explosive excavation. Its cross section has been found to be best approximated by a hyperbola. Its depth is somewhat less than the charge depth of burial except for charges buried at shallow depths.

The raised rim, or lip, surrounding the crater consists of uplifted material (upthrust) overlain with fragmented material that has been ejected from the crater. The fragmented material, ejecta, covers the original ground surface out to

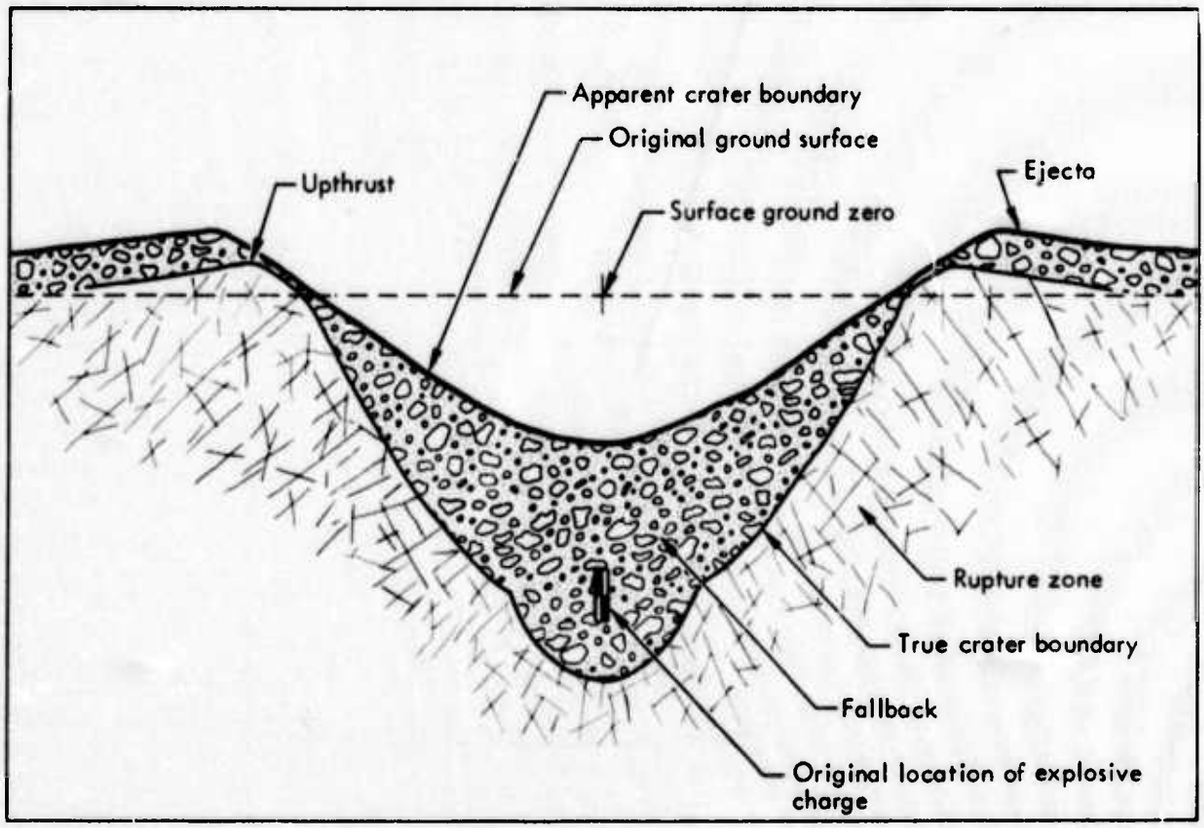


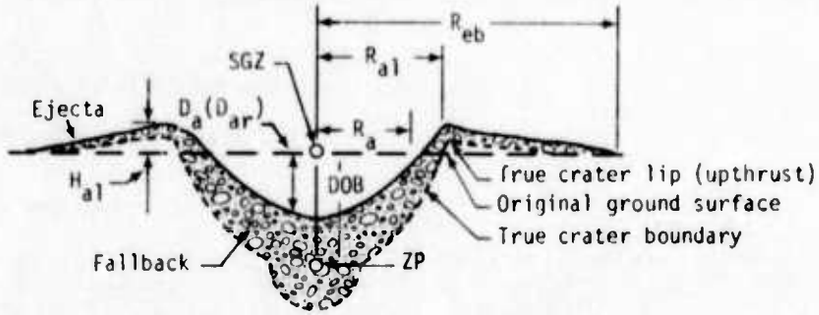
Fig. 1. Cross section of typical crater in rock, showing nomenclature.[1]

a distance approximately equal to three times the crater radius from surface ground zero.

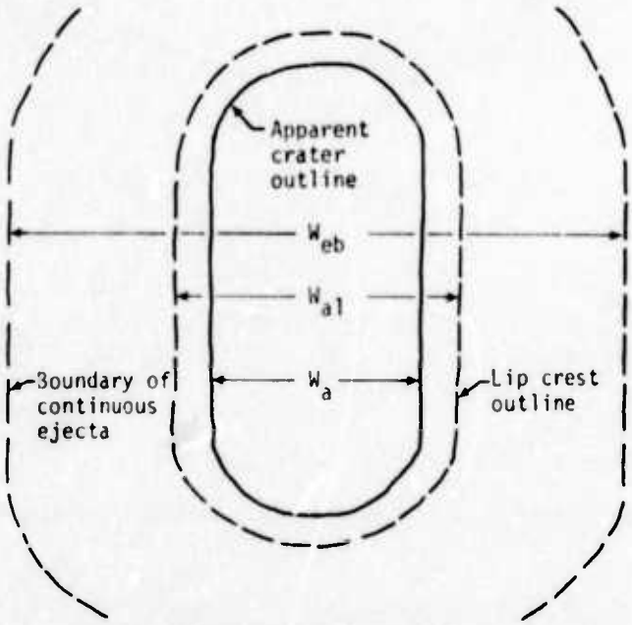
The true crater is the excavation that would exist if all of the broken material were removed from the crater. The cross section of the true crater has been found to be best approximated by a parabola. During crater formation, the true crater is partially filled with material, known as fallback, that forms the apparent crater. Thus, the size and the shape of the true crater are not easily discerned.

Surrounding the true crater is a rupture zone. Within this zone material has been displaced slightly upward and outward and has more fractures than in the natural state. The actual interface between the true crater and the rupture zone is poorly defined. It is thought to be more a transition region than a definite boundary. The outer limit of the rupture zone is also poorly defined. The blast-induced fractures diminish in number with distance from the detonation point until there are so few that they are indistinguishable from naturally occurring fractures.

In quantitative discussions of crater characteristics, it is necessary to have in mind the nomenclature that describes features of the crater and the notation used to represent measurable parameters. Figure 1 shows the preferred crater nomenclature. Figure 2 shows the notations used for craters produced by single charges and for craters produced



Cross section of single-charge or row crater



Plan view of row crater

Nomenclature which applies only to single-charge craters

R_a - Radius of apparent crater measured at original ground surface datum

R_{al} - Radius of apparent lip crest

R_{eb} - Radius of outer boundary of continuous ejecta

D_a - Maximum depth of apparent crater below and normal to original ground surface

Nomenclature which applies only to row craters

W_a - Width of apparent linear crater measured at original ground surface datum

W_{al} - Width of apparent lip crest

W_{eb} - Width of outer boundary of continuous ejecta

D_{ar} - Depth of apparent row crater

Nomenclature and definitions which apply to both single-charge and row craters

H_{al} - Apparent crater lip crest height above original ground surface

V_a - Volume of apparent crater below original ground surface

V_{al} - Volume of apparent lip

V_t - Volume of true crater below original ground surface

DOB - Depth of burst

ZP - Zero Point-effective center of explosion energy

SGZ - Surface Ground Zero (point on surface vertically above ZP)

NSP - Nearest Surface Point (point on surface nearest ZP; same as SGZ for horizontal surface)

Fig. 2. Crater notations.[1]

by a single row of several charges. As will be seen later, these notations can be adapted for use with multiple-row-charge craters.

In addition to the nomenclature and notations in Figs. 1 and 2, the following definitions will apply in discussions of the material properties of the media affected by a cratering detonation.

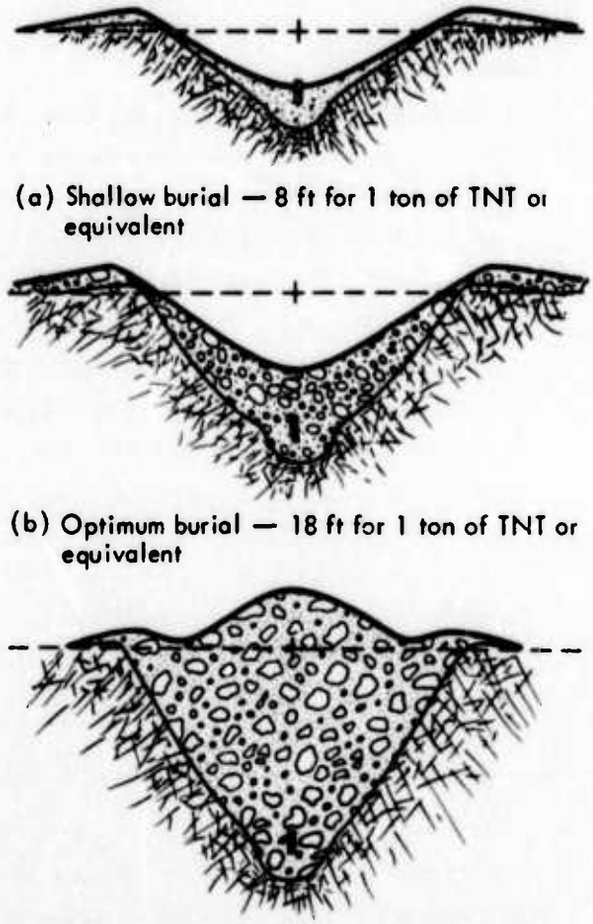
Rubble: The material comprising the fallback and ejecta.

Rupture Zone: The zone of blast-induced fractures and displacements from the true crater boundary outward to the relatively undisturbed in situ material.

Bulking Factor (BF): The ratio of in situ or preshot bulk density to post-shot bulk density. Bulking factor is used on military and civil works conventional construction projects to determine cut, haul, and fill requirements.

The crater shown in Fig. 1 is typical for a charge detonated at optimum depth of burial, which is defined as the depth at which the detonation will excavate the largest net volume of material. At optimum depth of burial an explosion will produce a large crater by fragmenting a large quantity of material and imparting sufficient velocity to eject most of that material from the true crater. An explosion at shallower depth will fragment relatively little material but will eject the material at high velocity. Conversely, a very deep explosion will fragment a large quantity of material but will eject very little of it.

Characteristic features of craters formed by explosions at shallow, optimum, and deep depths of burial in rock are shown in Fig. 3.



(a) Shallow burial — 8 ft for 1 ton of TNT or equivalent

(b) Optimum burial — 18 ft for 1 ton of TNT or equivalent

(c) Deep burial — 28 ft for 1 ton of TNT or equivalent

Fig. 3. Crater profiles for various depths of burial in rock.

This report does not attempt to establish a specific charge-weight limit for military applications; however, it is conceivable that explosive charges ranging in size from about 100 lb to several hundred tons will be employed.

In general, explosive excavation involves charges buried at depths ranging from the surface down to a point at which, upon detonation, there is little visible surface evidence of the detonation. The depth of primary interest for cratering applications that will be presented is the optimum depth of burial. The design technique that uses charges buried at

depths greater than optimum will also be discussed even though this technique is still under development.

Explosive excavation in the theater of operations will serve as an alternative to the standard conventional military engineering techniques employed during tactical operations as well as for military construction requirements. Whenever excavating, blasting, or removing rock are essential requirements, the use of large concentrated charges will offer the Tactical Commander substantial benefits. Among these benefits will be the advantages associated with rapid construction and the economy of men, equipment, time, and cost.

PRACTICAL APPLICATIONS

Initial research and testing, both commercial and military, indicate the following to be the most practical and beneficial projects for implementation in the theater of operations: tank traps, road craters, airfield destruction, bunker emplacements, harbors, sewage lagoons, large drainage ditches, quarries and expedient (blasted-into-place) dams. The desire of the engineer staff to satisfy more efficiently the requirements for such projects under combat conditions demands an improvement in existing excavation techniques.

PREVIEW OF CONTENTS

Before the combat engineer employs explosive excavation techniques, there are many questions to be answered. This report has anticipated some of these questions and provided some answers. However, the state-of-the-art has not reached the point at which all of the questions are readily answered. It is anticipated that this report will act as a catalyst to stimulate research and testing to take eventually the maximum advantage of the potential of chemical explosive excavation. Following this introduction and a look at some of the applications in Chapter 2, there are presentations in Chapters 3 through 6 of many of the parameters affecting this excavation technique, such as the explosives employed, the geologic media, and various explosive design techniques. Chapter 7 is devoted to a technique for the sequential minimization of cost or time for the rapid emplacement of single-charge craters using the information presented in previous chapters. Appendix A provides results from barrier demonstration projects accomplished to date.

Chapter 2 Applications

SCOPE

The employment of explosive excavation techniques in the theater of operations will provide a valuable nonnuclear military engineering tool. During the past ten years bulk explosives have been used effectively in commercial and civil works excavation projects, such as waterways, harbors, railway and roadway cuts, channel plugs, and quarries.

This chapter briefly reviews some of the engineering considerations for an excavation design and discusses several demonstrated and conceptual applications of large explosive charges to barriers, target destruction, and military construction in both forward and support areas of a theater of operations (see Fig. 4).

MILITARY USES OF EXPLOSIVE EXCAVATION

General

Demolition in the military sense is the destruction by any means (fire, water, explosive, mechanical, etc.) of structures, facilities, or materials to accomplish a military objective. Demolition has offensive and defensive purposes; for example, the removal of enemy barriers to delay or restrict enemy movement. The military engineering applications of explosive excavation are in three major areas: barriers, target destruction, and construction.

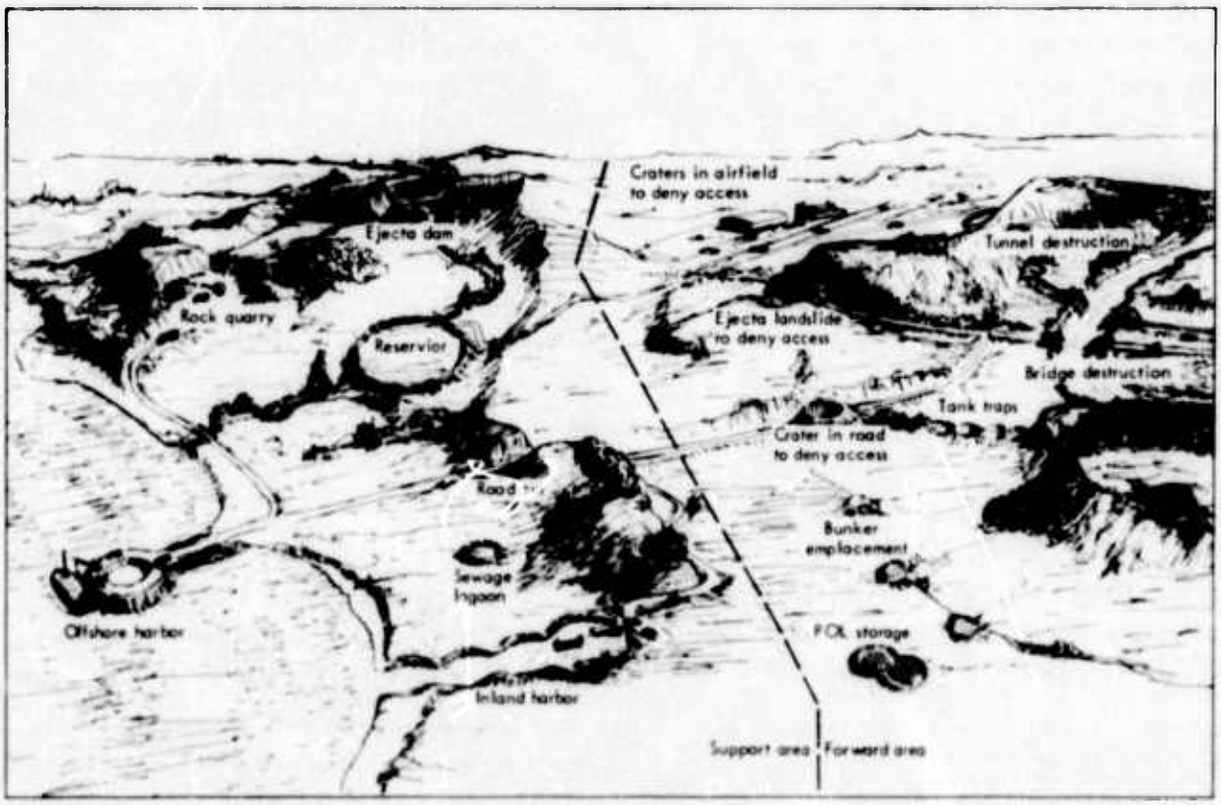


Fig. 4. Applications for commercial bulk explosives in a theater of operations.

Barriers

The principal advantage of using explosives in barrier construction is savings in equipment, time, and manpower. The demolition of any target that would impede the enemy's advance can create an obstacle. Explosively produced antitank obstacles, such as craters or fallen trees, are used extensively to tie in with other natural or man-made obstacles to form a barrier. More extensive obstacles are floods and landslides created by the demolition of dams or by the undercutting of steep hillslides.

Target Destruction

Explosives have proved extremely effective against military targets, whether soft or hard. Hard targets usually require some prior planning, because charge size and placement are often critical. The explosive power is designed to cut or pierce steel, concrete or other materials, to apply great heat or pressure to some area, or to create craters and fractures. Examples include the destruction of bridges, fortifications, culverts, buildings, tunnels, obstacles, and airfields. Soft targets, which may include many of those listed above, are those easily destroyed by explosives. Examples include captured munitions, wooden structures, hastily constructed obstacles, captured supplies, or other targets in rapid denial operations. Explosives have also been used effectively against underwater targets.

Construction

Explosives have proved to be quite versatile in many phases of construction work. Explosives can produce large quantities of fractured rock for construction and yet still provide clean-cut rock faces when required. Currently, the removal of rock outcroppings along road lines or other horizontal datum may be accomplished more quickly and economically with explosives. Explosives have been used extensively in both clearing and ditching operations. Research by EERL has shown that large and rapid excavations can be accomplished with explosives. Large excavation projects accomplished to date include railroad cuts, harbor construction, inlet deepening, channel construction, and quarrying. The ability to create rapidly these large, predictable excavations has excellent potential in the military construction field. As with target destruction, explosives are quite useful in underwater construction and the demolition of unwanted structures.

In the remainder of this chapter there appear the discussions of applications, both tested and conceptual, that are or appear to be feasible with explosive excavation technology in the theater of operations. Specific projects that illustrate the applications are included to give the reader an appreciation of the size of the excavation that can be accomplished.

BARRIERS

A barrier system is a coordinated series of obstacles. Military obstacles are features, either natural or man-made, that impede the maneuverability of military personnel and equipment.[2] The basic reason for the use of obstacles and/or barriers is to increase the

combat effectiveness of the forces that employ them. The obstacles of specific interest for this discussion of chemical explosive excavation are tank traps and road craters. The military engineer is vitally interested in the creation of terrain barriers that will deny or delay access of the enemy's tactical vehicles. To be effective, these obstacles must be too wide for the enemy's track-laying vehicles to span and too deep and steep-sided for any vehicle to pass through them.

Road craters are considered effective antitank obstacles if the tank requires three or more passes to traverse the crater, thereby providing sufficient time for antitank weapons to disable the tank.[2] Single charges buried at optimum depth, as discussed in Chapter 1, will produce road craters or tank traps of adequate size to be considered vital elements of any tactical barrier plan. It is also possible to fire several charges simultaneously, producing a linear crater. The design procedures for producing single- or multiple-charge craters are presented in Chapter 6.

Project Tank Trap

An excellent example of a tank trap or armored vehicle obstacle is the series of craters produced at the Nevada Test Site and tested in Project Tank Trap.[3] The purpose of the project was to evaluate the trafficability of the craters produced. Project Tank Trap is discussed further in Appendix A. An excellent example of an untrafficable crater in hard rock is shown in Fig. 5. The crater diameter from lip to lip measured 310 ft, and the depth at the center was 150 ft.

Fig. 5. M-60 tank unable to maneuver in hard rock of Pre-Schooner crater (Project Tank Trap).[3]

Project Diamond Ore IIA

Another series of craters was produced under Project Diamond Ore IIA, conducted at Fort Peck, Montana. Three craters were produced with slurry explosives as a part of an experimental cratering program carried out by EERL and are discussed in greater detail in Appendix A. Figure 6 gives an indication of the size of the crater produced by 10 tons of an aluminized slurry explosive. The lip-to-lip diameter of this crater measured 150 ft, and its apparent depth was 30 ft.

Project Armor Obstacle I

As a followup to the Diamond Ore IIA series, Project Armor Obstacle I was also conducted at Fort Peck, Montana, to determine the obstacle effectiveness of the three craters produced in the Diamond Ore IIA series. This mobility study was also conducted by EERL and is presented in greater detail in Appendix A. Figure 7 shows that the slopes of Crater IIA-2 were insurmountable obstacles.

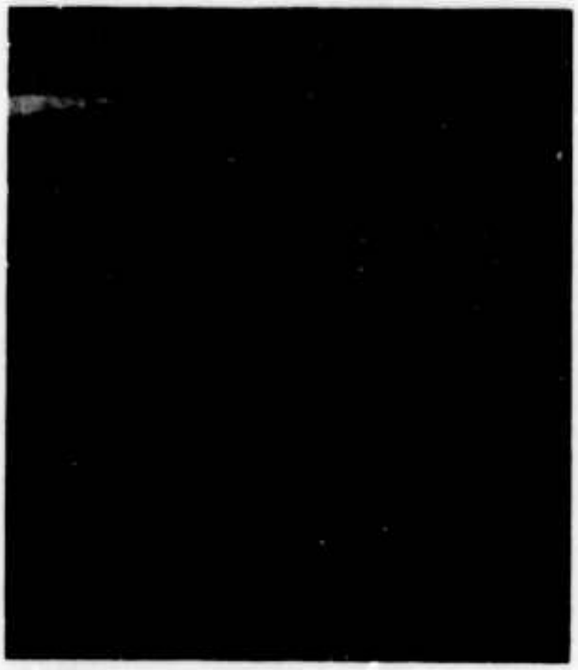


Fig. 6. Aerial view of Diamond Ore Crater IIA-3 (vehicle to right of crater is M-48 tank).

Fig. 7. M-48 tank unable to exit Diamond Ore Crater IIA-2 (Project Armor Obstacle I).

TARGET DESTRUCTION

Although no experiments in the destruction of military targets have been performed with bulk explosives, it is obvious that cratering can be used to destroy any target for which the removal of large amounts of earth or rock, on, or under, the target would result in its demolition. Thus, the military engineer can apply cratering techniques to the demolition of tunnels, bridges, dams, and other earthen structures. The following paragraphs discuss some of the conceptual applications of bulk explosives in this role.

Tunnels

Railway and highway tunnels are extremely vulnerable to explosive demolition and therefore are considered prime military targets. The damage criteria for large underground tunnels are based on blocking the tunnel for a distance of 30 m (100 ft) with the rock and debris from the detonation. This blockage necessitates an extensive and time-consuming operation to clear the tunnel.[4] One of the most critical factors in explosive tunnel demolition is the type of rock through which the tunnel is constructed;[5] the weaker the rock, the better the results.

Bulk explosives in the standard T-design would probably be a very effective means of producing serious damage to a tunnel.[5] As illustrated in Fig. 8, several 750-lb charges of a high explosive are normally considered adequate in a T-design with a 15-ft burden. Because of the effectiveness of aluminized slurries (see Chapter 3), it is conceivable that these 750-lb charges

of TNT could be replaced with 450 to 500 lb of a high-energy slurry.

It is also conceivable that tunnels with chambers outside the tunnel or in the roof and walls designed expressly for the purpose of future demolition could be readily destroyed by placing and stemming bulk explosives in these chambers.

The subject of tunnels cannot be dismissed without the mention of the complex tunneling systems encountered in the Republic of Vietnam. Since the consistency of slurry explosives can be varied from a rigid gel to a liquid, it is conceivable that slurries could have been pumped or poured into the openings of these tunnels and detonated, neutralizing the tunnel network and trapping the occupants who failed to exit prior to the detonation.

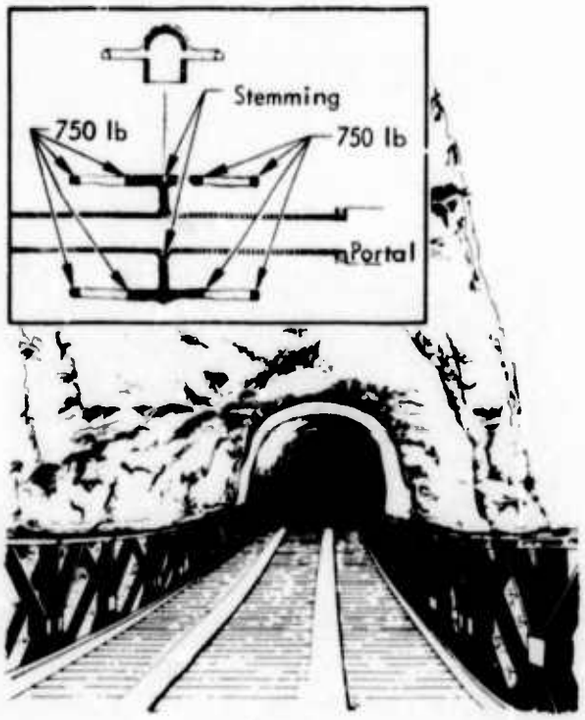


Fig. 8. Typical T-design for tunnel destruction.[5]

Bridges

Because of the many configurations and different materials used in bridge construction, there is no simple formula or general rule of thumb that predicts the optimum extent of demolition for bridges. In order to effectively destroy a bridge with a minimum amount of time, equipment, manpower, and explosive, it is essential that the demolition team be familiar with the construction of various types of bridges and the location of their most vulnerable points. Depending upon the characteristics of the bridge and the strategic placement of external high explosives on the bridge substructure or superstructure, an obstacle can usually be created by destroying the bridge. Because bulk explosives have not been tested as external charges, it is difficult to predict adequately their effectiveness in particular roles. However, as in the case of tunnel destruction, bulk explosives can always be used as a cratering charge by burying an adequate sized charge beneath an abutment or pier.

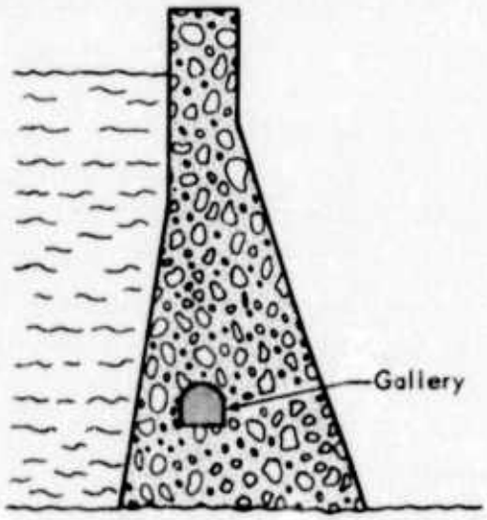
A promising technique presently under development for destroying bridge spans is the use of a water plume. The basic idea is to detonate explosives beneath the surface to create a plume of water with enough uplift force to knock a bridge span off its pier or abutment. Figure 9 is an artist's conception of the water plume technique.

Dams

The size of many dams makes it very impractical to employ the quantity of explosive required to destroy the entire structure. This is the reason that these

Fig. 9. Artist's conception of water plume destroying span of concrete bridge.

structures are likely ADM targets. It is conceivable that large charges of bulk explosives could be used to destroy the machinery and the equipment that control the gates on the dam. For concrete gravity or arch dams that contain inspection galleries, as illustrated in Fig. 10, it is conceivable that liquid explosives could be pumped or poured into the gallery and a portion of the dam could be destroyed. Again, relying upon the excellent cratering ability of bulk explosives, it may be possible to destroy an earthfill dam by burying several charges in a row at optimum depth in the center of the core as illustrated in Fig. 11.



Dam
Typical section

Fig. 10. Inspection gallery of dam used as explosive emplacement cavity.

This brief discussion of bulk explosives for destroying tunnels, bridges, and dams is by no means the limit of targets that the military engineer could destroy with these explosives. However, preliminary research indicates that the best results will occur when the charges are placed beneath the ground or water and the resulting cratering action is allowed to demolish the target. The possibility of broadening the scope of slurry usage as

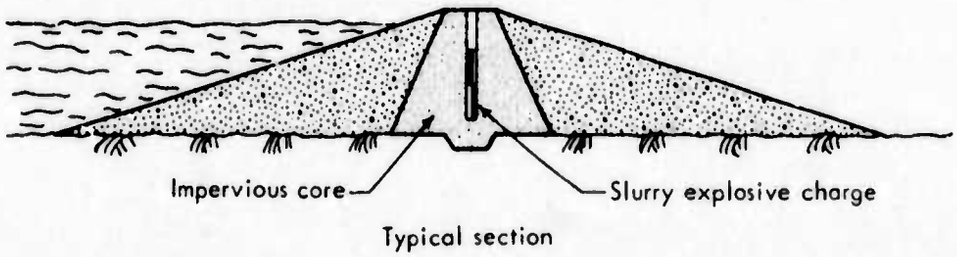


Fig. 11. Typical earth dam with row of explosives buried in core.

external charges will be evaluated by EERL upon completion of a series of small-scale tests to be conducted by the Weapons Effects Laboratory of the Waterways Experiment Station. While slurry explosives may not prove to be as effective in this role as some of the Army's present military explosives (i.e., Composition C-4, TNT), they could conceivably prove to be an advantageous substitute and, in some particular cases, more effective because of their physical characteristics, consistency, and ability to couple efficiently their energy to the surrounding material.

MILITARY CONSTRUCTION

The applicability of explosive excavation techniques is limited only by the ingenuity of the engineer staff and the tactical situation. However, the use of explosives for military construction is currently restricted to quarrying work and related drilling and blasting.

Quarries

Military operations usually require the upgrading of primary routes of communication. This upgrading obviously leads to a large demand for rock and rock products. Within the United States, ANFO and slurries have largely replaced dynamites in quarries and open-pit mines. Production has been increased as larger boreholes and increased spacing have been used. An artist's conception of the use of a row of several large-yield chemical explosives in quarrying work is presented in Fig. 12. A quarry produced in this manner would decrease considerably the daily requirement for drilling and blasting.

Field Fortifications and Storage Facilities

The capability to excavate material rapidly and to form a crater of predictable dimensions is being refined.

Fig. 12. Quarrying work using larger yield chemical explosives.[6]

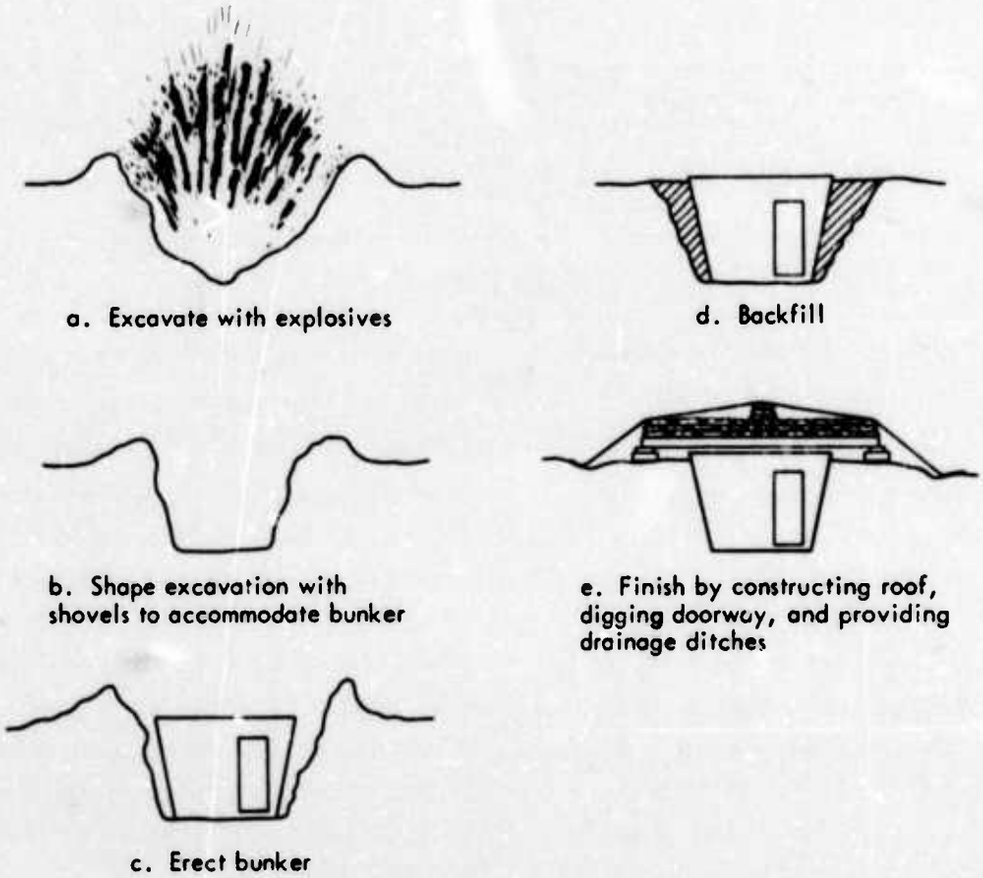


Fig. 13. Portable bunker installation procedure.[7]

The creation of emplacement holes has always been a problem, especially for the installation of bunkers or other structures that require the protection afforded by underground emplacement.

Fig. 14. Erection of portable bunker walls in crater (floor is covered with loose earth at this stage of construction).[7]

Rapid excavation with explosives is ideally suited to situations in which environmental conditions preclude the use of heavy earthmoving equipment or in which equipment is simply not available. Such methods may also be advantageous if hasty excavation is necessary due to immediate operations demands. Explosives can create sites that have stable bases for the construction of concrete bunkers or other buried structures. The ejecta from the resulting crater can be used as backfill material. A portable bunker tested by the Waterways Experiment Station is an example of a rapidly constructed structure that could be placed in an explosive crater for tactical use[7] (see Figs. 13 and 14). In addition, as demonstrated in an EERL study of the

protection of POL storage facilities,[8] the crater configuration is ideal for the protection and safety features required for the storage of POL products in standard steel tanks or in collapsible bladders in the forward area (see Figs. 15 and 16). Tests have also been conducted to evaluate explosives for producing instant foxholes.

Roadway Cuts

Row cratering with small charges can excavate ditches for construction drainage, culvert emplacement, or any other drainage structures. In conjunction with row craters, the technique of directed blasting may be employed. It has been found that, especially on side hills or sloped areas, by varying the charge weights in adjacent rows and by using delayed firing the ejecta can be deposited in predetermined configurations downhill. This technique can result in cuts for roadways or other structures that would require little cleanup prior to completion. In road building, especially, this technique could result in savings in time and equipment. This procedure has been demonstrated on several projects by EERL. Another possible application of directed blasting is the creation of temporary small dams either to divert streams or to hold them long enough so that culvert or bridge work can be accomplished (see Fig. 17). In areas subjected to periodic flooding, this technique could possibly be used to protect critical areas or structures. Small dams could also produce holding basins as water sources for water purification units operating in arid and semiarid areas.

Obstacle Removal

Often in this mechanical age the fact is lost that explosives are efficient means of demolition, and obstacle removal is often overlooked. Explosives have been used commercially to safely demolish

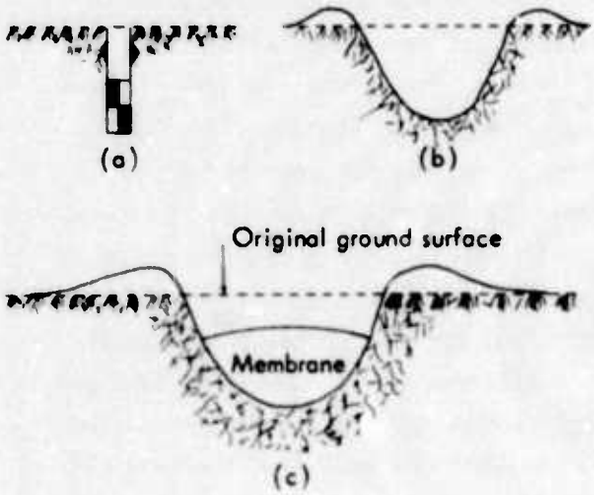


Fig. 15. Single crater for emplacement of POL storage facility.[8]

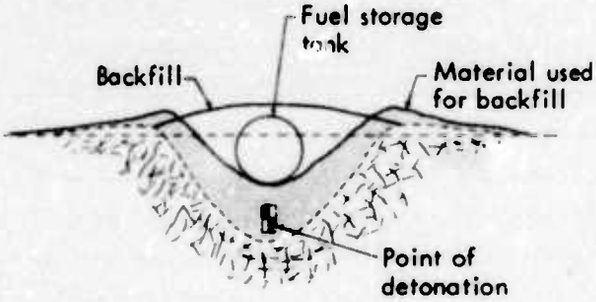


Fig. 16. Row crater for emplacement of buried fuel storage tank.[8]

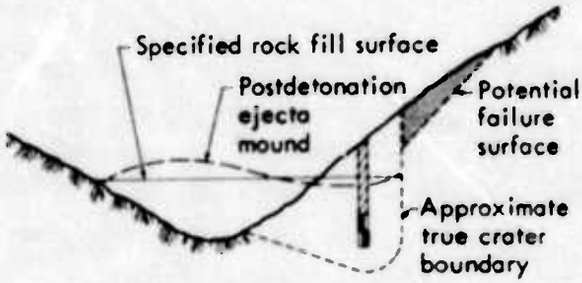


Fig. 17. Cut-and-fill operation using cratering techniques.

buildings, bridges, and other structures in populated areas. Militarily, these same techniques can be used to clear areas for new construction, to remove obstacles to vehicular or water traffic, or to prevent damage to existing bridges from debris accumulation. Explosives are uniquely suited for removing ice that threatens to clog or close ferry sites, harbors, or other water facilities.

The potential for explosives to supplement mechanical construction operations is just now being explored. Shaped charges can concentrate the power of an explosion into an energetic jet of metal particles sufficient to penetrate earth, rock, and concrete. Explosives can be used to cut steel and other strong materials. In certain media, drill holes have been enlarged by means of the "hole springing" technique.[9] Hole springing is a method of enlarging the bottom portion of a blast hole so that a large charge may be concentrated without increasing drilling diameter. Successively larger charges are loaded and fired until the desired volume is obtained. More exotic uses of explosives that may have military construction applications are soil compaction, metal working and forming, and explosive welding.

EERL Research

Research conducted by EERL has been concentrated on the applications of large-scale uses of bulk explosives in construction. The central idea is that explosives can be made to do more work than just break up rock; various types of excavations can be designed and produced safely, quickly, and in many cases, more efficiently than by the use of other techniques.

Studies, augmented by laboratory and field experiments, indicate the following to be the most attractive practical projects at this time: canals, waterway connections, harbors, channel deepening and widening, and highway and railway cuts, and quarries. Limited studies indicate that expedient (blasted-into-place) dams and overburden removal are potential applications for explosive excavation techniques. These concepts and some of the actual experiments conducted will be discussed briefly in the following paragraphs. For details on specific projects, the reader is directed to the list of references.

Canals

A canal constructed by explosive excavation can be formed by a linear crater created by one or more rows of charges. The charges might be detonated all at once or in an ordered sequence, or the length could be divided into short segments and each segment detonated independently of the others. The crater cross section could be used with little or no modification. The design would ordinarily specify a certain width of excavation at a specified depth.

Important factors that will have an influence on the explosive excavation design include the route of the canal, the engineering properties of the site medium, and such engineering requirements as length, width, and depth (navigation prism), and side slopes of the canal.

An excellent example of a canal excavated by chemical explosives is the Pre-Gondola series of row craters at Fort Peck, Montana.[1] These craters were produced as a part of an experimental

cratering program carried out by EERL. Figure 18 gives a good indication of the size of the completed canal. Its total length measures 1370 ft. The crater width at water level averages 150 ft, and the water depth at center line averages 26 ft.

Watercourses

A watercourse is defined as a canal, channel, or ditch for the specific purpose of moving water from one geographic area to another. Examples of watercourses include irrigation and drainage ditches, floodways, and spillways.

The shape of explosively formed row craters is well-suited to some applications of this type. The craters for a watercourse would be similar to those used for the canal. A navigation prism would ordinarily not be required.

The primary design consideration in these applications is the discharge capacity of the watercourse. This capacity is established by project criteria and is reflected in the cross section, depth, and gradient of the channel. Side slopes must be stable. Other factors important in the design include those discussed for a canal.

Spillways or connecting channels associated with dam and lake projects are typical types of watercourses that have been studied and found to be amenable to explosive excavation technology. Projects of this type can be accomplished by means of the design criteria presented in Chapter 6.

Harbors

The construction of a harbor for small-to-medium-sized boats, on land adjacent to water or offshore in shallow water, is within the capability of chemical explosive excavation technology. A harbor complex will normally consist of a mooring basin, a turning basin, and an inlet channel. The major considerations in harbor excavations are the required size and depth of the basin and channel. In certain regions the tidal range will be an important factor in fixing the required depth.

An example of a harbor produced by explosive excavation is Project Tugboat, a small-boat harbor excavated offshore in coral material at Kawaihae, Hawaii, by EERL in April and May 1970.[10] Figure 19 is an aerial view of the harbor site. The design requirement was for an entrance channel 12 ft deep and 120 ft wide, connected to a berthing area 12 ft deep and 240 by 240 ft in area (see Fig. 20). The coral, a low-strength porous rock, was under 6 ft of water. The excavation design used a row of eight 10-ton charges for the channel and a square array of four 10-ton charges for the berthing basin. The results shown as the shaded area of Fig. 20 proved this design to be conservative.

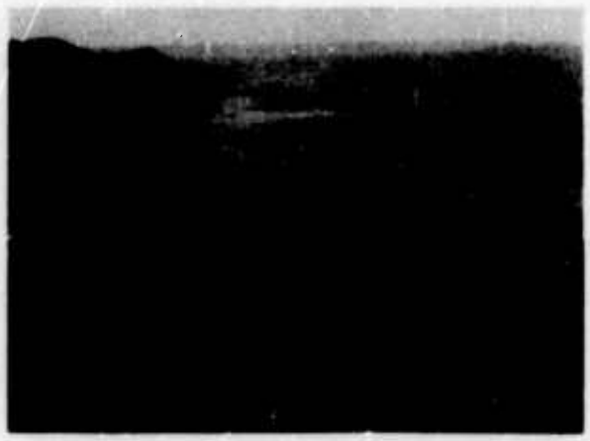


Fig. 18. Completed Pre-Gondola canal (boat is 42 ft in length).[1]

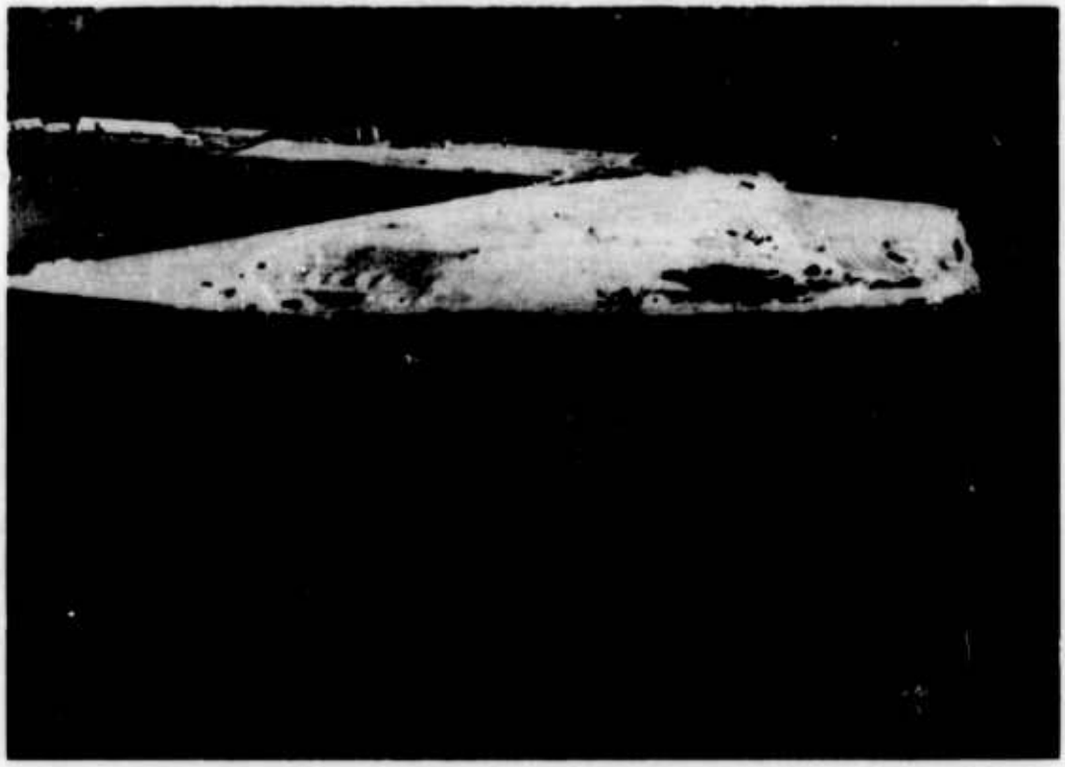


Fig. 19. Aerial view of Project Tugboat Site, Kawaihae Bay, Hawaii, showing areas excavated with explosives.

Channel Improvements

Explosive excavation shows great promise as a means of widening and deepening existing river and intracoastal waterway channels and of removing navigational hazards, such as rock outcrops and shoals. The depth and width of the navigation prism are the principal criteria affecting an explosive excavation design. These criteria, in turn, are dependent on the vessels using the waterways and the traffic plan, and will be influenced by local site conditions to include meteorologic, hydrologic, and geologic data in the same manner as harbors. Single-row craters may be used in some instances, but it is expected that most projects of this type will be underwater rock-removal projects requiring multiple rows of charges fired with delays between rows.

Highway and Railway Cuts

The principal criterion for the development of an explosive excavation design for highway or railroad cuts is the required width of the cut at the finished grade elevation. The required width at a specified depth criterion is somewhat analogous to the navigation prism requirement for canals. Proper drainage measures must be provided throughout construction.

Highway and railway cuts may be excavated with single- or multiple-row-charge craters. The fallback and ejecta are potential sources of subgrade material. If the rubble material is of

suitable quality, the costs of producing, hauling, and placing aggregate from another site can be eliminated.

A railway cut was excavated by EERL in December 1970 at the Trinidad Dam and Lake Project in Colorado.[11] The design used two parallel rows of charges. One row contained eighteen 1-ton charges, which were detonated simultaneously. The second row contained twelve 2-ton and two 1-ton charges, which were detonated simultaneously 150 msec later. Approximately 18,000 yd^3 of sandstone-shale was excavated leaving a broad, relatively flat-bottomed crater more than 400 ft long and up to 300 ft deep virtually coincident with the predicted size and shape. Figure 21 shows a typical cross section resulting from the detonation. This cut is an excellent illustration of the capability of explosive excavation for projects of this type.

Quarries

A variation of explosive quarrying is the "bulking" or "mounding" of rock with

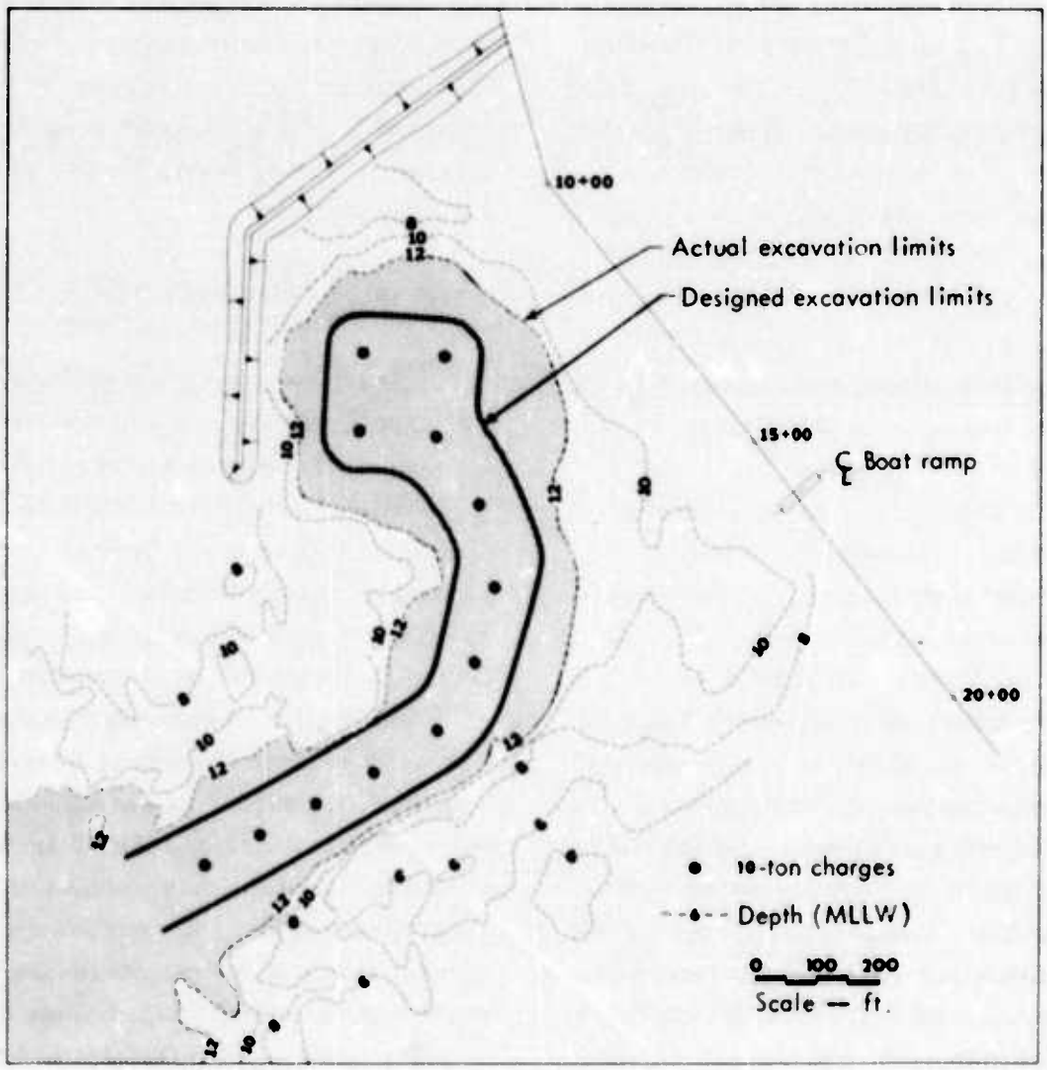


Fig. 20. Planned area of excavation and final 12-ft depth contours after remedial detonations, Project Tugboat.[10]

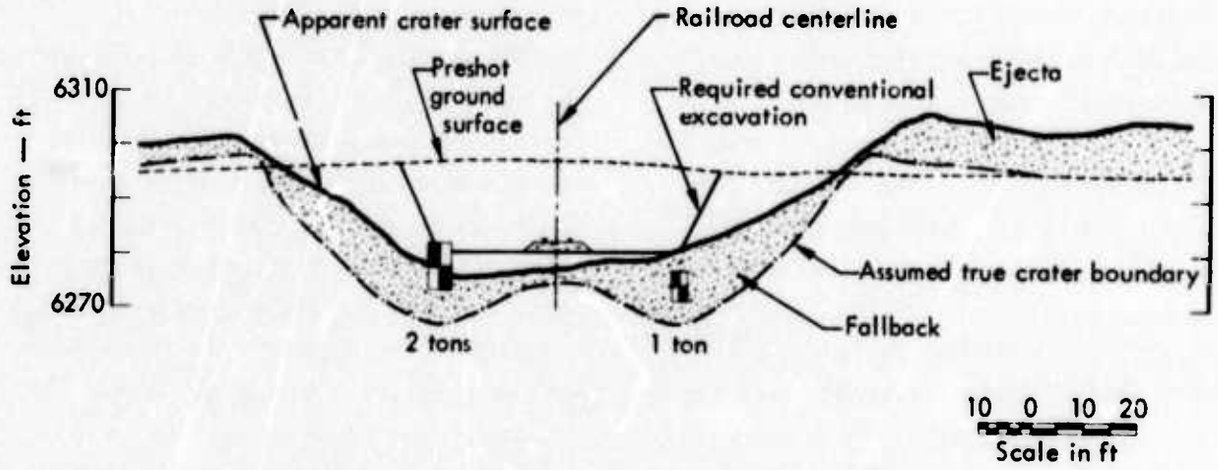


Fig. 21. Typical cross section of Project Trinidad railway cut (at Station 94+60).[1]

charges detonated at or near quarrying depth—see Fig. 3(c). The broken material can be removed from the true crater by conventional means. This approach appears to be more economical than cratering when the material excavated must be used for fill. For mounding detonations, it may be possible to control the limits of fracturing by controlled blasting techniques, such as presplitting. Further testing is required to prove this concept.

An example of a quarrying application in Australia was reported in Ref. 12. According to the report, 500 tons of explosives were used to produce 1.5 million yd^3 of fill for the Ord River Dam.

Overburden stripping with single- or multiple-rows of explosive charges has great potential for exposing mineral deposits and quarry rock. By careful control of the position and detonation of the charges, deep deposits of ore or rock can be exposed quickly with a possible savings in time and labor over conventional means. The depth of the deposit, the surface relief, and the geologic characteristics of the overburden are the important considerations in determining the excavation design. Time delays between the firing of rows of charges may prove to be advantageous in this concept. Testing is required to develop the technique.

ANALYSIS OF ENGINEERING PROPERTIES AND BEHAVIOR

Each explosive excavation design must be based on the engineering properties influencing the behavior of the geologic materials. Geologic investigations, materials testing in the field or in the laboratory if time permits, and engineering analysis and judgment – all contribute to the explosive excavation.

Slope stability, settlement, and seepage are three aspects of interest when one is considering the behavior of the geologic materials. The long-term behavior of an explosive excavation is controlled mainly by the environment and the nature of the material. An explosive excavation in clay shale or soil is more adversely affected by weathering, water movement, and frost-heave action than an excavation in rock. Explosive

excavations in rock are relatively stable and resistant to environmental factors. The engineering properties of strength, compressibility, and permeability are analyzed in the following paragraphs as they influence the behavior; i.e., slope stability, settlement, and seepage.

Strength and Slope Stability

Slope stability analyses commonly include field observations, comparisons of the behavior of other slopes in the same materials, and calculations to determine a factor of safety. Although influenced by settlement and seepage, the stability of rock slopes is primarily determined by the shear strength of the material when the unconfined compressive strength is greater than approximately 3000 psi.[13] The same is true for soils, except that the limiting value of unconfined compressive strength is much lower.

The factor of safety is often expressed as the ratio of shear strength to shear stress. Average values of shear strength and stress along an assumed failure surface normally make up this ratio; however, with the advent of the computer, values are sometimes assessed for discrete increments all along the assumed failure surface. If the shear strength is greater than the shear stress, the slope is stable, and the factor of safety is greater than one. The degree of stability is related to the amount by which the factor of safety exceeds one.

This section will relate shear strength and shear stress to the stability of the fallback and rupture zones and will provide an evaluation of the state-of-the-art in the assessment of slope stability.

Strength and Stress

In order for a slope failure to occur in a crater, large masses of material must be displaced. The displacement must involve sliding between the displaced and intact rock mass and must be of sufficient magnitude to impair the engineering use of the excavation. Mere surface movement does not ordinarily constitute slope failure. In general, the resisting forces along a failure surface are partially frictional in nature and partially due to cohesion. The frictional portion is predominant, except for clay slopes. For a rubble mass, there is no cohesion; therefore, the only contribution to strength is friction.

Stability in Fallback Zone

A strength analysis of the fallback zone is primarily an assessment of the stability of cohesionless rubble slopes. Since strength is a function of the angle of internal friction and the stresses are a function of the geometry, a factor of safety for a dry cohesionless slope can be expressed as:

$$F.S. = \frac{\tan \phi}{\tan \beta} , \qquad (1)$$

where

ϕ = angle of internal friction

β = angle of deposition or slope inclination.

If the angle of repose is substituted for ϕ, and the angle of deposition, β, is measured, the factor of safety can be calculated. The factor of safety for dumped large aggregate, a situation roughly analogous to the formation of crater slopes, normally varies between 1.1 and 1.5.[14]

Stability of Rupture Zone

For other than very weak rocks, the strength of the rock mass in the rupture zone is governed by the spacing and the orientation of the natural and blast-induced fractures; therefore, careful site documentation is necessary. The materials near the crater have been highly fractured by the explosion. In the rupture zone, natural discontinuities are disrupted and large subsurface displacements are common.[15]

In general, blast-induced fractures in the rupture zone will be clean, and the potential strength along the fracture can be compared to the frictional resistance of rock against rock. Exceptions are possible where existing clay-filled fractures are widened by the blast. Analysis of data for dry materials indicates that an existing throughgoing clean discontinuity dipping toward the crater would have to be inclined steeper than about 30 deg to develop an unstable block.[16] A postdetonation throughgoing plane of weakness steeper than 30 deg is unlikely because of the uplift of material toward the surface. This unlikely situation plus the added stability provided by the fallback at a relatively flat crater slope makes it most probable that the rupture zone will be stable.[17] A separate slope stability analysis would be required for any modification of the initial loading, such as cutting the toe of the crater slope, building of structures on the slope, or the development of additional pore-water pressure from rain or freezing.

Compressibility and Settlement

Knowledge of the settlement characteristics of the fallback is required to determine the suitability of fallback materials as a foundation for such engineering structures as road and railroad beds. This knowledge is also needed to determine the potential reduction in the total height of an embankment and the subsequent effect on the integrity of any impermeable surface material.

There has been little documentation of either the measured settlement or compressibility characteristics of crater fallback materials. Limited observations have been made, and the general pattern of behavior for cohesionless fallback is similar to that experienced for crushed rock and dumped rockfill materials.[18]

The major portion of the settlement will occur immediately following deposition of the fallback in the crater. Additional settlement of the fallback with the passage of time will be slow. Increased saturation or loading (particularly dynamic) can initiate a marked and sudden increase in settlement as in any other fill. The degree and extent of settlement will vary with the weather and geological location of the excavation. In areas of high precipitation, water can saturate the excavated slopes and accelerate the crushing of rock points by the weakening of the rock and the inducing of larger settlement. The crushed rock would then continue to settle at a decreasing rate. Settlement will tend to steepen the fallback slopes, but also will tend to increase the density of the materials with a resulting increase in shear strength. The former effect is detrimental to stability, but the latter is beneficial.

Analytical methods of assessing the magnitude of potential settlement involve

determining the compressibility characteristics of crater fallback material, determining the stresses involved, and applying the stresses to the compressibility characteristics. Techniques for laboratory measurements of compressibility of crushed rock have recently been developed, and research to extend this work is continuing.[11] Most of the information presently available on the settlement characteristics of large masses of rock aggregate is based on empirical data. Compression tests have shown that compressibility (1) increases with decreasing rock hardness, (2) is greater for angular, rough-surfaced particles than for rounded, smooth-surfaced particles, and (3) increases with decreasing uniformity of particle sizes.[19]

For the construction of an expedient (blasted-into-place) dam, the magnitude of total settlement may be estimated by the equation[19]:

$$S = 6.8 \times 10^{-5} H^{1.84},$$

where

S = settlement, ft

H = height of dam embankment, ft.

If a need develops to evaluate the compressibility of the rupture zone for foundations of projects such as bridges or spillways, the standard methods of testing in situ rock masses can be used. These techniques, including the plate jack test, pressure chamber tests, and borehole deformation tests, are discussed in Ref. 20. The compressibility of the rupture zone will be greater than that of the in situ rock mass because of the blast-induced fracturing.

Permeability and Seepage

Seepage and permeability characteristics of the excavation are a function of the local geology as well as of the disruptive effects of the explosion. The increase in permeability of the medium from the detonation can be expected to reduce the buildup of seepage pressures and pore-water pressures in most materials.

Permeability determines the ease of drainage and thus is an important factor in assessing long-term stability of excavations. Direct measurements of the permeability of fallback and ejecta materials have not been made in conjunction with postshot investigations of test craters to date.

Permeabilities have been estimated from grain-size curves, porosity, and certain empirically established relationships. In general, the permeability of the fallback is orders of magnitude larger than that of the preshot medium. Prior to the detonation in a relatively impervious rock, water flow, if any, is along fractures. Both the porosity and the permeability are increased significantly by the increased fracturing caused by the detonation. It is a logical assumption that the permeability of the rupture zone will approach that of the fallback near the true crater boundary, and that rupture zone permeability will decrease with distance from the crater in a manner similar to the decrease in fracturing and effective porosity (see Fig. 1).

Seepage data are required for all types of slope stability assessments and for the analysis of the flow-through characteristics of dam embankments. Present knowledge of crater seepage

pressures is limited, but additional information is available in Refs. 21 and 22.

On the basis of the permeabilities of the various crater zones and investigations of craters in clay shale, it is predicted that the groundwater surface for craters in rock will be depressed below its initial position. Such a depressed water table would be favorable to the potential stability of slopes. Flow from the water table into the crater, however, could cause an unstable drawdown condition to develop adjacent to the rupture zone. The water in or below the apparent crater will eventually reach an equilibrium level, with inflow from precipitation or by seepage from the surrounding area being equalled by outflow and evaporation. If the crater is composed of rocks that are susceptible to rapid disintegration due to weathering, the gradual breakdown of the rocks may, in time, fill the voids in the fallback and rupture zones. If such filling material is not washed out, permeability will be decreased, and the level of the groundwater surface may rise causing a less stable condition.

SUMMARY

The methods and procedures involved in postshot evaluation and the amount of additional construction required after an explosive excavation are dependent on the nature of the material and the proposed use of the excavation. Decisions of what data to collect must be made, methods of data collection must be devised, and knowledgeable individuals must be available to interpret the meaning of the data. Data collection can be simplified through the use of a list such as is presented in Table 1. The principles of soil and rock mechanics, engineering experience, and judgment are the essential elements in predicting the behavior of an excavation and the amount of postshot remedial work that may be necessary. Postshot construction methods are similar to those applied to projects accomplished entirely by conventional means.

A major portion of the work of EERL has been for civil works applications; however, the concepts and techniques may be applicable to present and future roles of military construction. Some of the suggested techniques presented in this chapter for using explosive excavation for producing obstacles, destroying targets, and supporting possible tactical military construction requirements have been tried and proven. Other applications of commercial explosives are still conceptual. The military engineer would conceivably have the option of undertaking an excavation or demolition project with the concepts that have been presented as alternatives to conventional explosive and mechanical excavation techniques. In many cases, both techniques will appear to be feasible; however, the final selection will depend on such factors as construction time available, available manpower and equipment, explosive requirements and availability, location, and the threat of enemy action. Under some tactical conditions, the use of explosive excavation or demolition techniques could be the only logical choice.

Explosives are a general tool in the military construction inventory and they should be utilized to the fullest extent. With a few exceptions, the information

Table 1. Project criteria and site data required to develop preliminary designs and to analyze feasibility of using explosive excavation techniques.

Criteria or data required	Application								
	Canals	Watercourses	Harbors	Channel deepening, widening; obstacle removal	Highway, railway cuts	Quarries	Expedient dams construction	Tunnel, bridge, and dam destruction	Tank craters
A. Project Criteria									
Project width at project depth	x		x	x	x				x
Alinement limitations	x	x	x	x	x				x
Gradient limitations		x			x				
Subgrade specifications		x			x		x		
Cross-section area requirements		x					x		
Plan dimensions			x	x			x	x	
Volume requirements						x	x		x
Gradation limitations						x	x		
B. Site Data									
Project topography	x	x	x	x	x	x	x	x	x
Hydrographic survey data	x	x	x	x			x		
Currents	x	x	x	x			x		
Tides	x		x	x					
Wave statistics	x		x	x					
General regional geology	x	x	x	x	x	x	x		x
Local geology (as necessary)									
Type of material	x	x	x	x	x	x	x	x	x
Density	x	x	x	x	x	x	x	x	x
Moisture content	x	x	x	x	x	x	x	x	x
Strength	x	x	x	x	x	x	x	x	x
Seismic velocity	x	x	x	x	x	x	x		
Bedding attitudes	x	x	x	x	x	x	x	x	x
Structural discontinuities	x	x	x	x	x	x	x	x	x
Joint and fracture pattern	x	x	x	x	x	x	x	x	x
Water table	x	x	x		x	x	x	x	x
Nearby structures survey	x	x	x	x	x	x	x	x	
Structural characteristics	x	x	x	x	x		x	x	
Wind aloft velocity statistics	x	x	x	x	x		x		

required to evaluate a military construction or demolition project using slurry explosives is the same as that required when these tasks are to be accomplished by conventional methods. As a result, little alteration of present data collection plans is required to study the feasibility of using explosive excavation techniques. Giving adequate consideration to the time available to complete the required mission and the local enemy situation, the engineer staff could utilize Table 1 to develop preliminary designs and to analyze the feasibility of slurry explosive excavation.

Chapter 3 Explosives

SCOPE

This chapter discusses explosives and the feasibility of their military employment. Explosives are surveyed in general, with particular emphasis placed on existing military explosives. Several nonmilitary explosives, including ANFO and other commercial explosives, are investigated with regard to possible military applications. Both types of explosives are then compared to determine where these nonmilitary explosives could fit in the Army's family of explosives.

EXPLOSIVES AND THEIR PROPERTIES

Explosives

Chemical explosives are oxygen-bearing compounds or mixtures that react violently (*i.e.*, detonate) when subjected to sudden shock and heat, and within microseconds are reduced to a mixture of gases at high temperature and intense pressure. Chemical explosives may be categorized as primary high explosives, secondary high explosives, and propellents.

Primary high explosives are very unstable substances that are extremely sensitive to ignition by heat or shock. The critical or minimum diameter at which detonation occurs is microscopic for primary high explosives. Sensitivity is the measure of the ease with which an explosive can be initiated. Primary high explosives are so sensitive that they must always be handled with caution, stored in small quantities, and stored separately from other explosive material. The three major primary explosives (lead azide, mercury fulminate, and lead styphnate) are the essential elements used in detonators. Detonators, such as fuses and delay blasting caps, may also contain timing and safety mechanisms. Lead azide, being relatively insensitive to flame, is a poor primary explosive for fused initiators, but is excellent for exploding bridge-wire electric blasting caps. Lead styphnate is used as an additive to improve the flame sensitivity of lead azide. A mixture of mercury fulminate and 10 to 20% potassium chlorate is used in fused blasting caps.

Secondary high explosives are sensitive substances that in practical application require the use of detonators and sometimes boosters for the initiation of explosive action. A detonator consists of a heat-sensitive primary high explosive and is used to initiate the main charge, whereas a booster is a sensitive secondary high explosive that reinforces the detonation wave from the detonator and delivers a stronger shock wave to the main explosive charge. Although the practical use of secondary explosives requires detonators for initiation, it is possible for secondary explosives to detonate by means of flame, friction, or other heat sources or shock. In some cases, detonation does not follow ignition by flame or heat, unless large masses of explosives are heavily confined. Blasting agents are secondary high explosives that are primarily mixtures of inorganic nitrate oxidizers and carbonaceous fuels and may contain additional nonexplosive substances,

such as powdered aluminum or ferrosilicon. Slurries, sometimes called water gels, contain high proportions of ammonium nitrate in an aqueous solution and, depending on the remainder of the ingredients, can be classified as either blasting agents or explosives. Secondary high explosives are of primary concern for military engineering applications.

Propellents, or deflagrating explosives, can be initiated by an igniter, flame, spark, or priming agent. Deflagration is a reaction in which the reaction front is moving through the unreacted explosive slower than the speed of sound for that substance in contrast to detonations in which the front is moving faster than the speed of sound. The major distinction between these agents and high explosives is that they do not undergo the transition from deflagration to detonation under normal conditions. Black powder, nitrocellulose, and dinitrotoluene are propellents.

Properties

Knowledge of several explosive properties is necessary to determine how they may be best utilized. Among these are critical diameter, density, detonation velocity, detonation pressure, available energy, water resistance, and fume class.

Critical Diameter

Any explosive has a "critical diameter" below which detonation cannot be maintained. The critical diameter is related to the thickness of the reaction zone that exists directly behind the detonation wave and is usually the same order of magnitude. Confining the charge underground or in a strong jacket helps reduce the critical diameter, since the lateral expansion of the gaseous products is thereby retarded.

At charge diameters slightly above the critical diameter, detonation velocity and energy release are below values produced by a larger charge. At larger charge sizes, detonation velocity and energy release cease to be a function of charge dimension and a condition called "ideal detonation" ensues. Whenever detonation properties depend upon charge size, the detonation is termed "nonideal."

Density

The density of an explosive is usually expressed in terms of specific gravity (the ratio of the density of the explosive to the density of water under standard conditions) or cartridge count (the number of 1-1/4- × 8-in. cartridges in a 50-lb box). Cartridged explosives are encased for convenient handling and for protection against accidental ignition, release of fumes, and moisture intrusion. The specific gravity of commercial explosives varies from 0.6 to 1.7 with corresponding cartridge counts of 232 to 83.[23] The density of free-running explosives is often expressed as pounds of explosive per foot of charge length in a given size blast hole. Denser explosives almost always produce higher detonation velocities and pressures. An explosive with a specific gravity of less than 1.0 or a cartridge count greater than 140 will not sink in water.

Detonation Velocity

The detonation velocity of an explosive is the speed at which the detonation wave

travels through the explosive. It is dependent on the density of the explosive, the ingredients and their particle size, the charge diameter, and the degree of confinement. Detonation velocity may be expressed as either a confined or unconfined value and is usually expressed in feet per second (ft/sec), or meters per second (m/sec). Decreased particle size, increased charge diameter, and increased confinement all tend to increase the detonation velocity.

Explosives are available with confined detonation velocities on the order of 1,500 to 7,500 m/sec. Since it is usually not possible to provide complete confinement with cartridge explosives, confined detonation velocities generally are not attained. Unconfined velocities are usually 70 to 80% of confined velocities.[24]

Some explosives, particularly blasting agents, are more sensitive to diameter changes than others. In charges with large diameters, the velocity may be high, but as the charge diameter is reduced, the detonation velocity is reduced. For charges with diameters less than the critical value, detonation is no longer assured and misfires are likely.

Detonation Pressure

The term brisance is often used to describe the property now attributed to detonation pressure. Brisance is defined as shattering action, or the ability of an explosive to demolish a hard object when fired in direct contact.

Detonation pressure, probably the best indicator of the ability of an explosive to break solid hard rock, is proportional to the product of density and the square of the detonation velocity. The detonation pressure is an important explosive property because it affects the strength of the shock wave transmitted into the medium. The reflection of this shock wave at a free face is an important mechanism in rock breakage, particularly in hard rock. The relationship of detonation velocity and density to detonation pressure is somewhat complex. It has been approximated by the following equation:[25]

$$P_D = 4.18 \times 10^{-7} \left(\frac{S_g D^2}{1 + 0.80 S_g} \right),$$

where

P_D = detonation pressure, kbar (1 kbar = 14,504 lb/in^2)

S_g = specific gravity

D = detonation velocity, ft/sec.

Explosives are available with detonation pressures up to 300 kbar.

Available Energy

The energy per unit mass released when a chemical explosive detonates is called the heat of detonation and is measured in calories per gram. The ability of an explosive to perform work depends on this energy, which is maximized when there is sufficient oxygen for a complete reaction but not an excess acting as a diluent. Total energy is comprised of two components, shock and bubble energy. Shock energy results from the transmission of the detonation wave into the surrounding medium, while bubble energy is the work associated with the expanding gases sometimes referred to as PV (pressure-volume) work.

Water Resistance

The water resistance rating of an explosive is an indication of its ability

to withstand water without deteriorating or losing its sensitivity. Water resistance is expressed as excellent, very good, good, fair, or poor. If the time period an explosive is to be exposed to water is relatively short, a rating of good is adequate. If longer periods of exposure to water are required, a very good-to-excellent water-rated explosive is required.

Flume Class

In addition to water vapor, carbon dioxide, nitrogen, and undesirable poisonous gases (fumes), such as carbon monoxide and nitrogen oxides, are usually formed when an explosive is detonated. The fume class indicates the type and quantity of these undesirable gases. The better ratings are given to low fume-producing explosives.

Sensitiveness and Sensitivity

Sensitiveness of an explosive or blasting agent is a measure of its propagating ability. It is not to be confused with sensitivity, which is the measure of ease of initiation.[26] Increased sensitivity does not necessarily lead to improved propagation characteristics or blasting action. It can, however, lead to decreased safety requirements because these compositions generally are not handled in equipment or with methods developed for sensitive explosive compositions.

Explosive Ingredients

The ingredients used in the manufacture of secondary high explosives are classified on the basis of their functions as explosive bases, combustibles, oxygen carriers, antacids, absorbents, sensitizers, and flame depressants. Since some ingredients perform more than one function, they fall into more than one class. An explosive base is a solid or liquid that, upon the application of sufficient heat or shock, breaks down into gaseous products with an accompanying release of heat energy. Oxygen carriers and combustibles are added to an explosive to achieve oxygen balance. A combustible combines with excess oxygen in an explosive mixture to prevent the formation of nitrogen oxides. An oxygen carrier assures complete oxidation of the carbon in the explosive mixture to prevent the formation of carbon monoxide. The formation of nitrogen oxides or carbon monoxide is undesirable because these fumes are poisonous and their formation results in a heat of detonation lower than does the formation of carbon dioxide and nitrogen. A lower heat of detonation means a lower energy output and less efficient blasting. An antacid is added to an explosive to increase its stability in storage, and an absorbent is used when needed to absorb liquid explosive bases. Nonexplosive sensitizers improve the ease with which high-density slurry blasting agents will detonate. Flame depressants lower the heat of detonation of permissible explosives used in underground mines and, therefore, minimize the chance of ignition of gas or coal dust in the atmosphere of a mine. Table 2 lists some of the ingredients used in explosives.

MILITARY EXPLOSIVES

Characteristics

Many explosives have been studied for possible suitability for military use, yet

Table 2. Ingredients used in explosives.

Ingredient	Function
Ethylene glycol dinitrate	Explosive base; lowers freezing point
Nitrocellulose (guncotton)	Explosive base; gelatinizing agent
Nitroglycerin	Explosive base
Tetranitro-diglycerin	Explosive base; lowers freezing point
Nitrostarch	Explosive base; "nonheadache" explosives
Organic nitrocompounds	Explosive base; lowers freezing point
Trinitrotoluene (TNT)	Explosive base
Black powder	Explosive base; deflagrates
Pentaerythritol tetranitrate (PETN)	Explosive base; caps, detonating fuse
Lead azide	Explosive base; used in blasting caps
Mercury fulminate	Explosive base; once used in blasting caps
Ammonium nitrate	Explosive base; oxygen carrier
Liquid oxygen	Oxygen carrier
Sodium nitrate	Oxygen carrier
Potassium nitrate	Oxygen carrier
Ground coal	Combustible
Charcoal	Combustible
Paraffin	Combustible
Sulfur	Combustible
Fuel oil	Combustible
Wood pulp	Combustible; absorbent
Lampblack	Combustible
Chalk	Antacid
Calcium carbonate	Antacid
Zinc oxide	Antacid
Kieselguhr	Absorbent; prevents caking
Metallic power	Fuel-sensitizer for high-density slurries
Sodium chloride	Flame depressant (permissibles)

only a few have been found acceptable for such use and some of these have certain characteristics that are considered to be serious disadvantages. Requirements are such that only a few explosives can qualify for standardization. Properties essential to their functioning are as follows:

(1) Relative insensitivity to shock or friction

(2) Detonating velocity adequate for the purpose

(3) High power per unit of weight

(4) High density (weight per unit of volume)

(5) Stability adequate to retain usefulness for a reasonable time when stored in any climate at temperatures between -80°F and +165°F

(6) Positive detonation by easily prepared primers
(7) Suitability for use under water
(8) Convenient size and shape for packaging, storage, distribution, and handling by troops
(9) Capability of functioning over a wide range of temperatures

Principal Military Explosives

Military explosives are used to accomplish a large variety of objectives. Table 3 lists the characteristics of the principal explosives used for demolition. A wide range of characteristics is available. Some of the more important explosives and their utilization are[27]:

TNT

Trinitrotoluene (TNT) is one of the most important of military explosives. It is a stable, cap-sensitive compound, although not extremely sensitive, and has good water resistance. TNT can be either cast or pressed. In cast form it has a density of 1.56 g/cm^3 and a confined detonation velocity of approximately 23,000 ft/sec. When the TNT is in pressed form, densities and velocities are slightly lower. Molten TNT may be combined with other products to form different explosive mixtures. Because of its high detonation velocity, TNT is used in cutting and breaching and as a main or booster charge for general demolition in combat areas. It may be recast to form a charge to fit special munitions. It can be burned in the open in small quantities without exploding. If an attempt is made to destroy it by burning when it is confined or in large quantities, it will explode. Detonation may be accomplished by military electric and nonelectric blasting caps. TNT also serves as the base explosive in determining relative effectiveness factors of other explosives.

RDX (Cyclonite)

RDX is the base charge in the M6 and M7 electric and nonelectric blasting caps. It is highly sensitive and brisant and is the most powerful military explosive.

Composition C-3

C-3 is a plastic explosive consisting of 77 ± 2% RDX and 23 ± 2% explosive plasticizer. It is a yellow, putty-like solid that has a density of 1.60 g/cm^3. Because of its plasticity and high detonation velocity, Composition C-3 is ideally suited for cutting steel structural members. It may be easily molded in close contact to irregularly shaped objects and is an excellent underwater charge if enclosed in a container to prevent erosion. Detonation is accomplished with military electric and nonelectric caps. Composition C-3 has several drawbacks, however, such as its hardening at -29°, and its slight volatility and hygroscopicity at normal temperatures. These drawbacks led to the development of Composition C-4.

Composition C-4

C-4 contains the following ingredients:

	Percent
RDX	91
Polyisobutylene	2.1
Motor oil	1.6
Di-(2-ethylhexyl) sebacate	5.3
	100.0

It is a putty-like material of dirty-white to light-brown color that, because of its

Table 3. Characteristics of principal U.S. military explosives.[5]

Name	Principal use	Smallest cap required for detonation	Relative effectiveness as external charge	Velocity of detonation, fps	Value as cratering charge	Intensity of poisonous fumes	Water resistance	Packaging
TNT	Main charge, booster charge, cutting and breaching charge, general and military use in forward areas	Special blasting cap, electric or non-electric	1.00	23,000	Good	Dangerous	Excellent	1 lb, 50 or 56 to box
Tetrytol, M1, M2			1.20	23,000	Fair	Dangerous	Excellent	16 2½ lb blocks in wooden box
Composition C3 M3, M5			1.34	25,000	Excellent	Dangerous	Good	16 2½-lb blocks in wooden box
M5A1 Composition C4 M112			1.34	26,000	Excellent	Slight	Excellent	24 2½ lb blocks in wooden box
Ammonium nitrate (cratering charge)	Cratering and ditching	Special blasting cap, electric or nonelectric	0.42	14,800	Excellent	Dangerous	Poor	40 lb charge in metal can
Sheet explosive M186, M118 charge demolition	(See C-4)	(See C-4)	1.14	24,000	Poor	Slight	Excellent	80½ lb sheets/box 25 lb roll
Military dynamite M1	Quarrying stumping-ditching	(See C-4)	0.92	20,000	Good	Dangerous	Good	½ lb 100 to box
Straight dynamite (Commercial) 40%	Land clearing, cratering, quarrying, and general use in rear areas, such as ditching and stumping	No. 6 commercial cap, electric or non-electric	0.65	15,000	Good	Dangerous	Poor	102 Sticks per 50 lb box
50%			0.79	18,000			Good	103
60%			0.83	19,000			Excellent	106
Ammonia dynamite (Commercial) 40%			0.41	8,900	Excellent	Dangerous	Good	110 Sticks per 50 lb box
50%			0.46	11,000			Good	110
60%			0.53	12,700			Good	110
Gelatin dynamite 40%			0.42	8,000	Good	Slight	Good	130 Sticks per 50 lb box
50%			0.47	9,000	Good		Very Good	120
60%			0.76	16,000	Good		Very Good	110
PETN	Detonating cord	Special blasting cap, electric or nonelectric	1.66	20,000 24,000	NA	Slight	Good	
	Blasting cap	NA						
Tetryl	Booster charge	Special blasting cap electric or nonelectric	1.25	23,400	NA	Dangerous	Excellent	
Composition B	Bangalore torpedo	Special blasting cap, electric or nonelectric	1.35	25,000	Good	Dangerous	Excellent	Bulk
Amatol 80/20	– do –		1.17	16,000	Excellent	Dangerous	Poor	
Black Powder	Time blasting fuze	NA	0.55	1310 Max Depends on Confinement	Fair	Dangerous	Poor	Bulk
Nitrostarch	Substitute for TNT	Special blasting cap, electric or nonelectric	0.80	15,000	Good	Dangerous	Satisfactory	1 lb blocks

high detonating velocity and plasticity, is used for cutting steel and timber or breaching concrete. It may be formed in sheets with an adhesive compound on one face for attachment to targets. Composition C-4 is slightly more powerful than C-3 and does not have C-3's disadvantages. It performs well under water and may be detonated by a military electric or nonelectric blasting cap.

Sheet Explosive

This plastic explosive comes in thin strips with an adhesive compound on one face. After the protective cover strip is pulled off, the sheet of explosive may be quickly pressed against any dry surface at a temperature higher than 32°F. A supplemental adhesive has been developed for colder, wet, or underwater targets. It is particularly suitable for cutting steel and may be used in bulk or cut to accurate width and uniform thickness. A military blasting cap is used for detonation.

PETN

Pentaerythnite tetranitrate is one of the most powerful military explosives, almost equal in force to nitroglycerine and RDX. In various degrees of granulation it is used as a priming composition in detonators, as a base charge in blasting caps, and as a core in detonating cord. When used in detonating cord, it has a detonation velocity of 24,000 ft/sec and is relatively insensitive to friction and shock.

Other Explosives

Composition B, Amatol, Pentolite, Ednatol, and Tetrytol are explosives formed with different combinations of the previously mentioned pure explosives. They are used in special purpose demolition equipment, such as bangalore torpedoes and shaped charges.

Dynamites and Ammonium Nitrate

Although classified as military explosives, dynamites and ammonium nitrate (AN) are characterized by low-to-medium detonation velocities and are less effective as external charges. Ammonium nitrate is used as the main ingredient in cratering charges. Dynamites are used for excavation, cratering, and general demolition in rear areas. Dynamites are not generally used in forward areas, but are acceptable in emergencies when other more suitable explosives are lacking. Dynamites of both commercial and military formulations are available. Commercial types of dynamite are straight, ammonia, gelatin, and ammonia-gelatin. All commercial dynamites must be handled with caution because they may be exploded by flame, sparks, friction, and sharp blows, including impact from bullets or projectile fragments. They must also be inspected and turned periodically to avoid separation of the nitroglycerine from the porous base.

Straight Dynamite

Straight nitroglycerine dynamite consists of nitroglycerine, sodium nitrate, an antacid, a carbonaceous fuel, and sometimes sulfur. Straight dynamites are named according to the percentage by weight of nitroglycerine they contain. Because of the tendency of nitroglycerine to freeze at low temperature, a substitute explosive oil usually replaces part or all of the nitroglycerine. Due to its high

detonation velocity, straight dynamite has an excellent shattering action, though poor fume qualities make it unacceptable for underground use or in poorly ventilated areas. Higher grades resist water well, but this resistance depreciates in lower grades. Fifty percent straight dynamite is commonly referred to as ditching dynamite. It has proved advantageous in ditching operations where sympathetic detonations between adjacent charges eliminate multiple cap or detonating fuse operations. High cost, sensitivity to shock and friction, and high flammability have contributed to the curtailment of the use of this type of dynamite in recent years.

Ammonia Dynamite

Ammonia dynamite is similar to straight dynamite except that a portion of the nitroglycerine and sodium nitrate is replaced by AN. Ammonia dynamite is also classified by an equivalent percentage of nitroglycerine even though it contains little or no nitroglycerine. Ammonia dynamites are generally lower in detonation velocity and density, and have better fume and shock sensitivity characteristics. One of the most widely used cartridged explosives, it works best where the rock is not extremely hard and water conditions are not severe.

Gelatin Dynamite

Gelatin dynamite is a dense, plastic explosive consisting of nitroglycerine or other explosive oil gelatinized with nitrocellulose, an antacid, sodium nitrate, carbonaceous fuel, and sometimes sulfur. The percentage ratings of gelatin dynamite indicate its equivalence to straight dynamite. Gelatin dynamite has been used in very hard rock and as a bottom charge in an explosive column. The more economical ammonia dynamites have replaced gelatins in most applications, though gelatins are still used in underwater blasting and in deep well shooting.

Ammonia-Gelatin Dynamite

Ammonia gelatin is a straight gelatin with a portion of the nitroglycerine and sodium nitrate replaced by AN. Percentage classifications of ammonium gelatin are identical to ammonia and gelatin dynamites. It has better fume qualities and water resistance than gelatins, making it suitable for both underground and underwater work.

Military Dynamite

Military dynamite M1 is a general use, medium velocity blasting explosive to replace commercial dynamites in military construction, quarrying, and service demolition work. Because it contains no nitroglycerine, military dynamite is safer to transport, to store, and to handle than commercial dynamite and is relatively insensitive to friction, drop impact, and rifle bullet impact. It will not freeze in cold storage, nor exude in hot storage. Primary ingredients are:

	Percent
RDX	75 ± 1.0
TNT	15 ± 0.5
Grade SAE No. 10 oil plus polyisobutylene	5 ± 0.5
Cornstarch	5 ± 0.5

Military dynamite may be detonated by a military electric or nonelectric blasting cap or by detonating cord.

Ammonium Nitrate

Ammonium nitrate is the least sensitive military explosive. Because of its insensitivity, the 40-lb cratering charge is provided with a TNT booster to permit priming by ordinary means. The rate of detonation of AN is affected by its particle size, apparent density, degree of confinement, and the efficiency of the booster charge. The detonation rate is increased by decreased particle size, decreased apparent density, and increased confinement. Ammonium nitrate is inefficient for cutting or breaching operations because of its low velocity and low shattering power. On the other hand, it produces a pushing or heaving effect, which makes it an excellent cratering and ditching charge. Readily absorbing moisture, AN becomes less efficient and less sensitive to initiation as saturation increases. Not toxic, AN is, however, a fire hazard, since it is a powerful oxidizing agent and will increase the intensity of combustion of any flammable material mixed or adjacent to it.

Nitramon

Nitramon is a sensitized AN, which was first introduced in 1932 by DuPont. It is very similar to AN but by sensitizing, it achieves a better oxygen balance thus allowing a higher total energy to develop.

Special Purpose Explosives

The explosives discussed thus far are the principal military explosives. Several of these explosives are used in special configurations to achieve certain effects. Included in this category are special mine clearing charges, such as the bangalore torpedo or the projected charge demolition kit. To produce a highly penetrating jet of metal, 15- and 40-lb shaped charges are available. Military explosives can also be used to improvise charges, such as package or pole charges. The wide range of characteristics available with military explosives makes it possible to choose at least one that can accomplish a task with the likelihood that several others could also perform the same task adequately.

COMMERCIAL EXPLOSIVES AND BLASTING AGENTS

General

While military explosives encompass a wide range of properties, there are many explosives in use throughout the world that are not classified as military explosives. Primary among these are blasting agents that are cap-insensitive chemical compositions or mixtures containing no explosive ingredient and requiring a high-explosive primer for detonation. These agents fall into two general categories: dry blasting agents and slurries. Some of the blasting agents are actually classified as high explosives in that they contain high-explosive ingredients. Another category of commercial explosives is the new component high explosives. Sensitized and unsensitized nitromethanes are also used as explosives.

Dry Blasting Agents

Ammonium nitrate is currently the least expensive primary ingredient for cratering explosives. A common fertilizer, AN can be detonated only when properly confined and strongly initiated.

In dry powder form AN is susceptible to spontaneous combustion and possible detonation if stored in large quantities. For explosive applications, AN prills (i.e., pea-sized pellets with very high porosity), are usually employed because they are more easily handled and are more stable.

Pure AN contains an excess of oxygen. When detonated, it produces N_2, H_2O, and O_2 and liberates 327 cal/g. Oxygen balance can be achieved by adding 5.5% fuel oil, which is readily absorbed by the prills. A mixture of 94.5% AN prills and 5.5% fuel oil is termed ANFO and is the least expensive cratering explosive currently available. Upon detonation ANFO produces H_2O and N_2 and liberates 890 cal/g, or approximately three times as much energy as pure AN. Numerous firms manufacture ANFO commercially at prices ranging from 2 to 10 cts/lb. The cratering characteristics of ANFO are very similar to TNT and 60% gelatin dynamite. The cost of ANFO, however, is less than a third as much.

ANFO has two major drawbacks: (1) its low density (0.8 to 1.0 g/cm^3) requires larger and more expensive emplacement cavities than more dense explosives, and (2) its hydroscopy—ANFO readily absorbs moisture, which rapidly increases its decomposition rate and destroys its capacity to detonate. For example, dry ANFO will detonate satisfactorily in a 1-in. steel tube, but with 5% water, the critical diameter for this degree of confinement is increased to about 4 in. As a result, if groundwater is present, ANFO must be placed in sealed packages, thereby raising emplacement costs. Properties of AN and ANFO are listed in Table 4.

Metalized Ammonium Nitrate

Powdered aluminum in AN ideally produces H_2O, Al_2O_3, and N_2 upon detonation and liberates up to 1975 cal/g, which is about twice the energy release of ANFO. An oxygen-balanced mixture of AN and Al contains 18.4% aluminum by weight. More aluminum can be added if additional oxygen is available. Water is sometimes added to provide extra oxygen. Dry mixtures of ammonium nitrate and aluminum are called "ammonols."

Table 4. Properties of AN and ANFO.[1]

Composition	Density (g/cm^3)	Detonation pressure (kbar)	Detonation velocity (m/sec)	Heat of detonation (cal/g)
AN powder	1.07	44	4,100	327
Prilled AN	0.81	33	4,100	327
93% prilled AN 2% fuel oil	0.80	40	4,100	570
94.5% prilled AN 5.5% fuel oil	0.93	60	4,560	890
90% prilled AN 10% fuel oil	0.80	45	4,100	760

The aluminum reaction is very complex and does not produce Al_2O_3 directly. Usually Al_2O is formed initially in the reaction zone behind the detonation wave and the more stable Al_2O_3 is formed later in the detonation process. The time required for the aluminum reaction depends upon the surface-to-volume ratio of the aluminum particles or flakes. A low ratio (i.e., large particles) can actually actually reduce detonation pressure as the aluminum initially acts somewhat as a diluent. Since the reaction continues well behind the detonation front, relatively high explosive pressures can be maintained for a long time, the result being a detonation wave with a low peak pressure compared to ideal explosives but a comparatively flat pressure profile behind it. Once transmitted into the medium, such stress waves decay less rapidly than the spiked shocks produced by nonmetallized explosives so that their influence can be felt over a greater distance.

Although 18.4% aluminum in ammonol is the theoretical requirement for oxygen balance, the aluminum particles most always contain surface oxides. Depending upon size distribution and quality, between 20 and 25% aluminum is actually required to achieve oxygen balance with dry ammonium nitrate.

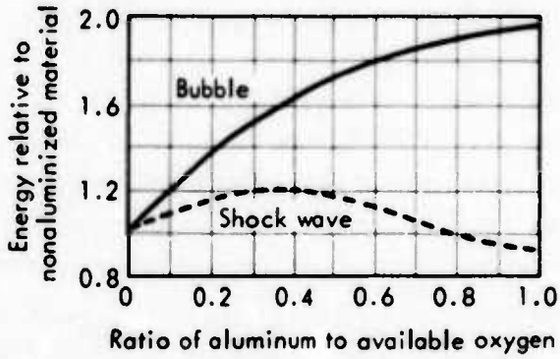


Fig. 22. Effect of aluminum on underwater shock and bubble energy.[1]

An important result of the aluminum reaction is a large increase in gas bubble energy. This reduction of the ratio of shock to gas bubble energy is desirable for cratering. The effect of aluminum on energy partitioning when the detonation is under water is shown in Fig. 22. Properties of some ammonols are listed in Table 5.

Slurry Explosives and Blasting Agents

The various nitrate-based explosives previously discussed (AN, ANFO, ammonol, etc.), typically suffer from one

Table 5. Properties of dry aluminized explosives.[1]

Composition	Density (g/cm^3)	Detonation pressure (kbar)	Detonation velocity (m/sec)	Heat of detonation (cal/g)
Ammonols:				
90% AN 10% Al	1.28	115	6,100	990
80% AN 20% Al	1.27	130	6,400	1,470
70% AN 30% Al	1.27	111	5,800	1,370

defect or another such as poor water resistance, excessive sensitivity, or poor storageability. Their desirability as explosives is thus impaired.

Through the addition of water, stabilizing agents, and gelling agents, not only are most of these problems overcome but the handling of the explosive is simplified. Such mixtures are called slurry explosives when they contain high-explosive ingredients; they are called slurry blasting agents when they do not contain such ingredients.

Slurry blasting agents contain non-explosive sensitizers or fuels (such as carbon, sulfur or aluminum) and are not cap-sensitive, whereas slurry explosives contain cap-sensitive ingredients (such as TNT) and the mixture itself may be cap-sensitive. Since the majority of the slurries presently manufactured are not cap-sensitive, all slurries, even those containing TNT, are often loosely referred to as blasting agents. This grouping is incorrect. The definition of blasting agent, established by the National Fire Protection Association,[28] requires that the mixture contain no ingredient that is classified as an explosive. The chemically active ingredients in most slurry explosives are AN, TNT, and water, while in slurry blasting agents they are AN, water, and sometimes sodium nitrate or aluminum.

Slurries are thickened and gelled with a gum, such as guar gum, to give them considerable water resistance. The gelling agent in slurries serves two purposes: (1) to insure a homogeneous mixture and to prevent the settling of components, and (2) to facilitate handling. The slurry and gelling agent can be mixed while the explosive is being pumped into the emplacement cavity where the slurry cures to a rubbery or jelly-like solid, which is water-resistant. Excellent coupling with the surrounding medium is thus assured, and void spaces within the explosive are minimized. Most slurries are heavier than water (1.2 to 1.6 g/cm^3) and, being highly water-resistant, may be emplaced under water (see Fig. 23). Some slurry emulsions have also been developed with similar properties.

Typical slurries contain 40 to 75% AN, 15 to 25% water, 1 to 5% stabilizing and gelling agent, plus aluminum or high explosive, or both. The addition of large quantities of aluminum produces an explosive with very high energy release at moderate detonation pressures. The presence of aluminum lowers peak pressures but provides higher sustained gas pressures during the expansion. Energy release per unit weight can exceed twice that of ANFO, and these slurries can excavate up to 80% more volume per unit weight than ANFO or TNT. However,

Fig. 23. Emplacing slurry explosive under water (used by permission of IRECO Chemicals, Salt Lake City).

energy release per unit cost is about the same as nonaluminized slurries and approximately one third that of ANFO. Nevertheless, aluminized slurries are often the best choice when emplacement costs overshadow explosives cost.[1]

Slurries require adequate priming with a high-velocity explosive to attain proper detonation velocities. Also, high-explosive boosters spaced along the borehole are often required to assure complete detonation. The consistency of most slurries ranges from fluid near 38°C (100°F) to rigid at freezing temperatures, although even at freezing temperatures some slurries maintain fluidity.

Ammonium nitrate slurries are frequently shipped and stored in plastic bags (see Fig. 24). They can be stored in most containers, including aluminum or steel (unless HNO_3 is a constituent); however, because the slurries are new, effects of long-term storage are not well known. Slurry blasting agents are very insensitive to flame and normally cannot be detonated with a No. 8 blasting cap. Their compressibility is low and thus they can be used under hydrostatic loads. Unconfined critical diameters are usually about 3 in. Properties of some selected slurries are given in Table 6.

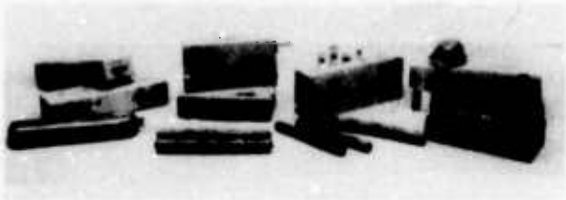


Fig. 24. Prepackaged slurries in plastic bags (used by permission of IRECO Chemicals, Salt Lake City).

Component Explosives

Component explosives generally refer to explosives that can be created in the field by the mixing of two separate nondetonating components. Both liquid and solid component explosives have been developed yielding a variety of explosive characteristics.[29] While most applications of this technique have been in the mining

Table 6. Properties of selected slurry explosives and blasting agents.[1]

No.[a]	Consistency	Density (g/cm^3)	Detonation pressure (kbar)	Detonation velocity (m/sec)	Heat of detonation (cal/g)	Percent aluminum	Contains HE?
1	Liquid	1.40	99	5,850	750	0	Yes
2	Liquid	1.40	104	6,050	730	0	Yes
3	Gel	1.30	60	4,300	750	2	No
4	Gel	1.33	66	4,500	1,110	8	No
5	Gel	1.20	85	5,700	1,450	20	No
6	Gel	1.50	81	5,000	1,950	35	No
7	Gel	1.65	80	5,200	2,050	—[b]	Yes
8	Gel	1.95	89	5,500	2,250	—[b]	Yes

[a] Brand names have been omitted.

[b] Aluminized, but percentage unknown.

Table 7. Confined detonation velocity and charge concentration of ANFO.[25]

Borehole diameter (in.)	Confined velocity (ft/sec)	Charge concentration (lb/ft of borehole)
1-1/2	7,000- 9,000	0.6- 0.7
2	8,500- 9,900	1.1- 1.3
3	10,000-10,800	2.5- 3.0
4	11,000-11,800	4.4- 5.2
5	11,500-12,500	6.9- 8.2
6	12,000-12,800	9.9-11.7
7	12,300-13,100	13.3-15.8
8	12,500-13,300	17.6-20.8
9	12,800-13,500	22.0-26.8
10	13,000-13,500	27.2-32.6
11	13,200-13,500	33.0-39.4
12	13,300-13,500	39.6-46.8

and quarrying fields, some high-performance liquid explosives have been developed. These explosives with high detonation velocities, densities, and energies compare favorably with the most powerful military high explosives. They may be shipped and stored as non-explosives. Additionally, component boosters are available that offer the same transportation and storage advantages.

Uses

Nitrate-based explosives, especially ANFO, have largely replaced dynamites in open-pit-mining and quarrying operations in which large boreholes are used. The specific gravity of ANFO varies from 0.75 to 0.95 depending on particle density and sizes. Table 7 shows how confined detonation velocity and charge concentrations of ANFO vary with borehole diameter. Pneumatic loading gives even higher values particularly in holes less than 3 in. Advantages of insensitive dry blasting agents are their safety, ease of loading, and low price. In the free-flowing form, they have a great advantage over cartridge explosives because they completely fill the borehole. This direct coupling to the borehole walls assures efficient use of explosive energy. As a cratering explosive, ANFO is the best in terms of cost and is slightly more effective than TNT. For bulk operations ANFO can be truck-delivered to the sites. Though ANFO requires additional priming, it presents a continuous explosive column; this characteristic simplifies the problem. Numerous methods have been devised to use ANFO in wet holes, although these methods tend to increase cost. Additional drawbacks of ANFO include increased drilling and loading costs due to low borehole pressures and low densities.

The development of slurries is continuing as a means to overcome the disadvantages of ANFO in certain mining and quarrying operations. Being denser

than water and water-resistant, slurries have proved most satisfactory in wet hole or underwater demolition.[30] When pumped downhole they displace the water with no loss of explosion effects. The detonation velocities of slurries range between 12,000 and 18,000 ft/sec, depending on ingredients used, charge diameter, degree of confinement, and density. However, the detonation velocity of a slurry is not as dependent on charge diameter as that of a dry blasting agent. Slurries consequently give the same advantageous direct borehole coupling as dry blasting agents as well as a higher detonation velocity and a higher density. Savings in costs realized by drilling smaller holes or using larger burden and spacing will often more than offset the higher cost per pound of explosive. Adding powdered aluminum as a sensitizer to slurries greatly increases the heat of explosion or the energy release. Aluminized slurries are used in extremely high-strength rock with excellent fragmentation results.

Slurries have also been used quite effectively to produce large craters and mounds in the experimental programs and civil works conducted by EERL. One advantage of slurries is that their higher densities result in low charge length-to-diameter ratios, thus allowing for more compact charges and less crater degradation. Slurries are also very easy to handle and load. They are packaged in plastic sausages or may be mixed and loaded directly in the borehole by truck (see Fig. 25). The range of properties available in slurries enables the engineer to choose the best combination of properties to fit his requirements. Highly

Fig. 25. Slurry mixing and pumping truck (used by permission of IRECO Chemicals, Salt Lake City).

sensitized slurries may be used as primers or boosters.

A combination of slurry and dry blasting agent may be used in the same borehole in "slurry boosting," with the bulk of the charge being dry blasting agent (see Fig. 26). In another application of slurry boosting, the slurry is placed in a zone in which fragmentation is difficult, such as a toe or a zone of high-strength rock in the burden. This technique will often give better overall economy than straight slurry or dry blasting agent. Slurry and ANFO have also been combined in a dual explosive cartridge in which ANFO is used as a central core surrounded by slurry.[31] Not only does this protect the ANFO from moisture, but by varying the amounts of each ingredient a range of explosive properties is available. Both slurries and medium-strength component explosives are packaged in convenient form for shattering large boulders in quarrying operations.

High-performance component liquid explosives have been developed for

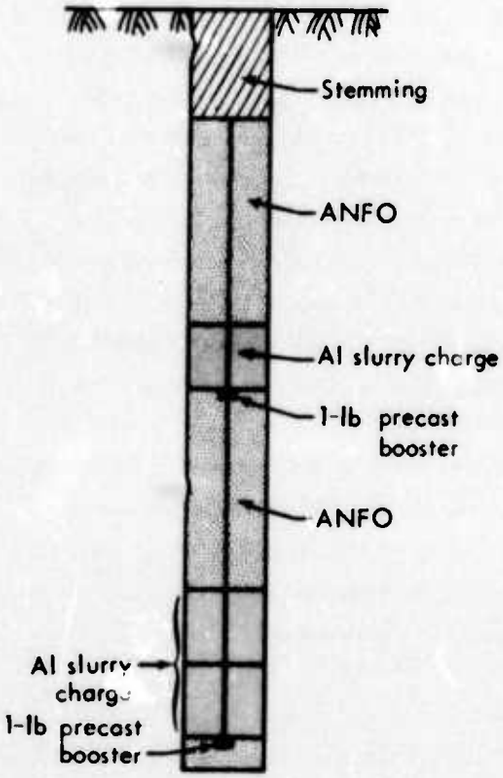


Fig. 26. Slurry boosting.

possible military use in tunnel destruction, minefield clearing, and expedient and barrier demolitions. Several varieties have been developed that can actually be absorbed into the ground and remain capable of detonation.

FEASIBILITY OF USING COMMERCIAL EXPLOSIVES IN MILITARY OPERATIONS

General

One of the development objectives of the U.S. Army should be to provide new and improved nonnuclear explosives and techniques for accomplishing a variety of military demolition tasks. Such tasks should include:

(1) Reduction of man-made and natural obstacles

(2) Creation of obstacles

(3) Reduction or excavation of natural materials for military construction

(4) Destruction and denial operations

Military demolition tasks are currently accomplished primarily by the use of block explosive charges and accessories, special explosive demolition charges and kits, and special purpose tools, mechanical devices, vehicles, and attachments. Skilled technicians with special training are required to employ explosives and their accessories and mechanical equipment. Explosives require special care and safety measures in handling, use, and storage. Army testing and adoption of new commercial developments, such as liquid, slurry, and gaseous demolition techniques and materiel, have not kept pace with the development of tactical doctrine, which emphasizes battlefield mobility, speed, and dispersion. The current doctrine requires decentralization of effort and increased capability of troops to perform demolition tasks. Greatly improved techniques must be developed so that presently available demolition materiel can be used with less logistic effort and special training. New demolition materiel and techniques are required so that troops other than trained engineers can easily and quickly perform the great number of demolition tasks that will be required.[32] It is within this general development context that the use of commercial explosives reviewed in the previous section will be analyzed.

Dry Blasting Agents

The Army has already adopted AN and nitramon as its cratering explosives. These explosives must be packaged in

waterproof metal cans. Normal employment concepts envision emplacement holes created by shaped charges except where prepositioned holes exist. The holes produced by these shaped charges are usually irregular in diameter and length. The cratering charges are designed primarily for the creation of obstacles, although they have been used for target destruction and construction on occasion.

ANFO with its low density, low detonation velocity, and large critical diameter is relatively ineffective as an external charge. In addition, due to its hydroscopy and low density, ANFO might prove cumbersome in large quantities on the battlefield. It does possess excellent potential for use in barriers as a replacement or supplement to the existing cratering charge. Although a water-tight metal container would provide excellent protection, it would also increase the cost of the charge. Industry has devised techniques for using ANFO in water, including the use of polyethylene tubing for protection. The flexibility of this tubing promotes effective filling of the borehole to improve media coupling. In explosively produced emplacement holes, ANFO offers a distinct advantage over canned cratering charges because it is rapidly loaded and assures excellent coupling. ANFO can be used in conjunction with existing cratering charges to fill all the voids and gaps, thus improving cratering results.

ANFO has become the leading commercial explosive in the United States, accounting for 75% by weight of all the explosives produced in 1970.[33] It has none of the safety problems of the various types of dynamites, it is more economical, and it has better borehole loading capabilities. In quarrying operations, large excavation projects, and ditching operations it could potentially replace almost all the commercial dynamites in the military system and be more economical than military dynamite. Disadvantages of ANFO include its requirement for protection in wet environments, extra boosting, and dependency on borehole diameter in developing maximum detonation velocities. Cost alone, however, merits careful consideration of expanded uses of ANFO in military operations, especially construction.

Slurries

Slurry explosives and slurry blasting agents are an assorted family of explosives that offer the versatility and flexibility so essential in military operations, and are safe and easy to use. New slurry formulations are continually being developed, each with distinct characteristics and advantages. A survey of current literature on 50 slurry products revealed the range of properties shown in Table 8.

Although this sample is incomplete and does not portray the combination of properties available for individual products, it does display the wide range of slurry mixes currently available commercially. This range of slurry properties

Table 8. Range of slurry properties.

Density, g/cm^3	1.11 - 1.60
Total energy, cal/g	470 - 1,171
Detonation velocity, m/sec	3,710 - 6,065
Detonation pressure, kbar	50.7 - 105.6

and products is a result of varying the ingredients in each particular slurry. Generally, the water, high-explosive, or aluminum content is varied. As mentioned previously, slurries may be truck-mixed. The Army is currently investigating this procedure to determine whether it is feasible for military use. A long-range goal of this program is to develop a series of slurry mixtures that can be produced by vehicle and are capable of performing a wide variety of military engineering missions.

The effectiveness of slurries as external charges has not been established. It is possible that certain slurry explosives or highly aluminized blasting agents would be effective external charges. A military family of slurries could include several explosives with high detonation velocity and detonation pressure and explosives with small critical diameters. Table 3 lists the relative effectiveness of various dynamites as external charges. Many high-energy slurries have higher detonation velocities and energies than any of the listed dynamites. In fact, some exhibit detonation properties similar to those of TNT. Because they are gels, they can be configured to establish intimate contact with any external surface. Tests should be conducted to determine the relative effectiveness of high-performance slurries as external charges.

Although slurries are more expensive than ANFO on a weight basis, their increased shock and gas bubble energy may result in a reduced charge size per hole and a reduction in the number of charges required per obstacle. Several selected slurries, ANFO, and TNT are compared in Table 9. Note that the bubble energy and cratering effectiveness of slurries increase with increased aluminum content. When cratering is carried on with large

Table 9. Measured properties and calculated parameters of representative cratering explosives.[1]

Explosive	Detonation pressure (kbar)	Bulk specific gravity	Detonation velocity (m/sec)	Impedance (m/sec)	Heat of detonation (cal/g)	Nominal cost ($/lb)	Excavated volume relative to equal weight of TNT[a]
ANFO	60	0.93	4,560	4,240	890	0.06 ± 0.04	1.0-1.1
AN slurry[b]	104	1.40	6,050	8,470	730	0.15 ± 0.05	1.0-1.2
AN slurry (2% Al)[c]	60	1.30	4,300	5,590	750	0.08 ± 0.05	1.0-1.2
AN slurry (8% Al)[c]	66	1.33	4,500	5,990	1,110	0.13 ± 0.05	1.2-1.4
AN slurry (20% Al)[c]	85	1.20	5,700	6,840	1,450	1.20 ± 0.07	1.5-1.7
An slurry (35% Al)[c]	81	1.50	5,000	7,500	1,950	0.25 ± 0.10	1.6-1.8
TNT	220	1.64	6,930	11,360	1,102	0.25 ± 0.05	1.00

[a]That is, "cratering effectiveness" as measured by small charge detonations in sand. Absolute cratering performance in terms of volume excavated per pound of explosive depends on the size of the shot; it is less for larger shots. Relative performance, on the other hand, is not as sensitive to charge size.

[b]HE sensitized.

[c]Slurry blasting agent.

charges, crater dimensions decrease as the charge length-to-charge diameter ratio is increased. Slurries with their higher densities permit the use of more compact charges for a given emplacement hole diameter. More importantly, their excellent water resistance makes slurries ideal for wet operations.

Principal applications for slurries may be in the combat engineering and military construction fields. Designed primarily to overcome the disadvantages of ANFO, slurries accounted for 9% by weight of all explosives produced in the United States in 1970.[33] Both ANFO and slurries would eliminate the Army's present quarrying doctrine of choosing the dynamite with the most advantageous cartridge size for use in quarry shots.

Component Explosives

Component explosives are relatively new and their use has been limited to specific applications. High-energy components have been developed for possible military use and are comparable to existing military high explosives. As liquids, they would complicate the Army's logistical system and would provide an explosive only as effective as those now available in the military system. Medium-strength, solid-liquid component explosives have been developed for use in secondary quarry blasts. The solid component is made in various sizes. After it is placed on the boulder, the liquid is added, and the explosive is detonated; results have been good. It should be noted that both ingredients are nonexplosive until combined. They are used only as small charges because mixing would be difficult for large charges. As component explosives are developed and their uses expanded they could become more important in filling the military's needs of the future.

SUMMARY

This chapter has provided a general overview of both military and commercial explosives in use today. The commercial explosives were evaluated in light of their possible addition to the military explosives family.

Target destruction generally requires an explosive with high energy, high detonation velocity, and high pressure capable of cutting hard materials (such as steel and concrete) when used in either an external or internal configuration. The Army currently utilizes TNT, Composition C-3, Composition C4, sheet explosive, and tetrytol for demolitions. High-energy component explosives are compatible with these and possess good safety characteristics. High-energy slurries also warrant additional investigation as possible candidates for target destruction applications.

Explosively produced obstacles are generally accomplished with military cratering charges of AN and nitramon. These explosives are effective; however, they have some disadvantages, which may be overcome by replacing or supplementing them with ANFO or slurries. Slurries, with their increased detonation velocities, pressures, and bubble energies may accomplish cratering tasks with less explosive.

Military construction demolitions are generally performed with either military or commercial dynamites. Commercial

excavation projects primarily use ANFO or slurries. ANFO has proven very economical and is excellent for dry hole blasting. Slurries were developed to overcome the disadvantages of ANFO and to offer a wide spectrum of explosive properties. They are water-resistant and may even be more economical than ANFO in hard rock because they can produce more rock with less explosive and fewer drill holes. Both ANFO and slurries have excellent handling and safety characteristics compared to commercial dynamites. Component explosives and slurries have been especially developed for secondary rock blasting in quarries.

The demand for rock and associated products has led to the development of cheaper, more efficient, and safer explosives to produce this rock. ANFO and slurries have demonstrated their superiority in these areas and are widely accepted by the explosives industry. These same explosives could be used for military construction and barriers and may even prove feasible for target destruction roles. A family of slurries could be as widely used on the battlefields of tomorrow as they are on construction sites of today.

Chapter 4 Geologic Media

SCOPE

This chapter presents the engineering properties of soil and rock materials that are pertinent to explosive excavation. The objective is to set forth those properties necessary to the evaluation of the feasibility of a proposed excavation.

The nomenclature of earth materials has always been a source of difficulty for the disciplines of geology and engineering. The existing Corps of Engineers Unified Soil Classification System,[34] does not meet all of the objectives listed above for explosive excavation. A limited literature search produced a variety of nomenclature for media classification already in existence.[35-37]

The engineering properties of soil and rock are important factors in nearly every civil engineering project. Soils are classified by index property tests. These tests do not measure the engineering properties of soil directly but were chosen because they are simple, relatively inexpensive, and reproducible tests that correlate with the engineering properties of the soil mass. The engineering properties of a rock mass cannot be predicted with the precision expected in soils. There are no widely accepted index properties that correlate with the engineering properties of the rock mass.

Rock masses are seldom monolithic but consist of a series of partially interlocking blocks usually referred to as joint blocks. The joint blocks are created by natural surfaces of weakness, such as joints, shear surfaces, bedding planes, and foliation surfaces. The spacing of these geological discontinuities depends on the rock type, the geological history, and the structure of the mass. The spacing and character of these surfaces determine whether the rock mass will have the properties of the individual joint blocks or those of the residual soil in the joints.

A CLASSIFICATION SYSTEM

General

This section outlines a media classification system developed by EERL in 1971 to be used in the design of explosive excavation projects. It does not put labels on rock types (as with soils, for example, in the Corps of Engineers Unified Soil Classification System[34]), but rather it provides a systematic approach to analyzing the engineering properties of the medium for a particular explosive excavation requirement. An outline of the classification system is presented in Table 10.

Primary considerations for classification are ease and expense of obtaining the data necessary to describe adequately the material for the intended purpose. Certain tests are normally run at any site at which a major project is contemplated. A complete engineering geology investigation of any site would include such items as core borings and logging, geophysical surveys, geologic mapping, and perhaps borehole photography and permeability measurements. Field samples would be subjected to various laboratory tests to complete the evaluation of the material properties.

Table 10. Media classification for explosive excavation.[42]

A. Primary Classification
I. Media
A. Common excavation
1. Soil
2. Clay shale
B. Rock excavation (generally requires drilling and blasting to excavate)
1. Weak rock (<4,000-psi unconfined compressive strength)
2. Intermediate-strength rock (4,000 to 16,000-psi unconfined compressive strength)
3. High-strength rock (greater than 16,000-psi unconfined compressive strength)
II. Lithology (for rock excavation) or soil classification (for common excavation) and geologic structures
III. Degree of saturation
A. Dry (<50% saturated)
B. Wet (50% ≤ % saturated ≤ 90%)
C. Saturated (>90% saturated)
IV. Joint spacing
A. Very close (less than 2 in.)
B. Close (3 in. to 1 ft)
C. Moderately close (1 to 3 ft)
D. Wide (3 to 10 ft)
E. Very wide (greater than 10 ft)
V. Thickness of bedding
A. Very thin (less than 2 in.)
B. Thin (2 in. to 1 ft)
C. Medium (1 to 3 ft)
D. Thick (3 to 10 ft)
E. Very thick (greater than 10 ft)
B. Secondary Classification
VI. Seismic velocity (compressional wave, V_p)
VII. Unconfined compressive strength values
VIII. Mass density
IX. Modulus of elasticity, E (taken tangent at 50% yield)
X. Abrasion
XI. Rock Quality Designator (RQD)

Note: Classification will be as refined as needed for the intended use.

The system described herein will use commonly performed tests as a framework for describing a material sufficiently for explosive excavation purposes.

Media

The medium is classified as to whether it is common or rock excavation. Such classification is in accordance with Corps of Engineers Guide Specifications CE-802.[38] Common excavation includes the satisfactory removal and disposition of all materials not classified as rock excavation. Rock excavation includes blasting, excavating, grading, and disposing of material classified as rock; it includes also the satisfactory removal and disposition of boulders one-half cubic yard or more in volume, solid rock, rock material that is in ledges, bedding deposits, and unstratified masses that cannot be removed without systematic drilling and blasting, and conglomerate deposits so firmly cemented as to possess the characteristics of solid rock that are impossible to excavate without systematic drilling and blasting.

If the medium is "common," the established Unified Soil Classification System is used. One type of material, clay shale, has been set apart as a separate entity in the development of the system. This action results largely from the extensive studies and experiments conducted in clay shale by EERL. Clay shale is listed under common excavation.

The degree of difficulty in excavating rock is related to the strength of the in situ rock. Numerous classification systems for in situ rock are available and are reviewed by Deere, et al.[20] These systems are assessed and provide valuable background material.

Three categories of strength were found to be sufficient for explosive excavation purposes. Unconfined compressive strength was selected as the basic index property because of its widespread use. These categories are:

Weak rock: less than 4,000-psi unconfined compressive strength

Intermediate-strength rock: 4,000 to 16,000-psi unconfined compressive strength

High-strength rock: greater than 16,000-psi unconfined compressive strength

These ranges of values are illustrated on the Miller-Deere Chart[35] shown in Fig. 27.

The physical properties of the medium in which a detonation takes place affect the fraction of explosive energy that is coupled into the medium. Predictions of ground shock are based on experimentally derived constants, which are normally classified according to the type of medium. Ground shock is an important safety consideration for large detonations. In general, more energy is coupled into rock such as granite than into soils.

Lithology and Geologic Structure

The lithologic description of a rock refers to the geologic name given to the rock type on the basis of its mineral composition, texture, and in some cases, its origin. Such names as granite, basalt, sandstone, etc., are generally understood by engineer and geologist alike. Lithology provides the engineer with basic identification of the material in which he must work

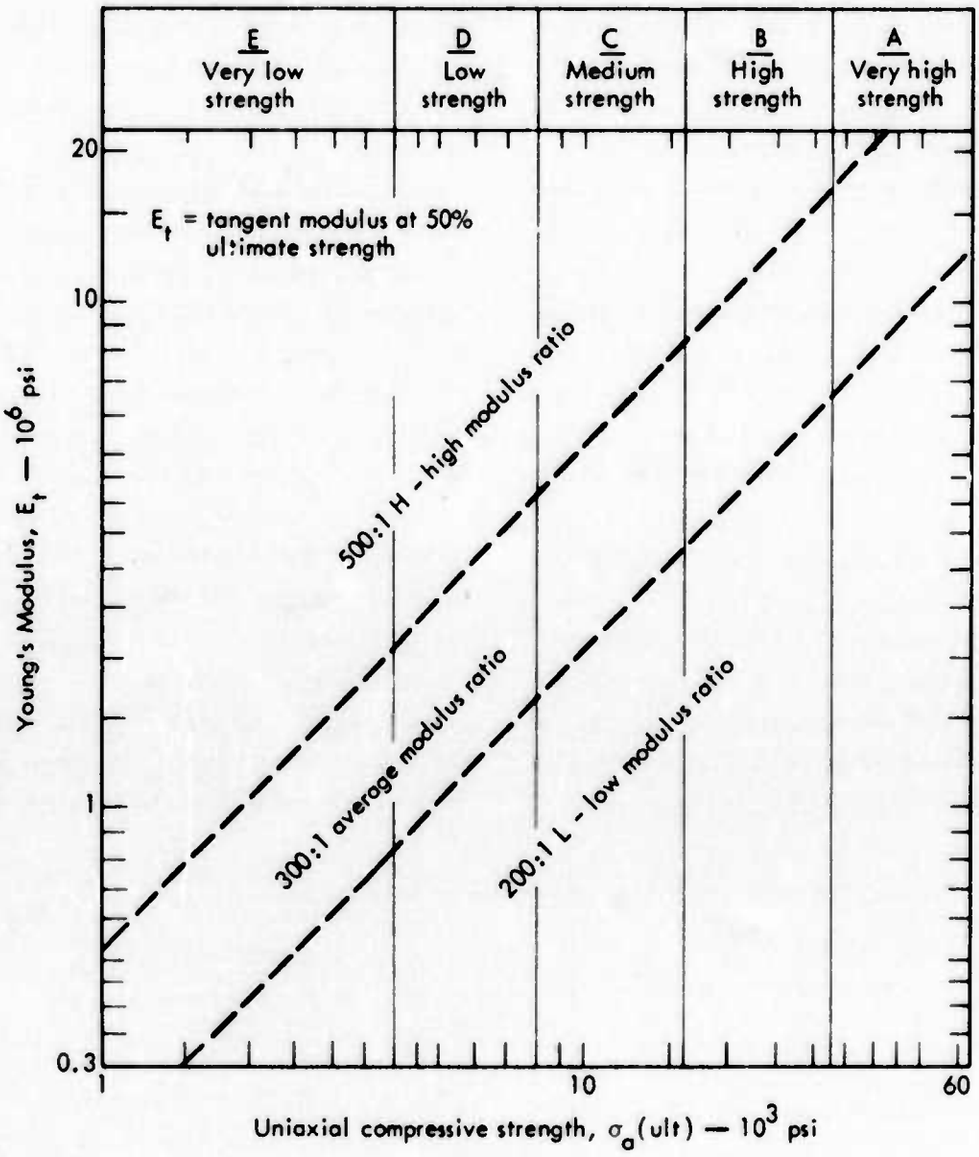


Fig. 27. Miller-Deere classification chart.[35]

and an initial link with past experience and behavior.

Geologic structure is another important feature. The common structural features of bedding and jointing are mentioned below. Additionally, such attributes as bedding attitudes and general structural complexity—whether the rocks at a given site are flat-lying or steeply dipping, and whether they are relatively undeformed or show evidence of shearing, faulting, or structural discontinuities of any kind, for instance—are important characterizations of any construction site and should be described.

Degree of Saturation and Water Content

Water content is included in this media classification system because of its observed effects on medium response to

nuclear detonations at the Nevada Test Site. Explosions in high-water-content rock caused larger cavities to be formed, and in cratering explosions higher peak spall velocities were measured when the detonation occurred in high-water-content rock.[39]

The effects of water on the dimensions of craters formed by large quantities of chemical explosives are not well known. Laboratory tests have shown that most rock types become significantly weaker when the specimens are saturated. For example, Figs. 28 and 29 show that the fracture behavior of Westerly and Cedar City granites is very much dependent on the saturation level.[40] It can be seen that up to 50% saturation has no significant effect on the strength of either rock. The fully saturated condition, however, reduced the fracture strength to essentially the value of the unconfined compressive strength, independent of the confining pressure.

It appears that the presence of the water is important for two reasons: (1) a near-saturated condition noticeably weakens the media being cratered, and (2) the presence of significant water enhances the explosive effects to some degree. Existing data are not sufficient to determine precisely the effects of water.

Water content is the ratio of the weight of water to the weight of solids, and therefore is only indirectly related to porosity and the continuity of the voids. On the other hand, the degree of saturation is the ratio of the volume of the water to the volume of the voids. A highly porous rock may have an apparently high-water-

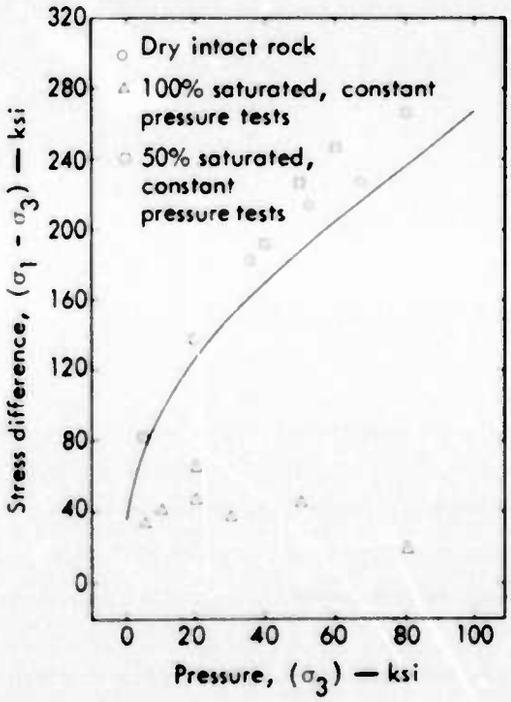


Fig. 28. Effect of full and partial water saturation on strength of Westerly granite.[40]

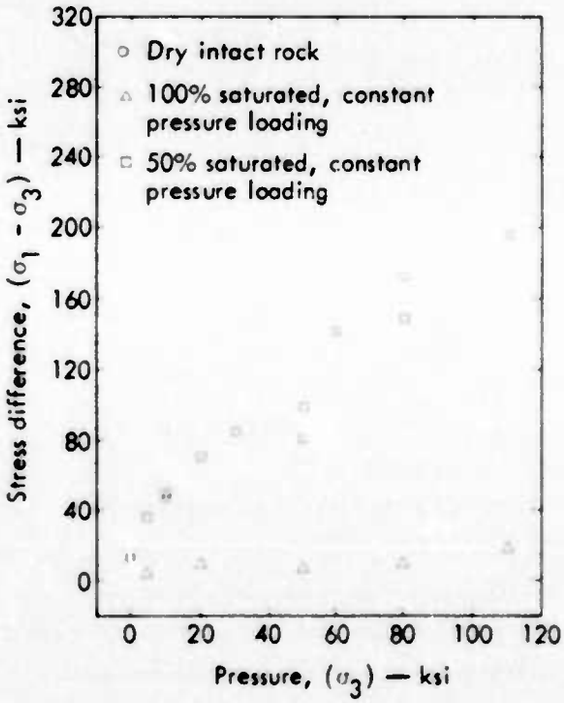


Fig. 29. Effect of full and partial water saturation on strength of Cedar City granite (tonalite).[40]

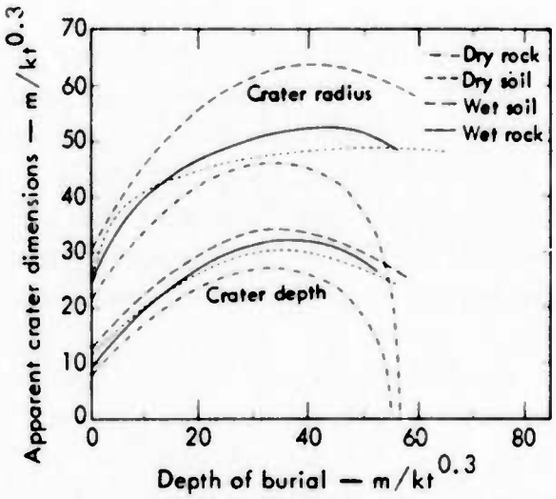


Fig. 30. Cratering curves for nuclear detonations (NCG, AEC, OCE, DASA approved curves as of March 1970).

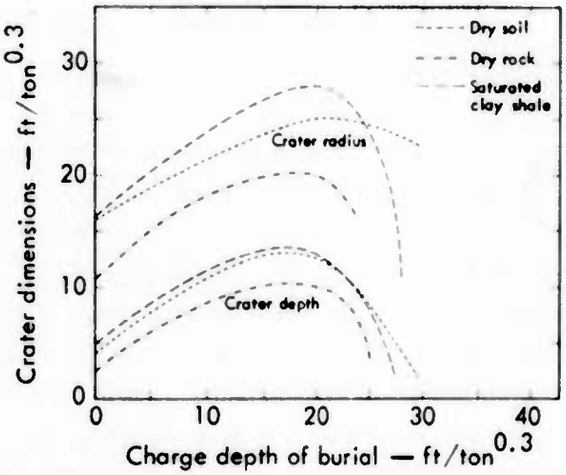


Fig. 31. Cratering curves for chemical explosives.[1]

content and still be far from saturated, whereas a dense rock may have a very low water content and be 100% saturated.

Either degree of saturation or water content is acceptable in the media classification system but each requires some explanation. A thorough discussion of all of the ramifications of water content as it affects the mechanical properties of the medium during the explosion is beyond the scope of this chapter. The effect of water content on nuclear explosions is discussed further in Refs. 39 and 41.

Only four sets of general cratering curves for nuclear detonations and three sets of chemical detonations are presently available (see Chapter 6). Nuclear cratering curves have been superimposed in Fig. 30, and chemical explosive curves in Fig. 31. These curves allow one to compare and ascertain the probable effects of the water on crater size. These curves have been obtained empirically, and the scatter of the data is such that a detailed analysis of the water content is presently not possible. Fortunately, cratering curves for individual projects are available in various publications and can be used as evidence of the expected effects of explosives in a similar medium.

An analysis was made of the extensive data obtained on water content and degree of saturation during the Interoceanic Canal Studies and other programs. Most of the rocks and soils tested that were less than 50% saturated showed a water content equal to or less than 3% for rock and 10% for soil.

Based on these and other data, three categories of saturation were selected. A dry medium is one in which the material is less than 50% saturated, or the water content is less than or equal to 3% for rock or 10% for soil; a wet medium is between 50 and 90% saturated, or the water content is greater than 3% for rock and 10% for soil; and the term "saturated medium" may be used for material that is greater than 90% saturated. The same cratering curves should be used for a saturated and wet medium pending future data refinements.

While degree of saturation is believed to be most pertinent to cratering, water content is much easier to obtain. Figure 32 provides a nomograph to aid in conversion from water content, w, to degree of saturation, s. Its use requires a knowledge of the specific gravity of the solids (grain density) G, and the void ratio, e, or the porosity n. To determine degree of saturation, locate G and e or n on the respective scales and connect with a straight line. Next, draw a straight line between w and the point at which the previous line crosses the pivot. This line defines s. The determination of density and void ratio or porosity is an additional laboratory requirement. Examples of the use of the nomograph are illustrated in the figure, one for a rock and one for a soil.

Joint Spacing

Physical discontinuities are present in rock masses in the form of planes or surfaces of separation. Geologically, these discontinuities are recognized as joints, faults, bedding planes, or rock cleavage planes. The permeability, shear strength, and deformability of a rock mass are all influenced by the number and kind of discontinuities existing in the mass. Cut slopes may be adversely affected unless the discontinuities are evaluated and their influence taken into account during design. A critical examination of rock cores can yield valuable data concerning the occurrence and the nature of the mechanical defects in the rock mass.

Table 11. Descriptive terminology for joint spacing (from Ref. 43).

Term	Spacing
Very close	Less than 2 in.
Close	3 in. - 1 ft
Moderately close	1 - 3 ft
Wide	3 - 10 ft
Very wide	Greater than 10 ft

Joints form the most common type of discontinuity and are found in all rock types. Sedimentary rocks contain, in addition, bedding planes that separate the rocks into layers.

Of more importance than the naming of the defects is the description of their spacing. In the case of joints, more than one set may be present; if the sets have different orientations it is often possible to distinguish between the sets, and the spacing of the joints in each set should be recorded. The closeness of the jointing may be expressed by the descriptive terminology in Table 11.

Thickness of Bedding

The bedding of sedimentary rocks may be described by the spacing between those bedding planes that are visible planes of weakness (i.e., where physical parting takes place). Bedding features in the form of color or textural banding may also be present. This banding should be described and recorded but should be distinguished from the prominent bedding planes. The terminology shown in Table 12 has been selected for the prominent bedding.

Other Descriptive Terms (Secondary Classification)

Thus far the description of geologic media has generally been oriented towards

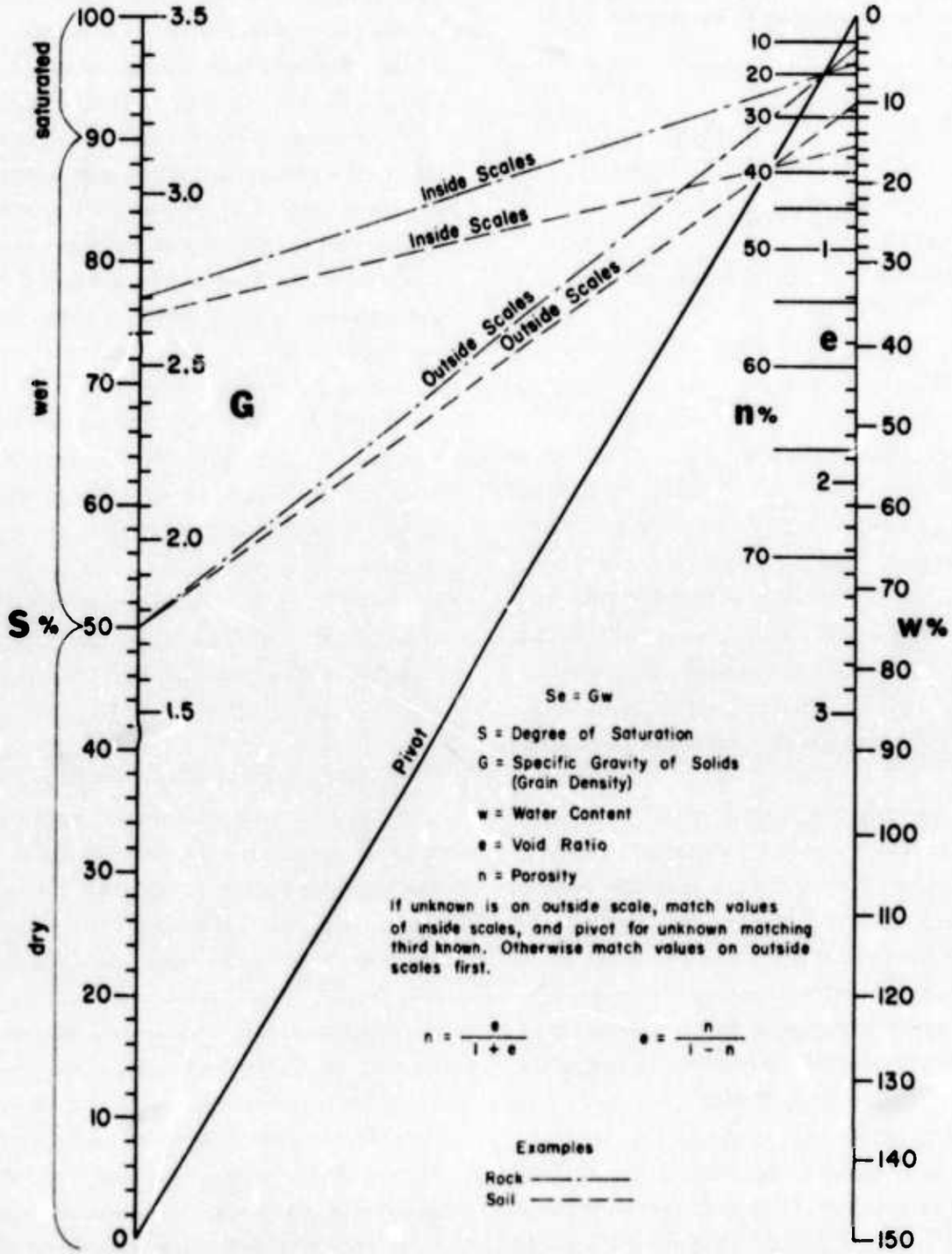


Fig. 32. Nomograph for conversion from water content to degree of saturation.[42]

a standardization of terminology. Most media can be described by a series of adjectives. For explosive cratering, however, certain average values of measurable physical properties are extremely helpful not only for evaluating probable explosive effects but also for analyzing explosive emplacement costs

Table 12. Descriptive terminology for bedding thickness (from Ref. 43).

Term	Thickness
Very thin	Less than 2 in.
Thin	2 in. - 1 ft
Medium	1 - 3 ft
Thick	3 - 10 ft
Very thick	Greater than 10 ft

and explosive selection. The following additional properties are considered the most valuable: (1) seismic velocity (compressional wave, V_p), (2) unconfined compressive strength, (3) mass density, (4) modulus of elasticity (tangent at 50% yield), (5) abrasion, (6) rock hardness, and (7) rock quality designator (RQD).

These properties are valuable in that they have been correlated with other aspects of engineering geology and can be used directly for purposes of analysis.

Seismic Velocity

Seismic velocity was selected for several reasons. This property can be measured either in the laboratory or in the field. The field seismic velocity is often used for estimating the in situ modulus of the rock mass. The primary advantage of the field seismic technique is that the measurement is made at a pertinent location in the field, and the seismic pulse is affected by the number and character of discontinuities present. A closely jointed or weathered rock will exhibit a lower velocity than will a sound rock mass. For example, if the rock substance is high-strength but is located in a closely jointed formation with possibly some weathering in the joints, the in situ seismic velocity will tend to be low. This observation introduces the ambiguity of whether the formation is composed of a high-strength rock substance in a closely jointed mass or a weaker material. Furthermore, if the fissures of a closely jointed formation are filled with water, the velocity will be only slightly affected by the joints. For the foregoing reasons, seismic velocity should be evaluated qualitatively in conjunction with the other parameters in this classification system.

Seismic surveys are routinely used for determining the depth of soil over a rock mass. It is possible through this technique to profile several layers under proper conditions. Seismographs are inexpensive, portable, and can be used at most job sites. Seismic velocities can be used with lithology to evaluate the rippability of a material with the aid of Fig. 33.

Unconfined Compressive Strength

Unconfined compressive strength is another parameter that is valuable in determining rock mass quality. Determination of this variable on intact core samples is accomplished readily in the laboratory. Unconfined compressive strength is widely used in soil and rock mechanics and is well understood. Like many variables when used alone, it does not give the complete story on the in situ rock mass. It does, however, provide an excellent indication of shear strength, a quantity that measures the degree of difficulty or resistance to cratering of the medium.

Mass Density

Mass density is a parameter that helps complete the analysis of the medium to

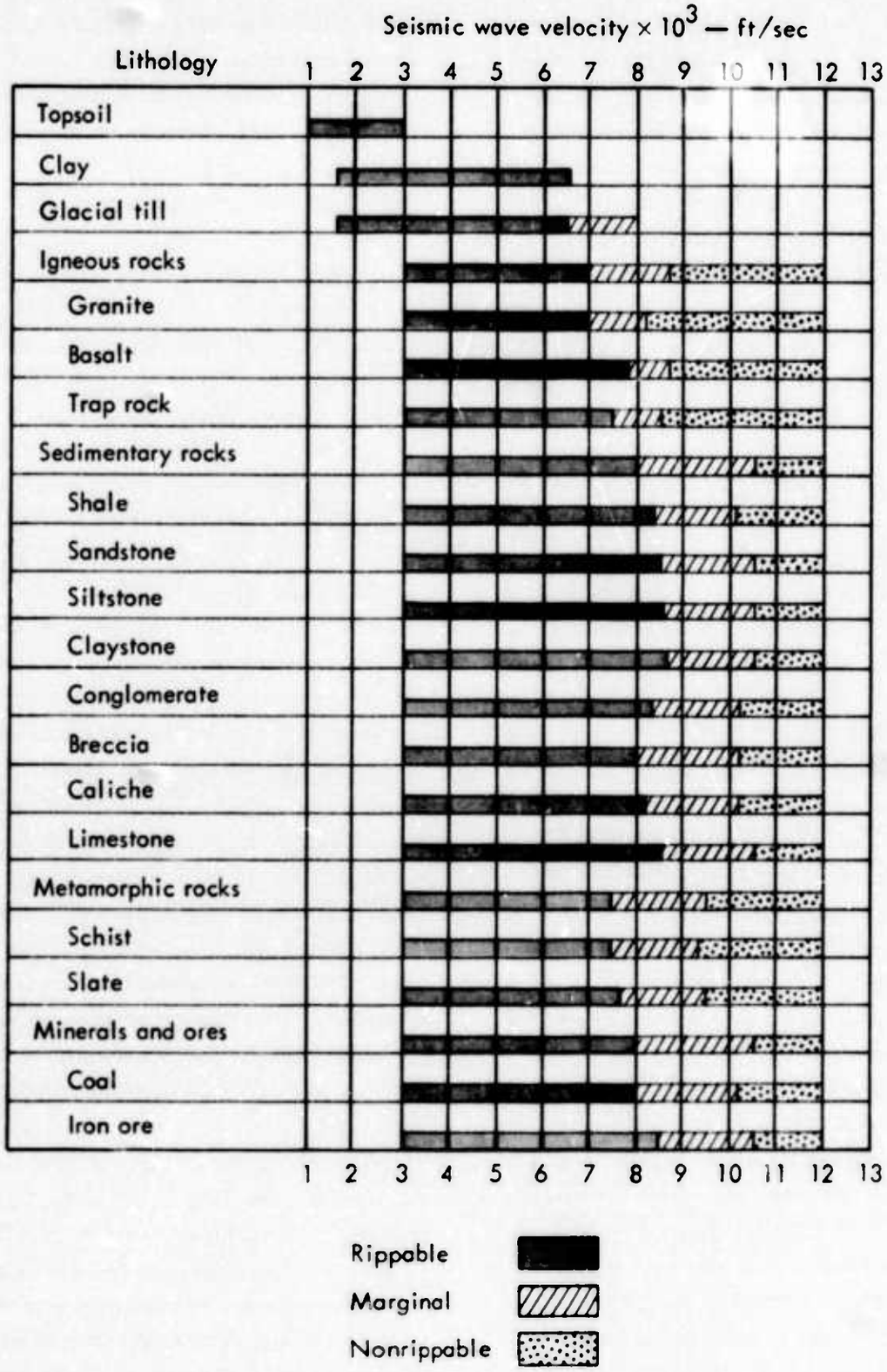


Fig. 33. Rippability and seismic wave velocity for D9G-No. 9 Series B ripper.[44]

be cratered. The specific gravity of the solids or the grain density from core samples is useful in determining the degree of saturation but is not representative of the mass density. Mass density values provide a real measure of the field density of the medium and aid in the evaluation of the seismic data.

Modulus of Elasticity

Modulus of elasticity (Young's Modulus) was selected as a relevant index property for rock because of its indication of relative competency. When determined statically in the laboratory, it is known as the elastic modulus, but when it is determined in the field by plate jacking tests or pressure chamber tests, it is referred to as the deformation modulus. Young's modulus for rock is a function of the applied stress and may depart from the simple proportionality of stress and strain that applies to many engineering materials. The stress-strain relationship of intact rock specimens is often linear at intermediate stress levels and curvilinear at low and high stress levels. Deere and Miller suggest that the slope of the stress-strain curve at 50% of the unconfined strength, E_{t50}, be considered the characteristic modulus of intact rock; they feel that the curvilinear relationship at low stress levels is caused by the closure of stress relief microfractures created during and after coring.[35]

For explosive cratering, the modulus of elasticity helps to evaluate the difficulty of excavation, the nature of the crater, and the probable postshot behavior of the crater and surrounding area. From the stress-strain curve determined during unconfined compressive strength tests, the value of the classic modulus tangent at 50% yield can be ascertained and plotted along the ordinate of the Miller-Deere Chart (Fig. 27). This chart is an excellent tool and is frequently used for comparing various geologic media.

Abrasion Data

Abrasion data are included in this classification system because of the relationships between abrasion, drillability, and explosive emplacement costs. Emplacement costs are a major component of the direct cost in explosive excavation. The type of medium, hole depth, and emplacement hole diameter will influence the time and type of drilling equipment utilized; hence, the cost. Bit abrasion and rock hardness are generally the parameters used by drillers for cost estimation. The term "hardness" is comparable to the various strength values used in this system but is more common to the driller's language. Hardness can be expressed in several ways; e.g., unconfined compressive strength, Moh's hardness scale, or a drilling index that compares various materials to some standard. The degree of abrasiveness is often estimated using the silica (SiO_2) content, which is normally a function of lithology.

Rock hardness is the ability of the rock to resist scratching. A means of quantifying rock hardness is the Moh's scale, Table 13. This scale lists minerals in descending order of hardness with the number assigned to each mineral an indication of its hardness. Each mineral can scratch the mineral listed below it but not the one above it. The column on the right indicates common materials of

Table 13. Moh's scale of hardness.

Mineral	Hardness	Common material
Diamond	10	
Corundum	9	
Topaz or beryl	8	
Quartz	7	Porcelain
Feldspar	6	Glass
Apatite	5	Knife blade
Fluorite	4	
Calcite	3	Copper penny
Gypsum	2	Fingernail
Talc	1	

equivalent hardness. The scale was developed to classify minerals but in some cases it helps to describe rock also.

Abrasiveness is a more critical factor in rotary drilling than in percussion drilling (discussed in Chapter 5) because the rotary bit is in constant contact with the rock. The abrasiveness of a given rock depends upon hardness, shape, and the bond holding the rock particles together. Hard angular particles bonded firmly together can gouge and scrape drill bits more severely than rounded, loosely-bound particles that tend to roll over the bit.

Practically the best source of information concerning abrasiveness (hence, drillability) is the investigative boring logs. The boring log will normally include the type of drilling apparatus and the bits used or the size of the stones in the diamond bit and the rate of drill advance. Bit information can be expressed in total number of feet per bit. Bit abrasion information together with lithology and the unconfined compressive strength are generally sufficient for emplacement cost analysis. Emplacement cost analysis is discussed in detail in Ref. 45.

Rock Quality Designator

Rock quality designator (RQD), developed by Deere,[20] was selected because of its value in measuring in situ properties of rock masses. The RQD is a modified core recovery percentage in which all the pieces of core over 4 in. long are counted as recovery. The smaller pieces are considered to represent close shearing, jointing, faulting, or weathering in the rock mass and are not counted. The RQD is a more general measure of the core quality than the fracture frequency. Core loss, weathered and soft zones, and fractures are accounted for in the determination. The RQD provides a preliminary estimate of the variation of the in situ rock mass properties from the properties of the "sound" portion of the rock core. Thus, a general estimate of the engineering behavior of the rock mass can be made. An RQD approaching 100% denotes an excellent quality rock mass with properties similar to that of a poor quality rock mass having a small fraction of the strength and stiffness measured for an intact specimen.

RQD together with lithology and joint spacing allow estimates of postshot particle gradation in the fallback and ejecta. Such information allows estimates of the engineering behavior (i.e., stability, seepage, and settlement) of the rubble.

SUMMARY

Cratering is influenced by a variety of material and explosive variables. The influence of each of these variables has

Table 14. Examples of media classification for explosive excavation.

	Sedan	Buggy	Pre-Gondola
A. Primary Classification			
I			
Media	Soil	High-strength rock	Clay shale
II			
Lithology/USCS geology structure	SM (Alluvium)	Horizontal basalt flow	CH
III			
Degree of saturation	Dry	Dry	Saturated
IV			
Joint spacing	N/A	Close jointing	N/A
V			
Thickness of beds	N/A	Thin bedding	Thin bedding
B. Secondary Classification			
VI			
V_p, fps	2,000	10,000	5,000
VII			
Unconfined compressive strength value, psi	1,000	16,000	900
VIII			
Mass density, lb/ft^2	113	162	140
Specific gravity	1.81	2.60	2.42
IX			
E, $\times 10^6$ psi	0.3	5	1.5
X			
Abrasion	Low	High	Low
XI			
Rock Quality Designator (RQD)	—	Unknown	Unknown

not yet been refined to the point of separating out individual effects. The cratering curves availa .e for various media from empirical data are limited but are considered satisfactory for general use once the nature of the material is ascertained.

Media for explosive excavation have been placed into five separate categories: (1) high-strength rock, (2) intermediate-strength rock, (3) weak rock, (4) clay shale, and (5) soil. Each category can be evaluated in the wet and dry condition. The system includes a description of the

medium as it appears in the field and index properties derived from soil and rock mechanics. Attempts were made to use familiar terminology and to keep the system compatible with the developing fields of rock mechanics and explosive excavation.

A summary of the organization of the system was shown in Table 10. Examples of three materials in which explosive excavation has been carried out are shown in Table 14. Elements such as lithology, bedding, and RQD provide in situ qualities; modulus of elasticity and unconfined compressive strength are determined in the laboratory and are good descriptors of the intact core material.

Examples of the use of this classification system as applied to previous nuclear and chemical cratering experiments are given in Ref. 42. Reference 46 catalogs the media properties for past experiments and draws some conclusions about the effects of the media on crater dimensions.

Chapter 5 Explosive Cavity Construction Techniques*

SCOPE

This chapter contains information on current techniques for rapid drilling of emplacement cavities used in chemical explosive excavation projects. Hole dimensions are limited to a depth of 100 ft and a diameter of 6 ft for practical purposes. The basic techniques covered in this chapter are percussion drilling, auger drilling, core drilling, rotary drilling, and novel drilling.

Large, deep cavities drilled by commercial rigs are employed extensively in quarry operations that use ammonium-nitrate-based explosives. The use of explosives as an earthmoving technique has many potential military applications; however, the Army's present capability of drilling large emplacement cavities for explosive excavation is, for all practical purposes, nonexistent. In a tactical situation the time required to drill the emplacement hole should be as short as possible; therefore, this chapter discusses the state-of-the-art in large-hole drilling techniques and explores some of the concepts being developed to improve penetration rates.

CONVENTIONAL DRILLING TECHNIQUES

Figure 34 outlines the drilling techniques studied in this chapter. Several novel techniques being developed by private industry show promise for improving drilling rates. However, the four conventional methods of drilling are percussion, auger, core or calyx, and rotary.

Percussion Drills

Percussion drills crush rock by repetitive impact from a piston or hammer that drives a chisel or wedge-shaped bit against the rock. A static axial load is applied to the drill steel in order to keep the bit in contact with the rock.

Rotary Percussion Drill

Rotary percussion drilling is a hybrid form of drilling that combines the impact associated with percussion drilling and the rotary motion of rotary drilling. The added impact force increases the drilling rate of the rotary bit.

Churn Drill

Churn or cable tool percussion drilling is an old technique in which the rock is broken through impact by dropping a heavy bit supported by a cable (Fig. 35). The bit is a heavy metal cylinder with the lower end forged and tempered to act as a cutting surface. The cuttings suspended in mud are periodically removed from the hole with a bailing bucket. Within the next 20 years, it is expected that the churn drill will disappear from all large-scale drilling operations; however, its use will be continued in areas in which the workload will not support modern, high-penetration-rate machines. The churn drill may be of military interest, because

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*This chapter is based on Ref. 45.

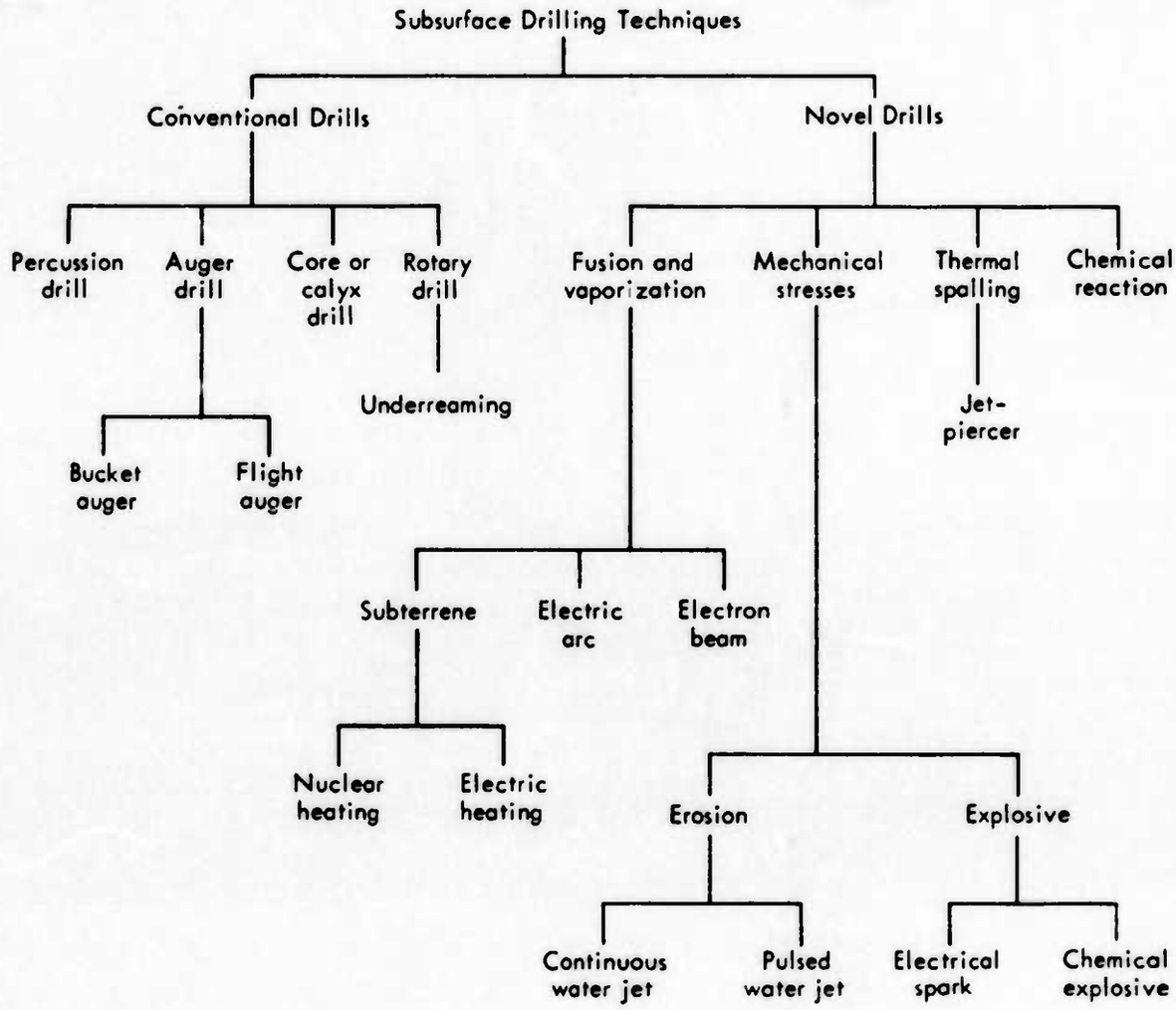


Fig. 34. Drilling techniques available for excavating subsurface cavities.

in underdeveloped countries in which forces may be deployed, some form of churn drilling will be in use even though it is a very low performance tool. From the military standpoint, it may also be advantageous because it is one of the lightest tools available for drilling large-diameter, deep holes.

Auger Drilling

Auger drilling is a rotary drilling technique accomplished by the use of two types of augers: the flight auger and the bucket auger shown in Fig. 36. Rotary power is provided to the auger by a square telescoping steel shaft, known as the Kelly. The maximum depth obtainable is controlled by the Kelly and is usually 100 to 130 ft. The auger is presently the most efficient and economical method of drilling in soil and weak rock. Care must be taken, however, to insure that the material being drilled is not so weak as to cave in around the auger, thereby preventing its removal from the hole. Average penetration rates for the auger are shown in Table 15.

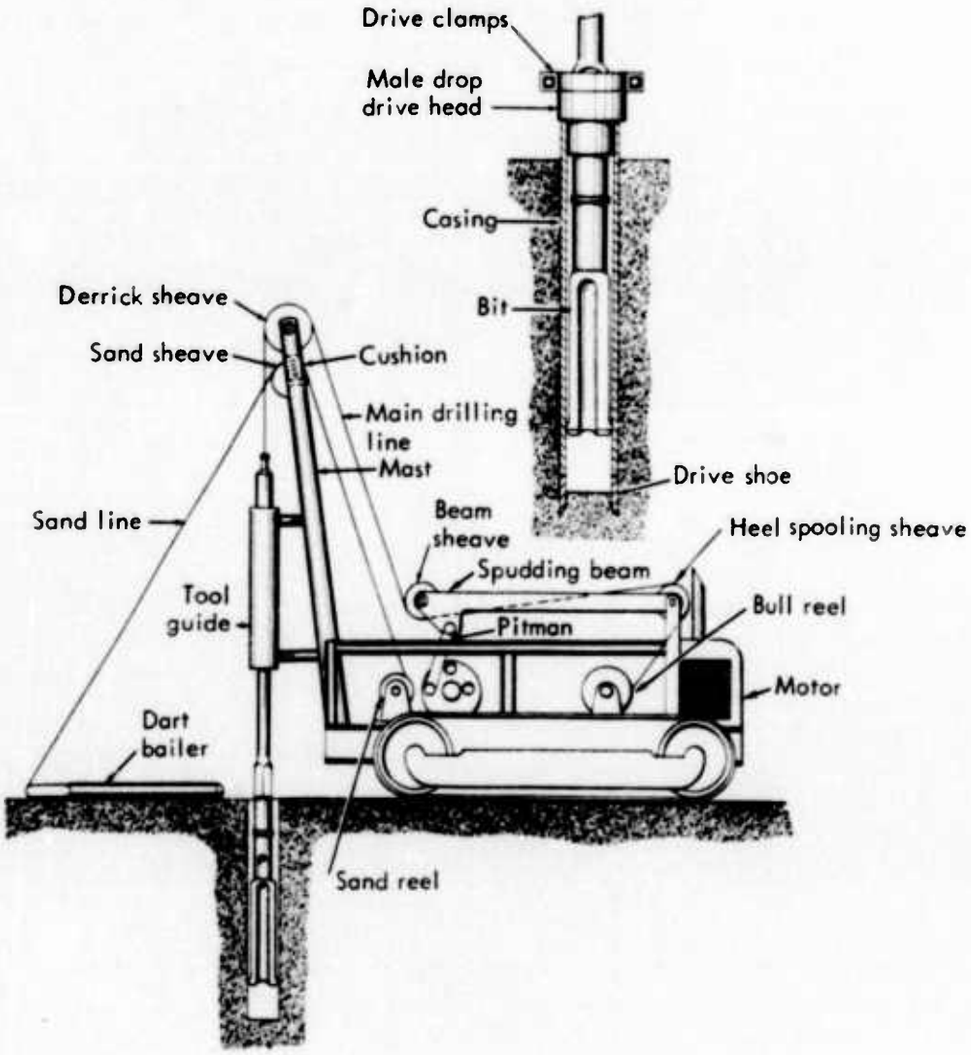


Fig. 35. Churn drill[36] (inset shows casing device).

Core or Calyx Drilling

Large-diameter core drilling (with steel shot) also has been used extensively in the United States especially prior to 1950. The shot drilling method uses chilled steel shot as the core cutting material. The core barrel is a steel cylinder with short slots cut at intervals in its lower edge (Fig. 37). The shot is fed down the drill rod to the inside of the core barrel cutting edge where it is crushed into angular shot fragments that

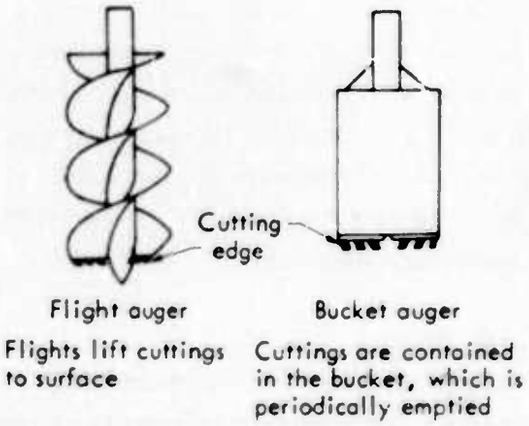


Fig. 36. Auger bits.

cut the rock. In some cases, the chilled shot is introduced with circulating fluid through the drill pipe.

The torque, hoisting, and circulation requirements for large-diameter core drilling are relatively low. As a consequence, mobile drill rigs can be used. Truck-mounted exploratory drills have successfully drilled 200-ft, 36-in. diameter core holes in limestone (Table 16).

Due to the periodic stops at intervals to remove the drill cores, net large-diameter coring penetration rates are low. Net penetration rates for 36-in. diameter shallow holes drilled in intermediate-strength sedimentary and metamorphic rocks on TVA projects ranged from 0.4 to 0.7 ft/hr. Net penetration rates for 36-in holes in intermediate-strength limestone averaged 1.5 ft/hr with chilled steel shot as the cutting material.[36] Research by the laboratory of the North Pacific Division of the Corps of Engineers in Portland, Oregon, has shown that penetration rates in intermediate-strength basalt with steel shot in the core barrel were as low as 1.8 ft during an 8-hr shift. By comparison, rolling cutter bits resulted in a penetration rate of 2.7 ft per 8-hr shift.

Fig. 37. Calyx drill bit with core sample.

Table 15. Average net penetration rates for auger drilling.[47]

Diameter (in.)	Average net penetration rate (ft/hr) Common excavation	Weak rock
36	48	19
48	25	11
60	17	7
72	10	4

The large-diameter core drilling method requires that the subsurface materials be relatively impermeable and of sufficient competence to allow the boring to stand without support. The technique would be most adaptable in the boring of moderately sized holes (36 to 48 in.) in rock that is impervious or in areas in which the water table is below the total depth of the boring. The application of the core drilling technique would probably be at the point of refusal for the auger drill (intermediate-strength rock). The application of core drilling in explosive cavity construction is limited because of the slow rate of penetration.

Rotary Drilling

This method is the most effective method of boring large-diameter holes to depths exceeding the practical limits

Table 16. Calyx drilling equipment and capability.[45]

Rig type	Diameter (in.)	Depth range (ft)	Remarks
Truck-mounted (200-300 hp)	36	200	Most suitable for conditions where torque, hoisting, and circulation requirements are low; normally uses chilled shot for cutting.
Crane-suspended auger attachment	72	500	Greater hoisting and torque capabilities permit larger diameters to be drilled; normally uses carbide inserts for cutting.
Drive-in (500-750 hp)	72	1,200	Depth limited only by core removal capabilities; use either inserts or rolling cutters and core catcher.

of the auger and calyx methods. This technique uses a rotary machine or a rotary table to transmit motion to the drilling shaft, which is equipped at its lower extremity with a cutting tool or bit. As the bit is rotated in the hole, the rock is broken into chips and continuously removed by circulation of a fluid or a gas. Large-diameter rotary drilling equipment and techniques have been adapted from oil well drilling practices and equipment. Drilling penetration rates generally vary from 0.5 to 8 ft/h, depending on hole depth, diameter, and formation strength and hardness.

The large-diameter drill rig is in most cases only slightly modified from the equipment used to drill oil or gas wells. The rotary drilling equipment consists basically of the drill rig (surface component), the drill string (in-hole components), and the circulating system.

Underreaming

Beyond the strength and depth limitations associated with augering, two main rotary drilling methods have been employed to drill large-diameter holes. The multiple-pass method utilizes a pilot hole that is of small diameter and is drilled to the desired depth. The pilot hole is reamed to successively large diameters until the desired diameter is obtained. Another version of this method accomplishes the drilling and reaming in a single pass with a stage type bit. The single stage or full-bore method uses one flat type bit of a larger diameter to obtain the desired opening in a single pass.

A relatively new technique has been developed for obtaining a large-diameter hole at depth without drilling a large-diameter hole from the surface. This technique has been termed "underreaming." The desired results are obtained by a specially designed expandable drilling tool called an underreamer. The underreamer consists of a long body with attached cutting arms that expand or bell outward to ream an area of larger diameter. Tooth or insert type rolling cutters are attached to the ends of the expandable arms. Circulation pressures control the movement of the cutters outward, increasing the annular volume between the Kelly and the upper body. Once open, the

cutters lock out to full-gage cutting position to ream the depth of hole desired. When collapsed, the cutting arms remain flush with the body and locked into place. Figure 38 illustrates the open and closed locked positions of an underreamer.

Three basic types of underreamers are available and are distinguished by the cutter attached to the end of the body:

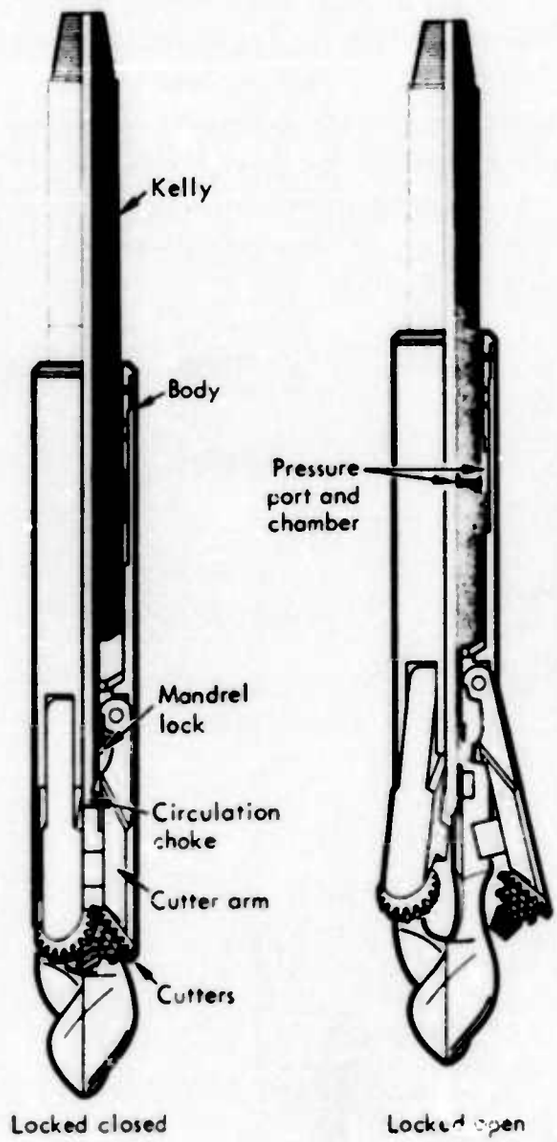


Fig. 38. Expanding hole opener (used by permission of Bakerdrill, Inc., Houston – identified as Baker Lockomatic, patented U.S. and Foreign).

(1) a type with no cutting attachment on the nose of the tool, utilized in a pilot hole of a diameter suitable to accept the diameter of a collapsed underreamer, (2) a type with a flight auger on the end, (3) a type that accepts a high-strength rock, rolling-tooth cutter at the end of the underreamer body.

The sizes and expansion selections of underreamers currently available are quite varied. Collapsed diameters range from 3-3/4- to 21-1/2 in., and the tools expand up to double the collapsed size. A few underreamers of very large diameters (56 to 90 in.) have been designed specifically to meet the needs of large drilling projects.

For the purpose of constructing emplacement cavities for explosive excavation, underreaming seems to be a very good procedure. It accomplishes the task of obtaining a large-diameter cavity at a specified depth, suitable for emplacing explosives. Figure 39 is a schematic representation of an underreamed cavity

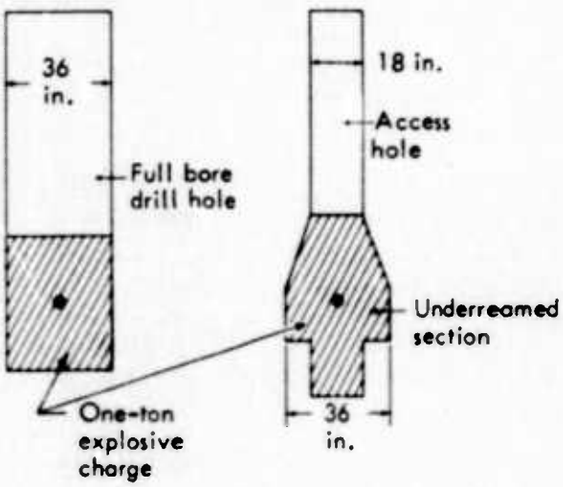


Fig. 39. Comparison of full-bore hole with underreamed hole, each with same size charge hole.[45]

compared to a full-bore hole, both obtaining the same desired cavity for the emplacement of required bulk explosives. Underreaming a pilot hole significantly reduces the volume of material that must be removed.

NOVEL DRILLING TECHNIQUES

The drilling rates of the conventional drilling rigs previously discussed are limited by the amount of energy that can be delivered to the cutting surface of the rock.

Novel drilling techniques are being devised that deliver more energy to the cutting surface. The four basic mechanisms of rock destruction employed are (1) fusion and vaporization, (2) chemical reaction, (3) thermal spalling, and (4) mechanically induced stresses.

Fusion and Vaporization Drills

These types of drills use electric arcs, electric or nuclear heating elements, electron beams, lasers, or plasma to melt any rock in the path (Fig. 40). Due to the high-energy requirement, test results have been limited to data collected from small laboratory models. One model, called the subterrene, developed by the Los Alamos Scientific Laboratory, has shown promise by melting a 2-in. hole through 50 ft of rock with an electric heating element.

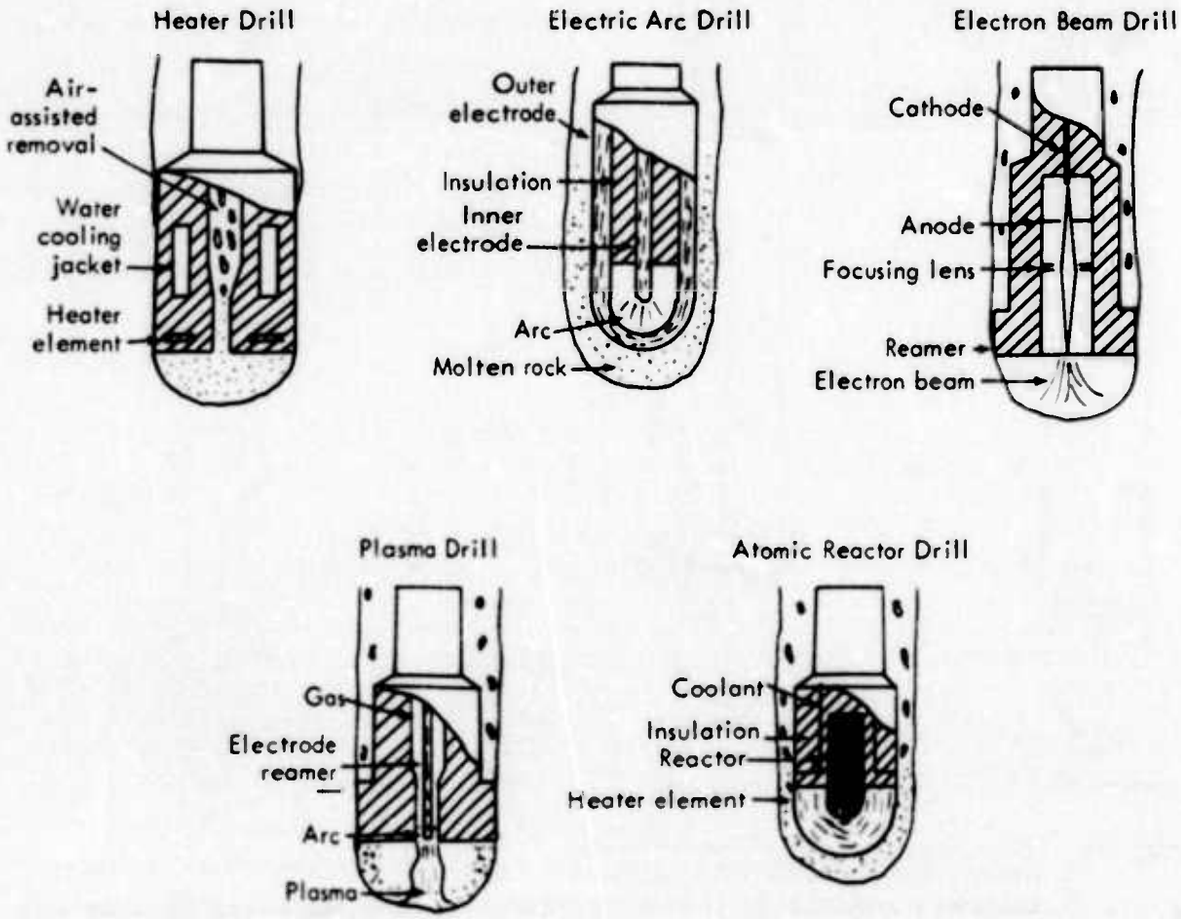
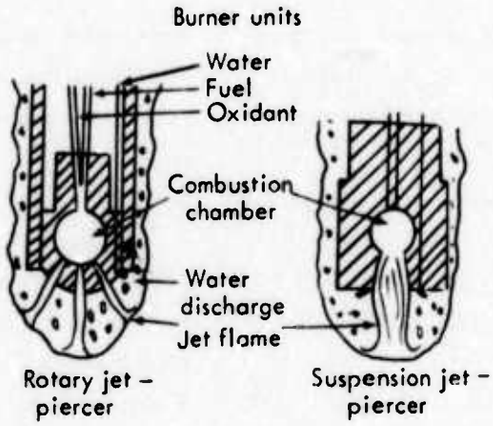
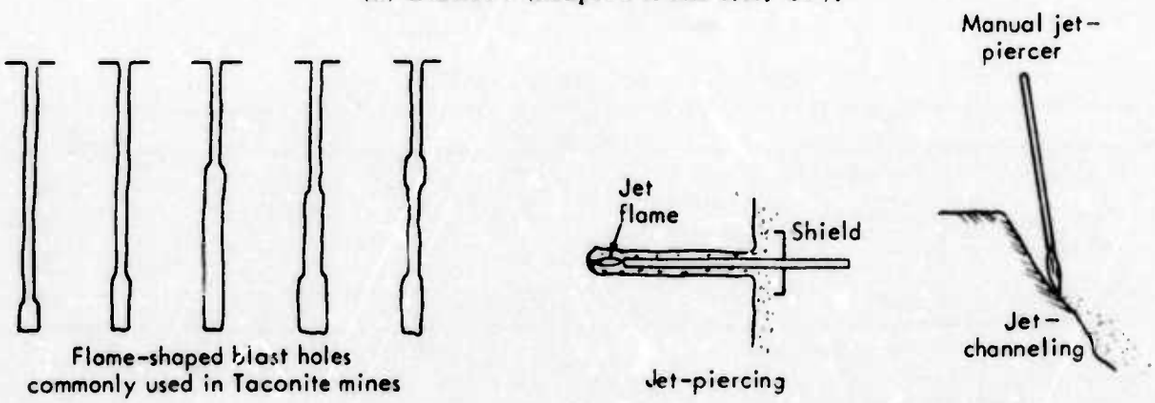


Fig. 40. Five types of fusion and vaporization drills (adapted from Ref. 48).



(a) Burners (adapted from Ref. 48).



(b) Applications.[36]

Fig. 41. Jet-piercer burners and applications.

Chemical Reaction Drills

These drills use highly reactive chemicals, such as flourine, to dissolve the rock. High-penetration rates have been obtained in laboratory and field tests; however, difficulty in handling and the high cost of chemicals has discouraged the development of this drilling technique.

Thermal Spalling

Jet-piercing is a thermal process that is dependent upon the spalling characteristics of the rock being drilled. Thermal spalling is the cracking of rock due to differential expansion induced by thermal stresses.

Thermal spalling is a very successful process used in the mining of highly spallable rock, such as taconite and quartzite. Most rock beds, however, contain some random inclusions of poorly spallable rock requiring mechanical assistance.

The jet-piercer is basically a combustion chamber in which fuel oil is burned with either oxygen or air. The resulting gases produced by combustion (3,000 to 4,000°F) are directed at supersonic velocity through a nozzle to the rock.

There are three classifications of jet-piercers: rotary, suspension, and manual, as shown in Fig. 41.

Some jet-piercing drills have a rotary unit mounted in a rotary rig and operate in a manner similar to the conventional rotary drill.

The suspension jet-piercer is suspended from a cable, which eliminates the need for a complex support rig. Unlike the rotary unit it cannot penetrate poorly spallable rock.

The manual jet-piercer is a small unit handled by one man. It is used for small holes and for secondary rock reduction.

Performance characteristics of the jet-piercer are listed in Table 17.

Mechanically Induced Stress

Drills that remove rock by mechanically induced stresses use the least amount of energy to remove a given volume of rock as compared to the other three methods—except for thermal spalling in high-strength rock, such as taconite.

Table 17. Jet-piercer performance.

	Rotary	Suspension	Manual
Average hole diameter, in.	9.5	8.0	2.0
Minimum hole diameter, in.	6.5	5.0	1.25
Fuel flow, gal/hr	36.0	36.0	6.0
Oxygen flow, ft^3/hr	10,000	10,000	1,000
Rock ore formation	Piercing speed - ft/hr		
Oxidized and altered taconite	14	—	—
Cherty-magnetic taconite	12-23	10-18	20-50
Slaty taconite	10-15	5-10	—
Michigan jasper	19-40	5-15	—
Labrador specularite	23-40	—	50-70
Michigan argillaceous iron formation	14	—	—
Quartzite	30-40	—	70-80
Michigan diorite	12	—	—
Michigan dolomite	20-35	20-30	35-40
Granite	20-50	10-30	25-50
Potsdam sandstone	—	20	45-50
Sioux quartzite	—	20-30	45-50
Rhyolite	—	15	—
Arkansas nepheline syenite	—	10-25	—
Quartz monozonite	—	5-15	—
Kyanite	—	—	30-40
Dunbar sandstone	—	—	50-60
Feldspathic quartzite	—	—	50-60
Diabase traprock	—	—	15-20
Labrador iron ore	—	—	25-50
Granite—gneiss	—	—	35-40

Several techniques in this category have been devised that have higher drilling rates than conventional rotary drilling:

The Erosion Drill

Erosion drilling requires a high-velocity stream or pulse of water to disintegrate the rock. Laboratory tests with a model of this drill have shown penetration rates up to twice those obtained with conventional rotary drills, but because of the high water pressures required (2,000 to 20,000 psi) special high-speed pumps and high-pressure hydraulic equipment must be developed to make this method practical.

Spark Drill

An electrical spark discharged under water produces an explosive-like pressure pulse capable of crushing rock. Although still in the experimental stage, the spark drill is capable of transmitting twice as much power to the rock as a conventional rotary drill.

Listed in Table 18 are some novel drills, and their potential for exceeding

Table 18. Estimates of maximum drilling rates for 8-in. diameter novel drills in intermediate-strength rock.[49]

Drill	Rock removal mechanism	Maximum potential drilling rate (ft/hr)
Erosion	Mechanical	70-280
Spark	Mechanical	70-280
Explosive	Mechanical	50-140
Rotary	Mechanical	30-170
Forced flame	Spalling	55-110
Jet-piercing	Spalling	18-36
Electric disintegration	Spalling	18-28
Pellet	Mechanical	8-28
Turbine (single-stage)	Mechanical	6-28
Plasma	Spalling	16-22
Electric arc	Spalling	8-16
High-frequency electric	Spalling	6-12
Plasma	Fusion	4-6
Electric heater	Fusion	2-6
Electric arc	Fusion	2-6
Nuclear	Fusion	2-6
Laser	Spalling	2-4
Electron beam	Spalling	2-4
Microwave	Spalling	2-4
Induction	Spalling	1-2
Laser	Fusion	0.6-1.2
Electron beam	Fusion	0.6-1.2
Electron beam	Vaporization	0.2-0.4
Laser	Vaporization	0.2-0.4
Ultrasonic	Mechanical	0.08-0.14

the drilling rates of the conventional rotary drill.

The application of novel techniques to assist rotary drills is also being explored. For example, a spark drill mounted on a rotary drill bit can be used to weaken the rock ahead of the roller cutters.

This technique of combining present methods with novel techniques shows possibilities in the near future for improving drilling rates.

FACTORS AFFECTING DRILLING RATE

The maximum rate of drilling for a large-diameter hole is obtained by selecting the proper bit cutters, maintaining the proper weight on the bit, rotating the bit at the proper speed, and using the most suitable circulation system.

Drilling Fluid

The circulatory system delivers the fluid, either gas or liquid, to the bottom of the hole and returns it to the surface (see Fig. 42). This fluid lubricates the drill string, cools the bit, and removes the cuttings. The direct circulation system pumps the fluid down the drill pipe, out through the bit ports, and up through the annular space between the wall of the hole and the drill string to the surface. The reverse system pumps the fluid down the annulus between the drill string and the wall of the hole, across the bottom of the drill hole, and back up through the drill pipe. Reverse circulation will be explained further under the section on drill bits.

Circulation fluid can be classified as liquid (ordinary water, water-based drilling mud, and oil-based drilling mud), gas (air), or foam. In terms of drill penetration rates, ordinary water is the best drilling liquid fluid, and the least costly.

Air Circulation

Air is used as the circulating fluid because it is the easiest to obtain and requires less support equipment than other drilling fluids. The disadvantages of using air are that it produces dust, which is a health hazard, and that it accelerates machinery wear. With air circulation, high air velocity must be maintained to keep cuttings in suspension while they return to the surface. As hole sizes increase, greater quantities of air must be used to maintain high air velocity.

Foam Circulation

To overcome these problems a foaming agent is used that is similar to commercial

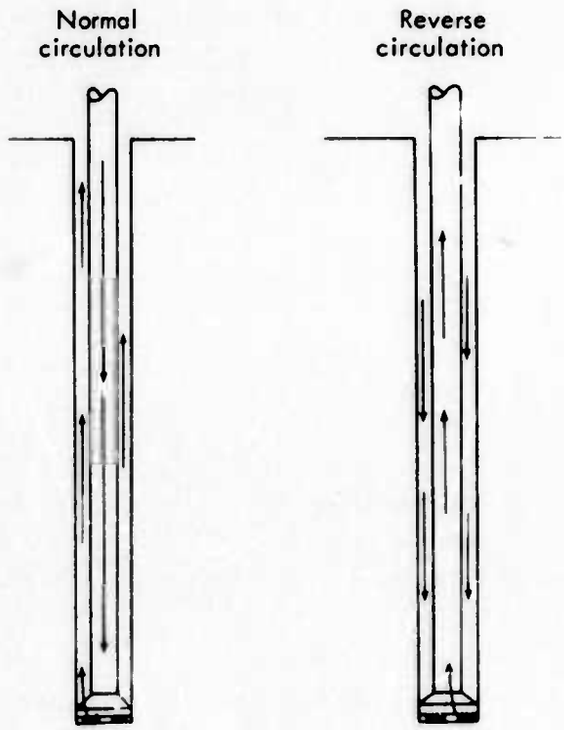


Fig. 42. Drill fluid circulation.

liquid detergents. Foams in an aqueous solution of inhibited alkylayol sulfate salts have shown significant results in reducing the dust associated with air circulation drilling.

Liquid Circulation

Liquid circulating fluid, either water or mud, is sometimes required due to hole size, cutting size, and downhole condition. As hole diameters increase, greater flow rates are required to clean the hole. Liquids, because of their greater viscosity, can suspend larger particles and require a lower flow rate. Circulatory liquids also require auxiliary support equipment not associated with air drilling. Water must be readily available, and a system for storing and recirculating the liquid is required.

The selection of the circulating fluid to be used with the circulating equipment is dependent upon: (1) the size of the drill hole, (2) the anticipated size and shape of cuttings, which are dependent on the mode of failure of the rock, (3) the permeability of the rock and the position of the water table, and (4) the volume of cuttings produced per unit of time as controlled by the drilling penetration rate.

Rotary Drill Bits

Drill bits may be classified by the shape of the cutting surface as conical, hemispherical, pyramidal, and prismatic. A refinement of this shape classification, however, is the set of names: rolling cutter bits, diamond bits, and drag bits. Applied forces transmitted to the rock through the bit are concentrated in the area of contact. The stresses at and below the contact break the rock. Experiments simulating the cutting actions of percussion and rotary drill bits indicate that rock fails in three distinct modes: crushing, chipping, and spalling (Fig. 43). Crushing and chipping are essentially static processes, whereas spalling is caused by stress waves.[50] Stresses under a chisel bit are essentially compressional.[51] Crushing is apparently due to failure of rock in a state of triaxial compression; chipping is due to fractures propagating from the vicinity of the crushed zone. Spalling is the mode of failure introduced when tensile stress waves are generated by the reflection of compression stress waves from a free surface.

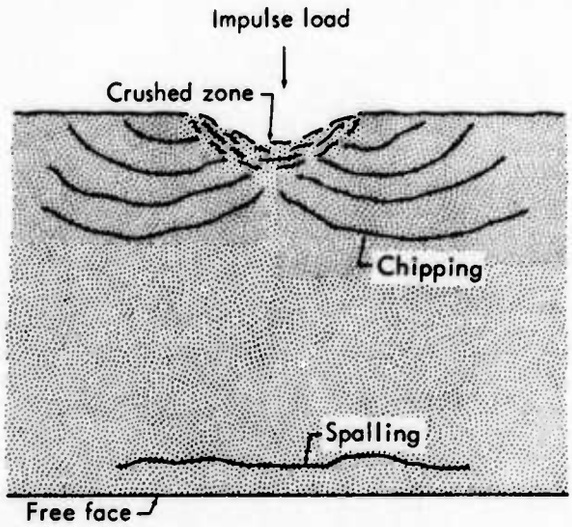
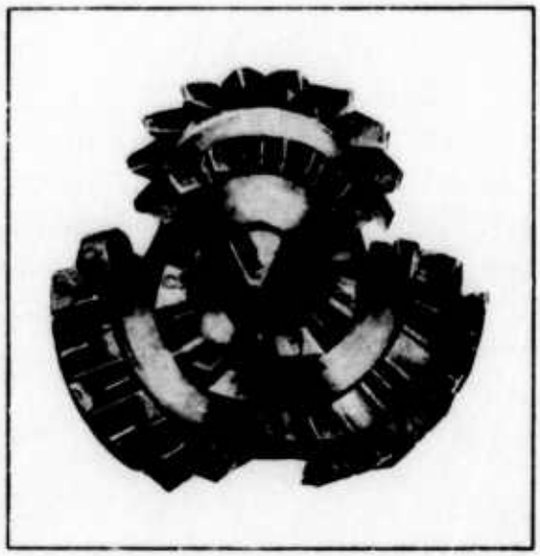


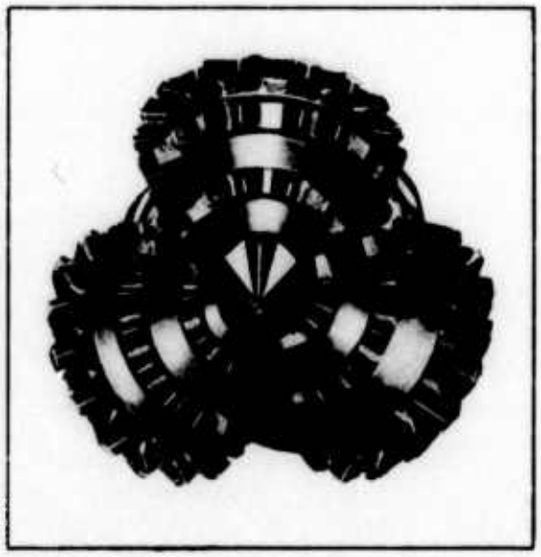
Fig. 43. Modes of failure induced by drill bit.[51]

Roller Cone Bits

Roller bits penetrate the rock by crushing and chipping. They have two general types of conical cutters, toothed cutters, and tungsten carbide insert cutters. Widely spaced teeth on tooth type cutters are oriented so that a scraping effect is obtained in weak rock.

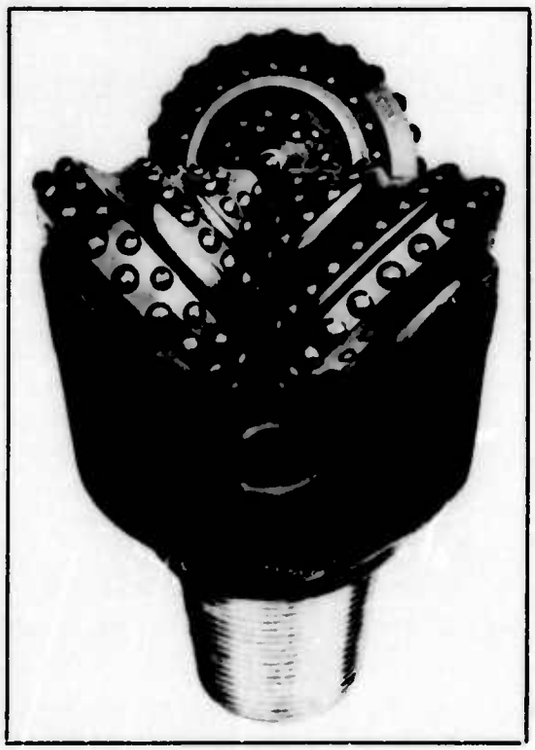


(a) Weak rock bit—for clays, salt, soft limestone, soft shales.



(b) Intermediate-strength rock bit—for medium-hard limestones, dolomite, slaty shales.

(c) Intermediate-strength rock bit—for pyrite, chert, dolomites, grantie, porphry.



(d) High-strength rock bit—for taconite, flint, basalt.

Fig. 44. Roller cone drill bits (used by permission of Varel Manufacturing Company, Dallas).

Higher strength materials require shallow, closely spaced teeth and an orientation that results in material failure by compression rather than by scraping. Tungsten carbide inserts are installed in place of teeth on the bit cone for use in the highest strength materials. Figure 44 shows roller cone bits used for weak, intermediate-strength, and high-strength formations. Roller drill bits are readily available in sizes ranging from 3 to 26 in. in diameter.

Diamond Bits

Diamond drill bits are cylindrical in shape with diamonds set in the contact arc (Fig. 45). They are usually of small diameter and are often used for coring. Diamond bits require greater rotation speeds but less thrust. Their expense prohibits their use in large-diameter drilling.

Drag Bits

Drag bits consist of two or more blades with tungsten carbide inserts interspersed throughout the blades (Fig. 46). They are used chiefly in unconsolidated or soft materials, such as clay shales. Drag bits range in size up to 26 in.

Large Hole Bits

Bits greater than 26 in. have been specially designed for large-diameter drilling. The most widely used drill bit for very large-diameter drilling is the flat or plate type. Large rolling-tooth or several tri-cone-type cutters are fixed to

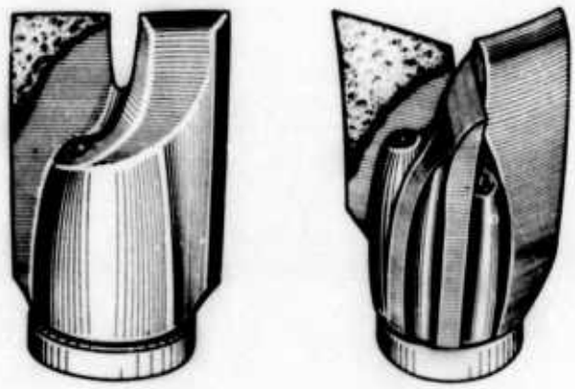
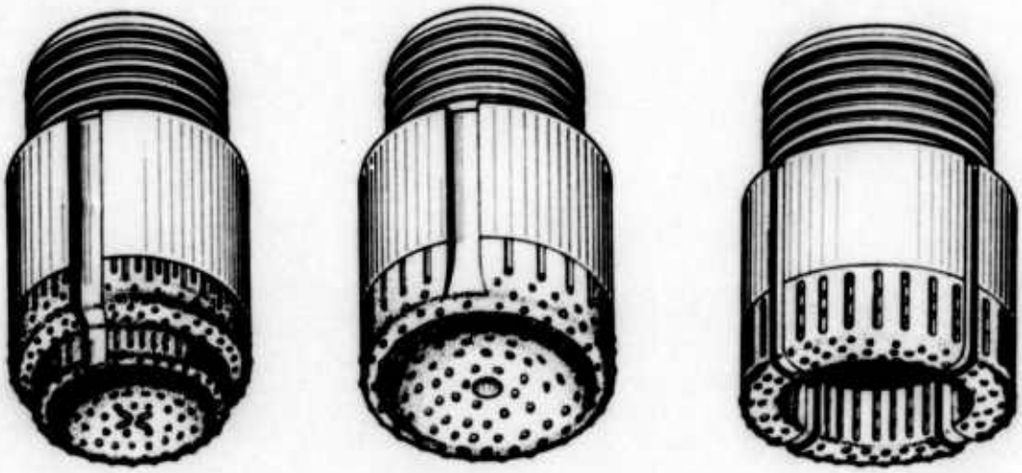


Fig. 46. Drag bits.[36]



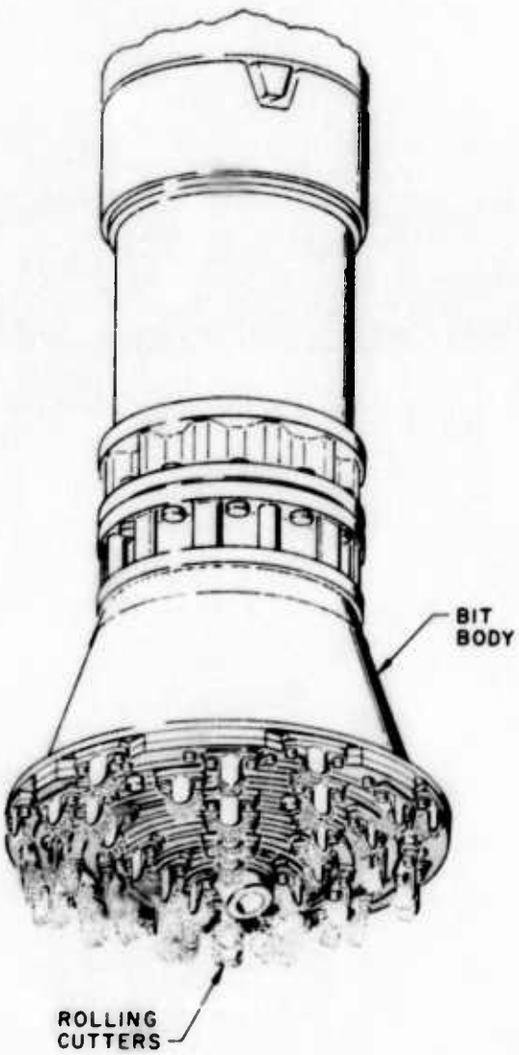
(a) Full-hole or plug bits. (b) Core bit.

Fig. 45. Diamond drill bits (used by permission of E. I. duPont de Nemours and Company, Wilmington, Del.).

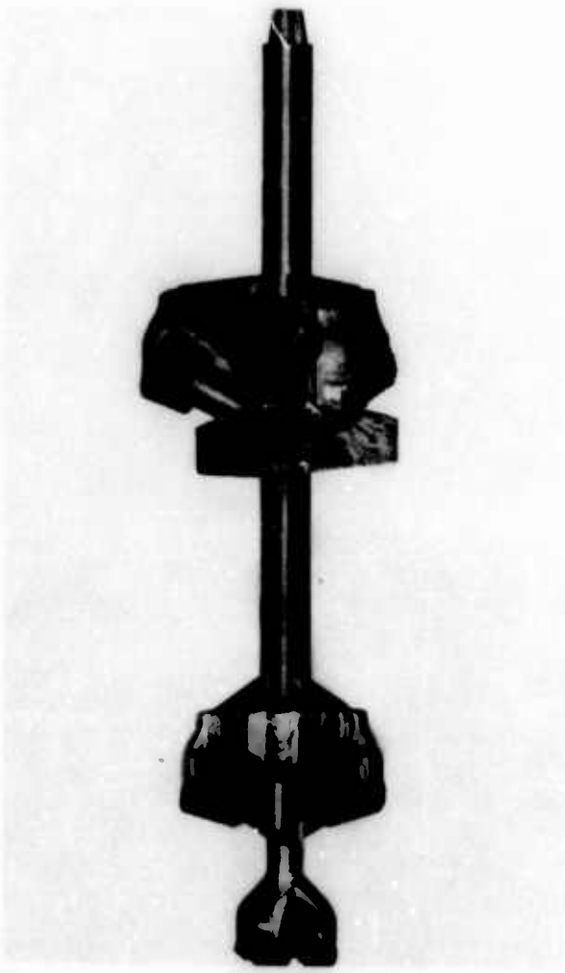
the flat bottom of a cylindrical bit body. Another large-diameter bit that incorporates rolling cutters is a stage-type bit comprising a series of progressively larger bits bolted together in a stacked arrangement. Figure 47 illustrates a flat type hole bit and a stacked type hole bit.

Before the advent of "large-hole bits," large holes were drilled with a multiple-pass system. First, a pilot hole was drilled, followed by a series of expansion tools until the desired diameter was obtained. This was a slow process, and occasionally a cutter arm on one of the tools broke off causing more delays.

The stacked drill bit solved the problem of large-hole drilling. A series of drill bits was stacked one on top of the other with the smallest at the bottom and the largest at the top. In operation each succeeding bit would enlarge the hole made by the bit beneath it,



(a) Flat type hole bit.



(b) Stacked type hole bit (used by permission of Smith Tool, Compton, Calif.).

Fig. 47. Large hole bits.

thereby eliminating the multiple-pass system.

In harder formations, reverse circulation must be used for proper bottom hole cleaning with air, air mist, water, or mud as the circulating medium. Because the stacked bit design is not conducive to reverse circulation, the flat bottom bit was developed.

Rotary Speed

The drilling rate for a given type and size of bit increases with increasing rotary speed. Large-diameter rotary drills require a high sustained thrust and a low rotary speed. In small-diameter borings, rotary speeds in excess of 200 rpm are common. Rotary speeds such as this in large-diameter borings produce high peripheral velocities on the outside cutters of large-diameter bits, which velocities would cause rapid tooth wear and bearing failure. Rotary speeds of this nature would require prohibitively high torque. A rule of thumb for the rotary speed in the drilling of large-diameter holes is $\frac{120}{D}$ = RPM, where D = the hole diameter in feet.[52]

Bit Weight

The rotary drill imparts axial thrust and torque through the bit into the rock. Each drill rig has an optimum thrust associated with the available torque for a maximum penetration rate in a specific medium. Operating below optimum thrust results in a decrease in penetration rates and may impart a grinding or polishing action on the rock. Operating above optimum thrust requires high torque, which can overexert and stall the drill. As the hole size increases, the weight on the bit is generally increased to maintain a satisfactory drilling rate. Figure 48 shows the effect of increased bit weights on the drilling rate with oil-field size, tri-cone rock bits on intermediate- to-high-strength material. Large-diameter bits, however, require bit weights different from individual tri-cone bits. Suggested bit weights for large-diameter drilling are shown in Table 19.

DRILLING EQUIPMENT

The previous sections of this chapter discussed various techniques used in drilling emplacement cavities. This section will discuss the capabilities and limitations of drill rigs presently in use. There are two main categories of drill rigs, fixed and mobile. The latter can be further categorized into methods of mobility.

Fixed rigs are those in which a semi-permanent structure must be built to support the drilling equipment. These are used when one is drilling deep holes that require weeks or months to complete.

Mobile drill rigs are used where a large number of relatively shallow (less than 1000 ft) holes are required. These units require only a small crew (2 to 4 men) to operate.

Table 19. Suggested bit weights for large-diameter drills.[36]

Formation hardness	Weight (lb/in. of bit diameter)
Weak rock	1,000
Intermediate-strength	1,500-2,000
High-strength	3,000

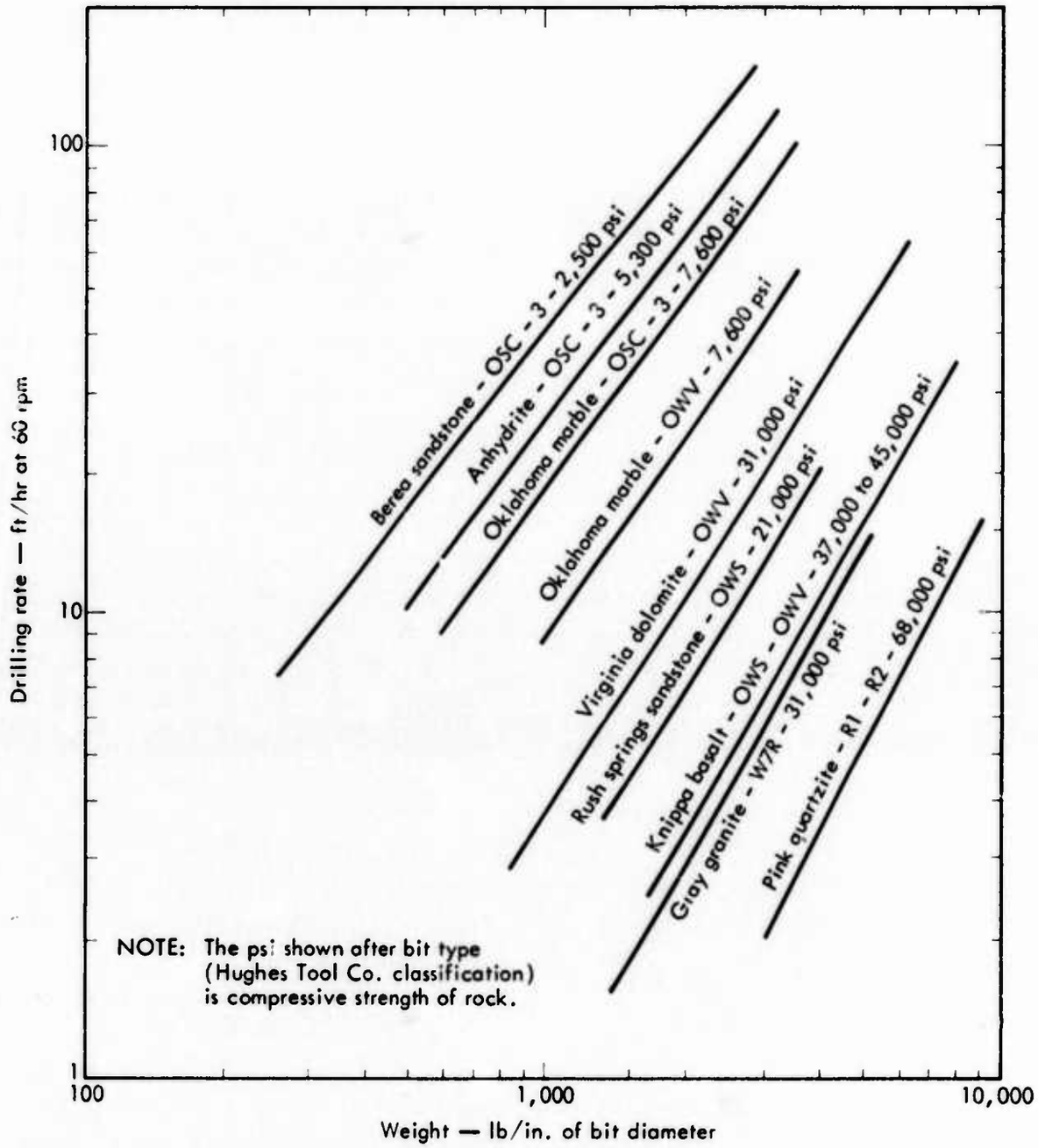


Fig. 48. Drilling rate as a function of weight for new rock bits.[53]

Truck-Mounted Drill

A drill rig mounted on the frame of a large truck makes it highly mobile (Fig. 49). Generally, two men are required to operate one rig, and set-up time is only the time required to position the drill at the desired hole location (about 10 min.). Approximately 10 days of training are required to give the operator sufficient skill to operate one of these rigs. Truck-mounted rotary, auger, and percussion drills are readily available from various manufacturers. Holes range up to several feet in diameter

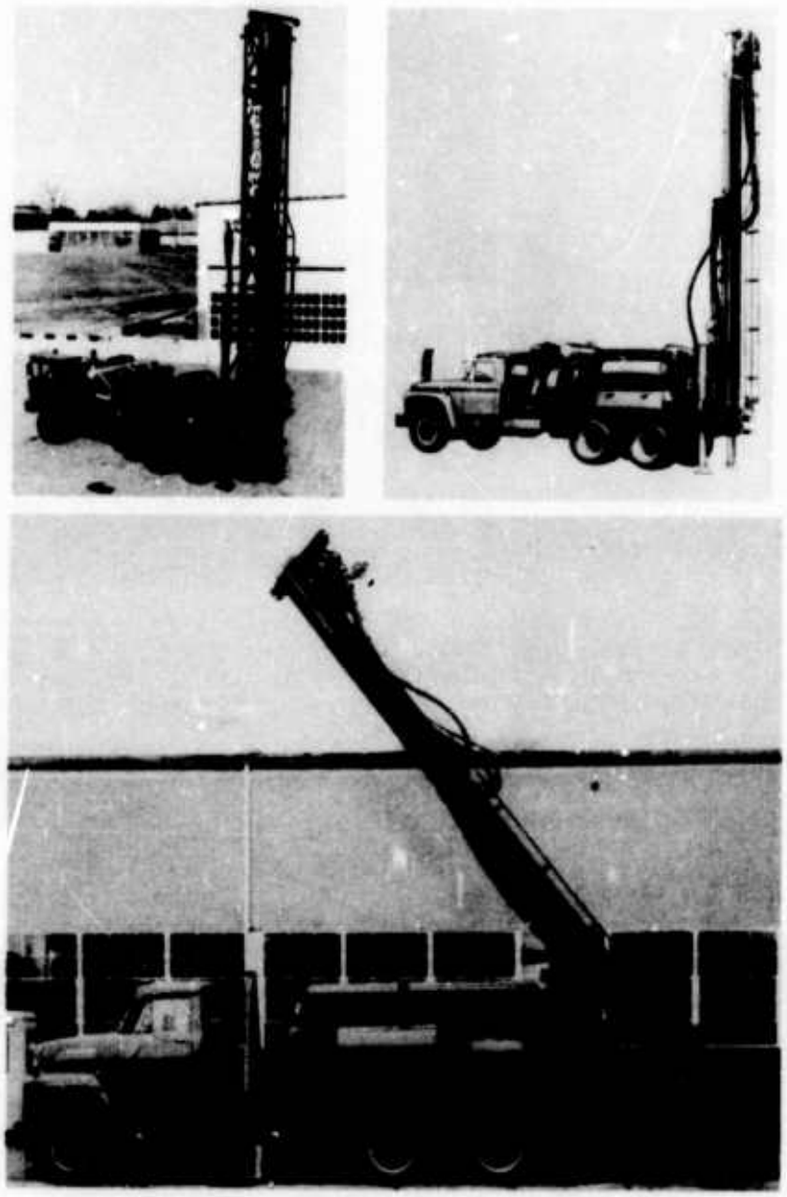


Fig. 49. Three types of commercially available truck-mounted rigs (used by permission of Schramm, Inc., West Chester, Pa.).

depending on the type of bit used and material being drilled.

Crawler-Mounted Drill Rigs

Crawler-mounted rigs are mounted on a unit with caterpillar tread (Fig. 50). These units are basically the same as the truck-mounted drill. However, they can operate in terrain trucks are unable to negotiate.

Portable Drills

These are small, compact units mounted on skids. These units are very versatile in that they can be mounted on various vehicles, such as trucks or trailers.

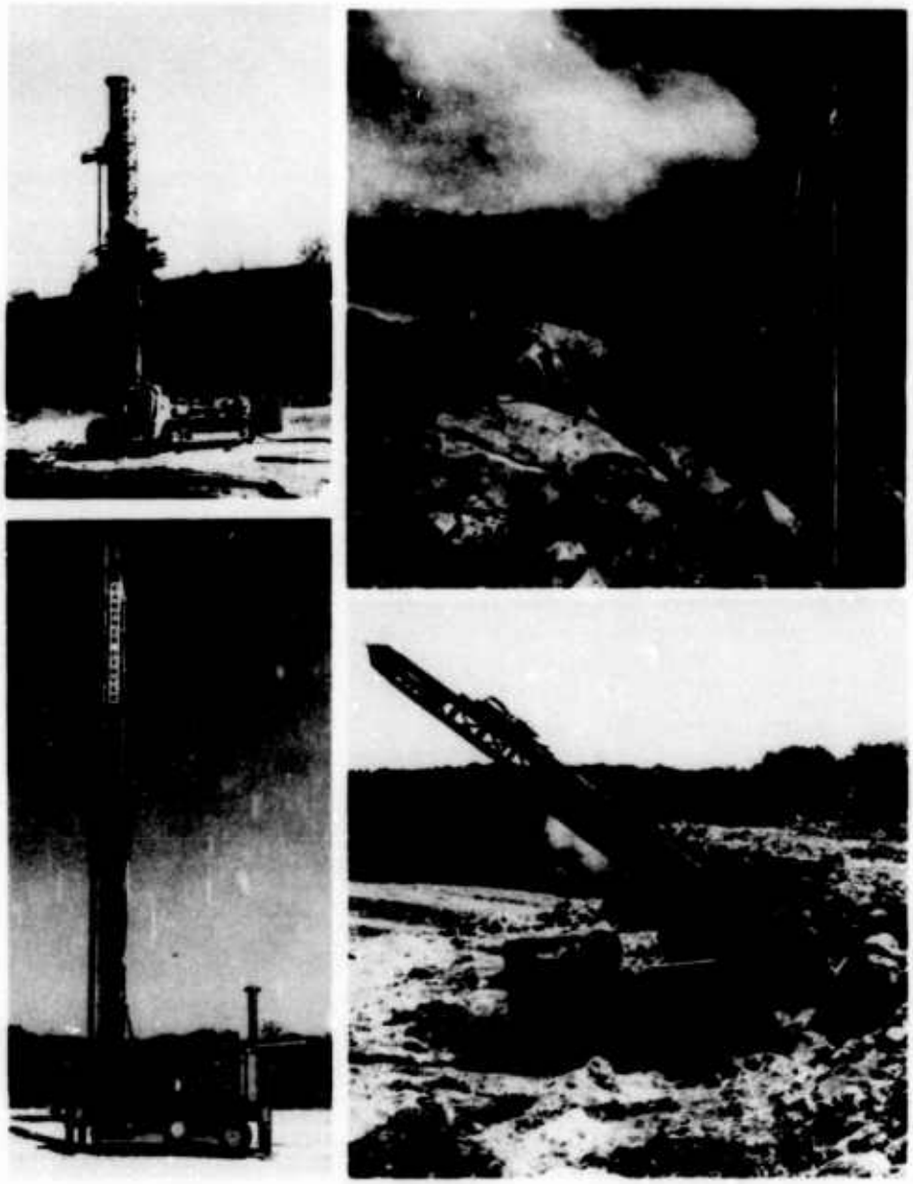


Fig. 50. Crawler-mounted rigs (used by permission of Gardner-Denver Company, Quincy, Ill.).

MILITARY DRILL RIGS

The following is a brief summary of military drilling capability.

Pneumatic Percussion Drill[54]

The pneumatic percussion drill is a lightweight (57-lb) hand-held drill capable of drill holes 1-5/8 to 2-in. in diameter to a maximum depth of 10 ft.

Wagon-Mounted Drifter[55]

The wagon-mounted drifter is a pneumatic rotary-percussion drill mounted on a wheeled frame, making it highly

mobile in level terrain or areas of easy access; it requires one operator. Maximum hole size is 4 in. in diameter and 24 ft in depth.

Crawler-Mounted Drifter[56]

The crawler-mounted drifter, like the wagon drill, is a pneumatic rotary-percussion drill. Unlike the wagon drill, it is a self-propelled unit powered by compressed air. A two-man crew is required to operate this unit, which is capable of drilling 2-3/4 to 4-in. diameter holes,

Earth Auger[57]

The earth auger is a skid-mounted, gasoline-powered unit requiring a crew of two to operate. It is capable of drilling holes from 9 to 20 in. in diameter but is limited to a maximum depth of 9 ft.

Well Drill[58]

The well drill, like the earth auger is gasoline-powered and skid-mounted. However, it is a percussion-type drill (operated by a three-man crew) capable of drilling holes 6 to 8 in. in diameter to a depth of 1,000 ft.

As can be seen, military drills have no capability for drilling large emplacement cavities for bulk explosives.

SUMMARY

Drilling requirements for large explosive emplacement cavities exceed the capabilities of present military equipment. The earth auger employed by the Army drills holes of sufficient diameter (9 to 20 in.); however, it is limited to use in weak material and to a depth of only 9 ft. The skid-mounted percussion drill (well drill), on the other hand, can drill holes to a depth of 1,000 ft, well in excess of hole requirements. However, it approaches only the lower limit (9 in.) of hole diameter required for large chemical charges.

In a tactical situation time is an important factor. Present drilling methods are limited to drilling rates of only a few feet per hour in high-strength rock. For this reason, novel drilling techniques, which have the capability of exceeding present drill rates, are being studied. The drill must be capable of high penetration rate with a minimum amount of energy and support equipment. The mechanically induced stress drills, although still in the developmental state, appear to satisfy the above requirements more completely than any of the other novel drill techniques.

Chapter 6 Excavation Design

SCOPE

This chapter describes the basic design procedures available to the military engineer for the use of chemical explosive excavation techniques. Although the emphasis is on cratering design, basic mounding design procedures will be included. First, the methods used to predict the dimensions of craters will be introduced, followed by the design procedures currently used for explosive excavations.

Empirical scaling is the method for predicting crater dimensions used throughout the text. It is the simplest of the methods in current use and most immediately applicable to practical engineering situations. Other methods, requiring complex calculations based on the conservation laws of mass, momentum, and energy, are used at the present time primarily as research tools. The presentation of empirical scaling includes experimentally developed curves relating crater dimensions to the depth of burial of the charges. The curves indicate the expected performance of a 1-ton charge of TNT in each of three geologic media. These results may be extended to other charge weights by the scaling rules, and they may be extended to other explosives and media by other considerations, including judgment.

The results from single-charge detonations can be used to predict the results from the detonation of rows of charges by the application of additional empirical rules. Going beyond the qualitative introduction to cratering and the representative parameters discussed in Chapter 1, this chapter presents basic quantitative cratering data and outlines procedures for applying the data to the design of single-charge and row-charge excavations, and (with somewhat less experimental verification) to multiple row-charge excavations. Related matters including interconnecting row craters, underwater cratering, charge shapes, stemming, and delayed row-charge detonations are also discussed.

PREDICTION OF CRATER GEOMETRY

Of the several means for predicting crater size and geometry, empirical scaling is the most practical for engineering. It offers accuracy comparable to that of elaborate computational methods while requiring only a fraction of the time and effort. In situations in which several charges are used to form a row-charge crater, empirical rules are the sole means for predicting the results.

Since the apparent crater forms the useful excavation, its size is of first importance. Other crater dimensions, such as lip height, are secondary but may also be important for evaluating the entire crater for certain engineering applications.

Although there may be few applications for single-charge craters, the capability to predict their size is essential. This capability forms the basis for predicting the size of row-charge craters.

Empirical Scaling of Crater Radius and Depth

Fundamental Scaling Parameter

The fundamental parameter in scaling is a quantity that represents the ability of an explosive charge to produce a crater. This quantity is not heat release, maximum pressure, or mass, but is some factor that takes into account all properties of the explosive and medium. As indicated in the description of crater formation, the initial shock and the pressure after some gas expansion are both important properties when an explosive is detonated in rock. In weak materials the gas pressure may be more important than the initial shock. To avoid complications in the scaling of crater dimensions, it is presumed that the explosive is TNT, with its particular set of characteristics, and that the proper scaling parameter is charge weight. If another explosive is used, its effectiveness relative to TNT, discussed in Chapter 3, may be introduced as an adjustment to the computation.

In charge-weight scaling, crater dimensions are known for a reference charge weight and are multiplied by a scaling factor to predict the results for other weights. The scaling factor may be the cube root or the fourth root of the ratio of the charge weights; or it may be a fractional exponent lying between the cube and fourth roots. The reference charge in this discussion and subsequent development is one ton of TNT.

Cube Root Scaling

Cube root scaling may be derived from dimensional analysis by neglecting the effects of gravity and dissipative conditions, such as friction. With this form of scaling a crater dimension, radius for example, would scale as follows:

$$R_a = r_a (Y/Y_0)^{1/3},$$

where r_a is the crater radius for a charge having the weight Y_0 (which is 1 ton), and R_a is the crater radius for a charge of Y tons. The crater depth and the charge burial depth would scale similarly.

Cube root scaling gives reasonably accurate scaling of those crater dimensions and explosion effects that are little influenced by gravity. It also scales all effects from small-scale explosions in low-strength materials. However, it fails to scale accurately the crater dimensions, depth in particular, for explosions of more than a few tons.

Fourth Root Scaling

Another form of scaling that may be derived by dimensional analysis and from principles of similitude is fourth root scaling. Although it is probably the fundamental form of scaling, a number of similar conditions that it requires cannot be met by explosions of less than several thousand tons.

Empirical Scaling

Empirical scaling has been developed to provide a reliable scaling rule over the range of charge weights most often used in practical engineering situations. In this form of scaling the scale factor for crater dimensions is the ratio of the charge weights raised to an exponent that is intermediate between the cube root and the fourth root. A commonly accepted value for the empirical charge-weight scaling exponent, 0.3, is used here. Another value, 1/3.4, may be found in the

literature. The two exponents are so nearly equal that predictions of crater dimensions would differ by only a few percent if one were used in place of the other. Extensive cratering tests have led to the conclusion that the empirical scaling exponent, 0.3, can be confidently applied at depths of burial near optimum. At very shallow or deep depths of burial this value would be used with less confidence. In these cases it would be more appropriate to conduct a series of small-scale tests to develop another empirical constant for the desired media and depth of burial.

To apply empirical scaling it is necessary to ascertain the crater dimensions for the reference charge weight in the medium being considered, and to multiply them by the scale factor. Usually, the depth of burial, the apparent crater radius, and the apparent crater depth are the only crater variables considered in scaling. Other crater dimensions, such as lip height, are expressed as some multiple of crater radius, crater depth, or depth of burial.

With a reference charge of 1 ton, the scaling factor, P, is determined very simply:

$$P = Y^{0.3} , \qquad (2)$$

where Y is the weight in tons of the charge, the crater dimensions of which are to be computed.

The charge depth of burial, DOB, the apparent crater radius, R_a, and the apparent crater depth, D_a, would then be computed from the relationships:

$$DOB = P\,(dob). \qquad (3)$$

$$R_a = Pr_a, \qquad (4)$$

$$D_a = Pd_a, \qquad (5)$$

where dob, r_a, and d_a are the depth of burial and the crater dimensions for a 1-ton charge of TNT.

Often in practice the charge weight required to produce a crater of specified dimension is the quantity to be computed. In this case the specified crater dimension is divided by the corresponding crater dimension for a 1-ton charge to obtain the scale factor P. Then the unknown charge weight, Y, may be computed by the simple relation:

$$Y = P^{3.33} . \qquad (6)$$

In this form the relationship between charge weight and scale factor clearly indicates that great quantities of explosive are necessary to produce large craters. In general, the linear dimensions of a single-charge crater are doubled when the charge weight is increased by a factor of ten. Crater volume, being proportional to the cube of linear dimensions, increases by a factor of 8 when the charge weight is increased tenfold.

The crater dimensions scaled to 1-ton TNT charges in dry rock, dry soil, and saturated clay shale are shown in Figs. 51 through 53, respectively. The curves for crater radius and depth are based on data from experiments involving charge weights from 0.25 to 500 tons, with charge weights from 0.5 to 20 tons being most common.

The cratering curves for dry rock are based primarily on data from experiments in basalt, a high-strength rock with some verification from experiments in rhyolite. The curves for dry soil apply to desert alluvium, loess, dry sand, and materials of similar physical properties. They also apply to certain low-strength sandstones. The curves for saturated clay shale are

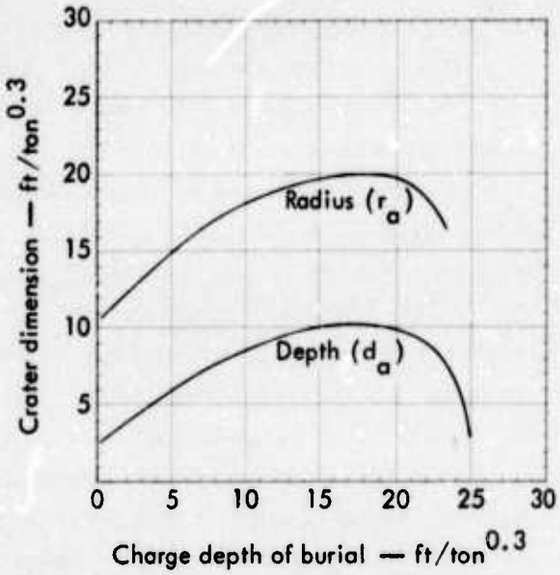


Fig. 51. Crater dimensions scaled to 1-ton (TNT) charges buried in dry rock.[1]

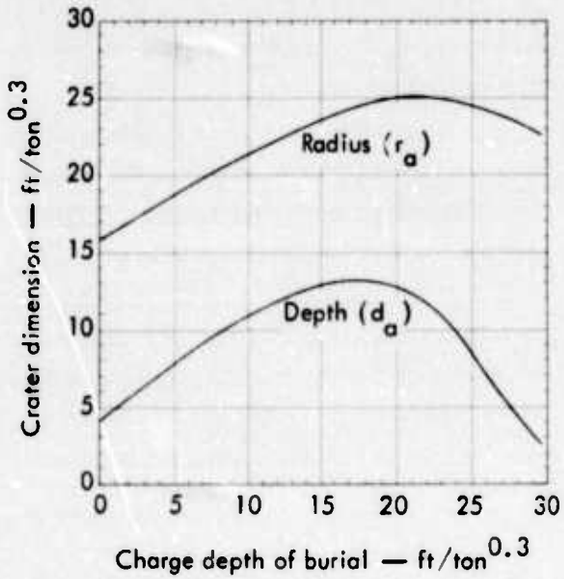


Fig. 52. Crater dimensions scaled to 1-ton (TNT) charges buried in dry soil.[1]

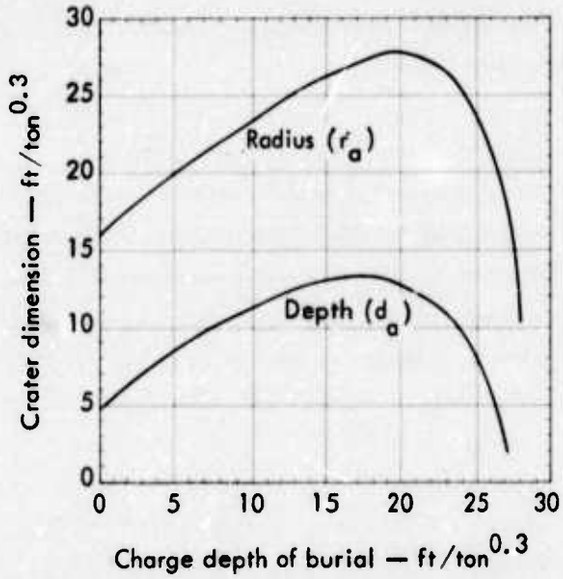


Fig. 53. Crater dimensions scaled to 1-ton (TNT) charges buried in saturated clay shale.[1]

also reasonably valid for saturated sand. The crater curves for dry rock and saturated clay shale may be regarded as the lower and upper limits, respectively, for crater dimensions in materials not mentioned above.

For maximum efficiency, a cratering change is ordinarily buried at a depth that will assure the greatest apparent crater volume. In the three materials considered here the optimum depths of burial and the resulting crater dimensions are listed in Table 20 for the range of typical charge sizes. The burial depths and dimensions for the 10- and 50-ton charges are simply those for the 1-ton

Table 20. Single-charge crater parameters for optimum depth of burial.[1]

Material	Charge size (tons)								
	1	10	50	1	10	50	1	10	50
	Depth of burial (ft)			Crater radius (ft)			Crater depth (ft)		
Dry rock	18	36	58	20	40	65	10	20	32
Dry soil	20	40	65	25	50	81	12	24	39
Saturated clay shale	18	36	58	28	54	87	13	26	42

charge multiplied by $10^{0.3}$ and $50^{0.3}$, respectively.

Supplemental Crater Parameters

Although the apparent crater radius and depth are the first criteria for explosive excavation design, parameters that describe the crater lip, the extent of the ejecta, and the extent of the rupture zone are useful. In Table 21 these parameters are given in terms of crater radius, or crater depth, and apply to craters produced by charges detonated at optimum depth. With the exception of lip crest radius, these parameters may vary over a considerable range. The lip height, for example, may vary by a factor of 2 around the perimeter of a typical crater. Comparable variation may be expected for the radius of continuous ejecta and the size of the rupture zone.

SINGLE-CHARGE CRATERS

Because maximum explosive efficiency is desired for excavation applications, the design information used here is based on burial at optimum depth; i.e., the depth below ground surface at which the detonation will produce the largest apparent crater. Explosive engineering projects, such as quarrying or underwater blasting, may require that charges be placed at a different depth.

Experimentally obtained single-charge crater dimensions for three materials are given in Figs. 54 through 56.[15,59-65] The optimum depth of burial and the radius, the depth, and the volume of the apparent crater are read directly from these figures. This information, which is used to select the weight and corresponding depth of burial for single charges, is also used to design row-charge craters and will be referred to frequently in this chapter. The optimum scaled crater dimensions are shown on these charts to permit computation of crater dimensions for charge weights outside the range of the graphs.

To illustrate the use of these charts, assume that a single crater with a depth of 27 ft in dry rock is desired. Figure 54 shows that a charge of 27 tons of TNT will excavate the desired crater, and that it will have a radius of 54 ft and a volume of almost 4,000 yd^3. The depth of burial is computed from the optimum DOB shown in the figure and the scaling factor:

$$DOB = 18 \times 27^{0.3} = 48 \text{ ft.}$$

Table 21. Supplemental single-charge crater parameters.

Parameter	Dry rock	Dry soil	Saturated clay shale
Lip crest radius (R_{al})	1.2 R_a	1.2 R_a	1.4 R_a
Lip height (H_{al})	0.25 D_a	0.15 D_a	0.45 D_a
Radius of continuous ejecta (R_{eb})	3.0 R_a	2.2 R_a	3.5 R_a
Radius of rupture zone at surface	4.4 R_a	—	4.0 R_a
Radius of rupture zone at charge elevation	1.1 R_a	—	2.0 R_a
True crater radius (R_t)	1.0 R_a	1.0 R_a	1.1 R_a

It is pertinent at this time to point out that the crater dimensions in Figs. 54 through 56 are based on averages derived from experimental data that are characterized by appreciable amounts of scatter. Considerable departures from the values given by these figures could be experienced; thus it is prudent to think of the crater dimensions as reliable to within a 20% margin.

When the material to be excavated varies widely from the materials represented by Figs. 54 through 56, it is suggested that test charges be detonated and the resulting crater dimensions be scaled up to the required charge-weight level.

CRATERING IN MEDIA OVERLAIN BY WATER

In some applications of explosive excavation, especially those related to navigation improvement, the rock to be excavated lies under water. The water

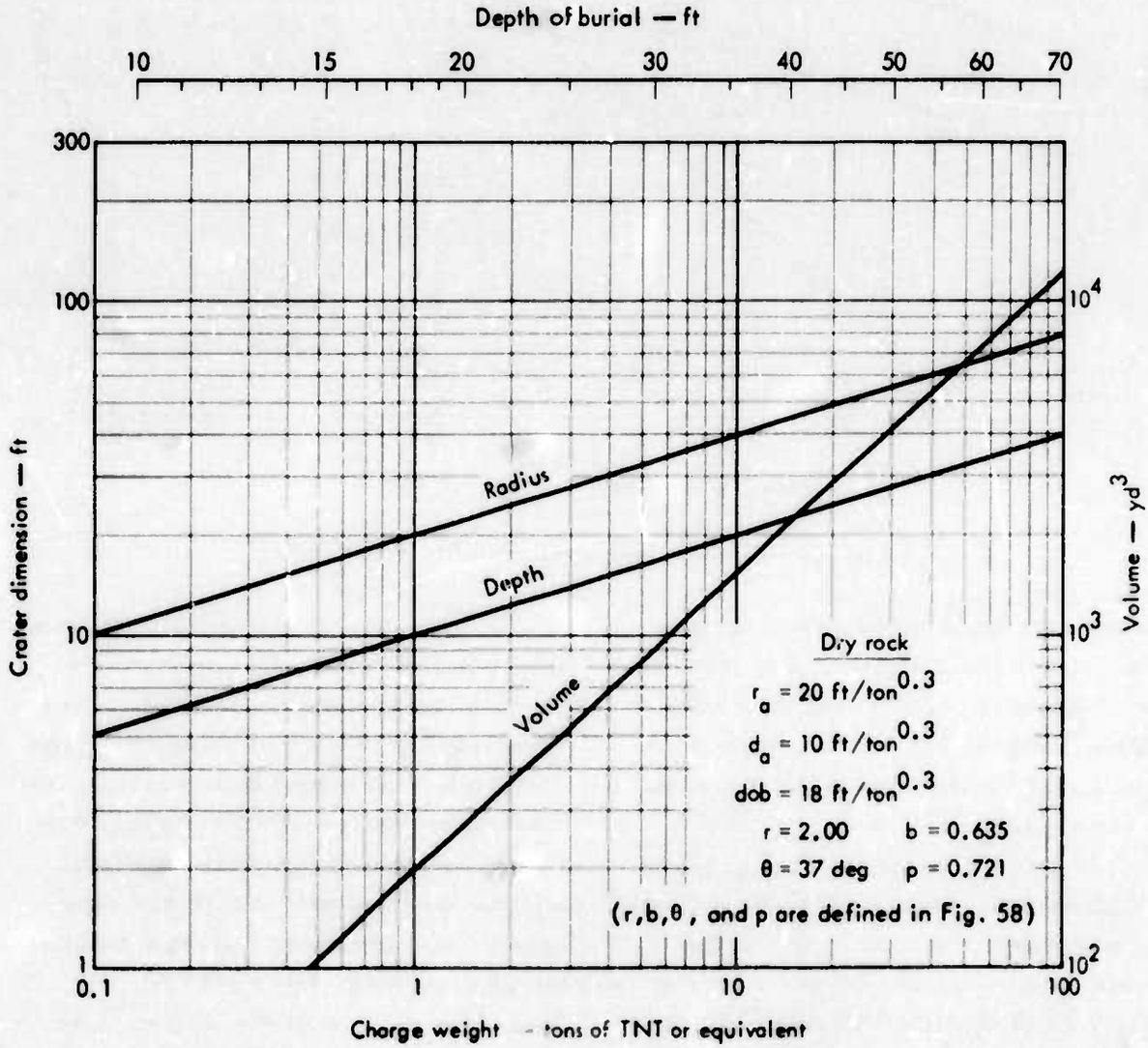


Fig. 54. Crater dimension data for dry rock.[1]

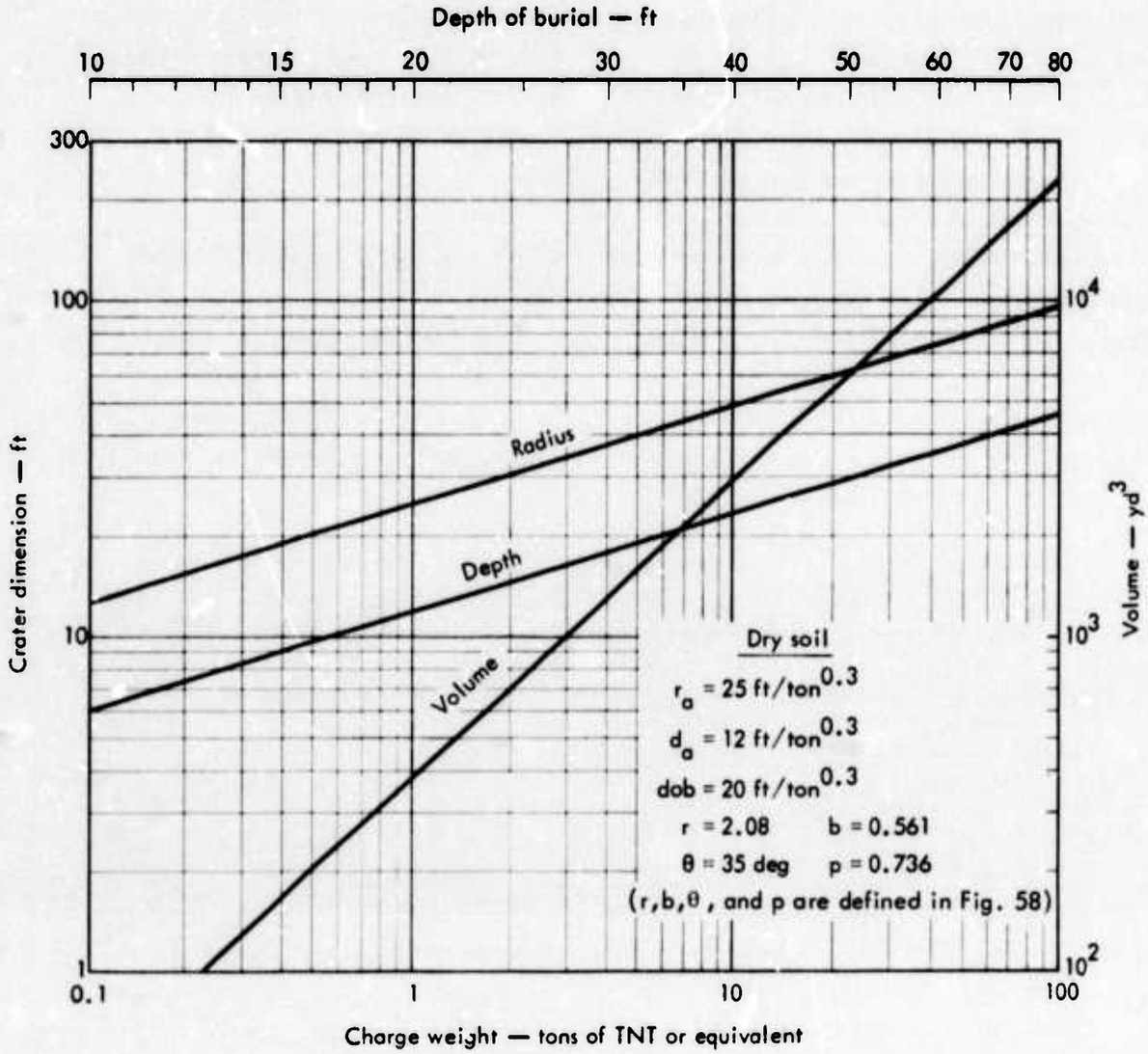


Fig. 55. Crater dimension data for dry soil.[1]

overburden has a pronounced influence on crater characteristics. The inrush of displaced water after the detonation redistributes ejecta and may wash material into the crater that would otherwise remain outside it.

There are no reliable scaling relationships for predicting the size and geometry of underwater craters. Although the crater radius scales well when half the water depth is utilized to determine the total charge depth of burial, the crater depth may not scale similarly.

To make use of underwater cratering on a practical basis, it is necessary to perform experiments to determine crater geometry under the particular conditions at the site. As a first approximation, the underwater crater radius is the same as for a land crater in similar material, and the depth is half that of the land crater. In determining charge depth of burial, the water layer may be regarded as a layer of the bottom material having a thickness equal to one-half the water depth.

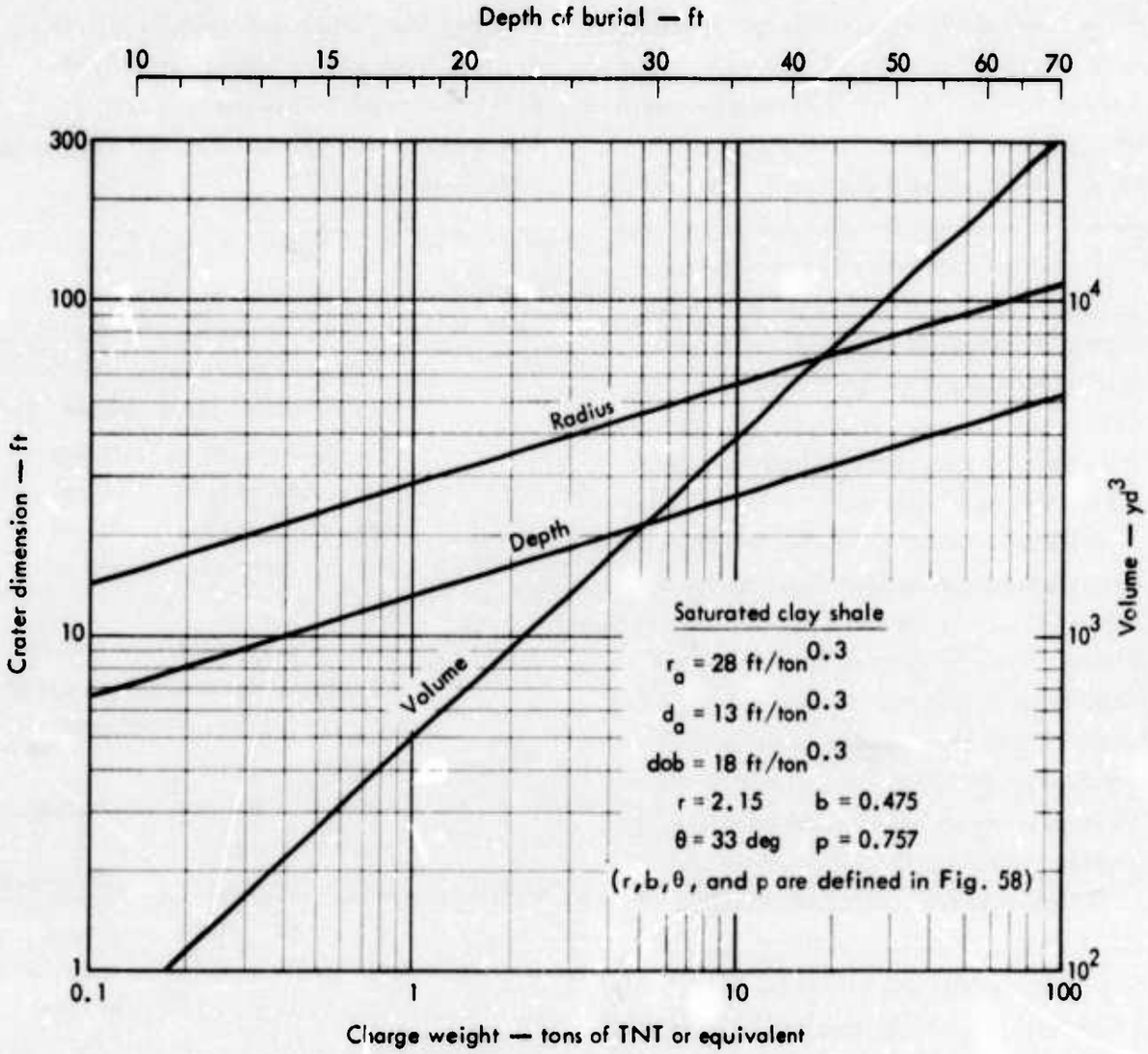


Fig. 56. Crater dimension data for saturated clay shale.[1]

ROW-CHARGE CRATERING

Concept and Background

Row Charges

In many applications, it is necessary to use an array of several charges to produce a crater of suitable geometry. The most simple and most common array is a row of charges. The charges, usually five or more in number, are buried along the alinement of the desired crater with a horizontal spacing approximately equal to the crater radius for a single charge having the weight of one row-charge member. A properly designed row of charges will excavate a trench having a smooth and uniform cross section even though the charges are separated by some distance. Furthermore, the excavated volume per ton of explosive in the row is greater than for a single charge in the same material.

Factors that influence the size and geometry of row-charge craters include

those involved in single-charge cratering with the added factors of charge spacing and time delay, if any, between detonations of adjacent charges. Although the depth of burial of a single charge may be varied over a wide range to alter certain crater characteristics, current row-charge design procedures have been tested for depths of burial of row charges over a fairly narrow range near optimum. There are as yet no reliable design procedures for row charges having depths of burial other than near optimum.

In the discussion of single-charge craters, it was noted that crater dimensions and depths of burial for any size charge may be scaled from the crater dimensions and depth of burial for a one-ton charge. Row crater dimensions and charge depths of burial may also be scaled. The manner in which this can be done will now be described and the scaling parameter developed.

A characteristic of row craters is that their width, W_a, and depth, D_{ar}, are generally larger than the diameter, $2R_a$, and depth, D_a, of a single-charge crater excavated by a charge equal in weight to one of the charges in the row. This increase in dimensions is called enhancement, and the size of a row crater can be expressed in terms of single crater dimensions by means of an enhancement factor (the scaling factor).[66] Because the enhancement of row crater dimensions increases as the charge spacing is decreased, the size of a row crater can be altered by changing the layout of the charges as well as their weights. The characteristic of enhancement is used in the design of row charges.

As previously mentioned, the excavated volume per ton of explosive in a row of charges has been found empirically to be greater than for a single charge in the same material. This experimental observation can be stated by the following equation:

$$A_r e^2 S = K V_a , \qquad (7)$$

where

A_r = cross-section area of optimum single-charge crater or unenhanced row-charge crater

e = enhancement of row-crater dimension relative to single-crater dimension

S = spacing between charges in row*

K = empirically determined ratio of volume excavated by charge in row to volume of single-charge crater (V_a)

V_a = volume of the optimum single-charge crater.

The terms on the left-hand side of Eq. (7) simply represent the volume excavated by each charge in the row. The enhancement factor, e, is squared because the width and depth of the row crater are enhanced by equal amounts. Equation (7) can be rewritten:

$$e = \left[\frac{K V_a}{A_r S}\right]^{1/2} , \qquad (8)$$

to show that the enhancement of row crater dimensions is inversely proportional to the square root of the charge spacing.

Further, Eq. (8) may be written as:

$$e^2 = \frac{K V_a}{A_r R_a (S/R_a)} , \qquad (9)$$

---

*Experiments have, to date, covered only a range of spacings of $0.55 < S/R_a < 1.4$.

in which charge spacing is now expressed in terms of the optimum single-charge crater radius, this being the most convenient method of expressing row-charge spacing. Now, it has been found that the dimensionless quantity, V_a/A_rR_a, which appears on the right-hand side of Eq. (9) has a value of approximately 1.1 for an extremely wide range of crater geometries. This factor can be considered a constant, and so Eq. (9) can be rewritten again as:

$$e^2 = \frac{1.1K}{S/R_a}. \qquad (10)$$

The factor K, which can be thought of as the efficiency of a charge in a row compared to a single charge, has been determined from several field tests. The best current estimate for the value of K is approximately 1.3, which means that a row charge is about 30% more efficient than a single charge. Putting this value into Eq. (10) results in the following approximation:

$$e^2 = \frac{1.4}{S/R_a}. \qquad (11)$$

Equation (11) relates the enhancement of row-charge crater dimensions to the charge spacing in the row. It can be seen that a spacing of $1.4R_a$ will result in a row crater with no enhancement. Equation (11) is relied upon to design row-charge craters.

An important adjunct of the concept of enhancement is the fact that the depth of burial of charges in a row must be the optimum single-charge depth increased by the amount of enhancement. Enhancement is discussed in greater detail in a later section.

The nominal length of a row crater formed by N charges is S(N + 1). The length of the crater segment having uniform cross section is equal to the distance between the first and last charges, or S(N - 1). If the distance between the first and last charges is less than twice the crater width, the crater will not be linear but will resemble a single-charge crater. For charge spacings of about 1 crater radius, a minimum of five charges is needed to assure that the distance between the first and last charges is approximately twice the crater width.

In theory there is no reason why the charge weight and spacing cannot be simultaneously decreased to the point where adjacent charges are in physical contact. In practice, however, it appears that certain end effects occur when a row of many small charges is substituted for one having a few large charges. These end effects are recognized by a reduction in depth over the end charges and the consequent flattening of slopes as illustrated in Fig. 57.

It is not necessary that the charge weight and spacing between charges within the row charge be uniform. In fact, there are many applications in which the charge weight and spacing must be varied to produce a uniform cut through varying terrain.

In the preceding discussion it was assumed that all charges in the row were detonated simultaneously. If time delays are used to reduce ground shock or airblast from the detonation, a reduction in crater volume results. The magnitude of this reduction will be influenced by the delay interval that is used. Experiments have indicated that time delay firings of charges in a row will tend to decrease the depth of the row crater. As the delay time is increased the decrease

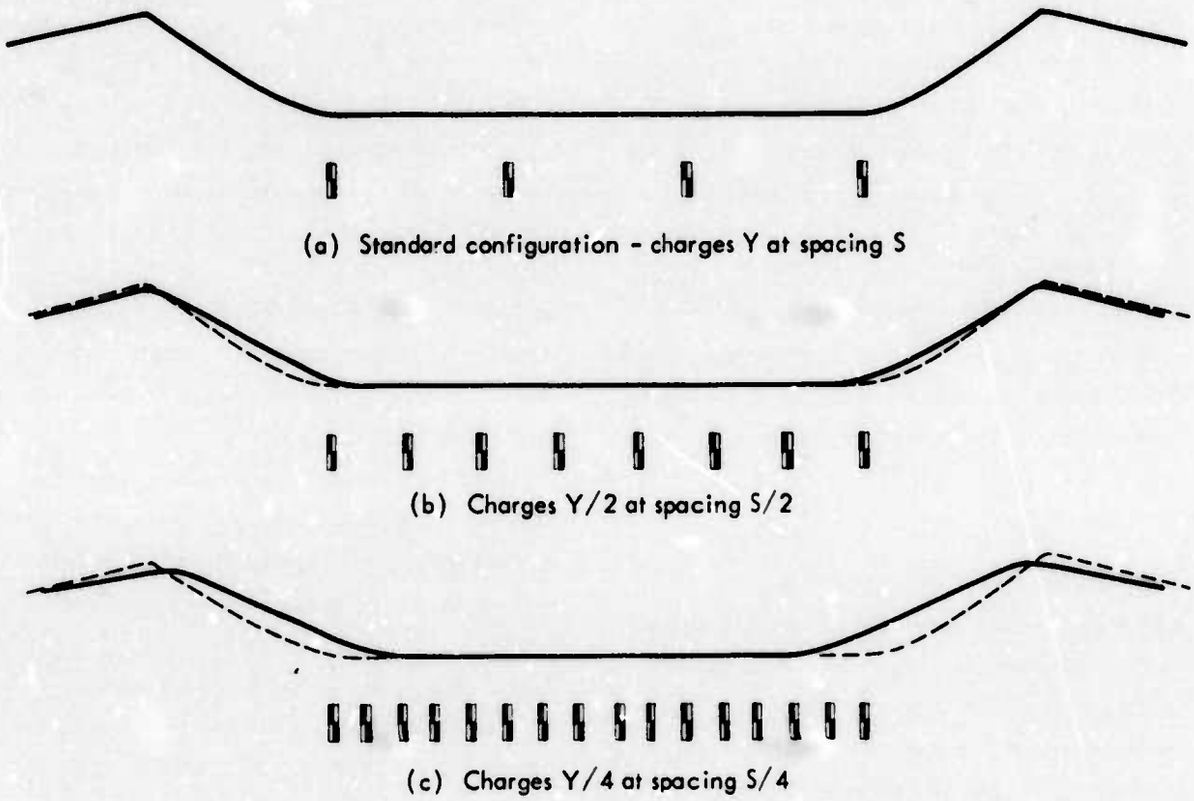


Fig. 57. Effects of charge size and spacing on row-crater end slopes (not to scale).

in crater depth approaches a limit as the delay is made indefinitely long. For long delays the crater depth is approximately half that for the same row of charges simultaneously detonated. The crater width is reduced only slightly. Criteria for use in designing delay intervals will be discussed later.

Multiple-Row-Charge Cratering

In applications in which crater width is more important than depth, it may be advantageous to use two parallel rows of charges. Such a charge configuration is detonated with a delay between the rows and can be designed to produce a crater typically one and one-half times as wide as the crater from either row of charges acting alone. The crater depth will be approximately the same as for a single row-charge crater. The design of two parallel rows is discussed in detail in a later paragraph. Investigations into the procedures for designing excavations with three rows of charges have just been initiated.

Design Procedures

As discussed previously, if a row of charges is properly emplaced and detonated simultaneously, a smooth channel or linear crater will be excavated. The design of row-charge craters is based on single-charge cratering data and the enhancement of row crater dimensions that occurs when charges are spaced closer together. The enhancement phenomenon permits flexibility in the choice of individual charge weights to excavate linear craters,

Flat Terrain

The design of a row-charge crater in flat, level terrain is less complicated than the design of a similar excavation through varying terrain, and will, therefore, be taken up first.

The first step in designing a row-charge crater in level terrain is to select the single-charge weight from the appropriate single-charge dimension chart (Figs. 54 through 56) that will produce the required half-width (i.e., radius) and depth of the cut. It is likely that only one of these two dimensions will be required to determine the weight; for example, a given weight may be adequate for the depth of a cut but not the width, and therefore the width requirement would govern the weight to be used.

After the single-charge weight has been determined, the weight and spacing of the charges in the row can be altered, if desired, in response to construction considerations involved in emplacing the charges. This also is done by considering enhancement which, from the derivation earlier in this chapter, can be expressed in terms of charge spacing:

$$e^2 = \frac{1.4}{S/R_a}, \qquad (12)$$

where

- e = enhancement of single-charge crater dimensions (both width and depth are equally enhanced)
- S = charge spacing
- R_a = (optimum) single-charge crater radius.

If $S/R_a = 1.4$, then $e = 1$ (no enhancement) and the single-charge weight determined from Figs. 54 through 56 would be the weight required for a row of charges spaced $1.4\ R_a$ apart that achieves the required width or depth. The charges would be emplaced at optimum depth of burial as for single charges of this weight. If a closer spacing is used ($S/R_a < 1.4$ and $e > 1$), it is no longer necessary to use the same weight charges to obtain the required crater dimensions; however, the smaller charges would still be emplaced at the depth of burial used for the $1.4\ R_a$ spacing (i.e., at the optimum depth shown for the single-charge weight on the crater dimension chart).

Assume that the single-charge weight that will produce the required half-width (radius) and depth of the row crater is Y_s, but that it is desirable to use charges of lower weight, Y_r, in the actual excavation of the row crater. The required amount of enhancement can then be expressed by:

$$e = \left(\frac{Y_s}{Y_r}\right)^{0.3} . \qquad (13)$$

When Eqs. (12) and (13) are combined, an expression for the proper spacing of the smaller charges, Y_r, is obtained:

$$\frac{S}{R_a} = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6} , \qquad (14)$$

where R_a is the single-charge crater radius of a charge of weight Y_r. The radius, R_a, is read from the appropriate single-crater dimension chart. The depth of burial for charge weight, Y_r, is obtained from the same chart, but it must be increased by the amount of enhancement. This increase in depth is equivalent to burying Y_r at the depth indicated for Y_s on the single-charge crater chart.

It may be noted that the enhancement phenomenon implies that if the charge weights determined for 1.4 R_a spacing are used at a closer spacing without increasing their depths of burial, the design no longer achieves the efficiency associated with optimum depth of burial. For a smooth excavation, charges in a row should not be spaced farther apart than 1.4 R_a; spacings greater than 1.4 R_a will result in cusping of the crater sides and bottom.

Example—Suppose it is desired to excavate a diversion channel 75 ft wide and 23 ft deep in dry soil. Figure 55 shows that the depth of the channel determines the charge weight, and that it will require a single-charge weight (i.e., unenhanced row-charge weight) of 11.0 tons. However, assume that 2-ton charges are desirable because of construction problems in emplacing charges larger than this size. Equation (13) gives the required amount of enhancement as:

$$e = \left(\frac{Y_s}{Y_r}\right)^{0.3} = \left(\frac{11.0}{2}\right)^{0.3} = 1.67.$$

The charge spacing for the 2-ton charges is found from Eq. (14):

$$\frac{S}{R_a} = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6} = 1.4\left(\frac{2}{11.0}\right)^{0.6} = 0.51,$$

where R_a is 31 ft for a 2-ton charge; therefore:

$$S = 0.51\ R_a = 0.51 \times 31 = 15.5 \text{ ft.}$$

The depth of burial of the 2-ton charges is the optimum depth obtained from the legend of Fig. 55 increased by the amount of enhancement and the charge weight scaling; i.e.,

$$20 \times 2^{0.3} \times 1.67 = 24.5 \times 1.67 = 41 \text{ ft,}$$

which, it will be noted, is the same depth of burial that would be required for an 11.0-ton charge, the original single-charge weight, Y_s. The 2-ton charges placed at a depth of burial of 41 ft and spaced 15.5 ft apart will produce a crater with the following width and depth:

$$W_a = 2\ (eR_a) = 2\ (1.67 \times 31) = 103 \text{ ft}$$

$$D_{ar} = eD_a = 1.67\ (14) = 23 \text{ ft.}$$

So far only the width or depth of the apparent crater has been considered as the design criteria; however, these criteria may not be sufficient for specifying the desired cut. If a project requires the excavation of a channel that can accommodate a specific navigation prism (i.e., one in which the width of the channel, L, is specified at some depth, D, below the original ground surface), then Eq. (15) in Fig. 58 can be used to compute the necessary single-charge weight. The concept of enhancement can be used as before to reduce this weight in a row of charges.

Example—Assume that it is desired to excavate a canal through hard, dry rock with a width, L, of 30 ft at a depth, D, of 10 ft below ground level. The single-charge weight is computed by Eq. (15) with the crater constants p and b given in Fig. 58:

$$Y_s = \left[0.72\left(\frac{10}{10}\right) + \sqrt{0.52\left(\frac{100}{100}\right) + \frac{900}{4(400)} - \frac{0.72}{1.63}\left(\frac{100}{100}\right)}\right]^{3.33}$$

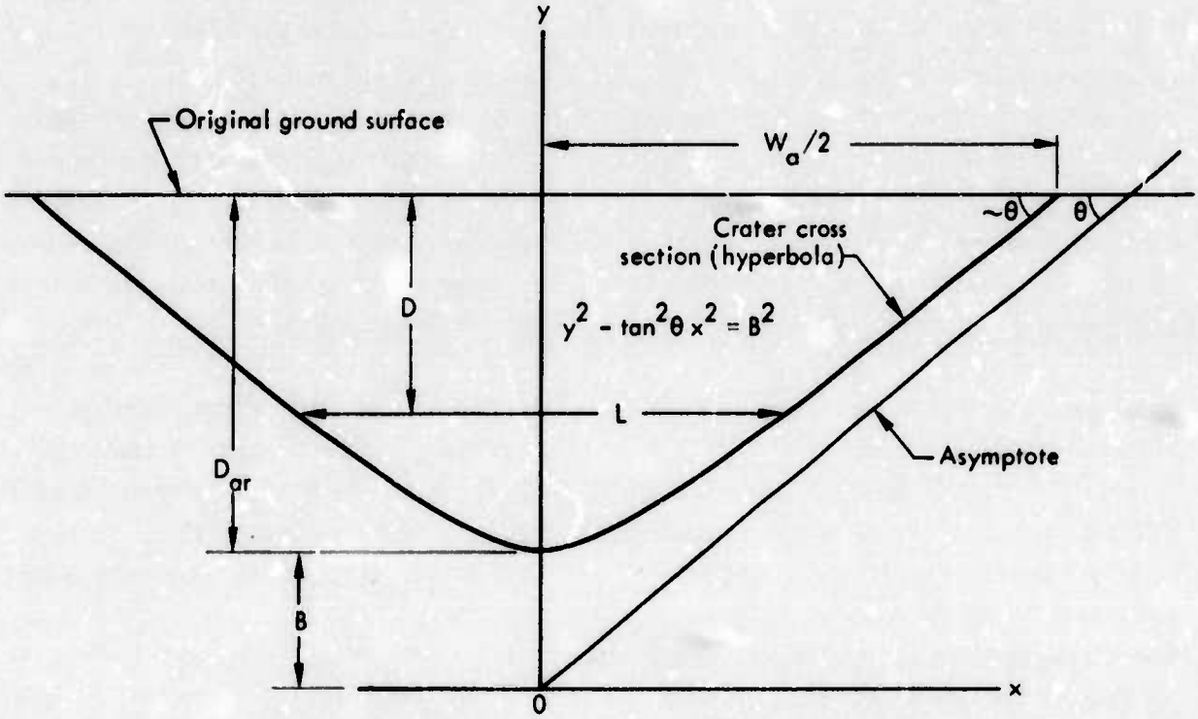
$= (1.52)^{3.33}$

$= 4.0$ tons.

This charge weight can be reduced by close spacing and enhancement with the procedures previously discussed.

Varying Terrain

The design of a row of charges to excavate a channel through varying terrain is an extension of procedures described in the preceding section. There are two design procedures that can be applied to a cut through varying terrain, and the choice of the one to be used in a particular case must be based on a consideration of operational factors. The two design techniques are very similar and differ only in the manner in which the concept of enhancement is applied. One method will result in a row of charges with varying weights, and the other will result in a row with all charges equal in weight. For convenience, the former is termed the "constant-enhancement"



$$Y = \left[p\frac{D}{d_a} + \sqrt{p^2 \frac{D^2}{d_a^2} + \frac{L^2}{4r_a^2} - \frac{p}{b+1}\frac{D^2}{d_a^2}} \right]^{3.33} \qquad \text{Eq. (15)}$$

where $p = \dfrac{b+1}{2b+1}$

$b = \dfrac{\tan^2\theta\, r^2 - 1}{2}$ (and $B = bD_{ar}$)

$r = \dfrac{r_a}{d_a}$

(Note: Values for the parameters r, b, p, and θ are given in Figs. 54 through 56 for the three cratering media discussed in this chapter.)

Fig. 58. Typical hyperbolic row-crater cross section.

method (i.e., constant S/R_a), and the other as the "constant-charge-weight" design. In some instances, particularly those in which the terrain relief is not great, it may be more economical to use the constant-charge-weight method because of the construction advantages of having a large number of charges or emplacement holes of the same size. If one is cutting through steep terrain for which only a few relatively large charges are required, the constant-enhancement approach may have a slight advantage.

Constant-Enhancement Method — The constant-enhancement approach is discussed first. The first procedure is the determination of the single-charge weight of the largest charge in the row, this being the charge directly beneath the point of highest elevation (or deepest cut). This charge weight is reduced to a convenient value by the use of close spacing and enhancement, if necessary, and then the remaining charges in the row are computed. The charge spacing (in terms of S/R_a) selected to reduce the largest charge to a convenient weight is maintained over the length of the detonation. Computing the weight and positioning of the remaining charges in the row requires a slightly different procedure depending on whether the excavation is a channel with a specified bottom elevation or one that will contain a navigation prism.

Consider, first, a cut with a specified bottom elevation. When the maximum single-charge weight has been determined according to the highest elevation (or more exactly, the greatest depth of cut) and the appropriate crater dimension chart, this weight can be reduced to the desired level by Eq. (14), which establishes the charge spacing in the row in terms of S/R_a. The weights required in the remainder of the row are determined by means of a modification to the crater dimension chart. The modification consists of shifting the depth curve up the vertical axis by the amount of enhancement, this amount being computed by Eq. (13). Weights for any other depth of cut can then be read directly from the modified graph. Charge spacings are the appropriate fraction (S/R_a) of the unenhanced radius for a given charge, and depths of burial are read at the intersection of the depth of cut and the unenhanced depth curve.

In the typical case of a cut through varying terrain, there will be adjacent charges of unequal weight. The spacing between adjacent charges should be the average of the spacings computed for each of the charges.

Horizontal positioning of the charges is then determined by the following process. Assume that the weight of a charge in a row is Y_m and that the crater radius for this charge is R_{am}. The spacing, S_m, for this charge would then be:

$$S_m = \frac{S}{R_a} R_{am} . \qquad (16)$$

Similarly, the spacing for an adjacent charge, Y_n, would be:

$$S_n = \frac{S}{R_a} R_{an} . \qquad (17)$$

The actual spacing between Y_m and Y_n is the average of S_m and S_n; i.e.,

$$S_{m-n} = \frac{S_m + S_n}{2} . \qquad (18)$$

This procedure is repeated for the remaining charges in the row. An example of designing a cut with a specified bottom elevation through varying terrain is given below.

Example—Assume that it is desired to excavate a channel with a constant bottom elevation through dry rock along the profile shown in Fig. 59. The deepest cut is 25 ft (i.e., 37 - 12), and from Fig. 54 the maximum single-charge weight (Y_s) required is found to be 21 tons. However, assume that emplacement considerations dictate a maximum weight of 5 tons (Y_r). The necessary amount of enhancement is given by Eq. (13):

$$e = \left(\frac{Y_s}{Y_r}\right)^{0.3} = \left(\frac{21}{5}\right)^{0.3} = 1.54 ,$$

and Eq. (14) gives the required charge spacing in terms of S/R_a as:

$$\frac{S}{R_a} = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6} = 0.61 .$$

The charge spacing of $S/R_a = 0.61$ will be maintained throughout.

It will now be convenient to modify Fig. 54 by shifting the depth curve vertically up on the figure by the amount of enhancement; i.e., a factor of 1.54. An appropriately modified form of the crater dimension chart can then be used to read required charge weights directly. A modified Fig. 54 for use in this example is shown in Fig. 60.

It has been established that the first and largest charge is 5 tons. From Fig. 60 it is found that R_a for a 5-ton charge is 32 ft. Designating this first charge as Y_1, then:

$$S_1 = 0.61 \times 32 = 19.5 \text{ ft.}$$

This charge of 5 tons will be buried at the same depth as the 21-ton single charge; i.e., 44 ft.

Proceeding with the design to the left of the highest point of the profile, one measures the depth of cut a distance S_1 (i.e., 19.5 ft) to the left of Y_1. This depth of cut is 17 ft, and from the enhanced depth curve in Fig. 60 Y_2 is

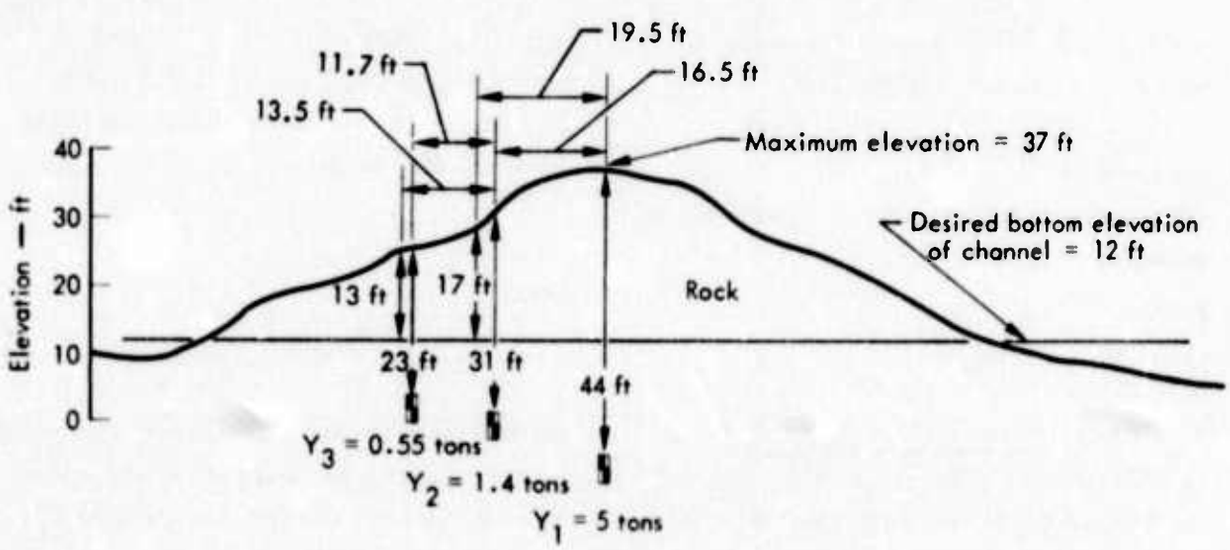


Fig. 59. Row-charge design to excavate channel with constant bottom elevation.[1]

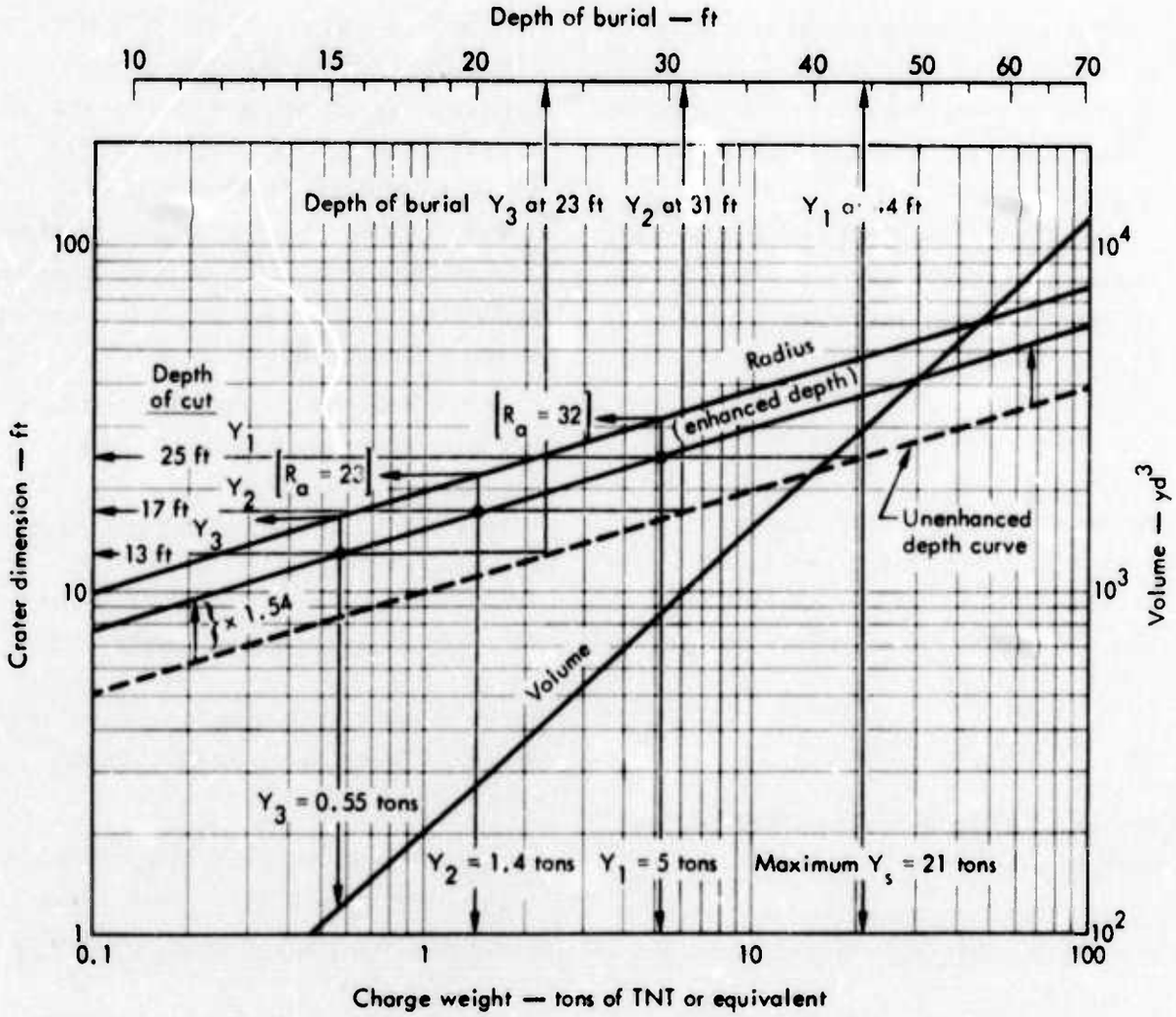


Fig. 60. Modified crater dimension chart to be used with example in Fig. 59.[1]

1.4 tons, R_a is 23 ft, and the depth of burial is 31 ft. It follows that:

$$S_2 = 0.61 \times 23 = 13.5 \text{ ft.}$$

The actual spacing between Y_1 and Y_2 is adjusted to be:

$$S_{1-2} = \frac{S_1 + S_2}{2}$$

$$= \frac{19.5 + 13.5}{2} = 16.5 \text{ ft.}$$

The weight of the third charge, Y_3, is determined by measuring the depth of cut a distance S_2 (i.e., 13.5 ft) to the left of Y_2. The depth of cut is 13 ft, and, according to Fig. 60, Y_3 is 0.55 tons, R_a is 16.5 ft, and the depth of burial for Y_3 is 23 ft. Thus:

$$S_3 = 0.61 \times 16.5 = 10 \text{ ft,}$$

and

$$S_{2-3} = \frac{13.5 + 10}{2} = 11.7 \text{ ft.}$$

As this procedure is continued, the design of the row charge proceeds to the end of the cut, to both the left and right of Y_1.

Consider now the design of an excavation that will circumscribe a navigation

prism. This design procedure requires more computations than the foregoing, although essentially the same steps are followed. Rather than the determination of charge weights directly from a modified crater dimension chart, the basic single-charge weights must be individually computed by means of Eq. (15) in Fig. 58. The enhancement, e, and spacing, S/R_a, used to reduce the largest weight to a convenient level is also used for the other charges in the detonation.

Example—The procedure for designing a cut to accommodate a navigation prism is most easily illustrated by an example. Assume it is desired to excavate a channel through dry rock represented by the profile in Fig. 61. Let the design specifications for the channel be a width of 25 ft at elevation zero. Starting at the highest elevation of 33 ft, the required single-charge weight is determined by Eq. (15)

$$Y_1 = \left[0.72\left(\frac{33}{10}\right) + \sqrt{0.52\left(\frac{1089}{100}\right) + \frac{625}{4(400)} - \frac{0.72}{1.64}\left(\frac{1089}{100}\right)}\right]^{3.33}$$

$$= 65 \text{ tons.}$$

If it is desired to reduce the 65-ton charge to 30 tons, then Eq. (13) gives the required enhancement:

$$e = \left(\frac{Y_s}{Y_r}\right)^{0.3} = \left(\frac{65}{30}\right)^{0.3} = 1.25 ,$$

and Eq. (14) gives the required charge spacing:

$$S/R_a = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6} = 1.4\left(\frac{30}{65}\right)^{0.6} = 0.9 .$$

The 30-ton charge will be buried at the depth for a 65-ton charge, which, from Fig. 54, is 62 ft. The appropriate charge spacing for Y_1 is S_1, and is computed as:

$$S_1 = 0.9 \times 56 = 50 \text{ ft,}$$

where 56 ft is the crater radius for a 30-ton charge.

When one proceeds to Y_2, the next charge to the left of Y_1, the elevation S_1 ft to the left of Y_1 is determined to be 22 ft. The single-charge weight for Y_2 is computed by Eq. (15):

$$Y_2 = \left[0.72\left(\frac{22}{10}\right)\right.$$

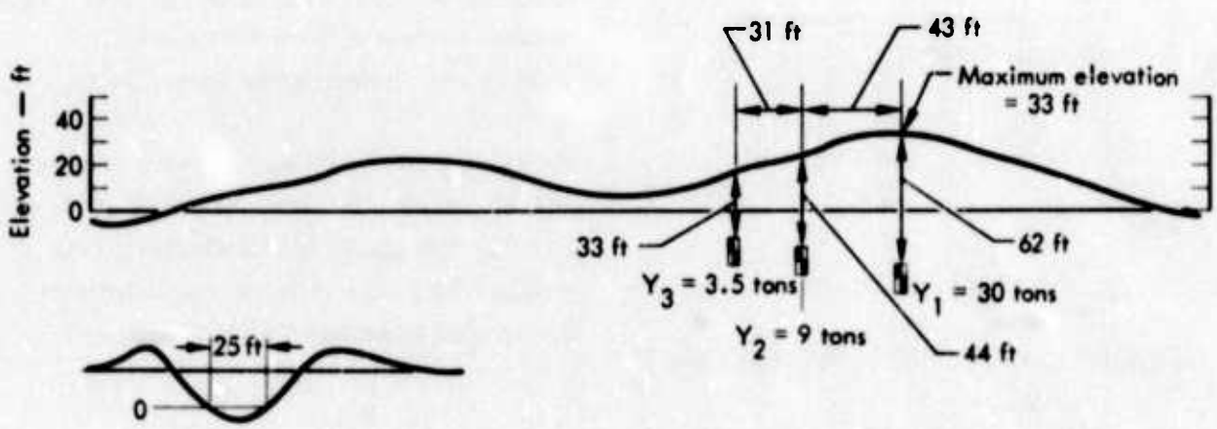


Fig. 61. Row-charge crater with navigation prism through varying terrain.

$$+ \sqrt{0.52\left(\frac{484}{100}\right) + \frac{625}{4(400)} - \frac{0.72}{1.64}\left(\frac{484}{100}\right)}\Bigg]^{3.33}$$

$$= 20 \text{ tons.}$$

This charge weight is reduced in the same proportion that Y_1 was reduced, so that now:

$$Y_2 = 20\left(\frac{30}{65}\right) \simeq 9 \text{ tons.}$$

The 9-ton charge is buried at the depth for a 20-ton charge, which is 44 ft. The appropriate spacing for Y_2 is S_2 and is computed as

$$S_2 = 0.9 \times 39 = 35 \text{ ft,}$$

where 39 ft is the crater radius for a 9-ton charge. The actual spacing between Y_1 and Y_2 is:

$$S_{1-2} = \frac{S_1 + S_2}{2}$$

$$= \frac{50 + 35}{2} \simeq 43 \text{ ft .}$$

When one continues to the third charge, Y_3, the elevation, a distance of S_2 (35 ft) to the left of Y_2 is determined to be 15 ft; so that:

$$Y_3 = \Bigg[0.72\left(\frac{15}{10}\right)$$

$$+ \sqrt{0.52\left(\frac{225}{100}\right) + \frac{625}{4(400)} - \frac{0.72}{1.64}\left(\frac{225}{100}\right)}\Bigg]^{3.33}$$

$$= 7.6 \text{ tons.}$$

The reduced weight will be:

$$Y_3 = 7.6\left(\frac{30}{65}\right) = 3.5 \text{ tons.}$$

The depth of burial will be 33 ft and the spacing for Y_3 is:

$$S_3 = 0.9 \times 29 = 26 \text{ ft.}$$

The spacing between Y_2 and Y_3 is then:

$$S_{2-3} = \frac{S_2 + S_3}{2}$$

$$= \frac{35 + 26}{2} \simeq 31 \text{ ft.}$$

This procedure is continued until the excavation design is complete.

Often it may be advisable to excavate the canal or channel by interconnecting row-charge craters. A connecting row-charge crater may use a different charge spacing, and often it will be advantageous to increase the charge spacing as the depth of cut decreases because fewer charges will be required. The connection of row craters is covered later.

Constant-Charge-Weight Method—In the constant-charge-weight method, the relative spacing between charges, S/R_a, is varied rather than the weight of the charges.

In the case of a cut with a specified bottom elevation, the initial step is to select the charge weight that is to be used throughout the detonation. The spacings between the remaining charges are determined by means of the appropriate crater dimension chart and Eq. (14). The method can be illustrated by using the profile and channel depth shown in Fig. 62.

Example—Assume that 3-ton charges will be used. The deepest cut is 25 ft, and Fig. 54 shows that a single-charge weight of 21 tons is required; therefore the charge spacing is given by:

$$S/R_a = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6} = 1.4\left(\frac{3}{21}\right)^{0.6} = 0.43,$$

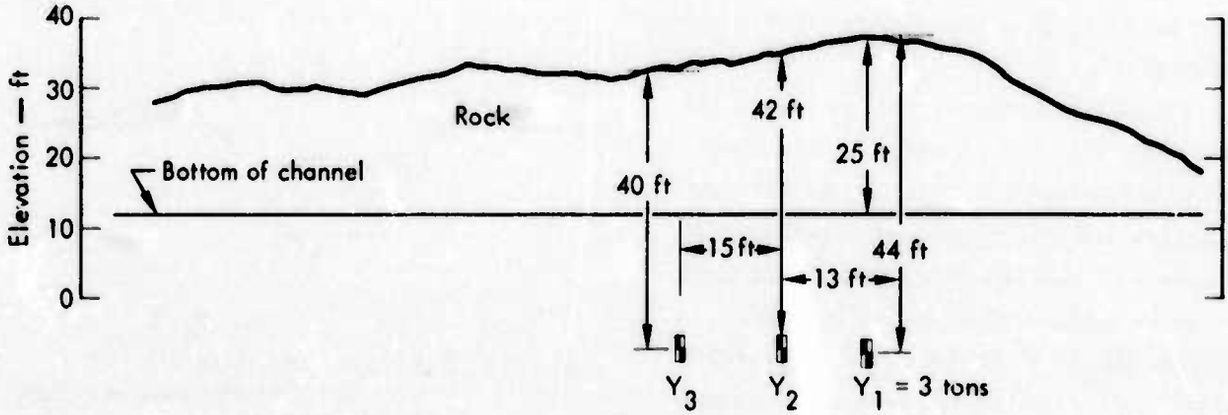


Fig. 62. Example of cut with constant bottom elevation and constant charge weights.[1]

and, because R_a for a 3-ton charge is 28 ft, then:

$$S_1 = 0.43 \times 28 = 12 \text{ ft.}$$

The 3-ton charge is buried at the depth for a 21-ton charge, 44 ft. The next step is to look at the depth of cut, S_1 ft from the first charge, and in Fig. 62 it is 23 ft. According to Fig. 54 this depth of cut requires a single-charge weight of 17 tons and a depth of burial of 42 ft. The spacing, S_2, for charge Y_2 is computed as follows:

$$S/R_a = 1.4 \left(\frac{3}{17}\right)^{0.6} = 0.50,$$

and therefore,

$$S_2 = 0.50 \times 28 = 14 \text{ ft.}$$

As with the constant-enhancement method, the actual spacing between Y_1 and Y_2 is the average of S_1 and S_2:

$$S_{1-2} = \frac{S_1 + S_2}{2} = \frac{12 + 14}{2} = 13 \text{ ft.}$$

To compute the spacing between Y_2 and Y_3, look at the depth of cut S_2 ft from Y_2. Because this depth in Fig. 62 is 21 ft, a single-charge weight of 14 tons and a depth of burial of 40 ft are required. S_3 is computed:

$$S/R_a = 1.4 \left(\frac{3}{14}\right)^{0.6} = 0.56;$$

therefore,

$$S_3 = 0.56 \times 28 = 16 \text{ ft,}$$

and

$$S_{2-3} = \frac{S_2 + S_3}{2} = \frac{14 + 16}{2} = 15 \text{ ft.}$$

The process is continued in this manner until the end of the cut. In some instances it may occur that the spacing (S/R_a) computed for a charge is greater than 1.4. Because this is the upper limit of row-charge spacing, it will be necessary to select a lower yield for the remaining charges such that S/R_a is everywhere less than 1.4. Otherwise, the procedure as described above has no exceptions.

The procedure for designing a constant-charge-weight row to excavate a crater for a specific navigation prism is similar to the above. Individual single-charge weights are computed by Eq. (15) rather than by obtaining them from a crater dimension chart, but the spacings are

computed in the same manner as the above example.

The use of enhancement to reduce individual row-charge weights is subject to an important constraint. As smaller charges are spaced closer together, the number of charges detonated at one time must be increased in order to avoid a short, elliptical crater. As a general rule, each row-charge detonation should result in a crater the length of which is at least twice its width. The following relationship gives the number of charges in a row that should be detonated to maintain this minimum ratio:

$$N > \frac{4}{S/R_a}, \qquad (19)$$

where N is the number of charges in a row.

Parallel Rows of Charges

The use of multiple rows will be warranted whenever a broad, shallow excavation is required and where a single row of explosives, of sufficient size to achieve the desired width, would result in unnecessary overexcavation of depth. At the current level of development of explosive excavation technology, the only experimentally verified alternative to a single row of charges is a set of two parallel rows, the two rows being detonated with a small time delay between them.[11]

The design procedure for two parallel rows is described below, and reference is made to Fig. 63 which shows a schematic cross section of the crater produced by the configuration. As shown in Fig. 63 the separation between rows should be 1.5 times the half-width of a crater produced by a single row of charges, and the width

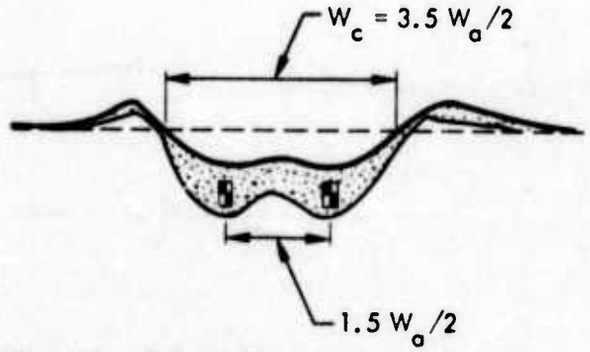


Fig. 63. Schematic cross section of crater produced by two parallel rows of charges.[1]

of the channel, W_c, at the preshot ground surface will be 3.5 times $W_a/2$. The bottom of the crater between the rows will be relatively flat with perhaps a slight mound of fallback.

The single-charge weight, Y_s (i.e., unenhanced row-charge weight), which will produce the required width, is determined as follows:

$$W_a/2 = R_{as} = \frac{W_c}{3.5}, \qquad (20)$$

where R_{as} is the crater radius corresponding to the single-charge weight of Y_s. With R_{as} known, Y_s can be read directly from the crater dimension charts. To use a smaller charge, Y_r, rather than Y_s, Eq. (14) gives the required charge spacing as:

$$\frac{S}{R_{ar}} = 1.4\left(\frac{Y_r}{Y_s}\right)^{0.6},$$

where R_{ar} is the crater radius corresponding to Y_r. The rows are separated by 1.5 R_{as} and the charges are buried at the optimum depth for Y_s.

The time delay between the rows is given by:

$$T_D = 150\ Y^{1/3}, \qquad \text{(msec)} \qquad (21)$$

where Y is the weight (in tons) of an individual charge in a row.

For example, when it is assumed that it is necessary to cut a channel 100 ft wide and 10 ft deep through soil, then from Fig. 55 it is found that a single row of 10-ton charges spaced 70 ft apart would meet the width objective, but the maximum depth of the channel would be almost 25 ft, a considerable overexcavation.

A design for the required excavation with a double row of charges would be performed as follows:

From Eq. (15):

$$W_a/2 = R_{as} = \frac{W_c}{3.5} = \frac{100}{3.5} = 28.5 \text{ ft},$$

and, therefore, $Y_s = 1.5$ tons (from Fig. 55). A double row of 1.5-ton charges, each with a charge spacing of 40 ft ($= 1.4 \times 28.5$) and separated by 43 ft ($= 1.5 \times 28.5$) would excavate the required channel with a maximum depth of approximately 13 ft.

The time delay between the rows is given by Eq. (21).

$$T_D = 150\ Y^{1/3} = 172 \text{ msec}.$$

The nearest commercially available delay cap is rated at 175 msec, and this rate would be acceptable.

In the example above, the single row would require 10 tons for 70 ft of channel, or 0.14 tons/ft; the double row requires only 3 tons/40 ft, or 0.075 tons/ft of channel, a reduction of approximately a factor of 2.

Interconnecting Craters

After a row of charges has been designed to excavate a channel, other considerations such as safety may limit the total weight that can be detonated at one time. It may be necessary to detonate the charges as a series of separate row charges rather than all at once. It is, therefore, desirable to connect smoothly one row crater to another. The designer should be aware of this requirement ahead of time in order to establish the limits of each detonation and to provide for the connection of the row craters. All charges in a long row may sometimes be emplaced before any portion of the row is fired, in which case provision for connections must be made prior to emplacement.

At the point of connection between two row craters, the distance between the end charges of the two rows should be the optimum crater radius of the larger of the two charges. This distance is obtained directly from the crater dimension charts. In the event that only one row of charges is emplaced at a time, the end charge of a follow-on connecting row should be placed beneath the end lip crest of the existing row crater. No adjustment to the charge weight is required to compensate for the lip material.

The technique of connecting one row to another is not well-developed. The connection of row craters may result in the deposit of a low mound of material in the first crater when the second one is formed. Remedial excavation with conventional methods will be necessary should the mound obstruct the channel.

Delayed Row-Charge Detonations

Some applications of explosive excavation may not appear to be feasible because of the proximity of buildings or other structures and the consequent risk of

ground-shock-induced damage. Should a first analysis indicate that a project is not feasible for this reason, consideration should be given to excavating by means of delayed charge firings. A limited amount of experimental data has been acquired on relatively large charges (1-ton) emplaced in rows and fired in some ordered sequence. These data indicate that if the delay between charges is great enough, the peak seismic motion generated is that due to the individual charges rather than the total.

For sequential detonations, a time delay on the order of milliseconds is introduced between charge detonations in the row. The detonation sequence begins at one end and progresses toward the other. Delay intervals can be achieved by using commercially available delay caps or delay connectors. The delay interval between charge detonations is computed according to the following equation:

$$T_D = 25\ Y^{1/3}, \qquad \text{(msec)} \quad (22)$$

where Y is the individual charge weight in tons, or the average charge weight in a row consisting of a mixture of charge sizes.

Delayed firings can be expected to result in a reduction of crater depth compared to a corresponding simultaneously detonated row. Widths are apparently not affected. This depth reduction can be compensated for in the design by increasing the charge weights 15% above those required for simultaneous firing. The ground shock from a delayed row can be predicted on the basis of the largest charge in the row, rather than the total weight of the explosive used in the row.

Underwater Cratering

Experience in underwater cratering is limited and considerable experimentation remains before quantitative design criteria can be fully established. For the present, the design procedures outlined for dry land cratering are assumed applicable; however, the water overburden must be accounted for. Because water is much less dense than earth materials and possesses no shear strength, less energy is required to displace water than is required to displace an equal volume of rock or soil. When cratering detonations under water are designed, the water overburden should be treated as an added layer of bottom material that is one-half as thick as the depth of water. This means that all crater depths and burial depths should be referenced to a hypothetical surface one-half the depth of water below the water surface.

The presence of water may drastically alter the process of crater formation because ejecta may be washed back into the crater. The extent of this washback will depend on the water depth and the material being cratered. Ejecta throwout ranges will also be less when water is present. Underwater craters will generally tend to be shallower and wider than a crater on dry land. It is strongly recommended that extensive experimentation be carried out prior to the designing of any large-scale underwater cratering project.

CHARGE GEOMETRIES AND STEMMING

Thus far, the charge has been implicitly assumed to act as a point source of energy. Many chemical explosive cratering experiments have been conducted using

spherical, center-detonated charges. There is evidence, however, that although a spherical charge is the most desirable shape, it is not a requirement for successful results. The degradation of cratering efficiency caused by minor departures from the spherical shape are not significant. The most recent experiments have used cylindrical charges. Cylindrical charges have been tested in cratering experiments with length-to-diameter ratios of almost 10 to 1, and the evidence suggests that ratios of 4 or 5 to 1 can be used without necessitating the increase of the charge weight to compensate for degradation of crater size. When the height-to-diameter ratio of the charge is 4 or more, the effective center of the charge is currently assumed to be at a point one-third of the charge length from the bottom, and depths of burial should be referred to this point.

All cratering charges should be stemmed to prevent loss of energy from premature venting through the emplacement hole. The emplacement hole should be filled with stemming material all the way to the surface. Stemming materials can be sand, gravel, or soil. If the explosive is a water-resistance slurry, the stemming material should be saturated with water prior to detonation. Water alone can be used for stemming if no other material is available.

SINGLE-CHARGE MOUNDING

A refined system for determining the depth of burial necessary to produce a mound of fractured material without throwout has not been developed to date. It is possible, however, through the use of the empirical scaling laws and associated figures for the referenced 1-ton TNT charge (Figs. 51 through 53) to determine the "threshold" depth for mounding. By threshold depth of burial is meant the depth of burial at which a specific charge would be as likely to produce a mound as to produce a small crater. The system functions simply—by extrapolating the curves to the point where the radius and depth curves appear to be zero. For the curves developed thus far, the information extrapolated corresponds to a scaled depth of burial of:

Dry rock	25 ft/(ton)$^{0.3}$
Dry soil	32 ft/(ton)$^{0.3}$
Clay shale	28 ft/(ton)$^{0.3}$

(Knowing this dob and the empirical scaling law, one can calculate the threshold depth of burial.)

Example—It is desired to use a 5-ton TNT charge to create a mound of broken rock that will be used as quarry rock. Use Fig. 51 or the value given above:

Threshold dob = 25 ft/(ton)$^{0.3}$

From Eq. (2), $P = (Y)^{0.3} = (5)^{0.3} = 1.62$

From Eq. (3), DOB = 1.62 (25) = 40.5 ft.

Experimental excavations have been performed using mounding charges in single and multiple rows. As with craters an enhancement factor appears to be involved with these tests. Research sufficient for the presentation of even general empirical relationships has not been accomplished. The effects of the media on mounding appear to be more pronounced than on cratering, because it is possible to fracture large amounts of rock in place, yet not be able to excavate all that is fractured. Additionally, the actual radius

and depth of broken rock are difficult to predict accurately.

SUMMARY

Empirical data have been used effectively to develop scaling laws that predict the size and shape of explosively produced craters to an acceptable degree of accuracy. This information is used as the basis for the design of explosive excavations. Single-charge craters are easily designed based on the required depth or radius and the design curves (Figs. 54 through 56). The design of row-charge craters is based on single-charge cratering and the enhancement of row-crater dimensions that occur when the charges are spaced close together. The enhancement phenomenon permits flexibility in the choice of individual charge weights to excavate linear craters. There are two design procedures that can be applied to a cut through varying terrain; the choice of which is to be used in a particular case must be based on a consideration of the operational factors. The two are very similar and differ only in the manner in which the concept of enhancement is applied. The constant-enhancement method will result in a row of charges with varying weights, while the constant-charge-weight method yields a row with all charges equal in weight. By coding these two design methods for a computer, it is possible to determine a wide range of feasible designs with various size charges and configurations. This coding allows greater flexibility in the final design selection. The technique of mounding, while still in the early development stages, appears to offer an attractive alternative in explosive excavation design.

Chapter 7 Optimization of Cost and Time for Single-Charge Cratering

SCOPE

A primary objective of the continuing research by EERL is to reduce the cost of emplacing large chemical explosive charges. Alternate methods to the expensive full-bore drilling of large-diameter emplacement holes are being investigated. Emplacement time for large explosive charges is another factor of tremendous importance, especially for military uses in the battlefield. One possible method of reducing emplacement cost and time for single-charge cratering is to use long cylindrical charges in relatively small-diameter emplacement holes. Another method is the relatively new technique developed for obtaining large-diameter charge cavities at depth without drilling a large-diameter hole from the surface. This technique, is known as an underreaming, and the desired results are obtained by a specially designed expanding tool (see Chapter 5).

Cost and time models have been developed by EERL for use in a computer code to generate cost and time curves. In order to analyze the cost and time for the two drilling methods, a parameter study has been performed on a variety of explosive types, charge sizes, drilling techniques, charge length-to-diameter ratios, and geologic media. This parameter study requires unit drilling cost relationships for full-bore drilling and underreamed charge cavities of the same diameter and charge length. With variations of these design parameters, the cheaper and faster drilling method is determined over a range of L/D values from 1 to 12.

COST AND TIME MODELS

Cost Optimization Model

The cost model is derived by first considering the two most important parameters of an explosive charge design, the depth of burial and the charge weight for a specified medium type. With these design parameters, charge geometry, and corresponding cost parameters, a cost model for a single-charge design can be derived, using cylindrical cavities.* For the full-bore drilling case depicted in Fig. 64, a cost equation for a single-charge design is:

$$PC = DOB\ (DPF) + \{L/2 \text{ or } L/3\} \times (DPF) + \frac{TTNT\ (DPT)}{E\ (VOLEFF)}, \quad (23)$$

or

$$PC = DOB\ (DPF) + \{1/2 \text{ or } 1/3\} \times (L/D)(D)(DPF) + \frac{TTNT\ (DPT)}{E\ (VOLEFF)}, \quad (24)$$

where:

VOLEFF, the cratering volume effectiveness, is given in Fig. 65.

When the charge length-to-diameter ratio, L/D, is 3 or less, the DOB is

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*A list of symbols used in this model may be found at the end of this chapter.

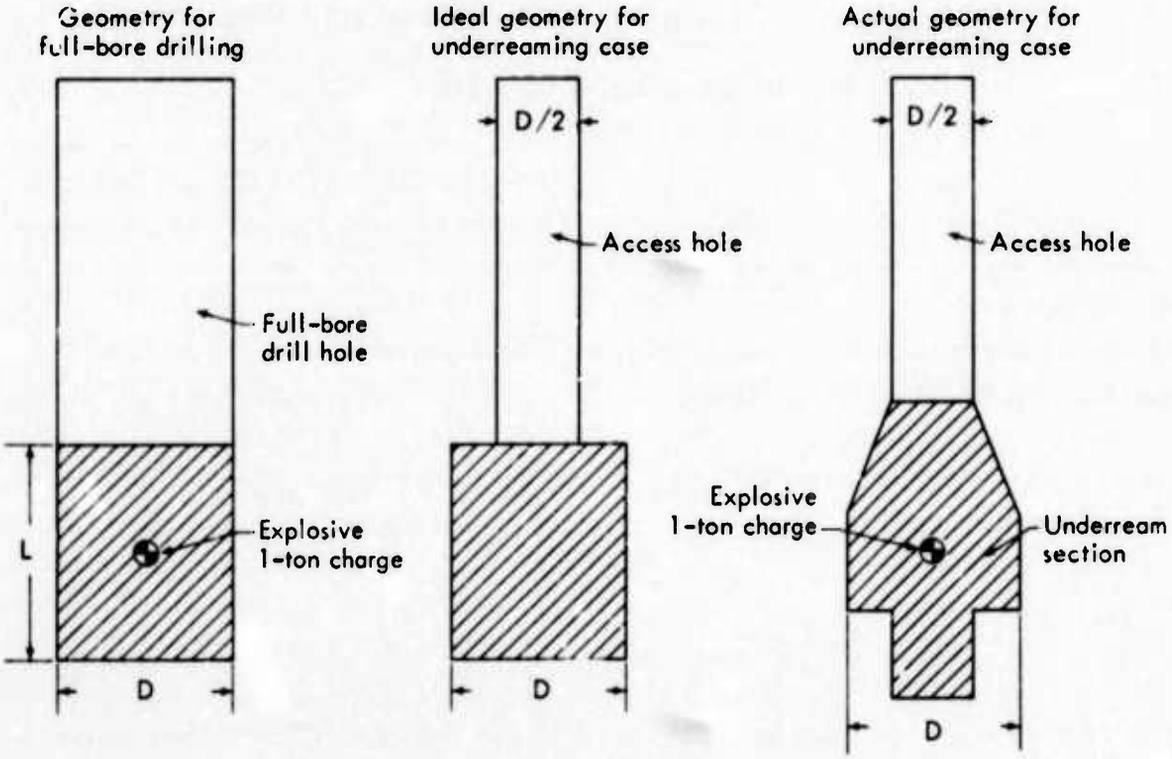
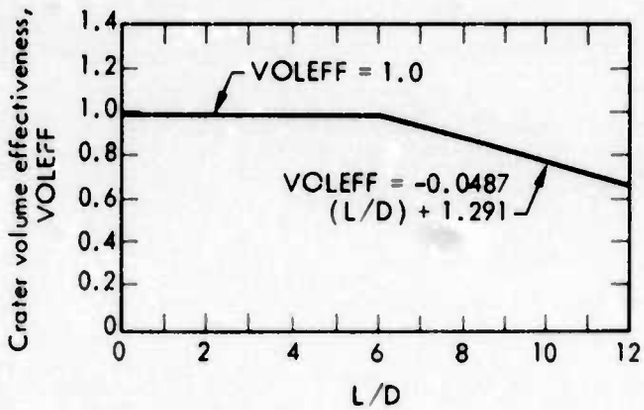


Fig. 64. Comparison of emplacement hole geometries.



For $L/D > 6$

$$\text{VOLEFF} = \frac{\text{Crater volume } (6 \leq L/D \leq 12)}{\text{Crater volume } (L/D) \leq 6)} = 0.0487\,(L/D) + 1.291$$

For $L/D < 6$

$$\text{VOLEFF} = \frac{\text{Crater volume } (1 \leq L/D < 6)}{\text{Crater volume } (L/D < 6)} = 1$$

Fig. 65. Cratering volume effectiveness degradation as a function of charge L/D.

assumed to be located at the center of the charge. However, when the L/D is greater than 3, the DOB is assumed to be located at the lower one-third of the cavity, which is L/3 from the bottom of the cavity. The cratering volume effectiveness, VOLEFF, shown in Fig. 65, illustrates that for an L/D more than 6, a greater quantity of explosive is needed to excavate the same crater. This volume effectiveness curve is an approximation of the degradation effect of the crater volume when L/D's greater than 6 are used.

Given a charge weight, TTNT, the required cavity volume is given as

$$CV = \frac{TTNT\ (2{,}000)}{\rho E\,(VOLEFF)} . \qquad (25)$$

For a given L/D, the volume effectiveness, and the diameter can easily be computed, after which the unit drilling cost for a given charge weight can be determined. Values of ρ and E for various explosives are given in Table 22, and VOLEFF is found by using the assumed L/D in Fig. 65. For a given L/D and computed cavity volume, the cavity diameter is calculated from:

$$D = \left[\frac{4\,(CV)}{\pi\,(L/D)}\right]^{1/3} . \qquad (26)$$

After the diameter has been computed, the unit drilling cost, DPF, is found for the diameter and media type from Table 23. With this approach (assuming L/D's and computing the cavity volumes and diameters for a given design), the full-bore cost—Eq.(23)—is used to compute costs for various L/D's of a given design. The results of this approach are presented later in computer-generated cost curves, with the lower value of the cost data shown in Table 23.

The alternate drilling method is underreaming, which produces a large-diameter

Table 22. Explosives data.[1]

Explosive composition	Specific gravity	Density ρ (lb/ft^3)	Cratering volume effectiveness[a] in relation to TNT, E	Required cavity volume (2,000/ρE) (ft^3/ton TNT or equivalent)	Cost, c (DPT) Delivered to site[b]	Cost, c (DPT) "Down-hole"[b,c]
ANFO[d]	0.93	58	1.0-1.1	31.35	40-200	120-320
2% Al-AN slurry[e] (blasting agent)[f]	1.30	81	1.0-1.2	20.57	60-260	140-380
AN slurry	1.40	87	1.0-1.2	19.16	200-400	280-520
8% Al-AN slurry (blasting agent)	1.33	83	1.2-1.4	17.12	160-360	240-480
20% Al-AN slurry (blasting agent)	1.20	75	1.5-1.7	15.69	260-540	340-660
35% Al-AN slurry (blasting agent)	1.50	94	1.6-1.8	11.82	300-700	380-820

[a]Based on small-scale tests in sand.

[b]Add $25 per 1,000 ton-miles for overland transportation from explosives manufacturing plant to site.

[c]Allows $80 to $120 per ton for emplacing, arming, and firing.

[d]Not suitable for use in wet environment.

[e]All slurries are suitable for use in a wet environment.

[f]Blasting agents have no high-explosive component.

Table 23. Estimated drilling costs.[1]

Material classification	Unconfined compressive strength (psi)	Rig type	Unit drilling costs for various hole sizes[a] (DPF) Diameter (in.) 12	24	36	48	60	72
High-strength rock (IV)	>16,000	Drive-in rotary[b]	15-30	25-40	35-80	50-100	70-140	95-185
Intermediate-strength rock (III)	4,000 to 16,000	Drive-in rotary[b]	8-20	15-30	25-45	35-60	45-80	70-140
Weak rock (II)	1,000 to 4,000	Drive-in rotary[b]	5-10	8-15	10-30	20-40	30-45	35-70
Common excavation (I)	<1,000	Drive-in auger[c]	3	—	4	5	6	7

[a]Data are based on drilling at least 300 linear ft of hole. Rig (24-hr day-rate), cutter, and drilling fluid costs are included, but no allowance is made for loss of circulation fluid. Rig day-rate includes equipment rental, profit, overhead, depreciation and crew salaries. Cost of mobilization, demobilization, site preparation, rig set up and rig tear down are not included.

[b]Mobilization and demobilization costs about $1.20/mi plus rig standby costs during travel time. Site preparation costs about $1,500 to $2,500.

[c]Mobilization and demobilization costs about $0.70/mi plus rig standby costs during travel time. Site preparation costs considered negligible.

cavity at a specified depth, suitable for emplacing the required explosives.[45] For the purposes of this study, the diameter of the access hole is assumed to be one-half the charge cavity diameter. This is the smallest possible ratio of access hole diameter to cavity diameter attainable with presently available equipment.[45] A schematic representation of an underreamed cavity is compared to a full-bore cavity in Fig. 64. This idealization of the underreamed cavity geometry is necessary to simplify the drilling cost relationship of the cavity. Based upon this simplification of the underreamed cavity geometry, the unit cost of producing a cavity of a fixed diameter is assumed to be a constant multiple, K_C, of the unit cost of full-bore drilling the same cavity. The drilling cost relationship is expressed as:

$$DPFU = K_C\ (DPF)\ , \qquad (27)$$

in which K_C is assumed to vary from 1 to 3 and DPFU is the unit drilling cost to underream a charge cavity.

The cost equation for underreaming a single-charge design is written as:

$$PC = DPFA\ (DOB) - \{L/2 \text{ or } 2L/3\} \times DPFA + (L)\ (DPFU) + \frac{TTNT\ (DPT)}{E\ (VOLEFF)}\ , \qquad (28)$$

or

$$PC = DPFA\ (DOB) - \{1/2 \text{ or } 2/3\} \times (L/D)\ (D)\ (DPFA) + (L/D)\ (D)\ (DPFU) + \frac{TTNT\ (DPT)}{E\ (VOLEFF)}\ . \qquad (29)$$

The negative L/2 or 2L/3 results from the terms (DOB - L/2) or (DOB - 2L/3), which are the actual lengths of the access hole depending upon the location of depth of burial.

The values of DOB, L/D, and the volume effectiveness degradation used in the full-bore case are also relevant to the underreaming case. By assuming

various L/D's and computing the cavity diameters for each, the underreaming cost, Eq. (8), is used to compute costs for various L/D's of a given design. The results of this approach are represented later in computer-generated cost curves* with the lower value of the cost data shown in Table 2?.

Time Optimization Model

A time model is derived with an approach similar to the derivation of the cost model. Again the two most important parameters of an explosive charge design, depth of burial and the charge weight for a specified media type,[42]† must be considered. With these design parameters, charge geometry, and corresponding drilling and emplacement time parameters, a time model can be derived for a single-charge design assuming cylindrical cavities. For the full-bore drilling case, the time equation is written as

$$PT = DOB\,(HPF) + \{L/2 \text{ or } L/3\} \times (HPF) + \frac{TTNT\,(HPT)}{E\,(VOLEFF)}, \quad (30)$$

or

$$PT = DOB\,(HPF) + \{1/2 \text{ or } 1/3\} \times (L/D)\,(D)\,(HPF) + \frac{TTNT\,(HPT)}{E\,(VOLEFF)}. \quad (31)$$

Note the similarity of this equation to the full-bore cost equation with HPF and HPT replacing DPF and DPT, respectively, to convert from time to cost. Downhole emplacement time, HPT, is a function of the explosive type considered and the method of emplacing the explosive. For the parameter study presented in the next section, estimates of HPT are of little significance in the total time. The values of the DOB, L/D, and VOLEFF used in the cost model, are relevant in the time model.

Equation (31) is used for computing the time, assuming various L/D's of a given design. The results of these calculations are displayed in computer-generated time curves.* The unit underreaming time of a cavity diameter is assumed to be a constant multiple, K_T, the drilling time factor. This time factor is expressed as:

$$HPFU = K_T\,(HPF), \quad (32)$$

in which the unit drilling time is for the cavity diameter, D. After the cavity diameter has been computed, HPF, a function of the cavity diameter and geologic medium, can also be determined. The access diameter is assumed to be one-half the cavity diameter. Drilling time rates have been produced from penetration rates for numerous sizes in various geologic media (described in Table 23) as follows:[36]

Media I	HPF = 0.002D (D, in.)
Media II	HPF = 0.003D
Media III	HPF = 0.005D
Media IV	HPF = 0.009D

*See Fig. 66.

†A condensed version of this report appears as Chapter 4.

*See Fig. 67.

The underreaming time model is written as:

$$PT = DOB\ (HPFA) - \{L/2 \text{ or } 2L/3\} \times HPFA + L\ (HPFU) + \frac{TTNT\ (HPT)}{E\ (VOLEFF)}, \quad (33)$$

or

$$PT = DOB\ (HPF) - \{1/2 \text{ or } 2/3\} \times (L/D)\ (D)\ (HPFA) + (L/D)\ (D)\ (HPFU) + \frac{TTNT\ (HPT)}{E\ (VOLEFF)}. \quad (34)$$

The procedures for developing the previous cost and time curves are also applied to the underreaming model for time.

PARAMETER STUDY

The purpose of this study was to analyze the cost and time of using full-bore drilling and underreaming techniques for L/D s in the range of 1 to 12. In the parameter study time curves were computed for a variety of explosive design parameters. The following questions were to be answered in this parameter study:

(1) What is the effect of changing the unit drilling cost relationship, K_C, in Eq. (6), for underreaming as compared to full-bore drilling a charge cavity of the same diameter?

(2) What effects do geologic medium, explosive type, charge size, and L/D have on the relative costs of underreaming and full-bore drilling a charge cavity?

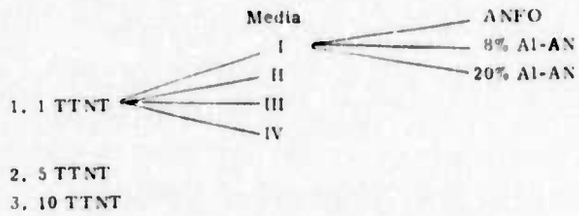
(3) Does underreaming lead to significant cost reduction for large values of L/D with cratering degration incorporated?

(4) For what L/D ranges, considering numerous K_C values for various explosive designs and explosive types, are each of the two drilling techniques most economical?

(5) For what L/D ranges, using assigned values[67] of K_T = 1.0 for Media I and II and K_T = 1.5 for Media III and IV, is underreaming faster than full-bore drilling?

(6) What is the effect of cratering degradation on the cost and time of drilling cavities with large L/D's?

The following parameters were considered for each of three cases as shown below:



Thus, each of the four media (Table 23) were considered for the explosive types listed above. For each medium the charge sizes 1, 5, and 10 tons TNT equivalent were used in the study. To evaluate these charge sizes, the optimum DOB's in Table 24 were used.

K_C is considered as 1.0, 1.5, 2.0, 2.5, and 3.0 for each of the designs and explosive types. The minimum value of K_C is 1; however, past experiences and limited available underreaming cost data show that K_C is greater than 1.[68]

RESULTS FOR SINGLE-CHARGE CRATERING

Results of Cost Optimization

Several observations can be made about the cost curves generated in the

Table 24. Optimum depth of burial for media types and charge sizes used in parameter study.

Media	DOB (ft)		
	1 ton, TNT	5 tons, TNT	10 tons, TNT
I	20	32	40
II	19	31	38
III	18	29	36
IV	18	29	36

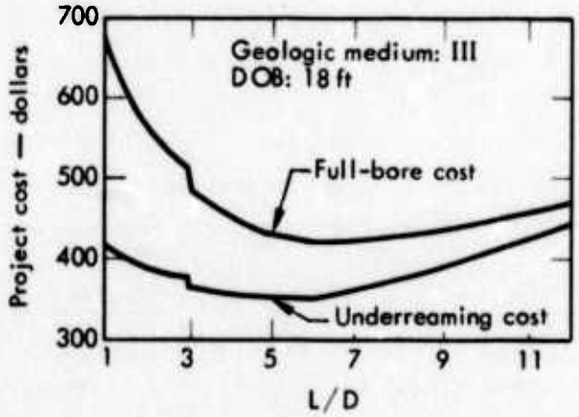
parameter study. With reference to the cost curves for ANFO in Fig. 66., the decrease in cost at the L/D ratio of 3 should be noted. This discontinuity appears in all the underreaming and full-bore drilling cost curves. In the derivation of the cost models it was assumed that if the L/D is greater than 3, the DOB is located at the lower third point of the charge cavity rather than at the center. Shifting the DOB to the lower third of the charge cavity results in an emplacement hole of L/6 less in total depth. As a result of the shifting of the DOB, a shorter emplacement hole is required resulting in a lower drilling cost. This rapid down shift of the cost curve occurs at the L/D ratio of 3.0.

The discontinuity is apparent in cost curves for both drilling methods; however, the cost decrease is more significant in the full-bore case. In the underreaming case the access diameter was assumed to be one half the charge cavity diameter; whereas, in the full-bore case the access hole and charge cavity have the same diameter. For the full-bore case a greater decrease in volume of material to drill at the L/D of 3 is apparent; therefore, the cost decrease is greater for full-bore drilling than in the underreaming case.

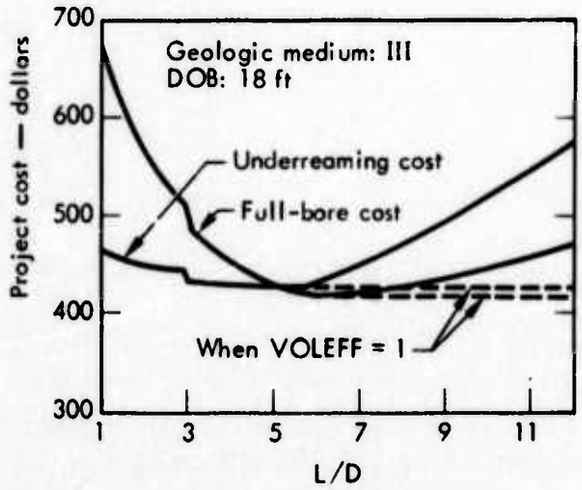
When the cratering volume effectiveness remains 1 for values of L/D greater than 6, cost curves level off as the L/D increases above 6 (see Fig. 66b). However, when crater degradation is assumed, the costs increase near the L/D at which the degradation commences. In this parameter study, the degradation factor given by Fig. 65 was used for L/D's greater than 6.

Several cost curves were generated varying the cost relationship, K_C, between the two drilling methods. L/D values at which underreaming is cheaper than full-bore drilling were determined. These results are given in Table 25 for ANFO, 8% Al-AN slurry, and 20% Al-AN slurry. Generally, for K_C values greater than 1 and a given medium and explosive type, the value of the L/D below which underreaming is cheaper increases as the charge size increases. As the value of K_C increases, the value of L/D below which underreaming is cheaper decreases, as shown in Fig. 66.

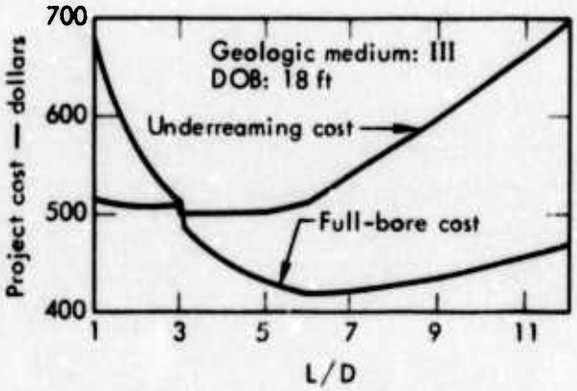
For a given explosive type, charge size, and K_C, the underreaming and full-bore cost differential for specific L/D's increases for Media I to IV, as shown in Fig. 67. These cost differentials at an L/D of 5 for the four media show that underreaming costs are greater than the full-bore costs. The increase in cost differentials for the higher strength media is a result of the increased cost of underreaming the charge cavity. For a specific medium, charge size, and explosive type, the cost differential of underreaming and full-bore drilling increases as the L/D increases to the right of the intersection of the curves.



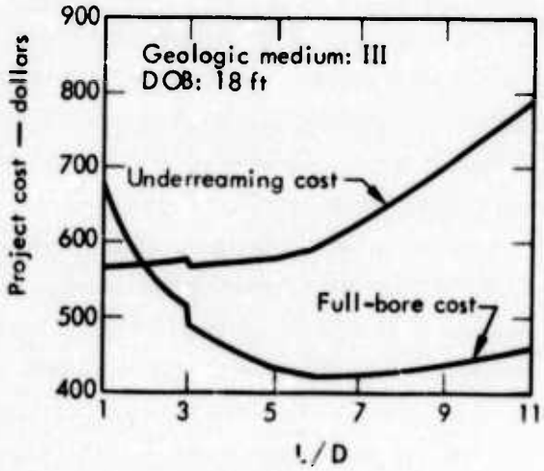
(a) K_C = 1.0.



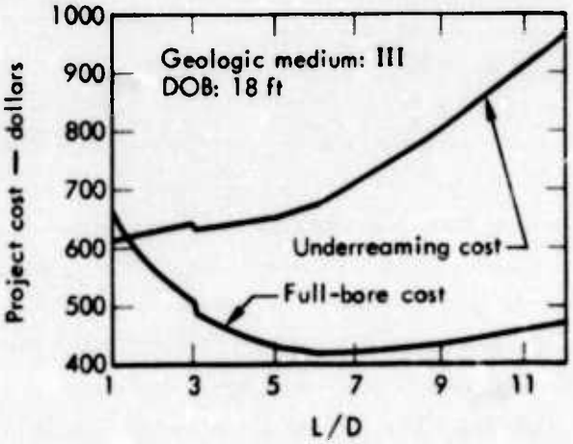
(b) K_C = 1.5.



(c) K_C = 2.0.



(d) K_C = 2.5.



(e) K_C = 3.0.

Fig. 66. Lower bound cost curves (underreaming and full-bore drilling) for Medium III using a 1-ton TNT-equivalent charge of ANFO.

As the L/D increases the cavity diameter decreases, resulting in a decreased unit drilling cost (DPF) for underreaming; however, the cavity length increases, resulting in increased cost to drill the underreamed cavity (see Fig. 67). In

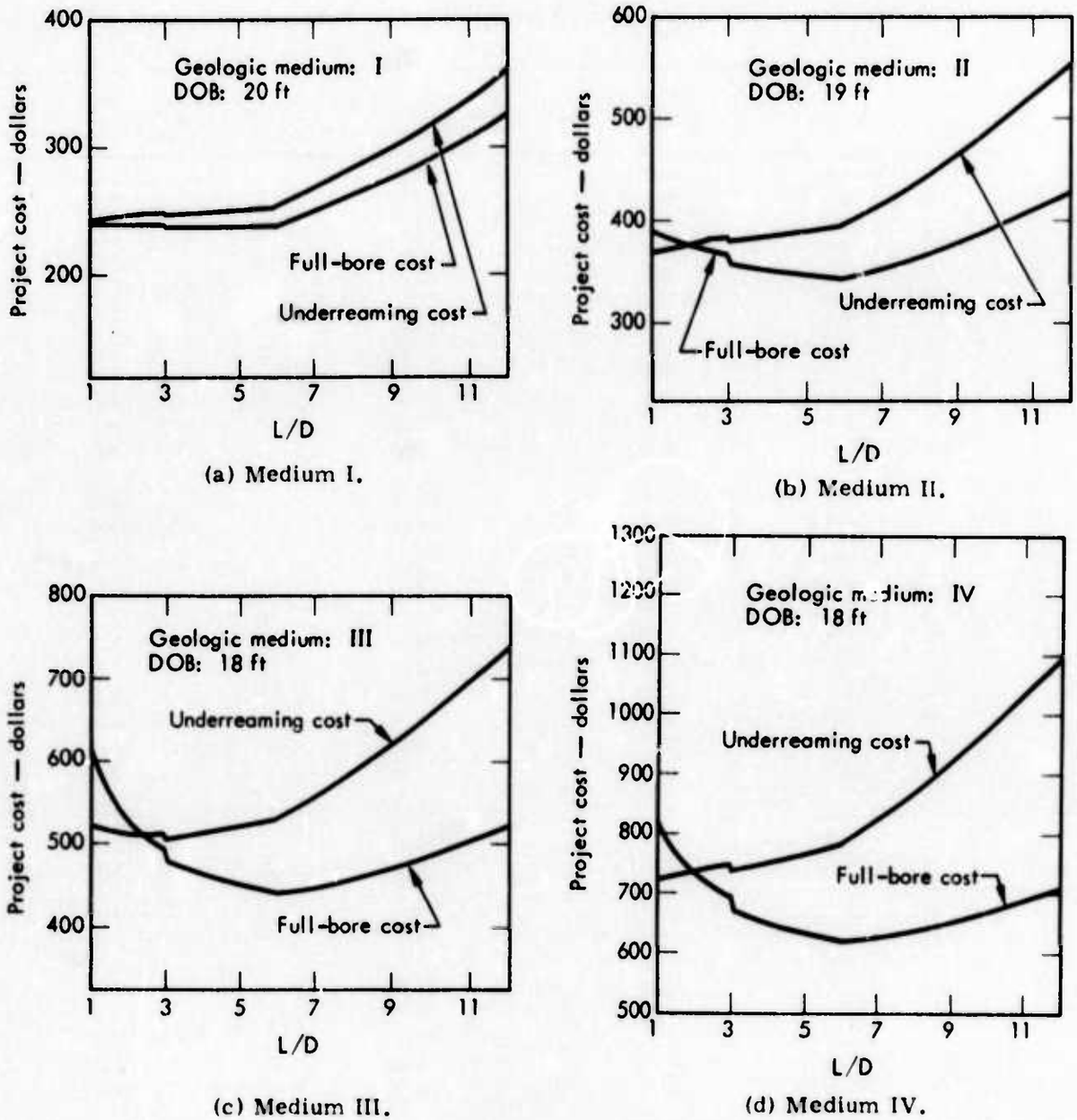


Fig. 67. Lower bound cost curves (underreaming and full-bore drilling) for 1-ton TNT-equivalent charge of 20% Al-AN (K_C = 2.5).

other words, as the L/D increases, an optimum L/D is reached at which the cost of lengthening the underreamed cavity and using smaller diameters is more significant than the cost of using shorter charge cavities and larger diameters. The underreamed cavity becomes longer as the L/D increases; therefore, higher K_C values for a specific medium, charge size, and explosive result in greater cost differentials.

Also, from Table 25, for various media, charge sizes, and K_C values, the ranges of the L/D at which underreaming is cheaper are generally greater for the more expensive explosives, 8% Al-AN and 20% Al-AN, than for the lesser expensive ANFO. ANFO has a greater

Table 25. Comparison of L/D ranges at which underreaming and full-bore drilling are cheaper.

K_C	Media	TTNT	L/D at intersection of full-bore and underreaming cost curves[a]		
			ANFO	8% Al-AN slurry	20% Al-AN slurry
1.5	I	1.0	2.2	3.0	3.0
	I	5.0	3.5	3.3	3.3
	I	10.0	4.2	4.2	4.7
	II	1.0	4.7	6.9	7.2
	II	5.0	5.4	5.6	5.9
	II	10.0	6.3	6.5	6.3
	III	1.0	5.3	7.1	7.3
	III	5.0	5.7	6.9	7.1
	III	10.0	5.8	7.1	7.3
	IV	1.0	4.6	6.6	6.9
	IV	5.0	4.6	5.9	6.0
	IV	10.0	5.3	5.8	5.7
2.0	I	1.0	1.2	1.3	1.0
	I	5.0	2.1	2.1	2.0
	I	10.0	2.5	2.7	2.7
	II	1.0	2.2	3.1	3.2
	II	5.0	3.4	3.3	3.3
	II	10.0	3.8	4.2	4.3
	III	1.0	3.1	3.5	3.7
	III	5.0	3.1	3.8	3.9
	III	10.0	3.1	3.9	4.0
	IV	1.0	2.5	3.1	3.2
	IV	5.0	3.1	3.1	3.1
	IV	10.0	3.1	3.4	3.5
2.5	I	1.0	1.0	1.0	1.0
	I	5.0	1.4	1.5	1.5
	I	10.0	1.6	1.8	1.8
	II	1.0	1.4	1.8	1.9
	II	5.0	2.6	2.7	2.6
	II	10.0	2.8	3.1	3.1
	III	1.0	2.0	2.4	2.5
	III	5.0	2.0	2.7	2.8
	III	10.0	2.3	2.7	2.8
	IV	1.0	1.5	2.0	2.0
	IV	5.0	2.0	2.2	2.2
	IV	10.0	2.2	2.5	2.5
3.0	I	1.0	1.0	1.0	1.0
	I	5.0	1.0	1.2	1.2
	I	10.0	1.2	1.3	1.3
	II	1.0	1.3	1.2	1.3
	II	5.0	1.9	2.1	2.1
	II	10.0	2.0	2.5	2.5
	III	1.0	1.4	1.8	1.8
	III	5.0	1.5	1.9	1.9
	III	10.0	1.7	1.8	1.8
	IV	1.0	1.0	1.4	1.4
	IV	5.0	1.5	1.6	1.6
	IV	10.0	1.7	1.8	1.8

[a]For L/D's less than the values given, underreaming is cheaper; for larger L/D's, full-bore drilling is cheaper (L/D range = 1 to 12).

required cavity volume per ton TNT equivalent than the other two explosives, resulting in a larger required charge cavity volume for an equivalent size charge. The higher underreaming costs for ANFO result in greater cost differentials between the underreaming and full-bore drilling cost curves (compare Figs. 66d and 67c).

One general conclusion that can be made from examining all the cost curves presented in this chapter is that full-bore drilling is the cheapest drilling method at the L/D in which the cratering volume effectiveness decreases less than 1. The results in Table 25 are based upon the lower drilling and emplacement costs given in Tables 22 and 23. Whenever these drilling and emplacement costs are changed, the results in Table 25 should also change slightly; however, the concepts should remain the same.

Results of Time Optimization

Similar to the cost curves, a discontinuity in time at an L/D of 3 is apparent in all the time curves for underreaming and full-bore drilling. As discussed in the previous section of this chapter, shifting the DOB at an L/D of 3 results in a shorter emplacement hole, which in turn decreases the drilling time. This downward shift in the time curves comes at an L/D of 3.0. At an L/D of 3.1 the curves level off to a small slope as shown in Fig. 68a.

In the full-bore case, this discontinuity or downward shift in time is greater than the downward shift for the underreaming time of the same design and explosive type. The reason for this difference in the time curves is the same as for the difference in the cost curves.

As previously discussed, the drilling time for underreaming a charge cavity as compared with the time for full-bore drilling the same cavity is described by a drilling time factor, K_T. In Media I and II the drilling time factor is $K_T = 1.0$, and in Media III and IV this drilling time factor is $K_T = 1.5$ (Ref. 67). When it is assumed that $K_T = 1.0$ for Media I and II, underreaming is always faster. This study used lower time estimates because the investigation was concerned only with the relationships of the time curves for the two drilling methods. The results of the time curves are given in Table 26 for ANFO, 8% Al-AN, and 20% Al-AN.

As the charge size increases for a given medium type (III or IV) and explosive type, the value of L/D below which underreaming is faster decreases for full-bore drilling.

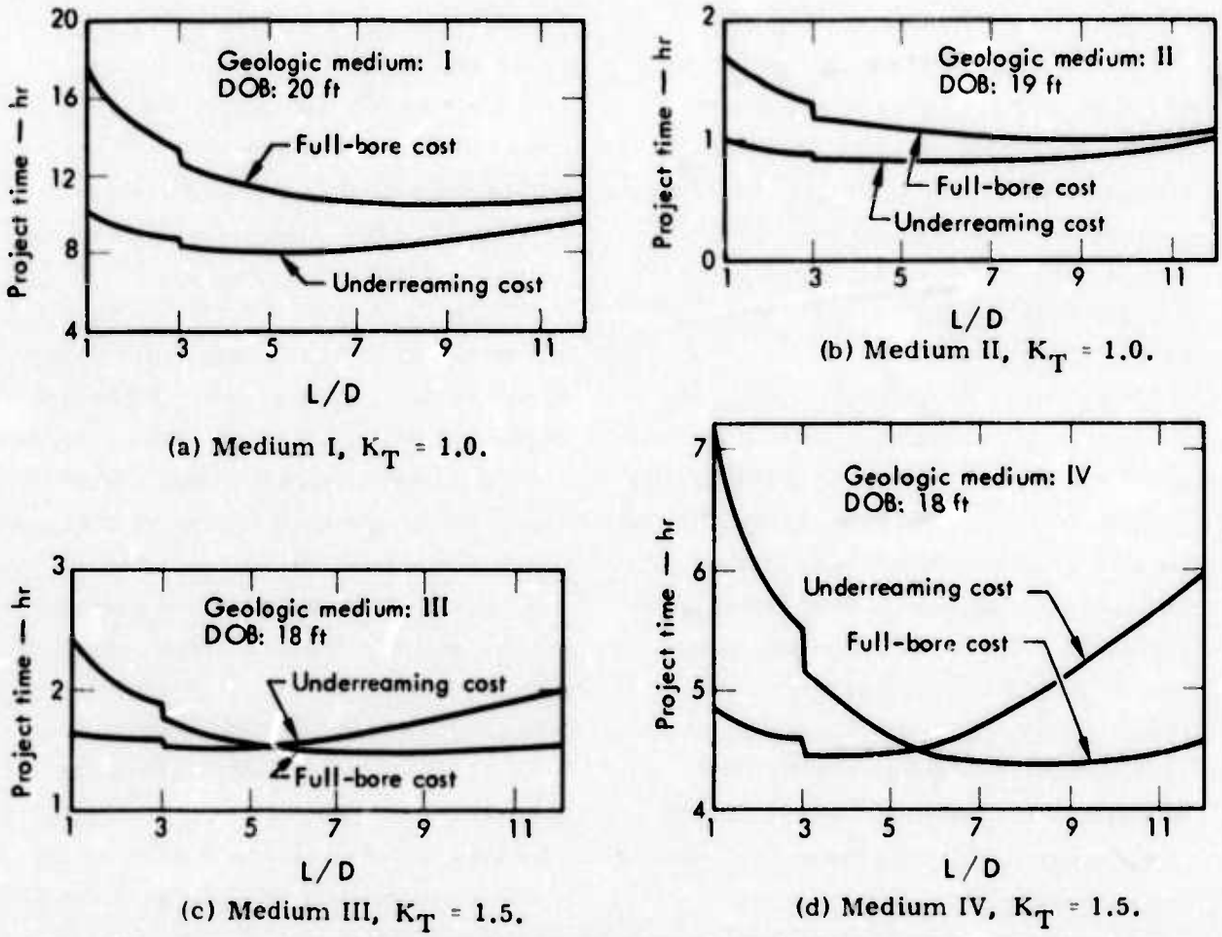
For Media III and IV and various charge sizes, the value of L/D below which underreaming is faster increases from ANFO to 8% Al-AN to 20% Al-AN because the required equivalent cavity volume (ft^3/ton TNT) decreases, resulting in less drilling times. As the required cavity volume increases for these explosives, the drilling times increase accordingly.

EXAMPLE PROBLEM: TANK BARRIER

The following problem is an example of a typical situation in which the method developed can be utilized:

Problem

A single-charge in dry rock with a depth of 20 ft and a radius of 40 ft is



(a) Medium I, K_T = 1.0.

(b) Medium II, K_T = 1.0.

(c) Medium III, K_T = 1.5.

(d) Medium IV, K_T = 1.5.

Fig. 68. Lower bound time curves (underreaming and full-bore drilling) for 1-ton TNT-equivalent charge of ANFO.

needed as a tank barrier. Two explosives, 8% Al-AN slurry and ANFO, are readily available and the properties of these explosives make either explosive acceptable for this project. Battalion Operations has available underreaming and full-bore drill rigs for cavity diameters ranging from 24 to 60 in.

Required

The requirement is to excavate the tank barrier in the minimum time possible.

Solution

First, the optimum depth of burial and charge size (tons TNT) must be computed. Figure 54 with the required crater dimensions, shows that a charge of 10 tons TNT equivalent buried at 36 ft will excavate the desired crater. Given the depth of burial, charge size, and medium type for the desired crater, one enters the computer-generated curves to find the L/D ratio, drilling method, and explosive type to minimize the time.

To save time the engineer on the battlefield would have available a manual including computer-generated curves for various explosive types and designs to aid him in selecting the optimum design parameters.

Table 26. Comparison of L/D ranges at which underreaming and full-bore drilling are faster.

Explosive type	Media	TTNT	K_T	L/D at intersection of full-bore and underreaming time curves[a]
ANFO	I	1.0, 5.0, 10.0	1.0	—[b]
	II	1.0, 5.0, 10.0	1.0	—[b]
	III	1.0	1.5	5.6
	III	5.0	1.5	5.2
	III	10.0	1.5	5.0
	IV	1.0	1.5	5.6
	IV	5.0	1.5	5.2
	IV	10.0	1.5	5.0
8% Al-AN	I	1.0, 5.0, 10.0	1.0	—[b]
	II	1.0, 5.0, 10.0	1.0	—[b]
	III	1.0	1.5	7.0
	III	5.0	1.5	6.9
	III	10.0	1.5	6.7
	IV	1.0	1.5	7.2
	IV	5.0	1.5	6.8
	IV	10.0	1.5	6.7
20% Al-AN	I	1.0, 5.0, 10.0	1.0	—[b]
	II	1.0, 5.0, 10.0	1.0	—[b]
	III	1.0	1.5	7.5
	III	5.0	1.5	7.1
	III	10.0	1.5	7.0
	IV	1.0	1.5	7.7
	IV	5.0	1.5	7.1
	IV	10.0	1.5	6.9

[a]For L/D's less than the values given, undrreaming is cheaper; for larger L/D's full-bore drilling is cheaper (L/D range = 1 to 12).

[b]Underreaming is always faster because K_T = 1.0 and the curves never intersect.

Solution for 8% Al-AN

Drilling method: Underreaming

L/D = 3.50

D = 2.64 ft

Drilling time: 5.33 hr as a lower estimate (refer to Fig. 69).

Solution for ANFO

Drilling method (full-bore drilling):

L/D = 7.5

D = 3.86 ft

Drilling time: 6.85 hr as a lower estimate (refer to Fig. 70).

The above information obtained from the time curves, Figs. 69 and 70, leads to the conclusion that the optimum solution would be the 8% Al-AN slurry using the underreaming technique with the charge emplaced in a 44-in. (3.64-ft) charge cavity with an L/D ratio of 3.5.

SUMMARY AND CONCLUSIONS

Mathematical models have been developed to optimize the cost and time for emplacement of chemical explosives for single-charge cratering. These models have been used in a computer code

to provide a fast method of determining the cost and the time of explosive emplacement. The models have been used in a parameter study to analyze the cost and time relationships of full-bore drilling and underreaming. Resultant cost and time curves can be used to select optimum L/D's for various designs and explosive types.

The L/D ranges at which each of the two drilling methods is cheaper and faster for various single-charge cratering designs was determined. Underreaming proves to be cheaper than full-bore drilling at smaller L/D's depending upon the design, explosive type, and K_C considered. An approximation of K_C should

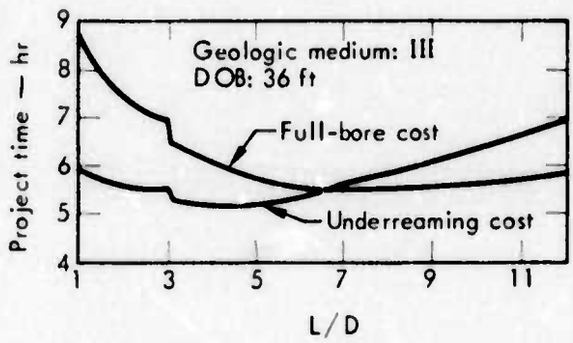


Fig. 69. Example problem time curves for 10-ton TNT-equivalent charge of 8% Al-AN.

be in the range of 2.0 to 3.0, depending upon the geologic medium. Large L/D's in underreaming produce no significant cost reduction regardless of whether

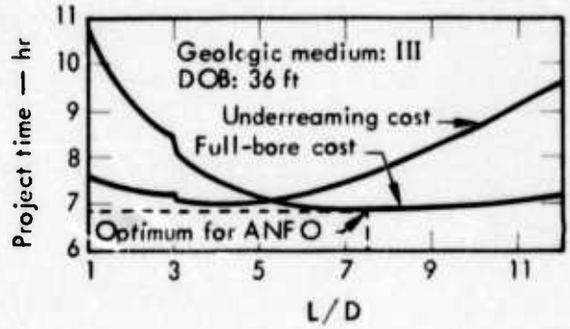


Fig. 70. Example problem time curves for 10-ton TNT-equivalent of ANFO.

crater volume effectiveness is considered or not. In fact, when this degradation effect is considered, the cost curves increase rapidly for the L/D at which the degradation commences.

One general conclusion is that full-bore drilling is the cheapest drilling method at the L/D ratio in which the cratering degradation commences.

The drilling time relationship, K_T, of underreaming a charge cavity as compared to full-bore drilling the same cavity was assumed to be equal to 1.0 for Media I and II and equal to 1.5 for Media III and IV. In other words, for Media I and II, underreaming is faster than full-bore drilling for the complete L/D range of 1 to 12. However, in Media III and IV, the ranges of the L/D at which underreaming is faster varies with the explosive type and charge size.

LIST OF SYMBOLS

CV	=	cavity volume, ft^3
D	=	cavity diameter, ft
DOB	=	depth of burial, ft
DPF	=	full-bore unit drilling cost, DPF
DPFA	=	unit drilling cost of the access hole diameter (D/2) for underreaming
DPFU	=	unit drilling cost to underream a charge cavity
DPT	=	downhole emplacement cost of the explosive, DPT
E	=	cratering volume effectiveness in relation to TNT
HPF	=	full-bore unit drilling time, hr/ft
HPFA	=	unit drilling time of the access hole diameter (D/2) for underreaming
HPFU	=	unit drilling time to underream a charge cavity
HPT	=	downhole emplacement time of explosive, hr/ton
K_C	=	drilling cost factor
K_T	=	drilling time factor
L	=	length of charge cavity, ft
L/D	=	charge length-to-diameter ratio
PC	=	project cost, $
PT	=	project time, hr
TTNT	=	charge weight, tons of TNT equivalent
VOLEFF	=	volume effectiveness as a function of the L/D
ρ	=	density of the explosive, lb/ft^3

Chapter 8 Conclusions

This report has presented a discussion of some of the factors affecting the potential employment of explosive excavation techniques in barrier and denial operations, target destruction, and military construction (e.g., bunker emplacement). The discussions have shown that commercial bulk explosives have demonstrated their potential applicability, effectiveness, and versatility for many military applications. The potential application of bulk explosives to target destruction is uncertain because there have been no experiments upon which to judge their effectiveness. Even those applications that have been demonstrated, such as excavation, would be difficult for the military engineer to implement. There follows a discussion of the three basic reasons for this problem with possible corrective action:

1. Procedures have not been established to permit the engineer to determine which explosive to use for a given application. Selection of the best explosive for a given project depends on many factors that are not fully understood. It is possible to develop a bulk explosive for military use that could be in the form of a single specified chemical composition or in the form of several inert ingredients, which, when mixed according to various "recipes," would yield a "family" of explosives, each suited to a particular mission or for use in a particular medium. It is not yet known whether it is better to have a single multipurpose explosive that performs well in all media, or to have a family of explosives that could be matched to the media or mission. In addition, the best packaging, or on- or off-site mixing configuration, must be determined.

2. A majority of the projects suggested for explosive excavation with bulk explosives require the capability to emplace rapidly large quantities of explosives in deep, large-diameter holes. The Army does not presently have the capability of drilling deep, large-diameter holes. Thus, the military engineer is presently unable to use explosives for large excavations. If the Army is to take advantage of the capabilities of commercial bulk explosives within the near future, an accelerated program of testing and evaluating portable drill rigs similar to those available commercially must be implemented. The drill rigs must be capable of creating in rock an emplacement hole up to 24 in. in diameter and 100 ft in depth.

3. An extensive prototype experimentation program must be conducted before many of the larger explosive excavation and target destruction projects can be approved and design procedures accepted as a part of Army practice. Despite the experience that has been gained from using commercial bulk explosives for large civil works excavation projects, it is still necessary to prove that these techniques are applicable to the battlefield.

Realizing the potential capabilities of commercial bulk explosives and the Army's present limitations for exploiting this potential, the Waterways Experiment Station has proposed a research plan that would investigate the feasibility of using

bulk explosives as a military engineering tool. The proposal is for Military Engineering Applications for Commercial Explosives (MEACE) and is directed toward evaluating the potential of commercial bulk explosives for future military applications such as barriers, target destruction, and military construction. The proposal addresses explosive and emplacement rig selection, cratering, and the associated effects, as well as application experimentation. Final action on the proposal by the Office of the Chief of Engineers (OCE) will depend on the results of an ongoing study into the anticipated applications of explosives in a theater of operations, the present and proposed explosive systems available for these applications, and a comparison of the relative merits of the present and proposed explosive systems.

EERL is continuing cratering and obstacle effectiveness experiments to maintain its expertise in the military and civil construction aspects of explosive excavation. Research in the area of new explosive systems and equipment to support them is also being conducted by other agencies. For example, Picatinny Arsenal is currently managing a development contract for a portable mixing unit for slurry explosives.

There is much research that remains to be done in this field. The Army can keep abreast of the progress with the MEACE Program. This report has merely introduced the concepts. Many important questions must be answered if the Army is to take advantage of the very great potential savings in time and manpower that are possible now with commercially available explosives and equipment.

Appendix A

Barrier Demonstration Projects

INTRODUCTION

Three recent field experiments and one planned for the near future are discussed in this appendix to illustrate some applications of slurry explosives for battlefield use. Project Diamond Ore, a nuclear simulation program, has demonstrated the capability of slurry explosives to produce tactically significant craters. Projects Tank Trap and Armor Obstacle I have evaluated craters produced by nuclear and chemical explosive charges as tactical vehicular obstacles. Project Armor Obstacle II is designed to reevaluate the Army's deliberate road crater and prechamber obstacle designs and to evaluate alternate designs with slurry explosives. The Diamond Ore program will continue to evaluate new explosive configurations to produce craters with large explosive charges. These projects mark the initial effort in evaluating slurry explosives for the military theater of operations.

DIAMOND ORE

General

Diamond Ore is the name for a series of explosive cratering experiments initiated by EERL in 1971 to simulate nuclear cratering detonations employing various stemming* materials. The initial tests, conducted in the fall of 1971 in Bearpaw clay shale at Fort Peck, Montana, consisted of three 10-ton detonations. One was at 6 m (20 ft) and the other two at a depth of burial of 41 ft. A photograph of the crater produced is shown in Fig. A1.

Diamond Ore Phase I

Diamond Ore Phase I was designed to determine the chemical explosive to be used in the experimental program and to define its detonation characteristics. This phase also developed the data acquisition and analysis procedures for the project. The explosive selected for Phase IIA was an aluminized, ammonium nitrate slurry. The explosive characteristics of this slurry are found in Table A1.

Table A1. Explosive data for Diamond Ore Phase IIA.

Type: aluminized ammonium nitrate slurry w/10% sand[a]
% Aluminum: 30%
Density: 1.64 g/cm^3
Total energy: 1,411 cal/g
a. Shock energy: 531 cal/g
b. Bubble energy: 880 cal/g
Detonation velocity 4,700 m/sec
Detonation pressure 80 kbar
Booster: Composition C-4 (10 lb)
TNT equivalence: 1.4
Cost: 64¢/lb (downhole)
Sensitivity to No. 8 detonator cap = No reaction
Effect of H_2O: No effect experience
Effect of storage: No known effect
Shipping class: Oxidizer

*Materials utilized to fill the emplacement hole above the explosive charge.

[a]Sand used for fallout simulation experiment.

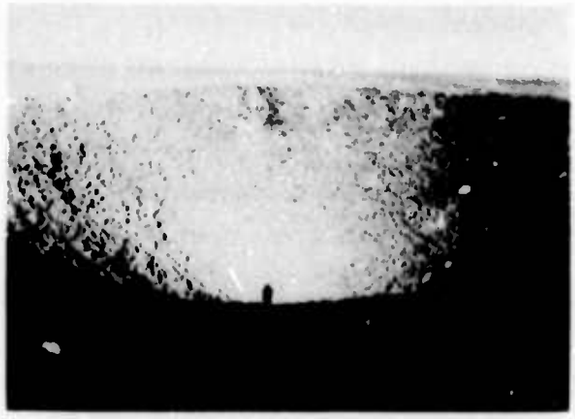


Fig. A1. Crater produced during Phase IIA of Diamond Ore.

Diamond Ore Phase IIA

Phase IIA consisted of a series of three cratering detonations. Two of these detonations were at a depth of burial of 41 ft with one detonation fully stemmed and one unstemmed (air-stemmed). The third detonation was at a 6-m (20-ft) depth of burial and was fully stemmed. Each detonation consisted of 10 tons of slurry explosive, which gave an energy equivalence of about 14 tons of TNT. The access holes were 36 ins. in diameter and the charge cavities were mined spheres. Charges were center-detonated. The stemming utilized was a specially prepared grout that closely matched the properties of the in situ material. Stemming was keyed into the access hole to prevent premature venting of the explosive. Each detonation was heavily instrumented to include surface motion measurement, close-in ground motion and earth stress measurements, uniformity of detonation measurements, seismic motion, close-in airblast, and stemming pressure measurements. Postshot surveys were conducted to determine crater size and shape, ejecta limits and composition, and missile ranges and densities. Features of the resulting craters are listed in Table A2.

Summary

The Diamond Ore nuclear simulation project is investigating the use of chemical explosives to model nuclear detonations. In conjunction with the testing program, computer simulation work is being carried out to refine the cratering codes to reduce the need for future large-scale testing. Additionally, Diamond Ore field testing has been scheduled to continue this study of explosively produced craters in various media and in various stemming configurations.

Table A2. Preliminary crater measurements for Project Diamond Ore Phase IIA.

Dimension[a]	Unstemmed 41 ft (A-1)	Stemmed 41 ft (A-2)	Stemmed 20 ft (A-3)
Apparent radius (R_a)	63.5	62.7	60.3
Apparent depth (D_a)	24.5	22.6	31.2
Lip height (H_{al})	11.5	12.9	8.4
Lip crest radius (R_{al})	83	81	74.5
Apparent volume (V_a)	4,900	4,200	5,500
Maximum missile range (R_{max})	1,040	1,110	2,180

[a]Linear dimensions, ft; volume, yd^3.

PROJECT TANK TRAP

General

Project Tank Trap was a joint research effort of the U.S. Army Engineer Nuclear Cratering Group (now EERL) and the Land Locomotion Laboratory (LLL) of the Army Tank Automotive Center.[3]

The purpose of Project Tank Trap was to determine the capability of selected tactical vehicles to traverse craters typical of those that could be produced with Atomic Demolition Munitions (ADM). The vehicles included in the test program were the M-60 tank, M-113 Armored Personnel Carrier, and an articulated two-unit general purpose vehicle called the Polecat. The characteristics of these vehicles are found in Table A3. The testing program did not involve the use of engineering effort (earthmoving, bridging, surface stabilization, etc.) to assist the entry and exit of the vehicles. Project Tank Trap was conducted in existing craters that had been created with chemical explosives at the Nevada Test Site to model possible nuclear tests. The Jangle-U crater was produced by nuclear explosion. Features of these craters are given in Table A4.

The original plan included the Danny Boy basalt crater (0.42 kt emplaced at a depth of burial of 110 ft) and the Pre-Buggy Row H alluvium crater (13-1,000 lb chemical explosives detonated in a row at near optimum depth of burial and at varying spacing). The vehicle testing results in the Pre-Schooner Bravo crater indicated it would be unsafe to attempt entry and exit of the Danny Boy crater and it was believed that little additional information concerning trafficability of rock craters could be obtained by so doing. After observing vehicle testing operations in the Scooter and Jangle-U craters, it was decided that the Pre-Buggy Row H crater (average radius, 23 ft; average depth, 13 ft) was too small to obtain meaningful test results. Vehicle testing, therefore, was limited to the Scooter, Jangle-U and Pre-Schooner Bravo craters. These crater sites were selected for the vehicle trafficability study for the following reasons:

(1) The craters are representative of those that could be produced by Atomic Demolition Munitions detonated at very shallow depths of burial (Jangle-U) and near optimum depths of burial (Scooter and Pre-Schooner Bravo).

(2) The media in which the craters were produced represent a wide range of materials (dry soil to hard rock); therefore, the test results could be used to predict the performance of tactical vehicles in several different types of material.

Table A3. Characteristics of test vehicles.

Vehicle	Gross weight (lb)	Vehicle track		Vehicle		
		Width (in.)	Ground contact length (in.)	Length (in.)	Width (in.)	Height (in.)
Polecat	9,100	20	156-1/2	317	81	98
M-113 APC	19,755	15	105	191-1/2	105-3/4	79-1/2
M-60 tank	95,300	28	166.7	273-1/2	143	126

Table A4. Description of test craters.

Code name	Cratering medium	Yield (kt)	Depth of burial (ft)	Depth of burial ($ft/kt^{1/3.4}$)	Radius R_a, (ft)	Depth, D_a, (ft)	Lip height (ft)	Slope angle (%)
Scooter	Alluvium	0.5/HE	125	(153)	155	75	12.5	30-35
Jangle-U	Alluvium	1.2/Nuc	17	(16)	130	53	8	20-32
Pre-Schooner Bravo	Basalt	0.02/HE	51	(160)	49	25	9	27-30

Scooter Crater

The Scooter crater is typical of craters produced by explosions at or near optimum depth in desert alluvium. Desert alluvium may be described as a loose silt-sand-gravel mixture with densities ranging from 1.5 to 1.7 g/cm^3. Particle sizes range from cobbles of 1 to 2 ft maximum dimensions through gravel and sand to very fine rock flour.

The Scooter crater was formed in October 1960, by the detonation of 500 tons of high explosive at a depth of 125 ft (scaled depth of 153 $ft/kt^{1/3.4}$). The Scooter crater has an average radius, at the original ground surface, of 155 ft and depth of 75 ft. The lip averages 12.5 ft in height. The crater slopes average 30 to 35 deg, flattening slightly near the top and bottom of the crater (Fig. A2). As is typical of craters produced by detonations at or near optimum depth, the slopes and bottom of the crater are composed of fallback material. This material is unsorted, very loose and lies at its angle of deposition. The cohesion of the fallback material as determined from field soils measurements ranges from 0 to 0.15 $tons/ft^2$ and the angle of internal friction (ϕ) varies from 30 1/2 to 33 1/2 deg.

Since the formation of the Scooter crater in 1960, the wind has blown some of the fine sand and silt sized particles in the fallback material down the crater slope and deposited them at the base of the slope. Although this action has altered the properties of the slope material somewhat, the fallback material in the region of the crater rim remains quite loose and moves freely when disturbed. No crust or hardened surface has formed on the crater slopes (see Fig. A2).

The Polecat was able to exit this crater without assistance.

The M-113 APC required winch assistance to exit the slopes of the Scooter crater. The force required to assist in exiting ranged from 3,200 to 4,000 lb. The rim angle of the M-113 relative to the slope increased significantly while the vehicle was exiting under its own power.

As shown in Fig. A3, the M-60 tank required assistance to exit the Scooter crater. Assisting forces of 2,000 and 20,000 lb were recorded during the two exiting operations. The slope requiring 20,000 lb of force appeared to have a considerable layer of loose gravel on the surface. This was not the case on the slope requiring 2,000 lb assistance. No other observations can be offered to account for the significant variation in required assistance on slopes with essentially the same inclination.

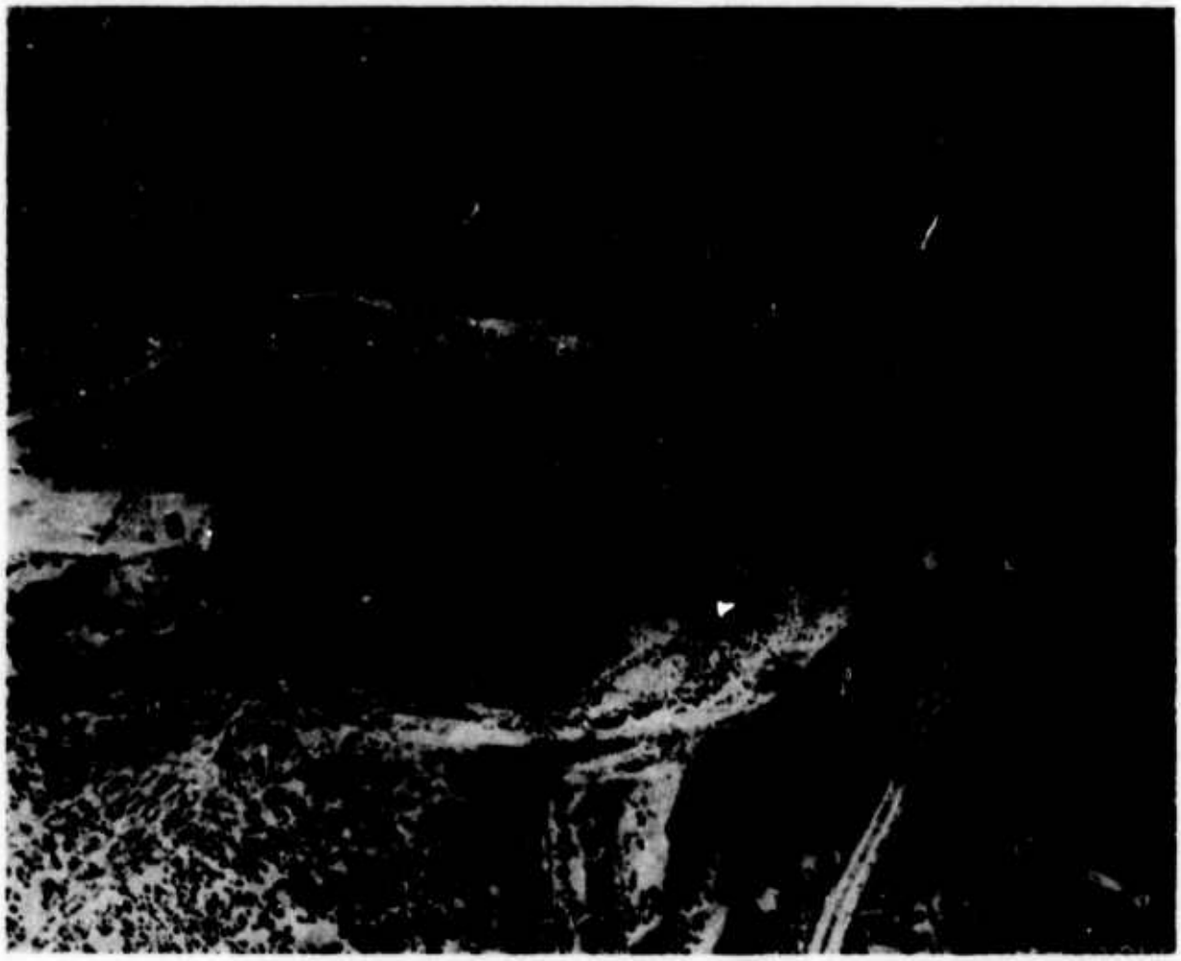


Fig. A2. Aerial view of Scooter crater.

Jangle-U Crater

The Jangle-U Crater is typical of craters produced in alluvium by explosions at very shallow depths. The alluvial material is essentially the same as that of the Scooter crater.

Jangle-U was formed in November 1951 by the detonation of a 1.2-kt nuclear device at a depth of 17 ft (scaled depth of .6 ft/kt$^{1/3.4}$). The Jangle-U crater has an average radius at the original ground surface of 136 ft and a depth of 55 ft. The lip averages 8 ft in height. The crater slopes, averaging 20 to 32 deg, are uneven and are characterized by irregular benching (Fig. A4). Field soils measurements indicated that the cohesion of the fallback material on the crater slopes ranges from 0 to 0.1 tons/ft^2 and the angle of internal friction (ϕ) ranges from 32 1/2 to 36 deg. Exposures of the true crater are evident near the rim of the crater, and fines are concentrated on the benched areas of the crater slope and at the bottom of the crater.

Since the formation of the Jangle-U crater in 1951, the fallback material has been blown by the wind from the steeper parts of the crater slope and deposited on the benched areas and in the crater bottom. As a result of weathering and

the removal of the loose sand and silt sized particles from the slope surfaces, a crust has formed on the crater slopes.

The crust on the crater slopes initially assisted the vehicles in exiting the crater, but the crust was easily broken and could not support the full vehicle weight. The fines that were deposited in the bottom of the crater constituted a soft area that reduced vehicle mobility.

The Polecat experienced no difficulty in exiting the Jangle-U crater slopes (see Fig. A5).

The M-113 APC was tested on the same routes as the Polecat. It required winch assistance only on the northeast slope. Winch assistance (400 lb) was required to negotiate the bench that occurs at approximately half the distance up the slope. The M-113 APC tended to labor in the fines on the bench. This phenomenon was also noticeable on the other route, but the vehicle was finally capable of exiting without winch assistance. The M-113 APC exhibited a high trim angle with reference to the slope angle while exiting the Jangle-U crater.

Fig. A3. M-60 tank exiting Scooter crater with assistance from M-88 VTR.

Fig.A4. Entry and exit slope of Jangle-U crater.

The M-60 tank was capable of exiting the southwest slope without winch assistance. On the northwest slope it required a winch assistance of 4,000 lb to negotiate the fines in the bench area. No assistance was required throughout the remainder of the slope. On the northeast slope the M-60 required an assisting force of 16,000 lb during the first quarter of the slope. After clearing the bench area the M-60 required no winch assistance.

Pre-Schooner Bravo Crater

The Pre-Schooner Bravo crater is typical of craters produced by detonations

Fig. A5. Polecat exiting Jangle-U crater.

at or near optimum depth in hard rock. The Pre-Schooner Bravo crater is located on Buckboard Mesa in Area 18 of NTS. The soil overburden on Buckboard Mesa consists of residual and aeolian sands and silts with large quantities of gravel and boulder-sized fragments of vesicular basalt. This overburden averages 3 ft in depth.

The Pre-Schooner Bravo crater was formed in February 1964 by the explosion of 40,000 lb of high explosive at a depth of 51 ft (scaled depth of 160 ft/kt$^{1/3.4}$). The crater has an average radius at the original ground surface of 49 ft and an average depth of 25.5 ft. The lip height averages 9 ft. Crater slopes vary from 27 to 30 deg with a general steepening in the lip area to 36 to 38 deg. The fallback material consists of angular basalt fragments ranging from sand size to fragments with maximum dimensions of 12 ft. This fallback material results in very rough and irregular crater slopes.

There has been no alteration of the Pre-Schooner Bravo crater since its formation. The slopes appear to be at the angle of repose for the fallback material.

Because of the large, angular, basalt rock that characterized the material in the ejecta field surrounding the Pre-Schooner Bravo crater, as well as the fallback material on the crater slopes, it was necessary to make a careful reconnaissance of possible routes of entry and exit through the crater prior to vehicle testing. The large boulders located on the crater slopes (Fig. A6) limited the choice of entry and exit profiles to two or three at the most.

The Polecat was not tested in the crater because it was believed that its sheet metal underbody would be severely

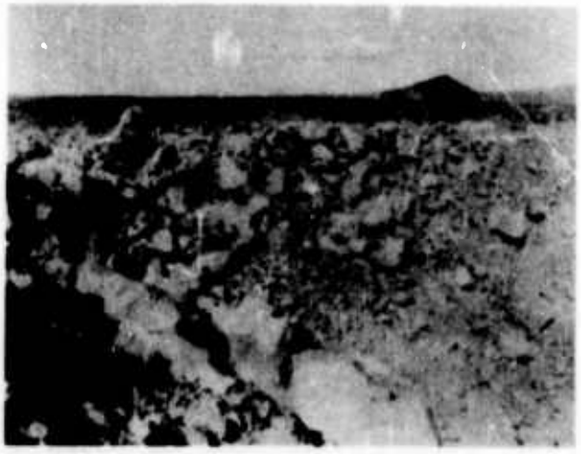


Fig. A6. Slope of Pre-Schooner Bravo crater showing entry profile used for M-113 and M-60 tanks.

damaged by the jagged edges of the basalt ejecta and fallback material.

The M-113 APC negotiated the lip of the crater under its own power with no difficulty and was then lowered into the crater by a M-88 VTR. After the APC reached the bottom of the crater, the driver attempted to maneuver around the large boulders in order to get into a position to exit but was unable to do so using the power of the vehicle only (Fig. A7). At this point in the testing sequence it was determined that the APC could not maneuver into a favorable exit position without assistance. By moving rocks ranging from 4 to 12 in. in diameter by hand and maneuvering the APC by means of the M-88 VTR winch it was possible to get the vehicle in a position to exit the crater. While attempting to exit the crater under its own power, the M-113 APC displaced much of the rock surrounding the tracks. As a result, loose rock became lodged in the track assembly. The M-113 APC was not able to exit the slope without assistance, and the M-88 VTR was used to winch the vehicle out of the crater.

Fig. A7. M-113 Armored Personnel Carrier immobilized in Pre-Schooner Bravo crater.

The M-60 tank negotiated the crater lip and was lowered into the crater in essentially the same manner as the M-113 APC. The large boulders in the bottom of the crater prevented the M-60 from maneuvering into a favorable position for exit; and, consequently, rock was moved by hand to fill in the voids between the boulders so that the M-60 tank could maneuver without becoming lodged on one or several of the boulders. The M-60 tank attempted to exit the crater without assistance but the tracks simply rotated in place and the vehicle became embedded in the rock (see Fig. 5 of this report). As the M-60 tank was attempting to exit the crater with assistance from the M-88 VTR winch, the right track jumped off the drive sprocket when rocks became lodged in the track assembly system. Efforts to get the track back on the sprocket while the tank was in the crater were unsuccessful, and the vehicle was winched from the crater with only one track providing traction. Inspection of the right track and sprocket showed that the track system was damaged considerably.

Summary

Based on the results of Project Tank Trap it is concluded that:

1. Craters formed in dry soil by the detonation of explosives at the surface and at very shallow depths of burial (approximately 20 ft/kt$^{1/3.4}$) do not present significant obstacles to tracked tactical vehicles. This is primarily due to the fact that these craters have flat slopes and are relatively shallow.

2. Craters formed at or near optimum depth of burial (160 ft/kt$^{1/3.4}$) in dry soil are a trafficability obstacle to tactical tracked vehicles. The slopes of this type of crater are greater than the slopes of very shallow depth of burial craters.

3. Craters formed in hard rock, such as basalt, cannot be negotiated by tracked vehicles without major modification of the crater and/or assistance by heavy duty equipment either mobile or fixed. The random arrangement and size (several inches to several feet) of the rocks ejected by the detonation cause the primary trafficability problem. Visual observation of craters formed in basalt by detonation of explosives in the ADM-yield range indicate that detonations at scaled depths of burial considerably less shallow than the Pre-Schooner Bravo depth of burial would also constitute formidable barriers to tracked vehicles.

4. A two-unit articulated vehicle such as the Polecat is able to negotiate crater slopes in dry soil much more readily than tactical tracked vehicles such as the M-113 APC or the M-60 tank. This is due primarily to the fact that the center of gravity of the two-unit vehicle lies forward of the rear unit (there is essentially no weight transfer between units of the articulated vehicle); and, consequently, the ground pressures at the rear of the second unit are considerably less than the rear pressures of a single-unit tracked vehicle.

ARMOR OBSTACLE I

General

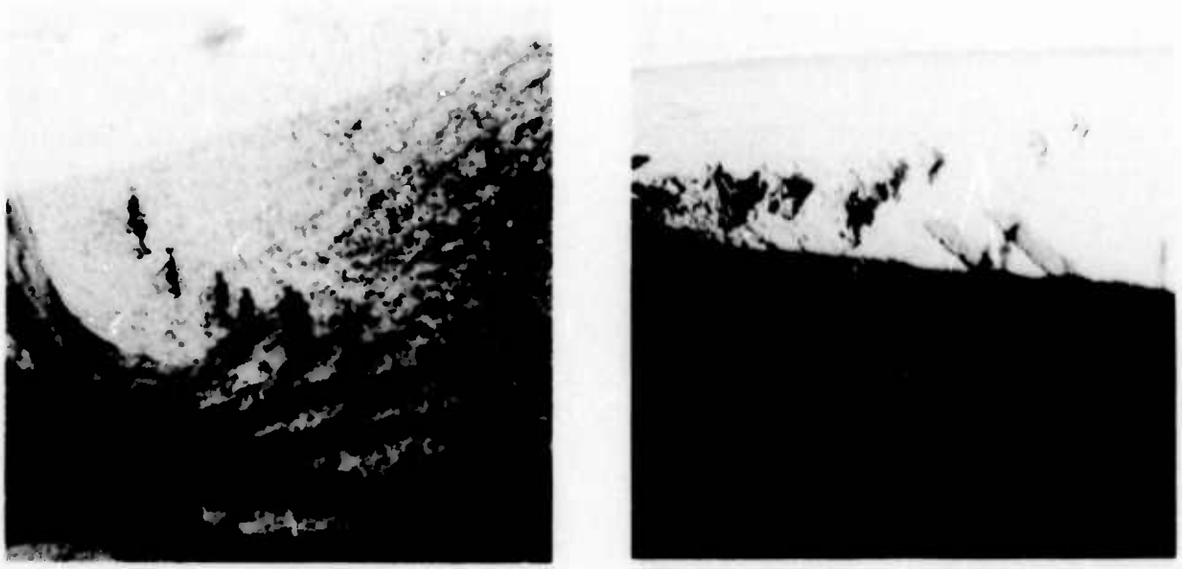
Armor Obstacle I was conducted in conjunction with Diamond Ore IIA during the fall of 1971 at Fort Peck, Montana. The three craters produced by the Diamond Ore detonations were tested to determine their value as obstacles to vehicles and men. Test equipment included an infantry squad, one M59 APC, and an M48 tank. The medium involved was Bearpaw clay shale.

Project Description

The dimensions of the craters produced in Diamond Ore IIA are listed in Table A2. The infantry squad experienced little difficulty in crossing the craters, their average traversing time being 1 min. Although these craters will only slow down the foot soldier's advance, the soldier, while crossing the crater, becomes a more vulnerable target (see Fig. A8).

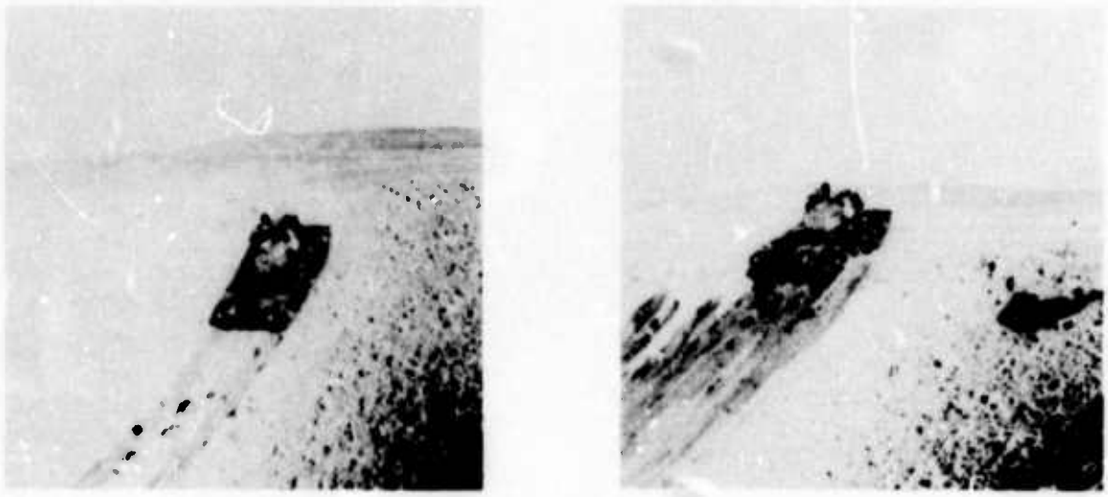
Unfortunately, the APC developed mechanical difficulties that limited its testing to the traversing of the crater lips. These it climbed with little difficulty, however. The tank also experienced little difficulty in climbing the crater lips. Inside the craters, however, there was a different story. The tank was only able to exit from Crater IIA (Fig. A9) under its own power. This exit required 13 attempts and took 15 min. During this time, its vulnerability to antitank weapons was high. In Craters IIA-2 and IIA-3, the tank was able only to traverse two-thirds of the way up the exit slope before stalling. After 20 min of unsuccessful exit attempts in each crater, the tank was winched out (Fig. A10).

After the tank was unable to negotiate Crater IIA-3, a dozer worked in the crater preparing an exit ramp. To simulate actual field conditions, the dozer pushed a ramp into the crater and back-bladed the enemy side of the crater to



(a) (b)

Fig. A8. Infantry squad crossing and exiting Crater IIA-2.



(a) (b)

Fig. A9. Tank exiting Crater IIA-1.

construct the exit ramp (Fig. 11). It took the dozer 41 min to complete the entrance and exit channels. The tank was finally able to exit after 12 attempts spending 9 min in the crater (Fig. A12).

Summary

The craters formed by an aluminized slurry explosive in Diamond Ore Phase IIA proved to be excellent obstacles when evaluated in Armor Obstacle I. Not only was it extremely difficult for the tank to exit these craters, but remedial dozer action proved only marginally effective and quite time consuming, not counting dozer mobilization time. The crater widths in all three cases exceeded the capacity of any known, rapidly installable

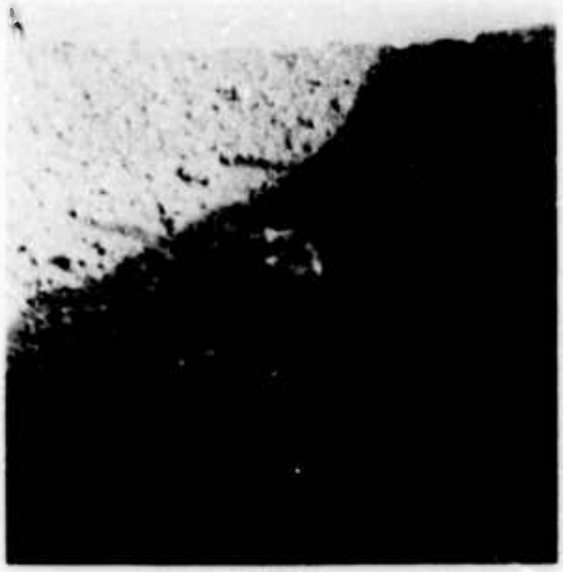
fixed bridging. Strategically placed, the obstacles demonstrated in Project Armor Obstacle I would have impeded the enemy's advance and created significant tactical or logistical delays.

Fig. A10. Signs of unsuccessful attempt by tank to exit unassisted.

(a)

(b)

Fig. A11. Bulldozer preparing exit channel in Crater IIA-3.



(a)

(b)

Fig. A12. Tank exiting Crater IIA-3 after bulldozer work.

FAUNA FIRST SERIES II

General

Project Fauna Frolicus II, conducted in the fall of 1974 at Fort [illegible], Maryland, was a series of structure damaging tests designed to evaluate and to improve upon the existing mathematical models used for predicting structural response and damage to buildings and other structures. Project Fauna Frolicus II is the first of a series of experiments planned to investigate the loading, design, and construction criteria for building shapes exposed to blasting loadings to improve existing models. Testing targets will be demolished, and concrete, and masonry

military engineering construction techniques.

Project Scope and Design

This project consists of three series of blasting experiments. A total of nine detonations were accomplished with explosive charges ranging in weight from 20 to 4,000 lb, placed in cavities from 6 to 30 ft in depth. The major elements of the project are noted in Table A8.

Each detonation in series I and II was instrumented to measure airblast and seismic effects. Following the detonations, each target was measured by standard surveying techniques to determine the damage. The damage effectiveness of

Table A8. Major Elements of Project Fauna Frolicus II

Series	Designation[a]	Description
I	FF: 1	Fuse 8, charge size: 20 lb (10 lb) at depth: 1,250 lb (10 lb) at depth; airblast/seismic detection, instrument
	FF: 2	Fuse 8, charge size: 20 lb at depth: 1,000 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
II	FF: 1[b]	Fuse 8, charge size: 100 lb (10 lb) at depth: 4000 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
	FF: 2[b]	Fuse 8, charge size: 100 lb at depth: 4000 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
	FF: 3[b]	Fuse 8, charge size: 100 lb at depth: 4000 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
	FF: 4	Fuse 8, charge size: 400 lb (10 lb) at depth: 100 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
	FF: 5	Fuse 8, charge size: 400 lb (10 lb) at depth: 50 lb (depth: 10 lb) at depth; airblast/seismic detection, instrument
III	AB ADDL 1[b]	Single charge: 10 lb (simulation bulkhead cavity)
	AB ADDL 2[b]	Single charge: 10 lb (simulation bulkhead and door)

[a] With the exception of FF: 1 and FF: 5, all shots are standard.

[b] Structure not changed in FF: 1, 2, and 3 and AB ADDL 1 and 2 were conducted and metal chamber changed.

the craters produced in these two series, as well as those produced in the Diamond Ore IIB series were determined by each crater's ability to stop or to impede the movement of an M-60 tank.

Series I

The PC series was designed to compare the cratering ability, in terms of crater dimensions and obstacle effectiveness, of equivalent quantities of TNT and an aluminized slurry in a specific charge configuration. In order to evaluate adequately some of the handling requirements associated with employing TNT and slurry explosives in specified charge configurations, the time, equipment, and manpower required to load a three-hole row was recorded. In addition, the effort required to unload one cavity of TNT and slurry was also documented to simulate the receipt of a mission abort order.

Series II

The DRC series was designed to investigate the potential advantages of using slurry explosives to create a DRC and to compare the cratering ability, in terms of crater dimensions and obstacle effectiveness, of small, equal, and equivalent quantities of ammonium nitrate canisters and an aluminized slurry.

DRC-1, -2, and -3. The Army's standard DRC design was followed, employing M-3 shaped charges to create the required emplacement cavities. Handling requirements for loading shallow cavities in terms of time, manpower, and equipment for a five-hole row were noted, recorded, and evaluated. Assuming the cancellation of a road crater mission, the effort required to unload one cavity of slurry was documented.

DRC-4 and -5. These detonations were designed to test the feasibility of using slurry explosives to produce a DRC with smaller charge weights of explosives or larger charge weights and greater hole spacings. An 8-in. diameter drill was utilized to construct the cavities for these detonations. The craters produced were compared with DRC-1, -2, and -3 in terms of crater dimensions and obstacle effectiveness.

Series III

The AN-ANFO series (ammonium nitrate – ammonium nitrate fuel oil) was designed to compare the crater dimensions, from a canister of straight ammonium nitrate with a similar canister of prilled ammonium nitrate and fuel oil. Two 5-ft deep cavities were constructed with an 8-in. diameter drill. After the detonations, the small craters were measured.

Figures A13 through A15 illustrate the designs of two of the three experimental series. The designs for Series I and II (Figs. A13 and A14) were based upon existing Army practices and the design procedures presented in Ref. 1. The experiments in Series I (Fig. A13) investigated the possibility of reducing the total weight of the slurry explosive by 30% and producing an obstacle equivalent in crater dimensions and effectiveness. A total of 1,320 lb of TNT and 1,000 lb of slurry was employed to produce two craters of equivalent size. Series II was also intended to verify the 1.3 cratering effectiveness factor for a 10% aluminized slurry on a smaller scale.

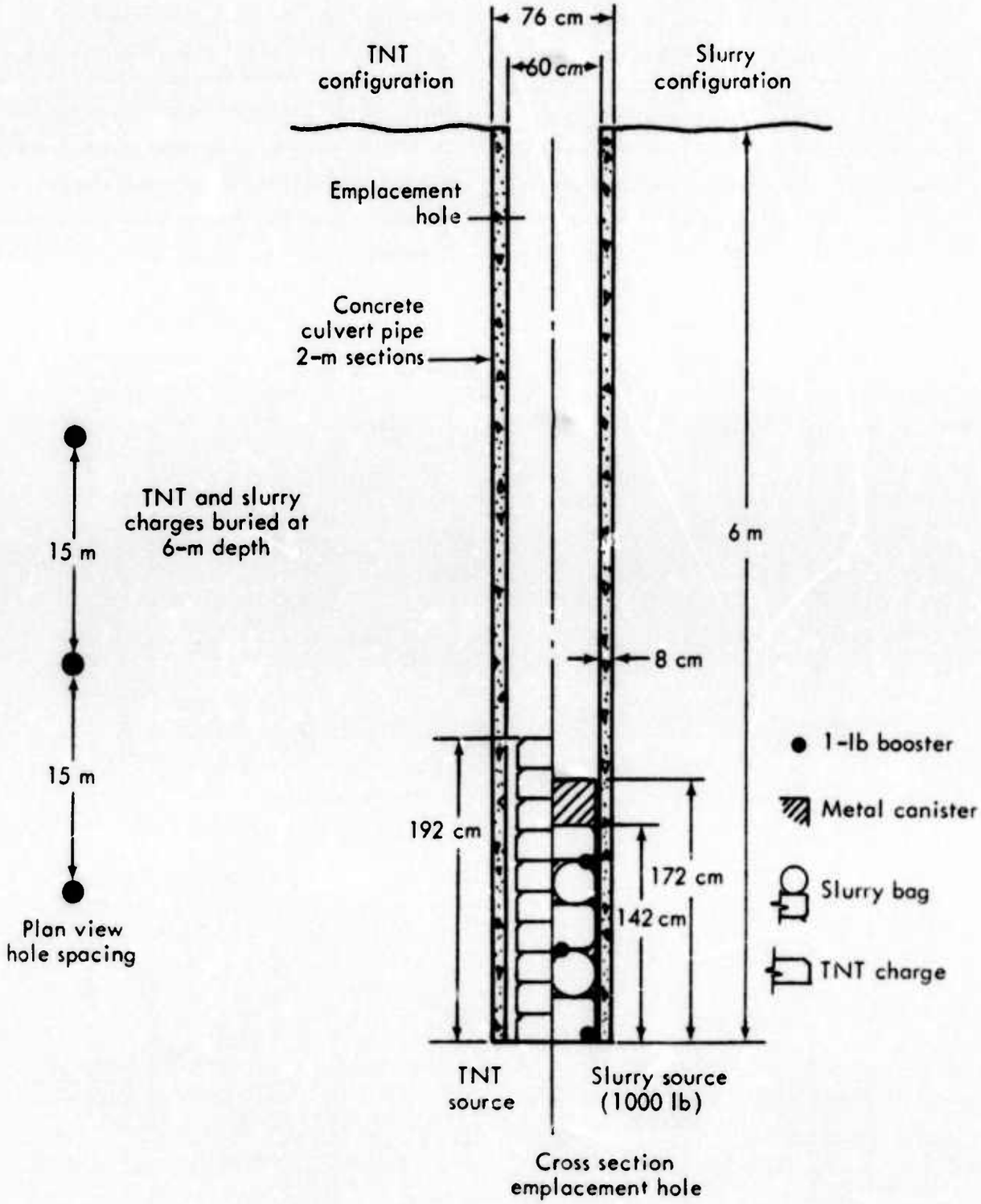
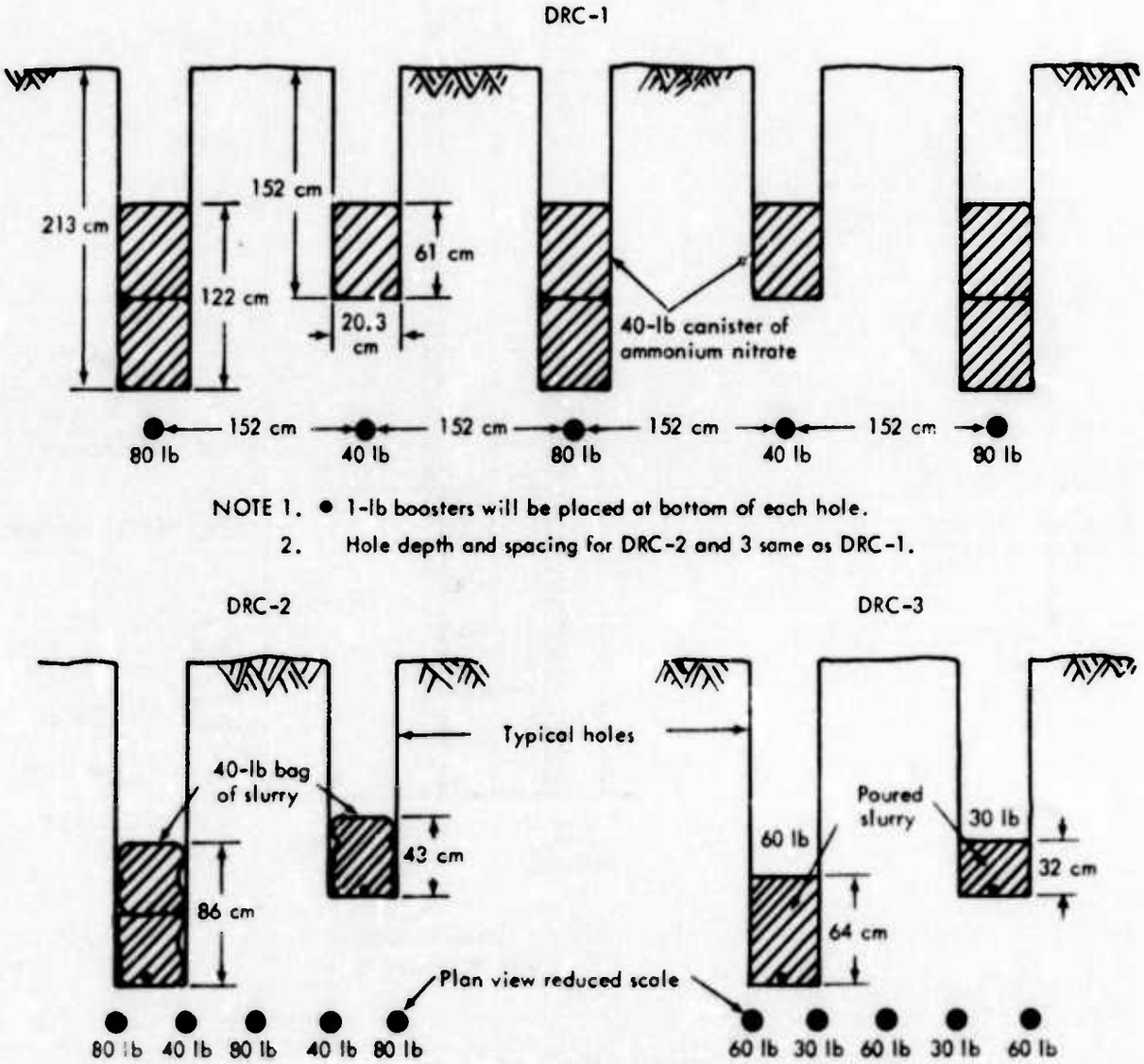


Fig. A13. Design for Series I (PC-1 and -2).

Army's Standard Deliberate Road Crater DRC-1 (Fig. A14a) used eight 40-lb ammonium nitrate canisters. DRC-2 (Fig. A14b) used the same hole size, spacing, and depth as DRC1 and was designed to determine whether bagged aluminized slurry produces the same size crater as the ammonium nitrate cratering charges. DRC-3 (Fig. A14c) also used the same design as DRC-1; however, the total weight of the explosive was reduced to 240 lb which was poured into the emplacement cavities according to the typical hole configuration (Fig. A14c).

Series II verified the effectiveness of changing the Standard DRC design by reducing the number of emplacement holes and the quantity of explosives. To produce a crater equivalent to the ones produced in DRC-1, -2, and -3, Experiment DRC-4



F. A14. Design for Series II (DRC-1, -2, and -3); same hole configuration for all three shots with varying charge heights.

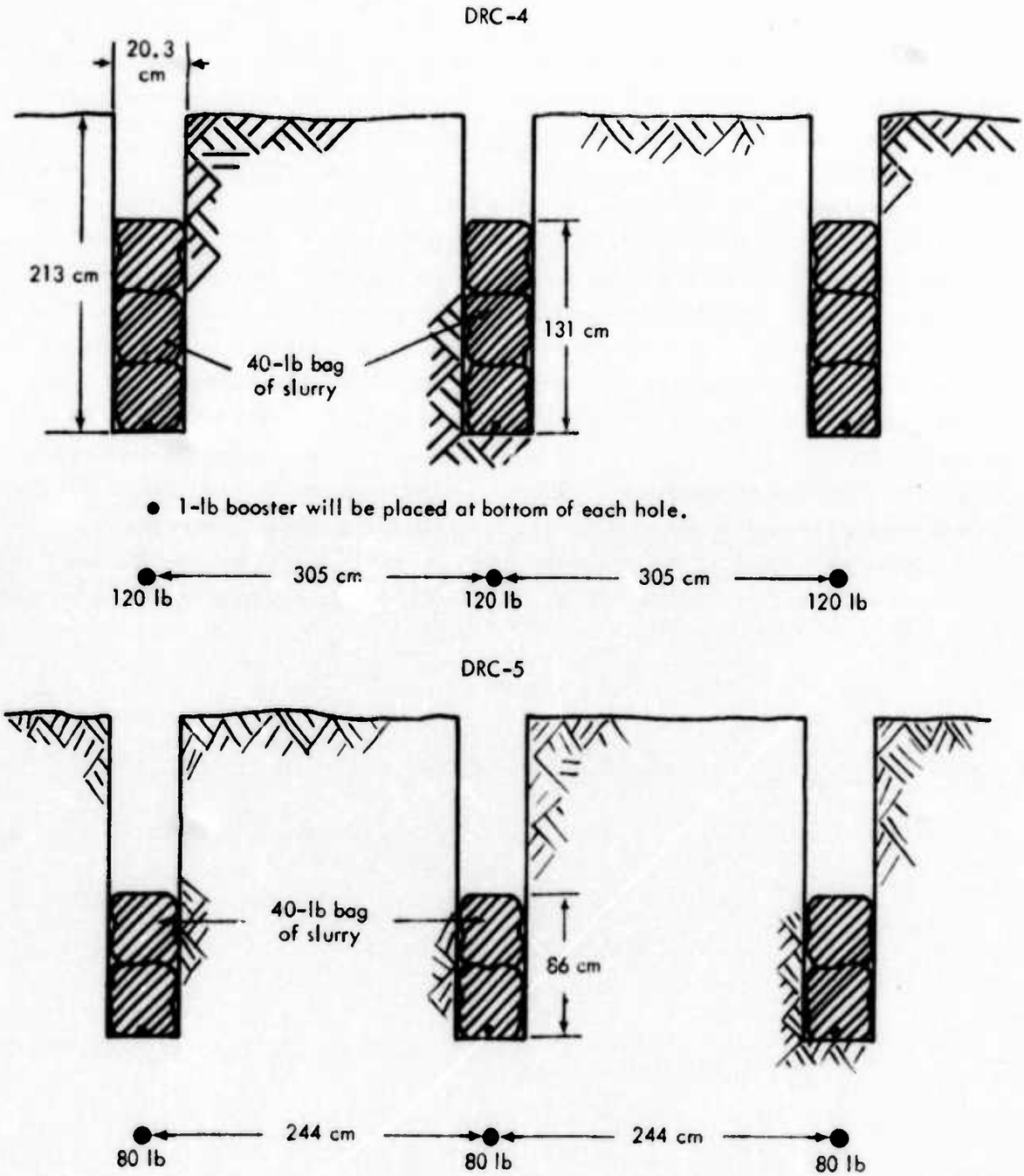


Fig. A15. Design for Series II (DRC-4 and -5); same size hole used in both shots.

(Fig. A15a), reduced the number of emplacement holes to three, increased the hole spacing by 153 cm, and had a constant hole depth of 213 cm, also increased the total amount of explosive required by 40 lb. The second experiment design evaluated was DRC-5 (Fig. A15b), which used the same design as DRC-4 with the exception of the hole spacing (reduced to 244 cm – 8 ft), and the total weight of the explosives (reduced by 80 lb).

The technical concepts of Project Armor Obstacle II are summarized in the following technical objectives:

1. To compare the cratering results of equivalent quantities of an aluminized slurry vs TNT in the same charge configuration in terms of crater dimensions and obstacle effectiveness

2. To evaluate the handling requirements for loading and unloading both large and small quantities of slurry explosives in deep and shallow emplacement cavities

3. To evaluate the utility of a field-expedient explosive container for loading and unloading bag or bulk slurry explosives in deep holes

4. To verify the advantages of using slurry explosives to produce a Deliberate Road Crater by comparing the cratering results of both identical and equivalent quantities of an aluminized slurry in plastic bags, 7 in. in diameter and 20 in. in length, vs 40-lb ammonium nitrate canisters in terms of crater dimensions and crater effectiveness

5. To test the feasibility of modifying the DRC design to accommodate slurry explosives in the same configuration with a view toward reducing the number of emplacement holes and the quantities of explosives required

6. To compare crater dimensions produced by a 40-lb AN canister vs a prilled (converted into spherical pellets) ANFO canister of equal weight and approximately the same dimensions

A final report covering the results of Project Armor Obstacle II is in preparation.

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