BuMines OFR 111-80

PB81-125577

U.S.B.M. Grant No. H0272020

RAPID EXCAVATION OF ROCK WITH SMALL CHARGES OF HIGH EXPLOSIVE

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Published January 1979

Final Report for May 1977-June 1978

Bureau of Mines Open File Report 111-80

Prepared for

United States
Department of the Interior
Twin Cities Mining Research Center
Twin Cities, Minnesota

i.a

50272-101

REPORT DOCUMENTATION PAGE	1. REPORT NO. BuMines OFR 111-80	2.	3. Recipient's Accession No. PB81 125577
4. Title and Subtitle Rapid Excavation of Rock With Small Charges of High Explosives			5. Report Date January 4, 1980
			6.
7. Author(s) George B. Clark, David W. Ashcom, and Kanaan Hanna			8. Performing Organization Rept. No.
9. Performing Organization Name and Address Colorado School of Mines Golden, CO 80401			10. Project/Task/Work Unit No.
			11. Contract(C) or Grant(G) No. (C) H0272020 (G)
12. Sponsoring Organization Name and Address Office of the Director--Mineral Resources Technology Bureau of Mines U.S. Department of the Interior Washington, DC 20241			13. Type of Report & Period Covered Contract research, 5/77--12/79
			14.

15. Supplementary Notes
Approved by the Director, Bureau of Mines, for placement on open file, February 15, 1980.

16. Abstract (Limit: 200 words)

The purpose of this investigation was to test a blast shield, develop a conceptual design for an automated drill and blast system (ADBS), and make an economic analysis of the ADBS compared with conventional drill, blast, and muck (DBM) systems. The ADBS tunneling concept was developed in an effort to design an excavation system of greater efficiency than the DBM systems conventionally used. The ADBS would minimize downtime and approach noncyclic efficiently by small charge blasting. Small charge blasting uses simultaneously detonated light charges in four to eight short holes, usually in a line. Limitation of the total explosive per blast reduced air blast overpressure, fly rock velocity, vibrations, and noise enabling the blast shield to adequately contain these side effects. Conceptual design for the ADBS included blast shield, chasis, hydraulic drills, automated explosive loading and firing system, and an armored cab. Horizontal and vertical blast shield alinement controls and ventilation are incorporated into the design. The economic analysis determined ADBS advance costs to be 17 to 20 pct lower than DBM on a per foot basis.

17. Document Analysis a. Descriptors
Excavation
Tunneling
Explosives

b. Identifiers/Open-Ended Terms

c. COSATI Field/Group 11F

18. Availability Statement	19. Security Class (This Report)	21.
Release unlimited by NTIS.	20. Security Class (This Page)	22. Price

(See ANSI-Z39.18) See Instructions on Reverse OPTIONAL FORM 272 (4-77) (Formerly NTIS-35) Department of Commerce

Foreward

This report was prepared by the Colorado School of Mines, Golden Colorado, under USBM contract number H0272020. The contract was initiated under the Advancing Metal and Nonmetal Mining Technology Program. It was administered under the technical direction of the Twin Cities Research Center with Jon K. Ahlness acting as Technical Project Officer. William P. Battle was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as a part of this contract during the period May 1977 to December 1979. This report was submitted by the authors on January 4, 1980.

TABLE OF CONTENTS

CHAPTER		PAGE

CHAPTER PAGE

LIST OF FIGURES

FIGURE NO.		PAGE

LIST OF TABLES

ABSTRACT

One portion of this project was to test a blast shield and to design some features for a novel tunneling method termed the Automated Drill and Blast System (ADBS). The ADBS tunneling concept was developed for the design of an excavation system of greater efficiency than conventional drill, blast, and muck (DBM) systems by minimizing the down time and approaching the non-cyclic efficiency of continuous tunneling by means of small charge blasting. Small charge blasting uses simultaneously detonated, small light charges (less than 200 grams per hole) in four to eight shallow (24 to 36 in. deep) holes usually in a line. Limitation of the total explosive per blast reduced air blast overpressure, fly rock velocity, vibration, and noise. Thus, shielded equipment can remain at the face and a tunnel shield permits support personnel to work outby the shield at all times.

The ADBS equipment will be composed of five components: (1) a chassis, upon which is mounted automated mucking equipment, and a frame which carries (2) the hydraulic drill(s), (3) load/blast system(s), and (4) control cab(s). The blast shield may have its own mounting. Ventilation, horizontal, and vertical alignment controls are incorporated in the system.

The blast sheild, which protects personnel outby the advance equipment from air blast overpressure, fly rock, and gases, was tested with full scale small charge blasting. A blast shield was designed, fabricated, and tested under conditions anticipated for a working model. The shield was designed to expand to fit the tunnel shape, to seal itself to the tunnel periphery, and to contain air blast, fly rock, and gaseous explosion products. In addition, the shield, which was lined on the inby side with rubber belting, reduced the blast noise level outby the shield to 113 dBA or less, which is very near permissible levels.

The test results show conclusively that the shield concept is viable.

Based partially on the shield tests, a blast resistant control cab was designed to be located on the equipment chassis inby the blast shield to protect operators during blasting, and to contain all the controls necessary for system operation and safety.

The economics of tunnel driving using the ADBS were compared with those of DBM. Four rock types, which determine advance rate, and four cost categories were examined. The results of this analysis are as follows:

1. ADBS advance rate is faster because equipment remains at the face and operation procedes semi-continuously.
2. ADBS bit and explosive cost (which accounts for 4% of the total advance cost) is higher than for DBM.
3. ADBS equipment cost, on a per foot basis, is lower than for DBM.
4. Due to a reduced labor force, ADBS labor cost is lower than for DBM.
5. Because of the overbreak reduction due to controlled blasting, ADBS tunnel support cost is lower than for DBM.
6. On a per foot basis, total ADBS advance cost is from 17% to 20% lower than for DBM, depending on tunnel diameter and rock type.

The small charge method greatly reduces ground vibration, increases personnel safety by limiting excessive fracture, and decreases tunnel support requirements by reducing overbreak. The overall analysis, tests, and results to date show that the Automated Drill and Blast System may be developed into a viable tunnel excavation method. The method also may be adapted to use in the mining of ore.

For some types of mining operations, the cyclic nature of conventional drill and blast methods may not be a disadvantage, either where the drilling and mucking may proceed simultaneously or where the operations are carried out in adjacent working faces. The proposed method would find application where continuous methods of excavation are desirable.

CHAPTER 1

AUTOMATED EXPLOSIVE SYSTEMS

Introduction

The demand for minerals is growing at an increasing rate. In the United States, where the per capita consumption of minerals is the highest in the world, the problems of mining at greater efficiency are compounded by many factors. The grade of newly discovered ore bodies is often lower than older ones and the more accessible ore is mined before more inaccessible deposits are developed. The result is that, in general, ore of decreasing grade must be mined under increasingly costly conditions.

Tunneling is also increasing for such uses as public transportation and utilities (water, sewage, etc.). The use of underground excavations is expected to increase for purposes such as storage and office space (Fairhurst, 1976), and military and civil defense installations have traditionally utilized subsurface sites.

The development and use of subsurface openings will continue to increase. Improvement of mining processes requires rapid and economic access to ore bodies and improved excavation methods. Underground excavations for civil purposes will, in many cases, be required in populated areas. This imposes more environmental restrictions on both the surface and subsurface excavation operations. These constraints, combined with increased labor and material costs, makes the development of improved, more economical excavation methods expedient.

The requisites for an improved excavation system are as follows:

1. Decreased excavation cost
2. Increased advance rate

3. Excavation of most types of hard rock
4. Reduction of support requirements
5. Provision of safe working conditions
6. Decreased noise, vibration, air blast, and overbreak
7. Accurate tunnel alignment
8. Allow support installation as close to the face as possible
9. Decreased maintenance costs and down time
10. Simplified disassembly of equipment for transport
11. Low energy consumption

Many of these factors are characteristic of the two major excavation methods currently in use, drill, blast, muck (DBM) and tunnel boring machines (TBM). Each of these systems, however, has fixed limitations, as described below.

Small Charge Blasting

Explosive excavation systems offer several advantages over mechanical systems, foremost of which is the ability of explosives to fragment even the hardest rock, and blasting is flexible, allowing continued operations when rock type, strength, or structure changes.

Excavation with explosives, as currently applied, has two serious disadvantages. Conventional blasting produces excessive air blast, vibration, and overbreak, which also causes decreased rock strength of the periphery of the opening and increased rock support requirements. The other primary disadvantage is the cyclic, inefficient use of manpower and equipment.

The small charge concept was developed in order to mitigate the disadvantages of conventional blasting. In this method, rock breakage is accomplished by the simultaneous ($\pm$ 100 μsec) detonation of several small (less than 200 grams per hole) stemmed charges of high explosives. The limitation of the charge size reduces air blast, vibration, and overbreak. Simultaneous detonation of adjacent charges increases explosive efficiency, thus reducing the powder factor. The effect of simultaneous detonation is superposition (reinforcing) of stress waves between adjacent holes, which has been successfully employed in presplitting and smooth blasting, some of the mechanisms of which are explained by photoelastic analysis (Dally & Khorana, 1971).

Reduction of the explosive consumption per hole requires a similar reduction of the blast hole depth, burden, and spacing. A round depth of 18 in. was successfully tested (Clark & Rollins, 1976) using a burden and spacing of approximately 12 and 18 in., respectively. The powder factor ranged between 1.3 to 2.7 pounds of explosive per cubic yard of rock broken. Powder factors for conventional tunnel rounds exceed 1.6 and usually approach 2.7 (Langefors & Kihlstrom, 1973). Research reported herein was conducted to verify effectiveness of 24 and 30 in. deep rounds (Hanna, 1978) which will reduce blasting costs and minimize many of the undesirable effects of conventional blasting.

In addition to the direct effects of smaller blasts on the rock, fracture control, and fracture augmentation, the blast effects can also be contained at the face, equipment can be shielded and remain at the face, and support personnel can work outside of the tunnel shield. In short, the method permits containment of the blast effects so that operations may be carried on in a continuous manner.

The type of round chosen for experimentation was a V-cut in which segments of the round which are in a line can be fired simultaneously to maintain the breakage effectiveness of this type of round with a small powder factor. The method can also be employed with other types of rounds. Two center lines of the V-cut are fired first, and each successive line blast generates new free faces to which the next line blasts may break. The peripheral holes, which are fired last, are spaced closer together and loaded using as small a powder factor as possible for fracture control and to avoid overbreak or fracturing of the rock in the periphery of the opening.

While this method was designed and tested for possible use in tunnels, it appears to have application in several types of mining. For example, it could be designed into a mining system to excavate a longwall face for selective mining. The same features would be retained, i.e., continuous drilling, blasting, and mucking, with personnel remaining at the face in protected cabs. Cyclic methods are not detrimemtal in some types of mining.

The simultaneous detonation of charges in an automated system requires the use of detonating fuse or exploding wire (EBW) caps since millisecond delay caps do not achieve the desired accuracy of detonation ($\pm$ 100 μsec). An EBW cap contains PETN and detonates only when a specific high energy electric pulse is applied, which results in a safer cap not subject to the usual causes of accidental initiation. Initiation of the cap is by means of a "firing set" (Reynolds Industries, 1977) which produces the necessary energy pulse. Currently, a firing set can detonate only six caps, which is also the desired number of small charges per blast employed in experimentation. Larger sets can be built if necessary.

Required types of slurry explosives are cap sensitive in diameters as small as 7/8 in. One inch diameter slurry charges were used in 18 in. deep rounds (Clark & Rollins, 1976); the experimentation described in this report also utilized one inch diameter slurry charges (Hanna, 1978). This type of explosive was chosen because (1) it is safer than straight dynamite, (2) it can be more easily handled by an automated loading system, and (3) it does not cause headaches when it is handled.

Small Charge System

The use of the small charges mitigates many of the disadvantages of conventional blasting. Noise, vibration, and air blast are reduced, which allows shielded equipment to remain at the face during blasting, and a tunnel shield behind the equipment to contain the blasting effects to the area near the working face. One aspect of this concept has been successfully demonstrated in field tests (Peterson & others, 1976) of the Rapidex spiral blasting method.wherein drilling, blasting, and mucking may proceed semi-continuously.

The small charge Automated Drill and Blast System (ADBS) is planned to operate as follows: While a set of properly spaced holes is being drilled to the desired depth, previously drilled holes will be mechanically loaded, primed, and stemmed. When the loaded holes are connected to the firing set, the drills and other ADBS components will be shielded and the explosive detonated. The muck produced by the blast can then be gathered and removed from the face area by a conveyor while drilling and explosive loading continue.

To obtain rapid advance rates, hydraulic drills will be utilized. Two automatic loading systems appear to be feasible, and a prototype of a third system has been successfully demonstrated by Peterson, et al (1976). Mucking will be accomplished with a conventional gathering arm or a similar type of equipment.

The operator(s) and controls will be housed in an enclosed, air conditioned cab, shielded from fly rock, noise, and vibrations mounted on the rear of the main chassis. The main blast shield will be located outby the other components. The purpose of the shield is to isolate the heading area so that support erection and other activities can proceed continuously, unaffected by the blasting operations. This is accomplished by sealing the shield to the invert, ribs, and back so as to contain fly rock, dust and gases, and blast overpressures inby the shield. Blast noise transmitted through the shield is attenuated to allowable levels. Besides being capable of sealing itself to the tunnel periphery, the shield must also permit the semi-continuous passage of muck from the face. In operating shields, access for men and material will be provided by means of a second access door. Utility lines (air, water, electricity, ventilation) will also pass through the shield. On an operational system, interlocking controls on the shield and the cab will prevent blasting until all doors and other openings are properly closed and sealed. The design procedures to account for the blast characteristics and their effects on the design of the blast shield and cab are straight forward (Appendix A). The design and testing of an experimental shield, and the design of a prototype system were two of the major objectives of this project (Chapter 2).

Comparison of Tunneling Methods

There are currently two proven tunneling methods for excavation in hard rock:

1. Drill, Blast and Muck System (DBM)
2. Tunnel Boring Machine (TBM)

In addition, two novel tunneling, blasting methods are the subject of research and development:

3. Spiral Drill and Blast Machine (SDBM)
4. Automated Drill and Blast System (ADBS)

Drill, Blast and Muck System. In the DBM method of tunnel or drift excavation, the procedure is to drill holes in the rock face, load the holes with explosive, blast, muck, and support the rib and back if necessary. Improved explosives, mechanized drilling and mucking, and advanced engineering have increased productivity, but the cyclic nature of DBM operations limits the effective continuous use of the labor force and equipment.

In spite of the inefficiency due to cyclic operations, the DBM is the method most used in tunnel and drift construction. This is due to the method's flexibility. That is, the DBM can operate successfully in conditions ranging from the hardest rock to softer rocks, and can readily be adapted to changing rock conditions which are frequently encountered.

Tunnel Boring Machines. Modern hard rock tunneling machines have developed rapidly since the three machines designed by James S. Robbins were successfully used for excavating tunnels at the Oahe Dam in 1954. The TBM concept is not new, however, as evidenced by the successful excavation of over 6,000 of 7 ft diameter tunnel by means of a compressed air driven fixed pick machine in 1884 (Muirhead & Glossop, 1968). Several other machines were successfully used in moderately hard rock (limestone and sandstone) prior to 1950.

Currently, TBM tunneling is limited to rocks of moderate strength and hardness. This includes most sedimentary rocks and some of the softer igneous and metamorphic rocks. Harder metamorphic rocks, such as schist and gneiss, can be drilled because of recent advances in cutter and bearing technology.

A TBM system must be able to (1) continuously cut rock, (2) collect and remove rock cuttings as they are produced, (3) excavate the tunnel to correct line and grade, and negotiate horizontal and vertical curves, (4) permit support installation as close to the face as necessary, and (5) control dust, water, and heat. In addition, it should be relatively simple and with a high degree of reliability, easy to maintain and disassemble, and allow rapid cutter displacement.

A TBM cuts the full tunnel face, normally producing a circular opening. (An exception is Atlas-Copco's "Mini'Fullfacer" which produces an arched back and invert and straight ribs using an under cutting technique). Machine advance is generally obtained by securing one section of the TBM to the tunnel periphery via hydraulic jacks and thrusting the cutter head assembly forward, using the stationary section as the point of forward thrust. (Some systems use a pilot hole anchorage system to pull the machine forward).

The cutter head assembly rotates and is thrust against the rock face. The excavated material is gathered by buckets or scraper conveyors and passed through the cutting head on conveyors where it may be removed from the heading by various means. TBM usage is increasing due to its rapid advance rate and low labor requirement. Principal disadvantages include high capital cost, long delivery time, and maintenance costs in hard rock.

Based on available data and estimates, the advantages and disadvantages of the four systems are summarized as follows, assuming that the two new methods can be made operational as predicted:

Drill, Blast and Muck System

Advantages -

- Blasting can break the hardest rock
- Adapts to changing rock conditions
- Excavates any tunnel size or geometry
- Individual components may be quickly replaced, avoiding prolonged down time
- Rapid equipment set up
- High reliability, availability
- Support can be installed close to the face

Disadvantages -

- Slow advance rate due to cyclic operations
- Labor intensive
- Relatively high powder factor
- High blast overpressure, noise intensity, vibration, fly rock velocity
- Excessive overbreak and wall fracture, creating safety hazard and increasing support requirements
- Large cost per foot

Tunnel Boring Machine

Advantages -

- Continuous advance at a rapid rate
- Creates stable opening, improving safety and decreasing support requirements
- Few operating personnel, reduced labor cost
- Simplifies muck removal
- Lower cost per foot

Disadvantages -

- High capital cost and long lead time
- Limited to moderately strong rocks
- Long assembly time
- Moderate reliability
- Requires large ventilation system
- High power consumption
- Cannot excavate very large openings
- Cannot excavate unstable rock

Spiral Drill and Blast Machine

Advantages -

- Semi-continuous advance at rapid rate
- Can excavate hard rock
- Adapts to changing rock conditions
- Excavates any tunnel geometry
- Relatively low powder factor
- Nominal to maximum support requirements
- Remote monitoring and control of advance
- Few operating personnel, reduced labor cost
- Low cost per foot - projected

Disadvantages -

- Remote monitoring and control may be a disadvantage
- High air blast, vibration, and fracture of the wall and overbreak
- Mechanically complex
- Moderate reliability, availability
- Support installation cannot be erected close to the face
- Cannot excavate very small or large openings
- Requires skilled labor

Automated Drill and Blast System

Advantages -

- Complete shielding of outby work area
- Semi-continuous advance at rapid rate allowing support erection close to the face
- Creates less fractured opening, improving safety and decreasing support requirements
- Reduced ground vibration, air blast, and noise

- Low powder factor
- Operator(s) can remain at the face
- Few operating personnel, reduced labor cost (urban areas)
- Excavates any tunnel geometry
- Low cost per foot - projected
- Incorporates many proven components

Disadvantages -

- Men continuously remain at the face (currently disallowed by MESA and USBM)
- Complicated rock removal from the face
- Cannot excavate very small openings
- Requires skilled labor

If the projected operational characteristics and economics summarized in Table 1 and the advantages and disadvantages outlined above represent a reasonable assessment, the ADBS utilizing small charges and shielding has the potential of becoming the more viable tunneling method for a wide range of rock conditions. That is, such an excavation system will provide flexibility of operation reduced costs, together with the other favorable characteristics described, and will utilize explosives as excavating agents. By far the greatest portion of excavation is now carried out with the drill and blast in spite of its several unfavorable characteristics. If these unfavorable factors can be reduced, most of the favorable factors which have given explosives such a broad application in the excavation industry can be retained.

TABLE 1-1

COMPARISON OF THREE EXCAVATION SYSTEMS - LIMESTONE

Excavation Method	DBM	TBM	ADBS
Excavation Characteristics	Cycle: drill, blast muck	Continuous cutting	Semi-continuous drill, blast, muck
Type of Rock	No limitation	6-30 ksi comp. str	No limitation
Availability (Machine)	70 - 95%	40 - 85%	60 - 90%
Advance Rate (ft/day)			
12' diam.	65	91	79
24' diam.	52	76	63
Powder Factor (lb/ft^3)	1.6 - 8.4	N.A.	1.35 - 2.7
Equipment Cost			
12' diam.	$330,000	$1,500,000	$325,000
24' diam.	497,000	3,000,000	487,000
Manpower			
12' diam.	15	2 - 4	5
24' diam.	24	2 - 4	9
Support Requirements	Maximum due to fracture and overbreak	Minimal due to excavation method	Nominal due to smooth blasting

CHAPTER 2

THE ADBM CONCEPT

Introduction

The concept of the ADBS was designed to overcome most of the disadvantages of conventional drill, blast, muck (DBM) tunneling. These disadvantages are summarized as follows:

1. Advance rate and efficient utilization of manpower and equipment is limited by the cyclic nature of the method.

2. Conventional blasting produces high blast overpressure, noise, and vibration.

3. Conventional blasting can cause excessive overbreak and wall fracture, creating a safety hazard and increasing support requirements.

4. The DBM system of advance has a high cost per foot.

The disadvantages associated with DBM tunneling are all directly related to the manner in which explosives are consumed. Because several holes containing a large amount of explosives are detonated at once, high overpressure, fly rock velocity, ground vibration, and noise are produced. Thus, men and equipment must be moved from the face to a safe location. This requires that the operation be cyclic in nature, which creates an inefficiency that results in a high cost per foot of advance.

In a small round, large delayed blasts are replaced with several smaller blasts. Small charges in 4 to 8 holes with the charges detonated simultaneously provide efficient rock breakage and limit blast overpressure, fly rock, and ground vibration. The total weight of explosive consumed per shot can be adjusted so that the air blast pressure is limited to a magnitude which can be safely contained. This allows shielded equipment to remain at the face and increases overall efficiency. The small charge round produces a smooth periphery in the same manner as in smooth blasting. This technique improves safety by creating a less fractured perimeter and decreases overbreak and support requirements.

The ADBS, which may remain at the face because of the utilization of small charges, will permit semi-continuous drilling, blasting, and mucking. A blast shield, which contains overpressure, dust, fly rock, and attenuates noise, will provide a safe working environment outby the advance equipment.

ADBS Components

Chassis. The ADBS chassis will consist of dual crawlers to advance the equipment, an apron and gathering arms to semi-continuously muck the broken rock, a conveyor to discharge the muck outby the blast shield, and an upper frame to carry the drill(s), load/blast system(s), control cab(s), and the blast shield support booms.

Drills. One or more (depending on the size of the face area) hydraulic drills will be mounted on the chassis frame. Each drill will be operated by controls located in the cab. Hydraulic drills, which have a higher penetration rate than pneumatic drills, will be utilized, with each drill being mounted on its own boom.

Load/Blast System. The proposed explosive loading and blasting system must be compatable with the total system operation. Two possible systems are proposed for investigation. The first will be patterned after the system which has been employed to date, i.e., detonating cord. A second system to be investigated will employ exploding blasting wire caps which are safe to use and have the required accuracy for simultaneous blasting of adjacent holes (see Chapter 7).

Control Cab. A control cab is designed to protect one drill operator and one load/blast operator. The purpose of the cab is to provide a safe, comfortable working environment for the operators. It will be heated or air conditioned as required, and will protect the operators from overpressure, dust, fly rock, and excessive noise. The cab, mounted on the chassis frame, will contain all production and safety controls.

Blast Shield. Blast overpressure, dust, fly rock, and excessive noise will be contained in the heading by a blast shield mounted at the rear of the chassis. The shield will consist of sections powered by hydraulic jacks, the sections conforming to the shape of the drift. A low porosity foam material will seal the periphery of the shield to the drift; a rubber lining on the inby face will attenuate blast noise. Doors will be provided for the muck opening and a manway. The shield will be attached to the rear of the chassis frame by two hydraulic booms.

Ventilation. The heading will be ventilated by means of the tunnel exhaust ventilation system. Except during blasting, the air inby the shield will be exhausted by a pipeline from the shield to the portal where the main fan will be located. The cab(s) will be ventilated by fresh air from outby the shield to circulate inby the shield. An auxillary fan may be required to clear the face of smoke in the desired length of time.

Component Shielding

The protection provided by the blast shield and control cab(s) against blast overpressure and fly rock consists of high strength steel plate and a rubber lining. Field testing of the blast shield proved these materials effective. Similar protection of the chassis is not required (Alpine, 1977), however, this should be verified during prototype testing.

Field testing of the SDBM (Peterson, et al, 1976) has shown that machine components can be adequately protected from blast damage by a heavy wire mesh shield. The hydraulic lines and cylinders on the drill and load/blast systems must be protected by steel plate. To provide protection for the load/blast systems, hydraulic drill, drill steel, and drill slide, an expanded metal shield will be used.

The shielding measures described above will protect all ADBS components from blast damage. However, one additional shielding system may be considered in future research. This would consist of a shield located at the face, fabricated from expanded metal (heavy wire mesh) or similar material. Such a shield, which would be hydraulically erected during blasting and collapsed during drilling, loading, and mucking, and would contain fly rock and provide additional component protection.

ADBS Operation

A typical round using the ADBS will be carried out as follows (Figure 2-1). Operators unlatch the manway door on the shield, enter the inby area and latch the door behind them, and enter the control cab and latch the cab door. Lights for the face and control console are turned on. The cab ventilation system is turned on, followed by the drill and load/blast system hydraulic pump and the chassis hydraulic pump.

While one operator is drilling a line of holes, the second operator is loading previously drilled holes. When drilling and loading are complete, the drill and load/blast booms are retracted from the face. The operators then close the control cab window shield, the butterfly valve in the main exhaust line, the cab intake and exhaust lines, and the muck discharge door.

Lights on the console indicate that all apertures are correctly closed (or indicate which is not closed) which closes the firing circuit. The

FIGURE 2-1

PLAN VIEW OF THE INTEGRATED ADBS

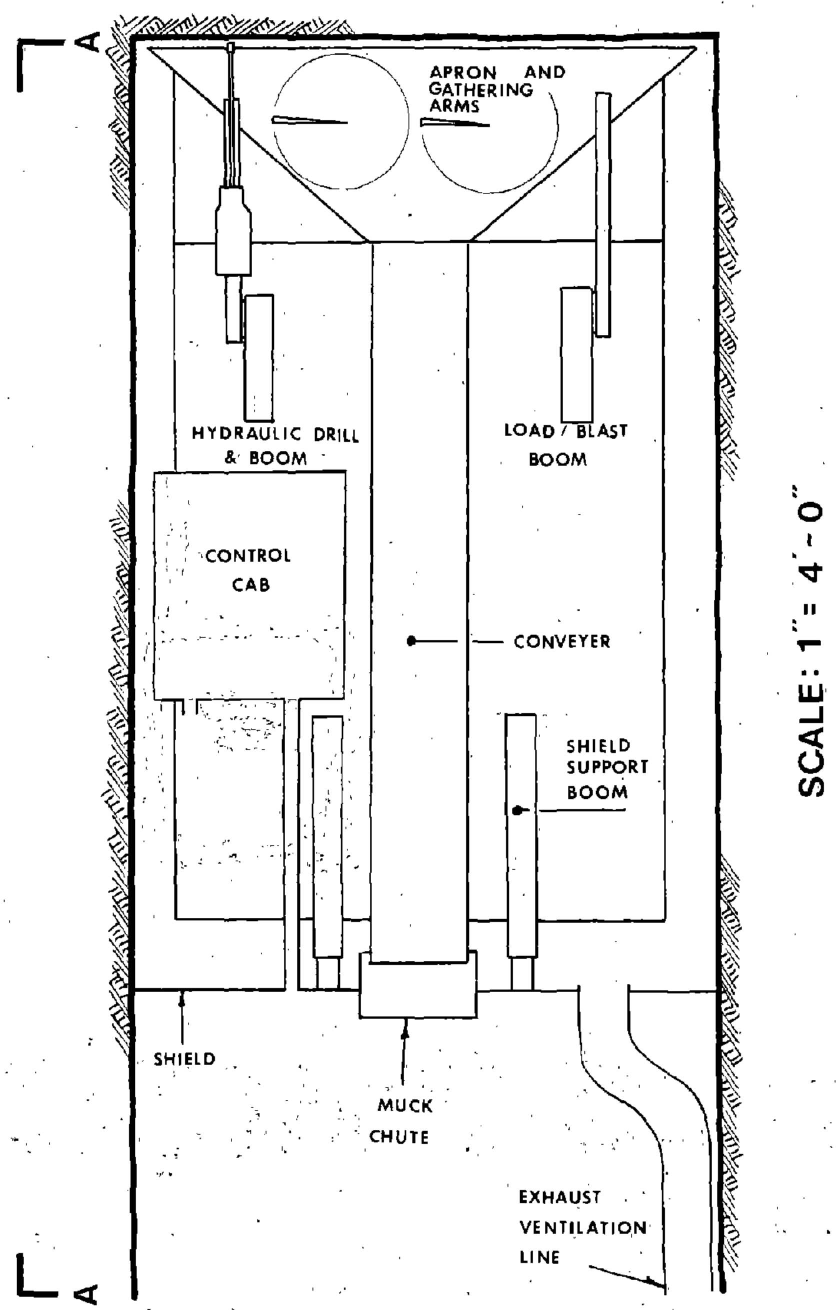
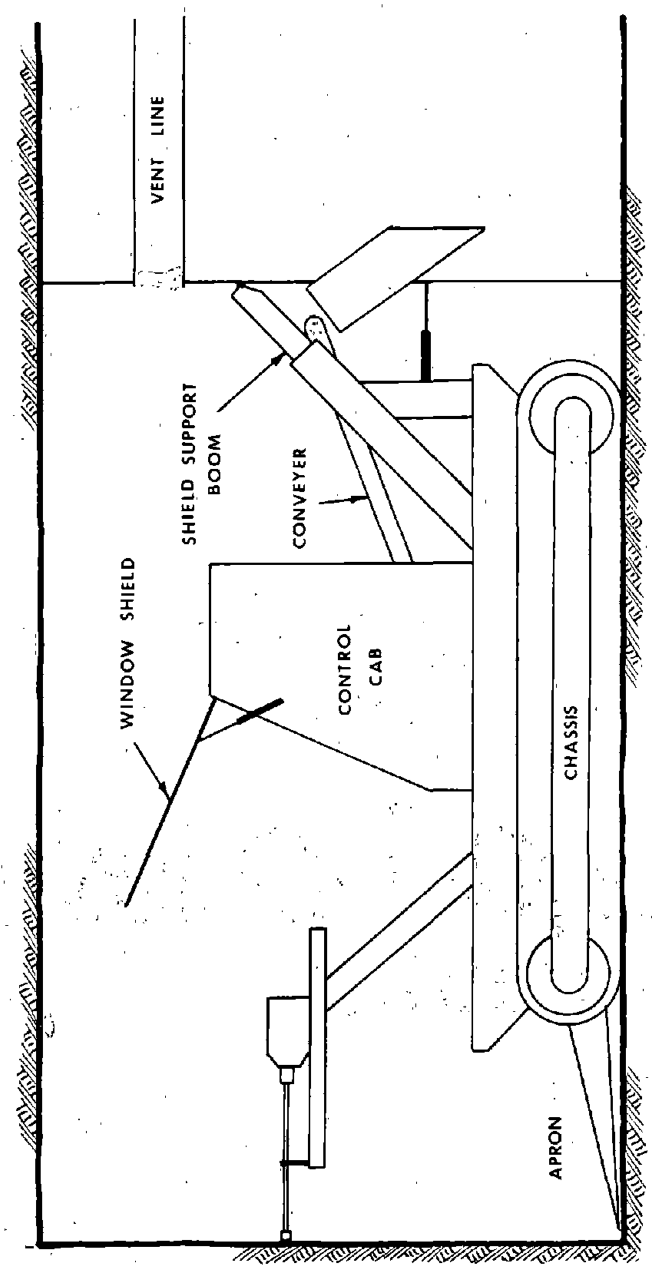


FIGURE 2-1 (CONTINUED)

SECTION A-A



SCALE : 1" = 4'-0"

shot is detonated by pressing a button on the firing set.

Immediately after the shot the control cab and exhaust ventilation systems are reactivated, and the muck discharge door is opened to complete the ventilation circuit. The control cab window shield is opened, and when the working face can be seen, drilling and loading commence at the same time the mucking system is activated.

When a complete round has been excavated, the shield sections are retracted and ADBS is moved ahead. At this time, men outby the shield measure line and grade and inform the operators of necessary correction in horizontal or vertical alignment. The two hydraulic booms position the shield in the center of the drift and the shield sections are extended. Men outby the shield verify that the shield is properly sealed in the drift via telephone.

While excavation continues, utility lines are extended and muck is transported from the outby shield area. Tunnel support erection, as required, and other activities proceed uninterrupted by the heading advance.

Contingencies

In addition to typical mechanical and electrical failures inherent in any mining equipment, several situations could possibly occur which demand consideration. These are misfires, electrical failure, and fire.

Misfires. In case detonation is accomplished by electric (EBW) caps, a misfire may be detonated by rewiring the shot. If the misfire is due to a cap malfunction, the problem becomes more complex. Since each hole must be stemmed in order to achieve the required explosive containment, top priming the misfire will not provide detonation. Two solutions to the misfire problem are possible. The first is to open circuit the shot and manually extract the stemming and explosive. However, this method is time consuming and dangerous. A better solution would be to provide the cab window shield with a slit fitted with high impact resistant glass. When a misfire occurs, the ADBS would be closed as during a blast and an adjacent hole drilled and shot. The misfired cap and explosive could then be collected from the muck and detonated with the next shot.

<u>Electrical Failure.</u> An electrical failure would render the ADBS completely inoperable and stop the ventilation system. If such a situation occurred, even immediately after a blast, the operators would be safe. Flash lights or cap lamps stored in the cab could be used while the cab and shield doors are manually opened. Self-rescuers, also stored in the cab, would provide the operators with protection from dust and blast gases except oxides of nitrogen while they exited the ADBS.

<u>Fire</u>. On any piece of mechanical equipment, fire is a possibility. Fire extinguishers located outby the shield and in the control cab will probably be able to extinguish any minor fire typical of such equipment. As in the case of an electrical failure, the operators can exit the ADBS using cap lamps and self-rescuers. It is emphasized that, due to the types of materials used in ADBS fabrication and because of MESA safety regulations, the possibility of fire is remote.

CHAPTER 3

BLAST SHIELD DESIGN

Introduction

One of the key components of the small charge Automated Drill and Blast System (ADBS) is the blast shield. This component of the system physically separates two segments of tunnel operations:

1. Inby shield operations
    a. Drilling
    b. Blasting, using the simultaneous detonation of several small charges of high explosives
    c. Muck gathering onto a conveyor
2. Outby shield operations
    a. Support installation
    b. Muck loading and hauling
    c. Installation of air, water, and ventilation lines
    d. Advance of track (if rail haulage employed)
    e. Other operations in support of face advance

The prime function of the blast shield is protection of outby personnel and equipment from blast induced fly rock, overpressure, and excessive noise. The shield must provide an access for men and materials (bits and explosives, etc.) to the face area and an exit mechanism for muck removal. Electric, communication, water, air, and ventilation utilities must also pass through the shield. All of these openings must be designed so that dust, fragments, and overpressure will be contained inby the shield and the transmission of noise and vibration through the shield will be limited to acceptable levels.

The minimum estimated distance from the face to the shield under operating conditions is 20 ft. If six holes, each loaded with one quarter pound of slurry explosive and stemmed, are detonated simultaneously, the peak overpressure predicted by equation is 10.5 psi. The design of the blast shield used in the field tests described below was based on an overpressure of 25 psi. The effective load applied to the shield by blasting is also determined by the duration of the blast pulse.

<u>High Velocity Fragments</u>. The impact velocities of particles on metals are divided into two regimes:

1. Hypervelocity - greater than 5,000 fps
2. High velocity - less than 5,000 fps

The velocity of fragments created by conventional blasting is far below the 5,000 fps range and lower portion of the high velocity regime applies to ADBS blasting conditions.

Within 20 ft of a blast hole, the fragment velocity is usually assumed to equal the initial particle velocity. A study of fly rock velocity from single cratering charges in sandstone showed a maximum fragment velocity of 260 fps (Duvall, 1957). Blasting tests using simultaneously detonated small stemmed charges of high explosives (Clark & Rollins, 1976) showed somewhat higher velocities. These higher velocity fragments may have been only loosely attached to the face, and therefore, exhibit a higher fragment velocity than would be expected from slabbing of larger fragments.

For simple cratering shots at the optimum scaled charge depth, the predicted fly rock velocity is less than 100 fps (Duvall, 1957). In the tunnel round tests conducted using small charges, small fragments (less than three inches in diameter) were thrown 50 to 100 ft. This indicates velocities of approximately 100 to 200 fps, and over 98% of the broken rock was within 14 ft of the blast.

Gross shield failure, characterized by bending or collapse, will not occur since the mass of the impacting fragments which reach the shield is small in comparison to the mass of the shield. Discing or flaking of the outby shield face will not occur because of low fragment velocity and strength, and because the test shield is constructed of high strength steel.

When mild steel projectiles penetrate mild steel plates, the penetration depth is predicted by (Healey, 1975):

$$Z = 1.63D\,(V_s)^{1.22} \qquad (3.1)$$

where

Z = penetration in inches

D = fragment density in pounds per cubic inch

V_s = fragment striking velocity in 1,000 feet per second

If shatter and ricochet (dominant characteristics of rock impacting steel) are ignored and a high impact velocity of 400 fps is assumed, the depth of penetration predicted by equation 3.2 is approximately 0.05-in. Previous tests with a fixed shield and small charges of high explosives showed only minimal pitting of the shield (Clark & Rollins, 1976).

<u>Vibration</u>. Vibration is of concern because of the possibility of gross blast shield failure in the form of bending or collapse, or local failure due to higher frequency response. Vibration amplitude, frequency, and duration determine the blast shield response. Natural vibration frequencies of thin flat plates of uniform thickness may be determined considering the shield as a mass-spring system (Harris & Crede, 1961). The blast shield used in the field tests, a 7 x 7 ft by 1/4 in. high strength steel plate with all edges clamped, was predicted to have a natural frequency of approximately 140 radians per second and a period of 44 milliseconds (22.7 Hz).

The period ratio is the relationship between the characteristic period of the forcing function (the blast pulse) to that of the oscillator (the blast shield). The anticipated blast pulse period for the charge sizes employed is 0.4 milliseconds (Clark & Rollins, 1976). Thus, the period ratio is 0.4/44 = 0.01. The maximax displacement, given as the ratio between maximum displacement due to vibration (X_M) and maximum displacement due to a static load (X_S), is approximately 1% of the displacement given by a static load equal to the peak blast pulse amplitude. This analysis shows how the pulse duration affects the pressure applied to the shield and why the pressure applied to the shield is relatively insensitive to variations in peak blast pressure unless the effective period of the blast pulse approaches that of the frequency of the shield.

Structural supports, utility fittings, access openings, and other shield appurtenances will tend to increase the resistance to vibration motion of the entire shield. Thus, blast overpressures and resultant vibrations induced in the shield are far below critical. For small charges, the factor of safety for bending failure of the shield is greater than 10.

Noise. Tests conducted in an unobstructed short tunnel (Clark & Rollins, 1976) recorded a maximum noise intensity of 114 dBA at a distance of 52 ft from the detonation of several small charges of high explosives. When a steel shield with plywood on the inby face and four lead-vinyl curtains outby were placed in the tunnel 15 ft from the face, blast noise was reduced to less than 110 dBA at 30 ft from a similar blast.

Allowable Blast Effects

Air Blast Pressure. Recent studies (Army, 1969) concluded that the critical human organ subject to blast pressure injury is the ear. The threshold of ear drum rupture occurs at 5 psi, while a 50% casualty rate occurs at 15 psi. For continued exposure, air blast overpressures must be less than 2 psi if human comfort is to be maintained.

Small Charge Blast Characteristics

The simultaneous detonation of several small stemmed charges (usually from four to eight) of high explosive produces air blast, high velocity rock fragments, vibration, and noise. The amplitudes of these four parameters dictate the design of the blast shield and is based on the following:

1. Structural strength to resist stress waves (precursors) which result from short period blast waves or high velocity fragments impacting the shield.

2. Shield strength, rigidity, and restraint to resist flexure of movement of the entire shield due to air blast overpressure or fragment impact.

3. A method of sealing the shield to the invert, ribs, and back so as to prevent blast overpressures and high velocity fragments from entering the outby work area, and to attenuate noise intensity to allowable magnitudes. The critical values of these parameters are determined largely by the magnitude and duration of impact (see also Appendix A).

Air Blast Pressure. Analysis of air blast overpressure from underground mine production blasts indicate that a cube root law describes the pressure decrease with distance from the blast (Olson & Fletcher, 1971):

$$P = 4.9\ (10)^3\ (D/W^{1/3})^{-2.15} \qquad (3.2)$$

where

P = overpressure in pounds per square inch

D = distance from blast in feet

W = zero delay charge weight in pounds

Studies conducted previously in a short tunnel yielded similar supporting results (Clark & Rollins, 1976). The pulse magnitude and duration observed in these two studies, which were conducted with stemmed charges, may be low due to the influence of cross cuts and nearness to a tunnel portal, respectively.

High Velocity Fragments. For safety reasons, all blast fragments must be stopped by the blast shield. It has been demonstrated that the depth of penetration of a high velocity rock fragment into the steel front face of the blast shield is small. For high velocity fragments, the ratio of the fragment mass to shield mass is small, and impact does not cause shield movement, cratering, or bending.

Vibration. Experiments with animals (Rice & Zepher, 1967) have shown that higher frequencies are less tolerable than lower frequencies. More specifically, higher frequencies are more tolerable at lower amplitudes, and peak accelerations are more tolerable at higher frequencies. It may be inferred that the effect on humans is similar. However, since the blast shield period is large and the vibration amplitude is small, the effects on humans of blast induced vibration will be slight and within tolerable limits.

Noise. Noise exposure standards have been established for both general industry and the mining industry (MESA, 1974; OSHA, 1976). Both federal codes set equal exposure limits (Table 3-1).

TABLE 3-1

PERMISSIBLE NOISE EXPOSURES

Duration Per Day (hours)	Sound Level (dBA) - Slow Response
8	90
6	92
4	95
3	97
2	100
1-1/2	102
1	105
1/2	110
1/4 or less	115

Field Test Blast Shield

The test shield was designed to attain four major goals:

1. Insure structural strength to resist predicted fly rock impact and blast overpressures.

2. Test the effectiveness of a support mechanism and sealing system which would prohibit the passage to the outby work area of fly rock, dust, gaseous products of detonation, overpressure, and excessive noise.

3. Test a method of mechanically expanding and contracting the shield as will be required on a working model.

4. Provide a shield which could be easily assembled and disassembled to facilitate transportation into and out of the heading.

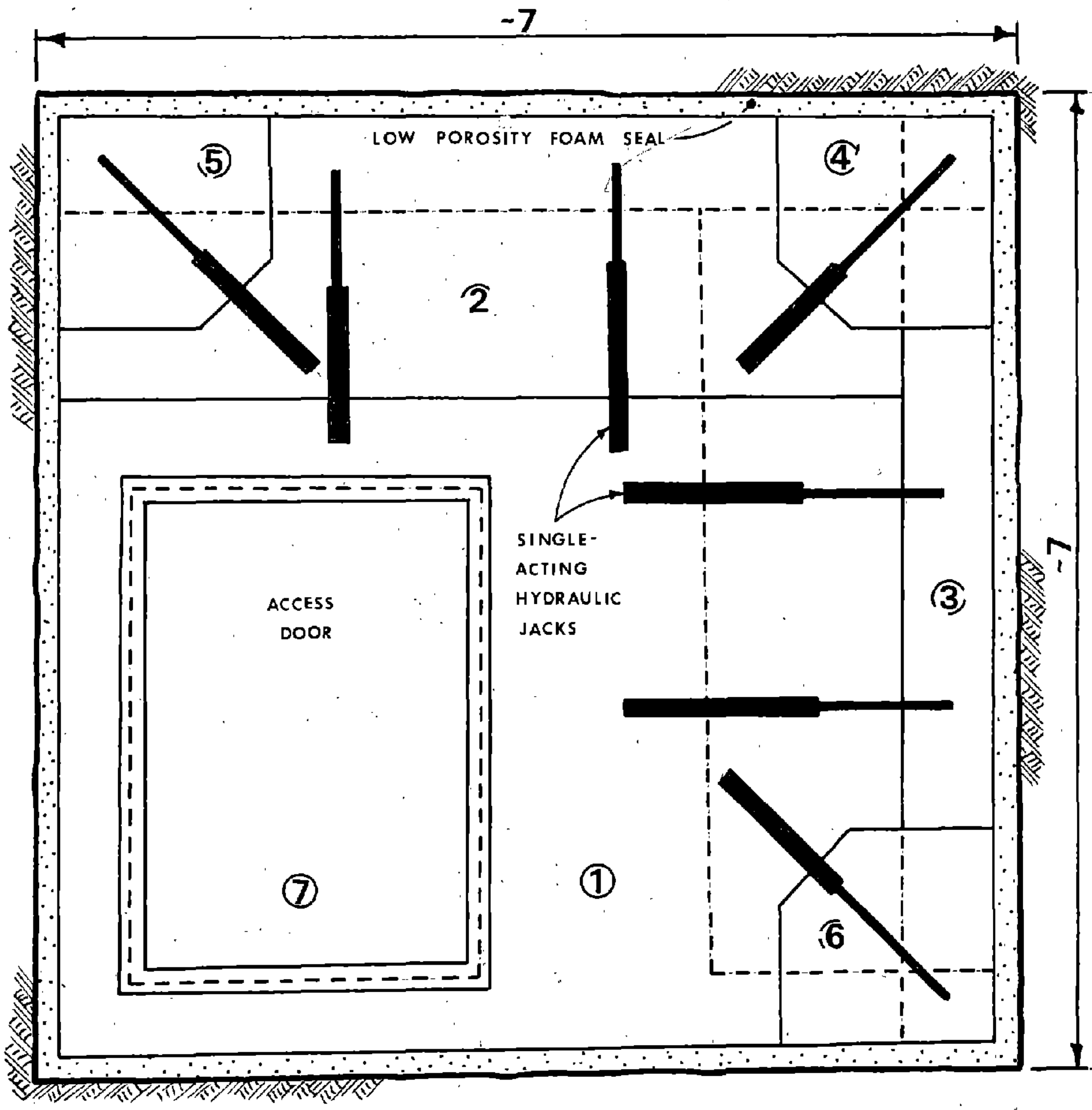
A 1/4 in. thick, high strength (Fy = 50 ksi) steel plate was the chosen shield material because of its prior successful application under similar test conditions (Clark & Rollins, 1976) and because analysis indicated a satisfactory factor of safety against bending failure. The shield consisted of seven overlapping sections. The sections were assembled as shown in Figure 1 and fabrication plans are contained in Appendix B. Pressure against the floor, ribs, and back was obtained by forcing each shield section against the main shield section and thence against the opposite surface of the drift. The force was obtained by hydraulic jacks, which may be considered for use in the fully automated shield.

In previous field tests of a similar blast shield (Clark & Rollins, 1976), fragment and air blast impact induced vibrations in the shield which resulted in a ringing noise. The impact and ensuing vibration were not harmful to the shield, however, the noise level was high. In order to mitigate this noise level, 1/4 in. thick synthetic rubber lining (used conveyor belt) was bolted to the inby side of the shield.

Sealing the shield to the floor, ribs, and back was accomplished by cold-bonding a low porosity foam rubber to the edge of the shield (Figure 3-1). When compressed, this material deformed to fill the irregularities of the rock surface.

FIGURE 3-1

GENERAL CONFIGURATION OF FIELD TEST BLAST SHIELD (OUTBY SIDE)



Note: The numbers indicate the order in which the sections were erected in the field. Sections 2 through 6 were then jacked into place.

A removable door was provided in the shield to allow access for drilling, loading of explosives, wiring of the round, and mucking. The door, also fabricated from 1/4 in. thick high strength steel and faced with lining, was flanged to provide effective sealing strength. An overlapping flap of rubber belt lining reinforced the seal.

This blast shield design, while not intended to be totally automatic, as will be ultimately required, contains most of the basic elements of an automated shield. The method of forcing the sections against each other and the tunnel periphery utilizing hydraulic jacks is proposed for a final shield as is a heavy foam rubber seal which functions as a sealing mechanism. A noise attenuating material, such as conveyor belting lining the inby side, is necessary to reduce blast induced high frequency noise which would otherwise be transmitted through the shield.

Blast Shield Erection. Field testing of the test blast shield was conducted in the Colorado School of Mines Edgar Mine in a cross cut approximately 7 x 7 ft in section.

The shield was located at a site 13 ft from the face for the following reasons:

1. The distance from the face to the shield is less than that for a machine-mounted shield.

2. The small floor area simplified hand mucking.

3. The drift at this location was roughly square in cross section.

The site was prepared by blasting protrusions and filling large voids around the periphery with concrete held in place with bolt and wire mesh reinforcement. Muck overlain by sand was used to cover the existing railroad tracks.

The shield was erected (Figure 3-1) with seven, ten ton single-acting hydraulic jacks extended until their full thrust was reached. This compressed the high density foam rubber, originally 6 in. thick, to an average thickness of less than 1 in. The maximum anticipated blast overpressure predicted by equation A-1 was 25 psi, assuming six holes each loaded with one-quarter pound of high explosive.

It was found that the total predicted peak blast force of 88 tons on the shield exceeded the frictional resisting force provided by the jacks, so six number 8 reinforcing bars were installed in drilled holes in the wall, roof, and floor and wedged against the shield.

Shield Tests. Field testing of the blast shield was carried out concurrently with tests of a 24 in. deep small charge round (Hanna, 1978). This arrangement exposed the shield to blast effects smaller than those anticipated. The 24 in. deep rounds were excavated by several shots. Each hole of the shot was loaded with a predetermined weight of slurry explosive, the weight being established by the volume of rock to be broken and an assumed powder factor. The holes were stemmed with caulking material. The explosive was initiated with 25 grain per foot detonating cord with the length of cord from each hole of the shot connected. The shot was initiated by a single blasting cap. This method caused all 8 holes of the shot to detonate simultaneously.

The first shot moved the top right corner of the shield several inches. Shield movement was caused by the lack of adequate frictional resistance against the drift periphery, and because of the fact that all four holes bootlegged, producing high overpressure.

To prevent continued shield movement, lengths of reinforcing bar were wedged into drill holes outby the shield, and the shield was timbered in place (not shown). As noted earlier, this method of supporting the shield at seven points, three bars along each side and one at the top, proved reasonably effective. However, high overpressure due to bootlegged shots required frequent replacement of reinforcing bars and timber, which failed in shear and bending, respectively. The first shot of the second 24 in. deep round (all eight holes of the V-cut) bootlegged, which caused all six bolts of the main shield section overlap to fail. These 1/2-in. diameter mild steel bolts were replaced with high strength bolts. This was the only shield component failure during testing. The problem of restraining shield creep was solved by placing a loaded granby car against the shield during blasting.

Test Results. In addition to round testing, both noise level outby the shield and particle velocities in the rock were measured (Appendix C).

Of the ten instrumented shots, four bootlegged one or more holes. Sound levels recorded five feet outby the shield ranged from 96 to 112 dBA. When all of the holes in a shot detonated properly, the maximum noise level was less than 108 dBA. This noise level compared well with noise levels recorded earlier for 18 in. deep small charge tests (Clark & Rollins, 1976). However, in the earlier tests, a fixed shield with 1/4 in. plywood inby and four lead-vinyl curtains outby was used (Appendix A). This arrangement, while proving the viability of the shield concept, is not practical for the ADBS. The tests conducted at the Edgar Mine used a newly designed moveable shield faced with inexpensive rubber conveyor belting as noise attenuation lining. Thus, adequate noise reduction can be attained by a lined shield designed to fit the shape of the drift, which was one of the key shield design criteria.

When the shield was braced to prevent movement, the compressed low porosity foam provided a good seal between the shield and the drift periphery. This was evidenced by the lack of smoke or dust in the drift outby the shield after the detonation. Usually, no particulate matter could be observed in the air, and only a faint odor of explosive was detected. The lack of significant quantities of smoke or dust indicates that the shield also adequately contained the blast overpressure. Thus, another criterion for the shield was met: dust, gas overpressure, and fly rock were contained by the shield.

The method of extending and retracting sections of the shield using hydraulic jacks worked well. One defect of this design was the tendency of one of the sections to laterally separate from the main shield section (Figure 3-2). This was caused by the moveable section being on the opposite side of the main section from its hydraulic jacks and only one set of restraining bolts in each slot. The separation did not, however, provide a path for the escape of blast overpressure and dust and can easily be rectified. The rubber sheeting apparently sealed the separation.

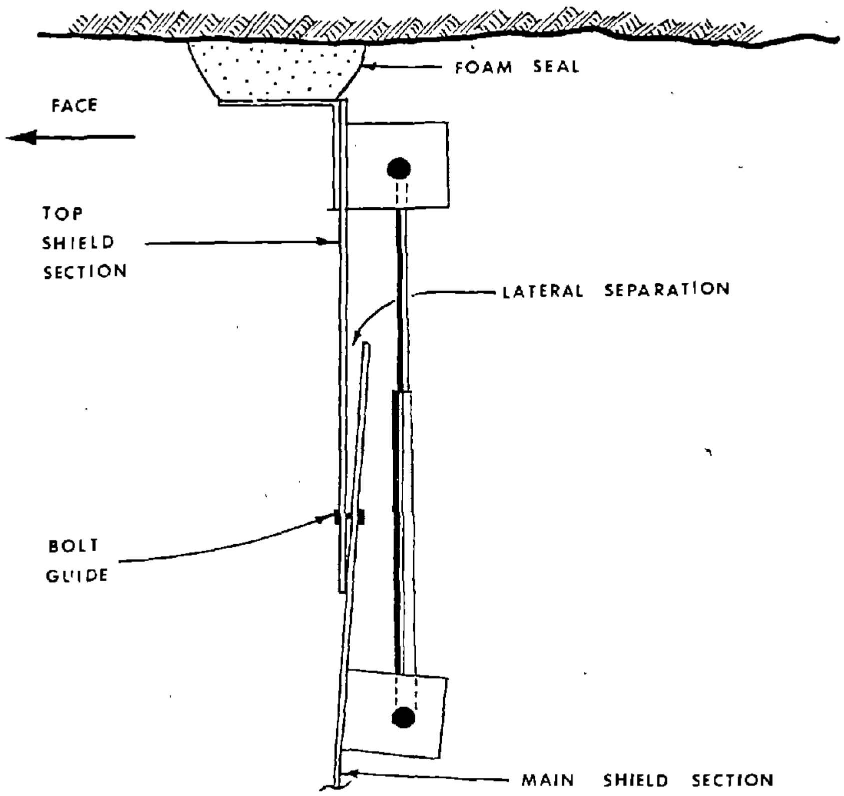
The force provided by the hydraulic jacks against the drift periphery was insufficient to prevent shield movement caused by excessive blast pressure. This potential problem was recognized in the design, and was prevented (the field tests showed that). In order for the peripheral seal to be effective, shield movement in the drift must be prevented. This can be accomplished by generating a notch in the periphery with the outside blast holes and/or with steel pins.

Recommendations

The rubber lining bolted to the inby side of the shield significantly reduced the outby noise level. The used conveyor belt lining employed also withstood fly rock impact well. In over thirty shots fired against the shield, only a few small holes (less than 1 in. in diameter) were made in the lining when it was not held tight against the shield. Such a material should be planned for the final shield design. Instead of replacing abraded lining on the shield, a second layer of lining may be fastened over the

FIGURE 3-2

CROSS SECTION OF THE LATERAL SEPARATION OF SHIELD SECTIONS
(Operating Face to Left)



damaged section. This maintenance method will also provide additional noise reduction.

The movement of parts of the shield probably preclude the use of an acoustic rubber sheeting on the outby side. This material functions properly only when it is cold-bonded to steel (Gates, 1977), which could not be done because of the wear as one shield section moves over another. However, a lining such as conveyor belting bolted to the outby face would provide additional noise reduction. This should be investigated in future testing.

The low porosity, high density foam rubber used to seal the shield to the drift in the field tests worked very well. The same material, a common furniture rubber, or equal would be incorporated on the final shield. A better bonding method or material may be required to attach the foam to the steel shield. The 4 in. wide by 6 in. deep cross section of foam was effective in the field tests.

As indicated above, the hydraulic jacks did not provide enough force to resist shield movement, but they did completely compress and successfully bond the foam seal in place when shield movement did not occur. The minimum force required to adequately compress the foam seal was not determined.

The bolt guides and slots to direct the movement of the shield sections were effective and such an arrangement should function on an operational shield.

One plan is for two separate systems to hold the shield securely in the drift. The first mechanism is to connect the shield to the chassis by means of two booms (Figure 3-3) mounted on the chassis. Each boom is equipped with two dual-acting hydraulic jacks to position the shield at the desired location in the drift. Tests of the blast shield showed that restraining the shield near its center of mass is very effective in reducing or preventing motion.

In the design for a 12 x 12 ft drift (Figure 3-3), the chassis connection points are located on the shield so as to provide three equally spaced spans. Plate deflection theory shows that the chassis connection will carry approximately 70% of the pressure load, and the periphery of the shield will carry the remainder. With the shield 20 ft from the blast, the peak overpressure is approximately 10.5 psi (Appendix A) or a total of 110 tons. Thus, the chassis load is 77 tons or 19.1 tons per support. High strength (F_y = 46 ksi) structural tubing will be effective for the booms and supports.

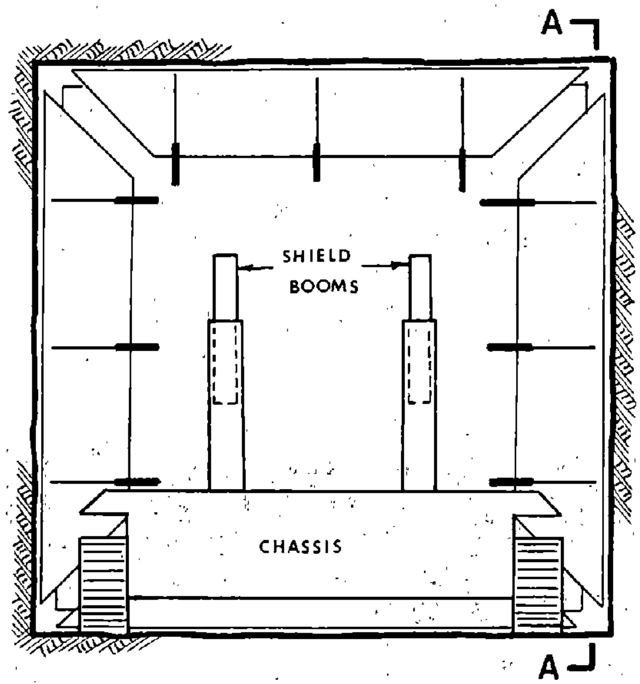
The movement of the upper jack on each boom will be operated by a single control so that both jacks move together. The jack will be pin-connected to the boom and to an arm which slides within the boom. This configuration allows the bending stress to be carried by the boom and slide instead of the jack. The two lower jacks, also operated by a single control, will be pin-connected to the boom support and the shield. Thus, only axial force will be applied to the lower jacks.

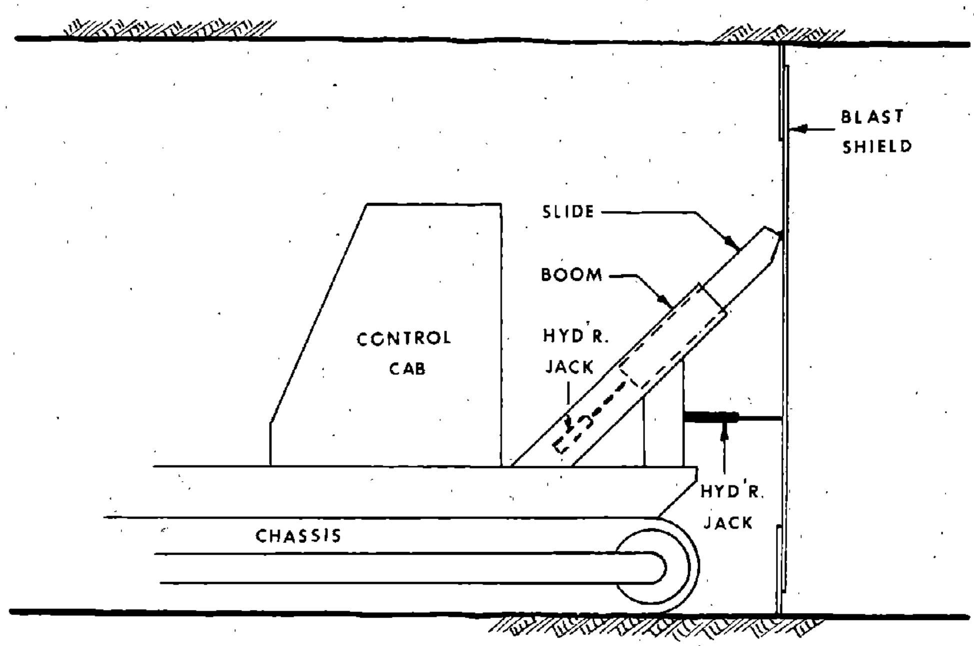
The booms, slides, and supports shown in Figure 3-3 were designed using the load, dimensions, and the steel shield indicated consisting of 1/4 in. plate. Allowable tension, compression, bending, and shear stress are per current AISC specifications (AISC, 1977) for structural steel design. Design caluclations and details are contained in Appendix B.

The second system for holding the shield is the peripheral support. One component of peripheral support will be provided by the frictional resistance developed at the rubber seal/rock interface. For the dimensions given in Figure 3-4 a 4 in. wide foam seal, and an assumed friction coefficient of 0.1, the frictional resistance developed can be calculated (Table 3-2).

FIGURE 3-3

INBY BLAST SHIELD TO CHASSIS CONNECTION

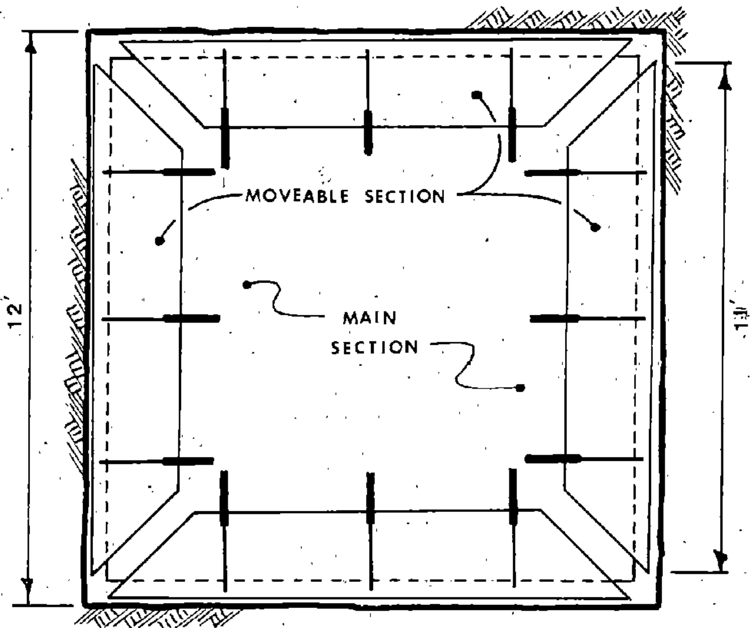




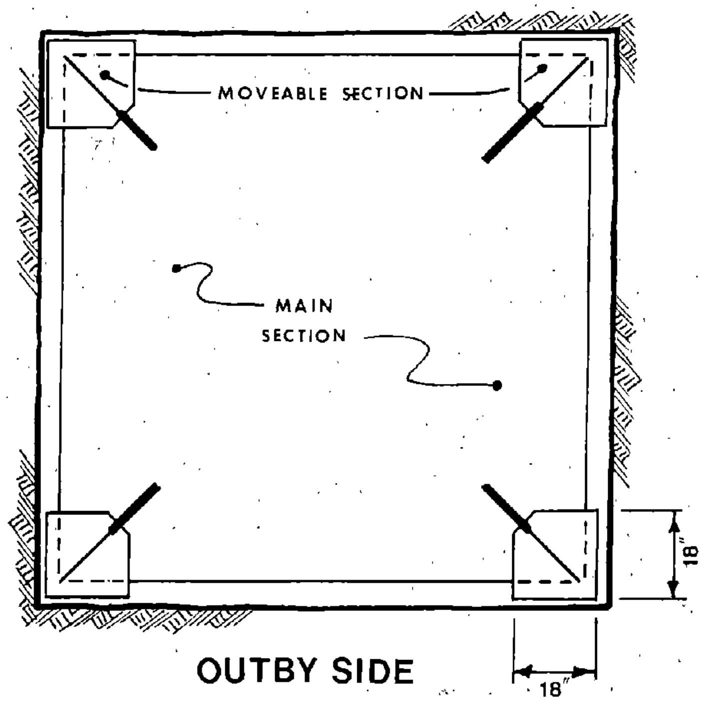
SECTION A-A

FIGURE 3-4

FINAL SHIELD DESIGN - MOVEABLE SECTIONS



INBY SIDE



OUTBY SIDE

TABLE 3-2

SHIELD FRICTIONAL RESISTANCE
(12 x 12 ft drift)

Shield Section	Contact Area ($in.^2$)	Pressure/Section (psi)	Frictional Force (lbs)
Sides, back or invert	528	133.6	7054
Corner	144	138.9	2,000

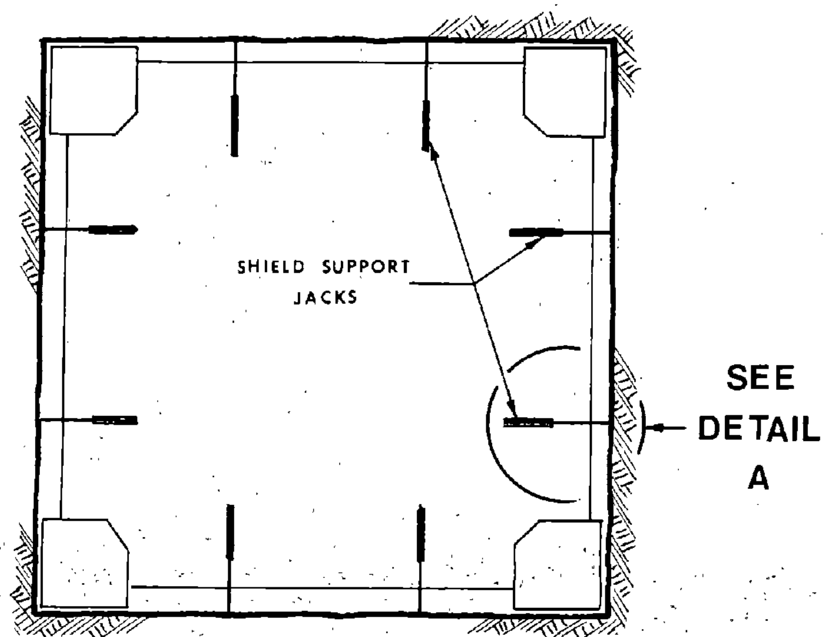
Thus, the total frictional resistance is approximately 16 tons. Out of the 110 ton overpressure force, 93 tons will be resisted by the chassis and frictional resistance, leaving 17 tons to be resisted by the second component of peripheral support. This will be provided by eight peripheral jacks, located on the outby side of the shield (Figure 3-5). Each jack cylinder will have a pointed end which acts as a stinger when pressed against the rock. Assuming a contact area of 0.25 sq in. and a coefficient of friction of 0.1, the functional resistance per jack is 4 tons or 32 tons total.

The muck discharge apparatus (Figure 3-6) is designed to be used in conjunction with a conveyor system outby. The door will be hydraulically activated and will be faced with rubber lining. The manway door will be hinged and swing in; the latch mechanism will be operated from either side of the shield. Water, ventilation, electric, and communication lines pass through the shield. These utility lines must be caulked or grommeted securely in the shield opening.

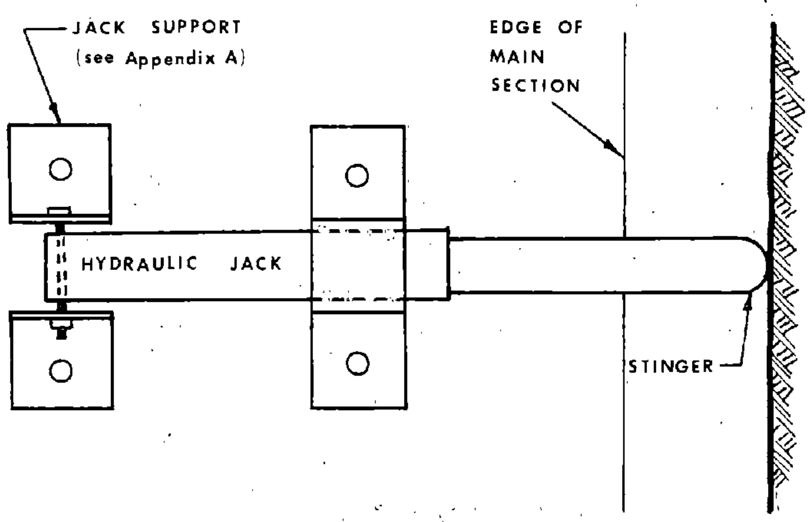
The shield design described above contains all the elements necessary to function efficiently with the ADBS. This design, combined with the controls and safety features described above, is safe, effective, and relatively inexpensive to fabricate.

FIGURE 3-5

SHIELD SUPPORT JACKS



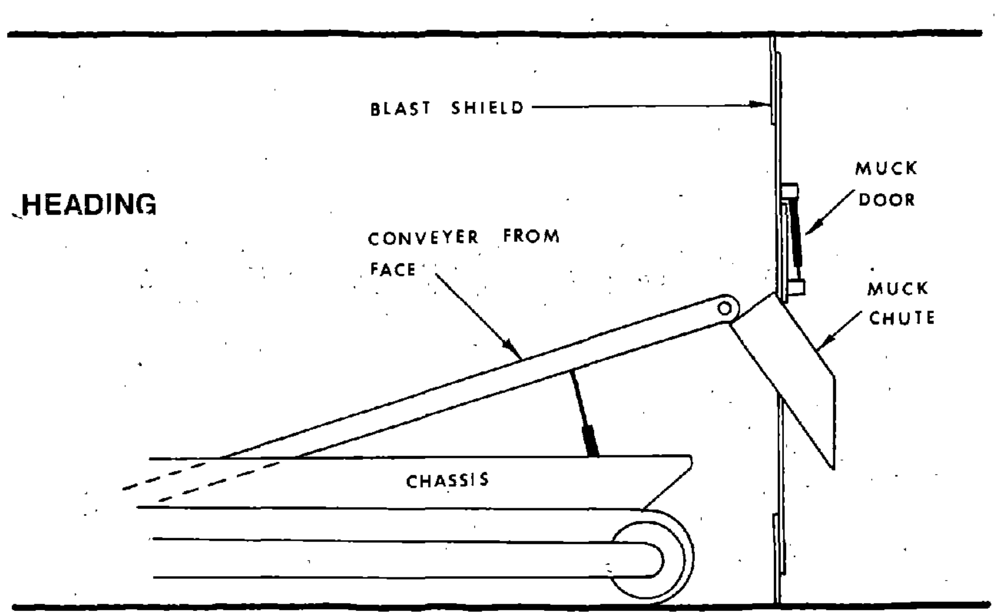
OUTBY SIDE



DETAIL A

FIGURE 3-6

BLAST SHIELD MUCK DISCHARGE CHUTE



CHAPTER 4

CONTROL CAB

Introduction

The operations will be monitored and controlled by operators in a control cab located to the rear of the ADBS equipment and inby the blast shield. Drilling, using boom mounted hydraulic drifters (Chapter 5), and explosive loading and detonation will be controlled from within the cab. Because the control cab will be located near the face, the operators must be shielded from the air blast and flying fragments and provided with a comfortable working environment. Safety and comfort requirements dictate that the cab be totally enclosed.

As indicated in Chapter 1, locating operators at the face has both advantages and disadvantages. The principal advantage is that an operator can continuously monitor the drilling, loading of explosives, blasting progress, and rock conditions. Most of the components are proven and relatively uncomplicated, and few failures are expected to occur. Continuous, direct visual inspection permits rapid preventive maintenance, decreased down time, and more efficient operation.

In spite of improvements in drilling equipment, explosives, and tunnel blast rounds, optimum drill hole spacing and powder factor are determined by geology especially for small charges. Interpretation of the geologic conditions and evaluation of their effects on blasting requires both experience and close observation of the face. The small charge method of blasting allows the shielded operator to monitor the results of each shot. This information, combined with an overall assessment of the face geology, permits refinement of succeeding round patterns or of hole placement in the same round. Continual round design refinement is critical particularly in the peripheral holes if excessive overbreak and fracture into the walls are to be avoided.

The chief disadvantage of having operators at the face is that such an arrangement (men in protective cabs near the face during blasting) is an untried innovation. Currently, government regulations require that "all persons shall be cleared and removed from areas endangered by the blast" (MESA, 1971). An interpretation of this regulation with regard to the control cab has not been requested. An additional disadvantage may be in initial operator acceptance.

Cab Design Parameters

Two basic considerations are required in the design of the control cab: shielding to resist blast effects and environmental control to provide operator comfort. The size will be based on the number of men per cubic space for controls and other factors discussed below.

Blast Shielding. The control cab must protect the operators and controls from fly rock, overpressure, and excessive noise. The design conditions for shielding the control cab (Table 4-1) assume that the cab is a minimum of 15 ft from the face and that each blast consists of six stemmed holes loaded with one-quarter pound of slurry explosive per hole based on detailed developement of the blast parameters (Appendix A).

TABLE 4-1

BLAST PARAMETERS FOR CONTROL CAB DESIGN

Blast Parameter	Magnitude
Overpressure	19.3 psi
Fragment penetration	0.05 in.
Noise level	115+ dBA

The cab is designed to resist failure by collapse due to overpressure. Also, the overturning force produced must be resisted by the cab weight and the mechanism connecting the control cab to the frame. The effect of fly rock impact with regard to penetration is insignificant (Appendix A).

The attenuation of blast noise intensity to allowable Federal requirements will probably require two noise reduction systems: both interior and exterior. Ideally, sound level will be reduced to below 90 dBA, which is the maximum for 8 hour exposure (Appendix A).

Cab Environment. The control cab must be totally enclosed in order to exclude gases, noise, and overpressure entering, and fresh air must be introduced and exhausted. Also, depending on tunnel environment, heating and/or air conditioning may be required.

Interior Size Design. Based on the production advance rate (Chapter 10 Table 10-6) the number of drills can be determined (Table 4-2). One man can operate two drills and one load/blast system operator is required for each drill operator, while an operator is not required for the mucking system. Mechanics, electricians, and other maintenance personnel may be stationed outby the tunnel shield.

TABLE 4-2

OPERATOR REQUIREMENTS

Tunnel Diameter (ft)	Advance Rate[1] (ft/day)	No. of[2] Drills	Number of operators
12	82	1	2
16	76	2	2
20	71	3	4
24	65	4	4

[1] Production rate of advance in sandstone

[2] Based on 36 in. deep round (see similar analysis - Chapter 7)

When one drill operator and one load/blast operator are required, a single cab can be used. For larger diameter tunnels, two cabs are required with one drill and one load/blast operator in each. Two cabs are necessitated due to space limitations imposed by the muck conveyor.

The drill and load/blast system is planned to be manipulated by conventional hydraulics, the controls for which can be installed on a console in front of each respective operator. Console lights will be required as well as safety lockout electronics. Each operator will have an adjustable chair.

Cab Design

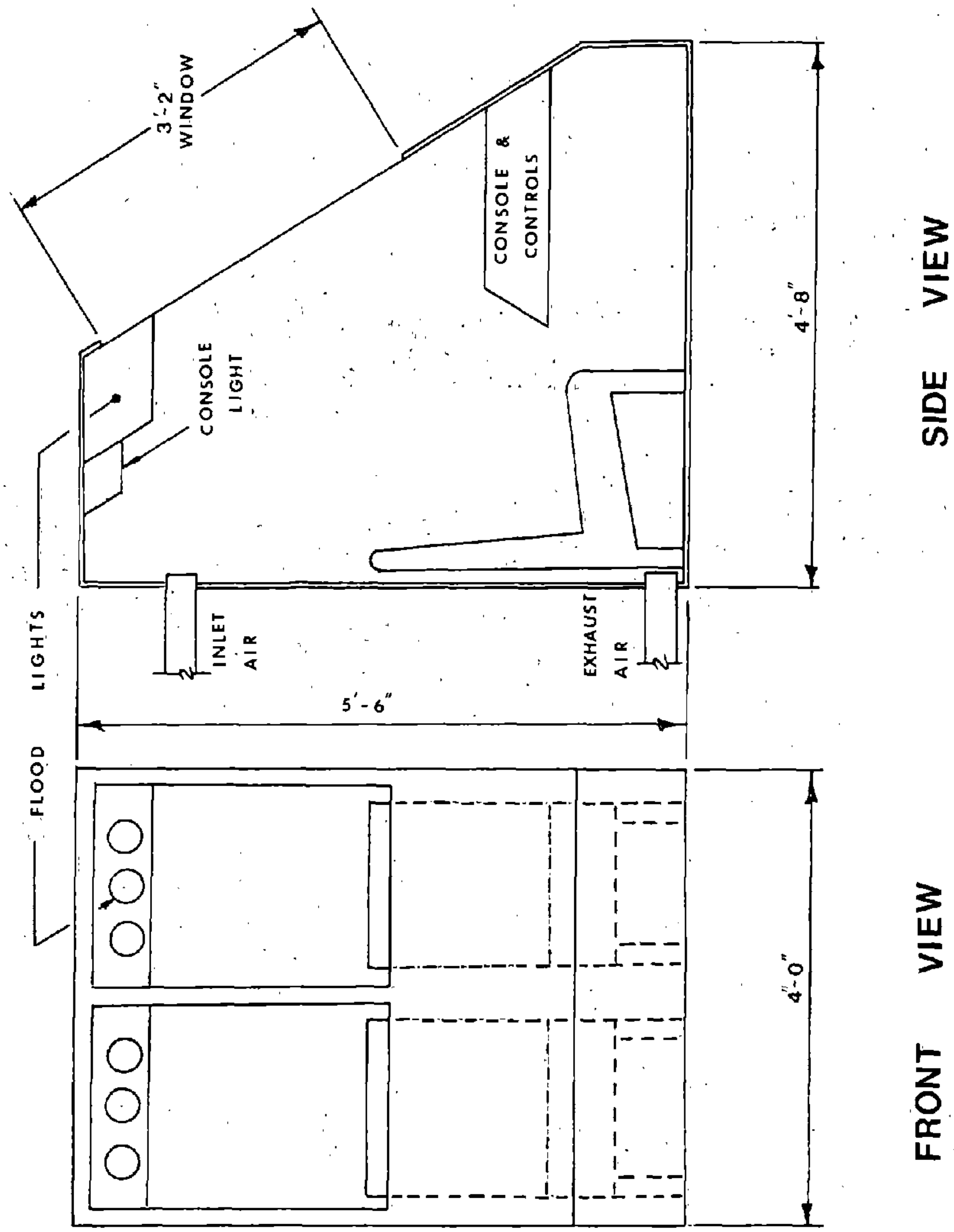
High velocity impact analysis (Appendix A) and blast shield testing (Chapter 3) indicate that high strength steel plate 1/4 in. thick will safely resist fly rock penetration. This material will form the cab shell, and during blasting, will totally enclose the operators. All joints will be full penetration welded so as to have strength at least equal to that of the high strength steel plate. Features of the cab interior (Figure 4-1) are placed to insure operator comfort. The windows are sized to allow visibility of the total face so that the tunnel face and ADBS equipment may also be monitored by each operator from the cab.

The face will be illuminated by spot lights mounted inside the cab against the roof. These lights are enclosed to prevent glare from the cab windows. Additional interior lights will illuminate the controls so the operator will not require cap lamps.

The cab will be ventilated by fresh air injected near the cab roof. The air source is the fresh air directly outby the shield, provided by exhaust ventilation (Chapter 5). Cab exhaust air will be discharged to the area inby the shield. Volume requirements can be estimated from the data presented in Table 6 in the work by Zabetahis (19??).

FIGURE 4-1

CONTROL CAB INTERIOR



The operators are assumed to be working moderately hard, thus consuming 1.74 cu ft per operator per minute or approximately 3.5 cfm total. The cab has a volume of approximately 66 cu ft. Thus, at the rate of 3.5 cfm, the air turnover rate is approximately 3 times per hour. A more desirable rate is 6 per hour or 6.6 cfm.

TABLE 4-3

APPROXIMATE RATE & VOLUME OF HUMAN RESPIRATION

Degree of Activity	Respirations (per minute)	Air Inhaled Per Respiration (cu in.)	Air Inhaled Per Minute (cu in.)
At rest	16	30	480
Moderate	30	100	3,000
Vigorous	40	150	6,000

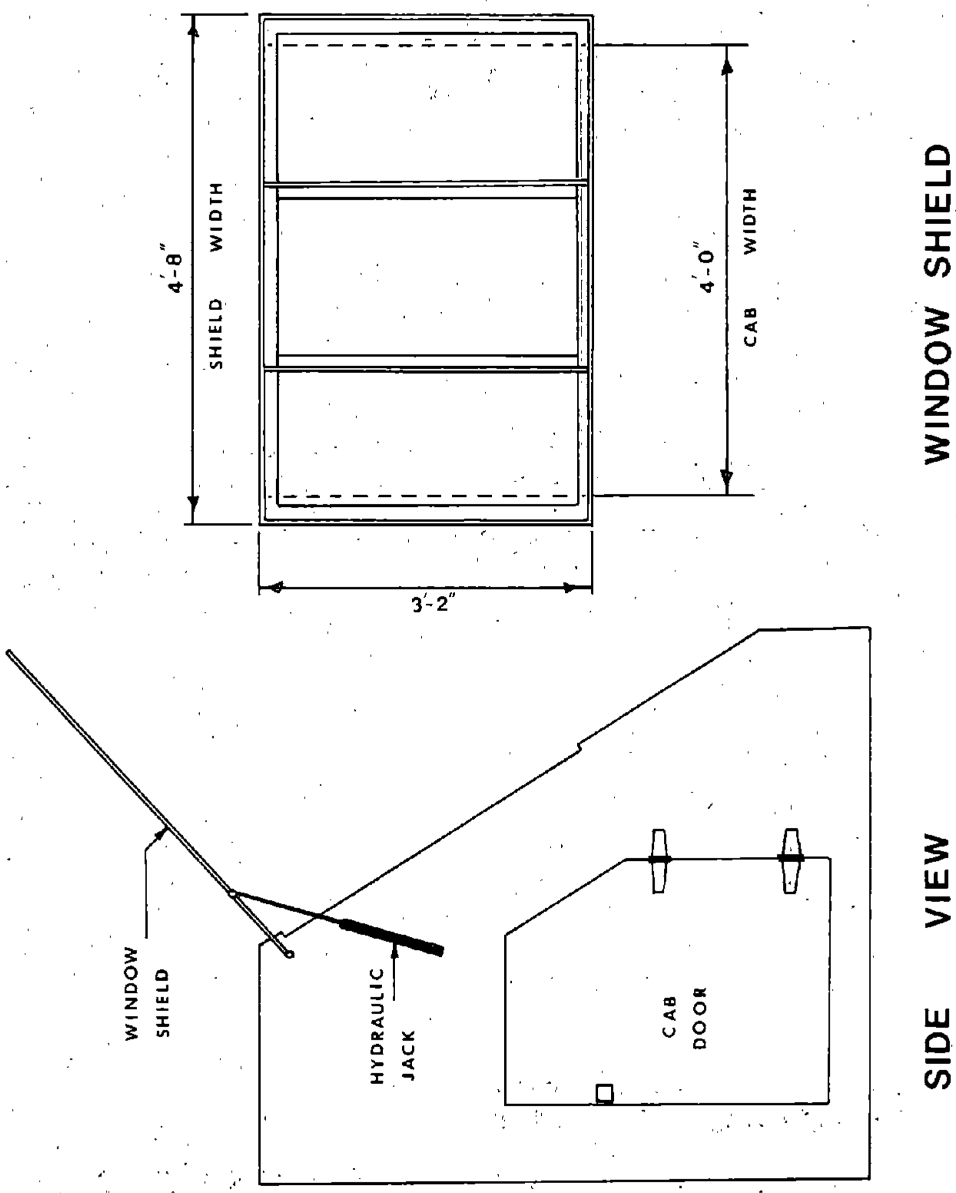
One door will be provided for each cab and will be loacted on the side adjacent to the conveyor. The door will swing toward the face (Figure 4-2) and will be provided with four interior dead bolts to secure the door tightly closed. During blasting, the windows will be shielded by a 1/4 in. thick high strength steel plate which will be operated by two dual-acting hydraulic cylinders actuated from within the cab. While drilling and loading are in progress, the shield will be raised out of the operators' sight (Figure 4-2).

Noise Attenuation. Based on the noise level measured outby the blast shield, as described in Chapter 3, a 1/4 in. thick layer of rubber sheeting will provide adequate noise attenuation. However, in order to insure noise reduction to lower levels and to reduce the startle effect produced by the blast, a second noise attenuation system will be included.

Several major rubber producers currently market rubber sheeting specifically fabricated to attenuate noise levels around machinery. One manu-

FIGURE 4-2

CONTROL CAB: DOOR & SHIELD



factures, a 1/4 in. thick synthetic rubber sheeting that, in tests, reduced noise as follows (Gates, 1977):

Noise level recorded behind 1/8 in. thick steel plate	104 dBA
Noise level recorded behind 1/8 in. thick steel plate with 1/4 in. synthetic rubber cold-bonded to the steel on the side opposite the noise impact	89 dBA

This reduction represents an 85% decrease in noise level.

A 1/4 in. thick synthetic rubber sheeting will be cold-bonded to the steel surfaces of the cab's interior walls, roof, and floor. This material is durable enough to withstand wear when placed on the floor (Gates, 1977). A layer of this rubber will also be bonded to the underside of the window shield, which will also protect the glass from the effects of vibration loading.

Thus, the cab interior will be protected by two noise suppression systems: (1) an exterior system consisting of a durable, 1/4 in. thick rubber lining (e.g., conveyor belt as used for the experimental blast shields) which, when the window shield is closed, will totally cover the steel cab shell, and (2) an interior system consisting of a 1/4 in. thick layer of synthetic rubber with a high noise reduction capability.

All hydraulic lines and utilities (ventilation, electric, etc.) will be shielded and grommeted and/or caulked to the frame opening. This will insure an air tight cab enclosure.

Cab Controls

Controls in the shielded control cab will be of two types:

1. Production controls for the drills, load/blast system, muck system, and
2. Safety controls for personnel protection both inby and outby the shield during blasting.

The controls for blasting will be accompanied by warning lights located on the console. Fail safe, interlocking blast controls will also be installed on the console as required.

Production Controls. Controls for production equipment will consist of the following:

1. On/off switch for chassis hydraulic pump which operates crawlers and automatic mucking system;

2. On/off switch for drill(s) and load/blast system(s) hydraulic pump;

3. Hydraulic controls to move each crawler forward or backward independently;

4. Hydraulic controls to manipulate the drill(s), load/blast system(s), and hydraulic booms;

5. On/off switch for automatic mucking system;

6. Hydraulic controls to retract the shield support mechanism (Chapter 2) and to compress the shield section (same controls used to reinstall shield in the new location in the drift);

7. Hydraulic controls to manipulate the shield boom (positions the blast shield in the drift).

Safety Controls. Controls to manipulate safety devices before, during, and after blasting will consist of the following:

1. Firing control button which initiates the simultaneous detonation of several small charges of explosives. This switch will be interlocked so as to be activated only when all of the following features are correctly positioned;

2. Open/close switch to operate muck conveyor door on blast shield (closed during blasting);

3. Open/close switch to operate control cab window shield (closed during blasting);

4. Open/close switch to operate butterfly valves in inlet and outlet ventilation lines to control cab (closed during blasting);

5. Open/close transducer on manway door in blast shield (closed during blasting);

6. Open/close switch to operate butterfly valve on exhaust fan (closed during blasting).

Each component listed above will be provided with a transducer and warning light located at the appropriate open/close switch to verify proper operation. The warning light will glow dimly to indicate that it is operational. In the event of a malfunction, the light will glow at full brilliance. This malfunction circuit will also open-circuit the detonation mechanism.

The protection control system will function as follows: After one set of holes has been drilled and another set loaded, the operators then close all doors and valves so as to isolate both the area inby the shield and the control cab. All the control lights will glow dimly, indicating that all valves and doors are properly secured. The light adjacent to the firing control button is glowing at full brilliance showing that the circuit is complete and that the round can be detonated. If a door or valve had not been properly seated, its warning light will have indicated the malfunction and open-circuited the firing system.

The protection control circuit is conceptual only. However, it does effectively monitor the components necessary to assure a safe detonation, both for the crew outby the shield and in the control cab. Such a system is proven, reliable, simple, and flexible. For example, should depressurization of one or more hydraulic jacks on the shield occasionally allow air blast overpressure to escape the inby side of the shield, individual transducers could be attached to the shield jacks to verify full pressurization. A prototype ADBS may indicate the need for additional controls on the firing circuit.

CHAPTER 5

CHASSIS AND AUTOMATIC MUCKING, HYDRAULIC DRILL, ALIGNMENT CONTROL, VENTILATION

Introduction

The Automated Drill and Blast System (ADBS) will consist of five major components:

1. Chassis and automated mucking equipment
2. Hydraulic drill(s)
3. Explosive loading and detonation system(s)
4. Blast shield
5. Control cab(s)

The explosive loading and detonation component is conceived as consisting of a hydraulically-driven, drifter-type boom, and a cap and explosive magazine coupled to an automated loading tube; research, design and testing of this equipment will be performed as another phase of this project. A similar system is currently being field tested (Peterson, et al, 1976). In addition to these components, continuous horizontal and vertical alignment and tunnel ventilation must be incorporated in the system.

Chassis and Automated Mucking

Automated semi-continuous mucking is a requisite for effective ADBS operation. The production rates of advance in different rock types (Chapter 10) are based on muck being collected and removed from the face as it is produced. Complete muck removal is most important when the lifter holes are to be drilled, and compressed air sweeping system may be required.

Currently there are successful automated and semi-automated continuous types of mucking equipment available for hard rock mining and tunneling. Two features are common to such systems: A gathering mechanism to pull the muck onto an apron and a conveyor which moves the muck from the apron to a discharge behind the machine. At the rear of the mucking machine or system several methods can be employed to transport the muck. Typical methods include rail haulage, conveyors, and rubber tire vehicles.

A continuous semi-automated mucking system currently in use is the "Haggloader," manufactured in Sweden (Hagglund, 1977). This equipment, whose only function is mucking, operates by pulling the muck ahead of the gathering arms, thus clearing the full width of the drift. Two operator-guided gathering arms direct the muck toward a chain conveyor. The machine only moves forward when mucking, either on rails or crawlers. Two constraints may prevent this particular mucking system from being incorporated into the ADBS. The primary reason is that the gathering arms interfer with the drilling and load/blast components of the ADBS. An additional disadvantage is that this machine is only semi-automatic, thus requiring an additional operator.

Alpine's "ROC-MINER" (Alpine, 1977) incorporates a continuous, automatic mucking system. This muck collection and removal system operates in much the same fashion as the "Haggloader," except that no operator is required for mucking. Two cam-operated gathering arms continuously sweep the apron and move broken material to the chain conveyor.

This system has two advantages relative to the ADBS. First, the apron and gathering arms have a low profile which will permit complete access to the face (including space for the lifters) at all times. The second advantage is that no operator is required. This allows a reduction in manpower and obviates the need for additional cab space.

The frequent detonation of small charges will produce muck semi-continuously. The maximum in situ volume to be excavated will be approximately 1.5 cu yd per shot (6 holes). A manufacturer (Alpine, 1977) has indicated that both the rock impact and broken rock volume can be satisfactorily accommodated by machines currently in use.

The Spiral Drill and Blast Machine (Peterson, et al, 1976), which is currently being field tested, contains a continuous, semi-automated mucking system. This machine uses a remote mannually-operated hoe to pull broken material to the chain conveyor. The chief disadvantage of such a system is that an additional operator is required.

Of the three mucking systems described, and in the light of the constraints imposed by the ADBS (continuous access to the face and minimum manpower), the continuous automatic gathering arm type mucker appears to be most adaptable. Such a system can be either rail or crawler mounted. Because rail installation involves down time, a crawler mounted machine may be preferable. (See also Figure 6-1).

The chassis of the ADBS then will consist of hydraulically driven crawlers, a frame to carry the chain conveyor, the apron and automatic gathering arms, and a frame on which will be attached other ADBS components. The gathering arms, chain conveyor, and the crawler drive are powered by their own hydraulic pump. This machinery will be separate from a similar unit used to power the hydraulic drilling and explosive load/blast components.

The apron will extend the total width of the drift and will abut the face, allowing almost complete muck collection. After each complete round (24 to 36 in.), the ADBS will advance so that complete muck removal is maintained.

The chassis and automated mucking system, as described above, is available as a single unit (Alpine, 1977) and can be readily modified to incorporate a full width apron and platform. Two hydraulic arms will also be attached to the rear of the chassis to carry the blast shield when the ADBS moves forward after each round (Chapter 3).

Hydraulic Drills

The drilling must be performed semi-continuously for optimum advance rates. A typical small charge round requires more lineal feet of drilling (of smaller holes) per foot of advance than conventional DBM. This increase in drilling is required because of the smaller spacing and burden required for small charges. Reduced blast energy per shot (the small charge concept) produces lower air blast overpressure, noise, and ground vibration which is requisite to ADBS operation.

The V-cut patterns (Figure 5-1) used in the Edgar Mine tests of the small charge round (Hanna, 1978) were 24 to 30 in. deep and had a low powder factor, good rock breakage, and an unfractured opening perimeter with relatively smooth walls. The rounds were blasted in Idaho Springs gneiss, which is moderately jointed and lightly fractured in the test section of the drift.

A conventional DBM round usually employed in the same heading is 4 ft in depth. Inspection of these two figures shows that the lineal feet of drilling per foot of advance is approximately 1.8 times as great for the ADBS. However, the increased drill hole length imposed by the small charge concept is compensated for by the reduced drill hole diameter. The ADBS will use a 1-in. diameter bit (Clark & Rollins, 1976). Because a conventional hole is drilled with a 1-1/2-in. bit, the small charge drill volume is 0.44 times the conventional drill hole volume. Hence, the total volume of rock removal by drilling smaller holes is 0.8 that for larger holes.

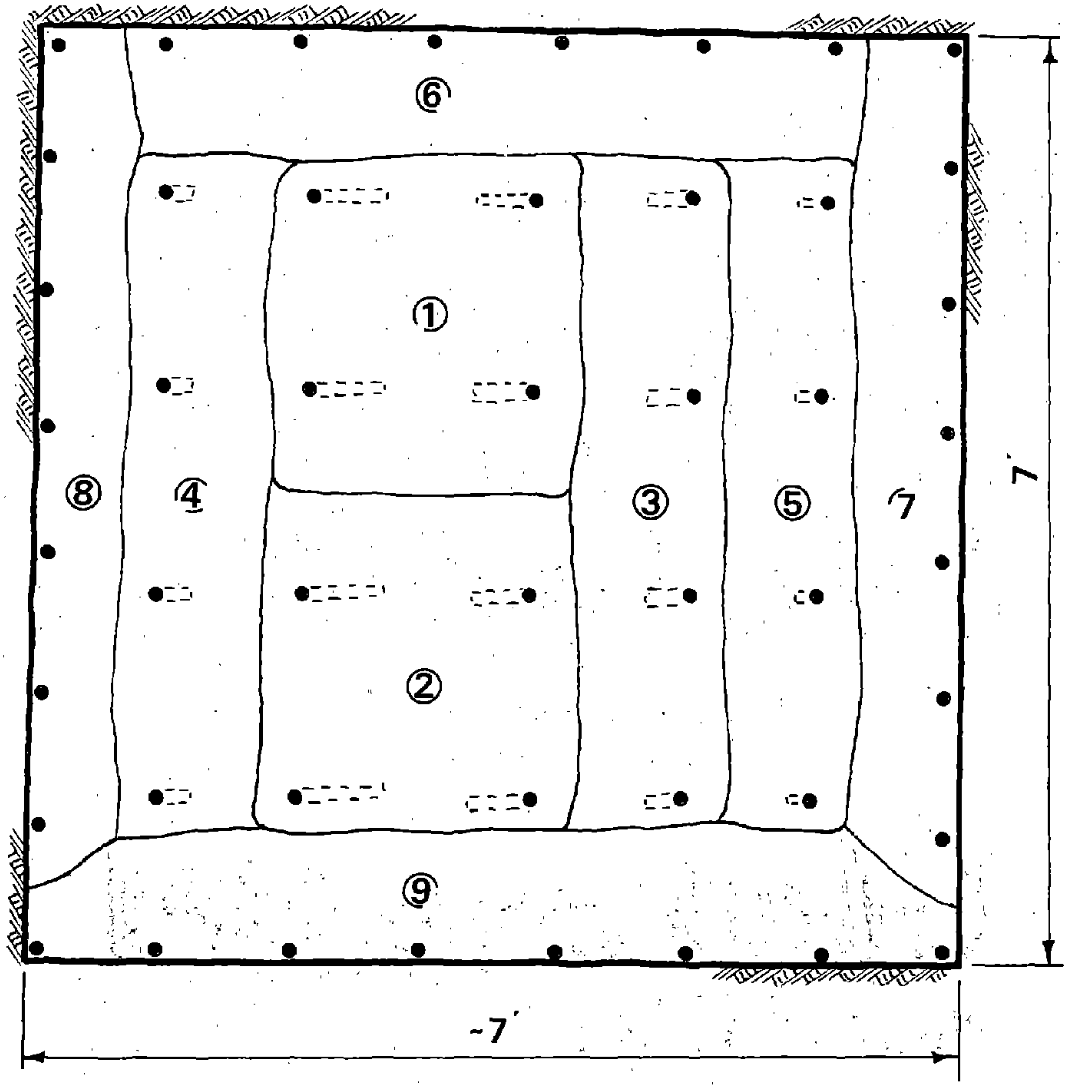
The number of hydraulic drills required is determined by the rock type and tunnel face area. Hydraulic drill selection criteria are:

1. Maximum penetration rate in the rock types attained in a given tunnel with a given drill.

2. Power matched to small diameter drill holes.

FIGURE 5-1

ADBS ROUND PATTERN, 24 INCH DEEP ROUND



Note: Numbers in Circles Indicate Shot Order
Maximum Holes Per Shot = 8
Total Holes Per Round = 48
Lineal Feet of Drilling = 96 Per Round
Lineal Feet of Hole Per Lineal Foot of Advance = 48

3. Drill boom designed for the short feed distance required by the small charge concept (18-36 in.).

4. Drill boom sized to cover the required face area.

5. Drill boom designed to accommodate the selected hydraulic drill.

6. Drill boom modified to provide blast protection for the hydraulic lines (such protection will consist of steel shields around hydraulic lines designed to allow normal articulation).

7. Drill controls mounted in the enclosed cab.

Alignment Control

Predicted rapid advance rates require that proper horizontal and vertical alignment be continuously verified and maintained. Proper alignment can be maintained through the use of laser instrumentation and telephone communication with the operators. The system will conceptually operate as follows: Surveyors correctly align a laser instrument mounted on one rib of the drift. This is typically accomplished by aligning the origin of laser light and two additional targets through which the light beam passes (these two targets allow the instrument to be adjusted if it is bumped). The laser beam impinges on the blast shield and from this point the distance to the rib and invert may be directly measured by personnel outby the shield. Based on this measurement, the operators can be informed of line and/or grade changes as required.

Ventilation

The ventilation system must be deisgned to furnish air to three separate operational areas:

1. Outby shield area (support erection as required, muck loading and haulage, utility installation, etc.).

2. Inby shield area - control cab (comfort and ventilation required for operators).

3. Inby shield area - drilling and blasting area to provide visibility for continued advance operations.

Air requirements for the outby shield area depend on the number of personnel and types of equipment operating. The requirement for the control cab(s) is 6.6 cfm which provides that the air be replaced six times per hour. The remainder of inby shield area will be occupied with gases and dust each time blasting occurs. Blasting liberates approximately 30 moles of gas per kilogram of explosive (5160 ft^3/lb) (Cook, 1958). The dust generated by blasting is 62.8 mppcf based on samples collected immediately after blasting in over 75 underground mines (Hartman, 1961). Ventilation requirements are based solely on the dust concentration, assumed to be 70 mppcf. That is, if the smoke time delay is assumed to be one minute per shot, the flow of air required to reduce 70 mppcf to 4 mppcf is:

$$Q(\text{cfm}) = 70 \text{ mppcf}/4 \text{ mppcf } (1 \text{ min})/1 \text{ cu ft}$$

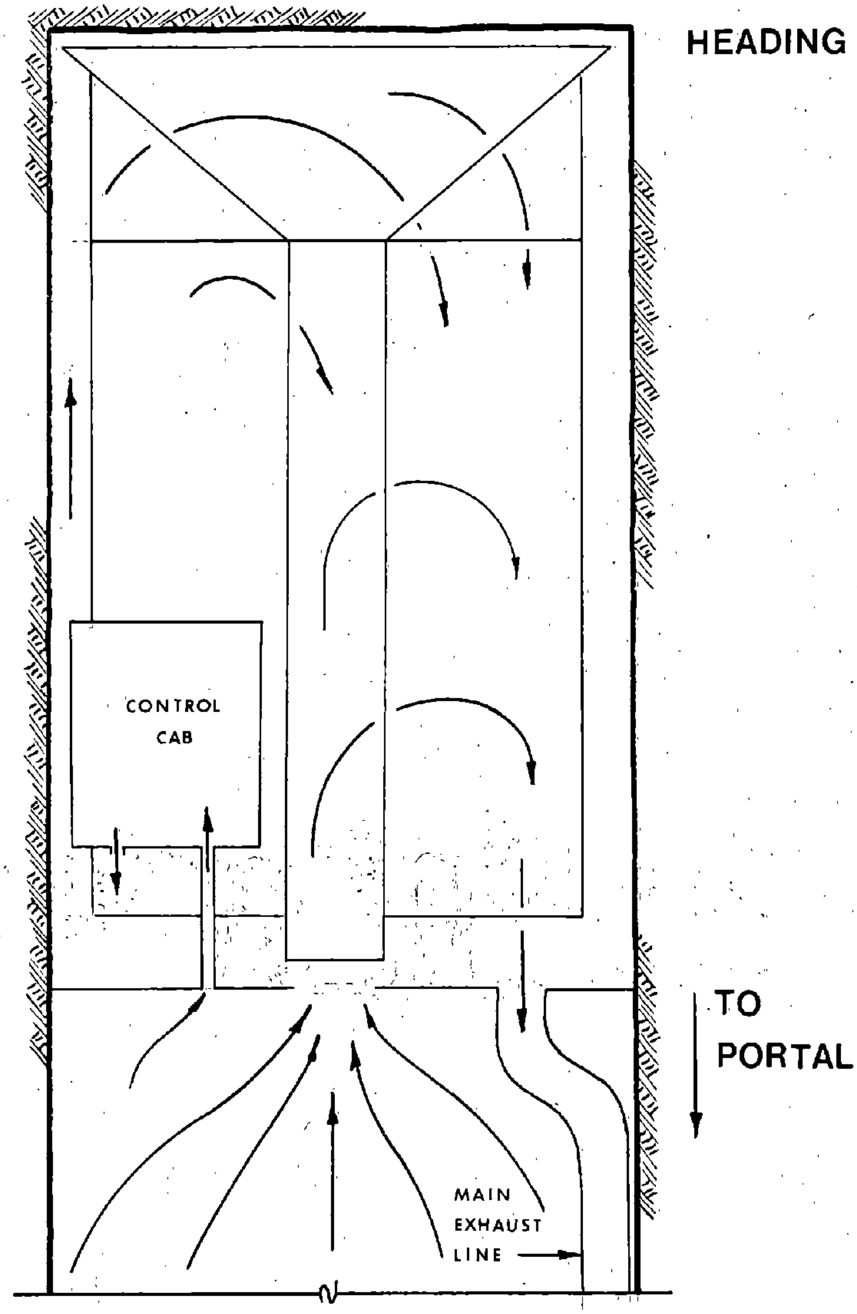
$$Q(\text{cfm}) = 17.5 \text{ cfm}$$

This amount of air must be exhausted from inby the shield and not allowed to contaminate the outby work area. Thus, an exhaust ventilation line is required from the shield to the portal. (Water sprays can be incorporated inby the blast shield to alloy the dust to provide more rapid visibility).

Since an exhaust line is required, the most efficient overall tunnel ventilation is provided by an exhaust system (Figure 5-2). A main exhaust line, sized for both inby and outby requirements, may be located on the rib or at the crown. The blast gases and dust may be discharged to this line via a flexible vent tubing. The vent line will be protected during the blast by means of a butterfly valve. After the blast, the valve will be

FIGURE 5-2

PLAN VIEW OF ADBS EXHAUST VENTILATION SYSTEM



opened and the dust exhausted. The control cab(s) will be continuously ventilated except during blasting when the cab will be totally closed. After each blast, all ventilation valves will be opened and fresh air circulated.

This ventilation plan assures that, even after the blast when the door on the shield is opened to allow muck removal, blast gases and dust will not enter the outby work area. An auxilliary fan may be necessary if the exhaust system does not remove blast gases and dust rapidly enough. Such a fan could be attached to the control cab and adequately shielded.

CHAPTER 6

NOISE AND AIR BLAST MEASUREMENT

Introduction

A second generation blast shield was designed and fabricated and was tested under conditions identical to those anticipated for a shield under operating conditions. The purpose of the shield is to contain blast overpressure, dust, fly rock, and to reduce the noise level outby the shield. The weight of power per hole and the number of simultaneously detonated holes per shot were determined by results from previous research (Clark & Rollins, 1976). The shield was located 13 to 17 ft from the face instead of over 20 ft as planned for an operational ADBS, the overpressure was higher than that predicted, and the pressure will be further reduced when operating equipment is between the blast and the shield.

Over thirty shots were fired against the shield. For shots 2 through 11, the seismic particle velocities in the rock were measured at three locations along the drift, and the noise level was measured 5 ft outby the shield.

Instrumentation

Longitudinal, transverse, and vertical particle velocities were measured at three locations in the wall of the drift by three component geophones. The geophones were bolted to a plate which was anchored to the rib by steel bars wedged in drill holes. Each instrument was then leveled with leveling screws.

The blast noise level was measured by an Air Wave Detector, Model SM-1 (Sprengether, 1977). This instrument is

> "a low frequency omnidirectional transducer that functions independently of electronic amplification. It is intended for use in applications where the monitoring of low frequency accoustic phenomena associated with explosions is required."

The three geophones and the Air Wave Detector were connected to a 12-channel refraction amplifier, SIE Model RS-44 (SIE, 1977). This instrument has individual channel gain and filter adjustment which allowed signal output to be adjusted according to the anticipated magnitude of the blast.

The output was recorded on an R Series Oscillograph (Dresser, SIE, 1977). This unit produced a permanent record of the output of geophones and Air Wave Detectors on light-developed recording paper. Both the oscillograph and the RS-44 were powered by 24 volt batteries (Figures 6-1 & 6-2).

Test Results - General

The shot for which particle velocities and noise were measured produced good rock breakage except for those where holes bootlegged. This problem was first attributed to inadequate stemming, and the single short length caulking was augmented by filling the hole to within 6 in. of the collar with moist medium grained sand. This stemming gave no improvement in blasting results. The slurry explosive strength was then compared with Hercules 65% "Unigel" dynamite. This was accomplished by placing equal weights of gel and dynamite in 1 in. diameter steel pipes resting on compacted soil. Three different weights (60, 75, and 90 grams) of each explosive were detonated and the width and depth of the crater thus formed were measured. No significant difference in crater dimensions was observed (the gel may have produced a slightly larger crater). The powder factor was then increased by 30% to 50% and no further bootlegs occurred. The bootleg problem was directly attributable to insufficient explosive energy for the burden and hole depth.

The noise level produced outby the shield by the ten shots ranged from 96 to 112 dBA. Inspection of the data (Chapter 6) shows that when all holes of a shot detonated properly, the maximum noise level was less than 108 dBA.

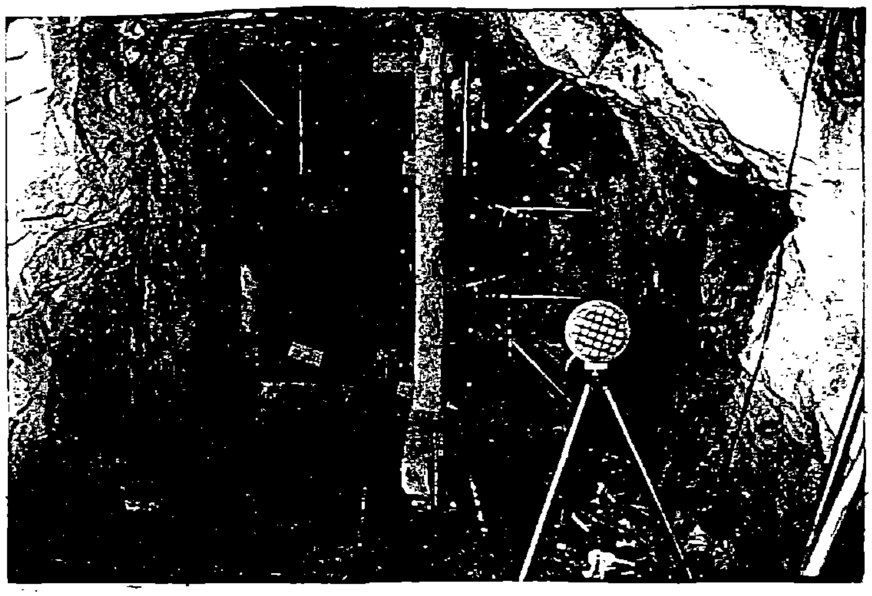


Figure 6-1 Typical Geophone and an Air Wave Detector Installation.

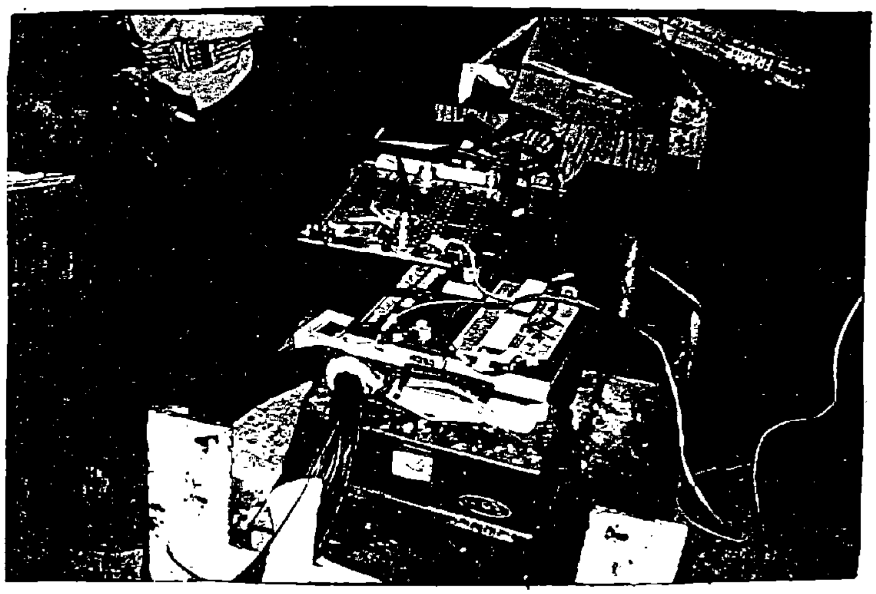


Figure 6-2 Recording Instrumentation Showing (Left to Right) Batteries, Recording Oscillograph, and Refraction Amplifier.

CHAPTER 7

BLASTING EXPERIMENTAL DESIGN

Introduction

In addition to the tests to determine the response of the shield to air blast and the level of noise outby the shield, a vital part of the research program was to investigate the details of round design, their relation to breakage, fragmentation, ground vibration, air blast, particle velocity, and related factors.

Small charge blasting, and particularly the experimentation, is designed to determine the lower limits of breakage in terms of powder factor, spacing of holes, burden, and geological structure.

Test Site and Geology

The investigation of the second phase of the factors involved in the design and testing of a small charge method of drill and blast system were conducted at the Experimental Mine of the Colorado School of Mines located at Idaho Springs, Colorado. The experiments were carried out in a cross cut from B-left drift (Figure 7-1) in a jointed and fractured granite or gneiss. The general geology of the Idaho Springs gneiss, which is a jointed, faulted, and foliated metamorphic rock mass, has been described by Moech (1964). The local rock mass consists of granite gneiss, pegmatite, and biotite gneiss. Several faults occur in the vicinity of the B-left drift, but no faulting is present in the immediate experimental area. No attempt was made to measure the direction of the joints except in the face of the experimental drift. The rock had a high modulus of elasticity, averaging 8.36×10^6, and its compressive strength varies from 16,500 to 25,700 psi unconfined and with higher strengths under confinement (Tables 7-1 & 7-2).

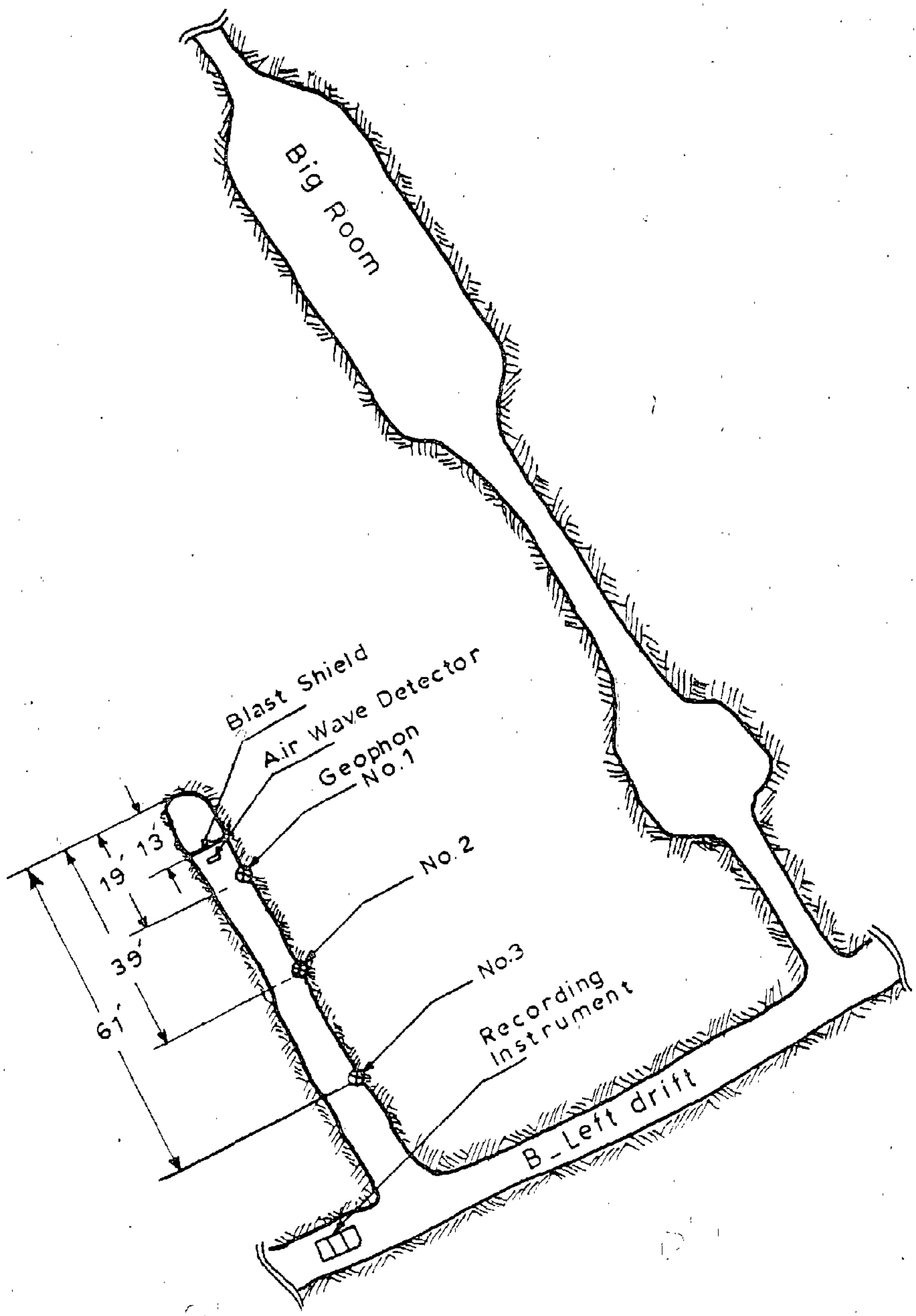


FIGURE 7-1 Map Showing Location of the Geophones and an Air Wave Detector in the B-Left Drift, Colorado School of Mines, Experimental Mine, Idaho Springs, Colorado.

TABLE 7-1

Elastic Moduli of Rock (Cox,Jr., 1971)

Specimen No.	Length-to-Diameter Ratio L/D	Young's Modulus 10^6 Lb/in^2	Poisson's Ratio
ER-1	3.07	6.88	.14
ER-2	3.07	7.83	.14
S-18-8	2.24	9.47	.13
S-22-6	2.29	8.53	.13
ER-3	2.53	7.47	.13
ER-5	3.06	7.08	.10

Average Young's Modulus = 8.36 $\pm$ .91

Average Poisson's Ratio = .15 $\pm$ .04

TABLE 7-2

Compressive Strength (Cox,Jr., 1971)

Specimen No.	Length to Diameter Ratio (L/D)	Compressive Strength (psi)
S-22-6	2.25	25,700
S-22-7	2.21	22,000
S-18-7	2.14	16,500
S-18-8	2.20	20,000
---	500 psi confining pressure	------
A2	2.24	17,200
B1	2.24	25,000
---	1,000 psi confining pressure	------
A1	2.13	24,000
A3	2.14	30,500
B2	2.19	27,000

Average unconfined strength = 18,200 $\pm$ 3600 psi
@ 500 psi confinement = 21,200 $\pm$ 4000 psi
@ 1000 psi confinement = 27,800 $\pm$ 2600 psi
Average density = 0.095 $\pm$ 0.001 lb/in^3 (164.2 lb/cu ft)
Average specific gravity = 2.63

Instrumentation

One of the unique characteristics of the small charge blasting method is the manner in which the charges are fired. That is, 2 to 6 holes in a line are loaded with as small a charge as possible to insure breakage, and the charges are fired simultaneously (within $\pm$ 100 μsec). The purpose of this type of loading and firing is to reduce the air blast and the ground vibration as much as possible, and instrumentation was selected to measure the air blast level outby the tunnel shield and the ground vibration (particle velocity) along the walls of the drift (Figure 6-1).

The three components of the particle velocity were measured at three locations (19, 39, and 61 ft) from the face of the drift on the rib. Three component geophones were bolted to steel plates, and the plates were anchored to the rib with short steel bars, the bars being wedged tightly in drill holes.

The air blast was measured 5 ft from the tunnel shield on the outby side with an Air Wave Detector, SM-1 (Sprengnether, 1977).

The three geophones and the Air Wave Detector were connected to a 12-channel refraction amplifier, SIE Model RS-44 (SIE, 1977). This instrument has individual channel gain and filter adjustments which allow the signal output to be regulated for the anticipated mangnitude of the air blast or particle velocity. The output was recorded on an R Series Recording Oscillograph (Dresser SIE, 1977), which produced a permanent record of the output of the geophones and the Air Wave Detector on light developed paper (Figure 6-2).

Round Design

A V-cut type round was selected for the basic tests of the feasibility of rock breakage with small charge rounds because it was desired to fire in line holes simultaneously and all of the segments of a V-cut round are made up of holes which are arranged in straight lines. Also, it does not require the drilling of dummy holes which are not loaded with explosive as is found in many burn cut rounds.

The basic principles of round design used in this experimentation are the same as those employed in the design of any conventional round. The explosive in the holes fired initially must be located so that the stress waves will break out a section of the face in such a manner that the breakage will create new free faces to which the subsequently fired charges may break.

The first phase of this experimentation, which was carried out in Missouri red granite in Missouri, utilized the face of a granite quarry in which most of the drift face was free from fractures and joints. One of the objectives of the second phase of this project, the results of which are described in this report, was to determine the effects of the geologic structure of the face upon the functioning of the small charge method of breakage. Of particular concern was the question as to whether the reinforcing effects of stress fields from adjacent blast holes would be decreased by the presence of the joints and fractures.

Hence, the factors to be evaluated in these tests were the relationships of the following to the geologic structure of the face:

a. Burden and spacing of holes

b. Depth of holes

c. Inclination of holes

d. Diameter of holes

e. Other types of cuts

f. Relation of rock and explosive properties

g. Method of initiation of explosive

h. Determination of optimum powder factor for each part of round

i. Determination of optimum number of charges per shot

j. Placement of holes with respect to joints and fractures

Rounds of 18 in. depth had been fired in earlier experimentation so it was planned to fire two rounds each of 24 and 30 in. depth. These rounds were designed to maintain a proper powder factor without increasing appreciably the number of holes. However, as the depth of a round is increased the spacing and burden may be increased, which in turn requires an increase of the weight of explosive per hole.

A general rule that has been followed in blasting is that the depth of the boreholes should not be any deeper than the minimum horizontal or vertical dimension of the face. This rule would hold for V, pyramid, draw, and slab type cuts, but the limit of depth of burn cut rounds is primarily determined by the depth to which holes can be drilled accurately, i.e., with minimum deviation in direction.

Thus, for very shallow rounds, the number of holes per unit area of the face is determined by the depth of the holes. For rounds other than burn cut, Langefors and Kihlstrom (1963) indicate that the number of holes per unit area increases more slowly than the increase in cross sectional area for smaller drifts, then reaches a constant value at a point which depends upon the rock properties and the diameter of the holes (Figure 7-2). The number of trim holes increases while the number of center holes remains constant.

In the experiments described below the minimum diameter of hole that could be employed was controlled by two factors. The first was that drills of diameters smaller than 1-1/4 in. are not readily available, and the second was that the slurry explosives have a critical diameter of [illegible] in. Consequently, for small charges and 1-1/4 in. diameter holes, the charge is concentrated in the bottom of the hole which may limit the depth, spacing, or burden.

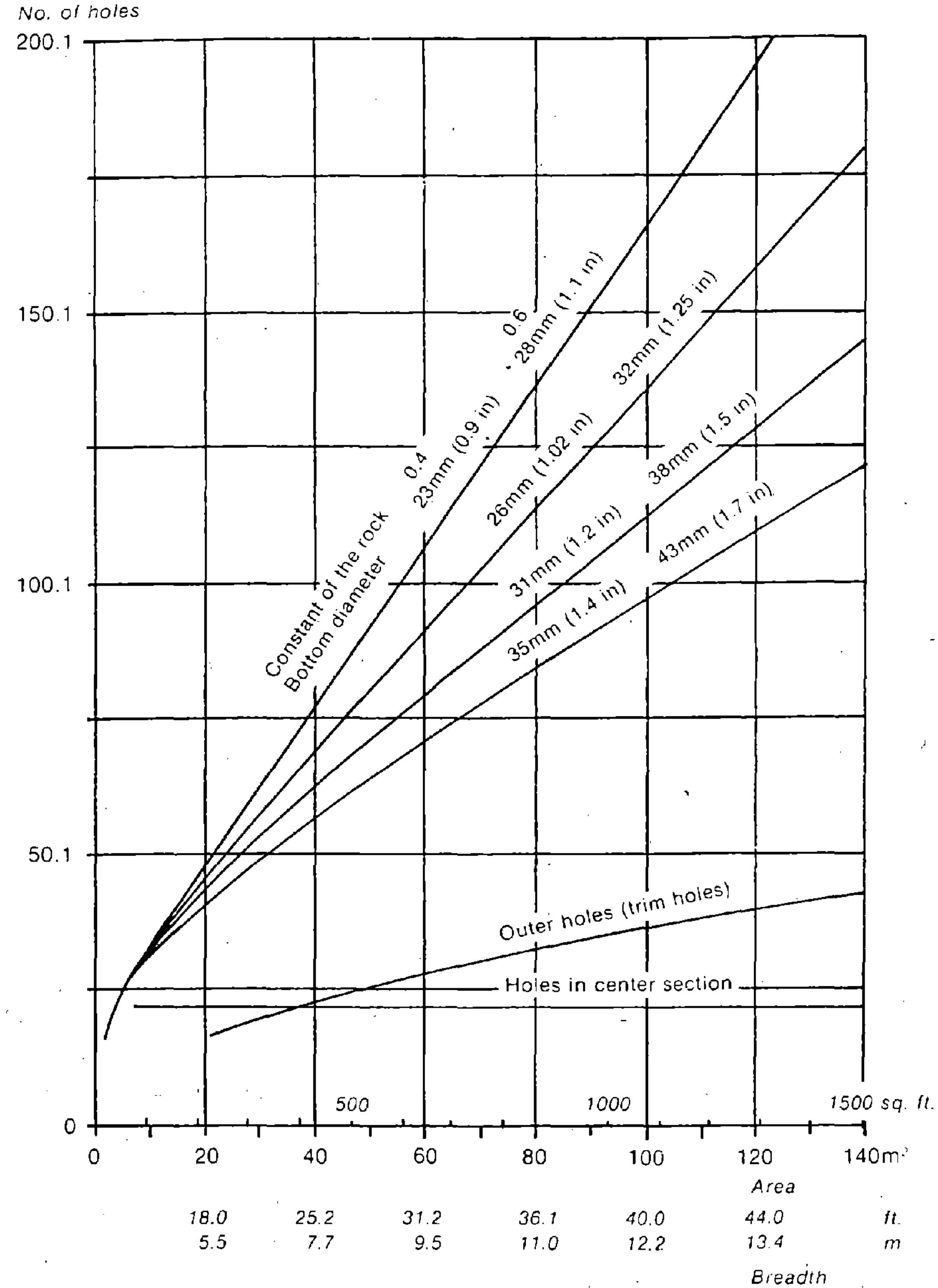


FIGURE 7-2 Number of holes as a function of tunnel face area for hole diameters for rocks with property constants C=0.4 and C=0.6 assuming a bore hole deviation of 0.85 ft (0.25m). Includes holes for smooth blasting.

Langfors & Kihlstrum, 1963

However, as discussed below, as the depth of the round is increased to 30 in. and greater, the results of the tests indicate that the spacing and burden can be increased with a consequent decrease in the amount of drilling and the number of blasting caps required.

The first round of 24 in. depth as fired to test round parameters as well as to test the blast shield and measurements were made of the air blast and the ground vibration. The goals of the first test were:

a. To smooth the walls and obtain a smooth regular face.

b. To determine effects of geologic structure upon small charge blasting.

c. To modify blast patterns based upon observed effects of geologic structure.

d. To optimize the number of holes to be fired simultaneously.

e. To test the shield.

f. To measure ground vibration and air blast.

This round required several shots consisting of blasting of groups of in line holes. The whole round consisted of about 40 holes, 1-1/4 in. diameter which were drilled with a jackleg. The cut portion of the round consisted of 8 holes collared in a vertical line, inclined at an 18 degree angle with the normal to the face, thus providing a 28 in. burden and 18 in. spacing at the collars. The weight of explosive per hole was calculated based upon the volume of rock burden on each hole with an assumed powder factor of 0.1 lb/cu-ft. After the initial cut was broken out, the blasting progressed by loading and firing each successive line of holes until the periphery of the drift was reached. For the best results, the value for the spacing/burden ratio was taken to be greater than one, i.e., for successful breakage to bench-like free faces, the spacing was 18 in. and the burden 14 in. The peripheral holes were drilled outwards at a 10 degree angle to form a 7 x 7 ft opening and spaced at a smaller distance to produce a smooth opening utilizing a powder factor of 1.35 lb/cu-yd.

The slurry charges were primed and fired with 25 grain-per-foot detonating cord with the cord leading to each hole in a group joined to one blasting cap. The holes were stemmed with a single shot length of caulking compound to a point within 6 in. of the collar. The use of detonating cord in this manner resulted in virtually simultaneous detonation of the charges in one group. The first four holes in the center of the cut bootlegged and the air blast moved the shield several inches, the blowout being due to an inadequate powder factor. The shield was reinforced as described in Chapter 3. With increased powder factor, the first round was successfully broken, creating a relatively smooth wall and face.

The second round differed from the first in the following respects:

a. Initial cut:

   1. The powder factor was increased to 5.4 lb/cu-yd.

   2. This was partially accomplished by decreasing the burden from 28 to 20 in.

b. Relievers and peripheral holes:

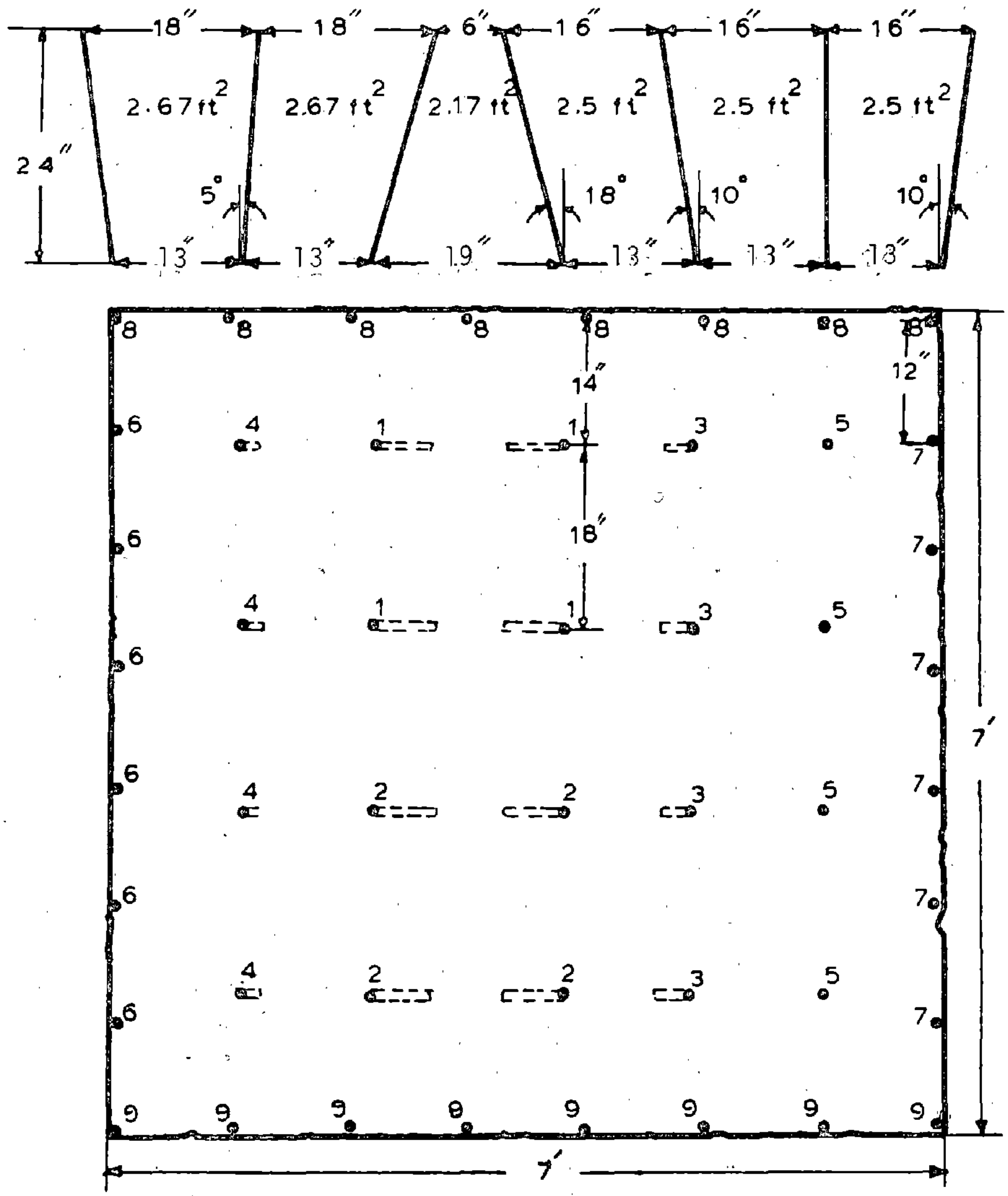
   1. The powder factor was increased to 2.4 lb/cu-yd.

   2. The number of peripheral holes was increased.

   3. The amount of explosive in holes close to joints was reduced.

All of the holes were stemmed to the collar by filling them with caulking material and moist medium-grained sand. The round was again 24 in. deep and consisted of 48 holes (Figure 7-3). To keep the amount of explosive per shot less than 1 lb, it was necessary to fire the holes in nine or more groups. The cut was fired in two sequences with 4 holes per shot, and the reaminder fired in the order shown in Figure 7-3. The above values gave good fragmentation, relatively low air blast pressure, and somewhat reduced ground vibrations, especially when blowouts did not occur. The data obtained from this test were used as a base for designing

Figure 7-3. Experimental Round Pattern (V-Cut) 24 Inch Deep Round.



NOTE: Numbers Indicate Shot Order. Maximum Holes Per Shot = 8. Total Holes Per Round = 48.

a 30 in. deep round.

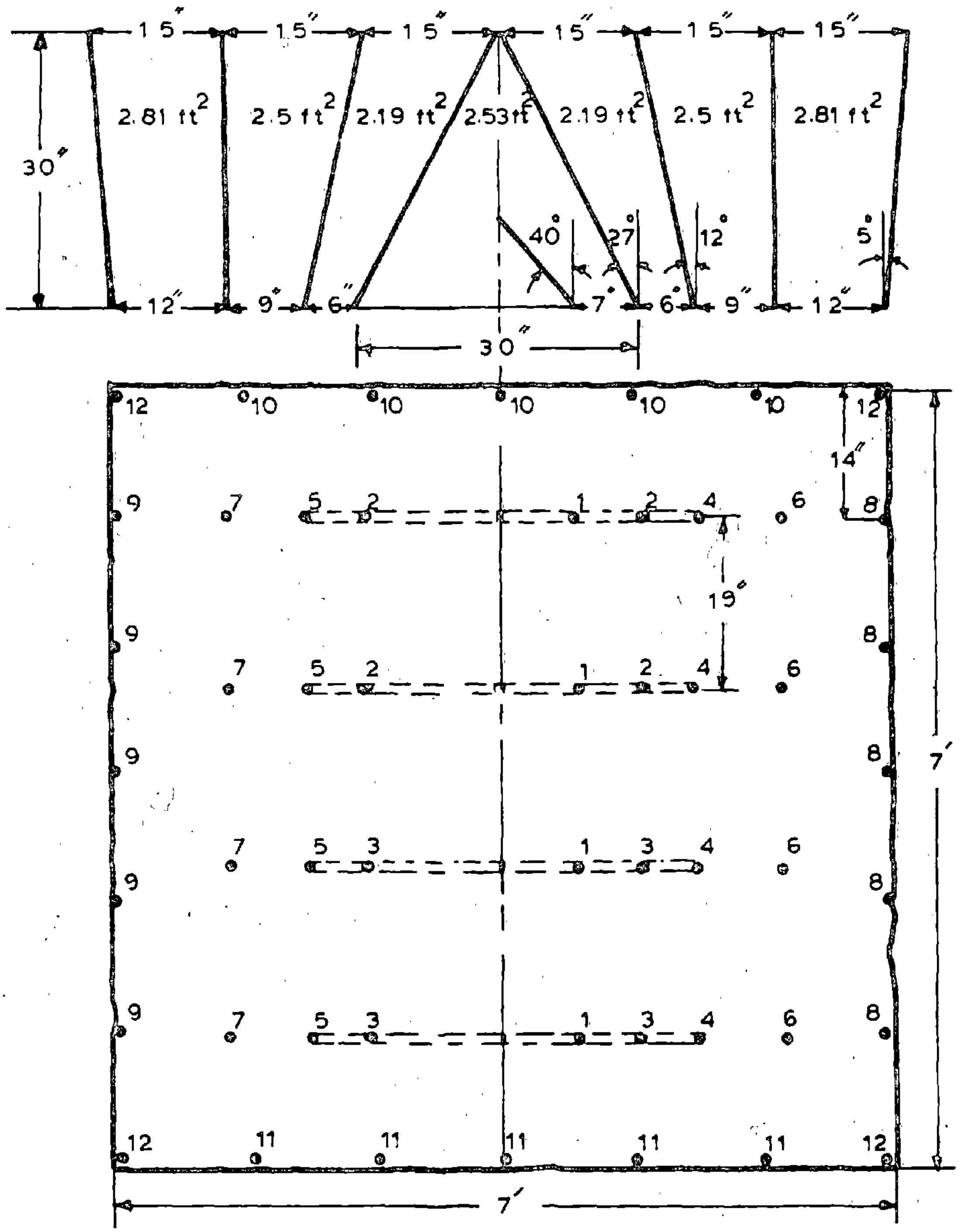
The purpose of firing a deeper round was to investigate the effects of increased depth in relation to geologic structure, spacing, burden, breakage, and general blasting performance. The pattern used a "baby" cut numbered 1 (Figure 7-4) to make the first break into the face and to keep the amount of explosive per shot below 1 lb. This procedure also reduces the depth of the first break and creates a new face to which the main cut may break. The "baby" cut was drilled with holes 13 in. deep at a 40 degree angle with the normal to the face.

The 30 in. round consisted of 52 holes (Figure 7-4) and the holes by line were fired in the order indicated. The powder factors used were 0.5 lb/cu-ft for the "baby" cut, with values ranging from 3.0 to 3.5 lb/cu-yd. The round broke successfully and the values employed for powder factors were again employed as guides for a second round 30 in. deep. However, the high powder factor required to break the rock in the first round was excessive because of misalignment of holes, and was consequently reduced to 0.21 lb/cu-ft for the second round.

In the 30 in. rounds, the number of holes was kept approximately the same as for the 24 in. rounds. For an infinitely large face, theoretically it should be possible to increase the burden and spacing linearly with the increase in depth of the holes. This would assume, however, that the geometry of the charges for the 24 in. rounds was of optimum size and shape. Also, the diameter of the drill holes would be increased in a like manner. In other words, cube root scaling should apply to the breakage effects.

As indicated earlier, the smallest diameter that can be used is dictated by the size of drills available and the critical diameter of the explosive. For blasting of drift or tunnel rounds, it is usually desirable

Figure 7-4 Experimental Round Pattern (V-Cut)
30 Inch Deep Round.



NOTE: Numbers Indicate Shot Order. Maximum Holes Per Shot = 5. Total Holes Per Round = 52. Lineal Feet of Drilling = 104 Per Round. Lineal Feet of Hole Per Lineal Foot of Advance = 52.

to have the explosive distributed along the drill hole instead of its being concentrated at the bottom of the hole. With the depths of round required for small charge blasting and the diameter limitations described above, the charges are still pretty well concentrated at the bottoms of the holes.

If, however, the spacing between the holes is increased in proportion to the depth, the volume of rock to be broken by each hole increases as the cube of the depth increase, and if the diameter of the hole is kept the same, the length of the explosive charge would likewise be increased by the ratio of spacing increase.

There are not enough data available at the present time to make an exact analysis to the relationship between the spacing, burden, powder factor, depth, face area and rock properties. Langefors and Kihlstrom have depicted the relation of number of holes and face area for two rock constants (Figure 7-5), but this analysis applies largely to very large tunnels.

A plot of the data for excavated tunnels described by Langefors and Kihlstrom (1964) indicates the general relationship between the number of blast holes per unit area of tunnel face and the depth of holes for conventional blasting, for application of the cube root law, and the values that have been used in small charge experimentation together with a projection for possible reduction in the number of holes with increased depth of blast holes (Figure 7-5). In general, the depth of the blast holes increases with the smallest dimension of the face, so that the depth of holes is approximately proportional to the square root of the face area. It is obvious that for the usual size of tunnels the depth of holes for small charge blasting is independent of the face area. The upper limit on depth is imposed by the restriction on the amount of explosive which

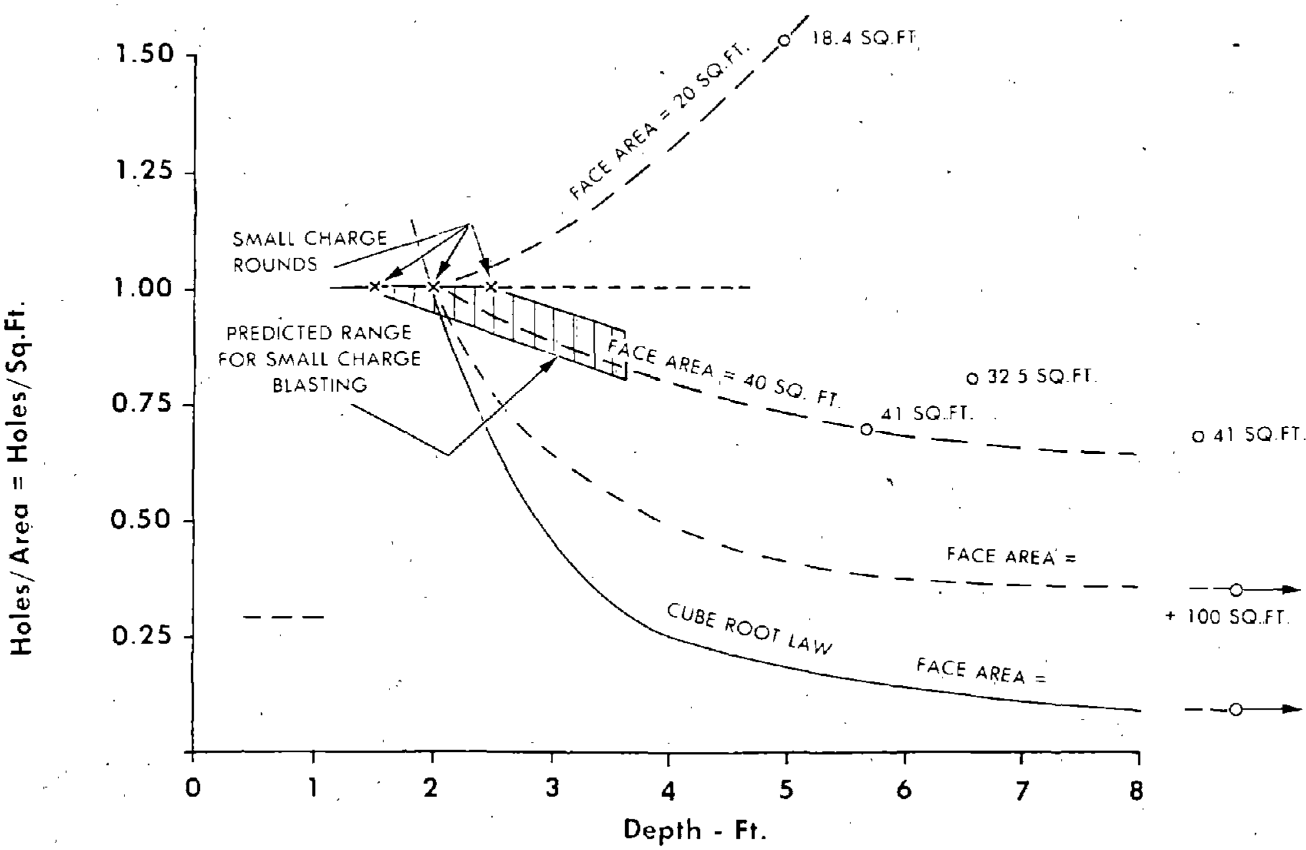


FIGURE 7-5 HOLES PER UNIT AREA AS A FUNCTION OF TOTAL FACE AREA AND DEPTH FOR CONVENTIONAL AND SMALL CHARGE BLASTING.

can be fired in one shot and the amount of explosive per hole. For good fracture control, it is usually desirable to fire 4 to 6 holes at one time, although there is a possibility that for some portions of the round that this could be reduced to 2 holes per shot.

The results of the experimentation to date indicates that the advantages of firing holes simultaneously outweigh the advantages of increasing the depth of the holes.

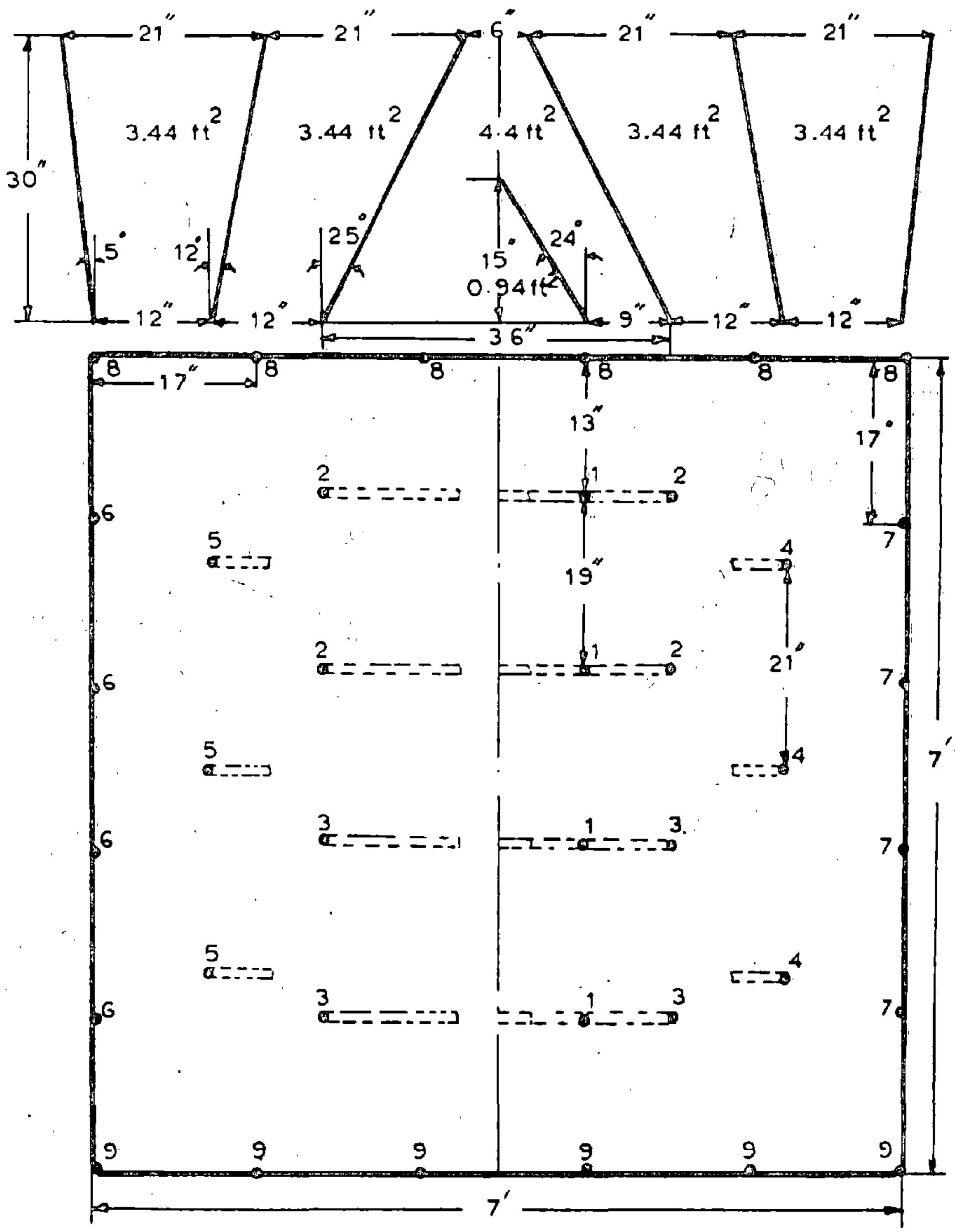
Thus, a round was designed utilizing 38 holes (Figure 7-6), but it was not tested. The spacing and burden were increased linearly with the depth of holes following the cube root law and with the same powder factors employed in the test rounds which were fired. The cut holes will contain 76 to 200 grams, the relievers 200 grams, and the trim holes 100 to 200 grams of explosive.

Mapping the Face

In order to determine the effects of face structure upon breakage and fragmentation, the geologic structure of the face was mapped before and after each set of holes was fired. In general, the successively exposed faces were moderately jointed and lightly fractured. The mapping of the face included a visual assessment of the irregularity of the joints, whether they were open or closed, and their average spacing. This information was used in the analysis of results and the planning of the layout of each portion of the round (see Chapter 8).

The number of holes per square foot of face varies from 0.8 to 1.0. For large faces, if four holes are fired for each shot, the number of shots per advance would be $n = 0.8A/4$. More holes can be fired per cycle in larger faces where two or more drills are employed and with delays between groups of holes to keep the amount of explosive per shot to a minimum.

Figure 7-6. Theoretical Round Pattern (V-Cut)
30 Inch Deep Round.



NOTE: Numers Indicate Shot Order. Maximum Holes Per Shot = 6. Total Holes Per Round = 38.

CHAPTER 8

BREAKAGE, THROW AND GROUND VIBRATION

Introduction

The experiments described in the previous chapter produced results which added to the information about small charge blasting which had been obtained in earlier experiments in Missouri red granite. The tests confirmed that the principle of simultaneous blasting of adjacent blast holes resulted in good fracture control even though the tunnel face was jointed and fractured. With proper design and correct drilling good breakage and fragmentation was likewise obtained, as well as a reduction of air blast. The placement of the geophones with respect to the tunnel face being blasted apparently had a marked effect upon the magnitude of the peak particle velocities.

The first 24 in. round was utilized to determine the effects of round design upon breakage and fragmentation as well as to make measurements of the air blast pressure outby the shield and the particle velocity at the geophone stations. The round was fired in eight separate line shots, which included those shots in which bootlegs occurred (Table 8-1). The cut, having 8 holes,was fired in two sequences, first, the bottom 4 holes and the top 4. The holes were inclined at 18 degrees with the normal to the face giving a 28 in. burden at the collar with an 18 in. spacing. The bottom holes were 24 in. deep and were loaded with charges varying from 50 to 200 grams (2.7 lb/cu-yd) per hole, depending upon the burden on the particular hole, which varied because of the irregularity of the face. All four holes bootlegged because the burden was too great. Holes were redrilled, reducing the burden to 20 in., and reloaded with 80 grams per hole (4.0 lb/cu-yd). They were well stemmed with caulking and wet sand. Good breakage was ob-

TABLE 8-1

24 Inch Deep Round, Instrumented Shots
(First Round)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) - (lb)	Comments
1(1)	V-Cut (bottom)	4	28(2)	18	18	2.7	295 - 0.65	Bootlegs in 4 holes due to large burden
2	V-Cut (bottom)	4	20	18	18	4.0	321 - .71	Good breakage, several large fragments
3	V-Cut (top)	4	20	18	18	4.0	152 - .33	Good breakage, small fragments Small explosive weight due to shorter holes
4	Relievers (left)	4	16	18	5	1.35	225 - .5	Bootlegs in bottom 2 holes due to large burden
5	Relievers (right)	4	16	18	5	1.35	212 - .47	Good breakage, several large fragments created by breakage to joints and existing fracture in the face
6	Relievers (left)	3	16	18	5	1.35	176 - .39	One extra reliever hole drilled. Good breakage

TABLE 8-1 (Cont'd.)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Hole Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) - (lb)	Comments
7	Trim (right)	3	14	14	10	1.35	180 - .4	One bootleg. Good breakage between other two holes
8	Trim (left)	5	14	14	10	1.35	256 - .56	Good breakage, smooth wall, small fragments
9	Crown Lifters	2 6	14	14	10	1.35	334 - .74	Good breakage at crown. Bootlegs in 3 lifter holes.
10	Lifters	3	14	14	10	1.35	170 - .37	Good breakage, smooth wall, small fragments

NOTE: (1) Not instrumented

(2) Assumed to be the total distance between two pair of holes

See Figures 8-2 & 8-3 for shot location.

tained while a few large fragments were formed about 2 ft on a side partially because of the position of the holes with respect to the joints, and because of fractures created by the bootlegged shots. The burden on the top 4 holes was smaller because of irregularities of the face and the rock between them was easily broken to extend the cut (Figure 8-1). The sides of the cut were relatively smooth, and the line of fracture between the holes extended beyond the cut several inches in both directions.

The remainder of the holes were fired proceeding outward from the cut. The second line of relievers was fired with 56 grams per hole (1.35 lb/cu-yd). The bottom two holes did not break to full depth, probably because of misalignment of the holes. The remainder of the relievers were fired with the same powder factor and all of them broke the rock successfully (Table 8-1).

In order to maintain smooth walls and the desired dimensions, the periphery holes were drilled with their collars on a 7 ft square with the holes inclined outwards at a 10 degree angle, and on 14 in. center spacing. The powder factor used was 1.35 lb/cu-yd with 30 to 70 grams/hole depending upon the depth of the hole. Only one hole bootlegged, probably due to excessive natural fracturing at the borehole which allowed the gases to escape. The crown and lifter holes were fired with an average of 40 grams per hole, resulting in good breakage at the crown, but with 3 bootlegs in the lifters, which were fired with 56 grams per hole, breaking the rock effectively. The face of the drift was moderately jointed with tight joints spaced on 5 in. to 1 ft intervals. The geologic structure of the face was mapped before and after each shot, because the presence of the joints could create large fragments in the muck pile and affect the propagation of controlled fractures between the blast holes.

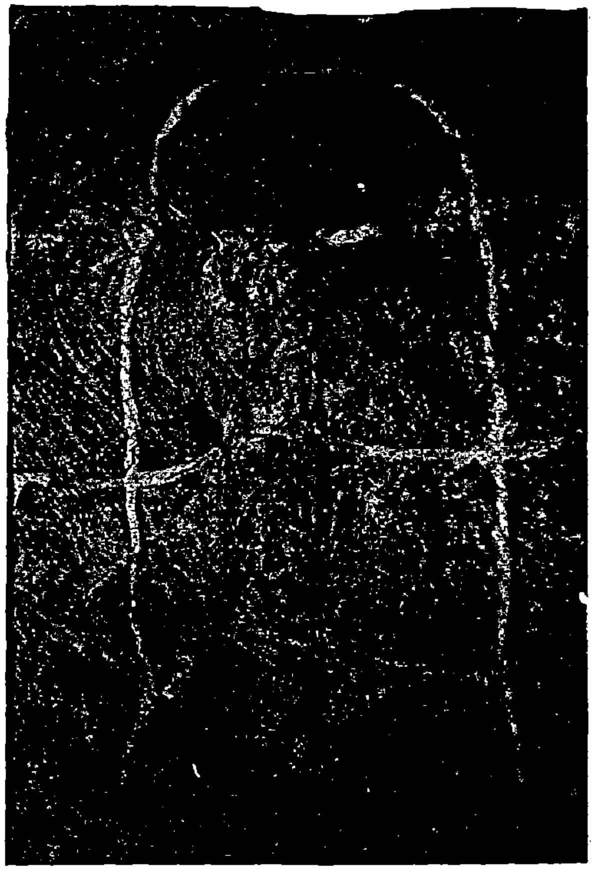
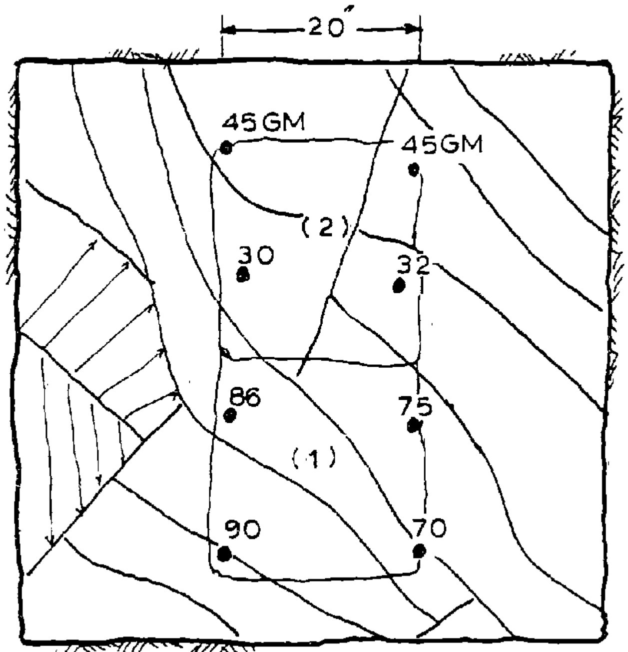


Figure 8-1 Breakage of V-Cut for a 24 in. Deep Round (Table 5.1)

The round was modified to adjust for the spacing and direction of the joints by placing the holes in the solid rock between the joints where it was possible to do so and adjusting the amount of explosive. The results of Shot Nos. 2 and 5 (Table 8-1) indicated that large fragments were created because of the joints and natural fractures (Figure 8-2). The results from the line shots showed that when the holes were placed in the solid good breakage from the face was obtained and consistently smaller fragments resulted.

The results of this round provided data for the design of the second 24 in. round, although the required powder factor was not firmly established nor the reasons for the occurrence of bootlegs determined. The objectives of the second were to further test the shield, to measure air blast and ground vibration, and to improve round design (Table 8-2). A powder factor of 0.10 lb/cu-ft was used in the central holes in the cut, and a factor of 0.125 in the corner holes of the cut, but all of the holes bootlegged which created both excessive air blast and ground vibration.

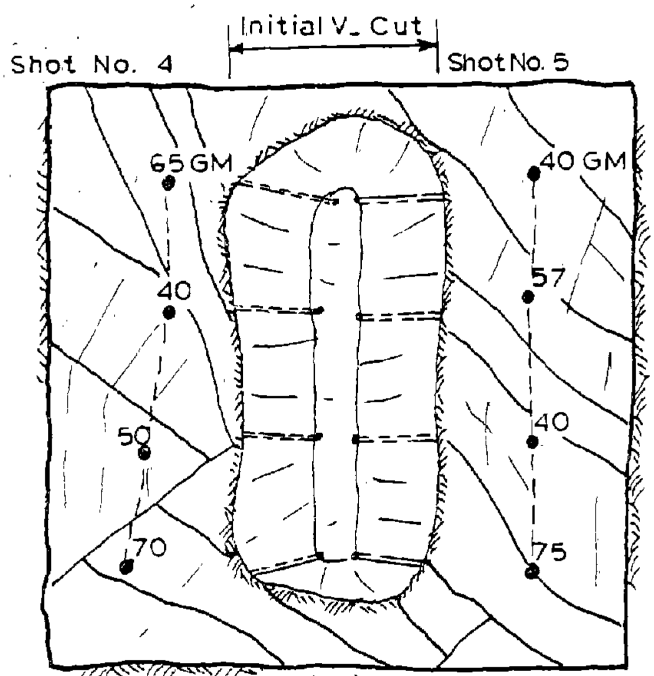
Shot Nos. 2 and 3 of the cut were loaded with 5.4 lb/cu-yd and broke to their full depth when they were fired. In the remainder of the shots, a powder factor of 0.09 lb/cu-ft was used, the corner holes being loaded with 10 to 20% more explosive which resulted in successful breakage and good fragmentation. The crown holes were loaded with less explosive because of the smaller spacing and the burden was small because of the irregularity of the face. However, the burden on the lifters was greater (Figure 8-3a) so the holes were loaded heavier with good breakage resulting. To minimize the amount of explosive fired in one shot, subsequent designs provided for firing the center crown and lifter holes first and the corner holes last.



(a)

a) Area 1 -- Location of Holes and Charge Size for Shot No. 2. Note that the Bottom Two Holes had Large Amounts of Powder. Shot No. 2 Resulted in Large Fragments.

Area 2 -- The Amount of Explosive was Reduced and Holes Placed Between Joints. Good Breakage and Better Fragmentation Resulted.

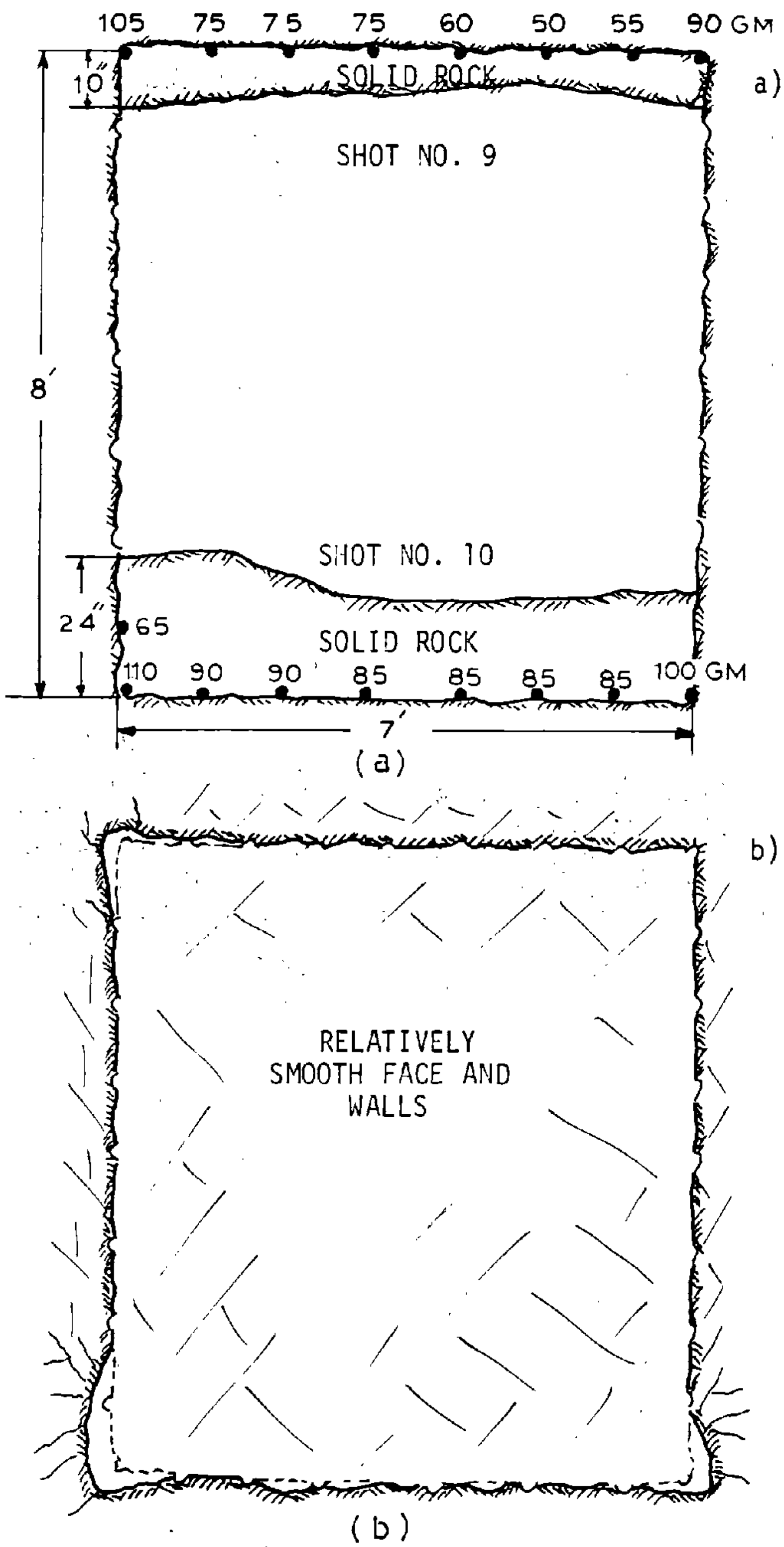


(b)

b) Bootleg Occured in the Bottom 2 Holes of Shot No. 4 Due to Large Burden.

Large Fragments were Created from the Joints in Shot No. 5 Due to the Intersecting of the Boreholes with the Joints.

Figure 8-2 Face Structure and Effect of Joints Upon Blasting (Table 5.1, First 24 in. Round).



a) The Crown Holes Loaded with Small Amounts of Explosive Due to the Unequal Burden. Note that One Extra Lifter Hole was Drilled at the Left Corner to Reduce the Burden.

b) After Shot Nos. 9 & 10, the Fractures Extended at the Corners and Damage in Bottom Corners Due to Heavier Loading in Lifter Holes.

Figure 8-3 Crown and Lifter Holes (a) Before, and (b) After Shot Nos. 9 & 10 (Table 5.2, Second 24 in. Round).

TABLE 8-2

24 Inch Deep Round
(Second Round)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) - (lb)	Comments
1(1)	V-Cut (central holes)	8	20(2)	18	18	2.7	540 - 1.19	6 bootlegs & partial breakage in the top 2 holes due to low powder factor
2	V-Cut (top)	4	20	18	18	5.4	460 - 1.01	Good breakage, several large fragments created by breakage to joints
3	V-Cut (bottom)	4	20	18	18	5.4	460 - 1.01	Good breakage, relatively small fragments
4	Relievers (right)	4	14	18	10	2.43	500 - 1.10	Shots No. 4 - 10 resulted in good breakage, small fragments & relatively smooth walls. Holes overloaded
5	Relievers (left)	4	14	18	5	2.43	433 - .95	
6	Relievers (right)	4	14	18	⊥	2.43	455 - 1.0	
7	Trim (right)	6	14	12	5	2.43	494 - 1.1	
8	Trim (left)	6	14	12	5	2.43	570 - 1.26	

TABLE 8-2 (Cont'd.)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) (lb)	Comments
9	Crown	8	14	12	5	2.43	580 - 1.28	
10	Lifters	9	24	12	5	2.43	795 - 1.75	

NOTE: (1) Instrumented shot

(2) Assumed to be the total distance between two pair of holes

*See Figure 8-4 for shot location.

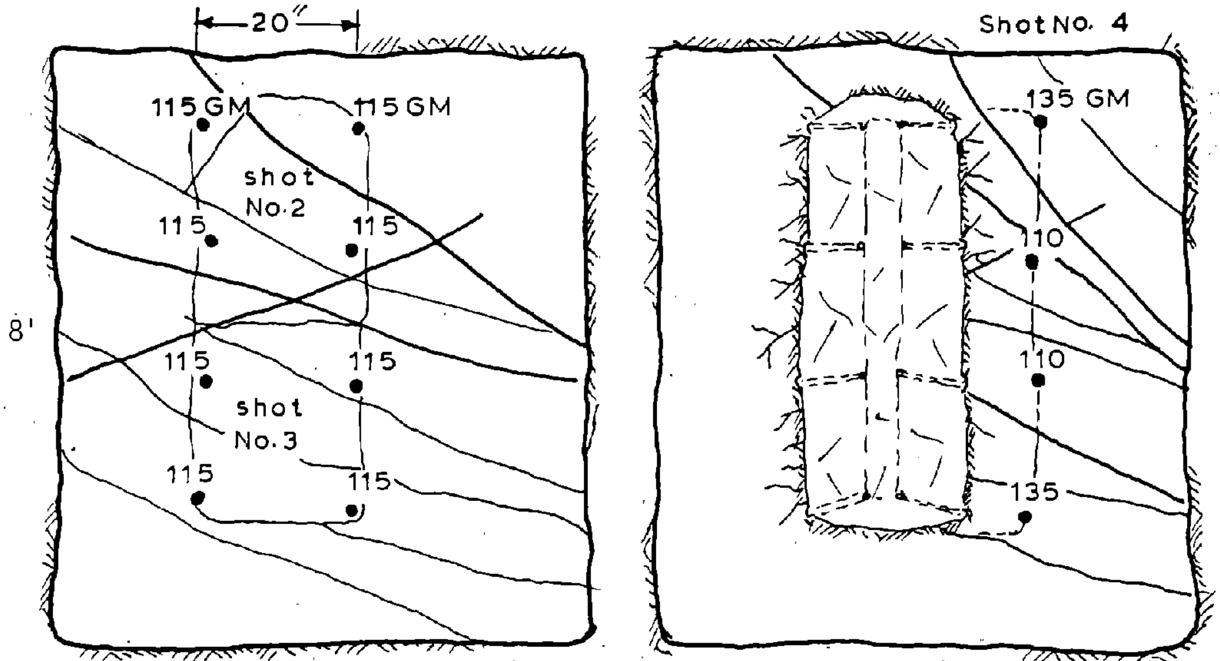
Good fracture control and relatively small fragments were obtained by using a modified hole placement pattern with the amount of explosive also adjusted to the structure of the face (Figure 8-4). Thus, the holes were placed in the solid rock between the joints wherever possible, and the weight of the charge was adjusted to the burden, spacing, and distance from the hole to the joint.

The test results (Figure 8-4a-d) showed that large joints caused the fractures initiated by the explosives to deviate and to extend beyond the "control" plane, and relatively large fragments were created because of two major joints.

The above experiments indicated that bootlegs were caused by low powder factor or misaligned holes or both.

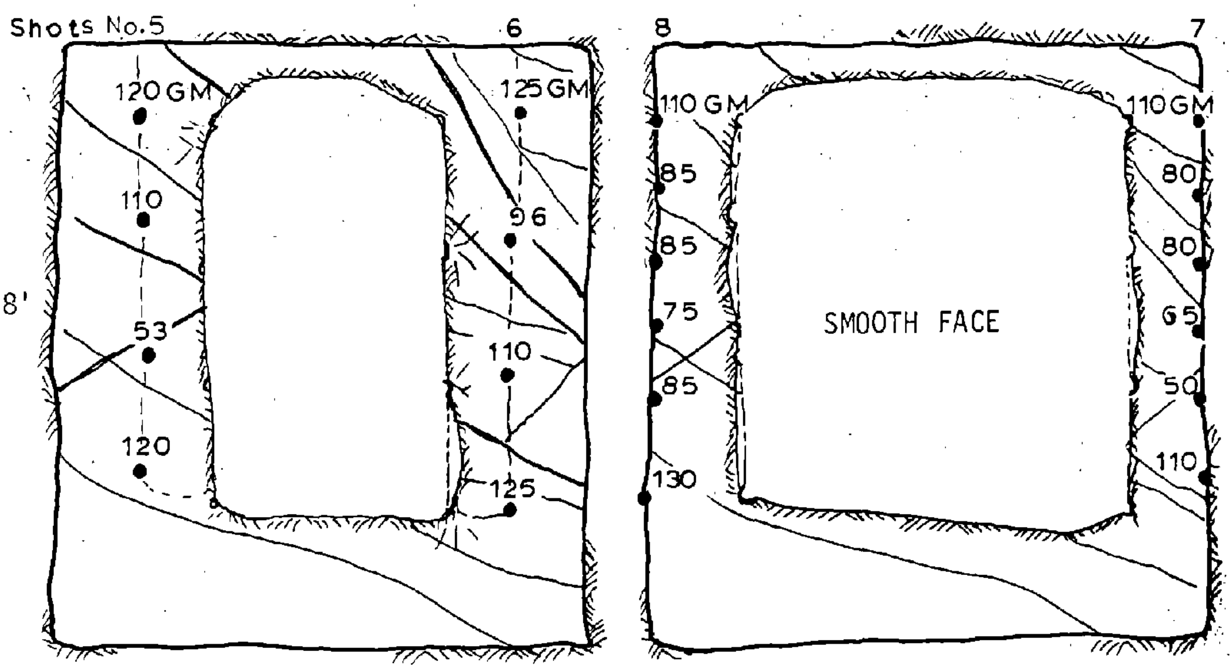
The primary purpose of tests with 30 in. deep rounds was to determine the behavior of the deeper rounds and breakage efficiency for the design of a prototype system. The round (Figure 7-4) was fired in eleven lines to keep the amount of explosive consumed at less than 1.5 lbs per shot. The "baby" cut (4 holes) was fired first. Then the 8 central holes were fired second, and the remaining holes were fired in the sequence shown in Figure 4.6.

Shot Nos. 1 and 2 (Table 8-3) bootlegged because of a low powder factor and the high strength of the rock, but caused some fracturing around the holes. Another line of holes was drilled on the right side 8 in. from the central line of the face, and the powder factor was increased from 0.2 to 0.35 lb/cu-ft. The holes were loaded using approximately 145 gm/hole, and the top and bottom two corner holes were loaded with approximately 10% more than the average. All 4 holes broke the rock to full depth, formed an evenly shaped V-cut, and small fragments were produced.



(a) Initial V-Cut

(b) Tearing Cracks After the Initial Cut was Blasted



(c) Tearing Cracks in the Right Reliever Line Due to Joints (Shot No. 4)

(d) Good Fracture Control from Shot Nos. 5 & 6

Figure 8-4 Face Structure and Effect of Joints Upon Blasting (Placing HOles in Solid Between Joints gave Effective Breakage. Table 5.2, Second 24 in. Round).

TABLE 8-3

30 Inch Deep Round
(First Round)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) - (lb)	Comments
1	Single Baby V-Cut (right)	4	8 (1)	19	40	2.7	130 - .29	Bootlegs in Shot Nos. 1 & 2 due to low powder factor
2	Single Baby V-Cut (right)	4	8	19	40	5.4	257 - .57	" " "
3	Single Baby V-Cut (left)	4	8	19	40	9.45	450 - .99	Good breakage, small fragments due to high powder factor
4	V-Cut (top)	4	30 (2)	19	27	10.8	1090 - 2.4	Bootlegs in Shot Nos. 4 & 5 due to low powder factor & misalignment of the holes
5	V-Cut (bottom)	4	28	19	27	10.8	1090 - 2.4	" " " "
6	V-Cut (top)	4	28	19	27	13.5	1362 - 3	Breakage to full depth. Large piece created by breakage to joints
7	V-Cut (bottom)	4	28	19	27	13.5	1360 - 3	Shot Nos. 7 - 12 resulted in good breakage, smooth walls, and small

NOTE: (1) Distance from the center of the drift

(2) Distance between two pair of holes

TABLE 8-3 (Cont'd.)

Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) (lb)	Comments
								fragments distribution. The small size fragments due to large powder factor
8	Relievers (right)	4	6	19	12	3.51	614 - 1.35	" " " "
9	Relievers (left)	4	6	19	12	2.97	520 - 1.15	" " " "
10	Relievers (right)	4	9	19	⊥	3.51	700 - 1.54	" " " "
11	Relievers (left)	4	9	19	⊥	3.24	647 - 1.43	" " " "
12	Trim (right)	5	12	14	5	2.97	667 - 1.47	Shot Nos. 12 - 15 resulted in good breakage
13	Trim (left)	5	12	14	5	2.97	667 - 1.47	Smooth walls & good fragments distribution
14	Crown (central holes)	5	14	14	5	2.97	680 - 1.5	" " " "
15	Lifters (central holes)	5	14	14	5	2.97	680 - 1.5	" " " "
16	Corner Holes	4	14	14	5	2.43	560 - 1.23	Good breakage & no damage in the ribs

To complete the free faces, the cut (8 central holes) was then fired in two sequences, the top and then the bottom 4 holes. The holes in Shot Nos. 4 and 5 bootlegged, which initially indicated that a higher powder factor might be required for this depth of cut. The holes (Shot Nos. 6 & 7) were reloaded with more powder, approximately 340 gm/hole, to give a powder factor of 13.2 lb/cu-yd. Rock breakage was successful and good fragmentation was obtained.

Two main factors were responsible for the high number of bootlegs. The first was the type of rock. It appears that the rock in the face was solid, confined, moderately jointed, and had high strength. The second and most important was the misalignment of the drill holes (Figure 8-5). The holes should be drilled in such a way that the bottoms of the pairs of holes meet at the apex and have the same spacing. However, after the blasting described above, it was discovered that the holes deviated from the required direction, which caused each pair of holes to have about 6 in. burden at the apex and 27 in. spacing between the bottoms of some of the holes. These alignment errors caused unbroken rock to remain as convex projections on the face (Figure 8-5). It was believed that simultaneous detonation would reduce the effect of the misalignment somewhat, but most of the drill holes had such great deviations that it was not effective in avoiding bootlegs.

After the initial V-cut was completed, the reliever holes were fired with the same powder factor utilized in the reliever holes as for other lines of holes with the same burden. In other words, if the breakage was successful for a certain volume and geometry of rock, then the powder factor was reduced in the following line shot. The reliever holes in Shot Nos. 8 and 9 had the same volume of rock. The breakage in Shot No. 8 was successful and small fragments were produced. Thus, in Shot No. 9, the powder factor was decreased from 3.5 to 3.0 lb/cu-yd. Breakage to full

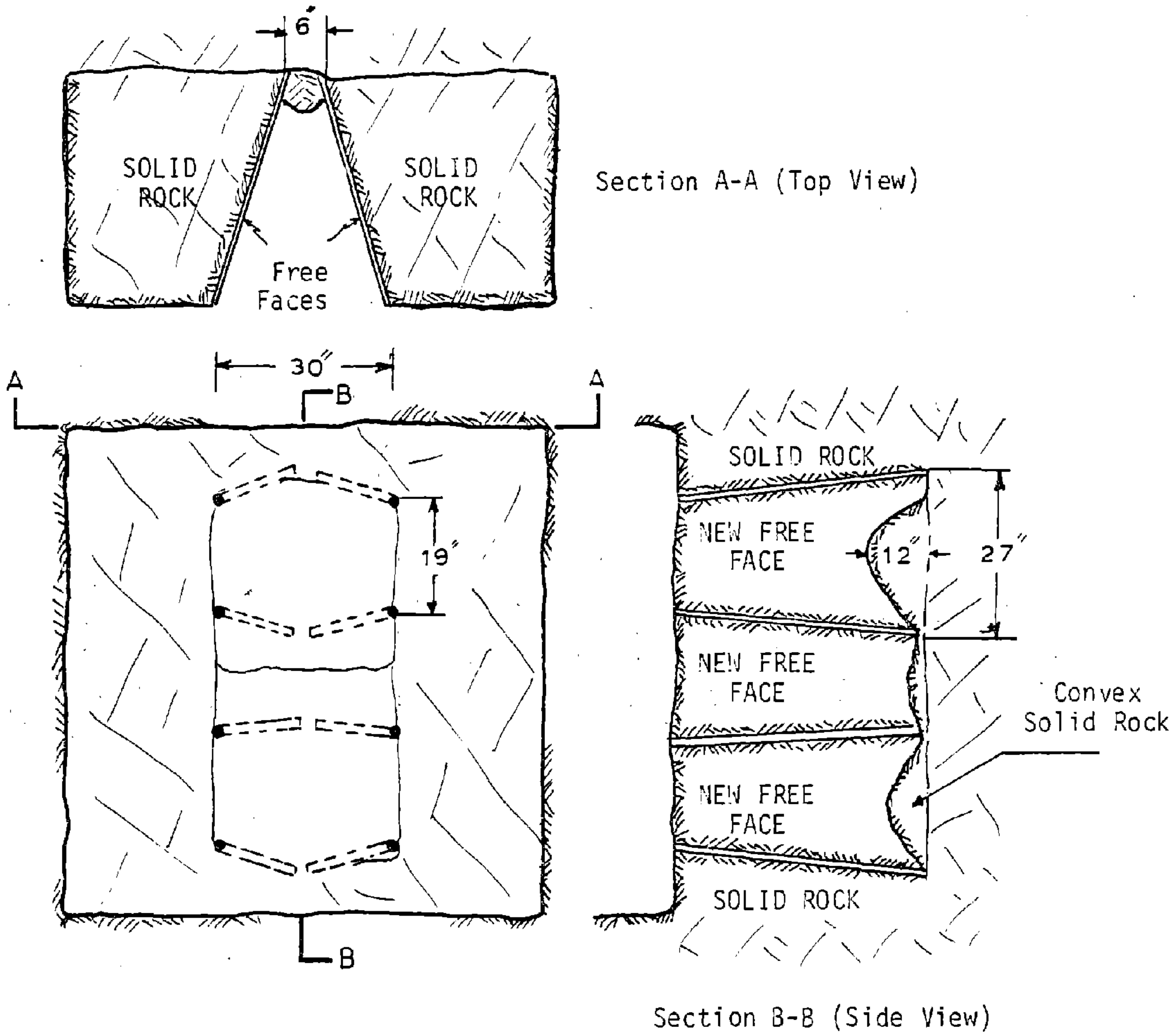


Figure 8-5 Effect of Alignment Errors of the Drill Holes (Table 5.4, First 30 in. Round).

depth, relatively smooth walls, and good fragment distribution resulted. Shot Nos. 10 and 11 were blasted in a manner similar to the previous two shots. The powder was decreased from 0.13 to 0.12 lb/cu-ft for Shot Nos. 10 and 11, respectively. Breakage to full depth was again achieved with good fragment distribution. It appears that the breakage obtained from the inclined holes (Shot Nos. 8 & 9) was better than those drilled perpendicular to the face (Shot Nos. 10 & 11) because of the greater angle of inclincation.

In order to obtain smooth walls and the desired size of the drift, the periphery holes were drilled outward at a 5° angle. In all of these holes, a powder factor of 0.11 lb/cu-ft was used which required 167 gm/hole. The breakage obtained from Shot Nos. 12 and 13 was successful and smooth walls were obtained (Figure 8-6).

As described earlier for the 24 in. deep rounds, it was found that the inside crown or lifter holes should be fired first, and then the corner holes would require less explosive. Thus, the five central holes in the crown and central lifters were fired first with a powder factor of 0.11 lb/cu-ft. Then the corner holes were fired using a powder factor of 0.09 lb/cu-ft. Figure 8-7 shows the blasting sequences and the amount of explosive per hole. Breakage to full depth and small fragments were obtained, and no damage was done to the ribs. The lifter holes must break and heave the rock to ensure good breakage and move the muck away from the face. Thus, the lifter holes were loaded with 10 grams more than average amount for the crown holes.

The geologic structure of the face as it was advanced changed from highly to moderately jointed. The newly exposed rock was relatively solid with few natural fractures. Figure 8-8 shows the face structure and the amount of explosive loaded in each hole. No large fragments were obtained because of the modified drill pattern and the utilization of more powder

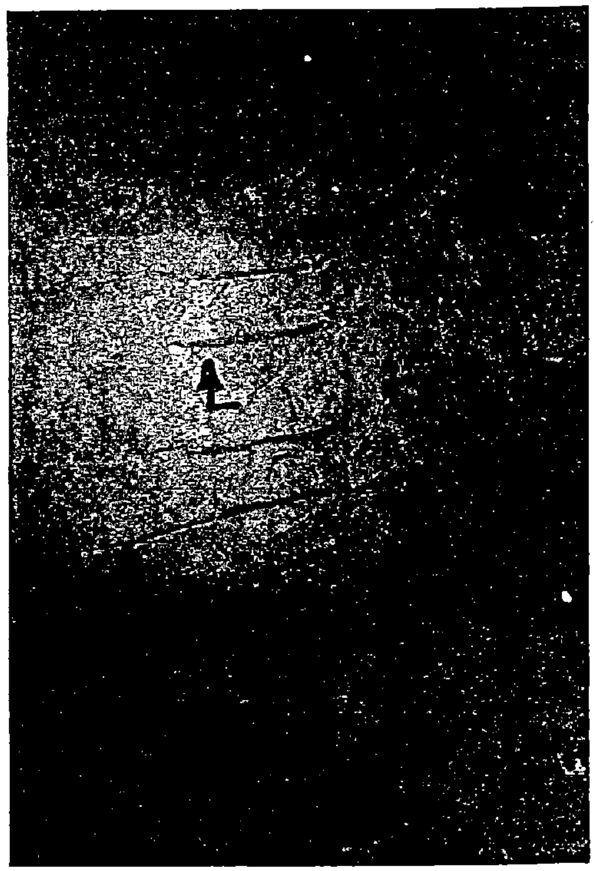
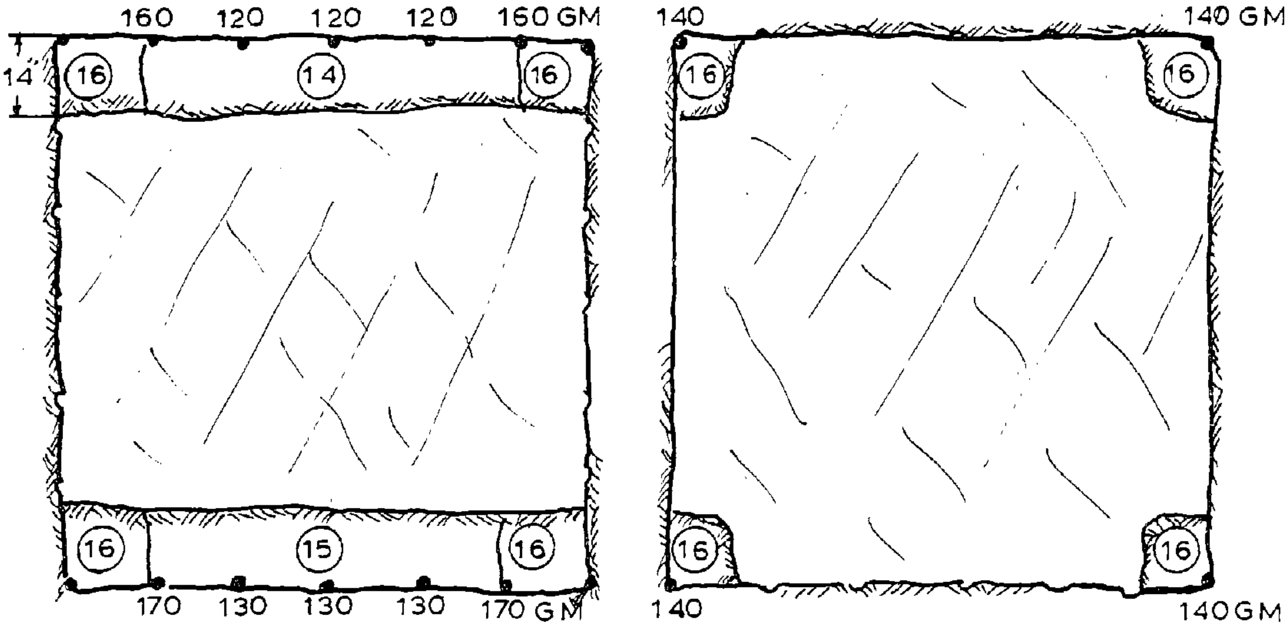


Figure 8-6 Smooth Wall Results from Blasting when Small Charge Method is Utilized in Hard Granite Gneiss Rock (CSM Experimental Mine).



(a) The Lifter Holes were Loaded with 10 gm more than Average Amount (136 gm/hole)

(b) Good Breakage Left Two Additional Free Faces at the Corners.

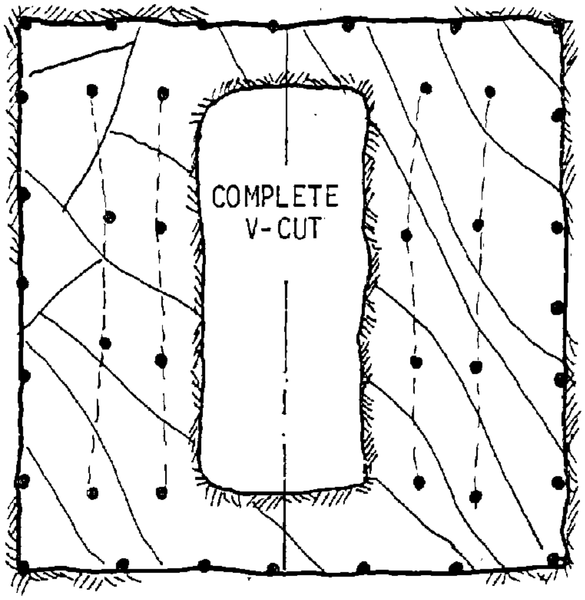
(c) No Damage was Done in the Ribs.

Figure 8-7 Firing Sequence in the Crown and Lifter Holes gave Smooth Unfractured Walls (NOTE: The Number in Circles Indicate Shot Order, Table 5.4, First 30 in. Round).

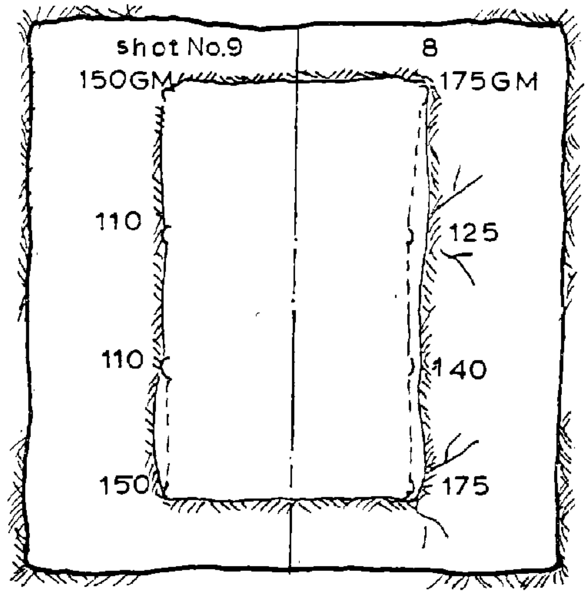
few natural fractures in the face. Figure 5.9 shows the face structure and the amount of explosive loaded in each hole. No large fragments were obtained because of the modified drill pattern and the utilization of more powder in each hole. All of the corner holes in a line were loaded 10% to 20% heavier than the average to ensure breakage. Experience showed that small joints did not cause the cracks to deviate. However, some cracks deviated and caused a small amount of damage to the walls. This was due to heavier blasting as in Shot Nos. 8 and 10, Figure 5.9. Also, the results indicated that the inclined holes (Shot No. 9) which used a lower powder factor were more effective in breakage than those drilled perpendicular (Shot No. 11) to the rock surface.

In general, the results obtained from the first round of 30 in. depth confirmed many of the specific requirements for the effectiveness of the small charge method. However, deviation of the holes in a line requires a higher powder factor for breakage. Thus, one additional line of holes in the initial V-cut of a second round was blasted in order to furnish more accurate data related to powder factor and drilling precision.

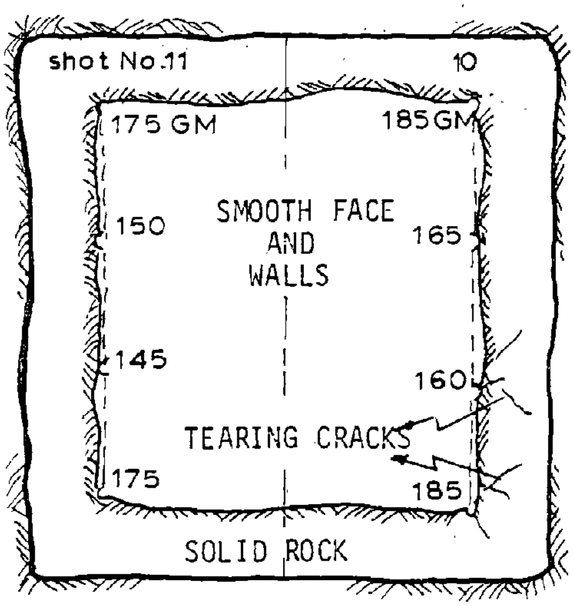
The holes of the initial V-cut were located and drilled the same as in the first 30 in. deep round, but the holes were carefully aligned before and during drilling in order to keep them in the target direction which permitted a decrease in powder factor. The shots and test results are presented in Table 5.5. Good breakage and good fragment distribution was obtained. The results of this test showed that the simultaneous detonation of well aligned holes produces efficient breakage and fragmentation. Typically, the powder factor was not the main reason for bootlegs (as in the first 30 in. round), but deviation in hole alignment required.



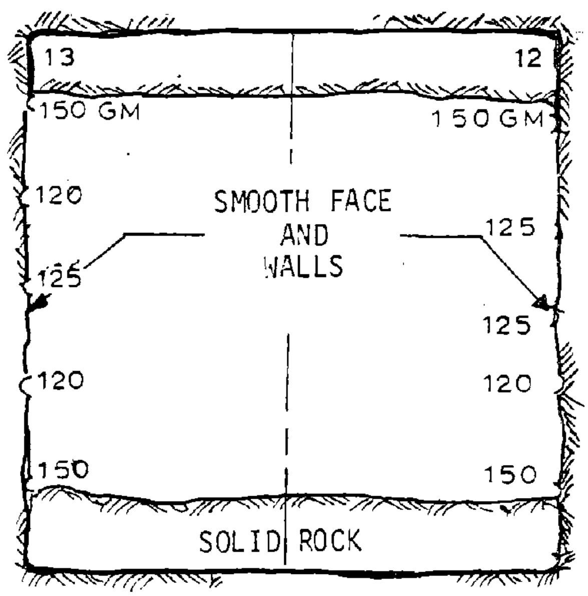
(a) Face Structure After the Initial Cut was Blasted (Shot Nos. 6 & 7).



(b) Tearing Cracks in the Right Reliever Line due to Joints and Heavy Blasting (Shot No. 8).



(c) Tearing Cracks due to Heavy Blasting (Shot No. 10).



(d) No Fracture Damage in the Walls After Shot Nos. 12 & 13.

Figure 8-8 Face Structure and Effect of Joints Upon Blasting (Placing Holes in Solid Between Joints gave Effective Breakage, Table 5.4, First 30 in. Round).

in each hole. All of the corner holes in a line were loaded 10% to 20% heavier than the average to ensure breakage. Experience showed that small joints did not cause the cracks to deviate. The results again indicated that the inclined holes (Shot No. 9) which used a lower powder factor were more effective in breakage than those drilled perpendicular (Shot No. 11) to the rock surface.

In general, the results obtained from the first round of 30 in. depth confirmed many of the specific requirements for the effectiveness of the small charge method. However, deviation of the holes in a line which creates a greater burden requires a higher powder factor for breakage. Thus, one additional line of holes in the initial cut of a second round was blasted in order to furnish more accurate data related to powder factor and drilling precision.

In a second 30 in. round, holes of the cut were located and drilled the same as in the first 30 in. deep round, but the holes were carefully aligned before and during drilling in order to keep them in the target direction. This permitted a decrease in powder factor. The shots and test results indicate (Table 8-4) good breakage and fragment distribution. The results of this test showed that the simultaneous detonation of well aligned holes produces efficient breakage, preventing bootlegs which occurred in the first 30 in. deep round where poor hole alignment required a greater amount of powder. Also, for the 30 in. deep round, it was found that the powder factor was from 7% to 16% higher than for the 24 in. deep rounds. The results obtained also indicate that the number of holes may be decreased by increasing the spacing and burden with the same powder factor and still maintaining efficient breakage and good fragmentation.

Evaluation of the size distribution of the fragments was important in order to accurately assess possible damage of the blast shield and the prob-

TABLE 8-4

30 Inch Deep Round
(Second Round)

*Shot No.	Location	No. of Holes	Dimension B(in.)	Dimension S(in.)	Holes Inclination (degree)	Powder Factor (lb/cu-yd)	Total Charge Wt (gm) (lb)	Comments
1	Single Baby V-Cut (right)	4	8 (1)	19	40	4.05	200 - .44	Shot Nos 1, 2, & 3 resulted in good breakage, unfractured smooth walls, & good fragments distribution
2	V-Cut (top)	4	30 (2)	19	27	5.8	586 - 1.3	" " " " "
3	V-Cut (bottom)	4	30	19	27	5.8	586 - 1.3	" " " " "

NOTE: (1) Distance from the center of the drift

(2) Distance between two pair of holes

*See Figure 7-6 for shot location.

lems of muck handling. The size and number of the largest fragments were measured as well as the throw (Tables 8-5 - 8-8). Figures 8-9 through 8-13 give the location and the various sizes of the fragments in the muck piles. The test results showed that the largest fragments (about 2 ft diameter) obtained from the initial cuts were found at a range of 2 to 9 ft from the face. The smaller fragments (less than 0.8 ft) were found against the shield and the drift walls. Therefore, the large fragments did not impact or damage the face of the shield. The effect of the impact of smaller fragments on the shield face was insignificant because of their low velocity and the high structural strength (Fy = 50 ksi) of the steel plate. Because the throw and velocity are decreased due to the small charge size and efficient utilization of the explosive, it is easier to contain the flying fragments.

The large fragments were usually created because of joints or existing fractures in the face (Tables 8-5 - 8-8). The number of large fragments was minimized in two ways. One approach was to reduce the amount of explosive in the holes close to joints and to place the holes in solid rock between joints and fractures. As noted above, this method resulted in efficient rock breakage, fracture control, and good fragment distribution (Figures 8-9 - 8-13).

It appears also that where gouge or intense fracturing is associated with joints with spacing greater than 1 ft, large fragments will be created. Also, the rock fracture was limited by joint planes as shown by the smooth surfaces of broken rock (Figure 8-11a).

The fragments obtained from Shot Nos. 1 and 2 of the second 30 in. deep round (Figure 8-13b) were very small. This was due to the high powder factor for these particular shots. Therefore, it is estimated that the powder factor required for the initial cut in the 30 in. deep round can be reduced by 10% with proper hole alignment.

TABLE 8-5

Fragment Distribution
24 Inch Deep Round
(First Round)

Shot No.	No. of Holes	No. of Fragments	Throw From Face (ft)	Dimension (ft)	Comments
2	4	4 Many Smaller	9 7-13	<2 <1	Large pieces created by breakage to joints & existing fracture in the face
3	4	1 6 Many Smaller	10 2-10 Against the shield	<2 <1 ≈0.5	Good fragment distribution
4&5	4 holes/shot	3 10 Many Smaller	2 2-10 Fragment line against the ribs	2 <1 <0.5	Large pieces created by breakage to joints & existing fracture
7&8	3 holes/shot 5 holes/shot	4 Many Smaller	Fragment line against the ribs	<1.5 <0.7	Relatively large fragment due to many intercept joints
9&10	8	2 2 4	Near the face	≈2 1.2 <0.5	Good fragments distribution

TABLE 8-6

Fragment Distribution
24 Inch Deep Round
(Second Round)

Shot No.	No. of Holes	No. of Fragments	Throw From Face (ft)	Dimension	Comments
2&3	4 holes/ shot	4 3 Many Smaller	2-5 10 Against the shield	<2 1 <0.5	Large pieces created by two major joints
4	4	2 10 Many Smaller	Fragment line against the ribs	1 <1 <0.7-0.4	Very good fragment distribution from Shot Nos. 4 & 5 due to high powder factor
5	4	3 5 Many Smaller		<1 ≤0.8 ≤0.6	" " " " "
6	4	1		1.5 x 1 x 1	Large piece created by joint
7&8	6 holes/ shot	10 5 Many Smaller		<1 <0.9 <0.7	Very good fragment distribution obtained from Shot Nos. 7, 8 & 9
9	8	3 Many Smaller	At the face	<1 <0.5	" " " " "
10	9	1 4 Many Smaller	At the face	2 x 2 x 2 ≈1 <0.5	Large piece due to large burden & not enough powder

TABLE 8-7

Fragment Distribution
30 Inch Deep Round
(First Round)

Shot No.	No. of Holes	No. of Fragments	Throw From Face (ft)	Dimension	Comments
3	4	1 Many Smaller	4 Against the shield	$\approx$1 $\lesssim$0.5	Very good fragment distribution
6	4	1 5 Many Smaller	6 10 Against the shield	2x1.5x1 <1 <0.6	Large piece created by breakage to joints
7	4	3 5 Many Smaller	8 Against the shield	<1.5 <1	Relatively small fragments
8	4	3 2 Many Smaller	Fragment line against the ribs	1 <1 <0.5	Good fragment distribution from Shot Nos. 8 - 16 except for large pieces created by breakage to joints in the face
9	4	1 Many Smaller		2x1.5 <0.6	
10	4	4 Many Smaller		1 <0.6	
11	4	4 Many Smaller		<1.5 <0.7	
12&13	5 holes/shot	3 5 Many Smaller	Fragment line against the rib	1 <1 <0.8	

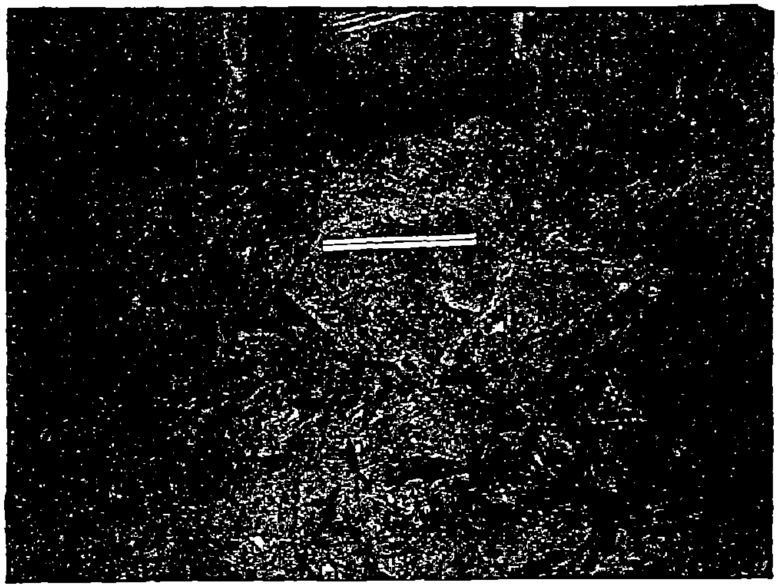
TABLE 8-7 (Cont'd.)

Shot No.	No. of Holes	No. of Fragments	Throw From Face (ft)	Dimension (ft)	Comments
14, 15 & 16	5 holes/shot 6 holes/shot	6 Many Smaller	At the face	<1 <0.9	

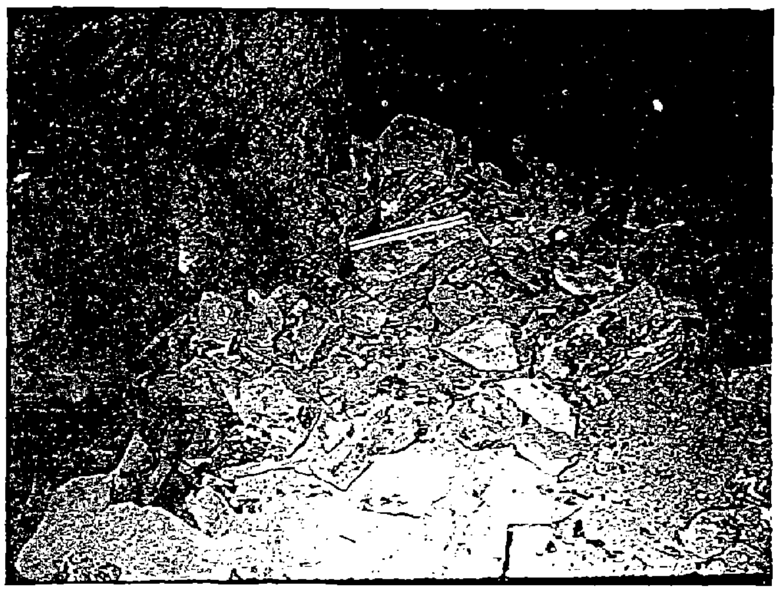
TABLE 8-8

30 Inch Deep Round
(Second Round)

Shot No.	No. of Holes	No. of Fragments	Throw From Face (ft)	Dimension (ft)	Comments
1	4	3 Many Smaller	8 Against the shield	<1 <0.8	Good fragment distribution obtained from Shot Nos. 1, 2, & 3. This is due to high powder factor
2	4	5 Many Smaller	6 Against the shield	1 <1	
3	4	4 Many Smaller	7 Against the shield	≈1 <0.7	



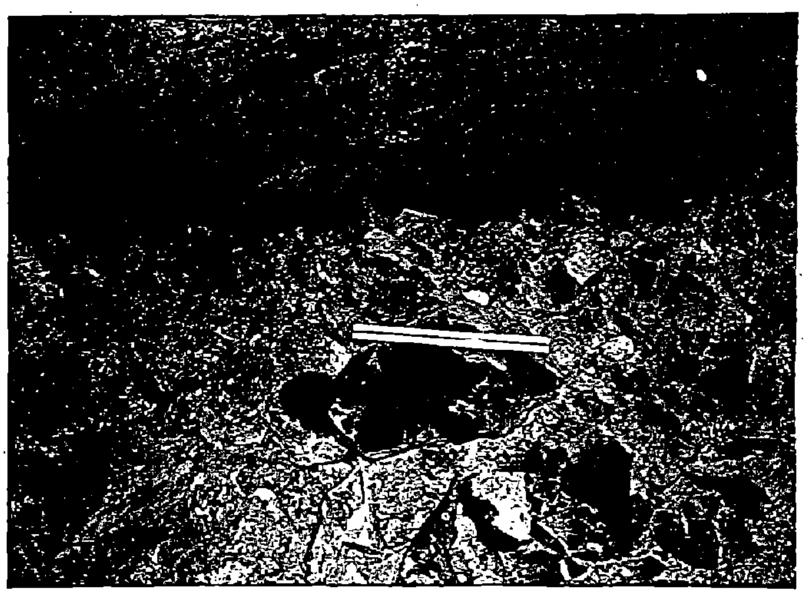
(a) Fragments Obtained from the Cut (Shot No. 2, Table 8-5).



(b) Fragments Obtained from the Reliever Holes (Shot No. 5, Table 8-5).

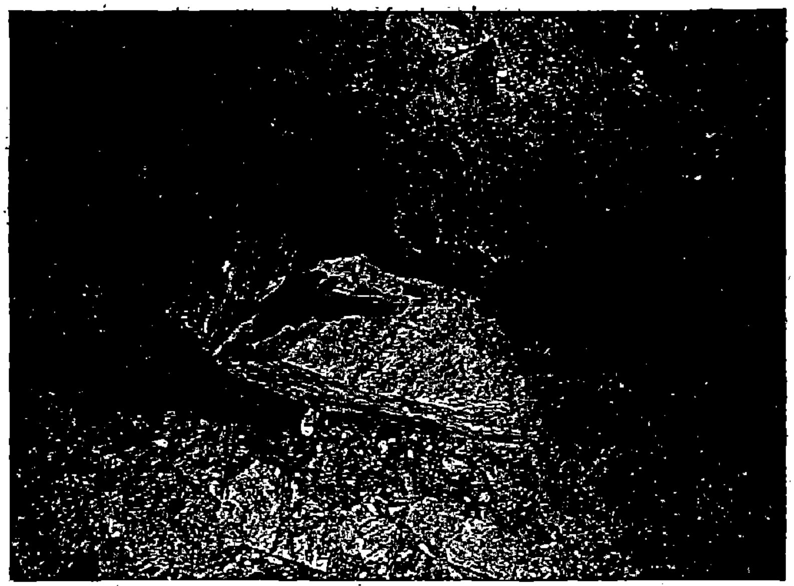
Figure 8-9 Fragmentation from Cut and Reliever Holes of the First and Second 24 in. Rounds, Respectively.

(a) Fragments Obtained from the Re-liever Holes (Shot No. 4, Table 8-6).



(b) Fragments Obtained from the Reliever Holes (Shot No. 6, Table 8-6).

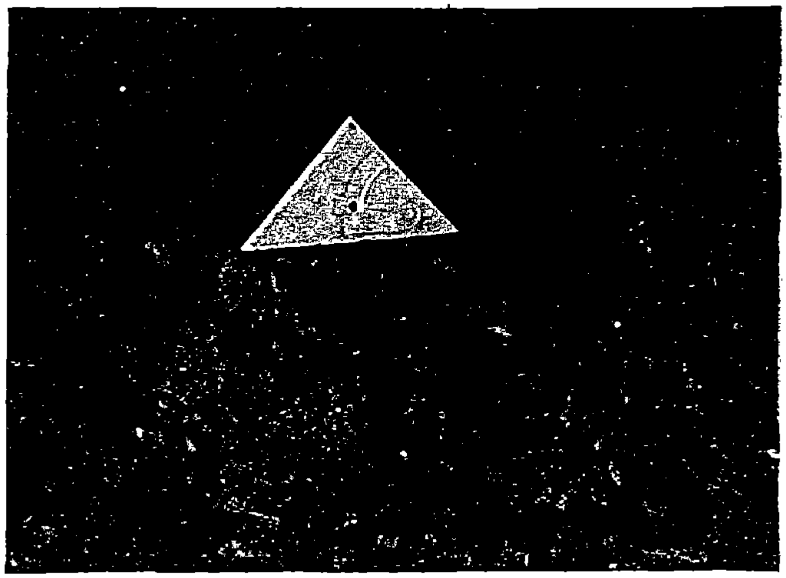
Figure 8-10 Fragmentation from Reliever Holes of the Second 24 in. Round.



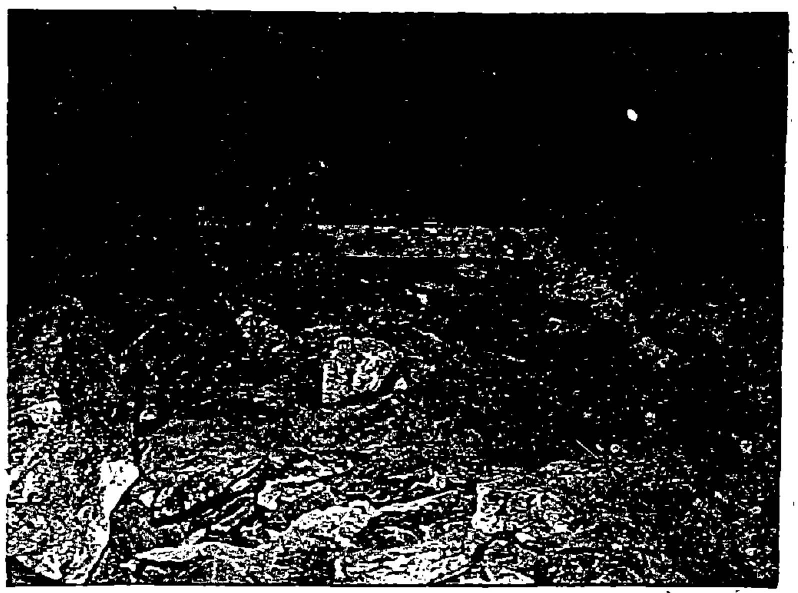
(a) Fragments Obtained from the Cut (Shot No. 6, Table 8-7).

(b) Fragments Obtained from the Cut (Shot No. 7, Table 8-7).

Figure 8-11 Fragmentation from Cut of the First 30 in. Round.

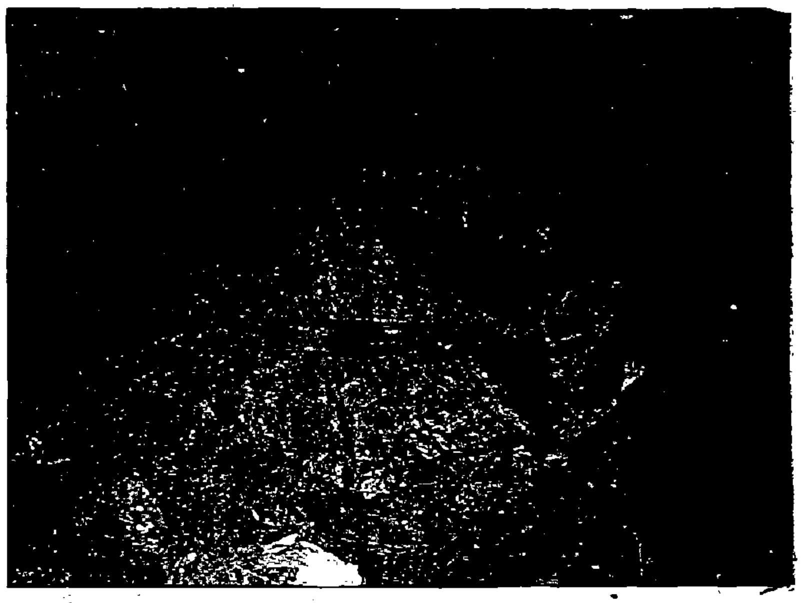


(a) Fragments Obtained from Reliever Holes (Shot No. 8 & 10, Table 8-7).

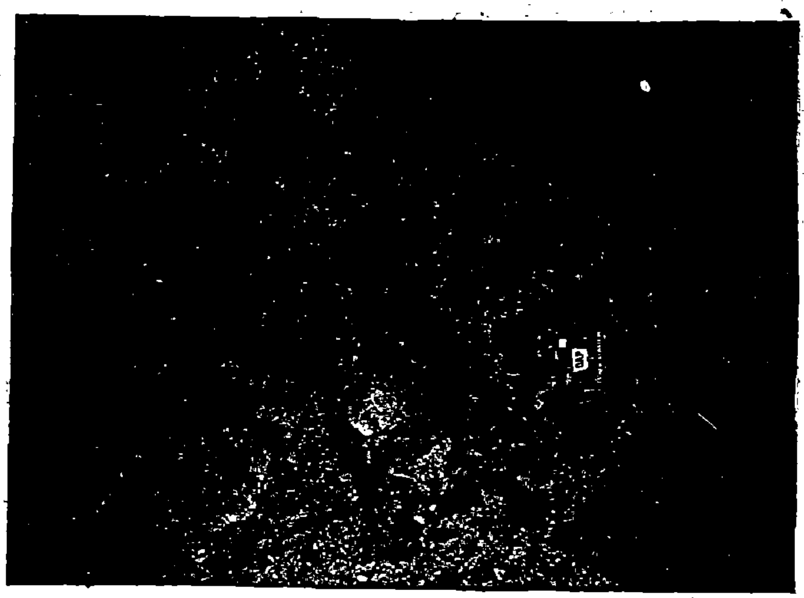


(b) Fragments Obtained from Reliever Holes (Shot No. 9, Table 8-7).

Figure 8-12 Fragmentation from Reliever Holes of the First 30 in. Round.



(a) Fragments Obtained from Reliever Holes (Shot No. 11, Table 8-7).



(b) Fragments Obtained from Cut (Shot Nos. 1 & 2, Table 8-7).

Figure 8-13 Fragmentation from Relievers and Cut of the First and Second 30 in. Rounds, Respectively.

Thus, for the small charge method utilizing well stemmed, confined charges, the fragment distribution is about the same as that for conventional blasting and provides for efficiency in muck handling.

Air Blast and Ground Vibration

During the initial tests, ten shots were instrumented in order to measure air blast (noise level) and wave particle velocity in the rock. Typical results of the noise level (Table 8-9) show that when the explosives were loaded and detonated properly, the noise was in the range of 96 to 107 dB. When one or more of the holes bootlegged, the maximum noise level was as high as 111 dB, due to the fact that the shield was moved several inches which resulted in reduced sealing.

The ground vibration (particle velocity) was measured at three locations along the drift and the square root of the sum of the squares (RSSQ) of the longitudinal, transverse, and vertical were calculated (Table 8-10). The values of the peak particle velocities (RSSQ) vs cube and square root scaled distance are summarized in Tables 8-11 and 8-12. A logrithmic regression analysis was made to determine whether the data could be scaled by either of the above two methods. The analysis showed that the regression factor for square root scaling was 0.84 and for cube root scaling was 0.81. This means that the scaling exponent b = 1/2 fits the data slightly better than scaling exponent b = 1/3 (Figures 8-14 & 8-15).

The equations for the data scaled by the square and cube root methods are:

$$V = 7.31\ (D/W^{1/2})^{-1.54} \quad (8.1)$$

$$V = 6.46\ (D/W^{1/3})^{-1.55} \quad (8.2)$$

TABLE 8-9

Noise Level Measurements
(5 Ft Outby Shield)

Shot No.	Total Charge Weight (gm)	(lb)	Noise Level (dB)	Comments
2	321 -	.71	96	Breakage to full depth (Shot Nos. 2 & 3)
3	151 -	.33	107	" " " " "
4	225 -	.5	112	Bootlegs in two holes
5	212 -	.47	106	Breakage to full depth (Shot Nos. 5 & 6)
6	176 -	.39	107	" " " " "
7	180 -	.4	105	Bootleg in one hole
8	256 -	.56	107	Breakage to full depth
9	334 -	.74	108	Bootlegs in three holes
10	170 -	.37	---	Recording problem
1*	540 -	1.19	111	Bootlegs in eight holes

*Indicates the shot of the 2nd 24 in. deep round

TABLE 8-10

Particle Velocity Measurements (in/sec)

Geophone No.	1				2				3			
Distance from Face	19 ft				39 ft				61 ft			
Particle Direction	Long	Tran	Vert	RSSQ	Long	Tran	Vert	RSSQ	Long	Tran	Vert	RSSQ
Shot No.												
2	.20	.01	.01	.20	.01	.05	----	----	.10	.05	.10	.15
3	.004	.004	.001	.006	.003	.003	.004	.005	.002	.001	.001	.002
4	.042	.042	.032	.067	----	----	----	----	.005	.005	.003	.008
5	.021	.002	.026	.034	.013	.006	.006	.016	.003	.004	.004	.006
6	.021	.016	.026	.037	.005	.011	.005	.013	.005	.003	.004	.007
7	.021	----	.026	----	.013	.008	.008	.017	.004	.007	.007	.011
8	.016	.011	.005	.020	.003	.008	.007	.011	.005	.009	.004	.011
9	.026	.021	.032	.046	.008	.013	.008	.017	.005	.009	.005	.011
10	.016	.011	.008	.021	.007	.007	.008	.013	.005	.008	.003	.010
1	----	----	----	----	.040	.016	.021	.048	.017	.012	.015	.026

TABLE 8-11

RSSQ Particle Velocities (in/sec) and Scaled Distance (ft/lb$^{1/3}$)

	Geophone No.		1		2		3	
Shot No.	Total Charge Wt (gm)	(lb)	RSSQ	D/W$^{1/3}$	RSSQ	D/W$^{1/3}$	RSSQ	D/W$^{1/3}$
2	321 -	.71	.20	21.3	----	43.8	.15*	68.5*
3	151 -	.33	.006*	27.4*	.005	56.1	.002	87.8
4	225 -	.5	.067	24.0	----	49.3	.008	77.1
5	212 -	.47	.034	24.5	.016	50.3	.006	78.6
6	176 -	.39	.037	26.1	.013	54.1	.007	83.7
7	180 -	.4	----	25.9	.017	53.1	.011	83.0
8	256 -	.56	.020	23.0	.011	47.2	.011	73.8
9	334 -	.74	.046	21.0	.017	43.2	.011	67.6
10	120 -	.37	.021	26.4	.013	54.1	.010	84.6
11	540 -	1.19	----	17.9	.048	36.8	.026	57.6

*Values not included in regression analysis

TABLE 8-12

RSSQ Particle Velocities (in/sec) and Scaled Distance ($ft/lb^{1/2}$)

Geophone No.				1		2		3	
Shot No.	Total (gm)	Charge	Wt (lb)	RSSQ	$D/W^{1/2}$	RSSQ	$D/W^{1/2}$	RSSQ	$D/W^{1/2}$
2	321	-	.71	.20	22.6	----	46.4	.15*	72.5*
3	151	-	.33	.006*	32.8*	.005	67.4	.002	105.4
4	225	-	.5	.067	27.0	----	55.4	.008	86.6
5	212	-	.47	.034	27.8	.016	57.1	.006	89.3
6	176	-	.39	.037	30.5	.013	62.6	.007	98.0
7	180	-	.4	----	30.2	.017	61.9	.011	96.9
8	256	-	.56	.020	25.3	.011	51.9	.011	81.2
9	334	-	.74	.046	22.2	.017	45.5	.011	71.1
10	120	-	.37	.021	31.0	.013	63.7	.010	99.7
1	540	-	1.19	----	17.4	.048	35.8	.026	55.9

- probable error in reading

*Values not included in regression analysis

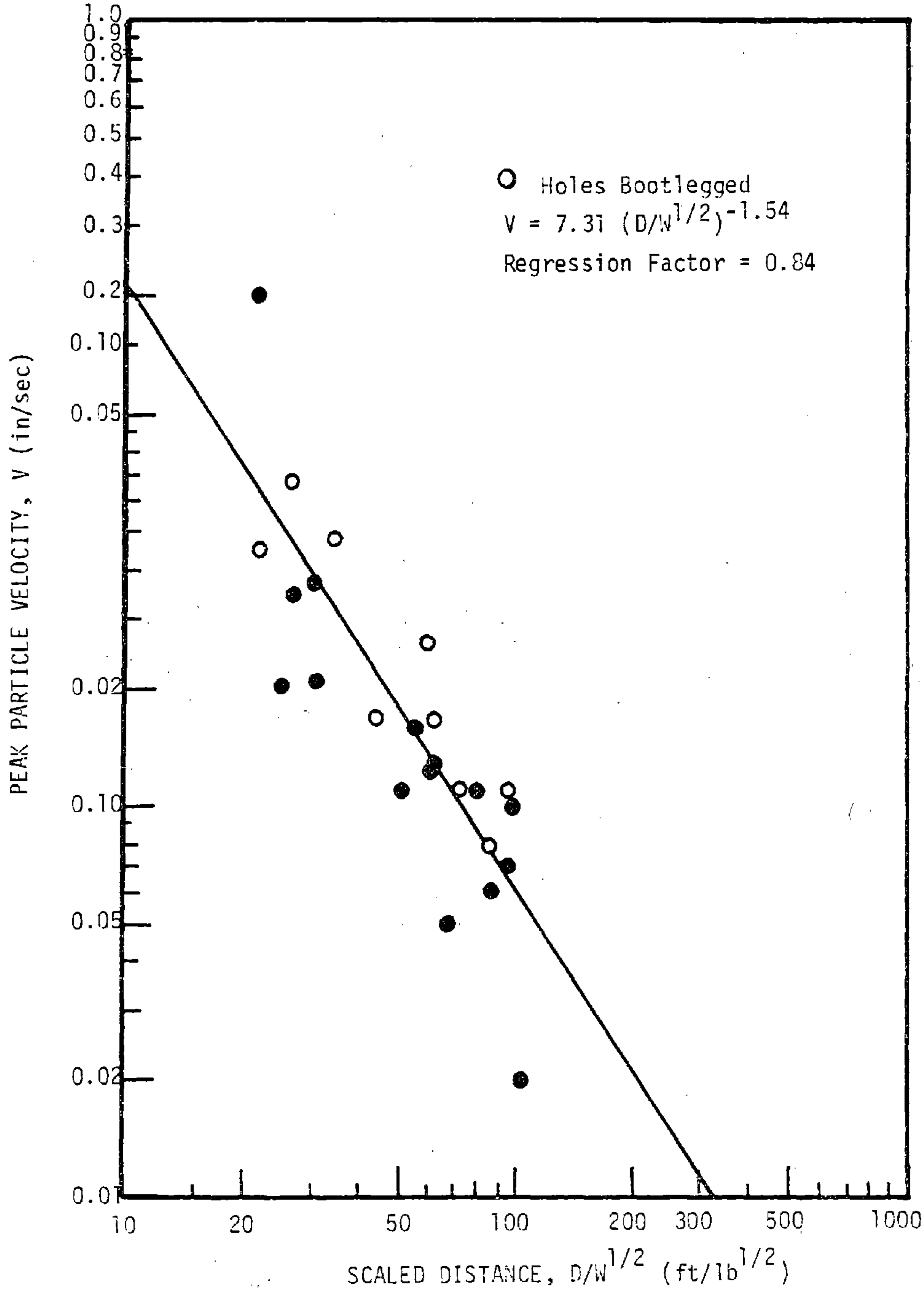


Figure 8-14 Particle Velocity vs Scaled Range for Small Charges (CSM, Experimental Mine) Square Root Scaling.

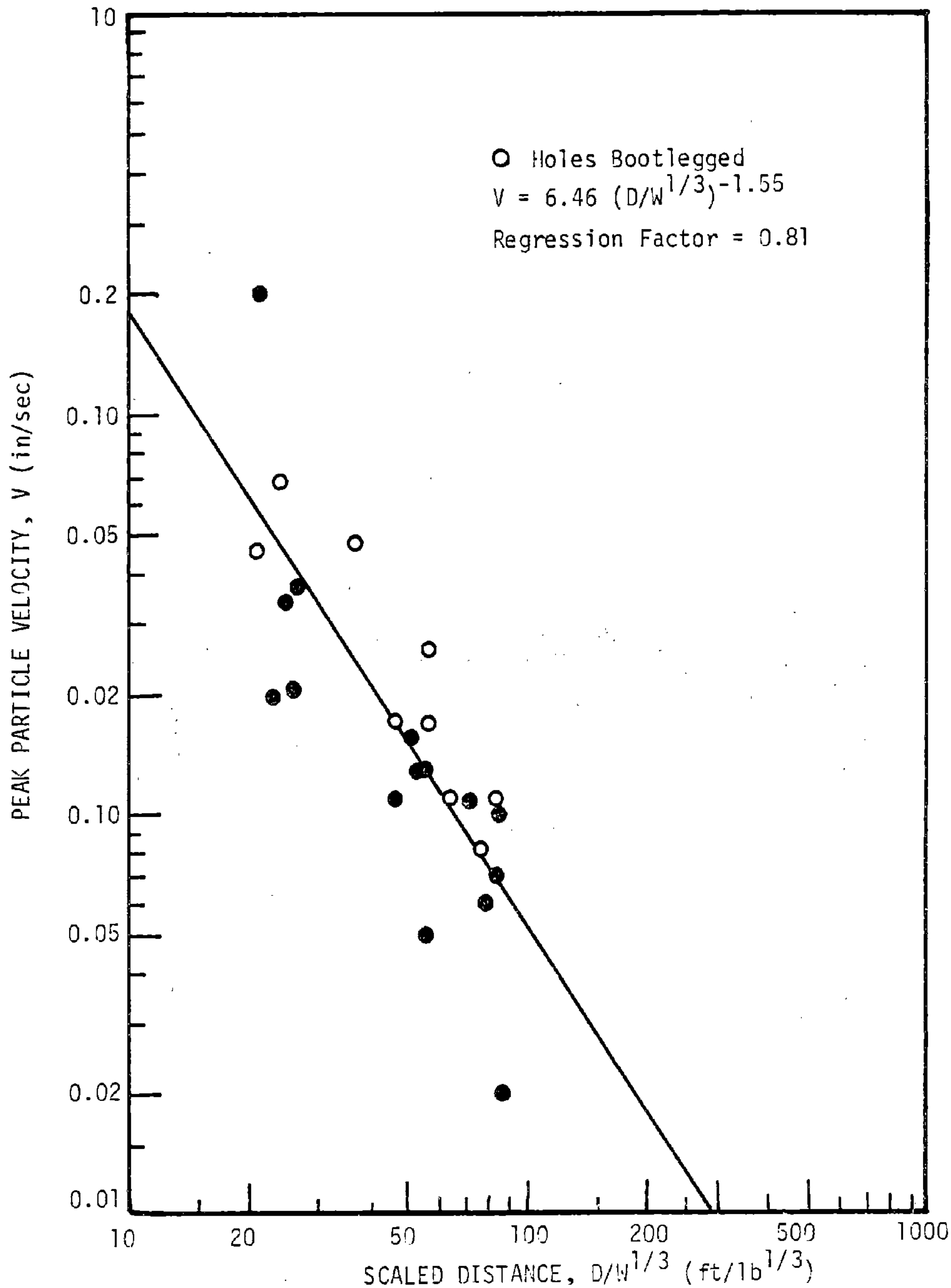


Figure 8-15 Particle Velocity vs Scaled Range for Small Charges (CSM, Experimental Mine) Cube Root Scaling.

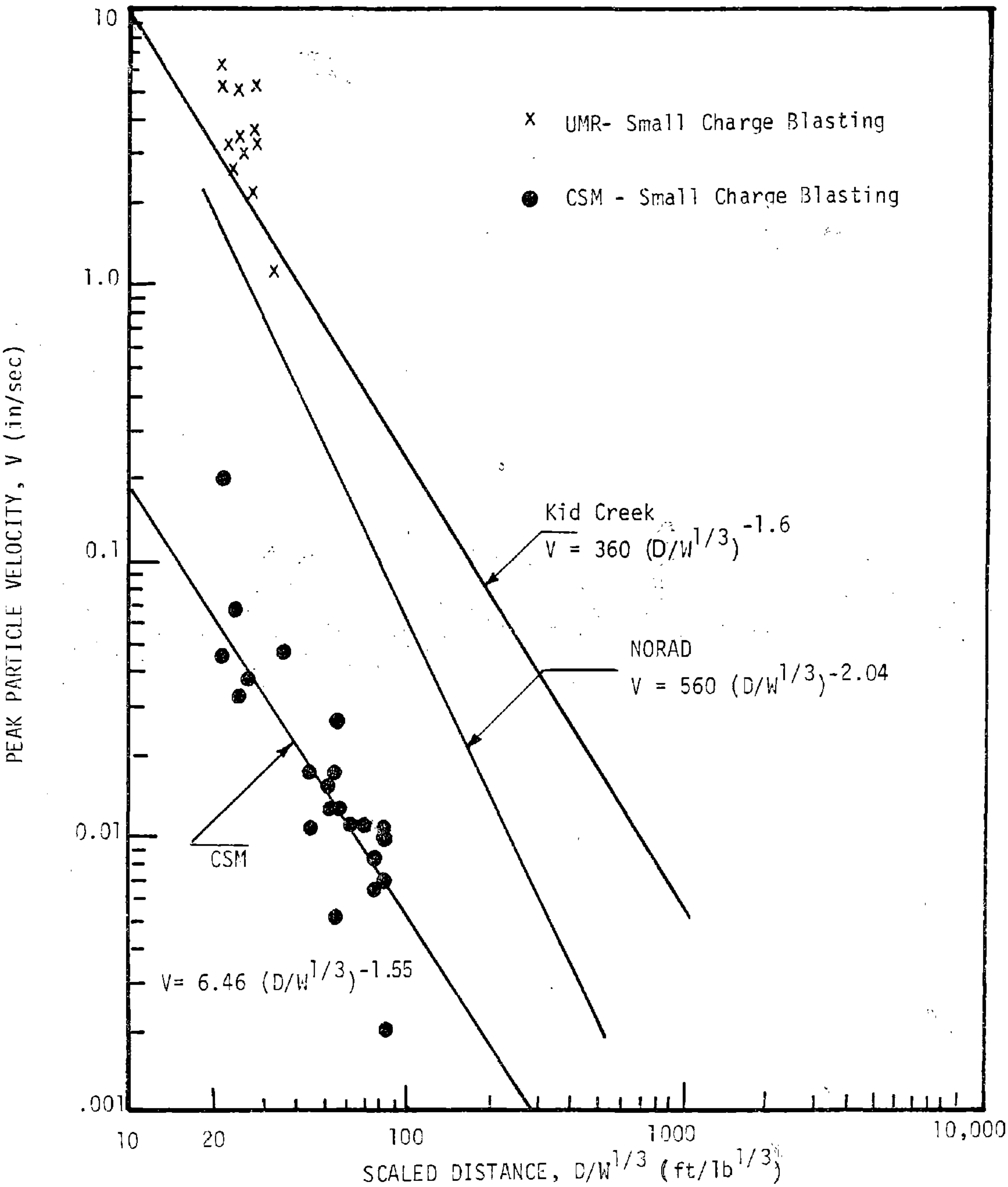


Figure 8-16 Comparison of Particle Velocities vs Scaled Distances for Blasting in Four Different Underground Operations.

cularly the geometry of the drift had a significant influence on the peak velocity because of the normal reflection of the wave. The vibration from conventional blasting also affects the support stability and causes fracture damage to the surrounding rock, while small charge blasting causes only minor damage to the rock support where the peak particle velocities are low. This was also evidenced by the stability of the rock on the roof and walls, which did not require barring down after blasting.

The fact that the peak velocities measured in granite-gneiss were so much lower than those at the other sites indicated in Figure 8-16 was in part due to the location of the geophones (Figure 7-1). These were placed on the rib of the drift leading to the working face, which meant that the attenuation of the wave before it reached the geophones was affected not only by properties of the rock and the distance traversed, but the geometry of the blast-geophone layout was such that the waves in essence had to "turn the corner" to reach the geophones.

The magnitudes of the peak velocities at these locations are important in determining the effects of blasting on the walls outby the shield, and the increased attenuation due to the geometry of the tunnel is another critical factor which is in favor of the installation of rock support outby the shield while blasting operations are proceeding at the face. The magnitudes of the peak velocities in the direction of the axis of the tunnel and in all directions beyond a plane coincident with the face would be of greater magnitude as shown by the upper curves in Figure 8-16.

The data obtained to date, therefore, indicate that the higher values would be applied to determine the effects on structures or personnel located in this half-space either in openings underground or on the surface if the blasting is being carried out at shallow depths. The effects in the walls of the tunnel appear to be at least an order of magnitude lower, which, as stated above, favors the acceptable operation of the small charge drill and blast system.

Calculation of Explosive Parameters

Only an aluminized slurry explosive was used in the experimentation except for comparative crater tests, and the last two shots of the second 30 in. deep round in which dynamite was utilized. The dynamite created a slightly larger crater than the slurry.

The detonation state and explosion state parameters of the gelatin dynamite 65% were determined using the composition data from the manufacturer. The composition of the slurry explosive was not available. To make predictive calculations regarding the performance of the dynamite explosive, a suitable equation of state and thermohydrodynamic equation, etc., were used for computer calculations. The calculation methods are described by Clark (1959). The output data for the detonation and explosion states are summarized in Table 5.14 for three different densities. The oxygen balance and the heat of formation were calculated from the composition, i.e., the gram atoms per 100 gram of the elements in the explosive. The primary constituent elements are hydrogen, nitrogen, oxygen, carbon, and sodium. Calculations showed (Table 8-13) that the dynamite has a slightly negative oxygen balance.

Stress Reinforcement and Effect of Joints and Fracture on Breakage

As described previously, the stress waves generated from simultaneous firing of two adjacent confined explosive charges reinforce each other and thereby produce additive stresses which enhances the initiation and propagation of the fractures between the holes, which result in effective breakage from the face and good fragmentation. The peak stress and wave energy

TABLE 8-13

Gelatin Dynamite and Its Detonation and Explosion Properties

Density gm/cc	Oxygen Balance	Heat of Formation kcal/kg	Detonation State				Explosion State		
			Temp °k	Pressure ATM	Heat (Q) kcal/kg	Velocity m/sec	Temp °k	Pressure ATM	Heat (Q) kcal/kg
1.1	−0.078	930	3,528.88	76,517.20	890.26	5,411.25	3,016.67	35,233.6	888.91
1.2	↓	↓	3,559.28	92,293.32	892.394	5,807.8	3,018.82	42,401.1	890.42
1.3			3,591.36	110,384.29	893.12	6,210.39	3,011.25	50,871.41	892.41

penetrating the walls and the roof of the drift is reduced because of the small size of charges. Small fractures along the line of holes were observed even when the holes bootlegged.

The analyses of the results of the blasting show that the structural geology of the face is a more important factor in affecting small charge blasting than in conventional blasting. The major joints and natural fractures affect the reinforcement of the stress waves, but only when extensive fractures or large open joints exist in the face, the energy of the stress wave is depleted and poor breakage results. In order to overcome these effects, a modified drilling pattern and powder factor are required. Thus, the blast is more effective if boreholes are placed in the solid rock between the joints with a slightly reduced powder factor, which procedure resulted in good fracture control and better fragmentation.

The joint and fracture patterns of the rock blasted in this experimentation were relatively consistent, and the breakage of the rock in each round indicated the adjustments in the positioning of holes which were required for efficient breakage. An experienced miner could readily make the minor adjustments in round design to conform to the geologic structure.

Finally, the simultaneous detonation of small charges has much in common with smooth blasting where the confining stresses in the rock around the opening acts to improve the direction control of the fractures, reduces the overbreak and deviation of the cracks, and enhances the stability of the walls of the excavation.

Chapter 9

CONCEPTUAL EXPLOSIVES LOADING AND FIRING SYSTEM

Introduction

One of the unique features requisite for the most effective breakage for the small charge blasting system is that two or more adjacent holes in a line be fired simultaneously. The advantages of this procedure have been well proven in several full scale tests, i.e., good fracture control, reduced ground vibration per unit weight of explosive, and containment of the air blast and flying fragments, which in turn will permit the system to be operated in a continuous manner.

The system as currently conceived will require the following:

1. Timing of charges in a single blast to within ± 100 microseconds.
2. That the charges be well stemmed.
3. That the charges be insensitive enough to be safe with mechanical handling.
4. That the blasting caps be insensitive enough to be inserted in the charge loaded mechanically.
5. That there be no excessive accumulation of explosives at or near the working face, i.e., inside of the shield.
6. That the gaseous detonation products be non-toxic.
7. That the explosive loading system can operate efficiently without breakdown.
8. That the loading system can load holes in any part of the tunnel face.

Explosive

As noted earlier in this report, an aluminized slurry explosive was utilized for most of the experimentation. This explosive has several advantages for use in the proposed automated system. First, all of the ingredients are non-explosive by themselves. Secondly, the strength and sensitivity of the explosive can be varied by a wide range. Its use for automated loading would require that one cartridge be loaded into the system at a time, which will require only moderate advances in the technology of the fabrication of explosives, and the problems appear to be easily solvable.

Explosive Loading and Firing System

In the experimental small charge blasting performed to date, detonation cord has been employed to obtain the precision timing required for simultaneous initiation of the charges in one shot. For an operational system, however, detonating cord has some disadvantages. The first is that the explosion of the cord is not confined in the rock, and hence, it contributes to the air blast behind the shield. The second is that considerable quantities of the cord would be required on both sides of the shield and a system to handle the detonating cord may be more complicated than that for EBWs. A third possible difficulty arises in the cutting, tying, and priming of the cord. In view of these and other factors, this method of ignition has not been considered further at the present time for the conceptual design of the proposed system. Its use is considered to be technically feasible, however.

From the outset of the small charge blasting project, it was felt that the most suitable method for precision timing of adjacent charges would be exploding wire blasting caps. The cost of high precision research type exploding wire caps is too high, but upon inquiry to an explosives manufacturing company, it was found that the EBW's of the required accuracy can be purchased at approximately the same price as standard blasting caps. As far as cost is concerned then, it will be a matter of reducing the number of caps to an optimum in keeping with the other requirements of the blasting. In any case, the total explosives costs per foot will be greater than for conventional blasting, but tradeoffs will more than compensate for this additional cost.

Thus, the most successful operation of the small charge blasting system requires that four or more small charges in line be fired within 0.2 milliseconds of each other, that the charge size be adjusted to the burden and geology of the face, that the charge be well stemmed, and that the explosive and initiator be safe to handle by an automated explosive loading device.

In the experimentation to date, the charge size has been varied according to the burden on the hole, its depth, and the solidity of the face around the hole. For a practical blasting system, it will be desirable to use only one or two sizes of charges, whereas in the research, most of the experimentation was specifically planned to determine the minimum size charge to break the rock effectively, and very few of the holes were overloaded. In operation blasting with a small charge system, the controlling factor will be that the charges be large enough to break the rock with virtually zero bootlegs.

One of the first conceptual designs of an explosive loading system included the plan for an apparatus for mixing the explosive ingredients of a slurry outside of the shield. However, the proper mixing of slurries requires careful control, and this type of operation could not be carried out as planned with the present state-of-the-art of explosive fabrication.

The proposed system will, therefore, be based upon the concept that the charges and the stemming will be prefabricated.

In the projected first experimental explosives loading and blasting system, it will not be possible to incorporate all of the features that are desired in the final design of a fully automated system. Provision for operational safety and efficiency can be made only after research experience and after more sophisticated devices have been designed, built, and tested.

The operations for which the system must be designed are:

1. Transport of explosives from the magazine to a position near the shield.

2. Transport of stemming and blasting cap assembly to a position near the face.

3. Storage of the above items near the shield for automated transport through the shield.

4. Transport of individual items through the shield.

5. Attachment of cap and stemming to the explosive cartridge.

6. Place assembled charge and stemming in loading device.

7. Load into hole and compress explosive and stemming.

8. Attach lead wires to current source and fire shot.

The transport of the explosives from a magazine to a position outside of the shield can be accomplished without any difficulty. It is planned that the explosive and stemming each be packaged, with the blasting cap already placed in the stemming. Large numbers of these can be placed in slotted containers which can be suspended from the wall of the tunnel near the shield. The explosive charges and the stemming packages with the cap in them can then be fed one at a time into plastic tubes individually on demand through the shield into the cab to be assembled by hand and placed into another tube leading to the explosive loading device.

Two alternative devices have been suggested for conveying the charge into the hole. One would be a plastic tube with water or air at low pressure employed to carry the assembled charge into the hole. A second device would be made up of half of a rigid split tube of plastic material which would be lined up with the hole at the collar with the assembled charge laid in it. The charge would then be pushed into the hole with a wooden or plastic tamping rod, provided with a groove to clear the lead wires. The length of wires required and the means of connecting them to the current source will be provided. The high capacity current source required for EBW's will be housed in the protective cab, with multiple outlets or connectors outside of the cab, shielded against air blast and flying fragments. The firing circuits must be arranged so that the leads from the caps be as short as possible to keep this type of firing circuit properly balanced, i.e., for high current EBW's.

Storage of Explosives

Explosives will be stored in much the same manner as they are in normal drilling and blasting, with the exception that it will be necessary to maintain a supply of cartridges, caps, and stemming units outby the shield during drilling and blasting operations. That is, it would be expected that in a tunneling operation, for example, that the main powder magazine would be outside of the portal of the tunnel, a smaller magazine located within the tunnel if the distance from the portal to the working face is great, and small stroage units just outby the shield where the parts of the explosive-cap-stemming unit can be kept in such a manner that they may be handled with automated equipment for moving, joining together, and for loading into the blast hole.

The first two types of storage magazines would require no special design, following the usual practice of keeping the caps and explosives separated.

The storage units outby the shield, however, constitute the first part of the automated loading and blasting system which must be considered. As an initial possible design, the blasting cap and the stemming may be fabricated together. This will serve at least two purposes. The first is that

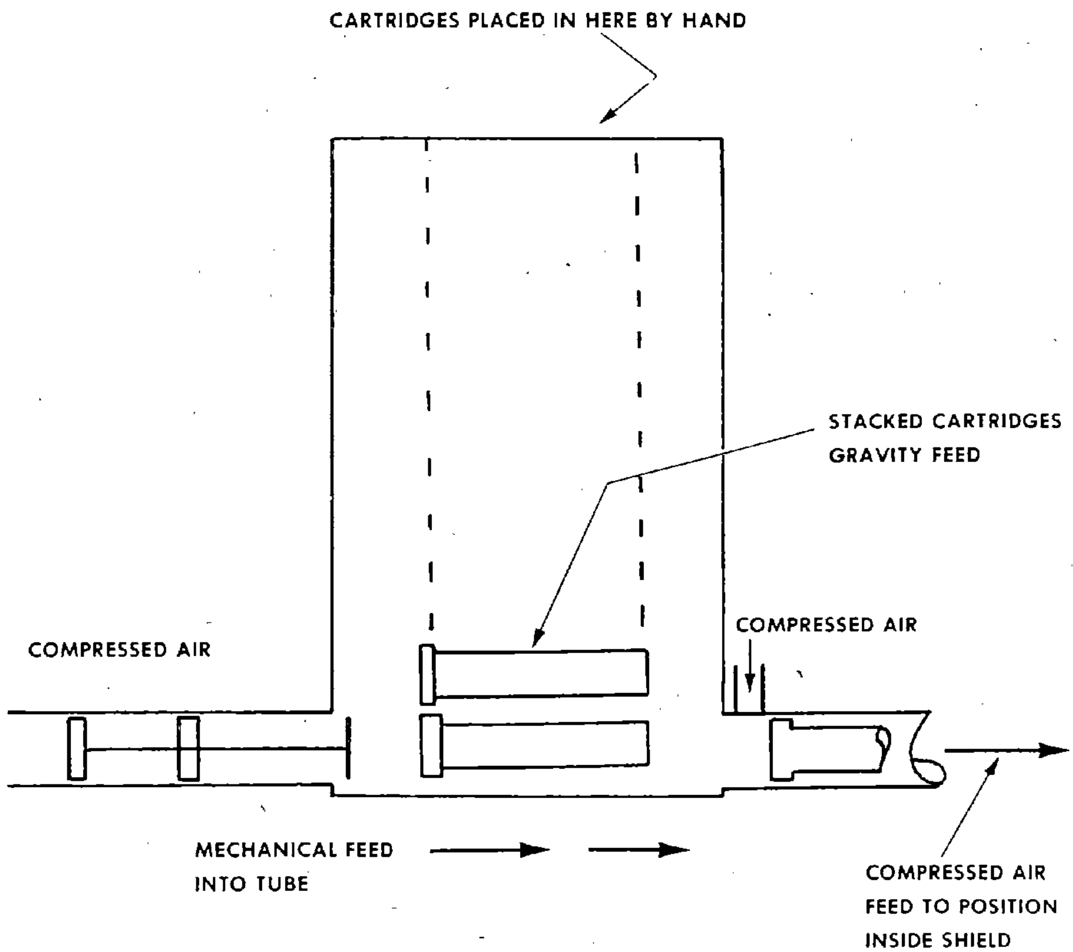


FIGURE 9-1 CONCEPT FOR FEEDING CARTRIDGES THRUOGH SHIELD, FOR BOTH EXPLOSIVE AND STEMMING.

the blasting cap will be relatively protected from impact, even though EBW's do not have any primary explosive in them. The second is that the cartridge of stemming serves as a means of conveying the stemming and the blasting cap from the storage units through the shield where they will be connected with the explosive cartridge. (Figure 9-1).

Explosive Charges

In the experimental work performed in this project, slurry explosives were employed for almost all of the shots. The reasons for this are that the explosive has the qualities needed for an automated system, i.e., it is safe to handle, its properties can be varied, it is cap-sensitive in small diameters, its fume properties are excellent, and its pliable so that it can be expanded to fill the bottom of the blast hole with moderate pressure.

For the initial concept of an automated explosive loading system, the type of charge construction is visualized as a cylindrical-shaped section of explosive enclosed in a moderately rigid plastic or waterproofed expandable casing (Figure 9-2). The end of the cartridge which will contact the stemming will be faced with a small rigid disc with a hole in the center just larger than the diameter of the cap. This will serve the following purposes: It will help keep the cartridge from expanding prematurely; it will act as a diaphragm or piston for compressed air to push against to convey the cartridge from containers through the shield and also into the blast hole, and will serve as a means of applying pressure to the cartridge to expand it in the bottom of the hole to give a high loading density.

Stemming-Cap Assembly

For the initial concept of the explosive loading system, it is planned to utilize a prefabricated assembly with the cap enclosed in a cartridge of stemming. This has several possible advantages, the first being that the stemming will serve as a means of conveying the cap through the system. The

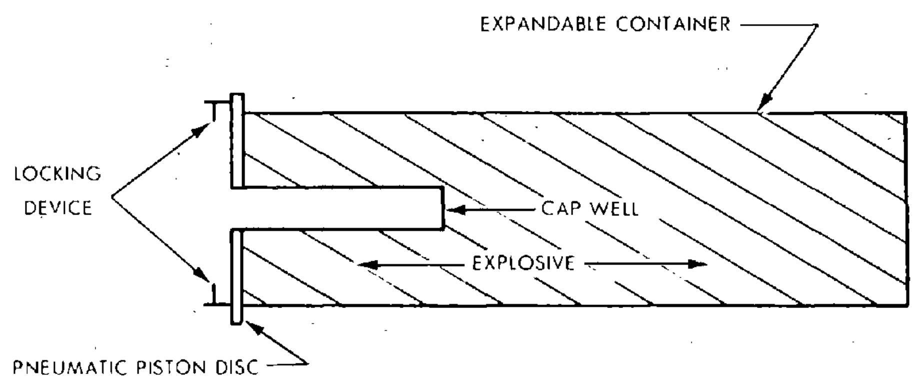


Figure 9-2 CONCEPT FOR EXPLOSIVE CHARGE.

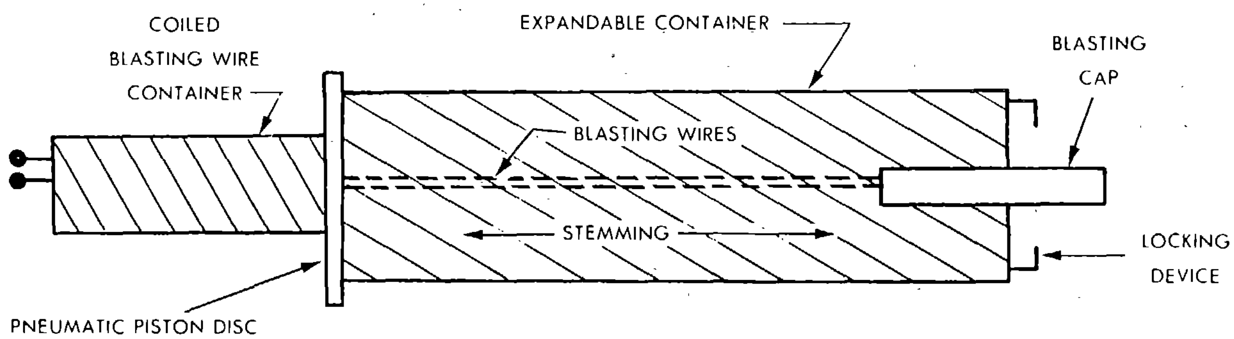


Figure 9-3. CONCEPT FOR STEMMING-CAP ASSEMBLY

casing for the stemming will be similar to that which holds the explosive, i.e., it will be made of semi-rigid plastic or paper which is folded or creased in such a manner that it will expand when pressure is applied to both ends of the cartridge. A rigid disc just larger in diameter than the cartridge will be fastened to the end opposite the cap to give rigidity to the cartridge during transport and to serve as a pressure surface when pressure is applied to expand the explosive and stemming cartridges at the bottom of the blast hole (Figure 9-3).

At the interface between the two cartridges a means will be provided to lock the two together with the cap inserted into the cap well in the explosive. In the initial phase of the development and testing program, it is anticipated that the two parts will be locked together by hand and placed into the loading tube leading to the blast hole. This will be further developed to permit automated assembly and transfer to the blast hole.

The blasting wires will be coiled within a small paper tube at the rear of the stemming. Inasmuch as the cartridge will be transported pneumatically, the wires in either a coiled or extended configuration will not interfer with the transport or compression in the bottom of the blast hole.

Charge Placement

The placement of the charge at the bottom of the hole will be one of the critical operations of the system. That is, the explosive must be expanded to give a loading density as near one as possible, and the stemming likewise expanded to furnish an optimum resistance to expansion of the explosive generated gases down the borehole.

One plan is to transport the charge-stemming assembly to the bottom of the hole through tubing, the latter section of which is rigid. This will be withdrawn to allow small dogs at the end of the tubing to drop or be forced

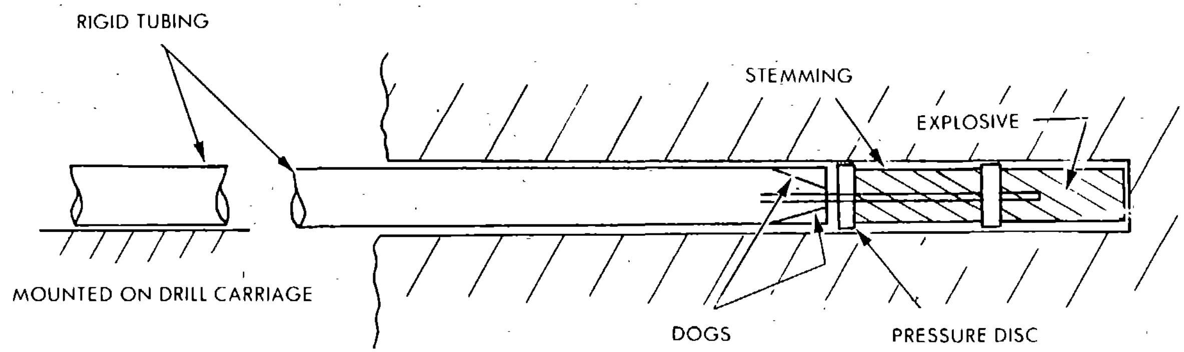


FIGURE 9-4 CONCEPT FOR LOADING AND COMPRESSING CHARGE AND STEMMING.

in place by small springs behind the rear disc on the charge assembly. The rigid portion of the tubing will have a device to lock onto the sides of the hole and will be forced forward either mechanically or pneumatically to compress the explosive and the stemming (Figure 9-4).

Careful alignment of the charge during all of the operation will be necessary. Also, the use of blasting caps which do not contain primary explosives will minimize the possibility of premature initiation.

Safety of Explosive System

As noted briefly above, the use of the proposed system is in conflict with current safety codes and rules for the employment of explosives underground. Aluminized small diameter slurries are safer than dynamites. In the higher strength aluminized slurries, the particle size of the aluminum is larger than that in those of lower strength, and is probably not explosible even when it is dispersed in air. The air-gap sensitivity is also low, which also minimizes the possibility of an accidental explosion of a cartridge. If the transporting system is properly designed, the storage chamber can be isolated from the transport conduits so that an explosion cannot be propagated back through these tubes. The bullet sensitivity is likewise very low.

The protective cab and the shield will be designed to resist the air blast even if all of the holes in a line bootleg. Misfires and hangfires can be handled safely because the operator will be able to observe the condition at the face from within the cab, which will expedite the process of taking care of any of the above occurrences.

As noted at the beginning of this chapter, the concept of having personnel remain at the working face while blasting takes place is a novel idea, and is contrary to the rules stating that personnel should be removed

from the blasting site to established distances. However, the results of the experimentation to date indicate that all of the blast effects experienced by operators in the protective cab or outby the main shield can be reduced to or below acceptable levels. The principles of the protection of human beings in hostile environments have been well established in the space program for astronauts, and these same principles can be applied to the environments which will be present with the small charge blasting system. If the system is properly designed and operated, the hazards would be less than those encountered in conventional drill and blast methods.

A somewhat parallel situation exists in an automobile, where hundreds of small scale explosions take place per minute within a few feet of the driver. The explosions are properly contained, and there is no concern for the safety of the people within the car.

CHAPTER 10

COST ANALYSIS

Introduction

The viability of an Automated Drill and Blast System (ADBS) will be determined by its productivity, safety, and cost effectiveness. In the evaluation of cost effectiveness, two requirements are indicated: An analysis of ADBS performance and technical feasibility, and a comparison with a proven tunneling method which can be used as a standard in order to show more effective operation at lower cost.

The ADBS tunneling concept was developed in order to design an excavation system of greater effectiveness than the Drill, Blast and Muck (DBM) System by applying the non-cyclic efficiency of continuous tunneling and the characteristics of small charge blasting. Since the ADBS method must be more than competitive with conventional DBM tunneling, it is logical that the latter be chosen as the standard.

Preliminary field research on the technical aspects of automated tunneling with small charges has provided some of the initial data for developing the economics of the ADBS (Clark & Rollins, 1976). The total direct cost is the sum of three expense categories: equipment (including supplies), labor, and tunnel support. A parallel breakdown is also applicable to the DBM method.

Projected cost savings of ADBS over DBM are as follows:

Direct Costs

Equipment:	Possible lower capital costs due to lower equipment costs and more rapid advance rates
Labor:	Less labor required due to nature of operation
Support:	Less support required because of reduction of wall fracture and overbreak

Indirect Costs

Reduction of overhead costs such as management and engineering salaries due to shorter period for project completion

Reduction of costs due to improved environment

The improvement in environmental conditions will accrue from changed conditions relative of DBM, both underground, and in some cases, on the surface. The blast shield will contain the gases from the explosives and may be more readily pumped from the working face by the ventilation fans. The containment of the air blast and the flying fragments will decrease the effects of blasting on rock structures, artificial support, equipment and personnel.

The reduction in ground vibration and air blast will make it possible to lower excavation costs in urban areas where vehicular tunnels are being driven. In the excavation of the Metro system in Washington, D.C., for example, blasting was severely restricted as to time of the day and nearness to surface structures. The use of small charge blasting will make it possible to carry out such blasting operations on a more continuous basis with obvious decrease in labor, equipment, and most other operational costs.

If the small charge method of blasting proves feasible for use in a longwall system of mining, some of the same advantages will also apply. That is, air blast will be kept to a lower level than is possible with conventional blasting, protected personnel can work continuously at the working face, and the operation may be made continuous without moving equipment and men from working place to working place while blasting is done.

The effects of small charge blasting on the walls of the excavation have been discussed in detail in earlier chapters, i.e., the reduction of overbreak and fracture of the surrounding rock can critically reduce support costs.

Because larger numbers of smaller diameter holes are required for the proposed small charge blasting system, the costs per foot of advance for both drilling and for blasting caps will be higher than for conventional blasting. Exploding wire (EBW) caps required for the ADBS in small quantities currently cost over $3.00 each. In large quantities, EBW's with the required accuracy of initiation can be produced for approximately the same price as electric blasting caps (EBC's).

Rate of Advance

The rate of advance is influenced by many factors such as tunnel configuration, equipment, manpower, water inflow, type and structure of rock, and method of rock breakage. However, the most influential factors are the rock type (Mayo, 1968) and method of rock breakage. For the purpose of this cost analysis, host rock is assumed to range from "intact" to"stratified or schistose" (Mayo, 1968) and to consist of a rock type such as granite or gneiss. Additionally, advance rates in stratified and jointed sandstone and limestone are considered.

DBM Systems. In tunneling practice in firm rock, small tunnels are driven full face, while large ones are driven "top heading" first, portal to portal, with steel ribs of the primary lining set in hitches on the side of the tunnel. The oepration cycle at the face consists of (1) drill, (2) blast, (3) muck, and (4) support (Table 10-1).

In conventional tunneling, blast holes are 4 to 12 ft deep. Typically, one hole is required for every 2 to 8 sq ft of face. For small tunnels, jack-leg drills are employed. Drill jumbos of various types are used to drive larger tunnels.

TABLE 10-1

OPERATIONAL CYCLE - DBM

Drill Cycle

Move in drill (jumbo), connect air & water lines
Drill blast holes

Blast Cycle

Disconnect air & water and move drill (jumbo) out
Load powder
Connect blasting leads
Move personnel to safe position
Ventilate (smoke time)

Muck Cycle

Hose down
Scale roof, clean fly rock
Move mucking machine into heading
Muck out round
Extend track as required
Move out mucking machine

Support Cycle

Move in materials & erection equipment
Install bolts as required
Erect set(s)

Where rail haulage is employed, changing of cars requires time and may delay the mucking cycle. Ventilation regulations usually require 100 cfm per man or 50 cfm per square foot of tunnel cross section. The waiting time for fumes to clear for safe breathing is usually about 20 minutes. For the proposed system, enough of the fumes must be removed to provide visibility only since the ADBS operator(s) will be enclosed in air conditioned cabs.

A study of the tunnel excavation operations at the Flathead Tunnel located in northwestern Montana (Skinner, 1974) for conventional drill and blast (Table 10-1) shows that the percentages of time per cycle were: drilling, 28 percent; mucking, 28 percent; support, 21 percent; loading, 13 percent; and miscellaneous, 10 percent. The rock encountered in the Flathead Tunnel ranged from "stratified" to "blocky and seamy" to "squeezing." However, over 90% was located in what may be considered as "moderately blocky and seamy rock." The average rate of advance was approximately 12.9 ft per shift or 38.6 ft per working day. The face area was approximately 600 sq ft (equivalent to a 27 ft - 8 in. diameter tunnel). This rate of advance is over 40% faster than predicted by Mayo (1968) under the best conditions. However, when total tunneling time including shutdowns is included, the average advance rate is 28.6 ft per day over the project life, which is within the range of rates described by Mayo.

The above analysis indicates that there are two time frames to be considered when determining advance rates. The first, as reported by Mayo, includes driving time and delay and shutdown time. This total time determines project duration.

The second time frame is the production (working) time which excludes holidays and other times when heading advance is stopped. This time will determine total labor cost for the project, neglecting salaries and overhead for security guards, maintenance people, and staff who work during shutdown time.

The total project time is a function of both delay time and rate of advance. The delay time is a function of shutdown time which, for this analysis, consists of holidays and weekends. Production time is composed solely of the time during which the heading crew is working.

The only available comprehensive source of tunnel advance rates is Mayo's (1968) report, which considers rates in granitic type rocks only. However, the production advance in massive limestone and sandstone can be determined by modifying the operational cycle (Table 10-2). The rock type will not significantly affect the blast or muck cycle. In the two sedimentary rock types, support would probably consist of shotcrete and rock bolts. This analysis assumes a 50% reduction in support erection time since steel sets would not be required.

The drilling rate affects the production advance rate more than any of the other operation cycle elements. All drill rates and equipment costs reported below assume that hydraulic drills are used (Chapter 4) since such technology is expected to become the industry standard within the next five to ten years. Drill penetration rates for hydraulic drills are significantly higher than for pneumatic drills (Table 10-2):

TABLE 10-2

DRILL PENETRATION RATES

Rock Type	Pneumatic Drill (ft/min) (Schmidt, 1972)	Hydraulic Drill (ft/min) (Bullock, 1974)
Granitic	2.00	3.00
Sandstone	2.25	4.50+
Limestone	2.50	4.00+

Drilling time includes both penetration time plus the time required to retract the bit from the drilled hole, realign the drill, and collar the next hole. DBM drilling times are based on the assumption of an 8 ft deep round and 30 seconds per hole for retracting, aligning, and collaring (Table 10-3). The determination of the number of holes per face is described below under Bit and Explosive Cost. The number of drills is based on typical current practice for a given face area which is determined by the desired rate of advance, manpower requirements, and available capital.

Thus, drill cycle times for pneumatic drills may roughly be reduced as follows for hydraulic drills:

granitic	30%
sandstone	44%
limestone	32%

TABLE 10-3

DRILL CYCLE TIME - 8 FT ROUND

Rock Type	Tunnel Diam. (ft)	No. of Drills	No. of Holes	Drill Cycle Time (hrs) Pneumatic	Hydraulic
Granitic	12	2	28	1.05	0.74
	16	2	35	1.35	0.95
	20	2	45	1.73	1.21
	24	3	58	1.50	1.06
Sandstone	12	2	28	0.95	0.53
	16	2	35	1.22	0.68
	20	2	45	1.55	0.87
	24	3	58	1.35	0.76
Limestone	12	2	28	0.86	0.58
	16	2	35	1.11	0.75
	20	2	45	1.42	0.96
	24	3	58	1.23	0.83

Thus, the drill cycle in granitic rock will occupy approximately 28% of the production cycle time, and because of the faster drill penetration rates (Table 10-3) drill cycle time is estimated to be 20% and 22% of the production cycle time for sandstone and limestone, respectively. Mayo's (1968) analysis indicates that the total rate of advance is a function of the tunnel diameter:

$$\text{TOTAL ADVANCE RATE (ft/day)} = M \times \text{TUNNEL DIAMETER (ft)} + B \qquad (10.1)$$

where

M is constant for the two granitic rock types ("intact" and "stratified or schistose")

B varies according to rock conditions (intact, fractured, etc.)

Mayo's (1968) value of M(M = -0.90) and the rate of advance achieved in the Flathead Tunnel (as indicated above) yield a predicted rate of advance in moderately blocky and seamy rock for pneumatic drills:

$$\text{ADVANCE RATE (ft/day)} = -0.90 \times \text{TUNNEL DIAMETER (ft)} + 64 \qquad (10.2)$$

When rock type, hydraulic drill penetration rates, and support erection times are considered, the rate of advance is predicted by equations These equations assume three eight hour shifts per day and were derived by reducing cycle times as described above.

$$\begin{aligned}
&\text{INTACT ROCK ADVANCE RATE (ft/day)} = \\
&\quad -0.90 \times \text{TUNNEL DIAMETER (ft)} + 63 \\
&\text{STRATIFIED OR SHISTOSE ROCK ADVANCE RATE (ft/day)} = \\
&\quad -1.60 \times \text{TUNNEL DIAMETER (ft)} + 74 \\
&\text{LIMESTONE ADVANCE RATE (ft/day)} = \\
&\quad -1.10 \times \text{TUNNEL DIAMETER (ft)} + 77 \\
&\text{SANDSTONE ADVANCE RATE (ft/day)} = \\
&\quad -1.13 \times \text{TUNNEL DIAMETER (ft)} + 79
\end{aligned} \qquad (10.3)$$

Determination of the overall rate of advance must include shutdown time. When 10 holidays are included with weekends, a total of 114 days of shutdown are obtained per year. Thus, a five day work week permits a production advance availability of 69%, which gives the predicted overall advance rates as:

$$\begin{aligned}
&\text{INTACT ROCK ADVANCE RATE (ft/day)} = \\
&\quad -0.62 \times \text{TUNNEL DIAMETER (ft)} + 43 \\
&\text{STRATIFIED OR SCHISTOSE ROCK ADVANCE RATE (ft/day)} = \\
&\quad -1.10 \times \text{TUNNEL DIAMETER (ft)} + 51
\end{aligned} \qquad (10.4)$$

LIMESTONE ADVANCE RATE (ft/day) =
-0.76 x TUNNEL DIAMETER (ft) + 53 (10.4)

SANDSTONE ADVANCE RATE (ft/day) =
-0.78 x TUNNEL DIAMETER (ft) + 55

Automated Drill and Blast System. The unit operations of the proposed automated system are essentially the same as for conventional drilling (Table 10-4).However, the drilling and mucking can proceed simultaneously, and ventilation can be performed while mucking is taking palce. Delays for bootlegs and misfires are not considered. Inasmuch as drilling time for DBM includes the moving of the jumbo to the face and out after drilling, and mucking time includes moving the mucking machine, there will be an appreciable saving of time for the ADBS which only moves a short distance ahead after round has been fired. Likewise, the support crews can work continuously outby the tunnel shield.

TABLE 10-4

CYCLE TIME DISTRIBUTION - ADBS

	DBM	ADBS	
	% of Cycle Time	Est. Time Saving %	
		min.	max.
Drilling time	28	0	0
Explosive loading time	13	2	7
Ventilation	5	1	6
Mucking	28	10	14
Support installation	21	2	6
Rail laying & lost time	5	5	10
	100%	20%	43%

The estimated time saving by the ADBS (Table10.4) varies from approximately 20% to 43% which means that the ADBS advance rate is faster than the

DBM advance rate by a factor of 1.25 to 1.75. Simultaneous mucking and drilling without the necessity of moving equipment from the face results in a lower cost per foot of advance for factors other than those which are constant per foot of tunnel. Thus, labor and overhead costs per foot are reduced.

An advance rate of 5 ft per hour was estimated in a preliminary design report for the Hercules system of automated blasting (Hercules, 1976), which should attain advance rates similar to the ADBS. However, for this analysis, rates are conservatively assumed to be 1.25 times the DBM rate of advance obtained in a given rock type (equations 10.5). Thus, the ADBS direct rate of advance can be predicted by:

INTACT ROCK ADVANCE RATE (ft/day) =
-1.13 x TUNNEL DIAMETER (ft) + 79

STRATIFIED OR SCHISTOSE ROCK ADVANCE RATE (ft/day) =
-2.00 x TUNNEL DIAMETER (ft) + 93 (10.5)

LIMESTONE ADVANCE RATE (ft/day) =
-1.38 x TUNNEL DIAMETER (ft) + 96

SANDSTONE ADVANCE RATE (ft/day) =
-1.41 x TUNNEL DIAMETER (ft) + 99

Similarly, the overall rates of advance can be predicted for the ADBS from equations 10.6, again assuming a 69% availability:

INTACT ROCK ADVANCE RATE (ft/day) =
-0.78 x TUNNEL DIAMETER (ft) + 54

STRATIFIED OR SCHISTOSE ROCK ADVANCE RATE (ft/day) =
-1.38 x TUNNEL DIAMETER (ft) + 64 (10.6)

LIMESTONE ADVANCE RATE (ft/day) =
-0.95 x TUNNEL DIAMETER (ft) + 66

SANDSTONE ADVANCE RATE (ft/day) =
-0.98 x TUNNEL DIAMETER (ft) + 69

Summary

TABLE 10-5

PRODUCTION (OVERALL) RATE OF ADVANCE (FT/DAY) FOR DIFFERENT TUNNEL DIAMETERS AND ROCK CLASSIFICATIONS USING THE DBM METHOD OF ADVANCE

Tunnel Diameter (ft)	Rock Classification Intact	Stratified	Limestone	Sandstone
12	52(36)	55(38)	64(44)	65(46)
16	48(33)	48(33)	59(41)	61(43)
20	45(31)	42(29)	55(38)	56(39)
24	41(28)	36(25)	51(35)	52(36)

TABLE 10-6

PRODUCTION (OVERALL) RATE OF ADVANCE (FT/DAY) FOR DIFFERENT TUNNEL DIAMETERS AND ROCK CLASSIFICATIONS USING THE ADBS METHOD OF ADVANCE

Tunnel Diameter (ft)	Rock Classification Intact	Stratified	Limestone	Sandstone
12	65(45)	69(47)	79(55)	82(57)
16	61(42)	61(42)	74(51)	76(53)
20	56(38)	53(36)	68(47)	71(49)
24	52(35)	43(31)	63(43)	65(45)

Bit and Explosives Cost

Differences in bit and explosive costs between DBM and ADBS result from differences in the unit operations of drilling and blasting. This is due to drill hole spacing and depth, and the larger number of holes and caps required for the ADBS.

DBM. For this analysis, a bit diameter of 1-1/2-in. was assumed. The number of holes per standard round can be determined from the following equation (Gustafsson, 1973), where "diameter" is the tunnel diameter:

$$\text{NO. OF BLAST HOLES} = 0.094\ (\text{DIAMETER (ft)})^2 - 0.877\ (\text{DIAMETER (ft)}) + 25 \qquad (10.7)$$

For the drilling rates given above and a 500 ft bit life (Cummins & Given, 1973), the cost for drill bits can be determined from the following equation based on a current cost of \$10 per bit (Ingersoll-Rand, 1977):

$$\text{BIT COST (\$/ft)} = (\text{NO. OF BLAST HOLES} \times 10)/500 \tag{10.8}$$

The above analysis assumes a throw-away bit.

The explosive cost depends on the number of blasting caps and weight of explosive used. The number of caps per round is equal to the number of blast holes calculated above, and the cost of explosives per foot may be determined from the following equation which conservatively assumes 0.075 pounds of dynamite per cubic foot of rock broken (Langefors & Kihlstrom, 1973) where "diameter" is the tunnel diameter:

$$\begin{aligned} &\text{EXPLOSIVE COST (\$/ft)} = \\ &\quad \text{CAP COST (\$/ea)} \times \text{NO. OF BLAST HOLES/ROUND LENGTH (ft)} + \\ &\quad \text{DYNAMITE COST (\$/lb)} \times (\text{DIAMETER (ft)})^2 \times 0.059 \end{aligned} \tag{10.9}$$

<u>ADBS.</u> In the Flathead Tunnel (Skinner, 1974), experiments were carried out with smooth-wall blasting to obtain a more uniform arch, the procedure being somewaht similar to that of firing perimeter holes in the ADBS.

In the first set of experiments, 7 ft holes were drilled above the spring line with 30 in. spacing. These were loaded with Hercules 60 percent trim powder with a loading density of 1/4 pound per foot and fired with No. 12 delay caps. The resulting rock breakage was largely along the joints, the rock being characterized as good (moderately to highly fractured but tightly jointed).

The hole spacing was reduced to 24 in. with the loading density maintained at 1/4 pound per foot of hole. This resulted in improved breakage, i.e., remnants of the drill holes were observed, no overbreak occurred, little scaling was required, and minimum rock bolting was used.

The smooth-wall blasting was discontinued because:

1. The time required for additional drill holes disrupted the normal drilling cycle.

2. The drilling required of the three top machines on the jumbo was increased.

3. Special powder was required.

4. Improvements due to smooth-wall blasting in closely jointed rock could not be determined accurately.

5. In highly jointed rock normal support would be required.

6. Increased stability resulting from smooth blasting could not be quantitatively determined.

In support of the use of smooth-wall blasting, as related to the ADBS, the following should be observed:

1. An additional drill on the top deck of the jumbo would have obviated the first two objections.

2. If smooth-wall blasting had been planned in advance, the handling of special powder for the top perimeter holes would not present a problem.

3. While No. 12 delay caps were employed to fire the top perimeter holes, the most effective results are obtained when the firing of caps does not vary more than 200 microseconds.

4. In jointed rock, the drill holes should be placed "in the solid" between joints as much as possible.

5. As compared with the ADBS, shorter holes in the latter permits better placement of the charge.

6. With closer hole spacing, the smooth-wall blasting was reported as giving beneficial results.

7. In the ADBS by comparison, smaller charges with closer spacing are required and give little overbreak in jointed rock (Hanna, 1978).

A drill diameter of either 3/4 or 1-in. has been suggested for the ADBS scheme. For this analysis, a diameter of 1-in. is assumed (Clark & Rollins, 1976). Bit life is 500 ft which is the same as for DBM. Again, assuming $10 per bit, the bit cost may be estimated by:

BIT COST ($/ft) = (NO. OF BLAST HOLES x $10)/500 (10.10)

As with the DBM system, ADBS blasting cost is a function of the number of caps and pounds of explosive used. The blasting cost may be determined from the following, conservatively assuming a powder factor of 0.08 lbs/ft^3 of intact rock (Clark & Rollins, 1976):

EXPLOSIVE COST ($/ft) =
CAP COST ($/ea) x NO. OF BLAST HOLES/ROUND LENGTH (ft) +
SLURRY COST ($/lb) x (DIAMETER (ft))^2 x 0.063 (10.11)

Explosive costs as of March 1977 are (Emerick & Hill, 1977):

Standard Electric Blasting Caps	$0.70/ea
Straight Dynamite (1-1/4 x 8)	.45/lb
Slurry (GEL) (1-1/4 x 15)	.55/lb

TABLE 10-7

BLASTING COSTS - DBM (8 ft round)

Tunnel Diameter (ft)	No. of Blast Holes	Bit Cost ($/ft)	Cap Cost ($/ft)	Explosive Cost ($/ft)	Total Cost ($/ft)
12	28	.56	2.45	3.82	6.83
16	35	.70	3.06	6.80	10.56
20	45	.90	3.94	10.62	15.46
24	58	1.16	5.08	15.29	21.53

TABLE 10-8

BLASTING COSTS - ADBS

Round Depth (ft)	Tunnel Diameter (ft)	No. of Blast Holes	Bit Cost ($/ft)	Cap* Cost ($/ft)	Explosive Cost ($/ft)	Total Cost ($/ft)	Total Cost-DBM ($/ft)
1.5	12	73	1.46	34.07	4.99	40.52	6.83
	16	121	2.42	56.47	8.87	67.76	10.56
	20	171	3.42	79.80	13.86	97.08	15.46
	24	236	4.72	110.13	19.96	134.81	21.53
2.0	12	64	1.28	22.46	4.99	27.73	6.83
	16	105	2.10	36.75	18.87	47.72	10.56
	20	148	2.96	51.86	13.86	68.62	15.46
	24	204	4.08	71.40	19.96	95.44	21.53
2.5	12	57	1.14	15.96	4.99	22.09	6.83
	16	94	1.88	26.32	8.87	37.07	10.56
	20	132	2.64	36.96	13.86	53.46	15.46
	24	182	3.64	50.96	19.96	74.56	21.53
3.0	12	52	1.04	12.13	4.99	18.16	6.83
	16	86	1.72	20.07	18.87	30.66	10.56
	20	121	2.42	28.23	13.86	44.51	15.46
	24	166	3.32	38.73	19.96	62.01	21.53

*Current cost of high precision EBW's (± 10 μsec accuracy) is $3.00. Cost of required EBW's (± 200 μsec) accuracy is approximately the same as standard caps.

The total explosive cost for small charge rounds is much larger than that for DBM (Figure 10-1), but decreases rapidly with the depth of round. The higher cost is due to the cost of blasting caps.

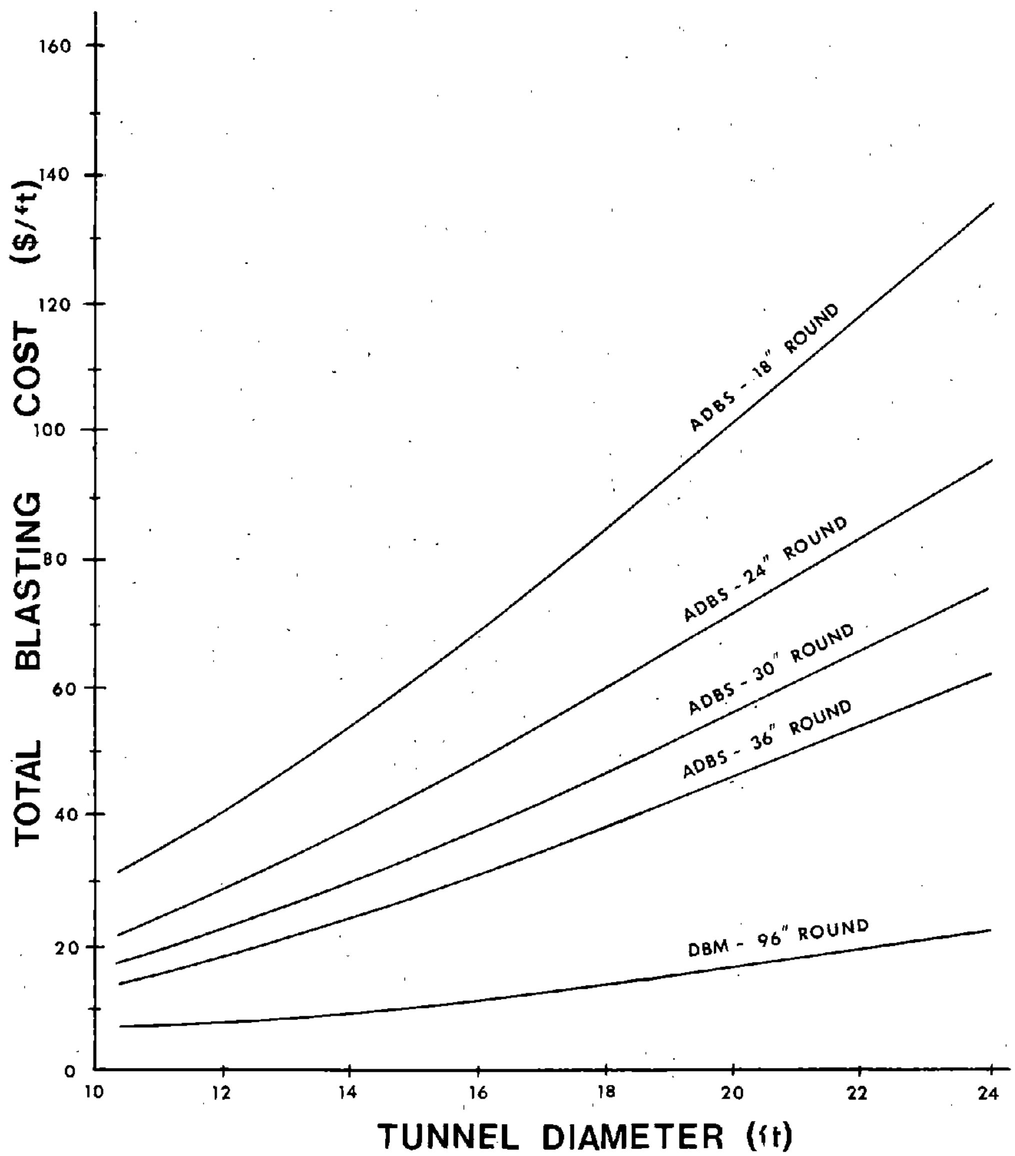
Capital Cost

The capital costs include those of mining equipment used at and near the tunnel face, and the supporting equipment and materials. The size of the equipment and the quantity of required materials is a function of the tunnel diameter, the method of excavation, and the rate of tunnel advance.

DBM. Equipment cost for the DBM method of advance is determined from the equation given below (Mayo, 1968). The curves in this reference were revised to reflect equipment only (labor was excluded) and were updated to

FIGURE 10-1

DBM AND ADBS BIT AND EXPLOSIVE COST



January 1977 dollars using the Engineering News Record index. The cost per foot is obtained by dividing the cost obtained from the equation given below by the appropriate advance rate.

$$\text{EQUIPMENT COST (\$/day)} = 515 \times \text{TUNNEL DIAMETER (ft)} + 7400 \quad (10.12)$$

Automated Drill and Blast System

ADBS. Equipment cost for the ADBS may be determined from the following relationship (Hercules, 1976), which was applied to the Hercules method of continuous advance for a 15 ft diameter tunnel:

Capital Cost - DBM	$1,110,000
Capital Cost - ADBS	900,000

Using the ratio of the above, ADBS equipment cost could be estimated from:

$$\text{EQUIPMENT COST (\$/day)} = 420 \times \text{TUNNEL DIAMETER (ft)} + 6000 \quad (10.13)$$

An alternate method of determining equipment costs is to price equipment on a per item basis. The following analysis is based on a 13 ft square tunnel (equivalent to a 15 ft diameter tunnel). Standard practice using DBM in a tunnel of this size is to use a two boom jumbo. Rail haulage and mucking is assumed and hydraulic drills are utilized. Costs are summarized as follows (Atlas-Copco, 1977; Elliot, 1977; Ingersoll-Rand, 1977; Tyson, 1977) assuming stratified and schistose rock:

1 - Drill jumbo complete but w/out drills, L.S.	$ 33,000
2 - Hydraulic drill w/boom and hydraulic pump, L.S.	146,000
10 - 8 cu yd muck cars (8 ft round) @ $5500/ea	55,000
3 - 15 ton locomotives @ $70,000/ea	210,000
2 - Eimco 75 mucker [$95,000/ea	190,000
1 - Support (steel & concrete) jumbo, L.S.	45,000
1 - Msc. equipment including compressors, building, fans, pumps, etc., L.S.	250,000
TOTAL DBM EQUIPMENT COST	$929,000

In stratified and schistose rock, the ADBS production advance rate is estimated to be 73 ft per day. If 2.5 square ft of face per drill hole is assumed, approximately 4900 lineal feet of drilling is required to advance 73 ft. With a 36 in. deep round, 1630 total holes are required. If 10 seconds are required to align the drill and collar the hole, total drill time delay is 4.5 hours. A standard shot will consist of six holes or 270 individual shots for 73 ft, or 11 per round. A one minute delay per shot yields a total shot time delay of 4.5 hours. Thus, 15 hours are left for drilling or 2700 ft per drill per day in granitic rock. Two hydraulic drills are, therefore, required.

Muck will be produced at the semi-continuous rate of 32 cu yd per hour with 73 ft of advance per day and a swell of 50%, as was also applied to DBM. The 50% swell is assumed to include overbreak volume. One train is being loaded each hour while a second train is being dumped, which requires four muck cars per train or a total of 8 muck cars.

Costs for ADBS equipment are as follows (Ibid; Alpine, 1977):

1 - ADBS drill platform and mucking system, L.S.	$200,000
2 - Hydraulic drill w/boom & hydraulic pump, L.S.	146,000
1 - Explosive loading system, L.S.	25,000
1 - Control cab w/controls & interlocks, complete, L.S.	20,000
1 - Blast shield w/hydraulics & muck thru-flow, L.S.	15,000
8 - 8 cu yd muck cars @ $5500/ea	44,000
3 - 15 ton locomotives @ $70,000/ea	210,000
1 - Support (steel & concrete) jumbo, L.S.	45,000
1 - Msc. equipment, L. S.	250,000
TOTAL ADBS EQUIPMENT COST	$955,000

The cost for the ADBS components, which includes mucking, drills, cab, shield, and controls is $406,000 or approximately 40% of total equipment cost. The total system cost would be lower if the ADBS were priced as a unit rather than adding component costs for various conventional systems.

However, in order to determine a conservative equipment cost, the ratio of the above equipment costs is applied to the DBM equipment cost equation (Equation 10.14) to obtain:

EQUIPMENT COST ($/day) = 529 x TUNNEL DIAMETER (ft) + 7600 (10.14)

Labor Cost

Labor consists of the crew involved in mining, personnel supporting the face crew, support crew, and the project administration staff.

DBM System. Labor cost for the DBM method is based on the estimated number of men necessary to drive a tunnel of a given size (Mayo, 1968). For example, a large tunnel requires more miners to obtain the desired advance rate than a smaller tunnel. Based on present wage rates and personnel requirements, the apparent cost of labor is obtained from the formula given below. This equation was derived from the data in the above references publication and current wage rates (Kiewit, 1977):

APPARENT LABOR COST ($/day) = 365 x TUNNEL DIAMETER (ft) + 2740 (10.15)

The labor cost obtained from the above formula must be adjusted to include the indirect labor costs such as fringe benefits and insurance. These costs typically amount to approximately 30% of wages (Tyson, 1977). Thus, the true labor cost becomes:

TRUE LABOR COST ($/day) =
475 x TUNNEL DIAMETER (ft) + 3560 (10.16)

ADBS. It is estimated that DBM manpower (not labor cost) can be reduced by 30% if the ADBS is utilized (Hercules, 1976). A more rational labor cost reduction can be quantified by eliminating from the DBM personnel (Mayo, 1968) those positions which are not required for the ADBS (Table 10-9). Typical positions include mucking machine operators, miners, and chuck tenders. When this is done, the reduction in labor cost is approximately 20%. Thus, the true cost for ADBS labor (including insurance and fringe benefits) may similarly be determined from the following, assuming a 20% labor cost reduction:

$$\text{TRUE LABOR COST (\$/day)} = 380 \times \text{TUNNEL DIAMETER (ft)} + 2850 \tag{10.17}$$

As with the DBM system, the above costs are given for January 1977 dollars. Labor cost per foot is determined in dollars per foot by dividing labor cost by advance rate per day.

TABLE 10-9

WAGE RATES AND PERSONNEL REQUIREMENTS
FOR A 12 FT DIAMETER TUNNEL

		DBM		ADBS	
Position	Hourly Wage	No. of Men	Cost/ Shift	No. of Men	Cost/ Shift
Shifter	$ 8.50	1	$ 68.00	1	$ 58.00
Nipper	5.00	1	40.00	1	40.00
Muck Machine Operator	7.75	1	62.00		
Oiler	7.10	1	56.80		
Motorman	8.25	3	198.00	3	198.00
Brakeman	7.10	3	170.40	3	170.40
Electrician	8.25	2	132.00	2	132.00
Compressor Man	6.85	1	54.80	1	54.80
Warehouse Man	6.85	1	54.80	1	54.80
Mechanic	8.25	2	132.00	2	132.00
Powder Man	7.75	1	62.00		
Laborer	7.10	4	227.20	4	227.20
Truck Driver	10.20	1	81.60	1	81.60
Walker	10.60	1	84.80	1	84.80
Office Help	6.85	4	219.20	4	219.20
Superintendent	14.50	1	116.00	1	116.00
Miner	8.25	5	330.00	4	264.00
Chucktender	7.10	5	284.00		
	TOTAL COST/SHIFT		$2,373.60		$1,842.80

Tunnel Support Cost

The need for some tunnel support is almost universal. Whether the tunnel is for water or wastewater conduits, or transportation, support is usually required for a portion of the tunnel. Tunnels or drifts driven for mining purposes often do not require permanent support but usually need some type for acceptable stability.

In the analysis that follows, the most costly case is assumed, that is, that maximum tunnel support is required. Current practice in rock requiring heavy support favors installing both steel sets (temporary support) and concrete lining (permanent support). Each of these two types of support is designed to carry the total design load with no consideration for the support provided by the other system and are applied to stratified and schistose rock as described below.

As indicated previously, limestone and sandstone may require minimal support consisting of shotcrete and rock bolts. Rock bolts are assumed to be required for only 25% of the tunnel length. This amount of minimal support system has also been applied to intact rock.

It is necessary also to analyze the effect of overbreak on support costs. Overbreak affects tunnel cost in three respects: increased steel set cost due to increased rock load caused by uncontrolled blasting, increased concrete yardage to fill the overbreak volume, and increased muck removal.

DBM. Steel support design in the U.S. is based on the rock classification and attendent load configuration developed by Terzaghi (Proctor & White, 1968). In this analysis, the rock conditions chosen ranged from "intact" to "stratified or schistose" and a density of 160 pounds per cu ft is used. Terzaghi's method determines a rock column height whose weight is applied to the set. Support requirements are predicated on steel sets on four foot centers.

Based upon these criteria, the set can be designed using design charts and a steel cost of \$0.50 per pound (Commercial Shearing and Stamping, 1977), which gives the following:

STEEL SET COST IN STRATIFIED AND SCHISTOSE ROCK (\$/ft) =
7.88 x TUNNEL DIAMETER (ft) - \$70 (10.18)

Concrete costs are divided into two categories: support concrete and overbreak concrete. Support concrete requirements are determined using Terzaghi's loading method and concrete with a 3500 psi compressive strength at 28 days. Grouting behind the concrete lining is included in the concrete cost and is estimated at \$200 per cu yd in place (Tyson, 1977; Appendix B).

CONCRETE COST IN STRATIFIED AND SCHISTOSE ROCK =
12.5 x TUNNEL DIAMETER (ft) (10.19)

The volume of overbreak is determined from a relationship which assumes overbreak at approximately 12% (Abel, 1975):

OVERBREAK VOLUME (yd^3/ft) =
0.072 x TUNNEL DIAMETER (ft) = 0.45 (10.20)

Cost to transport overbreak muck and replace this volume with concrete is calculated to be \$75 per cu yd of overbreak (Tyson, 1977). For intact rock, support concrete is not required. However, a 3 in. thick shotcrete lining is assumed to provide for the safety of the workmen. The equation given below utilizes an in place shotcrete cost of \$50 per cu yd based on premixed aggregate, manual loading, and wet-mix application (Conspray, 1977). Rock bolts are mechanically anchored, 6 ft in length, cost \$2.20 each including the shell (Union, 1977), and are installed on 4 ft centers (Appendix B):

SHOTCRETE AND ROCK BOLT COST (\$/ft) =
0.84 x TUNNEL DIAMETER (ft) + 0.18 (10.21)

For intact rock, no concrete lining is installed and the cost for mucking and transporting the overbreak volume is not included. Support requirements for limestone and sandstone are assumed to be the same as those for intact rock.

<u>ADBS</u>. The use of smooth blasting has proven effective since its introduction in the early 1960's because of the reduction in overbreak and the consequent reduction in the volume of concrete. Hence, small charge blasting results in significant cost savings as described above.

Smooth blasting in the Flathead Tunnel (Skinner, 1974) and small charge blasting (Hanna, 1978) both resulted in reduced overbreak. In these cases, lack of overbreak was evidenced by the visual remnants of the perimeter blast holes after blasting. In the small charge tests, the ribs and back did not require barring down after each round because of the sound rock conditions. Smooth blasting has also been effectively utilized in large, underground chambers to reduce overbreak and rock fracture around the perimeter of openings (Gagne, 1973; Hansen, 1968).

The reduction in overbreak can easily be quantified. For example, if proper drill alignment is assumed and the remnants of perimeter drill holes are observed, the reduction in overbreak may be estimated to approach 100%. The reduction in set load due to reduced overbreak is not currently quantified (Engineers, 1977; Reclamation, 1977; Skinner, 1974). However, such a reduction is generally recognized (Transportation and Communications Minestry of Ontario, 1976):

> "With standard drill and blast methods the thickness of the loosened zone has been observed to vary from about 2 feet in good quality rock to about 10 feet in low quality rock. This loosened zone will exert increased loads on the support system. Improved blasting techniques, such as smooth wall blasting, will reduce the thickness of the loosened zone, but field data are not available to evaluate this reduction quantitatively."

Thus, steel and concrete support cost for the ADBS can be conservatively determined (assuming no load reduction) from equations 10.18 and 10.19. A 50% decrease in overbreak volume is assumed for smooth blasting:

$$\text{OVERBREAK VOLUME } (yd^3/ft) = 0.036 \times \text{TUNNEL DIAMETER (ft)} - 0.23 \qquad (10.22)$$

As with the DBM method of excavation, overbreak muck removal and concrete replacement costs can be calculated at $75 per cu yd of overbreak.

The support cost in intact rock, limestone, and sandstone is predicted based upon the assumptions indicated above for shotcrete and rock bolts. Thus, equation 10.21 is also applicable to excavation by means of the ADBS.

TABLE 10-10

DBM SUPPORT COST ($/ft)

Rock Type	Tunnel Diam. (ft)	Support Cost(s) Steel	Concrete	Overbreak	Shotcrete	Total Cost
Intact	12				10	10
	16				14	14
	20				17	17
	24				20	20
Stratified & Schistose	12	25	150	31		206
	16	56	200	53		309
	20	88	250	74		412
	24	119	300	96		515
Limestone	12				10	10
	16				14	14
	20				17	17
	24				20	20
Sandstone	12				10	10
	16				14	14
	20				17	17
	24				20	20

TABLE 10-11

ADBS SUPPORT COST ($/ft)

Rock Type	Tunnel Diam. (ft)	Support Cost(s) Steel	Concrete	Overbreak	Shotcrete	Total Cost
Intact	12				10	10
	16				14	14
	20				17	17
	24				20	20
Stratified & Schistose	12	25	150	15		190
	16	56	200	26		282
	20	88	250	37		375
	24	119	300	48		467
Limestone	12				10	10
	16				14	14
	20				17	17
	24				20	20
Sandstone	12				10	10
	16				14	14
	20				17	17
	24				20	20

Discussion

The estimated costs for DBM and ADBS tunneling were determined considering three direct cost caregories: Equipment cost, labor cost, and support cost. Bit and explosive costs are higher for the ADBS (Tables 10-12 & 10-13). Equipment cost is obtained by applying equations 10.12 and 10.14 for DBM and the ADBS, respectively. The cost per foot is determined by dividing the equipment cost per day by the overall rate of advance in the appropriate type of rock. Equipment depreciation was not considered since the slightly higher ADBS equipment cost (3% higher) is insignificant and is offset by the more rapid overall advance rate. Total labor cost per foot is obtained by dividing the labor cost per day (Equation 10.14 & 10.15) by the production advance rate (Tables 10-12 & 10-13).

The sum of these four cost categories yields the total cost per foot (Tables 10-12 & 10-13). A 36 in. deep ADBS round was used for comparison.

TABLE 10-12

TOTAL DBM ADVANCE COSTS

Rock Type	Tunnel Diam. (ft)	Bits & Explo. ($/ft)	Equip. ($/ft)	Labor ($/ft)	Support ($/ft)	Total ($/ft)
Intact	12	7	377	178	10	572
	16	11	474	233	14	732
	20	15	571	290	17	893
	24	22	706	365	20	1113
Stratified & Schistose	12	7	357	168	206	783
	16	11	474	233	309	1027
	20	15	610	311	412	1348
	24	22	790	416	515	1743
Limestone	12	7	309	145	10	471
	16	11	381	189	14	595
	20	15	466	237	17	735
	24	22	565	293	20	900
Sandstone	12	7	295	142	10	454
	16	11	364	183	14	572
	20	15	454	233	17	719
	24	22	549	288	20	879

TABLE 10-13

TOTAL ADBS ADVANCE COSTS
(36 in. round)

Rock Type	Tunnel Diam. (ft)	Bit & Explo. ($/ft)	Equip. ($/ft)	Labor ($/ft)	Support ($/ft)	Total ($/ft)
Intact	12	18	310	114	10	452
	16	31	382	146	14	573
	20	45	478	187	17	727
	24	62	580	230	20	892
Stratified & Schistose	12	18	297	107	190	612
	16	31	382	146	282	841
	20	45	505	197	375	1122
	24	62	655	266	467	1450
Limestone	12	18	254	94	10	376
	16	31	315	121	14	481
	20	45	387	154	17	603
	24	62	472	190	20	744
Sandstone	12	18	245	90	10	363
	16	31	303	118	14	466
	20	45	371	148	17	580
	24	62	451	184	20	717

The results of the calculations (Tables 10-12 & 10-13) reveal several significant factors concerning tunneling in general and DBM vs ADBS tunneling in particular:

1. Tunneling in weak rock, as represented above by stratified and schistose rock, requires a significant expenditure for support. For both DBM and ADBS tunneling, support costs ranged from 28% to 32% of the total advance cost.

2. After support cost, equipment is the most costly portion of tunneling in any type of rock.

3. ADBS bit and explosive cost is almost three times as high as that for conventional DBM. However, bit and explosive cost is small compared to total advance cost (averaging approximately 1% and 4% for DBM and ADBS, respectively, in stratified and schistose rock).

4. When considered in terms of cost per foot, the ADBS equipment cost is lower than DBM. Similarly, ADBS labor cost, which is approximately 20% less than DBM on a daily basis, is significantly reduced due to the more rapid rate of advance.

5. The smallest advantage of the ADBS in terms of cost per foot over DBM shows cost savings of approximately 17% in stratified and schistose rock and savings of from 17% to 20% for the three other rock types.

Cost savings predicted for the ADBS are significant (Figure 10-2) for driving a 20 foot diameter tunnel in stratified and schistose rock as compared with the DBM system. The projected savings increase with increase in tunnel diameter.

A second method of tunneling cost analysis is based on the cost per unit of rock broken. In the United States, where more underground work has been carried out than anywhere else in the world (Sandstrom, 1963), tunneling costs are influenced by high labor costs, abundant labor supply, and

FIGURE 10-2

COMPARISON OF TOTAL DBM AND ADBS COSTS
FOR A 20 FOOT DIAMETER TUNNEL IN
STRATIFIED AND SCHISTOSE ROCK

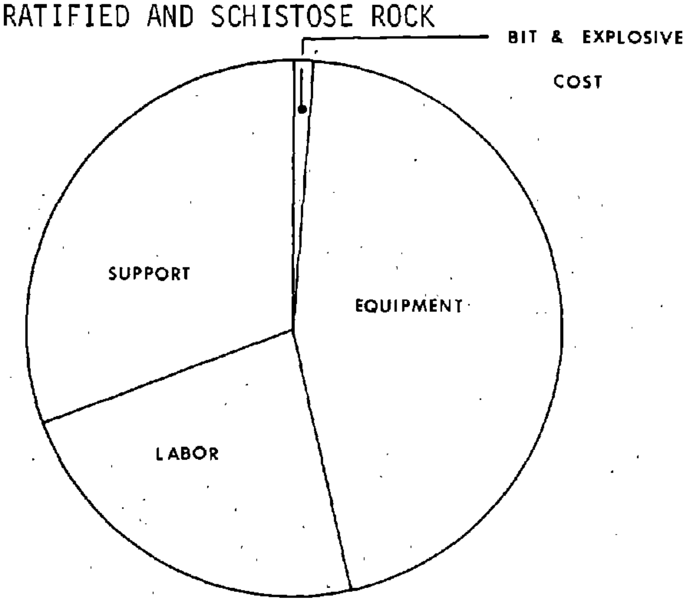


FIGURE 10-2a. TOTAL DBM ADVANCE COSTS

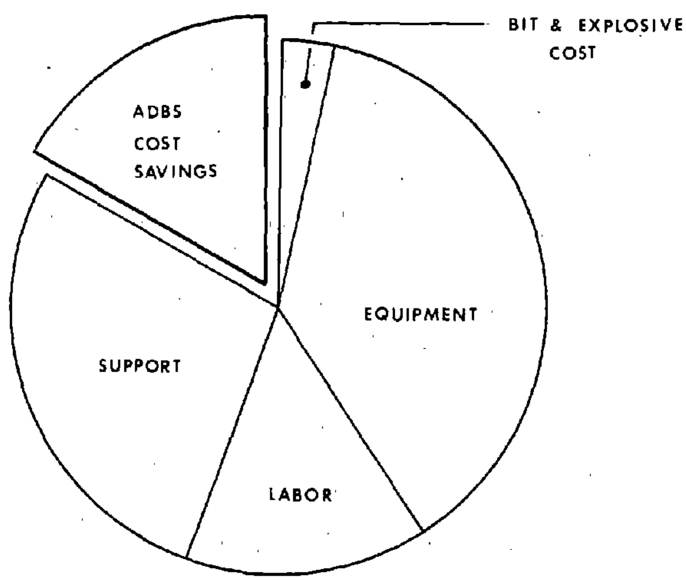
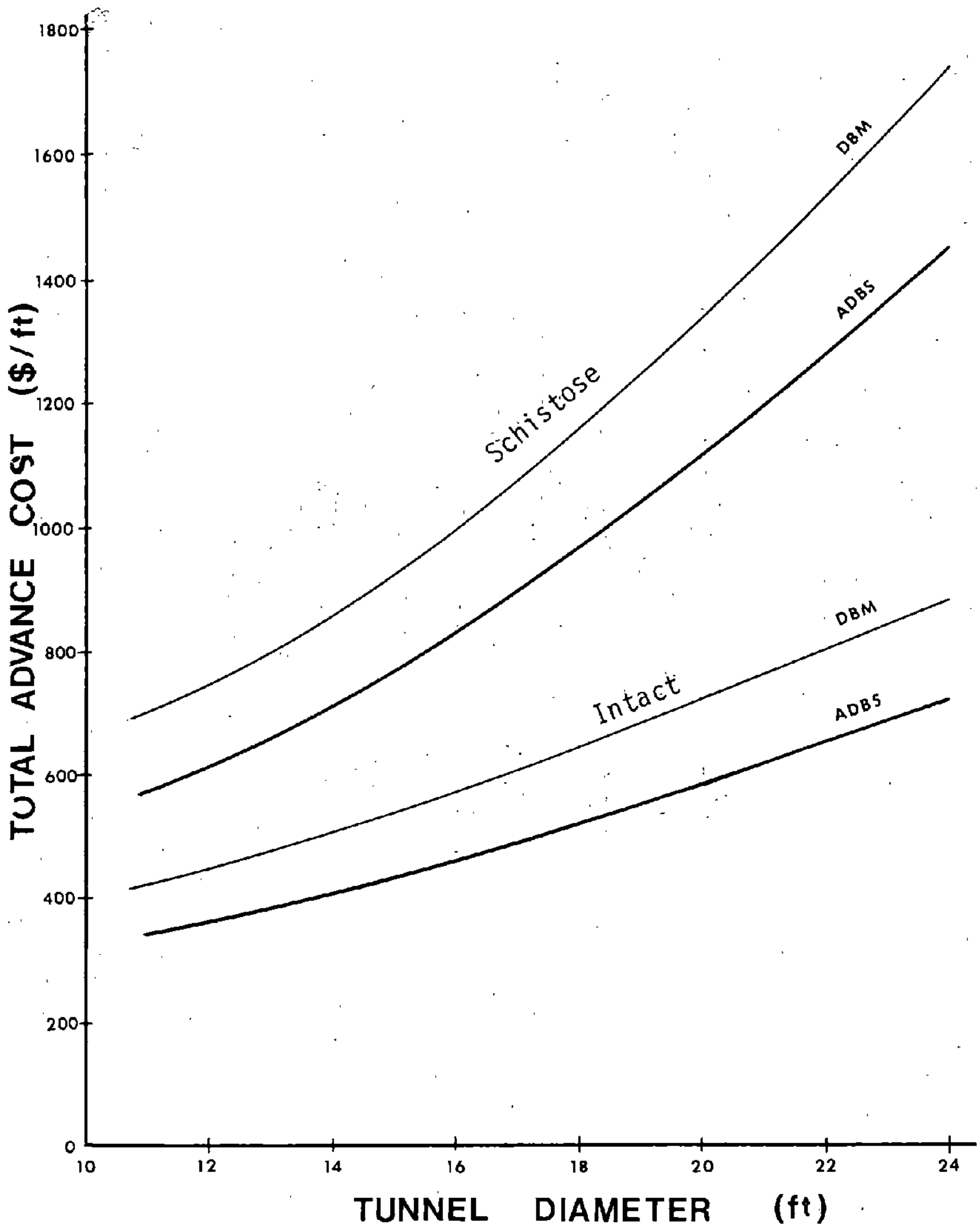


FIGURE 10-2b. TOTAL ADBS ADVANCE COSTS

FIGURE 10-3

RANGE OF TOTAL ADVANCE COSTS
FOR DBM AND ADBS



union policy. American machines are heavier and more labor and energy intensive, thus requiring a more rapid rate of advance to compensate for equipment capitalization and manpower cost.

Application of the European cost analysis to the DBM method and the ADBS yields an interesting picture. Assuming a two mile long tunnel in stratified and schistose rock with a face 13 ft by 13 ft (equivalent to a 15 ft diameter tunnel), overbreak as described above and all equipment written off, costs may be summarized as follows based on the equations and costs developed and described above:

DBM

Rate of Advance (production)		50.0 ft/day
Production Time		212 working days
Bit & Explosive Cost @ 9.90/ft		$105,000
Equipment Cost		$929,000
Labor Cost @ $10,685/day		$2,265,000
Steel Support @ $48/ft	$ 507,000	
Concrete Support @ $188/ft	1,985,000	
Overbreak Cost @ $47/ft	496,000	
Support Cost		$2,988,000
Total Tunnel Cost		$6,287,000
Total Volume Excavated		72,770 cu yd
Tunnel Cost		$86/cu yd

ADBS

Rate of Advance (production)		63.0 ft/day
Production Time		200 working days
Bit & Explosive Cost @ $28/74/ft (3 ft round)		$ 303,000
Equipment Cost		$ 955,000
Labor Cost @ $8550/day		$1,710,000
Steel Cost @ $48/ft	$ 507,000	
Concrete Support @ $188/ft	1,985,000	
Overbreak Cost @ $23/ft	243,000	
Support Cost		$2,735,000

Total Tunnel Cost	\$5,703,000
Total Volume Excavated	72,390 cu yd
Tunnel Cost	\$79/cu yd

Because the above analysis considers total volume of rock broken, including overbreak which is significantly more for DBM, the figures reflect an unrealistic cost. When the usable volume of tunnel (inside the support systems) is used, the following results:

DBM

Total Tunnel Cost	\$6,287,000
Usable Tunnel Volume	69,120 cu yd
Tunnel Cost	\$91/cu yd

ADBS

Total Tunnel Cost	\$5,703,000
Usable Tunnel Volume	69,120 cu yd
Tunnel Cost	\$83/cu yd

The above analysis shows a savings of 9% for the ADBS as opposed to DBM. Both methods of comparing ADBS vs DBM costs show significant savings in spite of many conservative assumptions utilized in determining ADBS costs.

Two additional aspects of ADBS tunneling represent significant, yet unquantifiable, cost savings. These savings are safety and reduction in materials committed to construction. The ADBS requires fewer personnel at the face. These individuals are protected in air conditioned, blasting resistant cab(s) which will provide assured safety. Personnel involved in support erection and other activities outby the ADBS are safer than under conventional conditions due to reduced rock fracture provided by controlled blasting.

The commitment of natural resources to construction projects is a major consideration as currently evidenced by environmental impact statement requirements. Powder factors lower than those currently applied to conventional tunnel rounds are anticipated for ADBS tunneling. Steel sets and concrete were considered for ADBS tunnel support in stratified and schistose rock, however, the system ideally lends itself to rock bolt and shotcrete support. This would materially reduce ADBS tunneling costs by reducing the materials required for construction.

Summary

The above analysis examined various aspects of DBM versus ADBS tunneling in order to obtain an equitable cost comparison. The specific categories considered were: (1) rate of advance, (2) bit and explosives cost, (3) capital costs, (4) labor cost, and (5) tunnel support cost. The analysis considered four of the many rock types and structures encountered in tunneling and are representative of a wide variety of conditions.

Succinctly stated, the results are as follows:

- The ADBS advance rate is estimated to be from 25% to 75% faster than DBM. The 25% increase was used in determining the costs reported above.
- Bit and explosive cost for the ADBS is significantly higher than for DBM. This higher cost is offset by a faster rate of advance, and lower labor and capital costs per foot.
- Capital cost for the ADBS is approxiamtely 3% higher than for DBM. The system cost would be lower if the ADBS were priced as a unit rather than adding component costs from various conventional systems. ADBS equipment cost, on a per foot basis, is approximately 18% lower than DBM.
- Labor cost per foot for the ADBS, due to a reduced labor force and more rapid advance rates, is approximately 37% lower than DBM on a per foot basis.
- ADBS tunnel support costs are from 0% to 9% lower than DBM, depending on rock type, due to decreased overbreak which results from controlled and blasting.

- On a per foot basis, total ADBS cost is from 17% to 20% lower than DBM, depending upon tunnel diameter and rock type.

- Two additional aspects of ADBS tunneling represent significant yet unquantifiable savings: Safety and reduction in materials committed to construction.

CHAPTER 11

CONCLUSIONS AND RECOMMENDATIONS

General

The increasing demand by society for minerals will significantly expand underground mining in the coming years. It is estimated that the United States industry must at least double its output if the nation is to retain its current ratio of domestic production to consumption. The mining industry abroad may have to triple its production to meet the demand (Malenbaum, 1973). In order to increase production rates, new methods, equipment, and technologies are required. In the fields of underground mining, safer, more efficient and more economical excavation techniques are necessary if the industry is to attain the desired production.

The research and preliminary development of a continuous small charge drill and blast excavation machine for mining and high speed tunneling has been significantly advanced in the past year. This improvement for tunnel technology has confirmed most of the projected capabilities of a machine which can remain at the tunnel face and continuously perform drilling, explosive loading, initiation, mucking, and other functions. Recent technological advances have been made with three novel methods of utilizing explosives for more rapid underground excavation. These are: (1) the spiral-Rapidex, (2) the decoupled charge-Hercules, and (3) the method utilizing small charge simultaneous firing. The latter appears to have the greatest potential for application.

Small charge blasting system has the potential of becoming a valuable mining tool for excavation of many types, including the mining of ore. Major benefits which may be obtained by utilizing this system for tunneling are a smoother, stronger drift perimeter, a more rapid advance rate, and a lower excavation cost. While only preliminary studies have been made to date, the effective use of the ADBS concept in longwall mining also appears to be feasible.

The amount of underground construction in the U.S. and overseas is also growing rapidly due to increasing congestion and inflated real estate costs in the cities and suburbs. The use of underground space for transportation and utilities is becoming relatively more economical. The ADBS is well suited for use in tunneling in urbanized areas since small charges produce only small ground vibrations and minimize the air blast and noise level.

The research reported herein had several objectives. The first was to design and field test a moveable blast shield whose perimeter can conform to the shape of a tunnel or drift, and which contains blast overpressure, fly rock, and attenuates noise to acceptable levels outby the shield. The second was to develop a conceptual design of the overall system and establish the basic configuration for the integrated components.

The primary purpose of another phase of the investigation was to determine the effects of geologic structure upon small charge blasting and ground vibration. The investigation of round parameters had the following goals: First, to design and field test 24 and 30 in. deep rounds, secondly, to evaluate the effect of the geologic structure in relation to the round parameters, and finally, to measure the air blast overpressure and ground vibration produced. A total of four rounds was tested, two 24 in. and two 30 in. rounds. The rock breakage was recorded and analyzed in terms of the effects of the cut, successive line shots, and geologic structure of the face. Safe effective slurry explosives were used and are available for the excavation conditions anticipated in full scale operations.

Finally, the economics of the system were compared with those of a conventional drill, blast, and muck system. This required analysis of advance rates in different types of rock, bit and explosives cost, and equipment, labor, and tunnel support costs.

Conclusions

The small charge blasting method, which consists of the use of simultaneously detonated, light weight charges in two to eight holes, is the key to ADBS operation. The semi-continuous blasting with small charges allows shielded equipment to remain at the face, which eliminates the cyclic inefficiency of the conventional drill, blast, and muck method.

The system will be composed of five components. The chassis and frame will consist of dual crawlers to advance the equipment, an apron and gathering arms to semi-continuously muck the face, a conveyor to discharge the muck outby the blast shield, and an upper frame to carry the drill(s), load/blast system(s), control cab(s), and the blast shield/support booms. One or more hydraulic drills will be mounted on the chassis frame. A concept for the explosive load/blast system was also developed. The explosive loading areas and the hydraulic drill(s) will be boom-mounted and operated from within the control cab. The control cab, located on the chassis frame, will protect one drill operator and one load/blast operator from blast overpressure, dust, fly rock, and excessive noise, and will contain all production and safety controls. The blast shield, mounted at the rear of the chassis, will contain blast overpressure, dust, fly rock, and excessive noise in the heading, thus allowing men to work safely outby the heading equipment.

The viability of the blast shield concept was critically examined. A review of civilian and military literature dealing with blast effects (overpressure, vibration amplitude, fly rock velocity, and noise level) indicated that the products of small charge blasting could be contained. Based on these parameters a blast shield was designed and fabricated. This shield contained many of the features which will be required for a working ADBS shield. It was mechanically expanded to fit the test drift, it sealed itself to the periphery of the drift, and was lined with a noise attenuation material.

The method of expanding various shield sections using hydraulic jacks and bolt guides in slots worked well. This mechanism functions best when the hydraulic jacks are located on the same side of the main shield section as the moving section.

The low porosity, high density foam seal between the shield and the rock effectively contained overpressure, dust, and fly rock when compressed and when the shield was adequately restrained in the drift. When the shield was moved by overpressure, the cold bond between the steel and the foam failed, thus destroying the integrity of the seal. Although the low porosity foam provides a good seal between the shield and an irregular rock surface, the shield must also be rigidly restrained.

The frictional force developed at the drift periphery was insufficient to prevent shield movement, and such movement was prevented when the shield was braced against steel pins set in drill holes, and against a loaded granby car. This support method approximates the recommended final design consisting of the friction due to the foam seal, hydraulic jacks applying direct force to the drift periphery, and the center of the shield to chassis frame connection.

The rubber lining attached to the inby side of the shield reduced the noise level outby the shield to a maximum of 112 dBA. When all holes of the shot broke the rock properly, the noise level outby was 108 dBA or less.

Field testing showed that an automated shield can successfully seal itself to a moderately irregular drift periphery by means of moving sections bounded by a foam gasket. If blast induced shield movement is prevented, the shield and seal effectively contain blast overpressure, dust, and fly rock. The rubber lining on the inby side of the shield provides significant noise attenuation and can be easily enhanced. The testing proved that a properly designed blast shield can be retracted at necessary intervals during ADBS advance, expand and seal itself to the drift periphery, adequately contain the blast products, and attenuate noise.

The analyses of the data obtained from round design and blasting showed that the V-cut pattern using small charges of 50 to 200 grams (0.12 to 0.16 lb/cu-ft) in 1-1/4 in. diameter holes fired simultaneously broke the granite gneiss with about 18 in. spacing and 7 to 14 in. burden. The V-cut pattern which consists of 2 to 6 holes per line detonated simultaneously makes effective use of stress wave reinforcement. The energy and the stress propagated from two adjacent charges caused high stress concentration along the line of holes and resulted in good fracture control, rock breakage, and effective fragment distribution. Also, the breakage into the walls of the excavation was limited because of the utilization of small charges and the geometry of the holes in a line. The fracture control was somewhat diminished and cracks were created in the rock when they intersected natural fractures or open joints. These joints and fractures may cause large fragments to be thrown into the muck pile.

Mapping of the face permits meaningful evaluation of the effects of structure and the determination of the correct amount of explosive needed for successive shots. Better fracture control and fragment distribution were obtained when a modified drill hole placement pattern was used. That is, the influence of the joint or natural fracture was overridden when the holes were placed in the solid rock between the joints. This resulted in better fracture control, and the size of the largest rock fragments was reduced for effective muck handling.

The maximum throw of the fragments from the tunnel face is a function of the powder factor employed. The large fragments up to 2 ft were always found near the face and no damage was done to the shield by the impact of the smaller fragments.

A regression analyses of the peak particle velocities showed that the cube root as well as the square root scaling law fit the ground vibration data for granite-gneiss and the geophone location employed. A comparative analysis of the particle velocities produced by small charges and conventional blasting showed a reduction of peak velocities by a factor of 55 compared to those obtained by conventional blasting (Kid Creek) and previous small charge tests. The geometry of the drift and the location of the seismometers, however, had significant effects on the transmission of the wave. Thus, a reduction of damage potential to rock structures and support systems caused by ground vibrations is obtained due to the small size of charges and the geometry of wave travel in the rocks in the walls of the drift.

In the beginning of the experimentation, several holes bootlegged, which resulted in high air blast overpressure and ground vibration. The results obtained from a crater test for measuring the effectiveness of two explosives were similar, showing that the slurry explosive was equivalent to a 65% dynamite, and that the explosive strength was not the cause of the bootlegs. Therefore, the powder factor was increased by 30% to 50% for some of the holes and no further bootlegs occurred.

Although slightly weaker than the dynamite, the slurry explosive has several advantages because it is safe for handling and will meet many of the requirements for automated operations.

The results of peripheral rock breakage by small charge blasting are similar to those of the smooth blasting technique. That is, the overbreak and fracture into the walls are limited because of the larger number of holes and smaller amounts of explosive, solid, smooth walls are produced which provide good sealing of the blast shield, and rock support requirements are significantly reduced.

Based on the preliminary design configuration of the ADBS, the economics of tunnel driving using this system were compared with conventional drill, blast, muck (DBM) tunneling. Four rock types (intact rock, stratified and schistose rock, limestone and sandstone), which dictated the advance rate, and four cost categories (bits and explosives, equipment, labor and tunnel support) were analyzed.

The ADBS advance rate is more rapid than DBM because the former can remain at the face and muck semi-continuously. The advance rate was estimated to be from 25% to 75% greater than that of the DBM, the 25% increase was used in cost calculations.

ADBS bit and explosive cost is relatively higher than for DBM. However, this cost accounts for only 4% of the total advance cost per foot and is offset by the faster advance rate, the lower per foot labor and equipment costs and possible lower rock support costs.

The estimated total capital cost for the ADBS is approximately 3% higher than for DBM. This cost would be lower if the ADBS were priced as a unit rather than adding component costs from various conventional systems. On a per foot basis, ADBS equipment cost is approximately 18% lower. Because of reduced labor force and more rapid advance rates, ADBS labor cost is 37% lower than DBM on a per foot basis.

Controlled blasting, which results from the use of small charges, reduces overbreak, which decreases ADBS support costs by up to 9%, depending on rock type. Controlled blasting also provides an unquantified reduction in the load to be carried by the support system. This decrease was not considered in the cost analysis.

On a per foot basis, the calculated total ADBS advance cost is from 17% to 20% lower than DBM, depending upon tunnel diameter and rock type. This reduction is based on an assumed conservative increase in advance rate, savings in labor, equipment, and support costs.

The ADBS has additional advantages whose cost savings cannot be easily quantified. Small charge blasting produces significantly lower ground vibrations than conventional blasting. This feature will greatly reduce complaints in urbanized areas. Controlled blasting also reduces overbreak and fracture which decreases muck haulage and backfill. Required ground support is also reduced, however, the magnitude of the reduction is not currently quantified. The ADBS provides a significantly safer working environment because fewer personnel are required in the heading. These men will be protected in air conditioned, blast resistant cabs.

Recommendations

All of the tests conducted to date indicate that the use of the small charge method of excavation will be technically and ecnomically feasible. Certain areas for improvement of the experimental components have emerged from the test results, and the requirements for further research and developments clarified.

The following improvements can be made on the shield now on hand, and the effects of these improvements evaluated in further tests at the CSM Experimental Mine:

a. Line both the inside and outside of the shield with rubber belting to obtain greater noise reduction.

b. Place a double line of heavier bolts on the sliding portions of the shield to prevent buckling.

c. Use heavy steel pins at about 8 positions around the periphery of the shield inserted into shallow drill holes to anchor the shield. Ultimately these would be automated.

d. Test the shield for stability with the maximum expected air blast if all holes in a shot bootleg.

e. A simple but heavy support should be provided against the center of the shield.

While the rounds utilized were effective, they should be designed in conformance with the curves in Figure to determine the optimum spacing vs depth and charge weight. If spacing can be increased further without increasing the powder factor, the cost of blasting caps will be decreased in proportion. That is, after the face area or the depth of holes reach a certain value, the number of holes per unit area of face for a given rock type and round design is constant. However, the air blast and ground vibration, with increase of explosive per shot, is critical.

Other round designs, such as draw-cut, pyramid, and burn-cut, should be analyzed to determine their adaptability to the small charge technique of blasting.

While the small charge method of blasting was originally conceived in connection with the rapid excavation of tunnels, there appears to be no reason why this cannot be applied to other blasting geometries, such as long wall mining, blasting in stopes, raises, shafts, and similar openings. In the longwall mining of ore, for example, the drill operator would be housed in a protective cab, a shield would be provided to retain the blasted material on a conveyor belt, and the operation could proceed in a continuous manner. Initial experimentation is this direction was conducted at White Pine and the blasting technique works well in breaking the ore and waste separately.

One of the next steps in the research will be to make a detailed design of a protective cab for operators, to construct it and to mount it on a simulated mobile chassis, and to test it under blasting conditions. The chassis would probably be made up of a heavy track type vehicular frame. The cab would be fabricated with door, window, and the necessary sound insulation. Instrumentation will include that for measuring the vibration of the components of the cab, the overall movement of the cab, the sound level within the cab, and its structural resistance to air blast and fragment impact.

The conceptual elements of an explosive loading and firing system have been described in the body of the report. The research, development, design, construction, and testing of such a system should be a priority item for the next phase of the research program.

The layout of the experimental drift in the CSM Experimental Mine is such that ground vibrations may be measured in a heading which is parallel to the test drift. It is highly desirable to measure ground vibrations along the test drift, but also in the parallel drift to determine the relationship of the magnitude of peak velocity and the direction of propagation of the waves in the rock with respect to the working face.

In summary, the next research areas or phases of the project should be concerned with the evaluation of the following:

a. Simple improvements of the shield which can be conducted with the shield on hand and their effectiveness tested.

b. Further tests should be made of design of the V-cut round of the relationships between depth, spacing, burden, powder factor, air blast, and ground vibration. Analyses should be made of the possible applicability of other types of rounds.

c. Further measurements should be made of ground vibrations in directions other than along the sides of the experimental heading.

d. A protective type cab should be designed, constructed, and tested under operating conditions.

e. An explosive loading and firing system should be designed, constructed, and tested. This would include a mixing and cartridge fabrication machine to be developed and designed by an explosives company and field tested at the CSM Experimental Mine together with the loading system.

f. An in depth feasibility study should be made of the possible application of the small charge technique of blasting to other types of rock breaking operations, including stopes, raises, winzes, shafts, etc.

g. Other phases of research pertaining to the successful application of the small charge method should be continued, such as updating the economic studies, etc.

APPENDIX A

BLAST PARAMETERS AND BLAST SHIELD DESIGN

The simultaneous detonation of several small stemmed charges of high explosives as employed in the ADBS produces air blast overpressures, high velocity rock fragments, vibration, and noise which are lower than those associated with conventional blasting. The characteristics of these four parameters dictate the design of the blast shield.

Air Blast Pressure

There are only a few references in the literature to measurement of air blast pressure in underground openings due to blasting. One recent analysis of air blast pressure from confined underground production blasts indicates that a cube root law equation describes the peak pressure decrease with distance (Olson & Fletcher, 1971):

$$P = 4.9\ (10)^3\ (D/W^{1/3})^{-2.15} \qquad \text{(A.1)}$$

where

P = overpressure (above atmospheric), psi

D = distance from the blast, ft

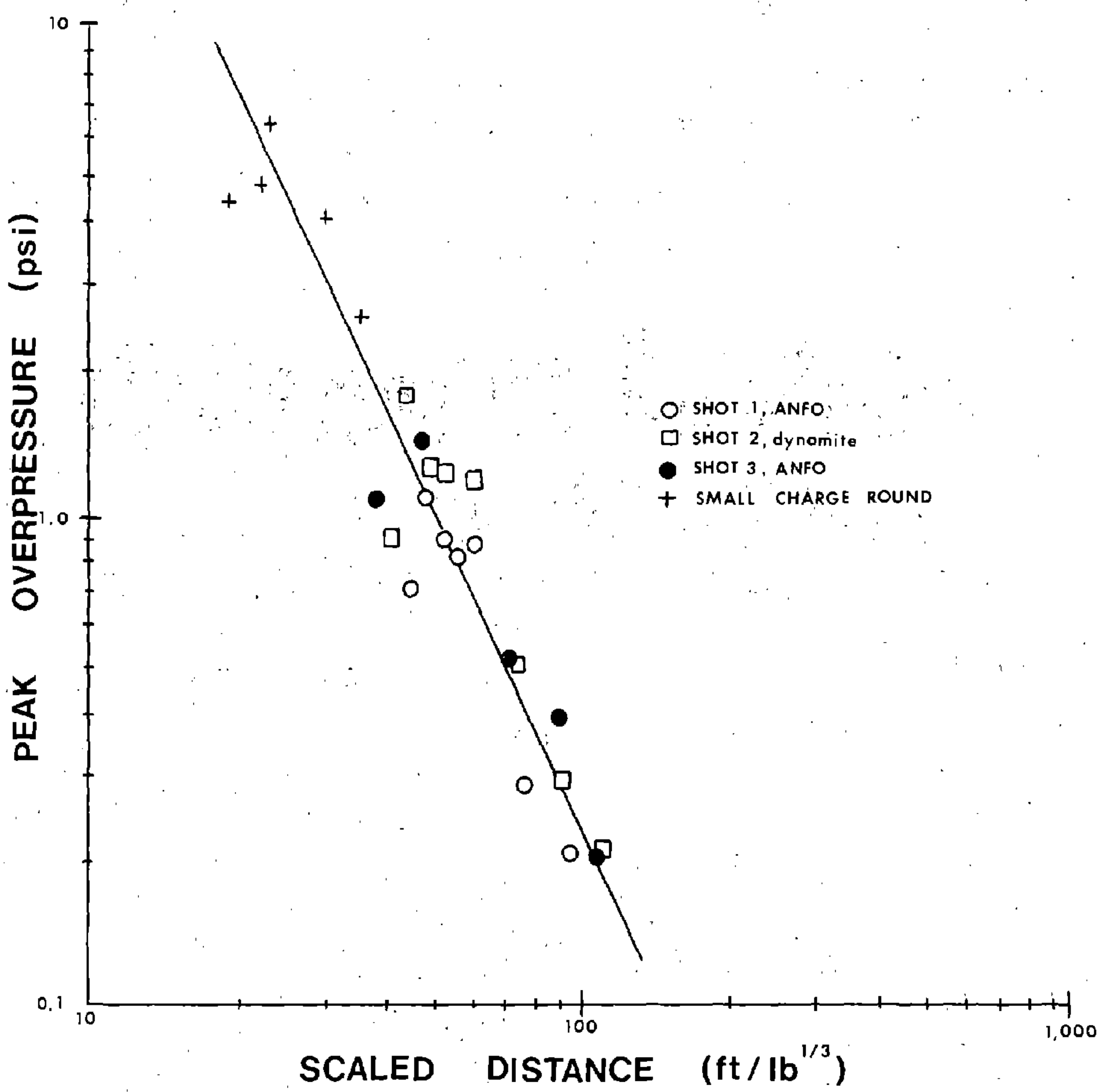
W = zero - delay charge weight, lbs

The above equation was developed from the data plotted in Figure , for distance from the blast to the gage station varied from 120 to 970 ft and generated a maximum peak pressure of 1.76 psi.

Studies conducted in a short 8 x 8 ft tunnel in granite (Clark & Rollins, 1976) yielded similar results (Table A.1). These data, included in Figure A-1, conform with the results of the prediction equation A.1 within experimental error, indicate that it can be used to predict the peak pres-

FIGURE A-1

PEAK OVERPRESSURE VS
SCALED DISTANCE



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sure acting on the blast shield. Also, the effective force acting on the blast shield can be calculated if the pulse shape and duration are also known. This is the force which must be resisted by the shield.

TABLE A-1

SUMMARY OF BLAST RESULTS - TUNNEL IN GRANITE

Shot No.	No. of Sticks of 40% Dynamite	Overpressure (psi)	Distance (ft)	Scaled Dis.[1] ($D/W^{1/3}$)
1	4	5.5	22.5	18.5
2	3	6	22.5	20.4
3	3	--	25.5	23.1
4	5	--	25.5	19.5
5	5	8	25.5	19.5
6	2	5	25.5	26.4
7	1	3	25.5	33.3

[1]Assumes one stick of 40% dynamite weighs 0.45 pounds.

Values of peak pressure (Table A-2), assuming a seven foot square shield, six blast holes detonated simultaneously, and each hole loaded with one-quarter pound of slurry explosive, decrease rapidly with distance from the blast. The blast is assumed to occur at the center of the face. The effective pressure is the average pressure distributed over the total shield area.

The above results and the values tabulated in Table A-2 are for stemmed charges and assume that a hemispherical wave front is preserved for a distance equal to one tunnel radius down the drift before reflections become important. This is probably accurate for charges detonated near the tunnel center line but may be inaccurate near the rib, back, and invert due to reflections.

The two sets of tests summarized above were conducted in a working mine and a short (25 ft) tunnel, respectively. For this reason, the reported overpressures may be low due to the influence of cross-cuts and the tunnel portal, respectively.

TABLE A-2

TYPICAL PEAK AND AVERAGE PRESSURE ON A 7 x 7 FT BLAST SHIELD AT VARIOUS DISTANCES PRODUCED BY SEVERAL SMALL STEMMED CHARGES OF HIGH EXPLOSIVE

	Distance From Blast (ft)					
	10	12	14	16	18	20
Peak Pressure (psi)	46.4	31.3	22.5	16.9	13.1	10.5
Effective Pressure (psi)	39.1	27.7	20.5	15.7	12.4	10.1

Blast shield design was based on the peak pressure of 25 psi of a triangular pulse of 0.40 msec duration, which will conservatively account for possible underestimation incurred by applying equation A.1.

High Velocity Fragments

The research concerning the impact of particles on metals may be divided into two general categories:

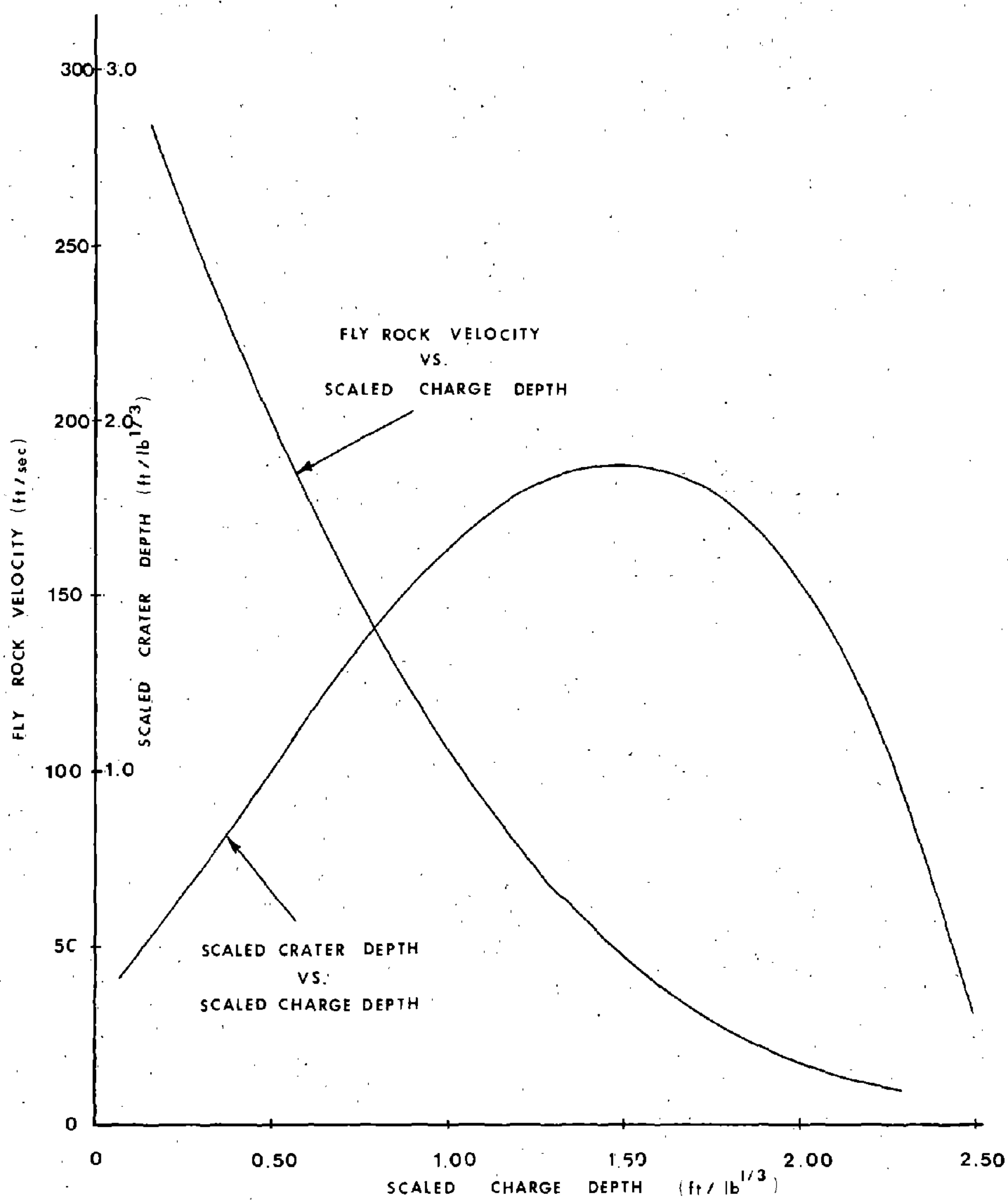
1. Hypervelocity - greater than 5,000 fps; and,
2. High velocity - less than 5,000 fps.

Hypervelocities result in the expected types of impact of micrometeorites on space vehicles. Further, the velocity of fragments created by blasting is far below the 5,000 fps range, and thus, the lower part of high velocity regime reflects ADBS blasting conditions.

Particle impact velocity is of primary concern for design purposes. Within twenty feet of the detonation, striking velocity is usually assumed to equal the initial particle velocity (Healy, 1975). A study of fly rock velocity from single cratering charges in sandstone showed a maximum fly rock velocity of 260 fps (Duvall, 1957). A charge size of 8 pounds was utilized in the tests and the shallow depth of charge burial resulting in the above velocity was only 0.44 ft (Table A-2, Figure A-1). Thus, the scaled depth of burial ($D/W^{1/3}$) was 0.22. The optimum scaled depth for maximum depth of cratering in sandstone was found to be approximately 1.6 (Figure A-2).

FIGURE A-2

PARAMETERS EFFECTING FLY ROCK VELOCITY



In order for the small charges to be effective, the explosive per volume of rock to be broken must be at or near optimum. Thus, based on the data presented in Figure A-2, fly rock velocity of large particles would be approximately 50 fps. Even assuming a gross drill hole spacing error, maximum fly rock velocity would not be expected to exceed 80 fps.

Tests using small charges were conducted and the resulting fly rock throw was filmed with a high speed movie camera (Clark, 1976). Larger fragments (15 to 17 in. diameter) were reported to have moved with velocities up to 171 fps, but this value appears to be in error.

TABLE A-3

FLY ROCK VELOCITY DATA FOR SANDSTONE

Explosive	Charge Depth (ft)	Charge Weight (lb)	$(D/W^{1/3})$	Fly Rock Velocity (ft/sec)
Semigelatin	0.9	8	0.45	210
	2.4	8	1.20	80
	2.4	8	1.20	90
	2.9	8	1.45	70
	3.9	8	1.95	13
	4.7	8	2.35	9
Gelatin	1.8	8	0.90	120
	2.5	8	1.25	70
	2.9	8	1.45	50
	3.9	8	1.95	19
Ammonia Dynamite	0.44	8	0.22	260

An additional consideration is fly rock throw. Small fragments (less than 3 in. in diameter) were thrown as far as 50 ft. which would indicate a maximum fragment velocity of approximately 200 fps., but over 98 percent of the broken rock was within 14 ft of the face. A maximum fly rock velocity of 80 fps is anticipated for small charge blasts. However, a velocity of 400 fps was utilized for shield design.

Fragment impact effects for military purposes are divided into two categories: Those on the front face of the object and those on the back face. Front face effects include fragment deformation, ricochet, shatter, and indentation. Back face effects deal with perforation and the residual particle velocity in the event that perforation occurs. The blast shield design must preclude back face effects for safety reasons.

The failure magnitude depends on the ratio of the mass of the fragment to the mass of the target and may be either gross or local. Gross failure occurs when large fragments impact flexible bodies and is characterized by bending or collapse. Such conditions are not anticipated since overall structural response to impact is insignificant where the mass ratio is small, as is the case with a blast shield.

For mild to hard steel, ductile (local) failure is of primary concern. In this case, the fragment plastically deforms the shield material during penetration without ejecting plate material. This phenomenon is frequently recognized on highway signs and explosives magazines subjected to bullet impact. Discing or flaking may occur on the back face. This type of failure is mainly of concern where plates of inferior quality steel are utilized. Thus, discing and flaking is not a problem for blast shield design.

When steel penetrates steel, where the hardnesses are similar, the probability of fragment shatter and ricochet is high. The equation for penetration of mild steel fragments into mild steel plates (Healy, 1975) is

$$Z = 1.63\ D(V_s)^{1.22} \tag{A.2}$$

where

Z = penetration in inches

D = fragment density in pounds per cubic inch

V_s = fragment striking velocity in 1,000 feet per second

In the case of the blast shield, with rock impacting steel, shatter and ricochet are the dominant characteristics of the impact. If shatter and ricochet are neglected and equation A-2 is applied, assuming a rock density of 160 pounds per cubic foot (0.093 pounds per cubic inch), and a velocity of 400 fps per the above discussion, the penetration is approximately 0.05 in. Tests conducted using a fixed shield and small charges (Clark & Rollins, 1976) showed very little pitting of the shield.

Shield Vibration

Vibrations generated by blast overpressure, and to a lesser degree fragment impact, are of concern because of the possibility of gross blast shield failure, probably in the form of bending or collapse. Failure of the shield support mechanism might also occur. Vibration amplitude, frequency, and duration determined the blast shield response. Natural vibration frequencies of thin flat plates of uniform thickness are given by Harris (1961):

$$\omega = B\ Et^2/\rho a^4(1 - \nu^2)^{1/2} \qquad \text{(A.3)}$$

where

ω = natural frequency in radians per second

B = constant based on plate geometry and edge condition

E = Young's modulus in pounds per square inch (30×10^6 psi for steel)

t = plate thickness in inches

ρ = mass density in lb-sec^2/in.^{4}

a = diameter or length of side in inches

ν = Poisson's ratio (0.25 for steel)

Values of the constant B are given in Table A-4 for selected plate geometries and edge conditions assuming a 7 x 7 ft plate, 1/4-in. thick.

In general, clamping the plate at the edge increases rigidity and the natural frequency. The same effect is obtained by reinforcing the plate with structural shapes such as angles or tees, and/or increasing the thickness of the plate. The natural frequency varies inversely as the square of the lateral dimensions. Neither the fundamental mode (as described below) nor any of the higher natural frequencies will be excited by the detonation of several small charges. However, precursors or local stress waves are generated. Such a localized stress may adversely affect bolted, riveted, or welded joints if the vibration amplitude, frequency, and duration are sufficiently high.

The above fundamental parameters of importance in the design of a blast shield are utilized in this study of response spectra by considering the plate to represent a mass spring system with a single degree of freedom (Jacobsen & Ayre, 1958). Pertinent definitions are:

1. The spectrum of a forcing function is the relationship between a given response of a single degree of freedom oscillator and the ratio of the characteristic period (or frequency) of the forcing function to that of the oscillator, i.e., τ/T, or ω/p, where τ is the period of the forcing function, T is the period of the oscillator, and ω and p are their respective frequencies.

2. The maximum displacement, velocity, and acceleration of the oscillator caused by the forcing function, i.e., the maximum displacement, etc., are denoted by X_M, $\dot{X}_M$, and $\ddot{X}_M$.

3. The maximum distortion caused by shield motion, or maximax distortion, is denoted by X_D.

4. The maximum displacement during the residual vibration period, residual amplitude, is denoted by X_S. This is given as a dimensionless ratio of the particular response to a transient to the maximum displacement, X_S, caused by a static load equal to the peak pressure of the transient.

TABLE A-4

NATURAL FREQUENCIES AND PERIODS OF
THIN FLAT PLATES OF UNIFORM THICKNESS

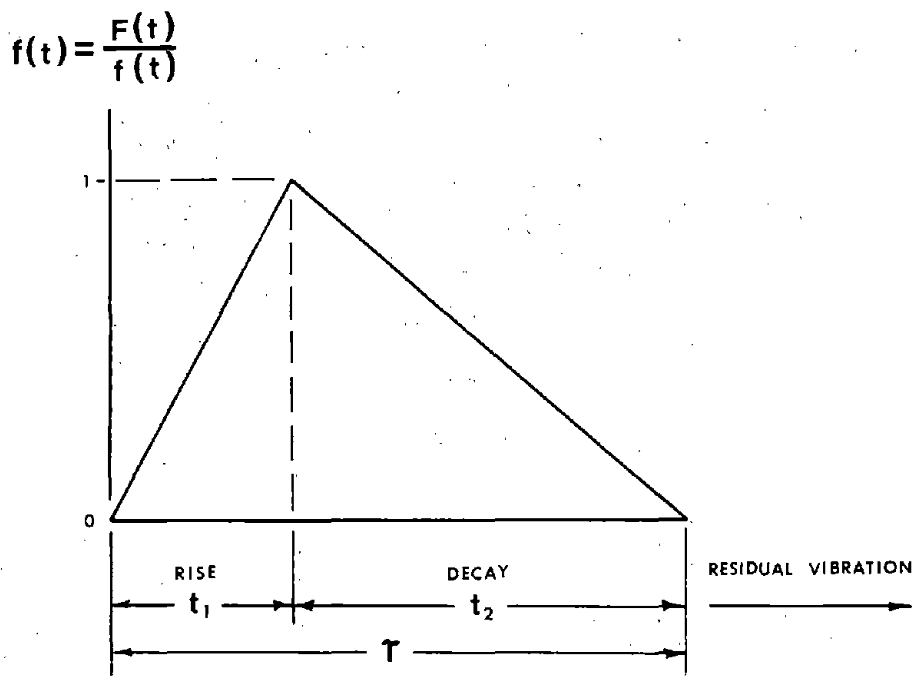
Shape of Plate	Edge Condition		Mode 1	2	3	4	5	6
Circular	Free	B	6.09	10.53	14.19	23.80	40.88	44.68
		ω_1	83.98	145.2	195.7	328.2	563.7	616.1
		T[1]	75	73	32	19	11	10
Circular	Simply Supported At Edge	B	5.90					
		ω_1	81.4					
		T[1]	77					
Circular	Clamped At Edge	B	11.84	24.61	40.41	46.14	103.12	
		ω_1	163.2	339.4	557.3	636.3	1422.0	
		T[1]	38	18	11	10	4	
Square	All Edges Free	B	4.07	5.94	6.91	10.39	17.80	18.85
		ω_1	55.3	80.7	93.9	141.2	241.9	256.2
		T[1]	114	78	67	44	26	25
Square	All Edges Simply Supported	B	5.07	14.26	22.82	28.52	37.08	48.49
		ω_1	77.5	193.8	310.1	387.6	508.9	695.0
		T[1]	81	32	20	16	12	9
Square	All Edges Clamped	B	10.40	21.21	31.29	38.04	38.22	47.73
		ω_1	141.3	288.2	425.0	517.0	319.4	648.7
		T[1]	44	21	15	12	12	10

[1]Period measured in milliseconds.

The motion of a flat plate is given by a fourth order differential equation. Such an equation may be solved for different boundary conditions (Jacobsen, 1958). However, when such a system vibrates in a natural mode, all segments of the system display simple harmonic motion. Thus, a simple spring-mass analysis is applicable. For triangular pulses (Figure A-3), which have been found to accurately model simple blast waves, the relationship of rise time, fall time, and pulse duration is expressed in terms of a factor $\sigma = t_1/\tau$. Thus, if $\sigma = 0$, the rise time is instantaneous; if $\sigma = 1$, the decay time is instantaneous.

FIGURE A-3

THE GENERAL TRIANGULAR PULSE



In the design of the blast shield, the residual amplitude, the maximax displacement, and the maximax distortion as functions of the period ratio are of particular importance (Figure A-4). The natural period, T, of the blast shield (7 x 7 ft x 1/4-in.), all edges clamped is 44 msec (Table A-4). The anticipated blast pulse period is 0.4 msec (Clark & Rollins, 1976). Thus, the period ratio is:

$$\tau/T = 0.4/44 = 0.01$$

The corresponding displacement, X_M/X_S (Figure A-4), is approximately 0.02. The maximax displacement for $\sigma = 0$ increases asymptomatically to a value approximately equal to 2. In other words, the plate would be displaced by the pulse approximately 1% of the maximum possible under a static load equal to the peak blast pressure. Assuming a blast shield of the above dimensions constructed of ASTM A440 steel (Fy = 50 ksi), the factor of safety against shear or bending failure is greater than 10.

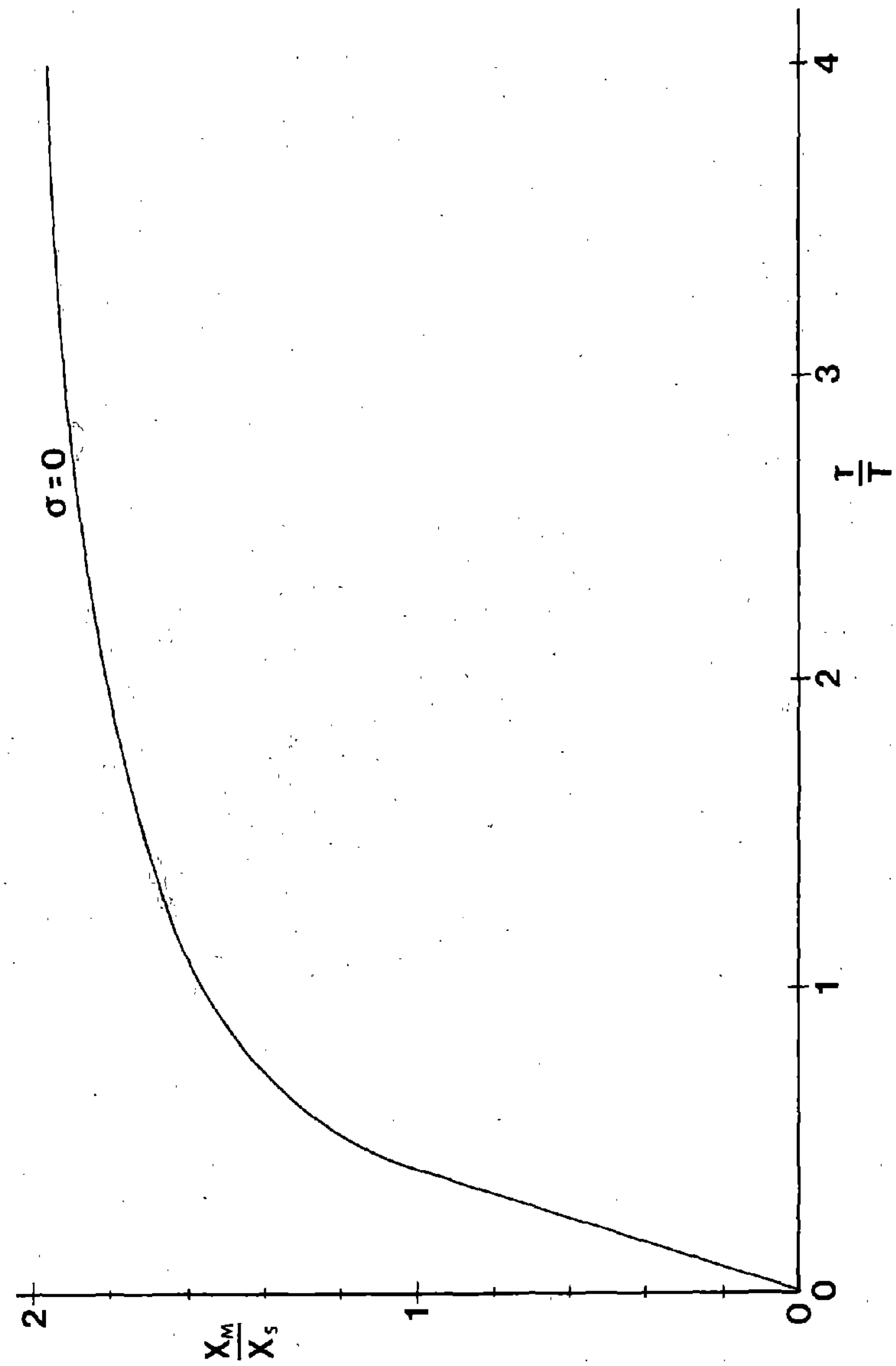
Based on this analysis, neither the fundamental mode or higher natural frequencies will be excited due to the small plate displacement. Structural supports, utility fittings, men and material access openings, and other appurtenances fastened to the blast shield tend to increase the resistance to motion of the entire shield. Thus, blast overpressure and shield vibrations are far below critical.

Noise and Its Effect on Humans

A study of short random bursts of noise to determine perceptual growth and decay of noise experienced by humans (Miller, 1968) found that a short term noise must be more intense to equal the effectiveness of a noise of longer duration. The threshold of hearing is reduced by the increase of noise duration up to one second. However, the loudness of an intense noise

FIGURE A-4

MAXIMAX DISPLACEMENT DUE TO A TRIANGULAR PULSE



depends on duration only up to 76 msec. Miller (1968) concluded that ". . . it is inferred that the auditory system acts as if the growth and decay periods of noise perception depend upon differences in latencies among the various meural paths in transmitting the chochlear activity to the higher centers of the brain. According to this hypothesis, the activity of the slowest pathways arrives at the center 65 msec after activity of the fastest pathways."

Figure A-5 shows that the growth of noise perception increases with time (Rice & Zepher, 1967). However, if faint short and long noises are amplified from 20 to 90 dBA, they are no longer perceived as sounding equal (Figure A-6).

Research indicates (Rice & Zepher, 1967) that a weighted energy function concept permits the subjective loudness of various types of single event transient sounds to be predicted from the Fourier transform of the wave and the frequency of the hearing mechanism. For these experiments, special earphones were used and the sound source was a 22 caliber blank producing a pulse of 50 msec duration. The wave form was determined and approximated as a triangular wave shape. The modulus of the wave form F (ω) was then written in terms of the frequency and the rise and decay time, from which a weighted energy density was obtained. This was then used to determine loudness, assuming a relationship based on empirical evidence.

The F (ω) of a pistol shot wave is essentially a flat distribution with regard to frequency. Thus, the energy density waves are functions of weighting factors of the hearing mechanism. These relationships predict maxima in frequency ranges of from 400 to 4,000 cycles per second. The choice of dBA level affects the location of the frequency maxima on the weighted density curves.

FIGURE A-5

GROWTH OF THE PERCEPTION OF NOISE
AS A FUNCTION OF TIME
(RICE AND ZEPHER, 1967)

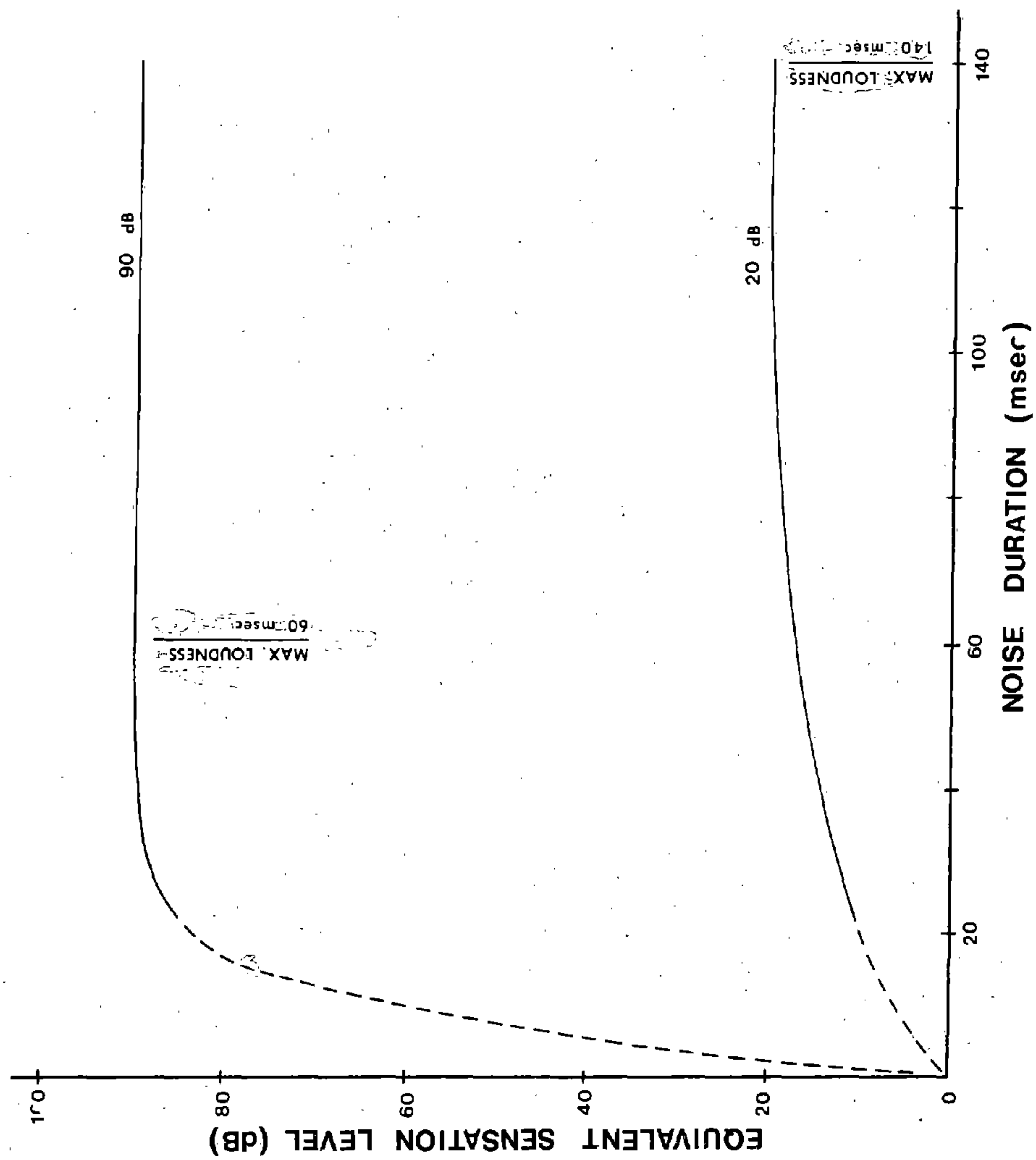
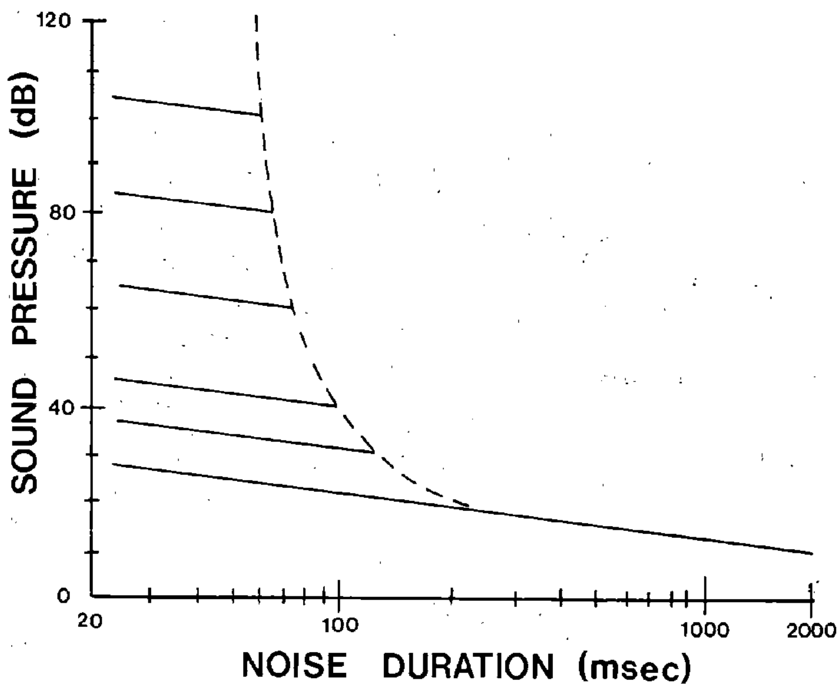


FIGURE A-6

EQUAL-LOUDNESS CONTOURS FOR NOISE
AS A FUNCTION OF DURATION



Note: The sound pressure of the shorter burst of noise, which sounded equal in loudness to the longer burst, is plotted as a function of the duration of the shorter burst. The dashed curve indicates the shortest duration for which loudness is independent of duration (Rice and Zepher, 1967).

In the initial portion of this project (Clark & Rollins, 1976), studies were conducted to determined sound intensity attenuation using a fixed blast shield and lead-vinyl curtains. Additionally, blast-induced sound intensities were measured at various distances from the blast with the shield and curtains removed (Table A-5). Distances were measured from the face and the blast shield was 8 ft from the face. Inspection indicates that unattenuated intensity, based on shot numbers 25 through 28, would exceed 120 dBA 20 ft from the face.

Studies (May, 1971) have shown the startle effect is proportional to loudness depending on the subject's expectancy, and is low where the background noise level is high. Research techniques employed in such studies are well developed and consider a large number of variables including frequency, background frequency, accoustic noise, subject vs experiment, presentation of stimulus, vision, footwear, sex, posture, subject restraint, and subject attention.

Further studies of the quantitative effects of these factors on humans working with the ADBS will be required to evaluate the acceptability of the noise and vibration levels in a cab and outby the tunnel shield.

TABLE A-5

BLAST-INDUCED NOISE INTENSITY WITH AND WITHOUT BLAST SHIELD AND LEAD-VINYL CURTAINS

Shot No.	Charge Weight (gms)	Initiation	Blast Shield	Pb-Vinyl Curtains	dBA
12	230	1 cap & primacord	std.	None	107 @ 30'
13	230	5 caps	std.	None	105 @ 30'
14	230	4 caps	std.	4	100 @ 30'
18	613	---	1*	4	109 @ 30'
19	230	---	1*	4	104 @ 30'
20	230	---	1*	4	105 @ 30'
21	230	---	1*	4	102 @ 50'
22	230	---	1*	4	96 @ 70'
23	230	---	1*	4	104 @ 30'
24	613	---	1*	4	108 @ 30'
25	613	---	Removed	None	114 @ 52'
26	184	---	Removed	None	108 @ 70'
27	184	---	Removed	None	108 @ 70'
28	184	---	Removed	None	107 @ 80'

*Indicates a 1/4 inch thick layer of plywood on inby side of blast shield.

Blast Damping Requirements

As was indicated above, the simultaneous detonation of several small stemmed charges of high explosives produces air blast pressure, high velocity fragments, vibration, and noise. The characteristics of these parameters have been established. The element of the ADBS most sensitive to these effects is the workman. All four of the blast effects can be reduced or contained so that a permissable working environment can be provided.

Air Blast Pressure

Recent tests (Army, 1969) have indicated that the lungs are one of the critical organs subject to blast pressure injuries. The release of air bubbles from disrupted alveoli of the lungs into the vascular system probably accounts for most injury and death. Based on present data, a tentative estimate of human response to fast-rise pressures of short duration (3-5 msec) is given in Table A-6. Human tolerance to blast pressure is governed by the lowest tolerance of a given body element which is the ear.

For continued exposure, air blast pressures must be reduced to less than 2 psi. As indicated above, the maximum anticipated peak air blast pressure on the inby side of the shield is 15 psi.

TABLE A-6

HUMAN TOLERANCE TO AIR BLAST PRESSURE

Critical Organ or Event	Overpressure (psi)
Ear drum rupture:	
Threshold	5
50% casualty	15
Lung damage:	
Threshold	30-40
Severe	80+
Lethality:	
Threshold	100-120
50% casualty	130-180
99%+ casualty	200-250

High Velocity Fragments

It has been demonstrated that the penetration of a high velocity rock fragment into the steel front face of the blast shield is negligable. All high velocity fragments will be stopped by the shield.

Vibration - Human Response

A study by Rice and Zepher (1973) established three levels of human response to vibration:

Level I - threshold of perception

Level II - threshold of discomfort

Level III - threshold of tolerance

Differences due to direction of application and subject attitude (standing, sitting, or lying) were noted. Higher frequencies were found to be less tolerable than lower frequencies (Figure A-7). More specifically, higher frequencies are more tolerable at lower amplitudes and peak accelerations are more tolerable at higher frequencies.

Studies conducted to determine the effects of short term vibrations on the cardiovascular system of animals (Zepler, 1973) were documented by monitoring the control and regional blood flow velocities and blood pressure of animals. The animals were anesthetized, restrained with spine vertical and vibrated along that axis for 30 seconds. The frequencies were varied from 2 to 12 Hz with accelerations 1, 2, and 3g.

It was found that the 3 to 9 Hz range resulted in minimum flow of blood rates for all accelerations. Extreme values occurred at 3g and 4 Hz and yielded (1) a maximum aortic flow rate of more than twice the control (normal) value, (2) a minimum aortic flow rate of 90% less than the control values, and (3) an increase in blood pressure of more than 5 times the control value.

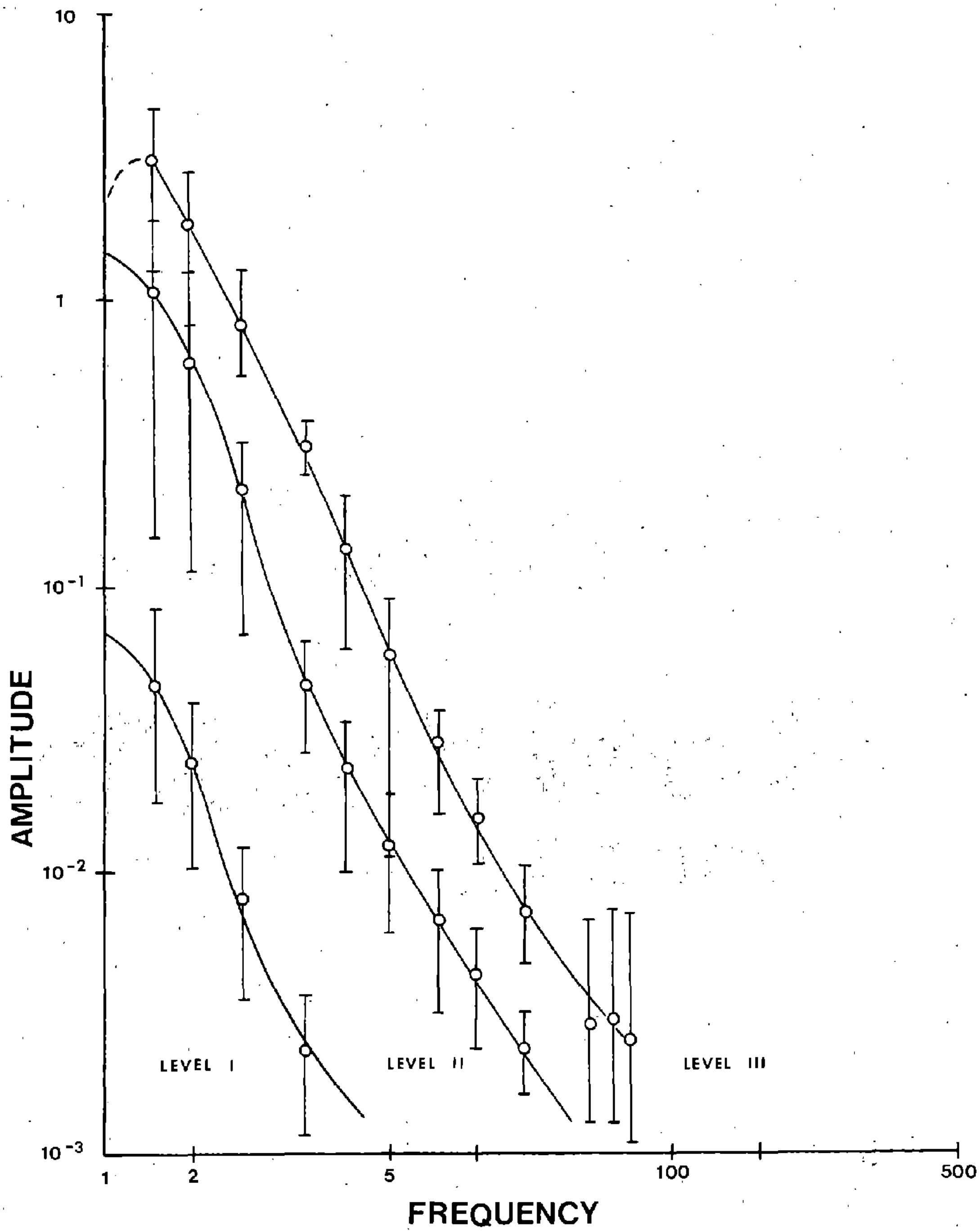
It may be inferred that the effect on humans is similar. However, since the blast pulse may be measured in msec and the resulting plate oscillation measured in seconds, human response to the anticipated amplitude and frequency of vibration will not exceed allowable limits.

Noise Measurement and Regulations

The two parameters which are usually employed to measure industrial noise are sound pressure and frequency (Bell, 1973). Sound pressure of interest varies from 10^{-9} to 20 psi. The decibel equivalent of pressure varies from 0 dB (3×10^{-9} psi) to 160 dB (3×10^{-1} psi). Levels of sound from different sources are combined by recursive methods, and frequencies are multiplied by different types of weighting scales based on octave or 1/3 octave separation of frequencies. Sound power, the rate at which accous-

FIGURE A-7

THREE LEVELS OF RESPONSE TO VIBRATION
(ZEPLER, 1973)



tic energy is radiated, is also important in noise evaluation and control, and is mathematically related to the decibel level. Many industrial noises have discrete frequency spectra, which are functions of the distance from the source. However, explosions generate noise of the explosive-impact type, which has a greater density of high frequency noise.

Current noise level regulations are in terms of the "A" scale or octave band noise levels. This is usually measured with octave band analysis or sound level meters. These instruments are designed to measure the weighted overall sound pressure level relative to a standard reference pressure of 0.002 dynes per square contameter. Precision meters have three weighting networks; the A scale corresponds most closely to the frequency vs amplitude response of the human ear.

Noise exposure standard have been established for both general industry and the mining industry (MESA, 1972; OSHA, 1976). Both federal codes use A scale measurements and set equal exposure limits (Table A-7).

TABLE A-7

PERMISSIBLE NOISE EXPOSURES

Duration Per Day (hours)	Sound Level dBA - Slow Response
8	90
6	92
4	95
3	97
2	100
1-1/2	102
1	105
1/2	110
1/4 or less	115

The "slow response" indicated above refers to the measuring equipment.

Personal hearing protection apparatus must be provided when noise levels exceed those given in Table A-7and must limit the sound level to permissible values. Where daily exposure is composed of two or more periods at different intensity levels, their combined effect must conform to equation :

$$\frac{C_1}{T_1} + \frac{C_2}{T_2} + \ . \ . \ . + \frac{C_n}{T_n} < 1.0 \qquad \text{(A.4)}$$

where

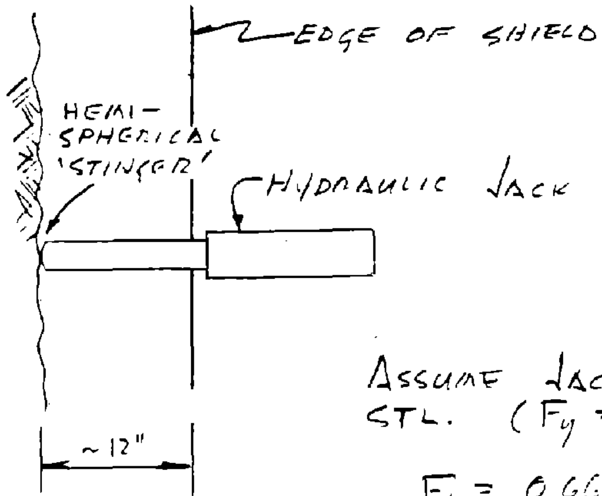
C = sound level dBA, slow response

T = time of exposure in hours

The Mining Enforcement and Safety Administration (MESA) regulations limit exposure to 115 dBA. For impact noises, 140 dBA is the maximum allowable.

APPENDIX B
BLAST SHIELD DESIGN DETAILS

N) CONT. APPROX. GEOMETRY:



ASSUME JACK CYLINDERS ARE HIGH STR. STL. ($F_y = 50$ KSI):

$$F_b = 0.66\ F_y = 33.0 \text{ ksi}$$

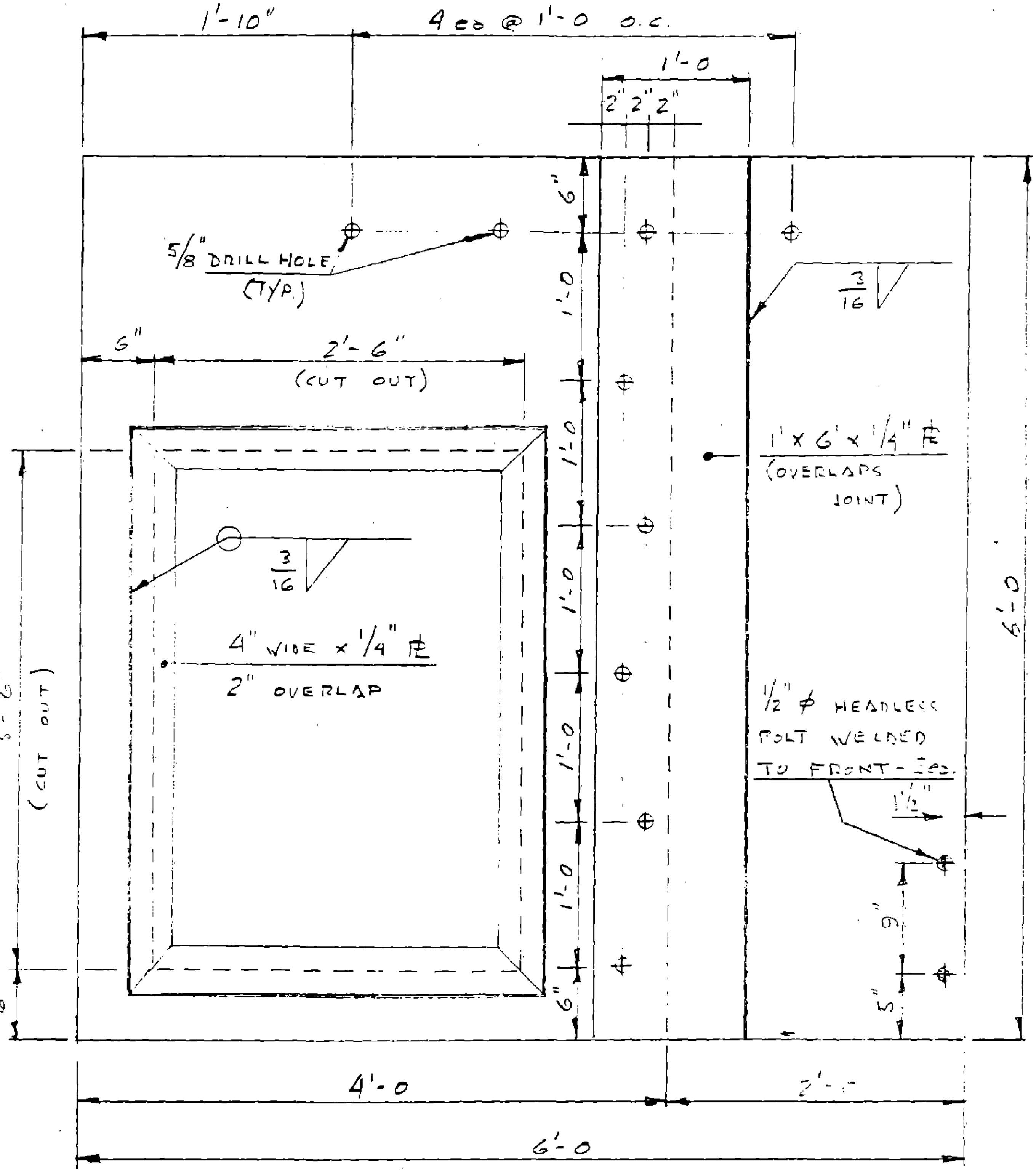
$$M = 4.25 \text{ k ft}$$

$$f_b = F_b = Mc/I = 4.25(12)\ \frac{c}{I} \implies \frac{I}{c} = 1.55$$

$$I = \pi d^4/64, \quad c = d/2$$

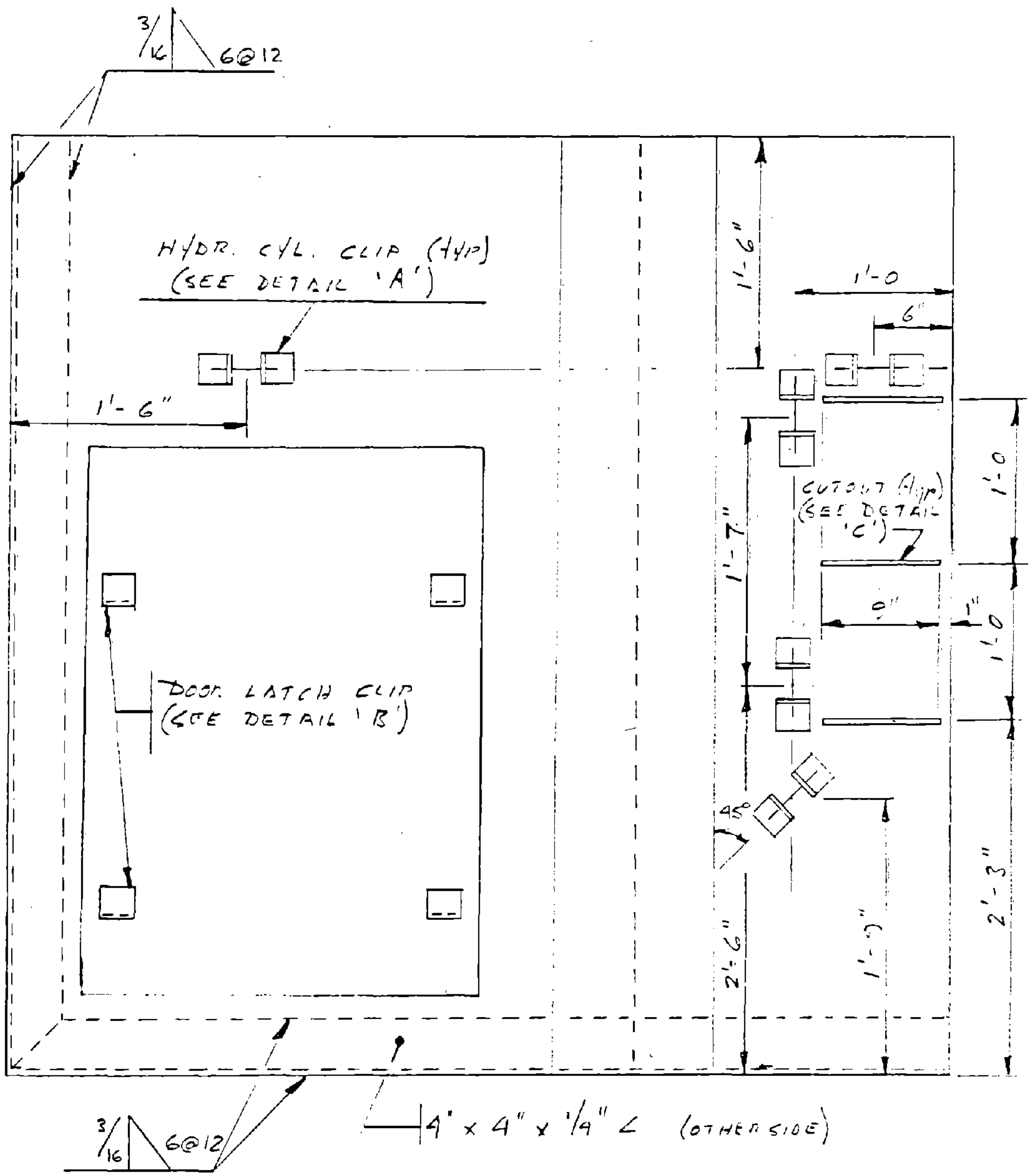
$$I/c = 2\pi d^4/64d = \pi d^3/32 = 1.55 \implies d = 2.5''$$

A) MAIN SHIELD SECTION - FRONT SIDE (OUTBY)



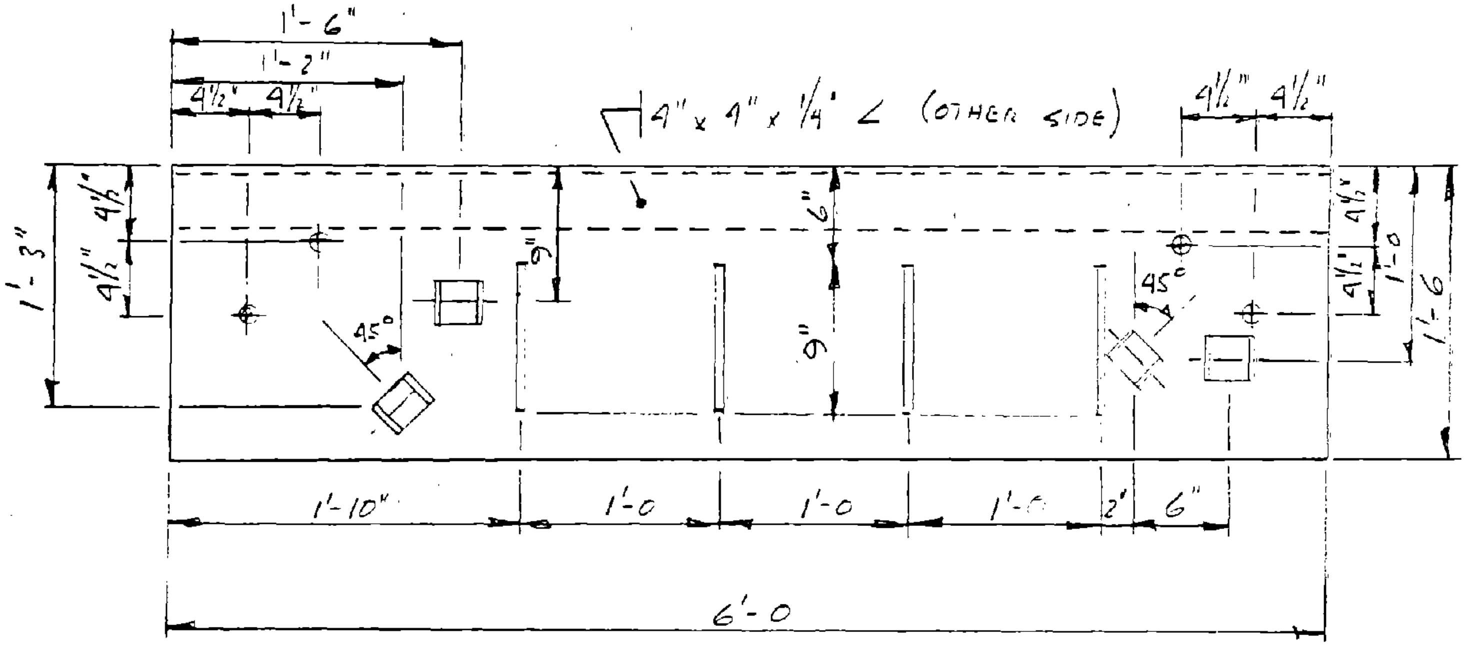
SCALE: 1" = 1'-0

A) MAIN SHIELD SECTION - FRONT SIDE
CON'T.



SCALE: 1"=1'-0

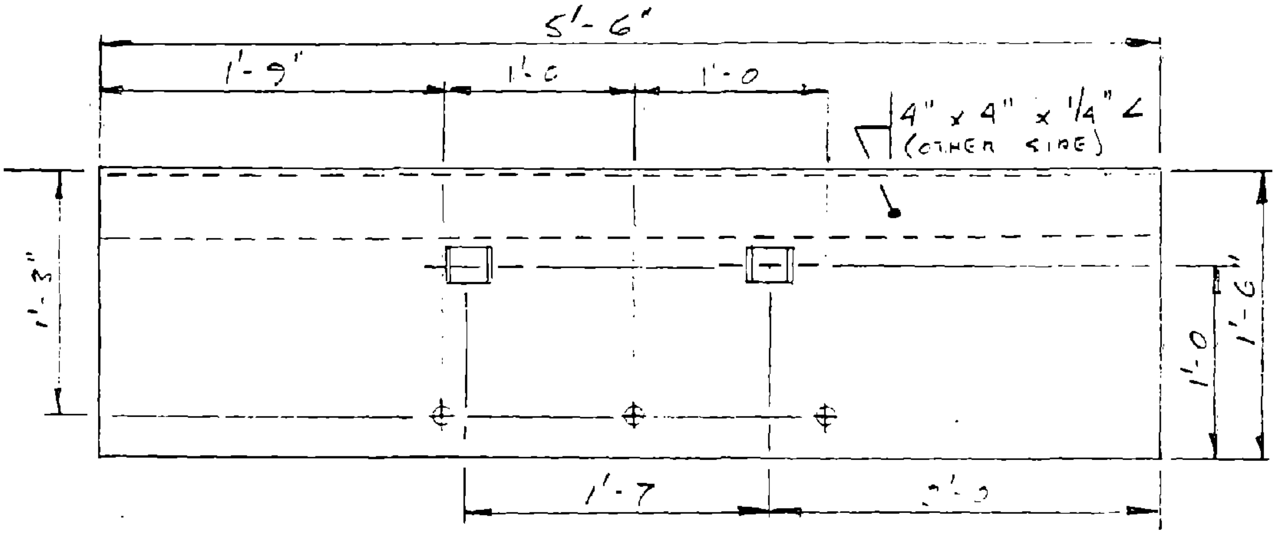
B) TOP SHIELD SECTION - FRONT SIDE



NOTE: ABOVE 4 ea HOLES ½" ϕ HEADLESS
WELDED TO FRONT; SEE DETAIL 'C' FOR SLOTS

SCALE: 1" = 1'-0

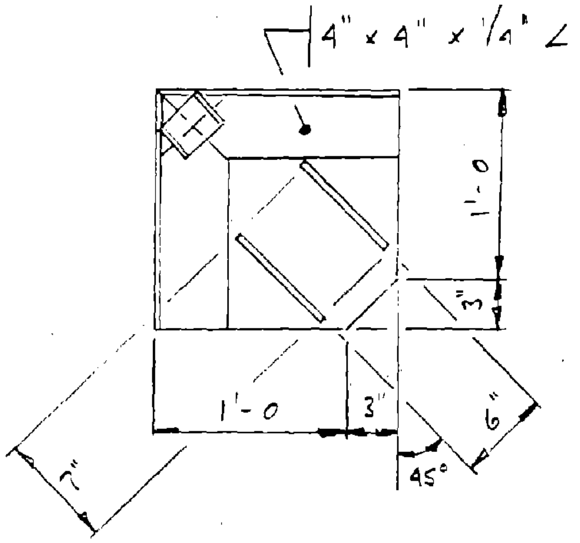
c) RIGHT SHIELD SECTION - FRONT SIDE



NOTE: 3 ea - D.H.'s 5/8" φ w/
1/2" φ BOLT HEAD WELDED
TO OTHER SIDE

SCALE: 1" = 1'-0

D) TOP LEFT CORNER SECTION - FRONT SIDE



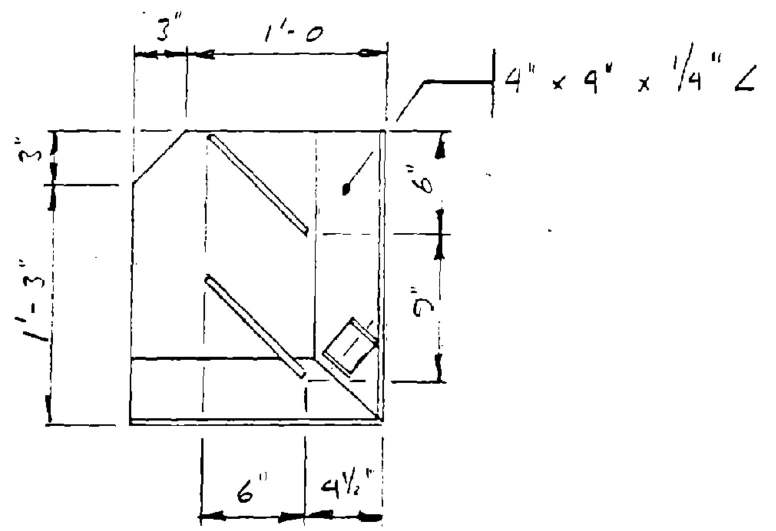
SCALE: 1" = 1'-0

NOTE: SEE DETAIL 'C' FOR SLOTS

E) TOP RIGHT CORNER SECTION - FRONT SIDE

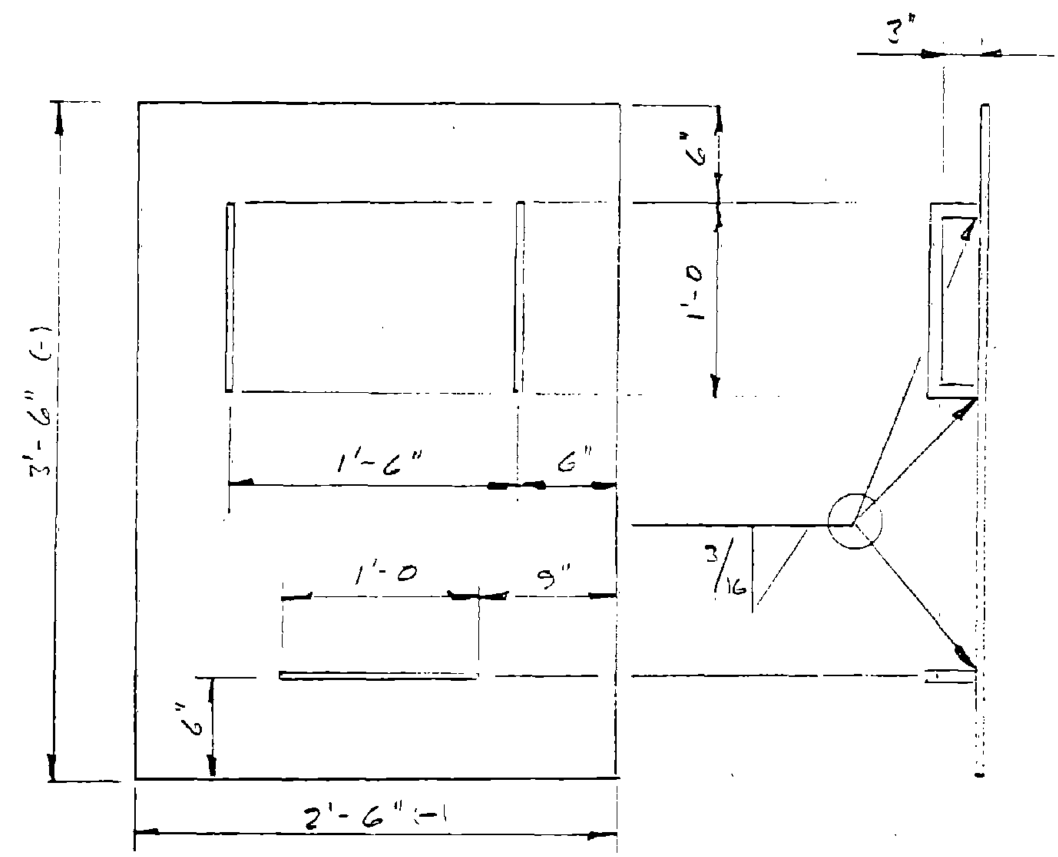
SAME AS ABOVE

F) BOTTOM RIGHT CORNER SECTION - FRONT SIDE



SCALE: 1" = 1'-0

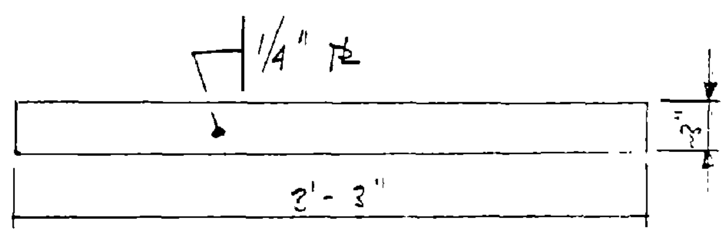
G) ACCESS DOOR - FRONT SIDE



SCALE: 1" = 1'-0

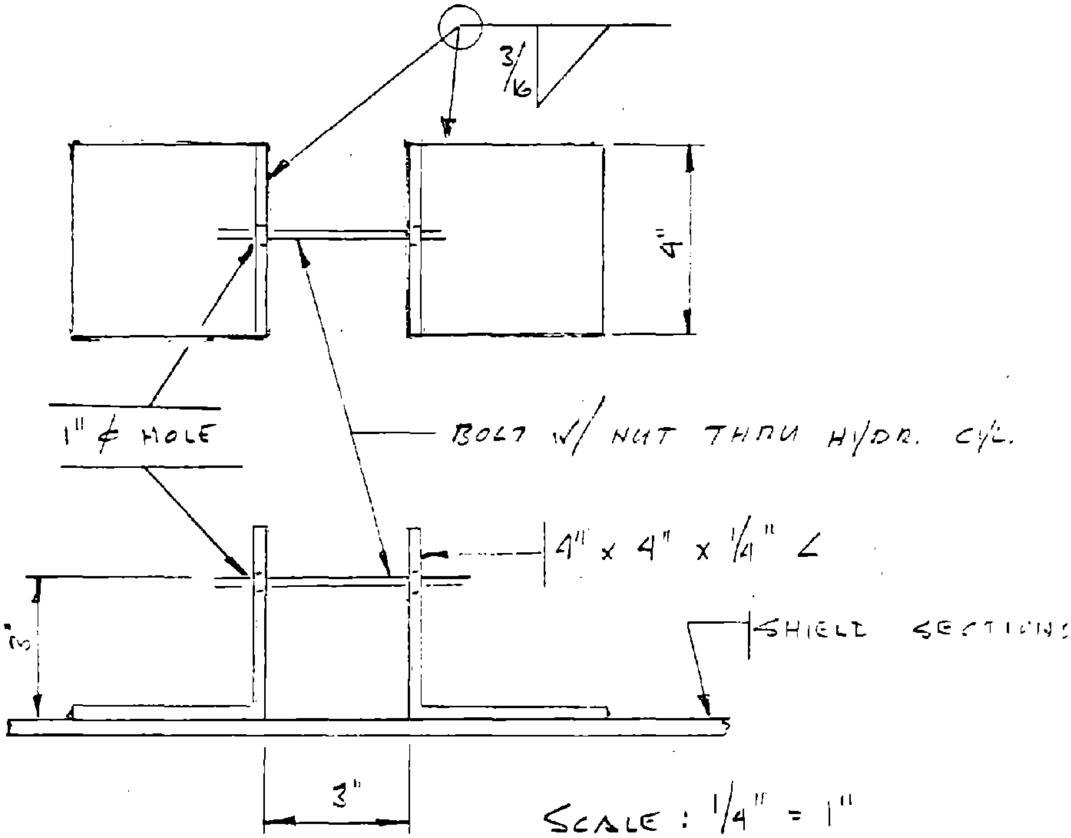
NOTE: FAB. LIFT HANDLES FROM #4 REBAR

H) ACCESS DOOR LATCH - 2 ea



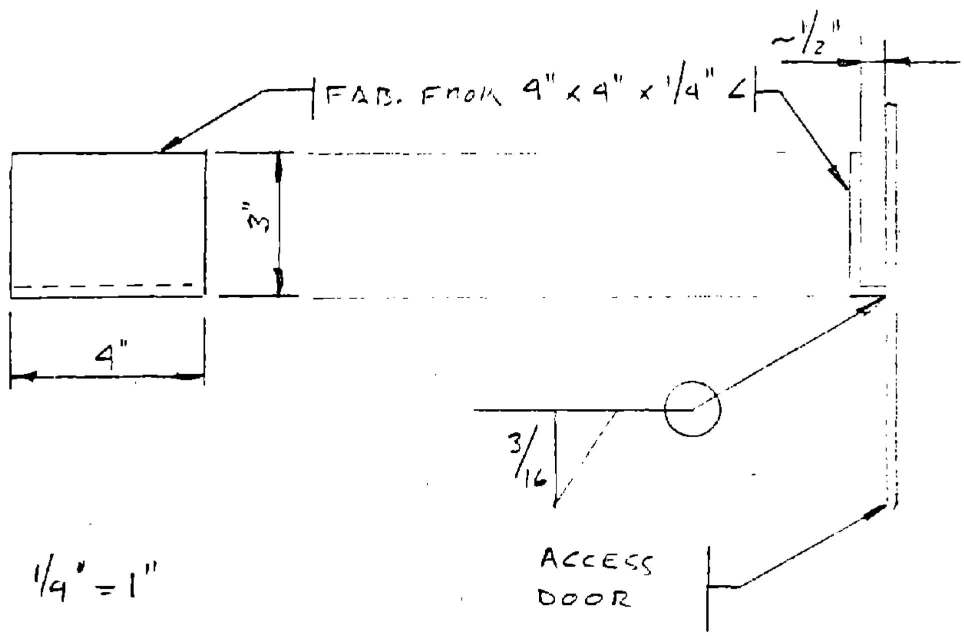
SCALE: 1" = 1'-0

I) DETAIL 'A'



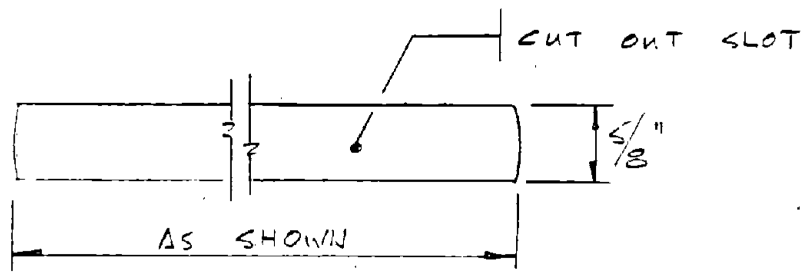
SCALE : 1/4" = 1"

J) DETAIL 'B'



SCALE : 1/4" = 1"

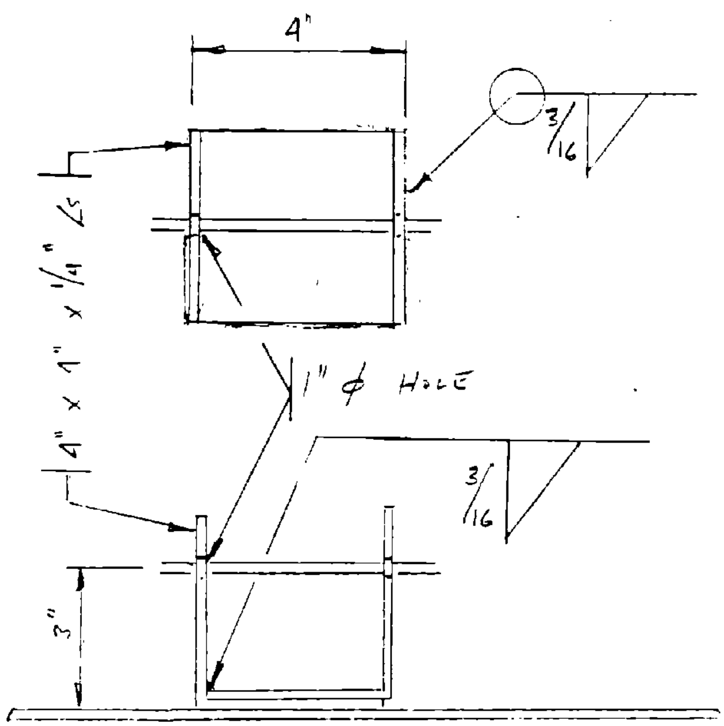
k) DETAIL 'C'



SCALE: AS SHOWN

NOTE: GRIND CUT SURFACE TO REMOVE BURRS

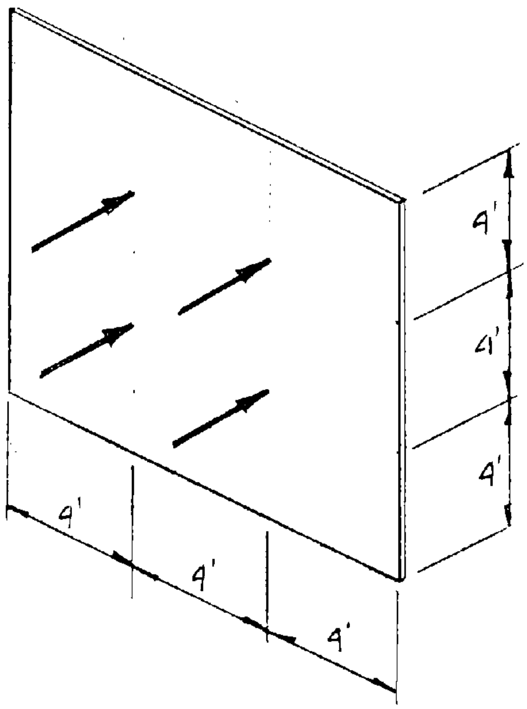
l) DETAIL 'D'



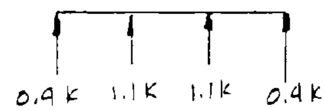
SCALE: 1/4" = 1'-0

NOTE: HYDRAULIC CYLINDER CLIPS MAY
ALL BE FAB. PER DETAIL 'D'

M) SHIELD TO CHASSIS CONNECTION



PER AISC, LOAD FOR 3 EQUAL SPANS IS:



∴ CENTER SUPPORTS CARRY

$2(1.1) / [2(1.1) + 2(0.4)] = 0.73$
OR ~70%

TOTAL LOAD =

$10.5\,psi\,(12^2)^2 / 1000 = 218\,K$

LOAD / SUPPORT $= 218(0.7)/4$
$= 38.2\,K$

APPROX. GEOMETRY OF CHASSIS & SHIELD:

THUS, THE MAX MOMENT IN 'A' IS:

$M_{(A)} = 2\,(38.2) = 76.4\ k\cdot ft$

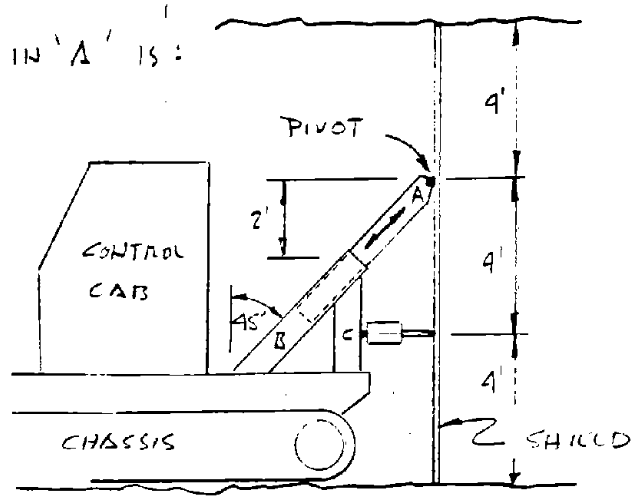
F_b (46 ksi) $= 0.67\,F_y$

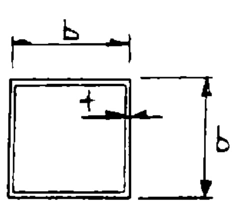
$= 30.7$ ksi

$f_b = F_b = M_{(A)}\,c / I$

$30.7 = 76.4(12)\,c / I$

$I/c = 29.9 = S$





FOR $t = 1/4''$, USE

<u>10 × 10 × 1/4</u> STRUCTURAL TUBING (AISC p 1-103)

M) CONT. HYDRAULIC JACK CONNECTION BOLTS:

F_v (50 ksi) = $0.4 F_y$ = 20 ksi

$f_v = P/2A$ where $f_v = F_v$, $P = 38.2$ K

$A = P/2F_v = 38.2/2(20\text{ ksi}) = 0.96 \text{ in}^2$

$.96 = \pi r^2 \Rightarrow r = 0.55$, $d = 1.1''$

USE $\underline{\underline{1\frac{1}{4}''}}$ ϕ H.S. BOLT

VERT. SUPPORT ('C'): USE $\underline{\underline{10 \times 10 \times \frac{1}{4}}}$ STR. TUBING

TILT-UP, PIVOT POINTS ON BOOM FOR BOLT SHEAR & TENSILE STRESS.

USE FULL PENETRATION (E70XX ELECTRODE) 1/4" FILLET WELDS (CONTINUOUS) FOR CONNECTIONS.

WEIGHT OF SHIELD

STL. PL:

$1.3(12)^2 = 190 \text{ ft}^2$	
$190 \times (\frac{1}{4}''/12) \times 490^\# =$	$1940^\#$
JACKS : 20 ea × $20^\#$/ea =	$400^\#$
LINING : $190 \text{ ft}^2 \times 0.5^\#/\text{ft}^2$	$100^\#$
MSC	$60^\#$
	$2500^\#$

∴ INSTALL STD 2-WAY, 10 ton JACKS FOR CHASSIS CONNECTION.

N) PERIPHERAL SUPPORT

JACKS MUST CARRY 17 tons; ASSUME 8 JACKS

LOAD / JACK = $17(2)/8 = 4.25^K$

APPENDIX C

DERIVATION OF EQUATIONS

A) Equation 10.18

For "Stratified or Schistose" Rock:

$H_p = 0.25B$, where B = Tunnel Width

w(assumed) = 160 pcf; Sets @ 4'-0" O.C.

$$\text{Load} = 4 \times w \times H_p = 4 \times 160 \times 0.25 \times B$$
$$= 160 \times B$$

Assume steel set is semi-circular with nominal diameter (B) equal to the tunnel diameter plus 1'-0":

Tunnel Diameter (final)	Set Diameter (nominal)	Set Load (lb/ft of width)
12	13	2080
16	17	2720
20	21	3360
24	25	4000

Design sets as wall plate rib (Proctor and White, 1968, Table 2, p. 240):

Tunnel Diameter (final)	Steel Set Based on Load	Weight of Set[1] (lb)
12	4 I 7.7	157
16	4 H 13.0	347
20	6 I 17.25	569
24	6 H 20.0	785

[1] Assumes semi-circular sets with a nominal set diameter as indicated above.

Set length = $1/2 \times \pi \times$ Dia (nominal)

Assume bolts, collar braces, foot plates, etc. add 10% to the set weight; steel cost = $0.50/lb (Commercial Shearing and Stamping, 1977):

Tunnel Diameter (final)	Total Steel Weight Per Set (lb)	Total Set Cost ($)	Steel Cost ($/LF)
12	174	87.00	21.75
16	382	191.00	47.75
20	626	313.00	78.25
24	864	432.00	108.00

Assume timbering adds 10% to the cost of the steel:

Tunnel Diameter (final)	Steel Cost ($/LF)	Total Set Cost[1] ($/LF)
12	21.75	25
16	47.75	55
20	78.25	85
24	108.00	120

[1] Rounded to nearest $5.00.

Thus, the cost equation is of the form

COST($/LF) = M x Diameter + B

Regression yields

COST($.LF) = 7.88 (DIAMETER) - 70

B) Equation 10.19

Since the concrete lining has the same nominal diameter and load-carrying requirements as the steel sets, the following is valid (see above):

Tunnel Diameter (final)	Concrete Arch Diameter (nominal)	Concrete Load (lb/ft of width)
12	13	2080
16	17	2720
20	21	3360
24	25	4000

Because the concrete support is an arch, design for compressive load; assume f'_c = 3500 psi. Because the loads are small (all less than 30 psi), assume the concrete thickness equals the depth of the steel section:

Preceding page blank

Arch diameter = nominal diameter + steel section depth

Assume the concrete is poured the full tunnel circumfrence:

Concrete volume = π x Arch diameter x Steel set depth x 1 LF / 27

Tunnel Diameter (final)	Arch Diameter (ft)	Concrete Volume (yd^3/LF)	Concrete Cost[1] ($/LF)
12	13.3	0.51	100
16	17.3	0.66	135
20	21.5	1.25	250
24	25.5	1.48	295

[1] Concrete cost (including grout) = $200/$yd^3$ (Tyson, 1977); rounded to nearest $5.00.

Regression yields

COST ($/LF) = 12.5 x Diameter

C) Equation 10.21

Shotcrete thickness = 3 in. Thus, the nominal tunnel diameter equals the tunnel diameter (final) + 0.5 ft. Assume chotcrete is applied above the spring line:

$$\text{Shotcrete Volume} = 1/2\left(\pi \times \left(\frac{DIA + 0.5}{2}\right)^2 - \pi \times \left(\frac{DIA}{2}\right)^2\right)$$

$$= \pi/8(DIA^2 + DIA + 0.25 - DIA^2)$$

$$= 0.39(DIA) + 0.10 \quad (ft^3/LF)$$

$$= 0.0145(DIA) + 0.0036 \quad (yd^3/LF)$$

Shotcrete cost is $50.00/$yd^3$ (Conspray, 1977):

SHOTCRETE COST ($/LF) = 0.73(DIAMETER) + 0.18

Rock bolts are installed @ 4'-0" O.C. or 16 ft^2 per bolt; bolts cost $2.20 per bolt; only 25% of the tunnel is bolted.

Bolt Cost ($/LF) = π x DIA x 1 LF x $2.20 x 25% / 16

BOLT COST ($/LF) = 0.11(DIAMETER)

SHOTCRETE AND ROCK BOLT COST ($/LF) = 0.84(DIAMETER) + 0.18

The slurries of the type used in this experimentation are sensitized with Al for use in small diameter boreholes. While they are used as a replacement for dynamites, they are much less sensitive and have other favorable characteristics. They resemble gelatin dynamites in texture and density, and have been produced in three grades of about 40, 60, and 80 percent strength by one company. Slurry technology permits extension of the grades above 100 percent equivalent dynamite strength and down to 40 percent.

These slurries have significant advantages over corresponding gelatin dynamites. They are non-toxic themselves, generate less toxic fumes, and while they are cap sensitive, they are much less hazardous than dynamites. In addition, the air-gap sensitiveness need be no greater than that required for satisfactory blasting performance. The critical diameter can be adjusted as small as 1/2 in. and the gap sensitivity in 1 1/4 in. charges to about 3 to 5 in. Their employment in place of dynamites requires careful adherence to instructions for use. The densities may be varied from 1.0 to about 1.15 g/cc without chemical aeration.

Thus, the advantages of such slurries are, (1) high strength/cost ratios, (2) safety, (3) good water resistance, (4) non-toxicity, and (5) cap sensitivity without the hazards of dynamites. They may be formulated to yield effectively no toxic fumes, i.e., they have very good fume qualities even with moderate oxygen unbalance which gives a minimum of noxious fumes.

The Iremite slurries are formulated for use in specific borehole sizes for given rock temperatures, the sensitivity control being accomplished by varying the ratio of fine to total aluminum present.

The most sensitive grade is designed for use in charge diameters of 7/8 in. to 1 1/2 in. and at borehole temperatures above 15^{o} C. The sensitivity is regulated by means of the ratio of fine, flaked (paint grade) to spherical, granular aluminum. The total amount of aluminum and the "strength," or available energy per pound, is the same for the 60 and 80 grades, but only fine alumimum is employed in the 40 grade and its strength is less.

This type of slurry is manufactured from non-explosive ingredients in a continuous flow system in which the aqueous solution and solid ingredients, both non-explosive, are fed with quantitative accuracy into a small mixing funnel and pumped with an air-operated diaphragm pump into polyethylene tubing. The lengths of tubing are then tipper-tied in the desired lengths. A fine spray of water is employed in the cutting and tying operation to maintain cleanliness and to wash away excess slurry. The safety features of this process are:

1. The most sensitive state is at the end of the mixing cycle when the slurry is mixed and partly thickened and the slurry is entering the tubing.
2. Until the solids come in contact with the liquid phase, there is no explosion hazard except for possible dusts and vapors which must be carefully controlled.
3. The aqueous solution is self-quenching in a fire, and the redox mixture formed after the solids and aqueous solution come together does not burn unless most of the water is lost by evaporation.
4. There is only a small amount of explosive present in the flow cycle at a given time if appropriate steps are taken to

remove the tied tubes far enough from the flow system that an explosion cannot propagate to the mixed explosive.

5. The finished product, although it is cap sensitive, is orders of magnitude safer than dynamite when exposed to heat, friction, and shock.

The critical diameter may be measured with the charges in a vertical position, initiated with a No. 8 cap in the top of the charge. Charges are usually contained in polyethylene tubing of 12 inch lengths. One method prescribes that d_c be determined to the nearest 1/4 inch for charges of 1 1/2 inch diameter or less, and to the nearest 1/2 inch for charges greater than 1 1/2 inches in diameter. The criterion for failure in the measurement of d_c is the existence of residue of undetonated explosive.

The detonation velocity vs. charge diameter curves are obtained from measurements of the velocity with at least 1 inch of the charge beyond the last point of measurement. Also, a length of at least 6 inches, or three charge diameters, whichever is the greatest, is required between the No. 8 initiator and the first measuring point.

<u>Bullet sensitivity</u> for this type of explosive is measured by means of a 22-50 caliber rifle, 48-grain jacketed bullet with a maximum muzzle velocity of 3,800 feet per second, with rifle being fired 100 feet from the sample. The velocity is decreased in 200 feet per second intervals until 50% initiations are obtained, initiation being witnessed with a 1/2 inch steel plate, although the sound is also a suitable criterion of go/no-go.

The <u>air-gap sensitiveness</u> at a specified temperature is obtained by measuring the maximum air gap for detonation by influence utilizing

4 inch lengths of the explosive for both the donor and the receptor. The witness is a one foot length of 20-50 grain per foot detonating cord inserted 1/2 inch into the far end of the receptor. The up and down method is used to determine the maximum gap for three successive detonations.

The thermal stability of cap sensitive slurries is determined by differential thermal analysis using samples greater than 1/2 gram. The requirement is that the explosive will not exhibit an exotherm below 130° C. and preferably not below 150° C.

The usual requirement for thermal stability is that an explosive will not show a significant weight loss when held at a temperature of 75° for 48 hours. However, this is not practical for slurries because they lose weight slowly by evaporation of water at 75° C. A modified test utilizes the criterion that there is no appreciable generation of permanent gases, i.e., those which will not condense at this temperature.

Slurries may be stabilized so that there is no appreciable reaction of aluminum with water in prolonged storage at 40° C. It has been found that one $AN/Al/H_2O$ slurry has a shelf life of at least one year.

One method of measuring the comparative strength, or the so-called "seismic strength" (Cook, 1970) consists of detonating one-gallon charges of explosive with a 380 gram booster in a large pond at a fixed depth and at a fixed distance from a seismograph on the ground near the pond. The relative strength is determined by comparison with amplitude vs. charge weight values for ANFO detonated under the same conditions. An increase of the percentage of aluminum from 3.5% to

10% increases both the weight and the bulk strength, and the temperature as well as the heat of explosion. The critical diameter of all three grades, the bullet sensitivity, and the thermal stability are identical. These properties are of particular importance for application in automated drilling and blasting because of the need for a low value of precariousness of the explosive, and the inertness of the ingredients.

Samples of IREMITE 40, 60, and 80 were subjected to fume tests by the Bureau of Mines (Cook, 1974). In the first series of tests, the OB was adjusted to zero for the slurry alone, and about -2% with polyethylene tubing. The test results (Table D-1) indicated that the tubing reacted to give a negative OB, while in a second series of tests, the OB design included the tubing and the resulting fumes were very low. The theory of the generation of fumes for Bichel gage, C-J, and borehole conditions is discussed by Cook (1974) in detail. He shows that the method of calculating explosion products utilizing a modified Abel equation of state, shock wave equations, and related thermodynamic equations gives very good approximations of the test results for amounts of gases in the detonation products (Table D-2 & D-3).

Table D-1- Properties of SHE Cap Sensitive Slurries (Cook, 1978)

	A	B	C
Inorganic nitrate	70	68	66
Total water	15	15	14
Total äluminum	3.5	7.0	10
Density (g/cc)*	1.20	1.24	1.29
Detonation velocity (m/sec)*			
1.25" diameter	3450	3550	3610
1.50" diameter	3620	3630	3670
2.00" diameter	3760	3800	3810
2.50" diameter	3940	3980	4010
"Sensitiveness"* in 1 1/2" diameter	3"	3"	3"
Critical diameter*	≤0.75"(S) ≤1.25"(L)	≤0.75"(S) ≤1.25"(L)	≤0.75"(S) ≤1.25"(L)
Minimum cap for initiation*	#4 or #5	#4 or #5	#4 or #5
Bullet sensitivity* (48 grain bullet at 3800 feet per second)	F	F	F
"Strength" (ANFO - 1.0)			
a. Weight	0.83	0.94	1.05
b. Bulk	1.25	1.46	1.70
Thermal stability			
DTA	+	+	+
48 hours at 75° C.	≠	≠	≠
Calculated properties			
Mols of gas per kg	35	33	28
Explosion temperature ($^{\circ}$ K.)	2500	2750	3000
Q [Heat of explosion (kcal/g)]	750	870	1040
A [Maximum available energy (kcal/g)]	690	780	870

* Measured at temperature of application.
\+ No exotherm below 130° C.
≠ No measurable permanent gas generation in 48 hours at 75° F.

Note: Tests are in general conducted on samples at least one day old.

Cook, M. H. (1978), Personal Communication.

Table D-3 - Data from Bureau of Mines Reports on IREMITE Fume Studies

Data	IREMITE	CO (l/kg)	Oxides of Nitrogen (l/kg)	CO + 6.5 Oxides of Nitrogen (l/kg)
A. September 16, 1970[a]	IREMITE 40			
	Bichel gauge	5	None	5
	Crawshaw-Jones gauge	11.2	1.2	19
	IREMITE 60			
	B gauge	14(1.2)[b]	None	14(1.2)
	C-J gauge	11.2(2.4)	0.2(0.5)	12.5(14)
	IREMITE 80			
	B gauge	10.5	None	10.5
	C-J gauge	8.1	<0.06	<12
B. June 8, 1971	IREMITE 40			
	B gauge	None	None	None
	C-J gauge	3.1	2.2	17.1
	IREMITE 60			
	B gauge	None	None	None
	C-J gauge	3.1	0.7	7.6

a - The IREMITES tested by the Bureau of Mines and reported on September 16, 1970, were all slightly negative in oxygen balance, whereas those reported on June 8, 1971, were oxygen balanced at zero (including polyethylene).

b - Results on second (replication) data sheet.

Table D-4- Gaseous Products Obtained from IREMITE (Cook, 1978) in the Bichel Gage and Crawshaw-Jones Apparatus.

Test Method	Bichel Gage	Crawshaw-Jones
Gaseous products, percent:		
CO_2	30.6	32.9
C_mH_n	--	0.2
H_2	--	1.1
CO	--	1.0
CH_4	--	0.2
N_2	66.5	63.8
NO_2	--	0.2
N_2O	--	258 ppm
O_2	2.9	0.6
Gaseous products per pound of explosives, cu ft1/:		
CO_2	1.39	1.50
C_mH_n	--	0.01
H_2	--	0.05
CO	--	0.05
CH_4	--	0.01
N_2	3.02	2.91
NO_2	--	0.01
N_2O	--	0.001
O_2	0.13	0.03
Total poisonous gases, cu ft/lb2/	none	0.06
Cu ft/200 grams	none	0.03
IME-Fume Class	1	1

1/ At 760 mm Hg and 0^{o} C.
2/ Includes CO and NO_2.

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