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Surface Mine Blasting

Proceedings: Bureau of Mines Technology Transfer
Seminar, Chicago, IL, April 15, 1987

Compiled by Staff, Bureau of Mines



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UNIT OF MEASURE ABBREVIATIONS USED IN THESE PAPERS

atm	atmosphere	(in/s)/in	inch per second per inch
°C	degree Celsius		
dB	decibel	K	kelvin
°F	degree Fahrenheit	kHz	kilohertz
ft	foot	lb	pound
ft ²	square foot	lbf/in ²	pound (force) per square inch
ft/lb ^{1/2}	foot per square root pound (scaled distance)	m	meter
ft/lb ^{1/3}	foot per cube root pound (scaled distance)	mg/L	milligram per liter
ft/s	foot per second	mi/h	mile per hour
G	gravity (32.2 ft/s ²)	min	minute
g	gram	μin/in	microinch per inch
gal/d	gallon per day	mm	millimeter
gal/min	gallon per minute	ms	millisecond
(gal/min)/ft	gallon per minute per foot (specific well capacity)	ms/ft	millisecond per foot
g/cm ³	gram per cubic centimeter	μs	microsecond
h	hour	mV	millivolt
Hz	hertz	ns	nanosecond
in	inch	pct	percent
in ³	cubic inch	s	second
in/s	inch per second	W	watt
in ³ /s	cubic inch per second	yr	year

SURFACE MINE BLASTING

**Proceedings: Bureau of Mines Technology Transfer Seminar,
Chicago, IL, April 15, 1987**

Compiled by Staff, Bureau of Mines

ABSTRACT

The Bureau of Mines has sponsored a comprehensive research program to enhance the safe, effective, and efficient use of blasting technology by the mining industry. Recent research results of the surface mine blasting program were presented at a seminar on April 15, 1987, in Chicago, IL. Many of the topics discussed at the seminar are presented in this proceedings. The new research described in these papers includes computer monitoring of rock motion, the influence of blast delay times on rock fragmentation, blasting effects on Appalachian water wells, blast vibration measurement near structures, and the reduction of accidents through improved blasting safety.

BUREAU OF MINES SURFACE MINE BLASTING RESEARCH

By Dennis V. D'Andrea¹

ABSTRACT

The Bureau of Mines Twin Cities Research Center has a comprehensive research program on the efficient and safe application of explosives in mining. Researchers combine an understanding of the basic principles of dynamic rock fragmentation with new blast design technology and recent developments in both methods

and equipment, for potential improvements in blasting practices. This paper outlines surface mine blasting research completed since the Bureau's last Technology Transfer seminar on blasting in December 1980. Three programmatic areas--productivity technology, blasting vibrations, and blasting safety--are reviewed.

HISTORY OF RESEARCH

Blasting research has been conducted at the Twin Cities Research Center (TCRC) since the center opened in 1959. Research during the 1960's and early 1970's established TCRC as the leading Bureau of Mines center in the area of blasting for improved fragmentation and increased productivity. During the period from fiscal year 1974 through fiscal year 1979, productivity research was on blasting to prepare ore bodies for in situ leaching. The major effort at TCRC from fiscal year 1975 through fiscal year 1983 was in the area of environmental effects of blasting (ground vibrations and airblast). Research since fiscal year 1983 has been mostly on blasting fundamentals for improvements in productivity. Blasting

safety research began at TCRC as one contract project in fiscal year 1978 and grew to involve four in-house projects during the years 1984 to 1986.

The heavily field-oriented blasting research program at TCRC has included 45 in-house and contract project efforts since 1975, resulting in 122 publications and numerous presentations at professional meetings. Report of Investigations (RI) 8507, on structural response and damage from blasting vibrations, won the 1981 Applied Research Award from the U.S. National Committee for Rock Mechanics. TCRC personnel have responded to over 650 requests for technical assistance and advice on blasting since 1981.

CURRENT RESEARCH PROGRAM

Mining Technology

Major research efforts at TCRC on improved productivity and blasting vibrations in surface mines are listed in table 1. Included are projects intended to improve mining productivity and to provide information on good blasting practices. The projects that started in the late 1970's addressed environmental issues, with indirect implications for mining costs and productivity. More recent long-range, high-risk research is

examining the fundamentals of blasting and blast-produced rock fragmentation. The fundamental research includes a study of high-precision delay initiators to improve fragmentation.

The Bureau is frequently asked to assist other Federal agencies, such as the Office of Surface Mining, the Bureau of Reclamation, and the Mine Safety and Health Administration, on environmental and safety issues associated with blasting practices. These assistance efforts usually do not involve pure research, but rather providing technical information and, on occasion, measurement, analysis, and advice.

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TABLE 1. - Blasting research at TCRC on improved productivity and blasting vibrations

Research area	Fiscal years	Key researchers	Significant publications ¹
Blasting effects on Appalachian water wells.	1978-80	P. R. Berger and Associates. ²	D. Robertson (1).
Contour mine blasting noise and vibrations.	1978-81	V. J. Stachura.....	Stachura, RI 8892 (2).
Standards efforts on vibrations, ANSI and ISO.	1980-81	D. E. Siskind, Acoustical Society of America.	ANSI Standard S3.29-1983 (3). Other ANSI and ISO Standards.
Fatigue from repeated blasting.	1979-83	M. S. Stagg, National Bureau of Standards. ²	Stagg, RI 8896 (4). Siskind (5). Woodward (6).
Vibration measurement near buildings.	1983	D. E. Siskind.....	Siskind, RI 8969 (7).
Blast designs to control vibrations.	1979-83	J. W. Kopp.....	Kopp, RI 9026 (8).
Blasting fragmentation fundamentals.	1984- (ongoing)	M. S. Stagg, Sandia Laboratories, ² University of Maryland. ²	Nutting (9).
Low-frequency vibrations.	1985- (ongoing)	D. E. Siskind, V. J. Stachura.	In preparation.

ANSI American National Standards Institute.

ISO International Standards Organization.

¹Only the senior author is listed here. Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

²Work conducted for the Bureau under contract.

Projects listed in table 1 range from minor efforts (such as providing an improved technology basis for the American National Standards Institute and the International Standards Organization to use in establishing standards) to major multiphase projects involving up to five supporting industry and Government service contracts (such as the fatigue study). The research on low-frequency vibrations is a TCRC technical assistance effort for the Office of Surface Mining on blasting vibrations above abandoned underground mine workings. *BLANDFORD*

Blasting Safety

Research on blasting safety at TCRC is concerned with safer blasting practices and blast designs primarily for surface mines and underground metal-nonmetal mines. The Bureau also conducts explosives research at the Pittsburgh Research

Center, which focuses on the safe use and evaluation of permissible explosives and permissible blasting methods for underground coal mines, and research on the properties and explosives chemistry related to safe explosive performance, storage, and transportation.

Table 2 summarizes the major blasting projects at TCRC in the Bureau's Health and Safety Program. Initially, all work was contracted out. More recently, the TCRC has conducted in-house research on a wide range of significant problems related to safer blasting procedures. Examples of this in-house research are blast designs for safer and more stable highwalls and the development of materials for blasters training.

Research on blasting safety is guided by the analysis of blasting accident statistics. This analysis now covers 8 yr, and has determined the most frequent causes of blasting accidents and

TABLE 2. - Research at TCRC since 1980 on blasting safety

Research area	Fiscal years	Key researchers	Significant publications ¹
Certification of blasters	1978	E. I. DuPont de Nemours & Co. ²	Coulson and Southall (10).
Blasting manual.....	1980-81	R. A. Dick.....	Dick, IC 8925 (11).
Misfires.....	1982	L. R. Fletcher.....	Fletcher (12).
Blast area security.....	1981-84	Mining & Marketing Associates. ²	Bennett (13-14).
Highwall stability.....	1983-84	V. J. Stachura, L. R. Fletcher.	Stachura, RI 8916 (15), RI 9008 (16).
Blasters training.....	1984-86	L. R. Fletcher and M. A. Peltier.	In preparation.
Flyrock control.....	1986	L. R. Fletcher.....	Fletcher (17).
Blasting accident analysis.	1981- (ongoing)	D. V. D'Andrea.....	D'Andrea (18). Peltier (19).

¹Only the senior author is listed here. Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

²Work conducted for the Bureau under contract.

identified where the most hazardous situations exist. Through this analysis, researchers have been able to determine the most critical industry needs and address

these needs through research efforts in areas such as blast area security, misfires, and flyrock control.

SUMMARY

The blasting research effort at TCRC is concerned with improved productivity, blasting vibrations, and blasting safety. The following papers summarize research projects on surface mine blasting that have been carried out and reported on since the Bureau last held a Technology Transfer seminar on blasting in December 1980.

Highlights of the TCRC research include evaluations and recommendations for vibration measurement methods near buildings and a comprehensive study of a test

house showing that low-level vibrations from repeated blasting did not damage structures. Blast design studies found that the best fragmentation was achieved when delays between blastholes were at least 1 ms/ft of burden. Highwall stability was improved using longer periods in the row of blastholes that formed the highwall. Accident analysis indicated that failure of the blast area security system is the major cause of blasting accidents.

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REDUCING ACCIDENTS THROUGH IMPROVED BLASTING SAFETY

By Larry R. Fletcher¹ and Dennis V. D'Andrea²

ABSTRACT

The Bureau of Mines investigated three of the major causes of mine blasting accidents: inadequate blast area security, excessive flyrock, and misfires.

Accidents resulting from inadequate blast area security occur during scheduled blasting because of failure to clear the blast zone, inadequate guarding, and failure of personnel to follow instructions, retreat to a safe location, and/or take adequate cover.

Excessive flyrock is produced when there is too much explosive energy for the amount of burden, stemming is inadequate, or the explosive energy is too rapidly vented through a zone of weakness in the rock. Geology, improper blast design, or carelessness can cause unwanted

flyrock. The operator must change blasting methods when shooting in geology that favors the production of flyrock.

Most misfire accidents are caused by drilling into bootlegs in underground metal and nonmetal mines. Improper disposing of misfires is the second most frequent cause of misfire accidents, and some accidents are due to impact initiation of explosives in the muckpile. Misfires are usually caused by misunderstanding, improper use, or some failure of the initiation system. Other causes are cutoffs, insufficient firing current, inadequate priming, improper explosive storage, and damage to the initiation system.

INTRODUCTION

Analysis of mine blasting accidents shows that the five most frequent causes of accidents are premature blasts, inadequate blast area security, excessive flyrock, misfires, and fumes. Over 70 pct of all blasting accident injuries from 1978 through 1985 have been caused by three of these: inadequate blast area security, flyrock, and misfires, all of which are discussed in this paper, based on earlier Bureau of Mines research (1-3).³

Each mining operation has a normal flyrock range, the distance from the blast at which flyrock can be expected, based on blasting experience at that operation. The distance of the normal flyrock range will vary from a few feet in an area strip coal mine blast to more than a mile

with poorly controlled shots. A safety factor is added to the normal flyrock range to determine the blast area to be cleared and secured before the blast is detonated. Rock that travels beyond the secured blast area is excessive flyrock (fig. 1). A distinction is made between injuries that occur within the established blast area and injuries that are the result of excessive flyrock projected beyond the blast area.

Anyone who remains in the blast area, such as the shot firer, must have adequate protection from flyrock such as that provided inside a blast shelter. The shot firer is frequently guilty of

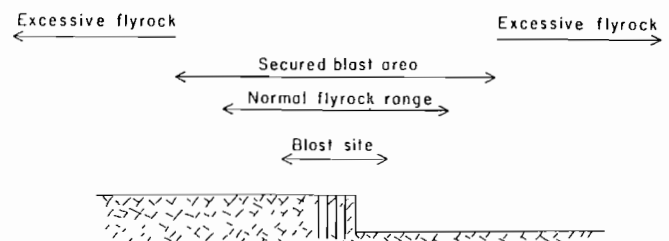


FIGURE 1.—Blast site, normal flyrock range, secured blast area, and excessive flyrock region for surface mine blast.

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

shooting and observing blasts from within the blast area without adequate protection. When a shot firer pays inadequate attention to the hazards of flyrock, a permissive attitude toward guarding and protecting the blast area is created.

A misfire results when explosives fail to detonate as planned during a mine blast. It is difficult, if not impossible, to determine how frequently misfires occur. When operators are asked, they usually reply, "Rarely, if ever, do we have a misfire." The truth is, however, that misfires are fairly common, but

because many people feel that misfires reflect the quality of their work, they are reluctant to report them. In addition, the increased use of non-cap-sensitive blasting agents with lower shock sensitivity has generated an attitude of indifference about misfires at some operations. Because of these two factors, it is rare that a misfire that does not involve an injury is ever reported.

This paper discusses the elements of effective blast guarding, the causes and control of flyrock, and the causes, detection, and disposal of misfires.

BLAST AREA SECURITY

The blast area security system is the means by which a mine operator prevents injury to people or damage to equipment when a scheduled blast is detonated. Most blasting accidents in surface and underground mines occur during scheduled blasting and are due to inadequate blast-site security. The result is often unnecessary injury and/or death, caused, in most cases, by flying rock. Mine personnel, visitors, and even trespassers can inadvertently wander into a blasting zone unless proper procedures exist to ensure that all personnel are cleared from the blast zone and kept safely away until after blasting is completed.

GENERAL REQUIREMENTS FOR BLAST GUARDING

The basic requirements of a blast-site security system at either a surface or underground mine are (1) to move personnel and equipment out of range of flyrock from the blast and (2) to prevent movement of personnel back into the blast zone. This includes visitors and trespassers, along with mine personnel.

The most effective procedures for accomplishing blast-site security objectives are to--

Have blasting personnel physically clear the blast-affected zone;

Account for personnel, to establish that no one is present in the blast zone at blast time; and

Place guards beyond the blast zone, at entries leading into the blast site, to

keep personnel from moving into the blast zone.

Elements contributing to the accomplishment of the blast-site security system requirements are as follows:

- Management commitment to safety.
- Training.
- Definition of blasting authority.
- Planning.
- Blast zone boundaries.
- Selection and placement of guards.
- Clearing procedure.
- Location of blast initiation site.
- Blasting time.
- Blast signals.
- Personnel accounting.
- Communication of blast location and time.
- Blasting crew communications.

Each of these elements is discussed below.

MANAGEMENT COMMITMENT TO SAFETY

The mines that have developed the best blast-guarding systems appear to be those in which management has made a definite commitment to safety. It has been recognized at those mines that it is management's responsibility to provide mining personnel with a systematic, safe blast-guarding method and to provide sufficient training to allow mining personnel to operate intelligently within the established system.

Blast-site security objectives are generally accomplished through the development of a blast-guarding standard operating procedure specifically designed to fit the requirements of a given mine.

TRAINING

Having developed a standard blast-guarding procedure, mine management is obligated to inform its employees of their responsibilities. Operations with good blasting security systems utilize information regarding proper blast guarding as training material for new hires and for retraining experienced miners. It is important that blast guarding be treated specifically and in detail as a real safety problem.

DEFINITION OF BLASTING AUTHORITY

One experienced individual must be responsible for the blast. Whether this person is a blasting superintendent, foreman, or crew leader, it is extremely important that one key individual direct the blast-guarding process for any given blast.

PLANNING

Preblast planning is an important aspect of blast guarding. Mines that have standardized their blast-guarding methods reduce the requirement for daily planning by making certain blast-associated decisions routine. This is particularly true at mines that have adopted central blasting systems and consistent methods for identifying crew member locations before a blast.

BLAST ZONE BOUNDARIES

The boundary around the blast beyond which personnel and equipment will be safe from flyrock must be determined. Guard posts must be located outside that boundary at all entries into the blast zone. Flyrock range, of course, is dependent on a number of factors including blast configuration, rock hardness, stemming, geology, etc., and is determined for each particular location and blast.

SELECTION AND PLACEMENT OF GUARDS

It is the general consensus throughout the industry that human guards should be used to guard blast-site entries whenever possible. Entrances to blast areas are sometimes left unguarded because it is either impractical or unsafe to place a person on post. When an entry is unguarded, a barricade or sign is used to warn people, but these methods are generally considered inadequate.

Persons selected as guards should be sufficiently trained to understand the serious responsibility they have to prevent any movement of personnel back into the blast zone. The absolute authority of the guard to prevent movement beyond the post must be established. The guards themselves must be placed in a location safe from flyrock and fumes.

CLEARING PROCEDURE

Mine management should establish a definite clearing procedure as part of the blast-guarding standard operating procedure. The clearing path will depend on mine layout, and considerable variance can exist within a given mine. Clearing personnel in surface mines are assigned to drive or walk through the blast zone prior to the blast. In mines where low-mobility equipment such as drills and shovels are left in the blast zone, this equipment should be inspected to be sure all personnel have left.

Clearing procedures in underground mines vary considerably. Since blasters frequently finish loading underground rounds well before the scheduled shift-change blast time, last-minute clearing must be done. If effective clearing prior to blasting is impractical, it is essential to use personnel accounting and keep miners informed of the time and location of the blast. Blast-guarding systems generally benefit from the inherent redundancies associated with requiring both clearing and personnel accounting in the blast zone.

LOCATION OF BLAST INITIATION SITE

A definite initiation site (blasting location) should be designated for all blasting areas in the mine or pit. Location of the blast initiation site should be based on a conscious management decision as part of the preblast planning process.

The purpose of designating location is twofold. First, it provides management with a means of fulfilling its responsibility to ensure that the blaster retreats to a safe location before blasting. The recent Bureau analysis of blasting accidents revealed that many surface and underground accidents were caused by unsafe location of the shot firer. Second, the location of a designated blast initiation station can easily be communicated to all mine personnel as the required checkpoint for personnel accounting when persons are entering or leaving that area of the mine or pit.

Minimum recommended requirements for a blasting site are as follows:

The site should be located outside the blast zone, protected from flyrock and upwind of blast fumes. Blast initiation from inside the blast zone should be permitted only if a secure blast shelter is used.

The site should be equipped with a highly visible sign giving the blast time. The sign should also notify visitors that they are required to check in and out at that station.

BLASTING TIME

Predictability is an essential element in a process with the destructive potential of blasting. Therefore, it is recommended that efforts be made to establish consistent blast times. Whether the blast is initiated at lunch time or the end of a shift, blast scheduling should be strictly controlled where possible. Timing for secondary blasting varies according to the needs of the particular mine. Many mines, both surface and underground, have developed systems in which secondary blasting is regularly

done concurrently with the production blast.

BLASTING SIGNALS

Audible signs are used at most surface mines to warn of the impending blast. In some cases, an air horn or electric siren is mounted at a fixed power source. In others, particularly when the pit is large, sirens or horns are mounted on vehicles used for clearing the blast zone. In either case, the devices used should be loud enough to be heard by personnel approaching the blast zone from any direction. In addition, consideration should be given to prevailing wind directions and velocities. Many surface mines use flashing lights on blast-associated vehicles as a supplement to sirens and horns.

The duration and pattern of audible signals used at the mines vary considerably. Some start 10 min prior to the blast, while others start 2 min before the blast. Some signals continue throughout the blast, while others are interrupted during the blast. A second signal after the blast signifies that all is clear.

The signal must fulfill three requirements, as follows:

It must be loud enough to warn personnel inside and near the zone of the impending blast.

In the event that the clearing process has failed and persons remain exposed to the impending blast, the signal should allow sufficient time for them to take cover or leave the area.

It must clearly indicate when it is safe to reenter the blast zone.

The air-powered whistle used at the Union Carbide Creek Pine Creek Mine (fig. 2) is a simple and effective means of warning that a blast is about to be initiated, particularly in areas where entries are left unguarded. A whistle improves blasting-site security when used as a backup to human guards and is a very cost-effective device. Easy to use, the whistle is simply screwed onto a drop on the air line and turned on as the blaster

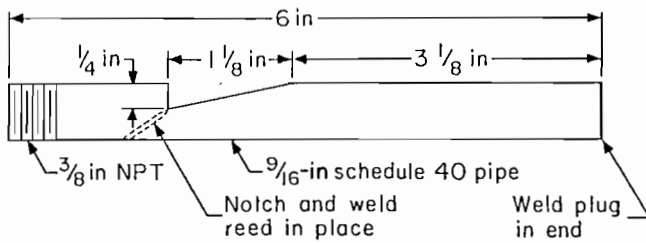


FIGURE 2.—Air whistle used at Pine Creek Mine (NPT—National Pipe Taper).

retreats from the face to begin the clearing process. It remains on throughout the blasting procedure and is turned off by the first person entering the area after the blast.

PERSONNEL ACCOUNTING

Personnel accounting refers to determination of the locations of personnel throughout the mine site. In essence, it is a tracking system and should be designed, at the very least, to account for the presence of personnel within the zone of a blast. Whenever reasonably possible, accounting stations should be established near all blast zones, preferably at the locations from which the blasts are regularly initiated. At a minimum, an effective personnel accounting system must establish that no personnel are occupying the blast zone at blast time. The personnel accounting system is an important backup to the clearing process. However, some large open pit operations that blast several times each day rely on clearing as the only practical method of ensuring that there are no people in the blast area. When clearing is the only method used, the clearing personnel must conscientiously cover the entire blast-affected area prior to every blast.

Many mines account for personnel by means of a head count at some safe location prior to the blast. While a head count is better than not accounting for personnel at all, it allows considerable potential for error.

The system of using a blasting board is applicable at many underground and surface mines. Blasting boards are placed at safe locations where miners congregate

during the blast. The board contains a tag for every person assigned to a particular station plus tags for visitors. When blast time arrives, supervisors at each safe location merely need to look at the board to ensure that all personnel have returned from the immediate blast zone.

Surface mines seem to place less emphasis on personnel accounting than do the underground mines. This is somewhat justified by the fact that clearing procedures are more effectively supervised and performed within surface blast areas because of better visibility. It is also impractical for some large surface mines that blast several times each day to account for all mine personnel before each blast. Fortunately, most of these mines have a limited flyrock range and can reliably use the clearing process. However, where practical, personnel accounting should act as a backup system to the clearing process. Blasting safety at many surface mines would benefit from an improvement in personnel accounting.

COMMUNICATION OF BLAST LOCATION AND TIME

It is important that all personnel entering a mine be aware of the exact location of blasting sites and the schedule for blast initiation. Surface and underground mines that blast on a daily basis can list the blasting times on signs at appropriate entrances into the mining areas. Since blast location changes frequently at most mines, mine personnel should be informed on a daily basis of the exact locations of all scheduled blasts.

Notices regarding blast-site locations in underground mines should be placed at the personnel accounting station for the blasting area. As personnel check in at the blasting station, they are informed of the blast time and location. This particularly benefits personnel who are not regularly assigned to a section for which blasting is scheduled but who are required to move throughout the mine.

BLASTING CREW COMMUNICATIONS

Some surface mines utilize hand signals to communicate between the shot firer and blast guards. However, hand signals do not provide the positive, instantaneous communication so necessary to blasting-site security. A hand signal will not always immediately attract the blaster's attention. Poor visibility, a blaster's not wearing corrective eyeglasses, or sun shining in the blaster's eyes are all cases in which a hand signal might not be easily seen.

Guards should use warning devices capable of quickly and positively gaining the blaster's attention. The two-way radio does this best. Safe separation distances between transmitters and electric blasting circuits can be determined

by reference to the Institute of Makers of Explosive (IME) Publication 20, "Safety Guide for the Prevention of Radio Frequency Radiation Hazards in the Use of Electric Blasting Caps" (4). Another source is the Du Pont "Blaster's Handbook" (5). Radio communication for blast guarding is safe if the IME guidelines are followed. Transmitters should be limited to 5 W or less, and a boundary should be defined around the blast pattern within which the use of radios is forbidden.

An alternative is the use of nonelectric blast initiation systems, which are widely used in the mining industry. These systems cannot be initiated by radio frequency energy and are completely compatible with close radio communication.

FLYROCK

Excessive flyrock is rock that is projected beyond the normal blast-affected area. It is generated when there is too much explosive energy for the amount of burden, when stemming is insufficient, or when the explosive energy is rapidly vented through a plane of weakness. Excessive flyrock is responsible for 24 pct of the blasting accident injuries that occur in surface mining (1978-85). Excessive flyrock can be the result of blast site geology and/or rock conditions, improper blast design, or carelessness. Flyrock control is achieved by careful attention to blast design, blast-site inspection, blasthole layout, blasthole drilling, and blasthole loading practices.

FLYROCK CAUSES

Geology and rock conditions can cause the generation of flyrock. Geologic features such as mud seams, natural joint or bedding planes, fractures, or cavities in the rock can result in excessive flyrock. Mud seams and fractures are planes of weakness through which explosive gases can rapidly vent and accelerate rock fragments. Cavities can accidentally

become filled with too much explosive for the amount of rock burden, resulting in large flyrock distances. Blastholes can penetrate openings from abandoned underground mines to create a dangerous condition similar to that resulting from natural cavities. Fracturing due to backbreak or overbreak from previous blasting can also cause dangerous planes of weakness. A ragged highwall face or an overhang can result in diminished burden along the front row of holes.

Excessive flyrock can be generated if blasts are not properly designed. Any blast design feature that results in insufficient explosive confinement or the rapid venting of the explosive gases can create a problem. Blast design errors such as too high a powder factor, an inadequate burden, too short a stemming region, failure to use stemming, improper delays between rows, or the wrong blasthole delay sequence can result in unwanted flyrock. The wrong delay sequence can cause diminished burden if the delay is too long. Cratering and blowouts can occur when back holes fire before front holes. A very short delay can result in too much confinement and, again, cratering and blowouts.

Unfortunately, carelessness is a leading cause of excessive flyrock. Carelessness during any part of the blast design, the blasthole pattern layout, the drilling of the blastholes, the loading of the blastholes, or the hookup of the initiation system can create a dangerous situation. The loading of explosives too near the collar of the blasthole is a common cause of flyrock.

CONTROL OF FLYROCK

The control of flyrock starts with proper blast design. The correct burden must be used. A small burden will not contain the explosive energy, while using a large burden may result in cratering and/or blowouts. The bench height, burden, and stemming region must be such that the blasted rock movement is primarily horizontal and outward, and not upward. In multiple-row shots, the delay between rows must be long enough to allow rock from an earlier row to move out so that the next row will have adequate relief. Insufficient relief can cause flyrock. However, the delay must not be so long that cutoffs occur and cause misfires that increase the burden on later firing holes, again resulting in blowouts and flyrock.

In designing a blast, relationships between charge diameter, burden, spacing, subdrilling, stemming region, and bench height are available for initial estimates (6). The type of explosive, the priming, and the initiation system must be selected. Toe priming reduces flyrock, compared with collar priming. Decisions must be made on the type of blasthole pattern, square or staggered, and on the delay sequence. The powder factor is calculated to ensure that the quantity of explosive being used is within the range of that normally used in surface mine blasting. These initial approximations must be modified for the particular blasting situation and may be further modified after experience with a number of blasts.

Before the blasthole pattern is laid out in preparation for drilling, a careful inspection of the blast site should be made. The face should be examined for

ruggedness, overhangs, fractures, zones of varying competence, and amount of toe burden. The blast site should also be inspected for backbreak, jointing, mud seams, voids, and other zones of weakness. Any of these blast-site features could cause excessive flyrock.

The layout of the blasthole pattern starts with the front row. If the vertical face has overhangs, is concave, or is irregular, the burden at some point may be reduced and violent cratering could occur. Faces with backbreak, open joints, weak zones, and mud seams will allow rapid venting of explosion gases with flyrock. In addition, the burden will not pull (be removed) as planned, causing an increase in the burden for later holes, which results in cratering at the top of the bench. If the face is sloped, the toe burden will be larger than the crest burden unless angled holes are used. If the normal column load is used when there is a sloped face, there will be flyrock because of the short crest burden. Also, the toe may not pull, producing a buildup in front of later holes, again resulting in flyrock. Adjustments in hole locations and powder columns in the front row should be made when conditions exist near the face that favor the generation of flyrock. Once the front row is established, the balance of the shot can be laid out. A tape should be used to ensure accurate spacing and burden distances.

Locating a blasthole close to an open fracture will provide a weak zone. The shot will break into the fracture, vent with flyrock, and produce poor fragmentation. The same kind of venting can occur when a hole is abandoned and a second hole is drilled a few feet away. To prevent this, the first hole should be backfilled. Where open fractures are present, they can be backfilled, but this is difficult and time consuming. The best way to handle fractures from previous blasting is to eliminate the cause of overbreak.

Accurate drilling is essential. The holes must be located in accordance with the blast design and drilled at the correct angle and to the proper depth. With a high face and smaller diameter holes,

extra care should be taken to ensure that holes are drilled at the proper angle. Deviated blastholes can result in burdens much smaller or larger than planned. The driller should provide a log of each hole that includes depth drilled, problem drilling zones, and any changes in penetration rate. This information could indicate voids or zones of weakness.

All holes must be checked before explosives are loaded, to ensure that their location and depth are in accordance with the blast design and the driller's log. If blocked holes are undetected, the explosive column can be loaded too near the collar, leaving an insufficient stemming region. Partially blocked holes can also be a problem and may not be detected with a weighted tape. On sunny days, a mirror may be used to check for obstructions. Before an attempt is made to load a short hole or a partially blocked hole, corrective action such as redrilling, changing the planned explosive load, or abandoning and backfilling of the hole should be considered.

Open joints and cracks can extend into a blasthole. The presence of cracks can often be detected by a lack of drill cuttings at the top of the hole. However, unless corrective action is taken during the loading of bulk products, these open fractures can be loaded with explosives, resulting in excessive energy for the hole. There will be less confinement of the explosive charge. This may affect the performance of some explosive products as well as provide a weak zone that may blow out.

Frequent checks of the powder-column rise during loading are important. A slower than normal rise may indicate a void, while a sudden rise indicates a blockage. When a blockage occurs during explosive loading, care must be taken to ensure that the explosive load above the blockage will detonate. This will require the placing of another primer and detonator in the column above the blockage. When a void is indicated, loading must be stopped. If the void is not unduly large, backfilling with stemming material will correct the problem. Another primer must be added and loading

continued. If the void is too large for this to be practical, the hole may have to be abandoned and another hole drilled nearby. Redrilling should not be done if there is a possibility of drilling into explosives. In some cases, the hole can be plugged just above the void. Once a plug is formed, the explosive loading can resume. In operations where voids are common, a special system for borehole plugging should be developed.

Checking the column rise will prevent accidental overloading of the blasthole. Maintaining sufficient stemming is an important factor in flyrock control. Stemming lengths of 0.7 to 1.0 times the burden are commonly used. When collar priming is used, the stemming length may need to be increased because of the greater potential for violence with top priming. Crushed and sized rock is the best material to use for stemming, but drill cuttings are commonly used because of availability and economy. Large pieces of rock or other material should never be mixed with the stemming as they can become missiles if there is a blow-out. Large rocks can also cut off or damage the initiation system and cause a misfire.

Care must be taken to ensure that the initiation system is properly hooked up and that the delays are correct. A final check of the hookup is imperative.

The secondary blasting of boulders too large for the loading equipment or crusher is required at some operations. Secondary blasting can produce dangerous flyrock even though the charges are small. Determining the blast area for this kind of shooting is very difficult, and careful clearing and guarding are required. Although secondary blasting is frequently done on-shift and as needed, it is best to shoot at a standard time such as during shift change. Secondary blasting can be done at the same time as primary blasting if the same immediate area is involved. However, if the blasts are widely separated, there will be two sources of flyrock to guard against.

The reshooting of misfired blastholes can generate dangerous flyrock when there is reduced burden and reduced confinement

of the explosive charge. When reduced burden, distance can exceed the normal flyrock range, from production blasts, and a larger-than-normal blast area is

required, which must be cleared and guarded. It is best to shoot misfired blastholes during shift change or at the same time as primary blasts.

MISFIRES

A misfire results when explosives fail to detonate as planned during a blast. A misfire has two basic effects on an operation: the safety hazard it presents and the increase in mining costs.

EFFECTS OF MISFIRES

To most people, a misfire primarily represents a safety hazard. With the increasing use of non-cap-sensitive blasting agents, the possibility of accidental initiation is reduced. However, based on U.S. Mine Safety and Health Administration (MSHA) data, there are still many injuries sustained because of misfire accidents. In an 8-yr period (1978-85), 56 misfire accidents resulted in 63 injuries and 6 fatalities. The majority of misfire accidents (75 pct) occur in underground mines, with 54 pct in underground metal-nonmetal mines. These numbers are not surprising because it is not the total amount of explosives used, but the number of shots or holes fired that provides the opportunity for misfires. Also, underground mines use smaller diameter charges, fired on smaller spacings, which are prone to misfire. In addition, visibility in underground mines is generally poor, so detection of misfires is hampered.

The remaining 25 pct of misfire accidents occurred in surface mines, with 11 pct in surface coal mines and 14 pct in surface metal-nonmetal mines.

Misfires that result in an accident with injuries, fatalities, and/or equipment damage involve obvious costs; however, there are other costs due to misfires that are not obvious. Because the effects of misfires will vary greatly, operators should conduct probable cost analyses of misfires at their mines. Those who do will place greater emphasis on avoiding misfires.

In addition to the cost of disposal of the misfire, there are the direct costs

in additional drilling, explosives, primers, detonators, and labor. In some underground mines, the failure of a "cut" hole could result in the loss of the entire round.

The handling of boulders that require secondary breakage is another cost factor. Increased digging time and greater wear and damage to equipment (especially bucket teeth) result in lower productivity and higher maintenance costs. Misfires frequently are the cause of high bottom, which results in reduced production and higher maintenance costs on mucking equipment. In addition, haulage vehicles traveling over rough terrain will increase the haulage cost and vehicle maintenance. Often, these humps must be drilled and blasted, which constitutes another cost.

The misfiring of one hole will increase the burden on a later hole, causing cratering with excessive flyrock and overbreak. Flyrock is a leading cause of personal injury and equipment damage. Overbreak may extend beyond the burden for the first row of holes for the next shot, which can cause problems in drill setup for the next shot. Overbreak at the final pit wall could produce ground control problems with very high cost and even loss of ore.

CAUSES AND AVOIDANCE OF MISFIRES

During the preparation and initiation of blasts, there are many aspects that may result in misfires. The most frequently stated cause of misfires is the incorrect use of the initiation system, a problem common to all initiation systems. A major contributing factor is the lack of understanding by blasting personnel of how the system works. Unless the blasting personnel have a full understanding of the initiation system, even minimal changes in a shot can result in poor blast performance and misfires.

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Damage to the initiation system or explosives column is another common source of misfires. Causes include poor work practices on the blast site and rock movement that produces cutoffs.

Damage can often occur while stemming is shoveled in and when wires, cords, or tubes are stepped on. Even driving over the initiation systems is not uncommon at some mines. The wiring-in or hookup of a shot, regardless of the initiation system used, is a very important part of blasting. It is important that each initiation system be checked after the hookup. The method of system checkout depends on the system used. All electrical hookups should be checked with a *blaster's* meter. All shots should be checked visually. Good housekeeping and neat and consistent hookup practices are helpful in accomplishing the system checkout.

Rock movement may cut off the explosives columns and result in misfired explosives. Uplift, as well as horizontal displacement, is a factor in cutoffs. An area surface coal mine may have 100 or more holes drilled and loaded, which are then divided into 4 or 5 shots. Where rock movement is generally up, with little or no horizontal movement, the uplift can damage loaded holes waiting to be wired in for the next shot.

The proper use of delays is very important in preventing cutoffs. The longer the delay between holes, the greater the probability of cutoffs. Shorter delay times are needed when surface delays are used. If longer delay times are required for fragmentation and rock displacement, in-the-hole delays should be considered. Many systems or combinations of systems are available to meet delaying requirements.

Geology is another factor that plays a major role in causing cutoffs. Fractures, faults, joints, and bedding planes are all zones of weakness that may cause burden movement to occur in much shorter times than considered normal. The use of decking and multiple primers is sometimes advantageous to avoid cutoffs. Rock falling from the walls of the boreholes can cause bridging and explosive column separation.

Hole blockage can also be the result of careless work habits, such as knocking material into the hole. Rapidly loaded cartridge products will often bridge, particularly in holes that are partially filled with water. Careless work practices are often due to the rush placed on the blasting crew to get the shot off by a given time.

Poor priming practices frequently cause detonation failure. Each product has minimum priming requirements. Even when proper priming is used, if the primer sinks into the mud at the bottom of the hole or water enters a hole loaded with non-water-resistant products, a misfire can occur. Many operators of surface mines using larger diameter blastholes place the primer at the floor level and not at the bottom of the hole in the sub-drilled region. In the event of a misfire, the more sensitive primer and detonator are more easily retrieved.

Storage is also an important factor in avoiding misfires, since improper storage can alter the performance of many products. The sensitivity of some products is dramatically reduced by low-temperature storage, which can result in malfunctioning. Malfunctioning of explosives products is usually due to improper storage rather than quality control problems in manufacturing, which are rare. Hydrostatic pressure as well as compression from the firing of adjacent holes has caused some products to lose sensitivity and misfire. If misfires are to be avoided, the blaster must have a complete understanding of the products and conditions under which they can be used. This information is available from the supplier.

DETECTION OF MISFIRES

Sometimes misfires are obvious, sometimes they are not. Each shot must be checked for misfires before mucking is begun. When the explosive is lying on the muck, detection of a misfire is no problem. However, when the explosive is buried in broken rock, visual detection is unreliable.

There are a number of clues that may indicate a misfire. Most operations have standardized blasting practices that have fairly uniform results from each shot. A change from the norm could indicate misfires. In surface mines where the shot can be observed from a safe location, watching and listening to the shot is worthwhile. A change in the sound--louder or quieter--may indicate misfires. Ejection of stemming, cratering, and flyrock may result from too much burden, due to misfiring of earlier holes.

In surface and underground operations, the muckpile profile can reveal areas of possible misfires. Muck lying mostly to one side of the shot, less displacement than expected, abnormal backbreak, and humps and valleys in the muckpile can all be due to misfires. Change in fragmentation is a very good indicator of possible misfires. Boulders across the top of the muckpile are easy to see and could indicate misfires. Operators of loading equipment should be aware that boulders uncovered in lower sections of the pile also may be the result of misfires. This is common where multiple decks are used.

Multiple priming can minimize misfires, although this method is sometimes not used because of additional costs. Considering the cost of misfires, perhaps multiple priming should be used more frequently. A double-trunkline or loop system must be used with detonating cord systems. Even with the two paths of detonation, all of the cord should be consumed in the blast. Finding detonating cord in the muck is a strong indication of a misfire.

In underground mines, detection is hampered by poor lighting. It is difficult for the loader operator to spot explosives in the muck. Checks for explosives should be made before and during the mucking operation. Every bootleg must be examined carefully for misfires. A misfire may have occurred even though there is no cap legwire or tubing protruding from the hole. Lifter holes are of particular concern because they slope downward and are often water filled, and it is easy for loose material to fall into them. These conditions promote misfires. In addition, lifter holes are the most

difficult to relocate and check for misfires. Because of potential misfires, blastholes must *never* be collared in bootlegs.

DISPOSAL OF MISFIRES

There are two basic methods used to dispose of undetonated explosives: to recover and destroy the explosives or to detonate the misfire in place. Any explosive product removed from a misfire is considered damaged and must be destroyed in a safe manner. The manufacturer is the best source of information on destroying an explosive product. Many water-based products do not burn readily and are difficult to destroy. The most common method is in-place detonation. This is good practice in underground mines where flyrock is less of a problem. However, in many operations, the disposal of misfires is the only blasting that is done on-shift, and this creates blast area security problems.

When the original initiation system is still intact, it can be used to refire the charge, except with cap and fuse blasting. When a misfire results with cap and fuse blasting, the blaster should never relight the fuse because it may have been shortened, causing unexpected premature initiation, or it may be damaged and could produce a hangfire. With cap and fuse blasting, repriming is essential.

Misfires are occasionally refired in surface mines, in which case flyrock is a major consideration. Because of reduced burden on the missed hole, violent flyrock may result. Normally, removal of the explosive load is recommended in surface mines. Mucking out a misfire must be done with caution, with the minimum number of personnel in the area, and under the supervision of a competent person.

Removal of stemming and explosives from blastholes is more difficult in vertical holes than in horizontal holes. Two common techniques used are washing material out with water or blowing it out with air. When the main charge is ammonium nitrate and fuel oil (AN-FO), washing with water has an advantage because water

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will desensitize the AN-FO. But when re-priming of the hole is planned, the use of water is a disadvantage and it is better to blow out the stemming with air. Also, air has an advantage in that it is readily available in many mines. The disadvantage of air is that it blows dirt and dust into the atmosphere, creating poor working conditions around the hole. When the explosive is to be removed, both air and water may deposit the charge in fractures around the borehole, which may create more of a problem than if the charge is left and dug out during the mucking operation.

The removal of an explosive charge by firing a nearby charge is another

technique that has been used. This method is not recommended and should be used only as a last resort, because the danger of drilling into the misfire is always present. In blasts with angled holes, such as vee cuts, or where hole length-to-burden ratios are high, this technique should not be used. When it is used, recovery and disposal of the explosive must still be performed in a safe manner.

If explosives from a misfire must be stored, all detonators should be removed and stored separately. Explosives and detonators removed from a misfire should not be stored with other explosives or detonators.

CONCLUSIONS

Blasting accidents are very costly in terms of human suffering, lost production, and damage to equipment. Blasting accident can be avoided if all persons involved in the blast have a thorough understanding of the conditions that can result in an accident and an appreciation of the hazards involved, and take the appropriate precautions.

The requirements of blast site security are common to both surface and underground mines. The two basic requirements of blast guarding are (1) to move mining personnel out of range of the blast and (2) to prevent the movement of personnel back into the blast zone.

Elements that contribute to the development of safe blast guarding are management commitment to safety, training, definition of blasting authority, preblast planning, definition of blast zone boundaries, selection and placement of guards, clearing plans, location of the blast initiation site, blasting time, blast warning signals, personnel accounting, communication of blast location and time, and blasting crew communications.

Many factors can affect the amount of flyrock produced by a blast. The blast design must be appropriate for the site. Blasthole pattern layout and drilling must be accurate and must take into account blast-site conditions. All holes

must be checked before loading of explosives, and loading must be monitored closely so that any problems encountered can be corrected.

Most misfires are due to some problem with the initiation system such as failure to make a connection, a broken lead, or simply not understanding the initiation system. Other causes of misfires are cutoffs, inadequate priming, and malfunctioning of the explosives due to improper storage.

Detection of a misfire is no problem if none of the holes detonate. However, if only a few holes or portions of a single hole fail to detonate, detection of the misfire can be very difficult. In these cases, visual inspection of the muckpile for undetonated explosives and boulders or other muckpile irregularities that suggest possible misfires is the most reliable detection method.

Disposal of detected misfires is accomplished by removing the explosives with water washing or air flushing, repriming, and reshooting, or by detonating a nearby charge. However, detonating a nearby charge can be very dangerous and is not recommended.

The best way to avoid misfire accidents and costs is to eliminate their causes. This can be done by knowing the characteristics of the explosives, delays, and

initiation system, by proper blasting design, by taking care in loading the shot and hooking up the initiation system,

and by good housekeeping practices at the blasting site.

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BLASTER'S TRAINING MANUAL FOR METAL AND NONMETAL MINERS

By Michael A. Peltier,¹ Larry R. Fletcher,² and Richard A. Dick³

ABSTRACT

The Bureau of Mines has developed a blaster's training manual for the metal and nonmetal mining industry. The material is divided into 6 chapters and 47 modules, with each module covering a single topic. (For example, the second chapter, which deals with initiation and priming, is subdivided into nine modules. One module covers initiations systems in general, another covers delay series, and one discusses priming. The remaining six modules deal with each of the six initiation systems.)

The modules were structured to enable mine training personnel to easily develop

a site-specific blasters' training program. Each module contains text material that comprehensively covers the topic, as well as a paraphrased section highlighting the major ideas of the text. Also included with each module are line drawings and test questions with answers.

The objective of this material is to increase hazard awareness and foster the use of safe blasting practices, with the anticipated end result being accident-free and productive blasting.

INTRODUCTION

Based on accident data obtained from the U.S. Mine Safety and Health Administration (MSHA), most blasting accidents are caused by human error, lack of hazard awareness, or lack of general blasting knowledge. A lack of understanding as to how explosives function can contribute to higher mining costs because of inadequate fragmentation or lost production.

Federal regulations require that every person who uses or handles explosive materials be experienced and understand the hazards involved. Trainees should do such work only under the supervision of and in the immediate presence of experienced miners. Federal regulations also require hazard and task training for miners. Most training given on mining

property is based on experience at that mine and is done without the aid of adequate training materials. An improved and more meaningful blasters' training program is essential in assisting operators to properly train blasters and meet MSHA training regulations.

The blasters' training material was developed to aid industry in the preparation of a site-specific training course and is based on a previous Bureau of Mines Information Circular titled "Explosives and Blasting Procedures Manual".⁴ The intent is to help individuals using explosives and blasting agents to develop a better understanding of the various aspects of blasting that contribute to a safe and efficient blast.

PREPARING A TRAINING COURSE

The blaster's training manual has been constructed to be easily used in developing a site-specific and comprehensive

blasters course. The material consists of discrete modules that contain text material, a paraphrased section, line drawings, and test questions with answers.

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Individual pages have been divided lengthwise with the comprehensive text material on the left-hand side of the page. Each paragraph of the text material is numbered for quick reference. The right-hand side of the page consists of paraphrased text material with a main heading and a paragraph number. The person preparing the training course can read the paraphrased material quickly in order to grasp the main ideas of the text material. If an explanation is needed, the individual, by noting the paragraph number, can go directly to the paragraph discussing a particular point.

Line drawings are included with the material to illustrate specific concepts. The line drawings can be easily converted to overhead transparencies for use in the training course.

The first step in preparing a blasters training course is to determine what material must be covered. This can be accomplished by talking with the blasting supervisor and blasters, and by observing the blasting operation. To help determine what topics need to be covered in the course, a checklist is included with the material. It is arranged to parallel the chapters. By completing the checklist, the trainer will be able to locate the modules to be included in the course. For example, under the "Chapter

1--Explosives Products" section of the checklist, ammonium nitrate and fuel oil (AN-FO) and emulsion may be noted next to the subsection "Blasting Agents." By reading the list of modules in the table of contents under the "Chapter 1--Explosives Products" section, the trainer will notice that module 4 discusses AN-FO and module 5 discusses emulsions.

The second step is to gather the training material needed. The information gathered from blasting personnel through the use of the checklist will indicate which modules should be included in the course. In addition to the modular material, slides and other visual aids from the actual operation should be used. Additional technical information concerning specific blasting products can be obtained from either the explosives supplier or manufacturer.

The third step is to write lesson plans for the course and arrange the training material into a cohesive unit. The writing of the lesson plans can be simplified by making extensive use of the paraphrased sections in the modules.

Since the experience, knowledge, and ability of individual blasters vary widely, both the length and amount of material to be included in the course will have to be determined by mine management.

CHAPTER CONTENTS

CHAPTER ONE--EXPLOSIVES PRODUCTS

Objectives

Purpose and Description

The purpose of this chapter is to help the blaster develop an understanding of various types of explosives. Chemical and physical properties of seven types of explosive products are discussed. Additional information explains nine properties of explosives that are used to determine how an explosive product will function under field conditions. Material explaining how to select an explosive product is included in this chapter.

Upon completion of this chapter, the blaster should be able to

1. Give a concise explanation of the nature of various explosive products;
2. List the basic reactive ingredients of an explosive product;
3. Explain how the detonation pressure and explosive pressure cause the rock to be broken;
4. Explain the importance of oxygen balance as it relates to both the energy released and the formation of toxic gases;

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5. Describe the individual characteristics of the explosive products the blaster may be using;
6. Briefly explain why a particular product is being used at the blaster's operation;
7. State and explain nine basic properties of explosive products; and
8. Relate the basic properties of explosives to the types of explosive products being used on the job.

Chapter Modules

<u>Module</u>	<u>Title</u>
1.....	Chemistry and Physics of Explosives.
2.....	Types of Explosives and Blasting Agents.
3.....	Nitroglycerin-Based High Explosives.
4.....	Ammonium Nitrate-Fuel Oil (AN-FO).
5.....	Slurries, Water Gels, Emulsions.
6.....	Heavy AN-FO.
7.....	Primers and Boosters.
8.....	Liquid Oxygen Explosives.
9.....	Black Powder.
10.....	Properties of Explosives.
11.....	Explosives Selection Criteria.

CHAPTER TWO--INITIATION AND PRIMING

Purpose and Description

The purpose of this chapter is to help the blaster develop an understanding of six initiation systems. The blaster will learn the various components of each initiation system, how each individual system functions, and the advantages and disadvantages of the six systems. Information about the two basic delay series and material concerning priming are also included in this chapter.

Objectives

Upon completion of this chapter, the blaster should be able to

1. Name the three basic parts of an initiation system;
2. Explain the difference in sensitivity to initiation between high explosives and blasting agents.
3. State the difference between an instantaneous and a delay detonator;
4. List the various components of the initiation system the blaster will be using;
5. Explain how the initiation system functions;
6. Explain how to check the final hookup of the system;
7. Discuss the potential hazards to the initiation system;
8. Give the definition of a primer;
9. Name some types of explosives used as primers;
10. Explain the proper procedure for making primers; and
11. Explain why the proper location of the primer in the borehole is important.

Chapter Modules

<u>Module</u>	<u>Title</u>
12.....	Initiation Systems.
13.....	Delay Series.
14.....	Electric Initiation.
15.....	Detonating Cord Initiation.
16.....	Detaline Initiation System.
17.....	Cap-and-Fuse Initiation.
18.....	Hercudet Initiation.
19.....	Nonel Initiation.
20.....	Priming.

CHAPTER THREE--BLASTHOLE LOADING

Purpose and Description

The purpose of this chapter is to examine proper blasthole loading techniques. The chapter discusses loading procedures for both small- and large-diameter blastholes. Also included in the chapter is material that discusses how to check blastholes for proper depth, water, voids, and obstructions, and how to mitigate these problems.

Objectives

Upon completion of this chapter, the blaster should be able to

1. Explain why blastholes should not be loaded and workers should retreat from the blast area during the approach or progress of an electrical storm;
2. Describe how to check the borehole for proper depth, obstructions, water, and voids;
3. Explain how to remedy problems, such as improper borehole depth, obstructions, water, and voids;
4. State why stemming is important and how to estimate the amount of stemming needed;
5. Explain when plastic borehole liners or water-resistant cartridges should be used;
6. Explain the proper technique for loading the explosive or blasting agent the blaster will be using.
7. Describe the characteristics of the type of pneumatic loading the blaster will use;
8. Explain the potential problem of static electricity if the blaster is going to use a pneumatic loader; and
9. List the advantages and disadvantages of using bulk-loaded products in large-diameter blastholes.

Chapter Modules

<u>Module</u>	<u>Title</u>
21.....	Introduction.
22.....	Checking the Blasthole.
23.....	General Loading Procedures.
24.....	Small-Diameter Blastholes.
25.....	Large-Diameter Blastholes.

CHAPTER FOUR--BLAST DESIGN

Purpose and Description

The purpose of this chapter is to examine the factors that influence safe and effective blast design. In addition to the discussion of design factors for surface and underground blasting, four controlled blasting techniques are also covered.

Objectives

Upon completion of this chapter, the blaster should be able to

1. Discuss how geology affects fragmentation;
2. Name the most significant geologic features to consider when designing a blast;
3. Discuss the importance of a well-detailed drilling log;
4. Explain how to determine the burden;
5. Explain why geologic structure is the major factor in determining blasthole diameter;
6. Explain how collar distance affects fragmentation size;
7. Explain the relationship of collar distance to airblast and flyrock;
8. Explain the relationship between burden flexing and rock fragmentation;
9. Discuss the problems of either excessive or insufficient subdrilling;
10. Explain how spacing is determined;
11. Explain the advantages of millisecond delays;
12. Discuss the two classifications of opening cuts;
13. Explain how to design an angled cut;
14. Explain how to design a parallel-hole cut;
15. Discuss the two types of delays for underground blasting;
16. Name the two main advantages of using controlled blasting;
17. List the four primary methods of controlled blasting; and
18. Discuss the advantages and disadvantages of the various methods of controlled blasting.

Chapter Modules

<u>Module</u>	<u>Title</u>
26.....	Introduction to Blast Design.
27.....	Properties and Geology of the Rock Mass.
28.....	Surface Blasting.
29.....	Underground Blasting.
30.....	Controlled Blasting Techniques.

CHAPTER FIVE--ENVIRONMENTAL EFFECTS
OF BLASTING

Purpose and Description

The purpose of this chapter is to examine the environmental effects of blasting. The material will discuss flyrock, ground vibrations, airblast, and dust and gases. Methods to reduce the potential health and safety hazards they may present will be discussed.

Objectives

Upon completion of this chapter, the blaster should be able to

1. Explain the importance of conducting a preblast survey, maintaining comprehensive records, and good public relations;
2. Discuss the causes of flyrock;
3. Discuss methods to alleviate flyrock;
4. Discuss the causes of ground vibration;
5. Discuss design techniques to minimize vibrations;
6. State some methods to monitor ground vibrations;
7. Discuss the causes of airblast;
8. Discuss methods to monitor airblast;
9. List techniques to reduce airblast;
10. Explain why an adequate amount of time must be given for dust and gases to be diluted before returning to the blast site; and
11. List the two common toxic gases produced by blasting and list techniques to reduce them.

Chapter Modules

<u>Module</u>	<u>Title</u>
31.....	Introduction to Environmental Effects of Blasting.
32.....	Flyrock.
33.....	Ground Vibrations.
34.....	Airblast.
35.....	Dust and Gases.

CHAPTER SIX--BLASTING SAFETY

Purpose and Description

The purpose of this chapter is to help the blaster develop a better understanding of blasting safety, by examining a number of auxiliary blasting functions. A number of precautions related to previous modules are mentioned. Four accident types that occur frequently are also discussed.

Objectives

Upon completion of this chapter, the blaster should be able to

1. Explain why a knowledge of all current blasting safety regulations is important;
2. Name the agencies that regulate and enforce the use and storage of explosives and blasting agents;
3. Describe the requirements for vehicles used to transport explosives and blasting agents from the magazine to the job site;
4. Explain the importance of marking the blast area and keeping nonessential personnel away;
5. Explain when to check for extraneous electricity;
6. Discuss why electrical storms are a hazard regardless of the type of initiation system;
7. Explain the importance of proper primer makeup;
8. List a number of checks to be made before borehole loading begins;
9. Describe various methods to check column rise during borehole loading;
10. Describe some precautions to consider before and during the hookup of the shot;
11. Explain some good methods for blast area security;
12. Describe the potential hazards to check for when reentering the blast site after the shot has been fired;
13. Discuss methods for disposing of misfires; and
14. Discuss the principal causes of blasting accidents.

<u>Chapter Modules</u>		<u>Module</u>	<u>Title</u>
<u>Module</u>	<u>Title</u>		
36.....	Introduction to Blasting Safety.	41.....	Borehole Loading.
37.....	Explosives Storage.	42.....	Hooking Up the Shot.
38.....	Transportation From Magazine to Job Site.	43.....	Shot Firing.
39.....	Precautions Before Loading.	44.....	Postshot Safety.
40.....	Primer Safety.	45.....	Disposing of Misfires.
		46.....	Disposal of Explosive Materials.
		47.....	Principal Causes of Blasting Accidents.

SUMMARY

A training manual for metal and nonmetal mining has been developed by the Bureau. This program consists of 47 modules or topics under 6 major headings (chapters). The modules consist of a text and outline on a single blasting topic, plus questions and answers.

Supplementing the modules are a 73-item bibliography, a list of regulatory authorities and their responsibilities, additional information on MSHA and OSM (U.S. Office of Surface Mining), a glossary, and 65 illustrations suitable for duplication.

DELAYED BLASTING TESTS TO REDUCE ROCKFALL HAZARDS

By Virgil J. Stachura¹ and Larry R. Fletcher²

ABSTRACT

The Bureau of Mines conducted delayed blasting experiments at a contour coal mine, which were designed to reduce overbreak without special drilling or significant additional costs. In the standard layout of the blast pattern at this mine, the ends of the rows formed the highwall. Overbreak was reduced by increasing the delays on the last row of holes at the highwall, which changed the effective delay pattern geometry and the direction of burden movement. These experiments resulted in smoother highwalls, which were also inherently safer because of the reduced likelihood of rockfall.

Three delay combinations were tested: 50 ms longer than the nominal design,

100 ms longer than nominal, and 50 and 100 ms longer in the two rows of holes nearest to the highwall. The mine's nominal blast design was a flat V-pattern with 17-ms surface delays between holes, 42-ms surface delays between rows, and 200-ms in-the-hole delays in each hole. All three test designs produced highwall improvements, compared with results using the nominal design, with occasional exceptions because of geologic variations. Observations and terrestrial photogrammetry showed that the delay changes produced generally smoother vertical profiles with less loose material.

INTRODUCTION

One of the major hazards found in surface mining is rockfall from highwalls. This hazard occurs in all forms of excavation in rock, especially where explosives are used. The explosive energy not only fractures the rock to be excavated but also damages the rock that borders the excavation. This reduces the stability of the highwall and increases the potential of rockfall. The rockfall hazard is normally attributed to blasting practices, geologic conditions, and adverse weather in 65 pct of accidents resulting from fall of rock (1).³ Of these three factors, only blasting is controllable, and therefore, blasting was the subject of this investigation.

In earlier research sponsored by the Bureau of Mines, Engineers International

Inc. evaluated blasting practices at nine contour mines (2). Almost all the mines visited had highwall instability problems that were aggravated by poor blasting practices. Engineers International conducted eight test blasts that demonstrated that good blasting practices did improve highwall stability. However, after the tests were complete and the contractor was off the site, the mine personnel reverted to their old blasting practices.

To avoid a similar result, the tests in this report emphasize simple, easily understood changes that minimize economic and procedural impact and maximize operator acceptance. The experiments are directed at reducing overbreak without special drilling or significant additional cost. They use simple changes in blast-hole initiation timing, which improve relief by changing the direction and time of burden movement. In this report, overbreak is defined as excessive breakage of rock beyond the desired excavation limit (3).

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

BLAST DESIGNS

The approach selected for devising the experimental blast designs was to take the design in use at the mine site and make minor delay-period changes. No other parameters were changed intentionally, but in the mine's normal blasting procedures the accuracy of spacings and burdens varied, more than one hole diameter was occasionally used, and powder column heights also varied. The mine where the tests were conducted used a Nonel⁴ (also called shock tube) initiating system, so the original delay times reflected those available using that system.

The blast design used by the mine, design 1, was a flat V with surface delays of 17 ms between holes, 42 ms between rows, and 200-ms in-the-hole delays (fig. 1). Designs 2 and 3, experimental designs used by the Bureau, were the same as design 1 except that in-the-hole delays of 250 and 300 ms, respectively,

were used in the highwall holes (figs. 2-3). Design 4, another Bureau design, was the same as design 1 except that 300-ms in-the-hole delays were used in the highwall holes and 250-ms in-the-hole delays were used in the second row of holes from the highwall (fig. 4). Figures 1 through 4 show the cumulative delay times for each blast hole and include arrows to indicate the observed direction of burden movement. In figures 2 through 4, this direction is perpendicular to the plane of the highwall, a sign of improved relief over that illustrated in figure 1. Design 4 was tried because of reports of overbreak extending far beyond a distance equal to one burden (4-5). It was anticipated that the individual delay in the second row of holes from the highwall would provide additional relief, reducing the damage to the highwall.

HIGHWALL EVALUATION

To determine a criterion for evaluating the test blasts, discussions were held with U.S. Mine Safety and Health Administration (MSHA) inspectors, mine superintendents, safety officers, and blasters. The general theme found in these discussions was that a smooth highwall of competent appearance is a safer one. A competent appearance is achieved by reducing overbreak. In addition to visual inspection criteria, stereo photography techniques were used to analyze the blasting effects in this study. The use of highwall profiles generated from stereo photographs provided for more analytical and consistent comparison of highwalls than was possible with visual inspection alone. The details of the procedure used to take and analyze stereo photographs may be found in RI 8916 and RI 9008 (6-7).

Typical examples of individual profiles from two different highwalls are shown in figures 5 and 6. Figure 5 shows a large ledge at the top of the test area, a result of shot design 1. Figure 6 shows an adjacent highwall, which resulted from shot design 2 and has a much smoother profile. Figures 7 and 8 show a test highwall and an array of profiles that are the result of shot design 1. Figures 9 and 10 show a test highwall and an array of profiles that are the result of design 2. The photographs (figs. 7, 9) were used to evaluate amounts of loose material and the appearance of competence. The arrays of profiles (figs. 8, 10) illustrate the degree of overhanging material and the size of ledges.

A total of 59 test blasts and their resulting highwalls were evaluated by various combinations of stereo photographs, on-site observation, photographs taken by the blaster, and notes made by the blaster.

⁴Reference to specific products does not imply endorsement by the Bureau of Mines.

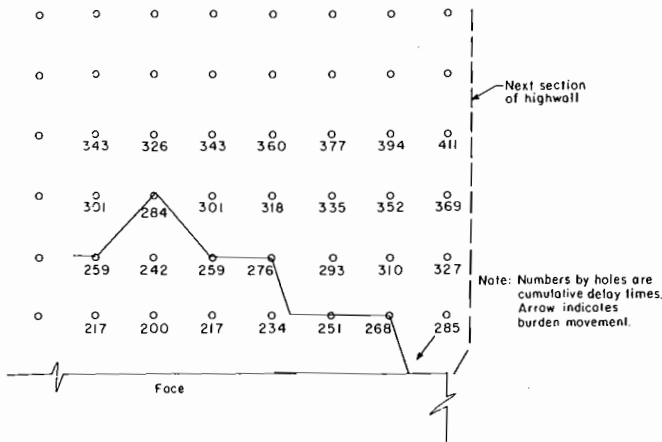


FIGURE 1.—Design 1: break line prior to 285 ms with 200-ms in-the-hole delays throughout.

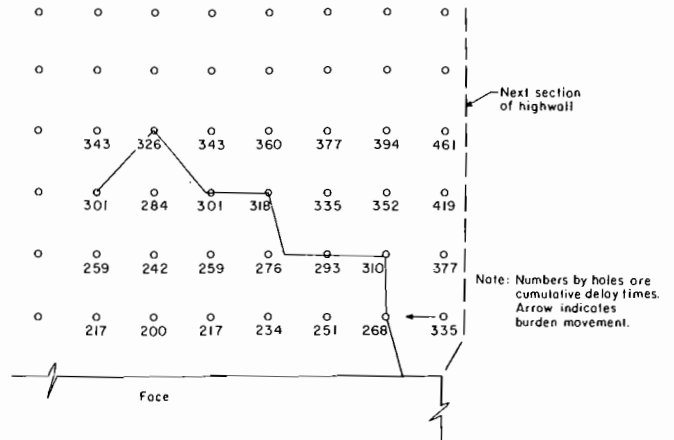


FIGURE 2.—Design 2: break line prior to 335 ms with 250-ms in-the-hole delays in highwall holes and 200-ms in-the-hole delays in remaining holes.

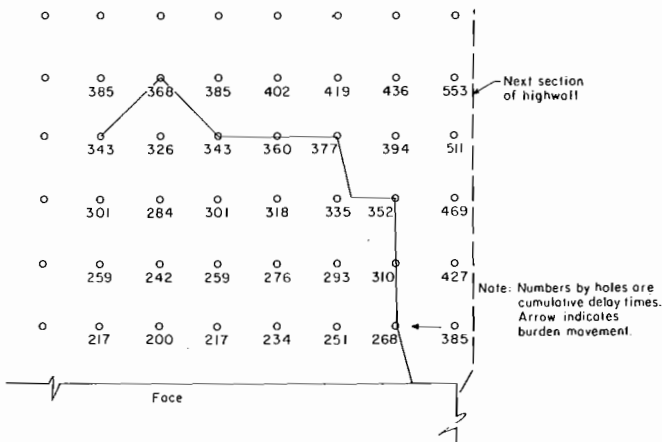


FIGURE 3.—Design 3: break line prior to 385 ms with 300-ms in-the-hole delays in highwall holes and 200-ms in-the-hole delays in remaining holes.

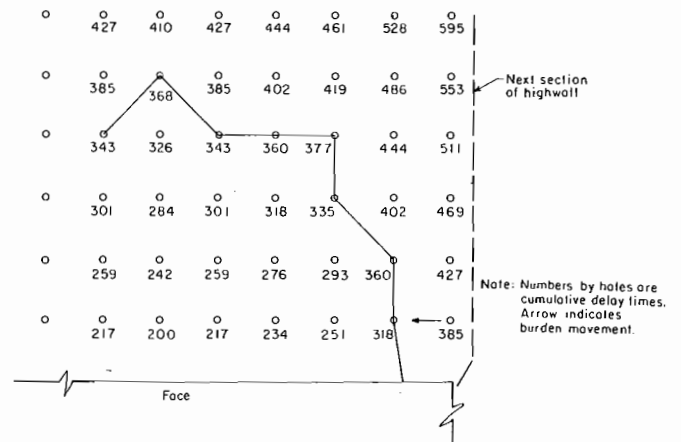


FIGURE 4.—Design 4: break line prior to 385 ms with 250- and 300-ms in-the-hole delays in two rows of holes nearest to highwall and 200-ms in-the-hole delays in remaining holes.

CONCLUSIONS AND RECOMMENDATIONS

The results demonstrated that greater highwall stability can be achieved by changing the burden movement to a direction closer to perpendicular to the plane of the highwall and by allowing more time for movement of the burden in front of the highwall blastholes. The changes in blast design described in this report can be implemented without increased costs or technical complication.

The burden movement was redirected by using delays in all highwall holes that

were either 50 or 100 ms longer than those previously used in the delay pattern. Another variation was to increase the delay time by 50 ms in the second row of holes from the highwall and by 100 ms in the highwall row of holes. The lengthened delays allowed more time for the burden to move, thereby reducing overbreak, which causes irregular and unstable highwalls. At the test sites used, the 50-ms-longer delays or a combination of 50- and 100-ms longer delays

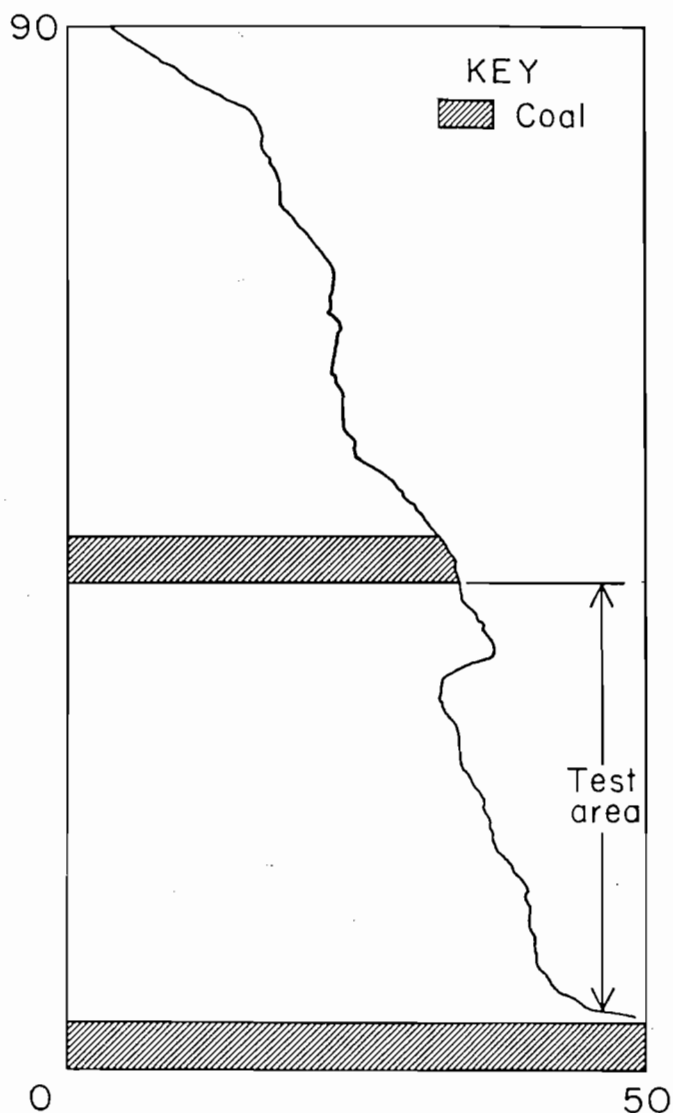


FIGURE 5.—Profile from stereo analysis, no additional delays. (Numbers indicate distance in feet.)

worked better than when the delay time was increased by 100 ms only. This was because of the better shearing action obtained with the 50-ms incremental increase. Since the test mine had one particular geology and used the Nonel initiating system for blasting, other mine sites and initiating systems may require a different adjustment of the delay time to obtain optimum results. The reader should also note that, for these tests, the highwall was located at the ends of the rows rather than the back row of holes.

The test results showed a general improvement in the highwalls even though drill-hole alignment and powder column

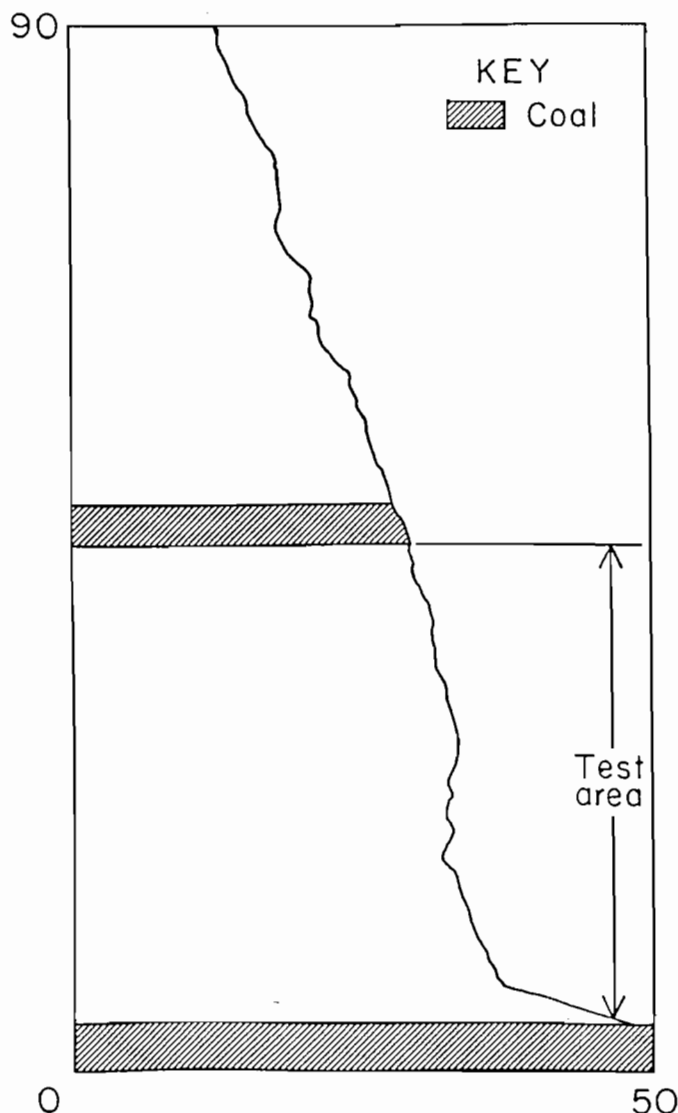


FIGURE 6.—Profile from stereo analysis, 50-ms-longer delays in highwall holes. (Numbers indicate distance in feet.)

heights varied, the geology changed continually (because the test site was a contour mine), and scaling practices varied. During the course of study, the safety officer and the mine operator both observed that the highwalls were noticeably improved and required less cleanup time by the dozers used to scale the highwalls at the test site. Precise evaluation of the blast effects on the highwalls proved to be difficult because of the variables mentioned above. However, after observing the results of 59 test blasts, the authors recommend an increased delay of 50 to 100 ms in the highwall holes, as described in the "Blast Designs" section.

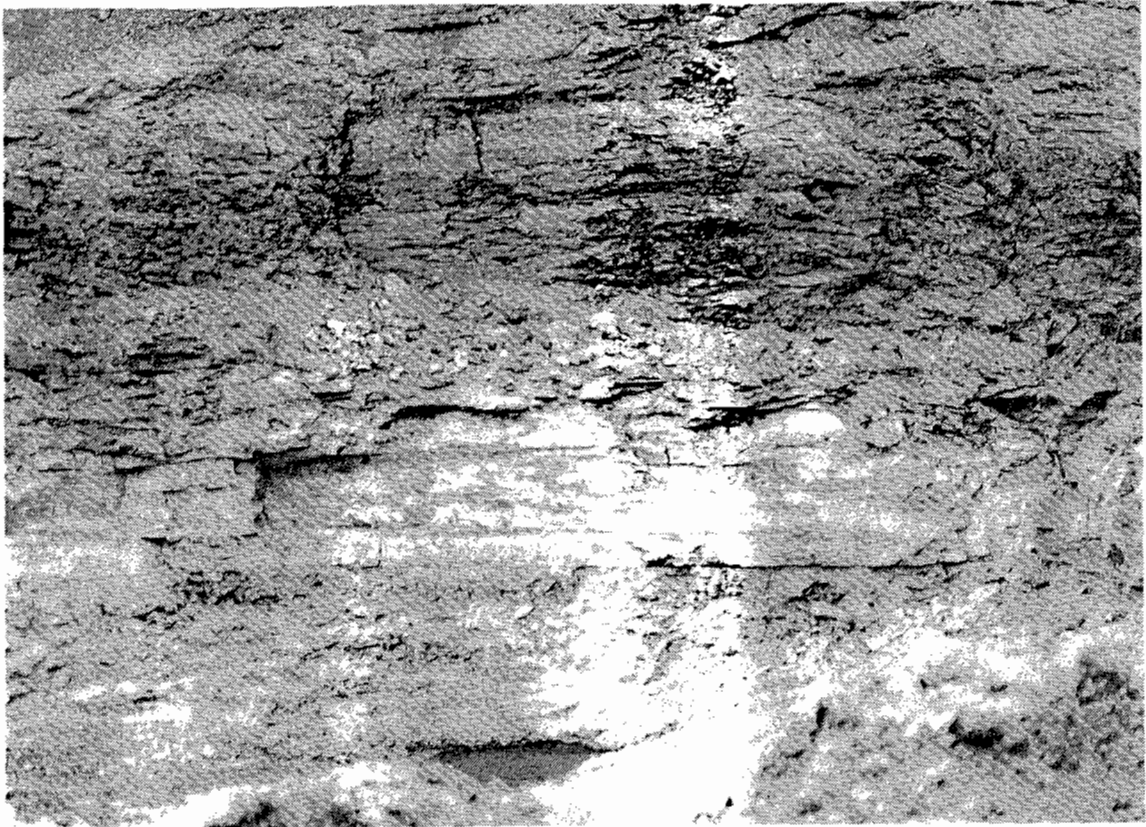


FIGURE 7.—Test highwall, no increase in delay time (blast design 1).

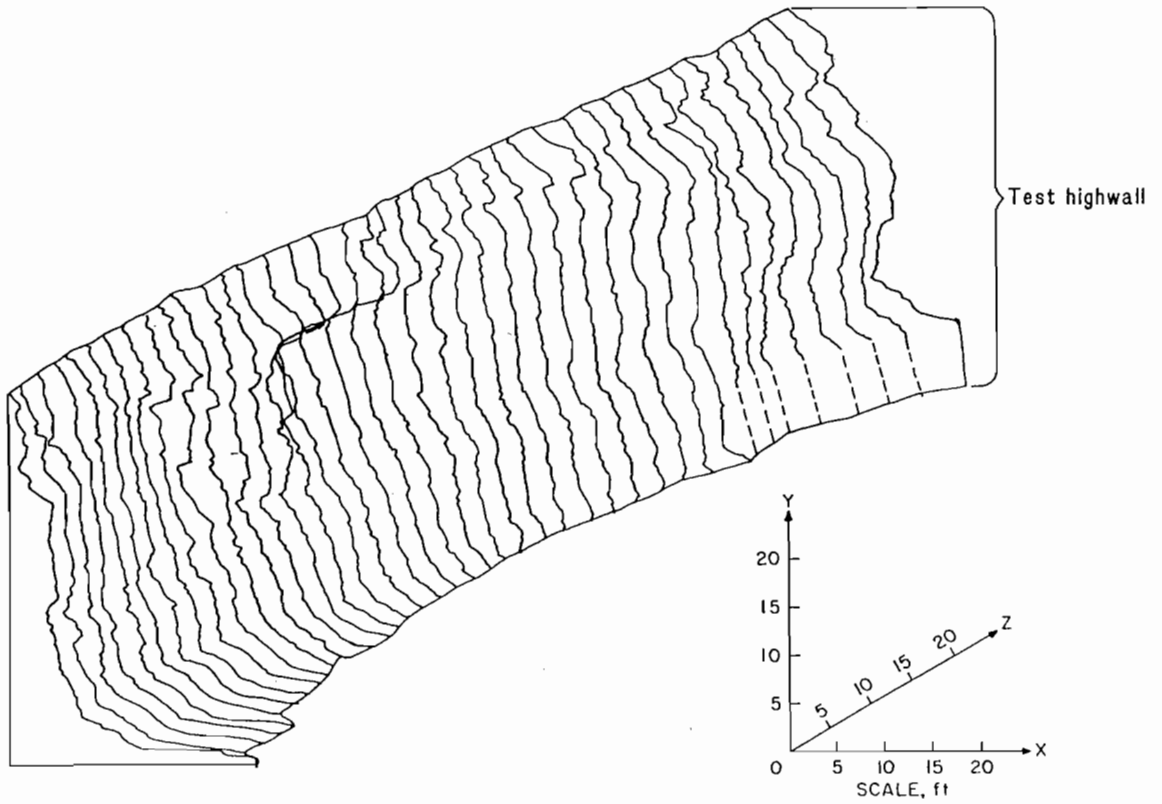


FIGURE 8.—Test highwall profiles, no increase in delay time.

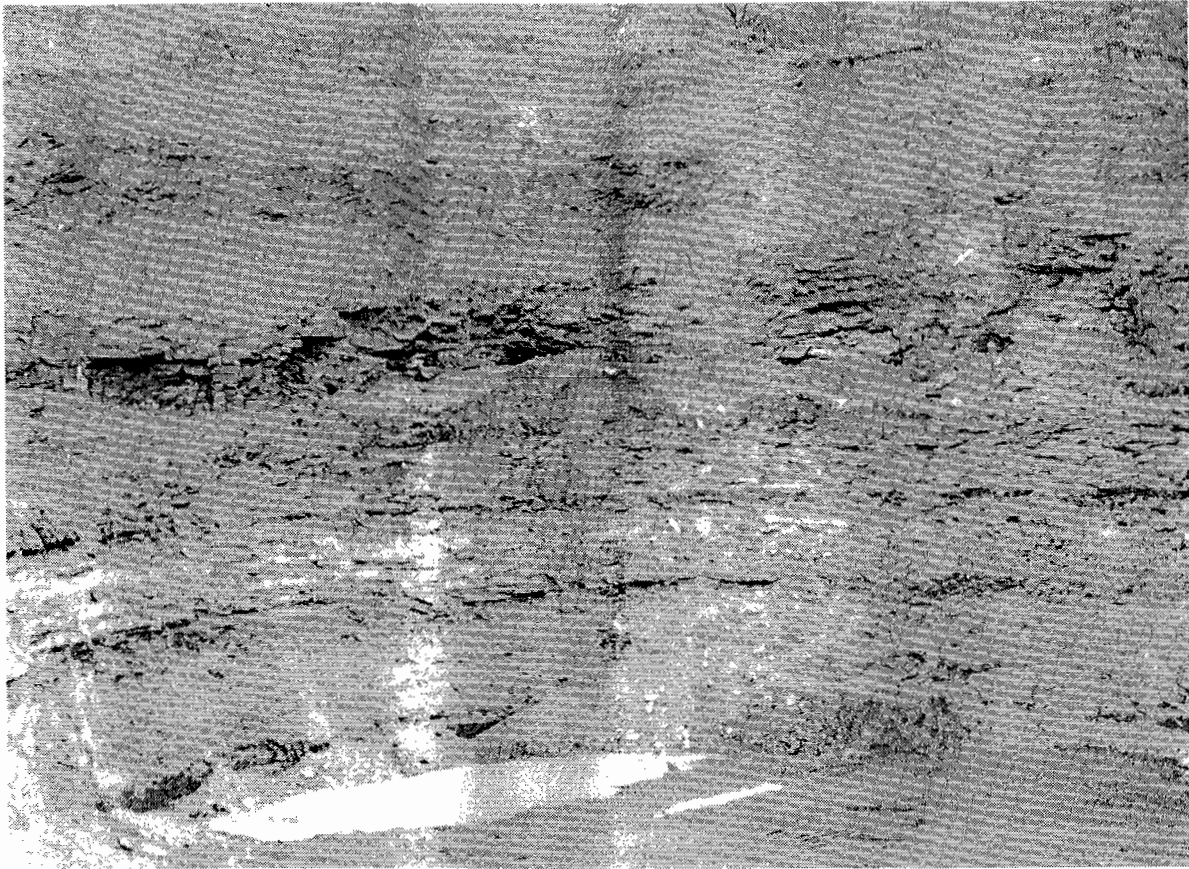


FIGURE 9.—Test highwall, 50-ms-longer delays in highwall holes (blast design 2).

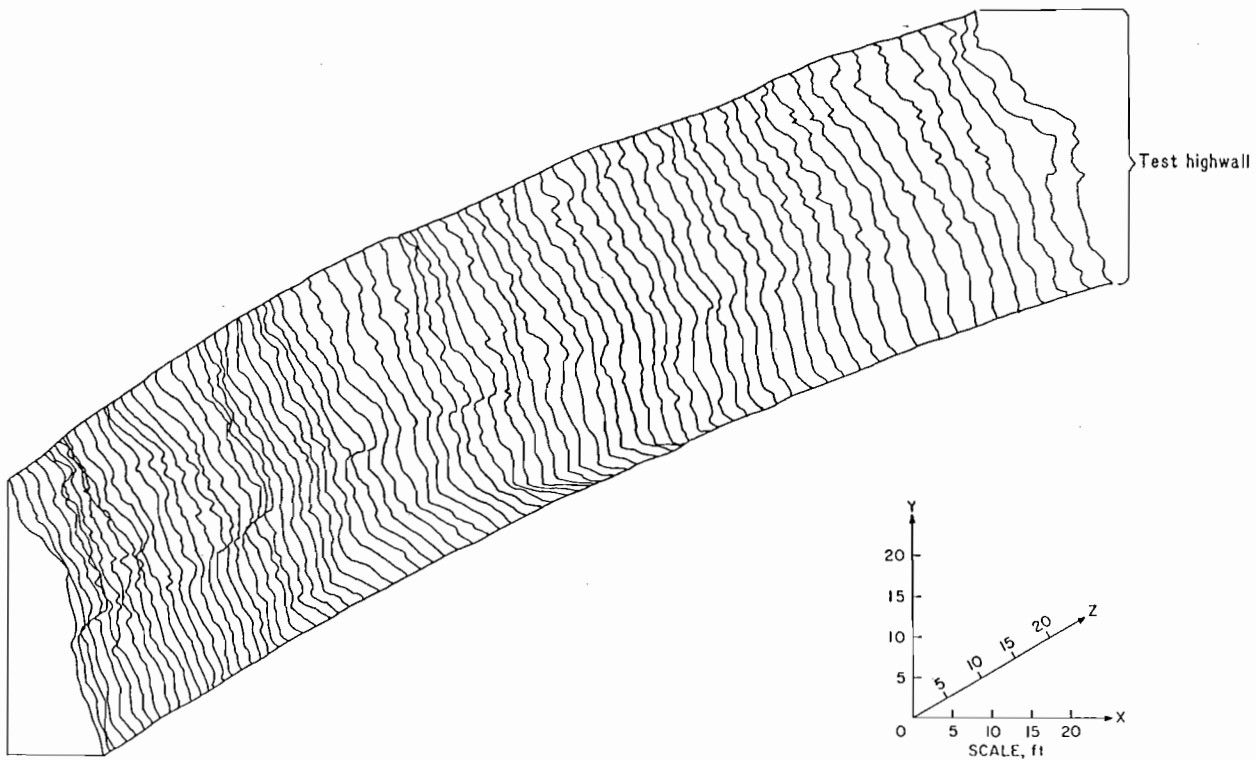


FIGURE 10.—Test highwall profiles, 50-ms-longer delays in highwall holes.

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EFFECTS OF BLAST VIBRATION ON CONSTRUCTION MATERIAL
CRACKING IN RESIDENTIAL STRUCTURES

By Mark S. Stagg¹ and David E. Siskind²

ABSTRACT

The Bureau of Mines studied the problems of blasting-vibration-induced structural response and cracking of low-rise residential structures in a series of research projects between 1976 and 1983. This paper summarizes the published Bureau findings and presents them from the point of view of the cracking and failure of the construction materials used for homes.

The damage data suggest that, for plaster and wallboard attached to the

superstructure, an increase in the rate of cracking is not likely to result from blasts generating vibrations of less than 0.5 in/s. Data on cracks in masonry walls suggest that blast-induced vibration levels of up to 3.0 in/s may be a threshold for local block-length cracks. However, additional data are needed to quantify vibration level effects necessary to generate stair-stepped cracks in masonry walls, which indicate loss of shear load capacity.

INTRODUCTION

Ground vibrations from blasting have been a continual problem for the mining industry, the public living near the mining operations, and the regulatory agencies responsible for setting environmental standards. Since 1974, when the Bureau of Mines began to reanalyze the blast damage problem, several field and laboratory studies have been conducted; the results of the most recent were published in RI 8969, in 1985 (1).³ The studies examined blast vibrations with respect to generation, propagation, structural response, cracking potential, instrumentation, and fatigue (2-4). A similar series of studies was conducted for airblast (5-6).

This paper summarizes the material on cracking of construction materials used in low-rise residential structures; the data are excerpted from two comprehensive Bureau vibration studies, RI 8507 on dynamic response and damage and RI 8896 on fatigue and long-term influences (3-4). Specifically, the paper discusses the cracking of plaster, wallboard, and masonry from blasting and other influences, giving an overall perspective to the blast vibration impacts as part of the total lifetime dynamic load for such materials.

CRACKING AND BUILDING PRACTICES

Current residential construction practices address basic human safety and not specifically the occurrence of

nonstructural minor or cosmetic cracks. Many of these practices were derived from allowable deflection criteria, in which material cracking potential is considered (7-9). However, cosmetic cracks do develop, and in 1948, Whittemore (10) discussed the lack of guidelines for vibrations of floors and pointed out that "deflection and vibration can be decreased, but only at an increase in price."

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

LONG-TERM CRACK RATES IN RESIDENTIAL STRUCTURES

PREVIOUS STUDIES

Structures crack naturally over time. Holmberg (11) analyzed blasting inspection reports to estimate a crack rate for apartment buildings in Sweden. Two apartment buildings were inspected for cracks three times between 1968 and 1980. The number of observed cracks is plotted as a function of time in figure 1. An average of 12 to 13 new cracks per year occurred for these particular structures. Holmberg did not report any specifics on the building construction, although concrete is a reasonable assumption.

The crack rate depends upon the type of structure. Rates for 11 wood frame houses that were subjected to 26 weeks of sonic booms and 13 weeks when there were no booms, as reported by Andrews (12), are listed in table 1. Crack rates at homes 1 through 4, which were studied during both periods, were generally lower during the 13-week nonboom period, which is similar to Bureau findings discussed later. The investigators also found evidence of the possibility that relative humidity and the number of sonic booms may together have had an effect on the occurrence of cracks.

The rates of 1.4 to 23 cracks per week during the nonboom period are quite high compared with the rate observed by Wall

(13) in a study of 43 single-story concrete block houses over a 26-week period; he reported a crack rate of 2.5 cracks per day for the 43 houses (<1 crack per week per house).

The large variation in the crack rates reported in the separate studies by Holmberg, Andrews, and Wall is indicative of the wide variation of susceptibility of houses to cracking. The rates ranged from near zero to 23 cracks per week. (The yearly rate reported by Holmberg indicates a cracks-per-week rate of less than one.) None of the investigators reported rates of zero. The large differences in the rates reported are partially a result of the difficulty of defining "cracks". For example, in Wall's report, shrinkage cracks were ignored, and only new cracks in the moderate (easily distinguishable) range were reported.

These data point out that when months pass between preblast and postblast inspections, any postblast inspection is likely to find some new cracks that are the result of natural aging.

BUREAU LONG-TERM FATIGUE STUDY

Blast effects on long-term crack rates were monitored over a 2-yr period at a Bureau of Mines test house (4). Bureau researchers developed two types of data in terms of the expected damage mechanisms: (1) fatigue damage from accumulated exposure, assessed by periodic inspections, and (2) triggering effects of discrete blast events assessed by inspections immediately before and after blasts, where the strains from blasting are added to already existing environmental strains. Researchers found that long-term repetition of the low-level blasts (peak particle velocity <0.5 in/s) produced no significant effect; however, blasts with velocities greater than about 1.0 in/s were associated with higher cracking rates, as shown in table 2.

The crack rate, or number of new cracks per inspection, along with the number of blasts that produced ground vibrations greater than 0.50 in/s and greater than 1.0 in/s, is shown in figure 2. Sixty

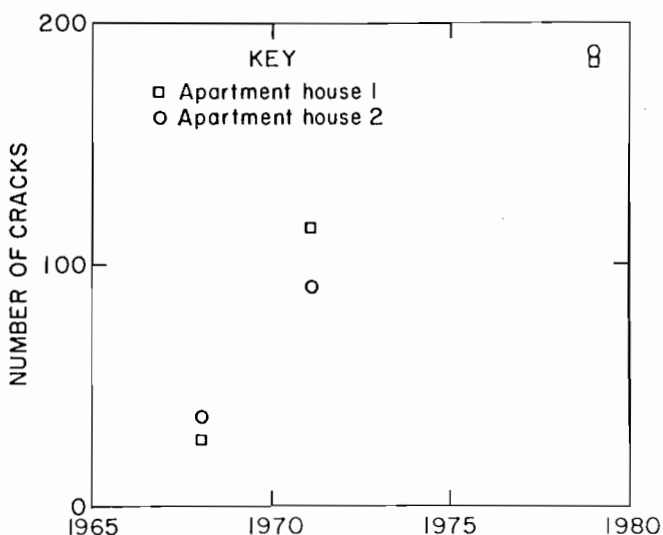


FIGURE 1.—Building age versus crack occurrences, after Holmberg (11).

TABLE 1. - Crack rates for houses subjected to sonic booms, after Andrews (12).

House	Number of stories	Area, ft ²	Foundation	Age, yr	Finish		Occupied	Number of cracks per week	
					Interior	Exterior		Boom period	Nonboom period
1...	1	1,560	Concrete slab.	5	Wallboard..	Brick...	Yes..	3.7	1.9
2...	2	1,750	...do.....	New	...do.....	...do...	No...	8.2	3.3
3...	1	1,470	...do.....	8	...do.....	...do...	No...	8.8	1.5
4...	1	1,160	Concrete stem wall.	18	...do.....	...do...	No...	6.1	1.8
5...	2	2,870	Masonry stem wall.	>50	Plaster and lath.	Asbestos siding.	No...	NM	23
6...	1	1,100	Concrete stem wall.	25	...do.....	Stone...	Yes..	NM	2.6
7...	1	1,090	...do.....	30	Lath and wallboard.	Wood lap	Yes..	NM	1.4
8...	1	1,280	...do.....	30	Plaster and lath.	Brick...	Yes..	NM	3.3
9...	2	2,000	Masonry stem wall.	40	Paper on plaster and lath.	Wood lap	Yes..	NM	3.0
0...	2	2,370	Concrete stem wall.	35	Plaster and lath.	...do...	Yes..	NM	14
1...	1	1,330	Concrete slab.	8	Wallboard..	Brick...	Yes..	NM	2.2

NM Not measured.

TABLE 2. - Crack versus vibration (4)

Shot level, in/s	Cracks per week	
	Without corner	Total
0.5.....	0.28	0.84
0.5, <1.0.....	.33	.89
1.0.....	1.0	1.8

shots had levels between 0.5 and 1.0 in/s, while 48 shots had levels above 1.0 in/s. Some of the crack rates shown in figure 2 include small hairline corner cracks, and some do not. The majority of corner cracks occurred in the first 8 months. Cracks were found in nearly every corner in the house, but were ignored until inspection period 15. Then it was decided to observe them rigorously despite their minuscule size. Corner cracks are an inevitable consequence of the curing of the tape compound and are enhanced by dynamic strains induced by

human activity. The data that exclude corner cracks are more realistic indications of blasting influences for homes other than new construction, i.e., within 6 months.

Differences were found in the number of cracks observed by the two teams of inspectors, Vibration Measurement Engineers (VME) and Bureau personnel, during periods 1, 15, and 36. The most pronounced difference was for period 15. The decision to include small corner cracks was made after VME had completed its inspection for that period but before the Bureau had completed its inspection for period 15. Other than for that period, differences in the number of cracks observed were an inevitable consequence of the difficulty of observing hairline (0.01- to 0.1-mm) cracks. Periods 1, 15, and 36 were omitted in calculations of crack rates. However, periods in which there were unusual external influences,

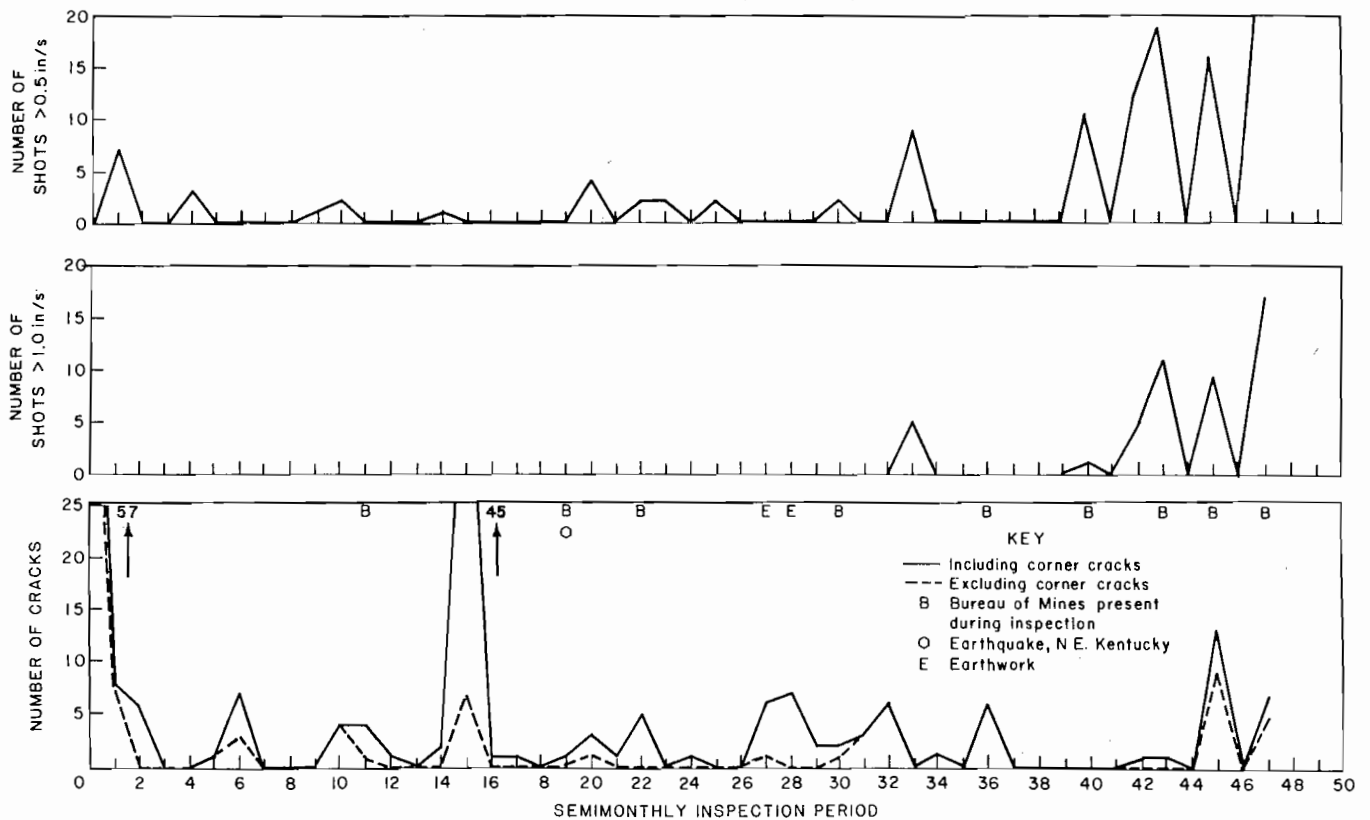


FIGURE 2.—Number of cracks and blasts >0.50 in/s and >1.0 in/s versus inspection period.

including an earthquake and soil removal by a scraper 40 ft from the test house, were included. The self-triggering seismograph recorded a 0.06 -in/s vibration for the scraper activity but did not trigger during the earthquake.

The increase in crack rate with ground vibration level indicates that the blasting produced a triggering strain, at

about 1.0 in/s. The low crack formation rates reported are reasonable since the test house was new, showed no differential settlement, and was not regularly occupied. These conditions resulted in low rates of natural crack formation, which allowed a few blast-related cracks to significantly affect crack formation rates.

CONSTRUCTION MATERIAL CHARACTERISTICS AND CRACKING

Cosmetic cracks result when a dynamic-induced strain (blast vibration or other transient vibration) added to a preexisting strain (static load) exceeds the strain level necessary to initiate a crack. Differential foundation settlement, excessive structural loads, and material shrinkage all induce strains that can produce random and/or patterned cracking. For analyzing blasting effects, these strain-inducing forces are considered static and the resulting strains are called "prestrains."

Stress-strain curves are used to describe response of materials under load

up to failure (cracking). Most materials, including masonry, plaster, and wallboard, respond linearly up to the initial yield point. A linear response means that deformation (strain) is directly proportional to load (stress). Beyond initial yield, plastic deformation or creep occurs until ultimate failure (fig. 3). The yield point damage is often not visually noticeable because of limited naked-eye resolution of 0.01 to 0.1 mm, particularly in textured surfaces such as masonry.

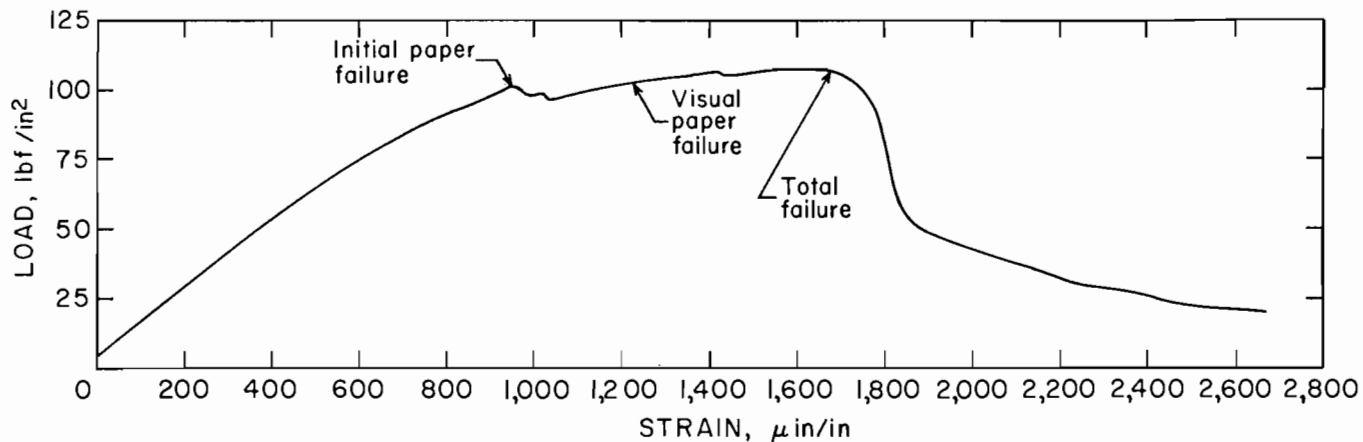


FIGURE 3.—Tensile stress-strain curve for 1/2-in-thick wallboard.

PLASTER

Plaster was not studied extensively because of its widespread replacement by wallboard for modern construction. However, many of the older homes analyzed for RI 8507 (3) were plastered and provided some insight into cracking potential. Also, wallboard is a gypsum plaster faced with paper on both sides. Tests run on stripped wallboard are suggestive of plaster failure (3). Of all construction materials, plaster is considered most susceptible to damage and exhibits fatigue at stress levels less than 50 pct of the static failure level (14).

WALLBOARD

The Bureau studied wallboard cracking both in the laboratory and as part of the fatigue study of the test house (4). For wallboard in the test house, researchers found threshold cracks occurring primarily in the wall corners and around nail heads. They found for wallboard--

1. The gypsum core failed at strains of about 350 μ in/in in tension and at about 1,000 μ in/in in bending, based on the nonlinear response points.

2. For visible cracking, paper failure was the controlling factor. Its nonlinear response point occurred at strains of 1,000 to 1,200 μ in/in (fig. 3). However,

visual observation of buckling or cracking was not possible until a slightly higher strain level was reached.

3. Strain rate seemed to affect ultimate or total failure, but the paper yield point was relatively constant. This allows comparison of various loading factors (e.g., blasting versus other activities and environmental factors).

4. Once the wallboard cracked, cyclic opening and closing of the crack of up to 0.1 mm was observed. These movements were unaffected by blasting activities.

5. Data on cyclic loading behavior of wallboard are limited, but results of tests on wood products indicated that fatigue effects can occur at stress (or strain) levels equivalent to 50 pct of static failure conditions, but over 100,000 cycles are required.

MASONRY

Bureau researchers also studied the cracking of concrete block walls, both at the test structure with its full-size basement and through a series of tests in cooperation with the National Bureau of Standards (NBS) in Gaithersburg, MD (4, 15). Generally, two types of cracks, local and steplike, were identified. Local block-length cracks less than 0.2 mm wide were difficult to discern from existing mortar joint separations and are usually not observed by homeowners. Steplike masonry cracks transverse the

wall along the mortar joint interface and, over time, open beyond 0.2 mm in width.

Previous work by Cranston (16), Green (17), and Wroth (18) noted that all brick walls have small, 0.1-mm cracks upon completion. Green stated that 0.1-mm cracks are difficult to see and "therefore, do not cause concern." As reported by Woodward (15), local cracks opened and closed throughout the cyclic and monotonic in-plane shear tests of a 5- by 5-ft concrete block wall. It was not until a steplike crack propagated the length of the wall specimen that shear load failure occurred.

Although findings by Bureau researchers on masonry failure provide some insight, further work at NBS on torsion and out-of-plane loading is recommended. Key findings for tests with masonry are given below.

1. Observations of tensile cracks at strain-monitored sites showed that such cracks were first detected visually at strain levels well above the first non-linear response point because of naked-eye limitations (~0.01 to 0.1 mm).

2. Strains read at the threshold of visual cracking using different gauge lengths gave different overall strain readings as illustrated below.

Based on the equation $\epsilon = \frac{\Delta l}{l}$,

$$\frac{0.01 \text{ mm}}{13 \text{ mm}} = 770 \text{ } \mu\text{in/in,}$$

but
$$\frac{0.01 \text{ mm}}{150 \text{ mm}} = 67.0 \text{ } \mu\text{in/in,}$$

where 13 and 150 mm are gauge lengths, and the visible crack width is 0.01 mm. Because strain gauge readings can be misleading, crack growth is best described in terms of displacement.

3. Local-site strains across the wall vary considerably from global strains. For in-plane shear failure, global strain is measured or calculated across the wall diagonally.

4. Local cracks can occur at low global strains, and global assessment of these cracks is not recommended. But, for the assessment of steplike cracks that propagate across the entire wall, the global strain approach appears reasonable.

5. Global failure strain levels for steplike cracks are not available. Limited testing to date has shown that in-plane shear failure may not occur in homes because of the relatively light vertical load available to prevent rotation from the shear couple and at least a partial conversion of the shear to tension.

6. For cosmetic cracks that do not affect load-carrying capacity, a crack-width criterion has been proposed (17). However, the acceptability of crack widths varies with material. For concrete, 0.25 mm is the limit of acceptability (19), while 1 mm is the limit of acceptability for brickwork (18).

FACTORS CAUSING STRUCTURE RESPONSE, STRAIN, AND CRACKING

Bureau researchers studied structure responses and cracking associated with blasting vibrations in an investigation involving a relatively few measurements at each of a wide variety of residential-type structures (3). Following this, fatigue from repeated loading of one house over a long period of time was studied (4). For both efforts, measurements were made of wall, floor, and racking responses, and observations of damage were made that could be correlated to specific vibration events. A significant part of the work was done near large surface coal

mines with thick soil overburdens and large-diameter blastholes, cases which had not been studied previously. In all, about 900 shots produced useful data on structural responses and damage potential from blast vibrations.

ENVIRONMENTAL STRAINS

Houses are subject to a variety of dynamic loads, in addition to static or slightly variable loads from settlement, soil changes, and aging. Among the dynamic forces considered significant are

daily and annual temperature and humidity cycles, wind, and human household activity. Bureau researchers monitored the weather and inside environment during the 2-yr test period and, in more detail, for short periods. For one test, they took readings at 3-h increments for a 2-day period, simultaneously measuring strain at site K₂, over a major doorway (fig. 4). Because there were at least four factors influencing the strain, researchers used multiple linear regression analyses. Maximum strains from daily environmental changes were found to be a significant fraction of those needed for wallboard core failure or paper cracking. The maximum strain observed at K₂ was +385 $\mu\text{in/in}$ or 39 pct of failure. The total maximum strains calculated from the correlation equation, assuming the worst case for each of the factors, are +675 to

-817 $\mu\text{in/in}$, or up to 82 pct of failure. "Failure" is defined as the strain level of 1,000 $\mu\text{in/in}$, found to produce wall-board cracking as previously discussed (fig. 3).

HUMAN-ACTIVITY-INDUCED STRAINS

Activities within the home can produce significant vibration and strain in local structural members (3-4). In severe cases, such as a hard door slam, the entire superstructure resonates producing strains in every wall, corner, and floor. By contrast, nail pounding produces a strong response only on the wall affected. Strains range up to about 100 $\mu\text{in/in}$, with typical values being 50 $\mu\text{in/in}$ in critical areas over windows and doorways.

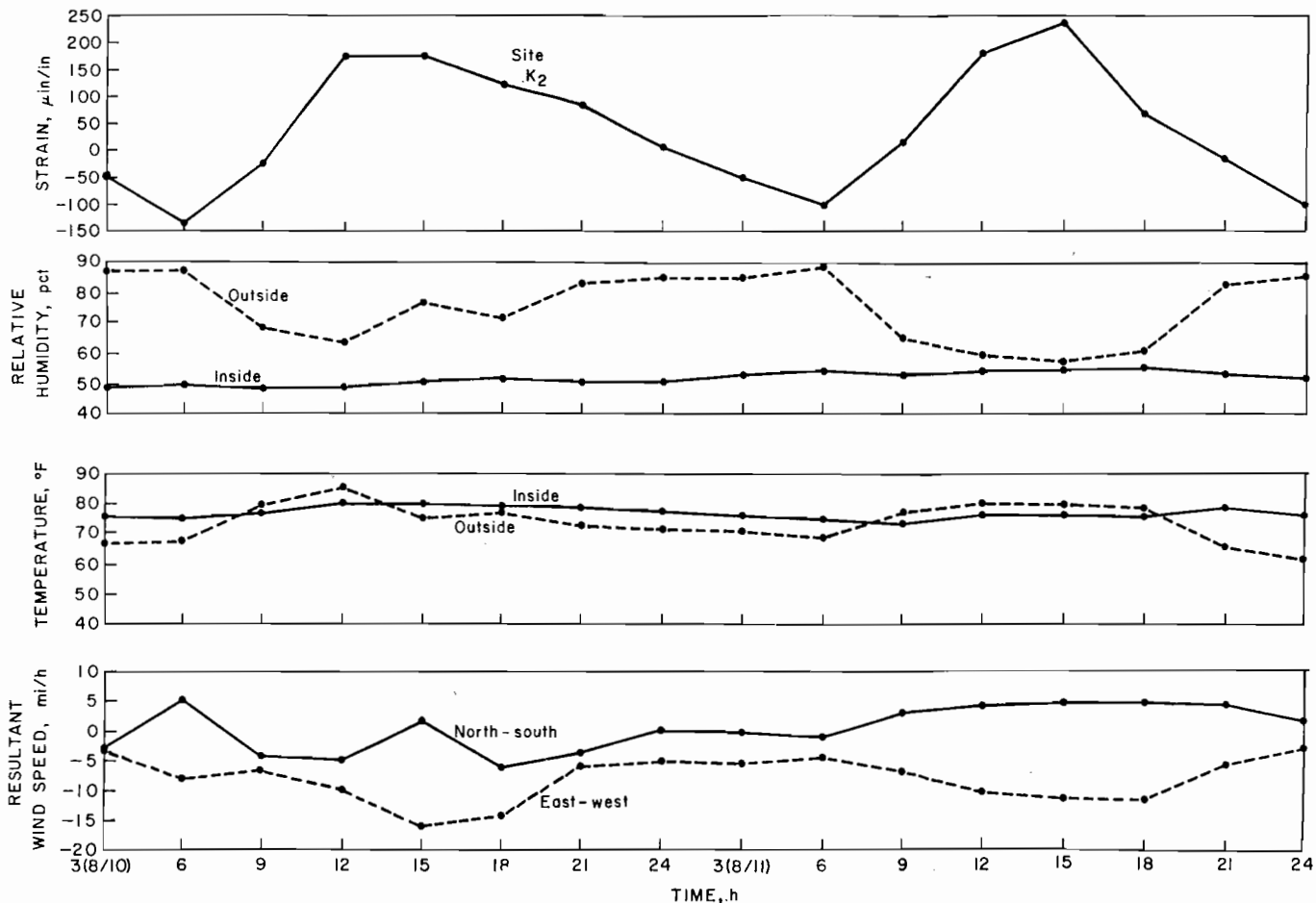


FIGURE 4.—Wallboard joint strain and environmental factors versus time, site K₂ over a doorway.

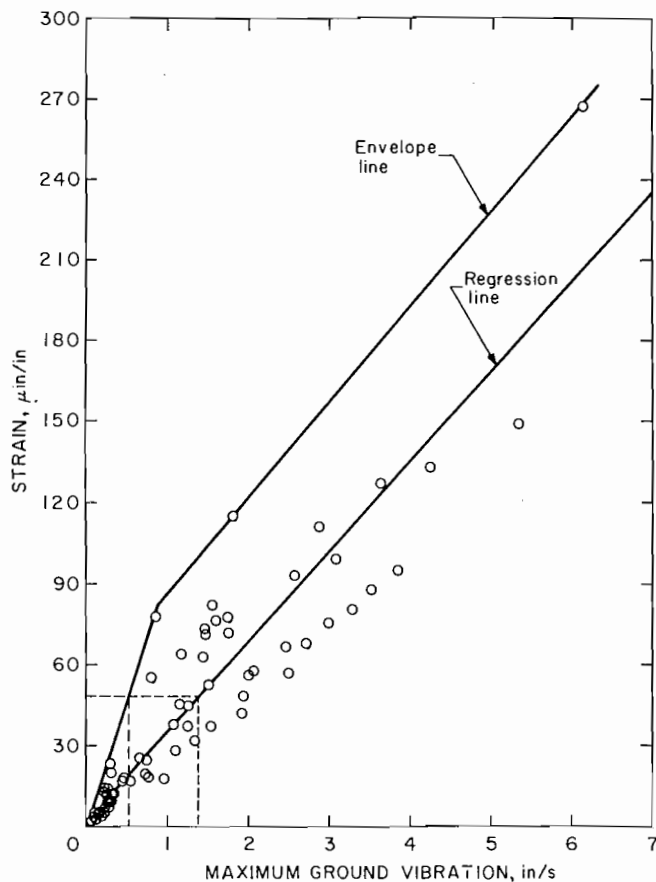


FIGURE 5.—Plaster and wallboard strain versus maximum ground vibration at site S₁ over a doorway.

BLASTING-INDUCED STRAINS AND COMPARISONS

Blasting responses and strains in residential structures were reported in detail in Bureau RI's 8507 and 8896 (3-4). An example of blast-vibration-induced strains from the fatigue study reported in RI 8896 is shown in figure 5. Structure vibration responses can be transitional, torsional, vertical uplift, or at times a combination of all three. In blasting, both the superstructure and foundation are typically affected. Non-blasting causes of vibration and strain act only on the superstructure, except for slowly acting soil changes and settlement. Because initial damage involves cosmetic cracks on superstructure interior walls, it is appropriate to compare superstructure strains from blasting and other sources (table 3). These comparisons are only approximate. A given vibration level does not always produce the same strain even at a single monitoring point, much less throughout the structure, probably because of different response modes for different blast angles, and wave characteristics.

TABLE 3. - Comparison of strain levels induced by daily environmental changes, household activities, and blasting (4)

Loading phenomena	Site	Induced strain, $\mu\text{in/in}$	Corresponding blast vibration level, ¹ in/s
Daily environmental changes	Bedroom midwall	149	1.2
Do.....	Over doorway...	385	3.0
Household activities:			
Walking.....	Over a window..	9	<.03
Heel drop.....	...do.....	20	.03
Jumping.....	...do.....	37	.28
Door Slam.....	Over doorway...	49	.50
Pounding a nail.....	Over a window..	89	.88

¹Vibration velocities are based on highest observation strains for a given velocity. Use of mean or "typical" values from regression analysis gives velocities considerably higher. For example, the door slam produces a level of strain typically observed at 1.44 in/s ground-measured particle velocity. See figure 5.

OBSERVED CRACKING FROM BLASTING

As discussed earlier, environmental factors induce most of the strain necessary for the generation of cracks triggered by household activities or blasting. Crack rates did not increase until blast vibration levels rose above normal threshold levels of 1.0 in/s. It is not surprising then that both wallboard and plaster cracked at low vibration levels, even though failure strain levels for wallboard are about 3 times those of plaster.

In reviewing both past and newly available data on dynamic vibration response, researchers noticed irregular and sometimes high-amplitude responses when the vibration frequencies matched structure resonances (fig. 6). A similar effect, noticed for the cracking data, was one of the most significant findings in RI 8507 (3). Consequently, coal mine and quarry production blasts that are typically 10 to 25 Hz produce a greater damage risk than smaller scale blasts often used for construction, excavation, and secondary blasting.

In a departure from earlier analyses and reports, the following review quantifies damage separately for each of the three major construction materials: plaster, wallboard, and concrete block. The reader is directed to the original reports for procedure and analysis details (3-4 15).

PLASTER CRACKS

Threshold and minor cracking data are summarized in figure 7 for pre-1975 studies and in figure 8 for recent Bureau research. All these data have been previously published in RI 8507 (3) and RI 8896 (4). However, in a departure from the earlier reports, these figures identify each data point as to source, degree of cracking damage, and type of material involved.

Langefors (23) presents the only significant amount of high-frequency data. These data suggest that vibration levels as high as 4 in/s may be safe for frequencies above about 70 Hz. In the descriptions of damage, in RI 8507's

table 10 (3), Langefors did not separate the cracking and fall-of-plaster cases. Dvorak's study produced observations of cracking at some of the lowest peak particle velocities, and questions have been raised about data reliability. However, Dvorak used the same seismic monitoring system as Langefors. Dvorak's brick structures were likely different from Langefors' unspecified structures (probably concrete), and the vastly different measured frequencies are indicative of a soil versus rock foundation. The lowest vibration level at which cracking was observed was 0.5 in/s with Dvorak's data and about 0.7 in/s without.

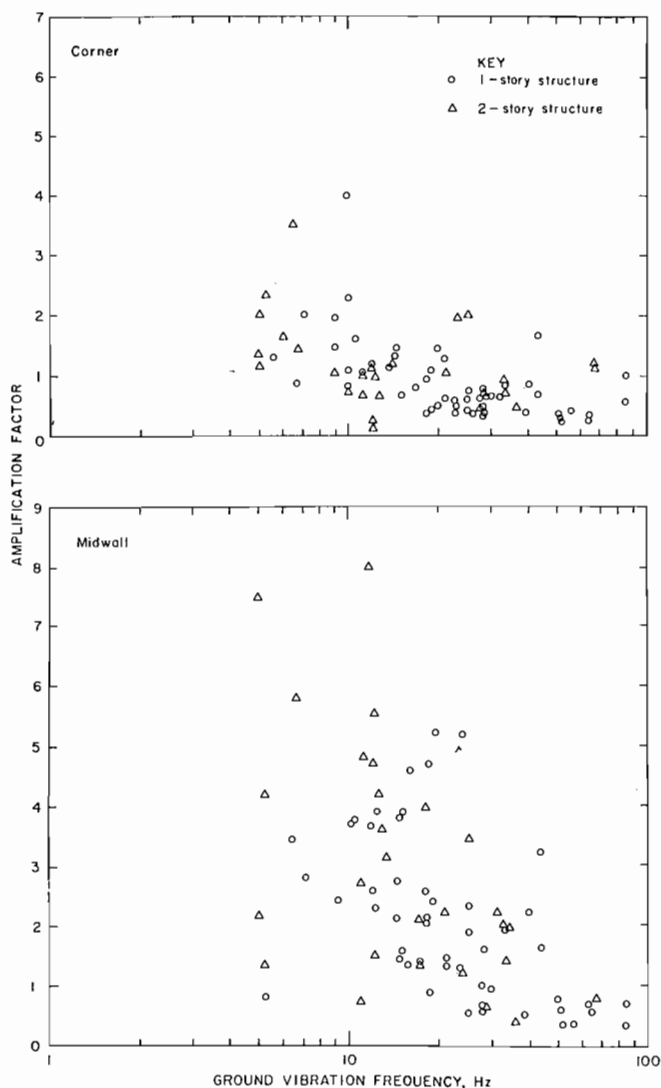


FIGURE 6.—Corner and midwall amplification factors (3).

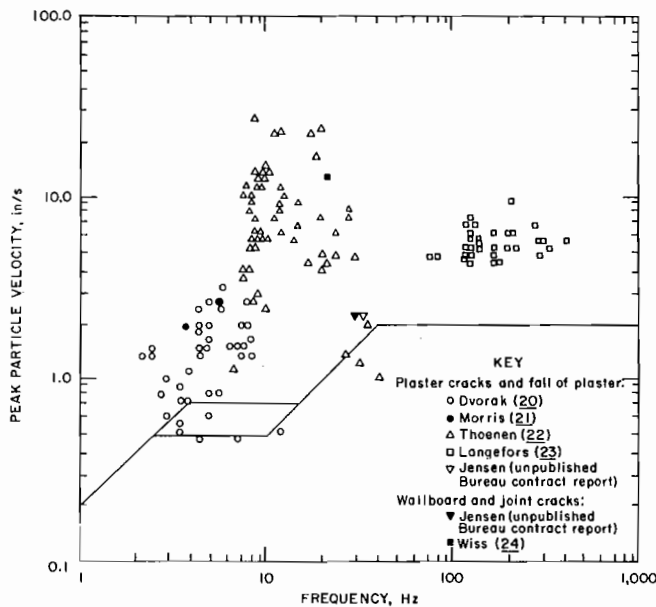


FIGURE 7.—Velocity and frequency levels for threshold plaster and wallboard cracking, pre-1975 studies.

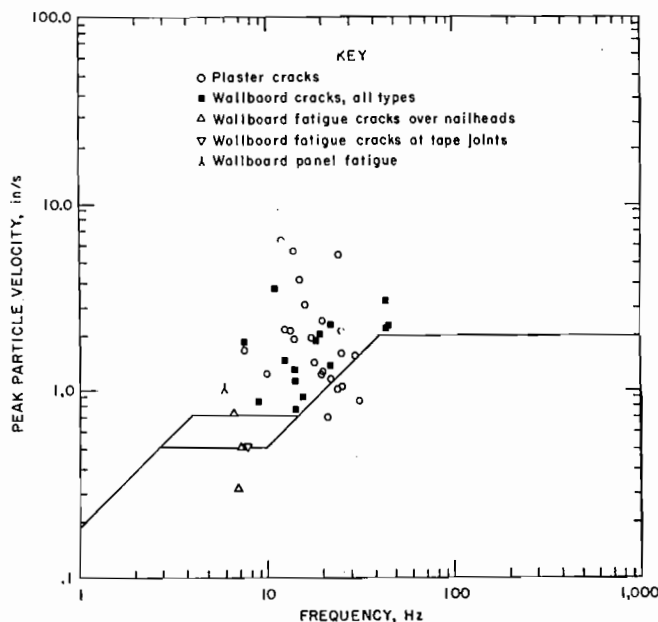


FIGURE 8.—Velocity and frequency levels for threshold plaster and wallboard cracking, recent studies (3-4).

WALLBOARD CRACKS

The state of cracking of wallboard is hard to identify because the interior plaster core will crack long before any surface effect is visible. Visible cracking of paper covering occurs at

strains about three times those required for core failure. Wallboard cracks are also influenced by how well panels are attached to the superstructure frame. Not being structural elements, they are not always put under in-plane stress when the frame flexes. The core around the nailhead is, at best, partially crushed upon attachment to the studs, and when the studs are uneven major core cracking can occur. The response from superstructure vibration is additional wallboard core crushing around the nailheads, resulting in a "loose" attachment.

At the test house, it was observed that cracks developed primarily at the plastered joints at wall corners and in plaster covering coating over nailheads (table 4). The high rate of naturally occurring cracks was caused primarily from curing of the tape compound. As the tests on the structure continued, a decrease of natural frequency of about 20 pct, e.g., 7.5 to 6 Hz at one location indicated a loss of rigidity and general flexure-induced loosening (4).

The lowest levels of observed blast-vibration-induced cracking occurred at a wall corner as crack extensions and when a new crack was observed beneath a window, at amplitudes of 0.79-1.1 in/s (fig. 8).

Fatigue-induced cracks were observed at 0.3 to 1.0 in/s. However, this cracking required a large number of vibration cycles, such as over 50,000 at a 0.5-in/s equivalent ground vibration. This equates to decades of typical blasting with one blast per day producing 10 cycles per blast.

MASONRY CRACKS

Cracks produced in block masonry walls by blasting are given in figure 9 for past work and figure 10 for recent Bureau studies (3-4). Most cracks observed were local, typically shorter than one block length, and about 0.2 mm in width. Cracks of this magnitude were observed from blast vibrations up to 6.2 in/s and were not of concern, being indistinguishable from normal construction and shrinkage effects. Their observation is difficult, which accounts for the high

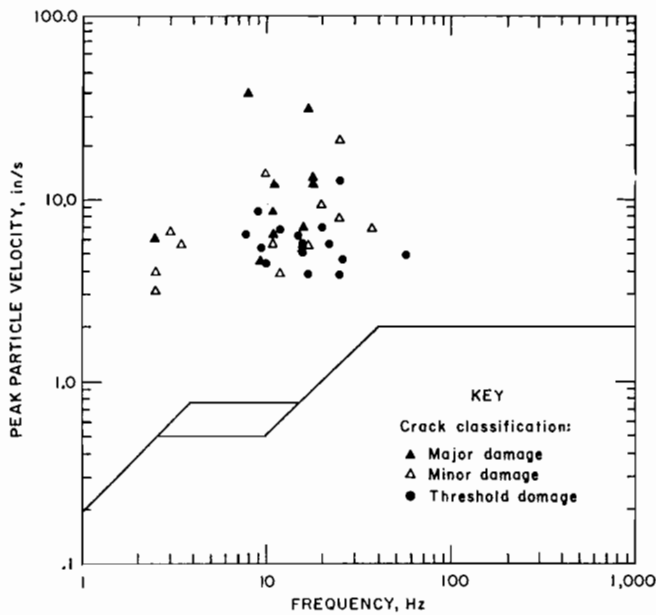


FIGURE 9.—Velocity and frequency levels for local masonry wall cracks, pre-1975 studies.

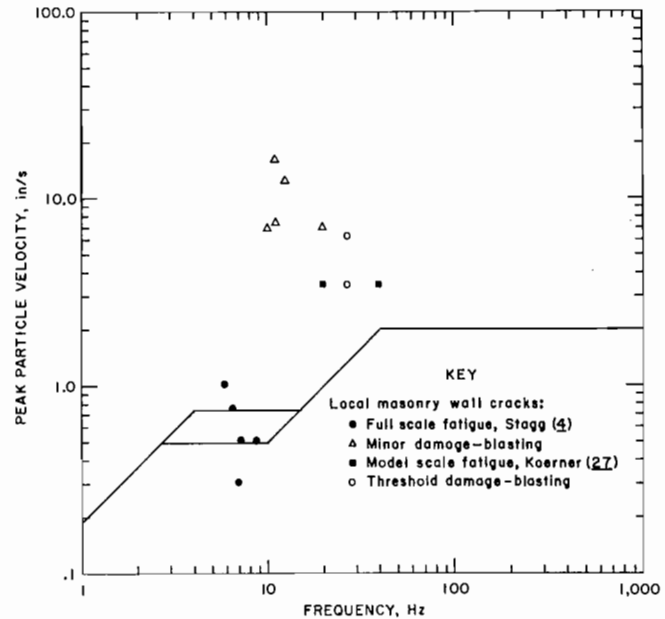


FIGURE 10.—Velocity and frequency levels for local masonry wall cracks, recent studies (3-4).

TABLE 4. - Wallboard cracks observed in fatigue test house (4).

Material	Initial cracks, before testing	Cracks developed during testing		Blasting level, in/s	Mechanical shaker tests ¹	
		Naturally occurring	From blasting		Cracks developed	Number of cycles at cracking
Taped corners...	39	35	5	0.88-3.5	ND ²	NAp
Nailheads.....	5	4	3	1.8 -2.2	>3	56,000, 339,500
Taped joints....	2	6	ND	NAp	1	56,000
Wallboard.....	3	6	ND	NAp	1	361,500

NAp Not applicable. ND None detected.

¹Shakers run at resonant frequency at equivalent vibration levels of 0.3 to 1.0 in/s.

²Corners almost completely cracked before shaker study.

TABLE 5. - Masonry wall mortar joint cracks observed in fatigue test house (4)

Material	Initial cracks, before testing	Cracks developed during testing		Blasting level, in/s	Mechanical shaker tests ¹	
		Naturally occurring	From blasting		Number of cracks	Number of cycles at cracking
Brick.....	20	28	7	3.4 -6.2	2	229,000, >293,500
Fireplace.....	21	11	1+	-6.9	2	56,000, >108,500
Block.....	NA	NA	5+	6.2 -6.9	3+	>339,500
Steplike crack.	ND	ND	1	.96-1.5	ND ²	NAp
Separation.....	ND	ND	2	6.9	1	>339,500

NA Not available--see text. NAp Not applicable. ND None detected.

¹Shakers run at resonant frequency at equivalent vibration levels of 0.3 to 1.0 in/s.

²Existing steplike crack functioned as an area of stress relief.

number of naturally occurring cracks (table 5). Also, these local cracks became more apparent during the cyclic test. Differential motion along the block interfaces was easily observed during continued cyclic motion, which accounts for the low vibration levels, 0.3 to 1.0 in/s. However, in the test house, a blast vibration of 6.9 in/s produced a crack of significant magnitude, widening a crack beyond the width that was observed in the absence of a blast.

SHEAR LOAD FAILURE

Shear load failure of the basement wall of the test house was observed after four shots in one day. A diagonal steplike crack propagated in the southwest basement wall, starting at ground level and proceeding upward. When these four shots were detonated, their vibration levels (ranging from 1.0 to 1.5 in/s) were the highest recorded in the study up to that time. But because observation of cracks in masonry is difficult, it remains unknown whether blasting or other events caused this steplike crack. It is noteworthy that no additional steplike crack

propagations were observed across brick or block walls. The existing steplike crack functioned as an area of strain relief during shaker runs. Energy transmitted by the shakers into the superstructure and foundation was primarily dissipated in areas of previous cracking.

Observations were also made of chimney and brick veneer responses during cyclic shaker tests. The masonry walls were relatively stationary, with the superstructure cyclically bumping the chimney and a brick veneer wall near the roof line. Mortar joint cracks developed at the chimney-roof interface and horizontally across the brick veneer just above door height.

Crack data from Edwards and Northwood (26) do not specify crack widths. If these crack data correspond to observations exceeding 0.2 mm (excessive crack widths), it would suggest that cracks can occur at particle velocity levels of 3 to 7 in/s with no effect of frequency. Additional data are needed to qualify frequency effects and the generation of stairstep crack patterns across the wall signifying shear load failure.

CONCLUSIONS

Bureau studies of the response and cracking of low-rise residential structures from blasting indicated that cracking of plaster and wallboard is not likely below about 0.5 in/s peak particle velocity for the worst case of structure condition and typical vibration frequency.

This safe-level criterion also appears independent of the number of blasting events and their durations. Researchers also noticed that high strains are produced in structure walls by normal weather conditions, such as wind, temperature, and humidity cycling. Dynamic events such as door slams or blasting produce additional strain, which can trigger a crack in a structure already under strain. Human activities, such as door slams, can be equivalent to blast vibrations of up to 0.5 in/s. The vibration level of 0.5 in/s thus provides a minimum value of concern for the impact

of external transient vibrations on wood-frame, low-rise residential structures typical of those studied by the Bureau.

Data on the response and cracking of masonry walls from blasting indicated that local cracking (block-length) may not be noticeable until particle velocity levels are up to 3.0 in/s. However, additional research is needed to quantify vibration levels that promote the generation of stair-stepped cracks that propagate across the wall and reduce its shear load capacity.

The authors encourage, where possible, direct measurements or assessment of strains or loads on members likely to fail. Alternatively, estimates of responses should be based on realistic transfer functions relating measured vibrations and reasonably expected responses. In particular, applications beyond the scope of the original Bureau studies are to be done only with caution.

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BLAST VIBRATION MEASUREMENTS NEAR STRUCTURES

By David E. Siskind¹ and Mark S. Stagg²

ABSTRACT

Blasting near structures often involves vibration measurement to assess damage potential. Several methods of measurement are used worldwide; however, there is no consensus as to which methods are technically sufficient and yet practical for all situations.

The Bureau of Mines studied five placement locations for vibration transducers to determine the best method for monitoring blasting vibrations. The locations were--burial in soil next to the structure, attachment to the foundation at ground level, to the basement floor, or to a surface slab, and burial at a distance from the structure in undisturbed soil. Typical surface mine production blasts were used as vibration sources.

With the exception of the basement floor measurements and some of the distant measurements, waveforms were similar and amplitudes were generally within 10 to 30 pct of each other. The low-frequency part of the wave (5 to 10 Hz) was particularly uniform in measurements obtained at all five locations. Differences in peak values were mostly from minor shifts in phase of the high-frequency components, which are less significant to structural response and potential damage than the low-frequency waves. Shallow surface burial resulted in good signal detection and the least chance of mechanically induced error.

INTRODUCTION

The Bureau of Mines studied vibration measurement methods applicable to production blasting in surface mines (1).³ Blast vibrations are routinely monitored for one of two purposes: (1) to assess damage risk for nearby structures and (2) to derive predictive equations for vibration generation and propagation. Despite years of practice and several published studies on measurement, the industry has not adopted a uniform and consistent methodology. Occasionally, those monitoring blasts fail to obtain reproducible and accurate vibration records.

Three measurement methods are in common use: (1) direct attachment of the transducer to the foundation of the structure to be monitored, at or near ground level,

(2) shallow burial of the transducer in the soil next to the foundation, and (3) measurement on a nearby concrete slab such as a driveway or walkway. The specific practices followed are often based on convenience. However, at sufficiently low vibration amplitudes, all three methods will give similar results. The many factors involved, such as transducer shape and size, soil or ground strength, and density, make for varied measurement requirements. Past Bureau studies did identify the need to anchor, attach, or bury vibration transducers for blasts exceeding 0.2 G acceleration amplitude (2).

Because of problems of practicality, cost, and site access, it is not likely that a single measurement method will be universally adopted, or needs to be. The Bureau, therefore, evaluated the most common methods to determine which give accurate indications of vibration energy transferred into structures while also being representative of the blast as a vibration source (1).

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

Previous research on blast vibration monitoring has been concerned with stability and slippage (2-4), spiking in soil (5-6), partial resonances (7-8), impedance matching (9), Rayleigh wave depth effects (6-10), burial in soil, (9, 11-12), and measurement for response spectra (13-16). The overall conclusions from all these studies are that burial of transducers is desirable and that caution is required for other methods,

particularly at high accelerations (combinations of high velocity and high frequency).

The research described in this report involves side-by-side comparisons of vibration time histories from a series of surface coal mine production blasts. A more complete description of the study and related background was published in 1985 in RI 8969 (1).

EXPERIMENTAL PROCEDURES

Four structures were studied at three operating mines, two near St. Clairsville, OH, and one near Evansville, IN. All had concrete block or stone basements with 5- to 7-ft depths. Researchers monitored a total of 23 blasts, containing 22 to 2,200 lb per delay, at distances from 425 to 4,900 ft.

Monitoring methods varied among the sites because of differences of accessibility and suitability. Concrete slabs were near the structures at two sites; however, at the other two, the slabs were 22 and 60 ft away from the main structure. Direct comparisons were made as closely as possible, by simultaneous monitoring, with transducers at the following locations:

1. Buried in the soil with 2 ft of the foundation,
2. Mounted on the foundation wall at or near ground level, usually inside.
3. On the basement floor,
4. On a nearby surface slab, a walkway, driveway, or garage floor, and
5. Buried near the structure but removed from the influence of disturbed ground around the foundation, 72 to 100 ft distant.

Special measurements were made of vibration propagation velocity to examine the implications of depth-dependent vibration amplitudes. Clark had described the decrease of Rayleigh wave amplitudes with increasing depth (10). For situations when the blast vibration is dominated by the Rayleigh surface waves,

this could be significant. The decrease depends not simply on depth, but on depth compared to wavelength (fig. 1). Therefore, measurements were made of propagation velocity, allowing the calculation of wavelengths from

$$\lambda = c/f,$$

where λ = wavelength, ft,

c = propagation velocity, ft/s,

and f = frequency, Hz.

According to Clark, a deeper foundation "sees" a decreased amplitude vibration. This is also consistent with observed low-amplitude vibrations on basement floors (10, 16-17).

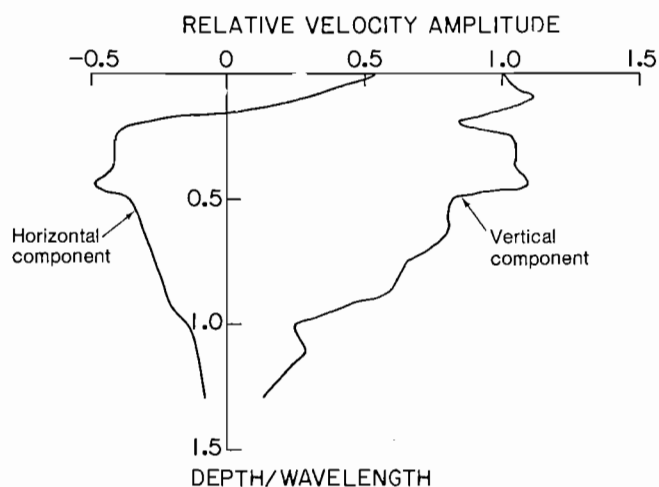


FIGURE 1.—Rayleigh wave amplitude profile versus depth, after Clark, Larson and Lande (10).

RESULTS

Waveforms were found to be very similar for measurements made near and on the structure, and also on a nearby surface slab. A set of vibration wave comparisons is shown in figure 2. Seven additional record sets are shown in RI 8968 (1). For structure response, the vibration's low frequencies (less than 20 Hz) are most critical. The low-frequency characteristics of the measured records were nearly identical, peak for peak and wiggle for wiggle. However, high frequencies present in the outside buried gauge records were absent inside, as the structure filtered the vibrations by its frequency-dependent response. These high frequencies provided minor changes in the waveform details and peak values. Records collected at distances from the house differ, mainly from phase changes between arriving vibration modes and travel-time shifts of energy from various parts of the blast.

Amplitudes varied more than waveform frequency characteristics. These amplitude differences of about 25 pct resulted from phase changes and minor amplitude differences from the more strongly affected higher frequencies. In terms of structure response, they are not significant. However, they do complicate monitoring by increasing the scatter in peak velocity values. Table 1 summarizes the peak vibration amplitudes. As expected, basement floors are consistently low, by about 25 pct. Most of the peak vibration amplitude differences in table 1 are from minor phase changes and variations in the relatively insignificant high frequencies. However, they do suggest the advantages of using some method of vibration analysis to compensate for these waveform differences, such as signal smoothing, weighting, velocity exposure, response spectra, or another frequency-dependent method.

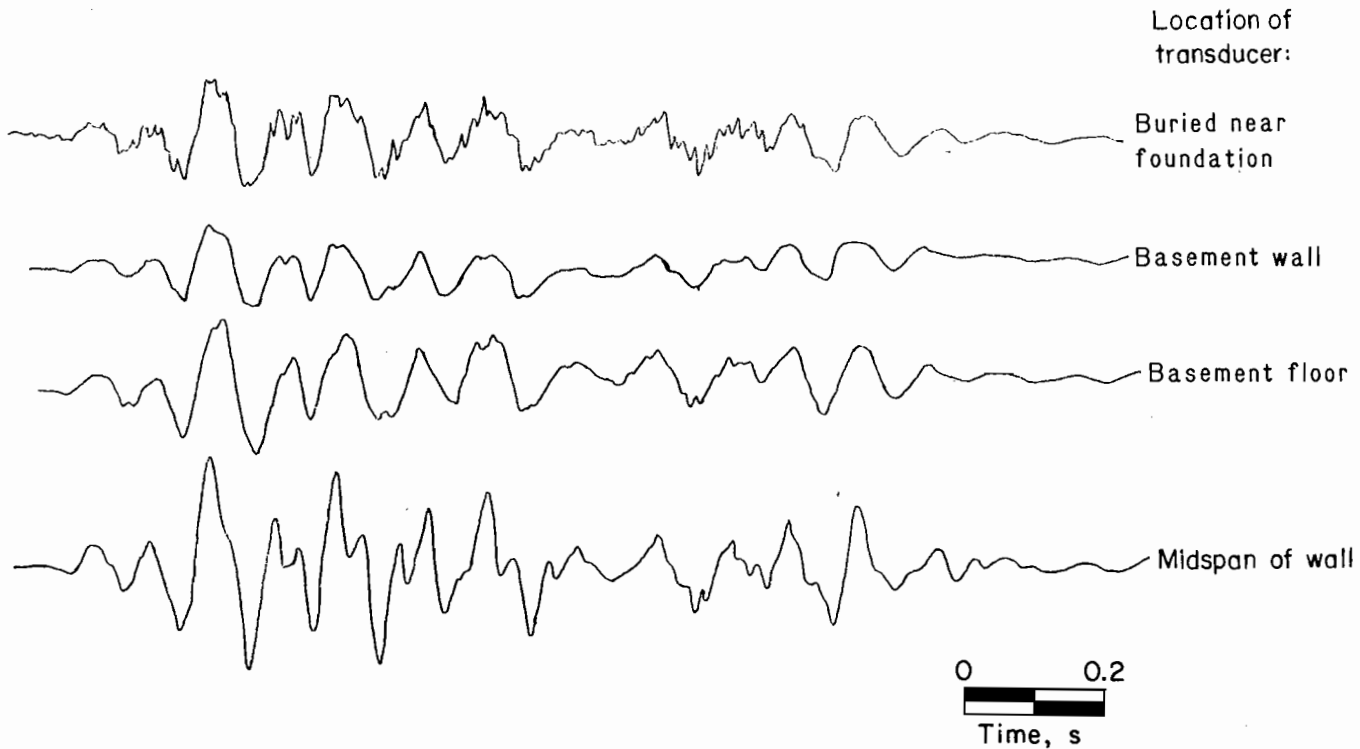


FIGURE 2.—Vibration records obtained at Fador house, shot 6, longitudinal component of motion, at different locations (1).

TABLE 1. - Vibration levels measured near and on test structures, peak particle velocities, inch per second.

Structure	Shot	Component	Measurement location				
			Ground level, buried	Ground surface, on slab	Ground level, on foundation	Basement floor	Away from structure, buried
Schnegg house...	1	V	0.32	NM	NM	0.21	NM
		L	.26	0.35	NM	.13	NM
		T	.28	.41	NM	.15	NM
Mine office.....	2	V	.12	NM	0.060	NM	NM
		L	.19	.14	.13	NM	¹ 0.11
		T	.18	.12	.15	NM	¹ .13
Fador house.....	4	V	NM	NM	.060	.054	NM
		L	NM	NM	.051	.040	NM
		T	NM	NM	.079	.072	NM
Do.....	5	V	.11	NM	.090	.080	² .090
		L	.13	NM	.14	.12	² .160
		T	.11	NM	.11	.080	² .160
Do.....	6	V	.13	NM	.12	.11	NM
		L	.18	NM	.17	.15	NM
		T	.13	NM	.12	NM	NM
Schnegg house...	6	V	.034	NM	.035	NM	NM
		L	.074	NM	.075	NM	NM
		T	.070	NM	.060	NM	NM
Training house..	13	V	.017	.015	.016	NM	³ .020
		L	.058	.060	.035	NM	³ .067
		T	.057	.059	.047	NM	³ .063

NM Not measured. ¹74 ft from structure.

V Vertical. ²72 ft from structure.

L Longitudinal. ³98 ft from structure.

T Transverse.

CONCLUSIONS

At the four sites examined in this study, the specific measurement methods used around structures appear not to be critical at low vibration levels. Five gauge locations were examined: on a surface slab, buried in the ground at the structure, mounted on the foundation wall, on the basement floor, and buried at a distance of 72 to 98 ft from the structures. The waveforms for all three components, vertical, longitudinal, and transverse, were found to be similar for the five measurement locations. This was particularly true for the low-frequency part of the waves, which is of most concern for vibrational response of structures. Low frequency for this study was 5 to 10 Hz.

Considerable differences were noted for the high-frequency part of the waves, mostly in the beginnings, which correspond to the multiple arrivals from the various delayed holes. It is likely that these differences were primarily a result of the varied wave paths to the different monitoring positions, leading to uneven and irregular wave interference. The high frequencies are of less concern for structural response, as discussed in a previous Bureau report on structure response from blasting (16).

Because peak values are influenced by the way specific vibration or wave modes interact, they were found to vary irregularly between the different methods. However, they were generally within

20 to 30 pct of each other and not consistent with the systematic 0.40 factor for foundation depths of 6 to 9 ft predicted by Clark (10). This demonstrates one of the weaknesses of a simple peak-particle-velocity criterion and over-reliance on interpretation of precise values.

Vibration levels below grade, such as on the basement floor, were consistently lower, suggesting differential

displacement for the wall top and bottom. Apparently, vibrational energy does decrease with depth.

Rather than recommending a specific measuring method, the researchers recommend that consistency be used at any one site. Where the option is available, shallow soil burial is still the desired method and was found least likely to introduce additional mechanical error.

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INITIATION TIMING INFLUENCE ON GROUND VIBRATION AND AIRBLAST

By John W. Kopp¹

ABSTRACT

A major concern with blasting at surface mines is generation of ground vibrations and airblast and their effects on nearby residences. This Bureau of Mines report looks at the use of millisecond delays in blast design and their effect on the resulting ground vibrations and airblast. A total of 52 production blasts were instrumented and monitored at a surface coal mine in southern Indiana. Arrays of seismographs were used to gather time histories of vibrations and airblast. The data were analyzed for peak values of vibration and airblast and for

frequency content. Various delay intervals were used within and between rows of blastholes. Delay intervals within rows were 17 and 42 ms, and those between rows ranged from 30 to 100 ms; these intervals are equivalent to burden reliefs of 0.5 and 1.3 ms/ft within rows and 1.2 to 4.3 ms/ft between rows. Subsonic delay intervals within rows reduced airblast by 6 dB. Large delay intervals between rows reduced the amplitude of ground vibrations; vibration frequency depended primarily upon the geology of the mine site.

INTRODUCTION

Millisecond-delay blasting caps were introduced to the mining industry in the 1940's and gained wide acceptance as a tool for improving rock fragmentation. The use of these delays also reduced ground vibration levels. The Bureau of Mines reported on this technique, which allowed the explosive in each delay period to be treated separately in its contribution to the ground vibrations, in 1963 (1).²

The Bureau undertook a major research effort during the 1970's to quantify these ground vibrations and their effects on structures (2). Out of this study came the fact that not only amplitude is important in preventing damage, but also the frequency content of the vibrations, because resonances were occurring at the

natural frequencies of oscillation of structures. A study was then undertaken at a surface coal mine to determine if the predominant frequency of ground vibrations could be controlled by an appropriate choice of blast delay intervals. This paper summarizes that study, published as RI 9026 in 1986 (3).

During the analysis of the Bureau's coal mine data, another study of delay control of blasting was undertaken by Reil (4) through a Bureau contract. Reil's study in two stone quarries involved precise timing control. Results were mixed with respect to both amplitudes and frequencies; results also appeared to be both distance and measuring-site specific.

EXPERIMENTAL DESIGN

INSTRUMENTATION AND MEASUREMENT TECHNIQUES

Airblast and ground vibrations from 52 production blasts were measured with self-triggered seismographs. These seismographs recorded three components of

ground motion and the airblast overpressure on standard cassette audiotapes. The tape recorder for each machine was automatically activated when the ground vibration reached a predetermined level

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²Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

The frequency range of the transducers used to monitor ground vibration was 1 to 200 Hz. The maximum amplitude that could be recorded was 4 in/s. For low-level signals, an alternative range could be selected with a maximum amplitude of 1 in/s.

The airblast channel used a 1-1/8-in ceramic microphone. The frequency response of the system was 0.2 to 200 Hz, with a maximum peak overpressure of 137 dB.

The blasts were also monitored using a 16-mm high-speed cinecamera. The rotating-prism camera was capable of speeds in excess of 8,000 frames per second, but a rate of 1,000 frames per second was sufficient for this study. The initiation system used was Nonel³ with surface delays. Nonel tubing was also tied into the delay detonators in order to provide a flash signal for the camera to record. This allowed computation of the firing time for each delay to the nearest millisecond.

TEST SITE

The project test site was a surface coal mine in southern Indiana. The mine utilized two large draglines to remove 50 to 100 ft of overburden from a 4- to 5-ft coal seam. The overburden was primarily shale with some sandstone intermixed, and required blasting to facilitate digging by the draglines. Blasting was accomplished using 12-1/4-in holes normally drilled on a 30-ft-square pattern and shot en echelon into a buffer. The terrain is flat to gently rolling hills. The layout of the pit is not influenced by topography; it is about 3 miles long in a north-south direction. The movement of mining is toward the west.

TEST PROCEDURE

This series of tests had two objectives. First, to determine if orientation of the shot affected vibration levels, seismograph arrays were established

³Reference to specific products does not imply endorsement by the Bureau of Mines.

in four directions from the shot. Each array used three instruments, located at distances of 300 to 5,000 ft from the blast. The complete ground vibration and airblast waveforms were recorded at each station. From these, the frequency spectra and peak particle velocities and airblast could be determined. Peak particle velocities were plotted versus scaled distance for each array direction. A least squares fit of the regression line was determined for each set of data. A one-way analysis of variance test was then performed on the data sets to determine if the blast parameter under study was significant.

The second objective was to determine the effects of varying the delay intervals between holes and rows. Airblast and ground vibration measurements were made as before with seismographs deployed in arrays in the four directions. Delay intervals used were 17 and 42 ms between holes in an echelon and 30, 42, 60, 75, and 100 ms between echelons. Normal mine production shots used 17 ms between holes in an echelon and 42 ms between echelons. The Nonel Primadet system was used for these delays for shots 1 through 36. Delay intervals between rows for shots 37 to 52 were obtained by using electric caps, all of one period, with a sequential blasting machine. The high-speed camera described previously was used to determine actual firing times for each hole. Propagation plots were made of the airblast and vibration data.

DIRECTIONAL EFFECTS

The blast design examined for directional effects was the one normally used by the mine. A delay of 17 ms was used between holes in the echelon, and a delay time of 42 ms was used between echelons. Ten shots were examined for this experiment.⁴ The ground vibration data are shown as a propagation plot in figure 1. The airblast data are presented in figure 2. A least squares regression analysis

⁴Shots 14 through 23. (Shots 1 through 4 are discussed in the following section; shots 5 through 13 are not discussed in this paper.)

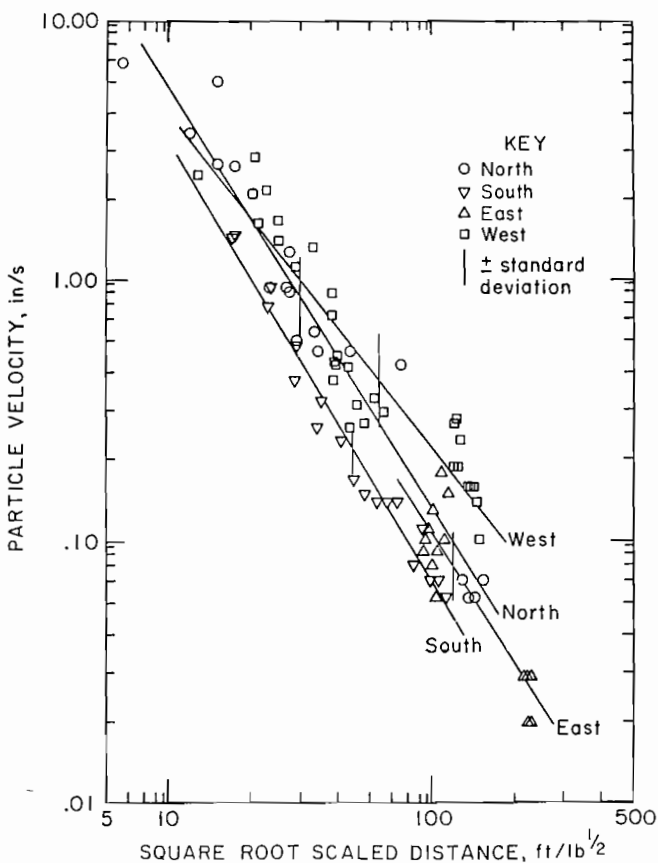


FIGURE 1.—Propagation plot of peak particle velocity showing directional differences.

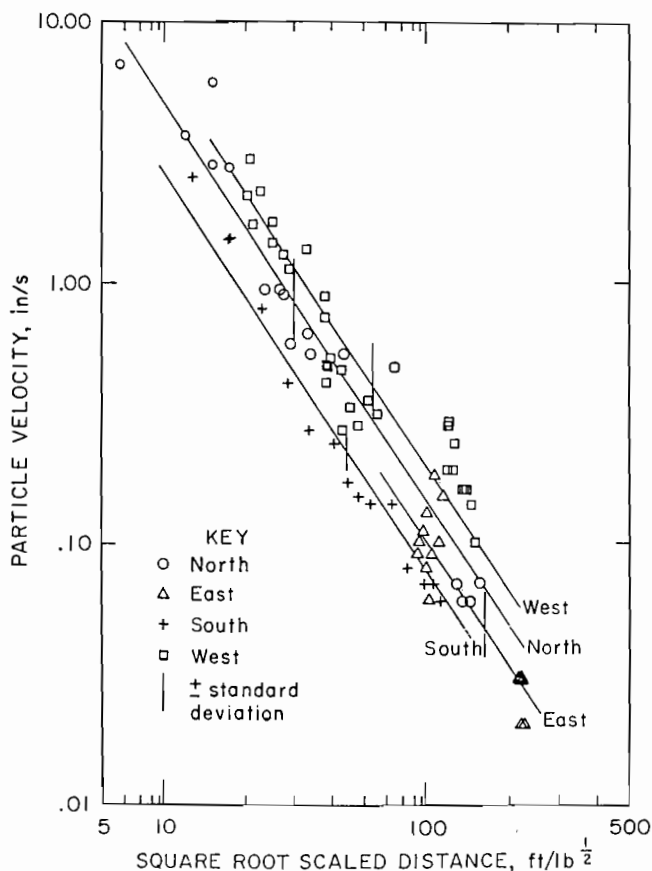


FIGURE 3.—Directional effects on propagation of ground vibrations. All regression lines have a common slope.

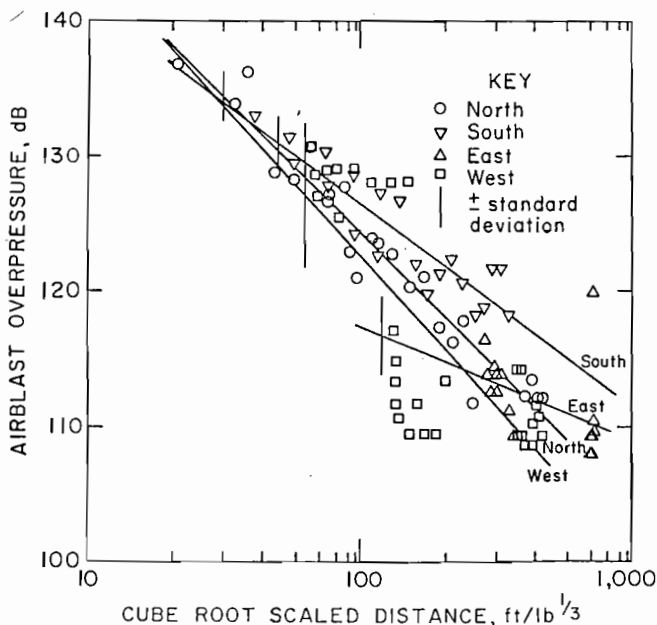
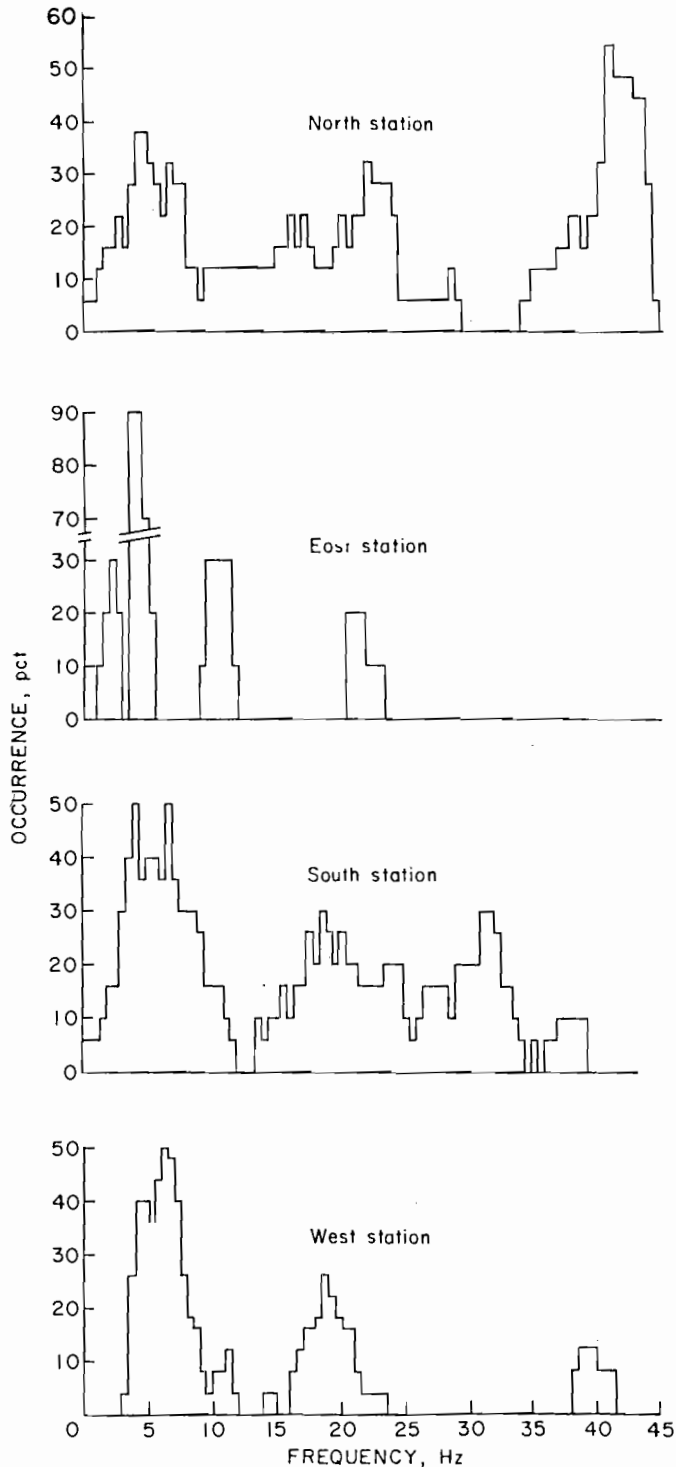


FIGURE 2.—Propagation plot of peak airblast showing directional differences.

was used to determine the regression line of each set of data. Analysis of variance tests performed on the vibration data determined that the data can be represented by four regression lines with a common slope (fig. 3). This indicates that the vibration level is dependent on direction from the blast, but attenuation of the vibrations is independent of direction, with the possible exception of the western direction. This may be due to a geologic anomaly west of the mine. The western part of the mine is overlain by lacustrine and sand and gravel deposits associated with a large creekbed drainage area. These deposits tended to produce lower predominant frequencies of ground vibrations (fig. 4) in the transverse axis than those produced for the other arrays on undisturbed ground (north and south directions). Frequencies of vertical and radial vibrations did not appear to be affected. The frequency of

vibrations in the reclaimed spoil or eastern direction was also predominantly lower.

The highest vibration levels were found in the western direction, with levels in the north array direction the next highest. The results in figure 3 suggest that vibration levels in the direction of initiation can be twice those in the opposite direction.



DELAY INTERVALS WITHIN ROWS

Two different delay intervals were used between adjacent holes in each echelon: 17-ms from shots 1 and 2 and 42-ms delays from shots 3 and 4. A 100-ms delay interval was used between rows. The blasts were all shot at the same location in the mine using the same blast pattern.

For these shots, the mine used a square pattern drilled on 25-ft centers. The pattern was fired en echelon, giving an effective burden of 18 ft and spacing of 35 ft. The actual firing times averaged 23 and 44 ms for the nominal 17- and 42-ms delays, respectively. This gave a relief of 0.5 ms/ft of spacing for the 17-ms-delay shots and 1.3 ms/ft for the 42-ms-delay shots. The burden/delays averaged 96 ms, giving a burden relief of 5.3 ms/ft.

The direction of the measurement arrays from the shot did not appear to significantly affect the airblast data. Analysis showed the design using 42-ms delays produced 6 dB less airblast than the 17-ms design (fig. 5).

An analysis of variance was also performed for each array direction comparing the two delay intervals. The data were sufficiently different to require separate regression lines with a common slope for the west and north arrays, but showed no difference in the east array (figs. 6-8). The south array had insufficient data for analysis. The direction of initiation of the holes in each row was toward the northwest. The airblast trace velocity for the 17-ms delay design was supersonic in the north and west directions but subsonic in the east direction. The airblast trace velocity for the 42-ms design was subsonic in all directions. The airblast from the 17-ms design was 7 dB higher in the north array and 6 dB higher in the west array, but no

FIGURE 4.—Predominant frequencies of transverse ground vibrations.

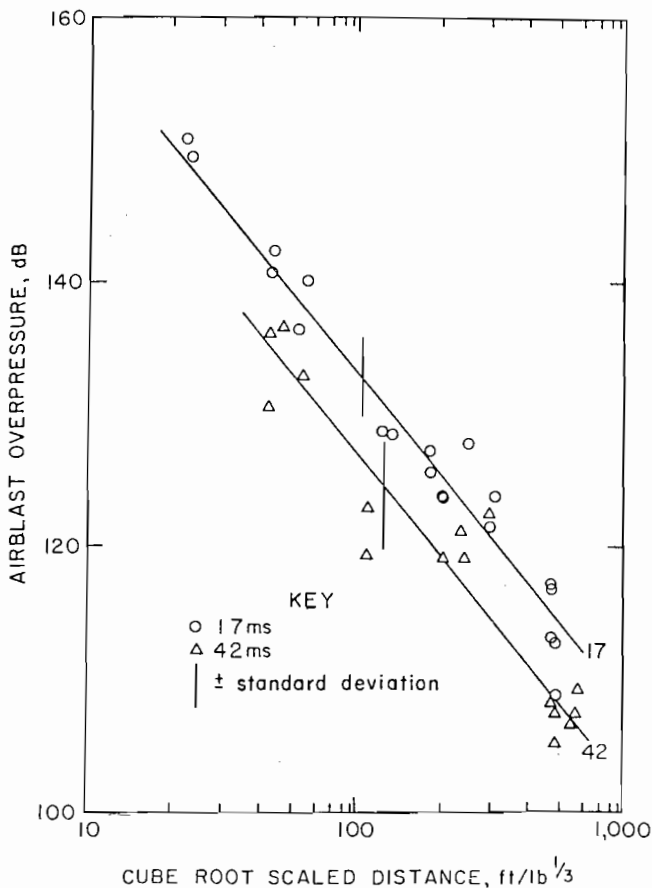


FIGURE 5.—Propagation plot showing differences in airblast levels for the two different blast designs.

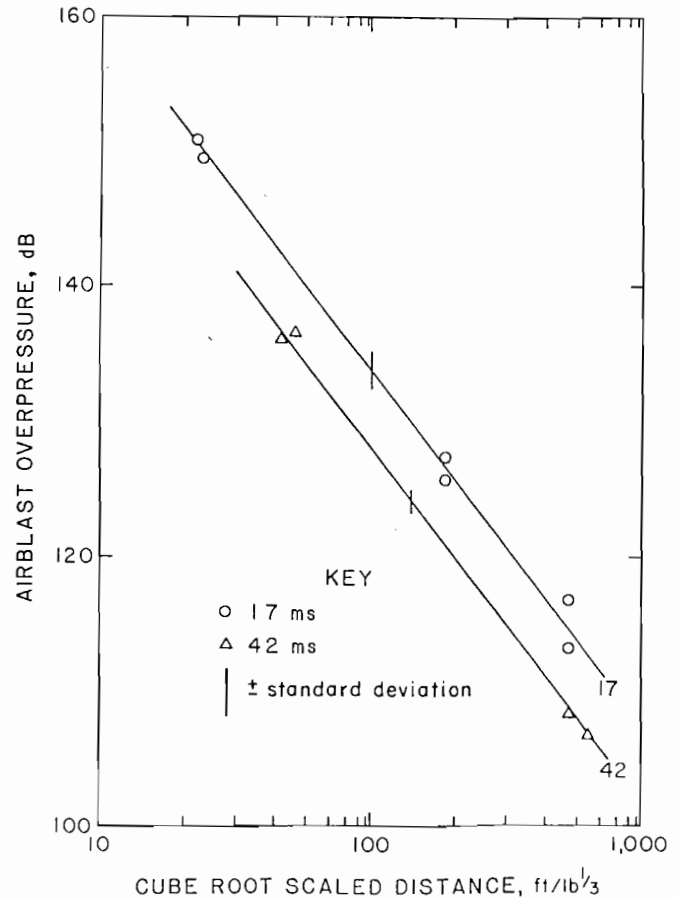


FIGURE 6.—Propagation plot of peak airblast for the west arrays.

different in the other directions. This would indicate that the reduction in airblast is attributable to the fact that the trace velocity along the free face was subsonic for the longer delay interval.

The two blast designs also show some difference in the predominant frequencies of the airblast. The design using 17-ms delays had more airblast energy in the 10-Hz range than the 42-ms delay design, as shown in figure 9.

Statistical analysis showed no significant differences in the ground vibration levels from the two blast designs. However, spectral analysis did show a difference in the predominant frequencies of the two designs (fig. 10). The 17-ms design has its predominant frequencies around 10 Hz, while the 42-ms design has more scatter in its predominant frequencies.

The delay interval between holes should be selected such that the trace velocity along the free face is subsonic. This resulted in a reduction of airblast of up to 6 dB in these tests. The delay interval between holes did not affect ground vibration amplitudes in these tests.

DELAY INTERVALS BETWEEN ROWS

Shots 24 through 52 used the same 17-ms delay between holes in a row (the average value of the actual delay interval between holes was 23 ms), but the delays between the rows were varied, in five steps from 30 to 100 ms. The shot pattern was 33 ft square, shot en echelon, giving an effective burden of 23 ft and effective spacing of 47 ft. Five different delay intervals between rows were used to study the effect of burden delay timing on vibration levels. The

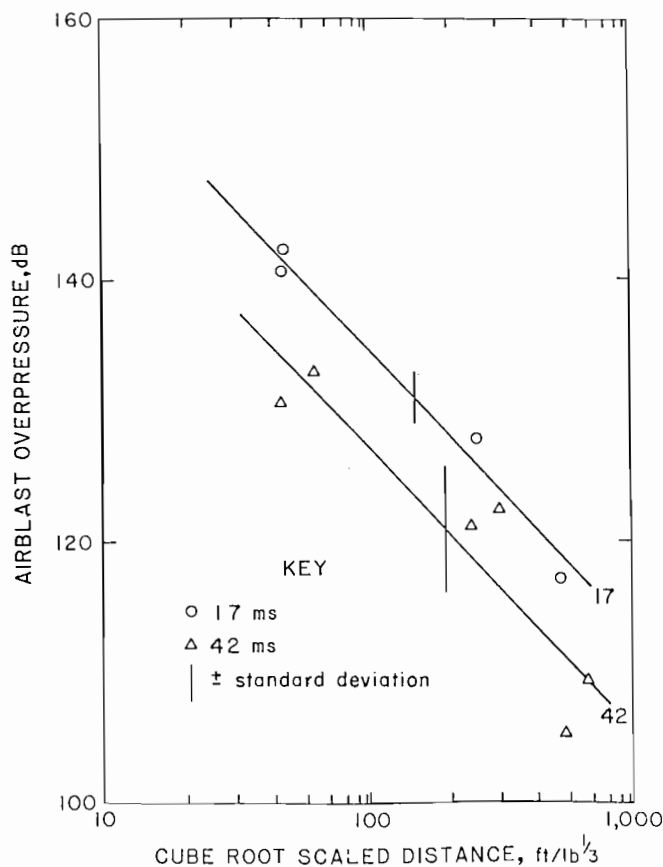


FIGURE 7.—Propagation plot of peak airblast for the north arrays.

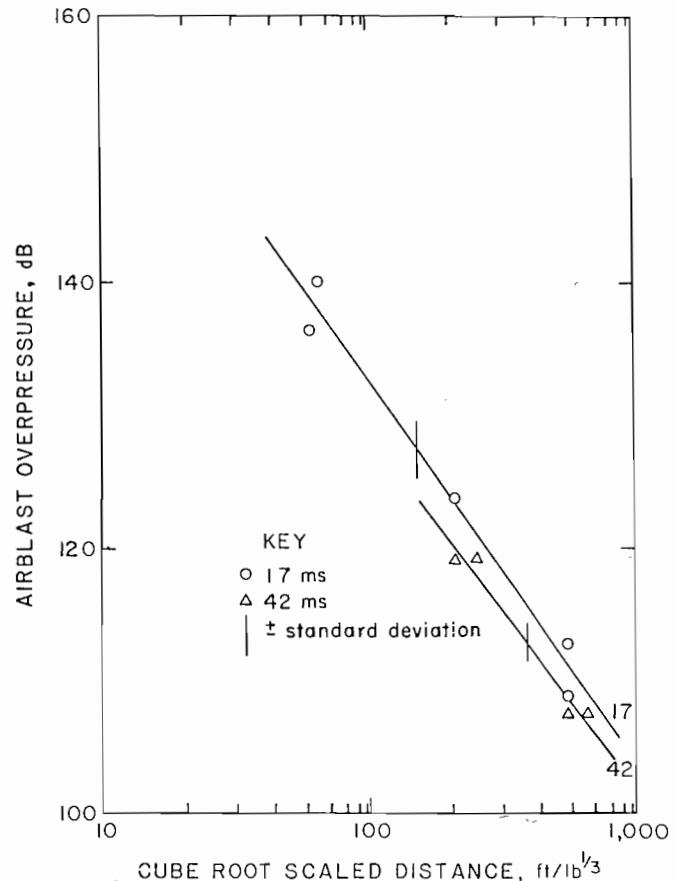


FIGURE 8.—Propagation plot of peak airblast for the east airways.

intervals used were 42 ms, which was the delay used by the mine, and 30, 60, 75, and 100 ms. The 42 ms was a pyrotechnic delay, while the others were selected using a sequential blasting machine. Table 1 gives values of burden relief for the different burden delays used.

Vibration data for each design were compared to determine if direction of orientation of the seismograph array was important. The eastern array (in the

TABLE 1. - Effective values of burden delay intervals¹

Shot	Delay interval, ms		Burden relief, ms/ft (actual)
	Nominal	Actual	
42-44	30	27.5	1.2
24-36	42	48.5	2.1
37-41	60	58.5	2.5
45-49	75	76.0	3.3
50-52	100	99.5	4.3

¹Effective burden was 23 ft for all shots.

spoils) had the lowest vibration levels. The highest levels were toward the west, where the ground was undisturbed. The vibration levels of the other arrays were intermediate between these levels. The western and northern vibration arrays were chosen for further analysis.

Vibration levels of the different designs were compared for the north and west arrays using regression analysis (figs. 11-12). Table 2 gives values of intercepts for regression lines with common slopes and shows that the two longest delay intervals resulted in the lowest vibration levels.

Analysis of the regression lines shows that only the two longest delay intervals resulted in significant reductions of ground vibration. This is probably the result of reduced confinement because sufficient time was allowed by the longer delays to move the burden before the next echelon of holes was initiated. This study found an average vibration

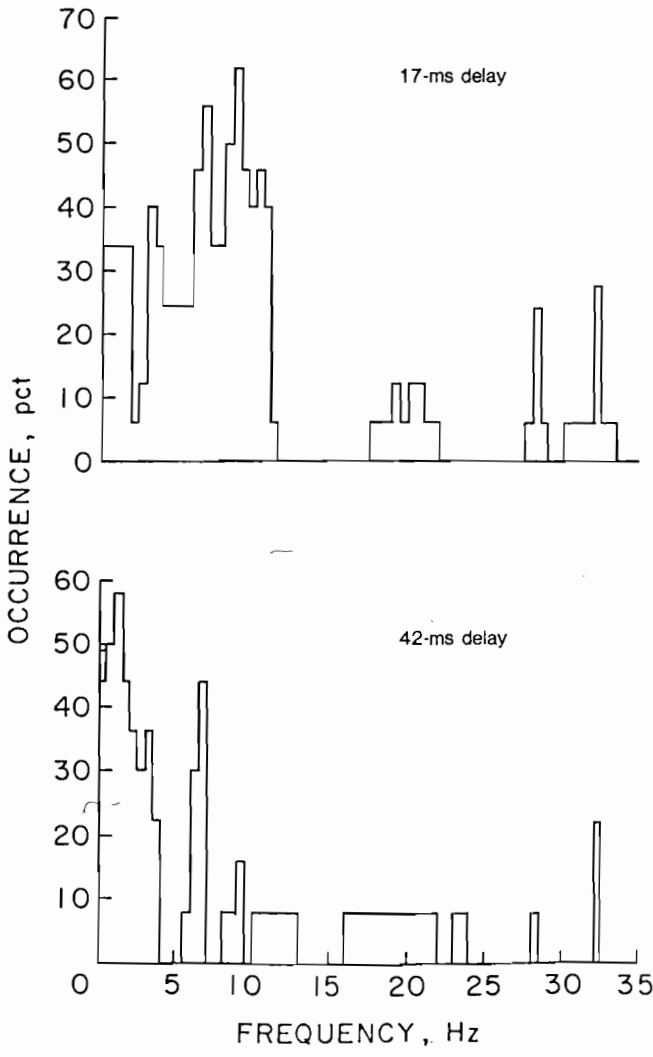


FIGURE 9.—Histograms showing the spectra differences of airblast for the two designs.

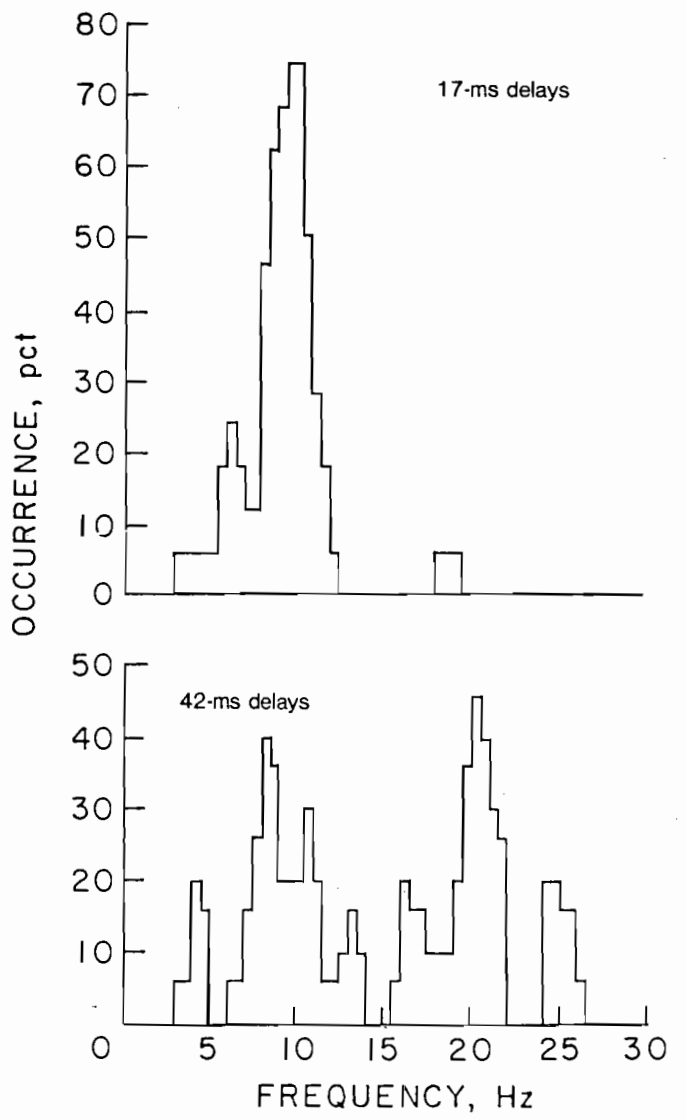


FIGURE 10.—Histograms showing the spectra differences of ground vibrations for the two designs.

TABLE 2. - Comparison of regression lines for various burden delay intervals

Shot	Burden delay interval, ms	Array direction	Regression line		Regression line with common slope	
			Slope	Intercept	Slope	Intercept
42-44.....	30	North.....	-1.69	214	-1.50	116
24-36.....	42	...do.....	-1.29	53	-1.50	104
37-41.....	60	...do.....	-1.44	102	-1.50	122
45-49.....	75	...do.....	-1.74	212	-1.50	87
50-52.....	100	...do.....	-1.41	51	-1.50	71
42-44.....	30	West.....	-1.11	42	-1.25	71
24-36.....	42	...do.....	-1.27	77	-1.25	73
37-41.....	60	...do.....	-1.47	171	-1.25	80
45-49.....	75	...do.....	-1.06	30	-1.25	63
50-52.....	100	...do.....	-1.03	19	-1.25	50

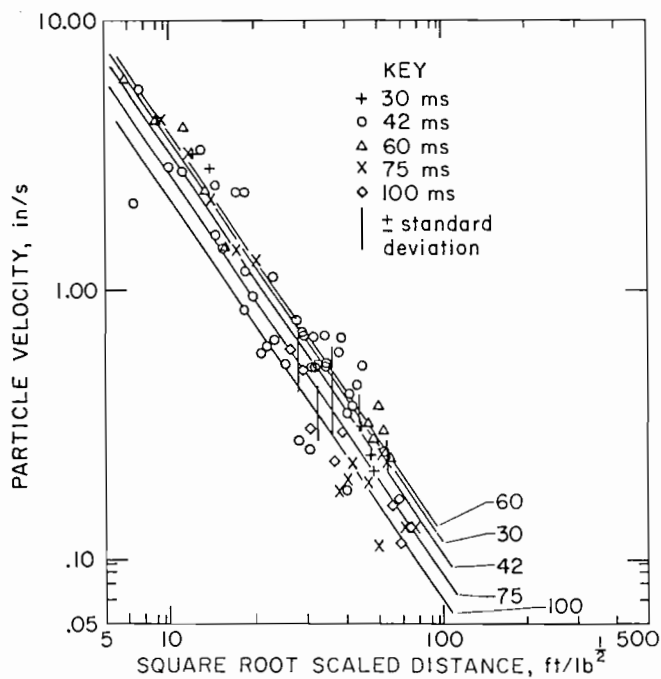


FIGURE 11.—Propagation plot of peak particle velocity showing differences caused by burden timing, for the north array.

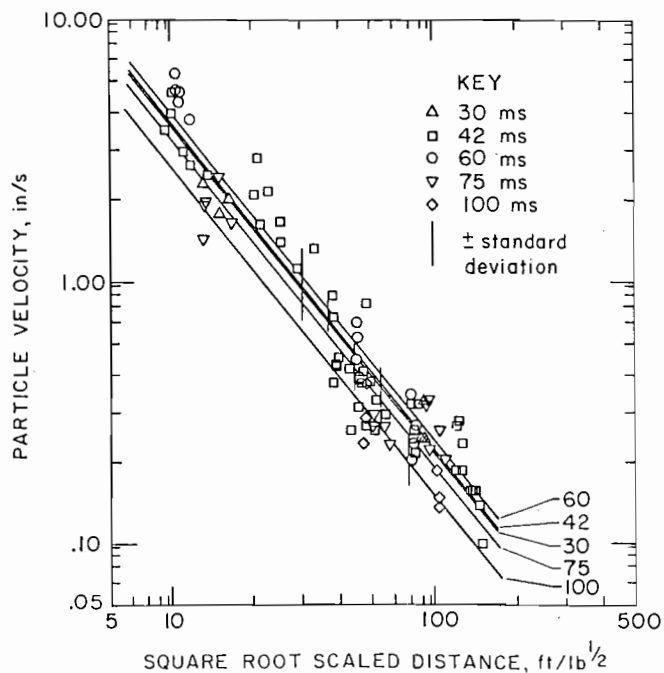


FIGURE 12.—Propagation plot of peak particle velocity showing differences caused by burden timing, for the west array.

reduction of 30 pct using the 100-ms design compared with the 42-ms design normally used by the mine.

The frequency content of the ground vibrations was also studied. No reduction in low-frequency vibrations was observed, which suggests that geology was the

predominant influence on the frequency of vibrations.

Airblast was also analyzed, but no significant differences were found in frequency or amplitude as delays between rows were varied.

CONCLUSIONS

Careful attention to blast design practices can help reduce airblast and ground vibrations generated by mine blasting. This study examined blasthole delay intervals and their effects on airblast and ground vibrations.

Airblast was influenced by the trace velocity along the free face. To reduce airblast, the trace velocity, which is a function of delay interval and spacing between holes in an echelon, should be less than the speed of sound in air. Airblast was reduced by about 6 dB by choosing delays giving a trace velocity of 80 pct of the speed of sound rather than a supersonic velocity.

Delays between holes in each row or echelon should be greater than 1 ms/ft of spacing, in order to prevent reinforcing

of the airblast wave fronts from the individual holes. Care must also be taken to avoid selection of delay intervals that can cause airblast frequencies equal to the natural frequencies of midwalls of nearby structures (about 11 to 25 Hz). Delay intervals of less than 40 ms will usually not present a problem.

Orientation of the blast and direction of initiation had a noticeable effect on the magnitude of vibrations. Vibration levels in the direction of initiation were about twice the level of those away from the direction of initiation. Vibration levels across the pit from the blast were also lower.

Vibration levels were also dependent on the delay interval between rows. Adequate time must be provided for burden

relief for each row. This investigation found that the delay interval between rows should be as long as practical for the burden involved. The longest burden relief value, 4.3 ms/ft, gave the lowest vibration levels.

The timing of delay intervals between rows had no influence on the frequency content of the ground vibrations. Geology was the controlling factor for predominant frequencies of ground vibrations in this investigation.

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VIBRATIONS FROM BLASTING OVER ABANDONED UNDERGROUND MINES

By David E. Siskind¹ and Vigil J. Stachura¹

ABSTRACT

Vibration wave frequency from surface mine blasting is an important influence on the response and potential cracking of nearby structures. The Bureau of Mines studied blasting vibrations in a midwestern coal mine that occasionally produces 4-Hz surface waves in its production blasting and has received numerous complaints from neighbors. The mine and nearby town are underlain by abandoned underground coal mines 85 to 325 ft below the surface.

Blast vibration measurements at the site and analysis of mine and regulatory agency records indicated that the propagating medium was primarily responsible for the vibration wave characteristics, including low frequencies, long

durations, and lower-than-normal attenuations of amplitude with distance. The observed low-frequency waves were consistent with predictive theoretical models of surface wave generation using the depths to the old mines.

Blast designs also contributed to the vibrations problem. The complex multilayered blasts generated vibration amplitudes up to three times those of same-weight-per-delay single charges, particularly for the echelon designs. By contrast, the heavy casting blasts generated more of the unusual low frequencies. Because of these low-frequency, long-period waves, the widely adopted 8-ms minimum charge separation criterion may not apply at this site.

INTRODUCTION

The Bureau of Mines studied a site at the western Indiana town of Blanford, where surface coal mine blasting was producing unusual low-frequency, long-duration vibrations. At the request of two regulatory agencies, the Indiana Department of Natural Resources (DNR) and the U.S. Office of Surface Mining (OSM), the Bureau investigated the influence of the ground structure, which includes extensive abandoned underground workings under both the active mining and the town of Blanford. These abandoned workings are at several depths, the most significant being the extensively mined coalbed No. 5 at 225 ft and the partially mined No. 4 at 325 ft (fig. 1). The local sedimentary rocks and the coal seams are essentially horizontal.

The initial objective of the study was to determine if blasting vibrations at this site were unusual, as local homeowners claimed. Researchers noticed that

some vibration records showed data unlike any from previous Bureau studies, with very low frequencies down to 4 Hz and long durations of over 4 s.

Causes for the unusual data were then sought, including the ground and structure conditions and factors in the surface blast designs. By identifying relative influences, design guidance could be provided to minimize the low frequencies generated at this and similar sites.

Seismologists have recognized that low-frequency vibrations are produced in soft, weak materials with low propagation velocity (1).² Thick soil layers, fill areas, and weak sedimentary rocks are examples of such materials. Furthermore, such layering typically has contrasting acoustic properties and produces surface waves of several types. These waves are inherently low in frequency and develop from multiple reflections of conventional body waves as they propagate in the rock

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²Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

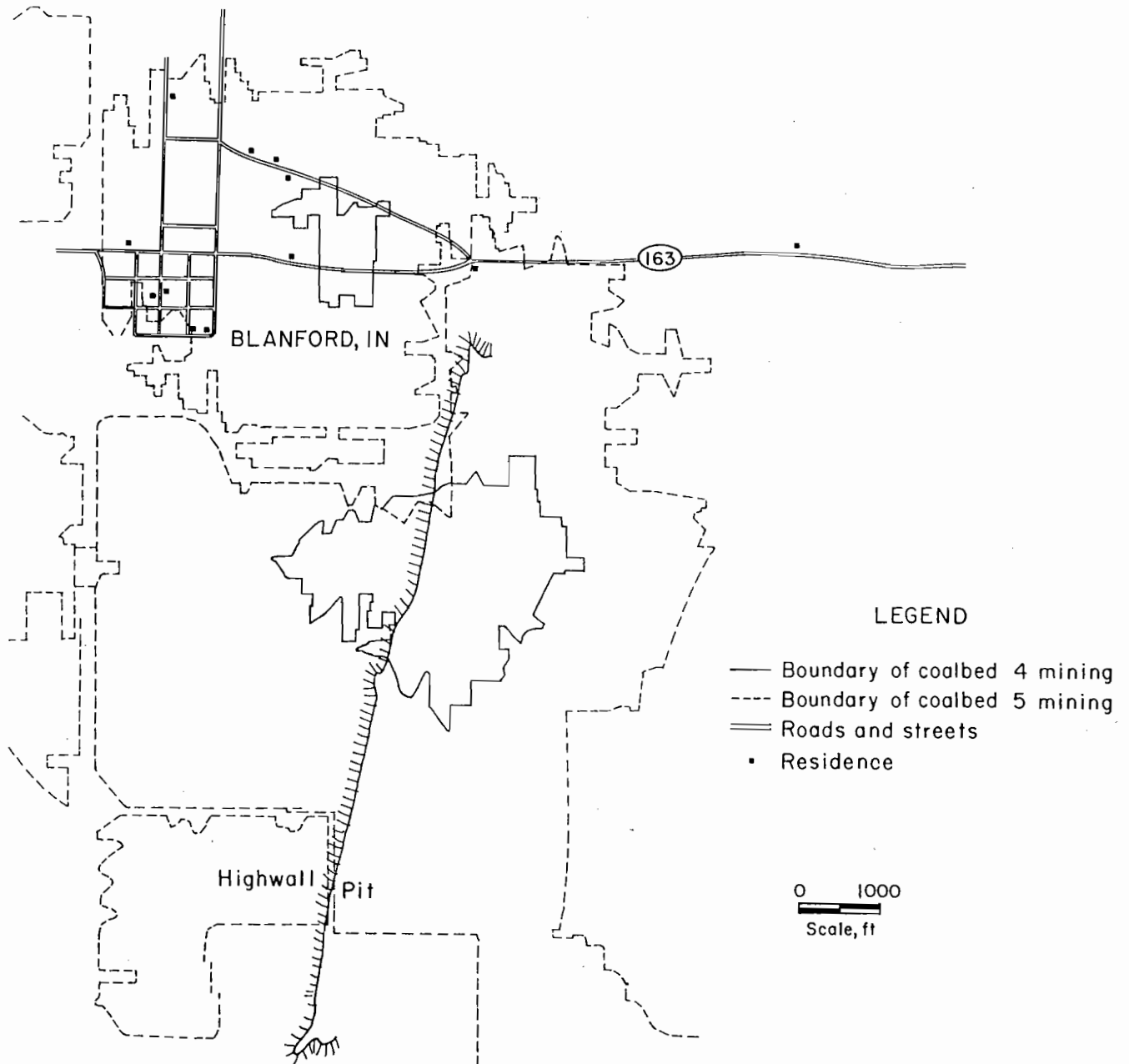


FIGURE 1.—Town of Blanford, current surface mining highwall, abandoned underground workings, and some of the homes studied for vibrations and settlement.

layers. Extensive horizontal mine workings also provide a strong reflecting layer.

Recent research suggests the selection of delay intervals as a useful method for controlling blast vibration frequencies. Early research by the Bureau investigated delay effects on vibration amplitudes (2). However, these researchers also noticed that the shorter delays of 0 and

9 ms (as opposed to 17 and 34 ms) produced lower frequencies.

Research by Anderson (3-4) and Reil (5) investigated the use of single-charge blasts to "calibrate" a site and provide a model record for seismogram syntheses. Anderson developed a "Fourmap" scheme, which uses superposition of appropriately delayed single-charge time histories that are then converted into Fourier spectr:

(6). Accurate delay times are assumed. This method graphically shows the relationship between delays and predicted spectral distributions and is used commercially by several blasting consultants.

Contrasting with Anderson's and Reil's work in rock quarries, a small amount of work was done by the Bureau in surface coal mines using standard production blasts. Here, the thick soil cover and relatively weak and complex sedimentary rock layers represent a far more influential propagation medium. Absorption, dispersion, and the generation of secondary waves through multiple reflection quickly complicate the wave character and make it more difficult to control through blast design. These conditions, plus the larger holes and charges, tend to generate lower frequencies, which are less amenable to change.

Wiss' extensive 1979 coal study did not specifically examine delays and vibration

frequency; however, he did notice the occurrence of constructive wave interference when using the 9-ms delay separation, but not when using 17-ms delays (7). Kopp specifically sought the influence of delay times on frequency for coal mine blasts (8). He concluded that vibration frequencies did not generally reflect the between-row or within-row delay intervals. However, his closest measurements at this thick overburden site were at 300 ft, and it is likely that, at that distance and in that material, the primary influence on the waves is the propagation medium itself. In the light of current knowledge, Kopp's field program in 1981-82 should have included closer-in monitoring and tests with more precise initiation timing.

This paper briefly summarizes an investigation in the Blanford area reported more fully in a recent Bureau Report of Investigations (9) and earlier as an unpublished Bureau report to OSM.

EXPERIMENTAL PROCEDURES

Vibration records from 235 production blasts were available for analyses of amplitudes, frequencies, and total durations. These were originally collected by the Indiana DNR and the surface mine (Peabody Coal Co., operating the Universal Mine) at seven Blanford homes over a period of 11 months. Distances of the homes from the blast source ranged from about 980 to 9,800 ft, and as many as six homes were monitored at one time (fig. 1).

For purposes of identifying the generation and propagation influences, Bureau researchers instrumented seven blasts, including two specially fired single-charge shots. For these tests, measurements were made as close as 56 ft to determine the wave characteristics before they were modified by the propagation medium. The single charges were used to identify the specific vibration-generation influences of the complex multideck, multidelayed blasts, as contrasted to the simple vibration sources represented by the single-charge blasts.

All vibration records were plotted for propagation showing the generation and

attenuation of particle velocity amplitudes. Researchers did this for each home, for various groups (by neighborhoods), and as an overall summary for all homes. Propagation plots were also prepared for various blast designs and, in the report to OSM, according to vibration frequency characteristics.

In addition to vibration amplitudes, frequency characteristics recorded at various homes and with various shot designs were compared. Single charges and close-in monitoring revealed the ground's natural response frequency of 8 Hz. Shot design data were compared for the four blasting methods employed by Peabody during the period covered by the records. This allowed identification of delay-sequencing influences on vibration frequency, as opposed to ground influence.

Other site data were collected to identify areas around Blanford and the nearby surface mine that had been previously undetermined. Depths to these deep coalbeds were available from Peabody, which is conducting an extensive study of one Blanford home. Bureau researchers conducted level-loop surveys of eight homes

in September 1985 and the following April to identify possible blast-vibration-induced subsidence and longtime stability. Finally, a predictive model was used to

determine if observed low frequencies were consistent with depths to the old workings (10).

RESULTS

VIBRATION AMPLITUDES

Propagation plots based on the Peabody and DNR records revealed higher vibration amplitudes than were found in previous Bureau studies at surface coal mines (1). Figure 2 is a summary of Peabody- and DNR-collected maximum velocities. The data are clustered, having been collected at seven Blanford homes rather than with widely spaced seismograph arrays. Most notable is that virtually all the measurements exceeded the mean of the previous coal mine vibration summary (shown as maximum velocity, RI 8507, coal). Some even exceeded the envelope line of highest measurements, which is approximately two standard deviations (2σ) above the mean.

Separate propagation plots were also made for the various blast designs (10). Figures 3 and 4 summarize these plots based on square root scaled propagation and all charge weights within 8-ms

intervals. Echelon blast amplitudes exceeded the historical coal mine summary

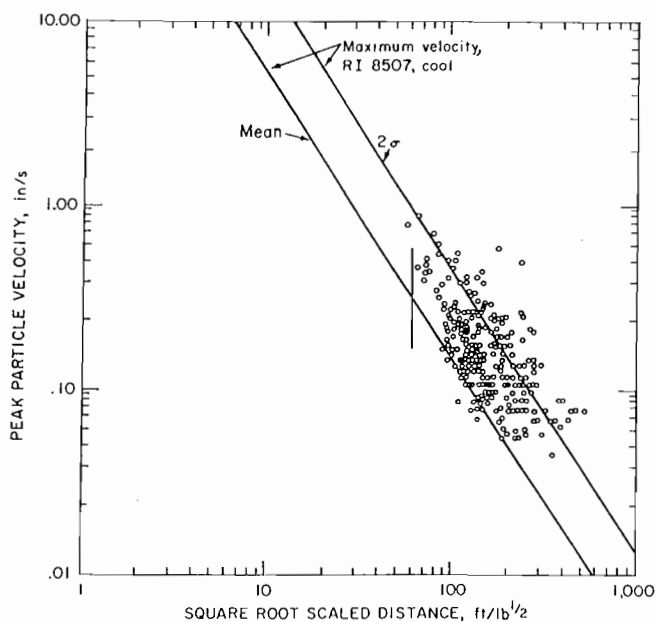


FIGURE 2.—Propagation plot of Peabody and DNR data for all homes compared with surface coal mine blasting data summary from RI 8507 (1).

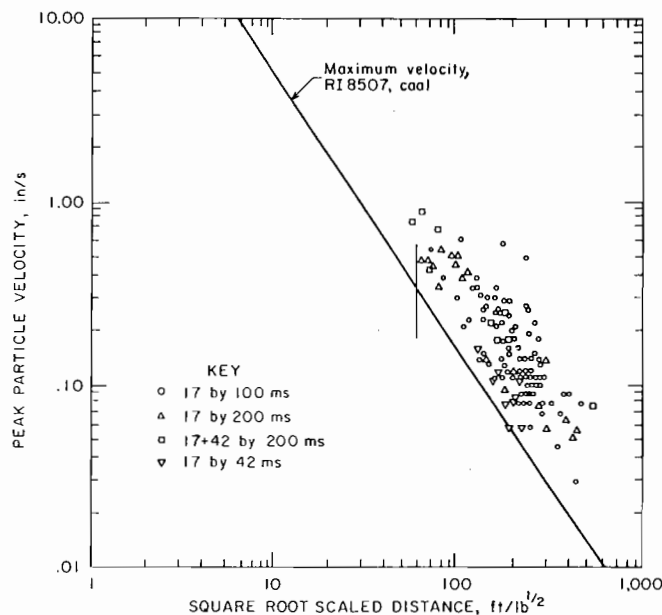


FIGURE 3.—Propagation plot of Peabody and DNR data for echelon production blasts.

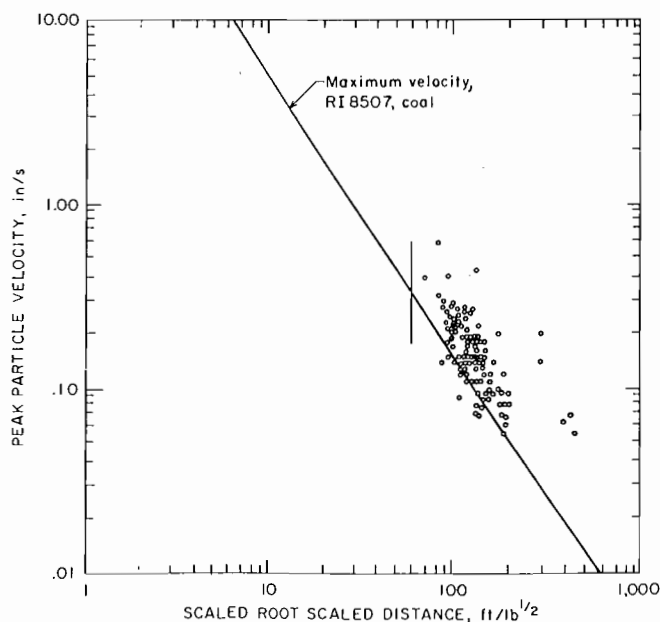


FIGURE 4.—Propagation plot of Peabody and DNR data for casting production blasts.

TABLE 1. - Production blast designs used at the Universal Mine,
July 1984 through April 1985

Design type and delays, ms ^{1,2}	Number of decks	Charge weight per delay, lb	
		Typical	Exception or maximum
Echelon:			
17 by 42.....	1	1,500	2,258 maximum.
17 by 100.....	2-4	325	Exception: 625 on 4-22-85
17 by 200.....	2-4	200- 400	Exception: 1,475 on 1-05-85 1,911 on 1-12-85
Casting: 8, 10, or 12.5 by 140 to 210..	1	2,000	3,842 maximum.

¹17 by 42 indicates 17-ms delays between holes in a row and 42-ms delay between rows.

²The 17+42 by 200-ms echelon had too few values to analyze separately.

from RI 8507 more so than did the casting blasts. This appears to be related to the blast complexity. Both the casting shots and the 17- by 42-ms echelon (17 ms between holes in a row and 42 ms between rows) were full-column charges. The other echelon blasts had 2 to 4 decks (table 1). The shots using full-column charges produced vibrations close to the historical mean, while the other three echelon designs produced most of the higher amplitude vibrations.

Although decking is used to reduce the charge weight per delay, the increased complexity and resulting charge interactions prevent a corresponding decrease in vibration amplitude. Hence, the vibration observed from these blasts was strong relative to the charge weights per delay. These charge weights per delay were computed in the traditional manner, based on a minimum time separation of 8 ms. Apparently, this amount of separation is insufficient to separate effects for such low-frequency, long-period waves.

It is logical to question if the abnormal amplitudes resulted from the dominance of low-frequency and inherently low-attenuation surface waves instead of constructive wave interference between delayed charges. Surface waves were judged a minor influence on vibration amplitudes (if at all) through the following analysis:

1. Most of the very low frequency cases were from casting blasts.

2. The casting blast vibration amplitudes, although high, were closest to the historical data of the various blast designs plotted.

3. If the high amplitudes were from low surface wave attenuation, the casting blasts would be the worst case, not the least abnormal. Apparently, the problem with the casting blasts was not their amplitudes but their unusual low frequencies and long durations.

Production and single-charge data were obtained on-site from the widely spaced array of seismographs (fig. 5). Vibration propagations from the single charges were close to those of previous Bureau studies. However, the production vibrations, for the same charge weight per delay, averaged three times higher. Evidently, vibrations from the delayed charges in the multihole, multideck production blasts were interacting. The somewhat shallower line slopes for the propagation means compared with those from previous studies summarized in RI 8507, are indicative of a low-vibration attenuation with distance. This is likely from surface waves being generated at the longer propagation distances. These results are consistent with the previous measurements. Based on constructive wave interference for various designs used by Peabody, the number of interacting charges were estimated from the delay sequencing shown in figures 6 through 10 (table 2). Sequence times are based on nominal delays as shown on the

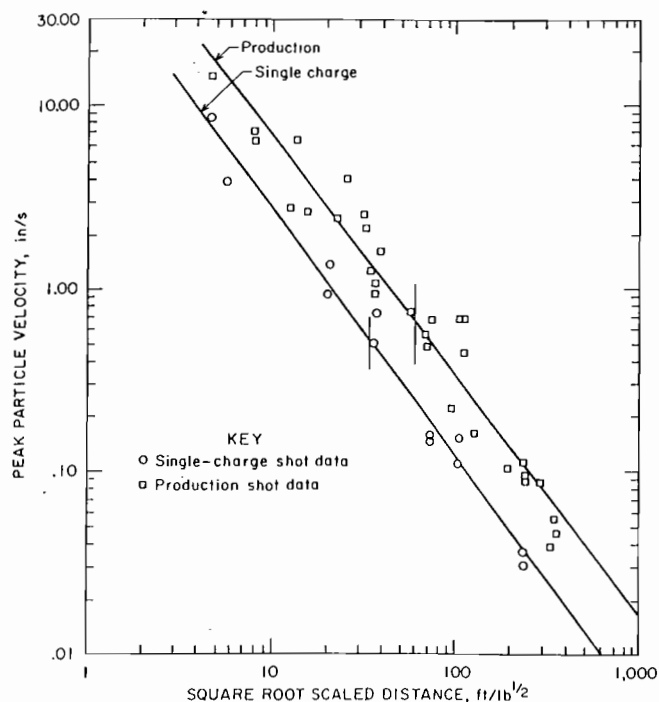


FIGURE 5.—Propagation plots of recent Bureau tests (9) compared with surface coal mine summary from RI 8507 (1).

tops of figures 6 through 10 and the occurrence of constructive interference when delays are within one-half the sinusoidal vibration period.

FREQUENCY OF VIBRATIONS

Production Blasts as Vibration Source Functions

All vibration measurements available as time history records were analyzed for frequency content. Some blasts had very low frequencies of about 4 Hz on all three components (vertical, longitudinal, and transverse) (fig. 6). Other blasts had from zero to two components with such low frequencies. The source function for the blast shown in figure 6 lasted about 0.7 s, which is also the approximate duration of the high-frequency part of the vibration record. The low-frequency wave began at about 1.7 s and lasted at least 2 s, for a total vibration duration of about 4 s at the measurement distance of 7,520 ft.

TABLE 2. - Analysis of charge weights for production blasts

(Maximum number of charges per time interval)

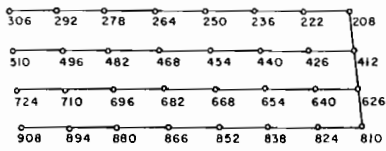
Design type and delays, ms	8 ms	17 ms	60 ms
Echelon:			
17 by 42.....	2	3	7
17 by 100.....	2	3	9
17 by 200.....	2	3	7
Casting:			
Short array.....	1	2	5
Long array.....	3	6	13

Researchers compared vibration records for various blast designs and monitoring locations (figs. 7-10). Although the larger casting blasts frequently produced low frequencies, the three tested echelon designs occasionally also did so. Generally, periodicities in the blast design, such as between-row spacing, were identifiable in some of the records (fig. 6) but not in others. At the larger distances of measurement, the vibrations were strongly influenced by the propagation medium. Researchers believe that low-frequency cases can be moderated by minimizing the amount of energy at such frequency that is produced by the blast design as an energy source (9).

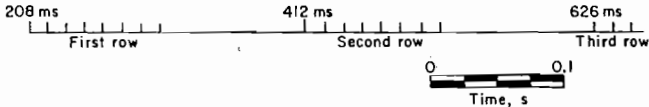
Vibration Wave Types

The researchers attempted to identify the types of waves observed, based on expected components and their phases. Rayleigh waves are vertically polarized with retrograde elliptical particle motions. They have significant motion in the longitudinal and vertical directions, and little in the transverse. The generation of these waves requires only a single free surface such as the ground-air interface. Rayleigh waves can also propagate where there is a sharp acoustic contracting layer at depth.

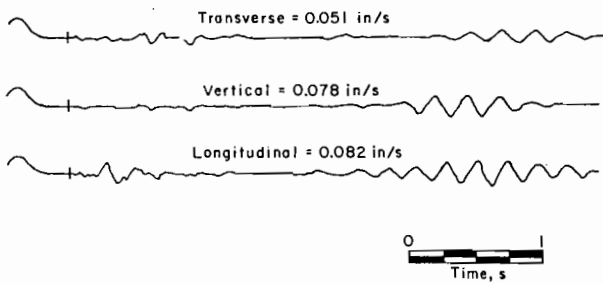
Love waves are horizontally polarized shear waves. They are strong only in the transverse direction. Generation of Love waves requires a layer with top and



PATTERN LAYOUT, HERCUDET SYSTEM, TIMES IN MILLISECONDS

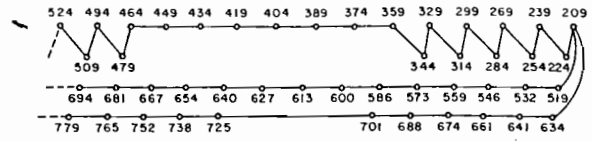


SEQUENCE OF CHARGES

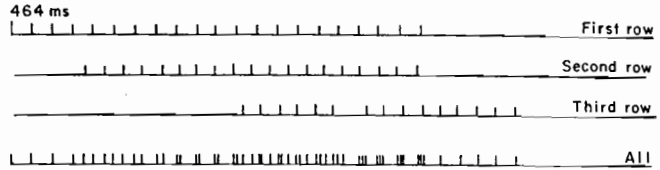


VIBRATION RECORDS, HOLLINGSWORTH HOUSE, 7,520 ft

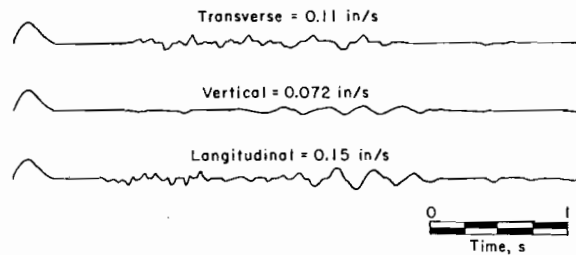
FIGURE 6.—Casting blasts, 200 ms between rows, 10 ms between holes in a row, January 21, 1985.



PATTERN LAYOUT, HERCUDET SYSTEM, TIME IN MILLISECONDS

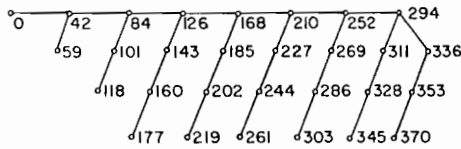


SEQUENCE OF CHARGES

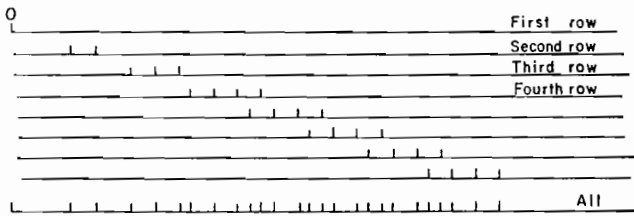


VIBRATION RECORDS, MASSA HOUSE, 4,960 ft

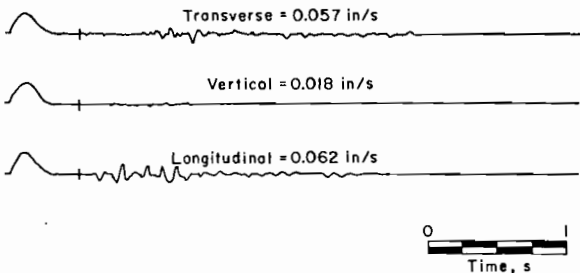
FIGURE 7.—Casting blasts, 200 ms between rows, 10 ms between holes in a row, February 16, 1985.



PATTERN LAYOUT, NONEL SYSTEM, TIME IN MILLISECONDS

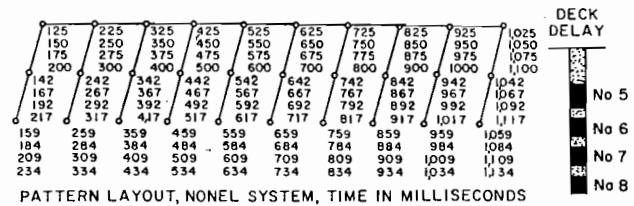


SEQUENCE OF CHARGES

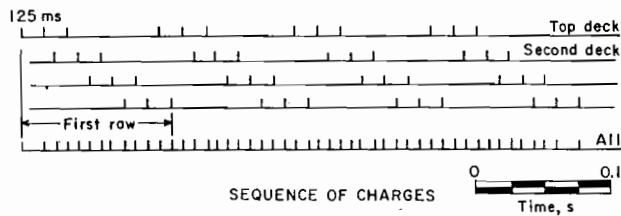


VIBRATION RECORDS, JACKSON HOUSE, AT 8,510 ft

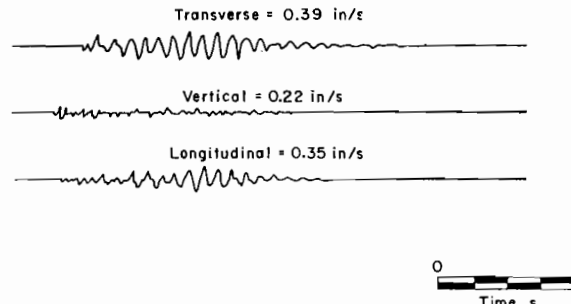
FIGURE 8.—Echelon blast, 42 ms between rows, 17 ms between holes in a row, January 4, 1985.



PATTERN LAYOUT, NONEL SYSTEM, TIME IN MILLISECONDS



SEQUENCE OF CHARGES



VIBRATION RECORDS, JACKSON HOUSE, AT 1,320 ft

FIGURE 9.—Echelon blast, 100 ms between rows, 17 ms between holes in a row, March 4, 1985.

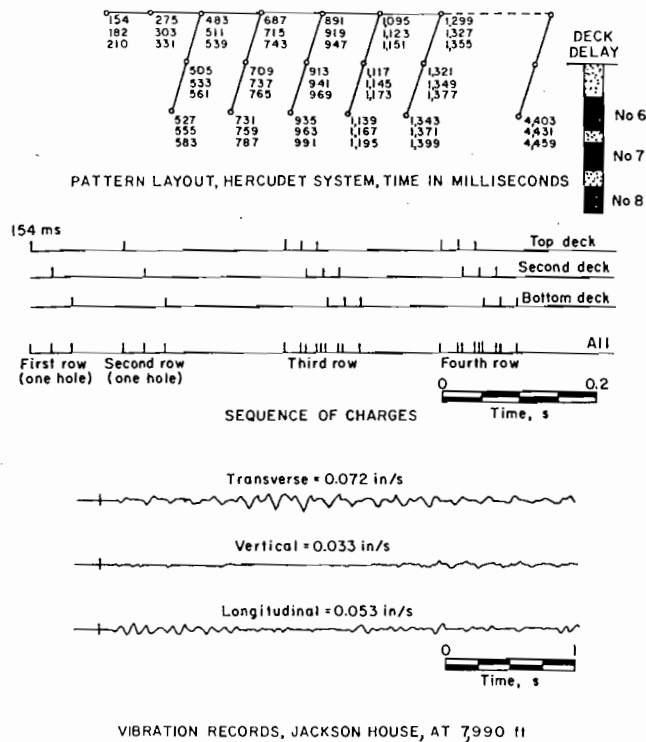


FIGURE 10.—Echelon blast, 200 ms between rows, 17 ms between holes in a row, January 12, 1985.

bottom boundaries having good reflecting properties. Extensive underground voids could provide such a reflecting surface, as could any low-velocity layer. The amounts of low frequency appeared to be related to the distance from the blast; however, trends were not consistent from site to site.

Site Differences for Vibration Waves

Six of the seven measurement sites had patterns of low frequencies resembling Rayleigh waves or Rayleigh and Love waves together. Two of these sites also occasionally had only high frequencies (e.g., 20 Hz) at the same time and for the same shots when low frequencies were being measured elsewhere. One site (Jackson) had every kind of combination at different times. The seventh site (Verhonik) had only transverse low frequencies, suggesting Love waves. This seventh site was in a different location from the others, east of the blasting rather than north. There was a 100-ft-thick layer of mine spoils between

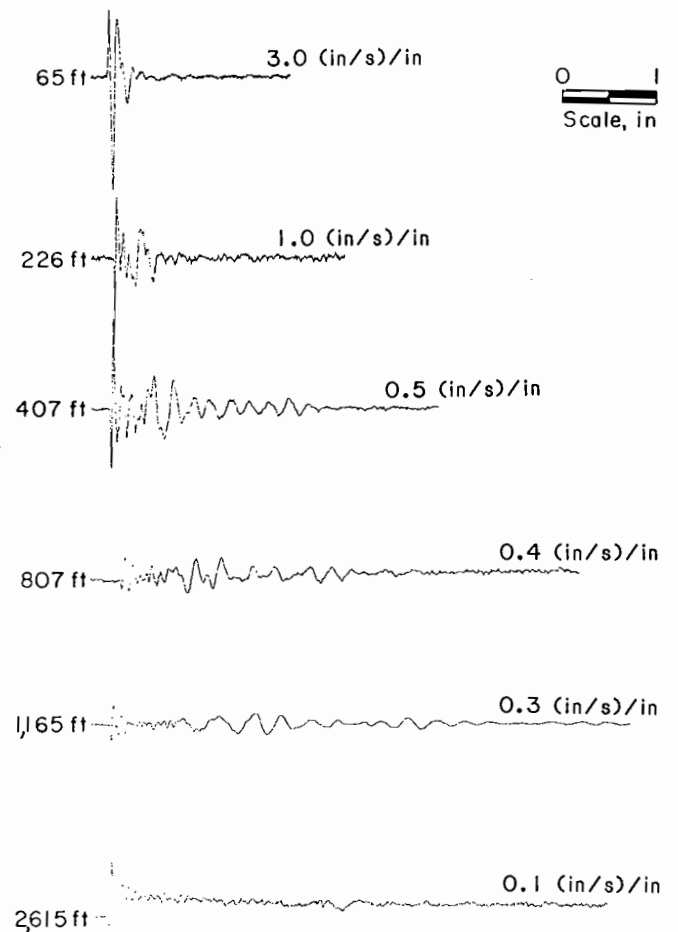


FIGURE 11.—Vibration records from single charge, vertical. Horizontal scale is 500 ms/in.

the site and the blasting. Table 3 summarizes all available time history records.

Single-Charge Blasts

The two special single-charge shots were fired to identify the blast initiation-sequencing influences on the wave forms as well as the previously discussed vibration amplitudes. They also showed the complexity of the propagating medium. Figure 11 is the vertical vibration record from a 1-m column of explosive that took about 0.3 ms to fully detonate. After propagating 65 ft, the vibration duration was over 150 ms. At a distance of 1,165 ft, the vibration duration was dominated by a 6.5-Hz wave lasting about 2 s. This observation suggests that any blast at this site is a potential

TABLE 3. - Vibration components showing very low frequencies (~4 Hz),¹
by measurement site and blasting method

Date	Residence						
	Massa	Volk	Hollings- worth	Zell	Polomski	Jackson	Verhonik
ECHELON BLASTS, 17 ms BETWEEN HOLES IN A ROW, 42 ms BETWEEN ROWS							
12-18-84.....		HF					
12-19-84.....		L, V					
12-22-84.....						HF	
12-28-84.....		V				HF	
1- 4-85.....		V				HF	
1- 9-85.....		L, V				HF	
ECHELON BLASTS, 17 ms BETWEEN HOLES IN A ROW, 100 ms BETWEEN ROWS							
3- 2-85.....						HF	
3- 2-85.....						HF	
3- 2-85.....	HF				HF	HF	
3-14-85.....						8 Hz	
3-16-85.....						8 Hz	
3-18-85.....						HF	
3-21-85.....						HF	
3-25-85.....						HF	
3-26-85.....						HF	
9-10-86.....	HF				HF	~10 Hz	
9-10-86.....	8 Hz				HF	~10 Hz	
9-10-86.....	HF				~5 Hz	~14 Hz	
9-12-86.....					HF	~9 Hz	
9-13-86.....	HF						
ECHELON BLASTS, 17 ms BETWEEN HOLES IN A ROW, 200 ms BETWEEN ROWS							
7- 2-84.....						T	
7- 3-84.....						T	
7- 6-84.....						L, V, T	
7- 7-84.....						L, V, T	
7- 7-84.....						L, V, T	
7-11-84.....						T	
7-11-84.....						T	
7-11-84.....						T	
7-13-84.....						T	
7-14-84.....						T	
7-28-84.....						L, V, T	
1- 5-85.....		HF					
1-12-85.....		L, V					
3- 5-85.....						HF	
3- 6-85.....	HF				L, V, T	HF	
3- 7-85.....						HF	
3- 9-85.....						HF	

See explanatory notes at end of table.

TABLE 3. - Vibration components showing very low frequencies (~4 Hz),¹
by measurement site and blasting method--Continued

Date	Residence						
	Massa	Volk	Hollingsworth	Zell	Polomski	Jackson	Verhonik
CASTING BLASTS							
8- 9-84.....				L,V,T			
8-13-84.....				L,V,T			
20-30-84.....						V	
11- 5-84.....						V	
11- 7-84.....						L,V	T
11-10-84.....	L,V,T					L,V,T	T
11-15-84.....	L,V,T						
11-17-84.....	L,V,T					L,V	
11-19-84.....	L,V,T					L,V,T	T
11-23-84.....		L,V,T				L,V	T
11-24-84.....						L,V	
11-26-84.....						L,V	
12- 1-84.....	L,V,T	L,V,T	L,V,T				
12- 4-84.....		L,V,T					
12- 6-84.....	L,V,T	L,V	L,V,T				T
12- 8-84.....							T
12-10-84.....	L,V,T	T	L,V,T				T ²
12-12-84.....		L,V,T	L,V,T				T ²
12-15-84.....		V					T ²
12-17-84.....		L,V,T	L,V,T				
1-12-85.....		L,V					T
1-14-85.....			L,V,T			L,V	T
1-17-85.....	L,V,T	L,V	L,V,T				T
1-21-85.....		L,V,T	L,V,T				T
1-25-85.....	L,V,T	L,V,T	L,V,T			V	T
1-28-85.....	L,V,T	L,V,T	L,V,T			V	T
1-31-85.....	L,V,T	L,V	L,V			L,V	T
2- 2-85.....	L,V,T	L,V,T	L,V,T		L,V,T		T
2- 6-85.....	L,V,T	L,V,T	L,V,T		L,V,T	V	T
2- 9-85.....	L,V,T	L,V,T	V		L,V,T	V	T
2-14-85.....	L,V,T	L,V,T			L,V,T		T ²
2-16-85.....	L,V,T	L,V,T					T ²
2-19-85.....	L,V	L,V,T			L,V,T		T ²
2-21-85.....	L,V				L,V	L,V	
3- 1-85.....						HF	

¹L Longitudinal (or radial). V Vertical. T Transverse.

Rayleigh surface waves would dominate L and V components; Love surface waves would dominate T component.

HF Higher frequency vibration with no clear components below 10 Hz.

²Vertical record may be defective; may have been present, but was not visible.

NOTE.--No entry means no records available.

TABLE 4. - Summary of two level loop surveys of eight Blanford residence.

House	Maximum elevation differences, ¹ ft		Maximum angular distortion ²	
	Sept. 1985	April 1986	Sept. 1985	April 1986
Ahlmeyers....	0.06	0.07	1/340	1/340
Albrecht.....	.12	.11	1/250	1/260
Finger, E....	.08	.07	1/300	1/340
Finger, O....	.09	.11	1/200	1/164
Jovanovich...	.30	.26	1/108	1/125
Marietta.....	.09	.09	1/560	1/533
Skorich.....	.06	.05	1/250	1/300
Zell.....	.37	.39	1/65	1/65

¹Accuracy is ± 0.01 ft.

²1/340 = Distortion of 1 part in 340.

low-frequency problem as the ground there favors such frequencies.

Theoretical Model

The O'Brien model computes surface waves generated by multiple reflections of compressive body waves in a low-velocity layer (10). Another version by Gupta is for shear waves. For a strong velocity contrast (strong reflector), the simplified relationship is

$$T = \frac{4h}{V_1}$$

where T is the surface wave period, h is the layer thickness, and V_1 is the propagation velocity for the low-velocity layer. Assumed for this simple model is that $V_1 \ll V_2$, with V_2 being the high-velocity layer.

At Blanford, researchers measured a V_2 of 10,000 ft/s and, using a 1.6-s arrival time difference, calculated a V_1 of 2,700 ft/s. For a 6.5-Hz surface wave ($T = 0.15$ s), an h of about 102 ft is indicated. Similarly, for a 4-Hz surface

wave, h is 167 ft. Depths to the coal beds as measured by Peabody at one site ranged from 85 to 394 ft. The extensively mined No. 5 was at 226 ft.

LEVEL-LOOP SURVEYS OF HOMES

Researchers surveyed eight Blanford home foundations to identify surface and structural changes from possible subsidence over abandoned workings (9). Only one of the eight homes was also monitored for vibration (Zell house). The primary selection criterion was the existence of a clear foundation horizon for surveying.

Most of the homes were out of level by significant amounts of up to one part in 65 (table 4). Boscardin cites that angular distortions or deflection ratios of 1 part in 300 can cause cracking in panel and load-bearing walls (11). A resurvey 7 months later showed no appreciable elevation changes. With these data alone, it is not possible to tell if the homes are distorted or strained, or if these differences were originally built in. Apparently, no significant long-term changes are occurring.

CONCLUSIONS

The propagating medium appears primarily responsible for the adverse vibration impacts in Blanford, through three mechanisms: (1) It favors generation of low-frequency surface waves of several types, with frequencies between 4 and 10 Hz, (2) it has the appearance of reduced

vibration attenuation (higher amplitudes) with distance compared with other coal mine sites, and (3) it produces interactions between delayed charges beyond those expected from the blasts as designed, because of constructive wave interference for these long-period waves.

Although further study of the subsurface conditions is needed in order to completely understand all the factors, the observed surface waves are consistent with a strongly reflecting subsurface interface at a depth of about 175 ft, or about the same as the depth of the extensively mined No. 5 coalbed. This agrees with theoretical models that predict low-frequency waves from strongly reflecting near-surface horizontal layers.

The existing geologic structure between the main part of Blanford and the active pit is another possible or contributing cause of the low-frequency blast vibrations. This region has a coal cutout or zone where the coal and other sedimentary rock beds are missing, replaced by fill characterized as sandy, gravelly drift. Vibrations propagating through such material often have abnormally low frequency. Such a medium could also explain the

rapid vibration amplitude attenuation observed between 407 and 807 ft (fig. 5).

Blast designs are also significant. The widely adopted 8-ms charge-separation criterion appears not to apply for this low-frequency site, as it was also previously found suspect in a 1979 study by Wiss of large-scale surface coal mine blasting (8). Vibration frequency characteristics appear to reflect periodicities in the blast design timing, particularly close to the shot. The larger casting blasts produced the clearest 4-Hz surface waves. By contrast, the more complex echelon blasts produced the largest vibration amplitudes for a given charge weight per delay. Results suggest that longer delays be used between charges to prevent constructive addition of vibration amplitudes for such low-frequency cases.

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COMPUTER MODELING OF ROCK MOTION

By Stephen A. Rholl¹

ABSTRACT

A computer model of rock motion due to blasting is presented. The code, CAROM, was developed at Sandia National Laboratories (SNL) to predict rock motion and final muckpile distribution. Researchers at the Bureau of Mines applied the code

to simulate bench blasting during full-scale fragmentation tests at a nearby rock quarry. Results of the code are shown for the first 2 s following explosive detonation.

INTRODUCTION

A contributing factor to the overall efficiency of surface mining is the final muckpile distribution of the blasted rock fragments. This is especially true in mining operations utilizing overburden casting to reduce stripping costs. As the overburden-to-coal ratio increases, it is essential, in many operations, that as much fragmented rock as possible be thrown into the pit or onto the spoil bank.

To optimize overburden casting, mine operators most often vary powder factor, explosive type, drill patterns, and/or delay timing. These variations are normally tested by trial and error based on the experience of blasting personnel. Unfortunately, this can often be time consuming and expensive. An alternative approach to the optimization problem,

which can quickly analyze technical problems, involves computer-simulated blasting.

Much of the early work to develop simple and effective computer models to simulate blasting was done by SNL. In 1983, researchers developed BUMP (1),² the first code to use very simple interaction laws to reduce computational times. A stabilized and improved code, CAROM (2), was later introduced. Both codes were written to study rock motion in oil shale crater tests.

The Bureau of Mines initiated contract JO245011 with SNL to modify these codes for use in modeling bench blasting. Subsequently CAROM was modified and used in blast design research. SNL staff also provided appropriate values for input parameters to execute the code.

MODEL DESCRIPTION

The CAROM code is a two-dimensional distinct-element code. That is, a group of distinct elements is used to describe the system of rock fragments undergoing motion. The shape of these elements is in theory arbitrary, but circles are most often used to simplify time-of-collision calculations. The size of the elements is unrestricted, and often a single element is used to represent a group of fragments. Studies by Gorham-Bergeron (2) have shown that the size of the material elements does not have a strong

influence on the calculated material motion for modeling oil shale cratering experiments.

The initial configuration of the elements used in the CAROM code is specified by the user. In addition, the initial conditions (both the velocity and acceleration of each element) must likewise be specified. CAROM simply lets the elements move about until the code detects a collision between two elements. The collisions are handled by the theories of classical mechanics, namely, conservation

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²Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

of linear momentum and Newton's collision rule. The collisions themselves can be treated elastically or inelastically, although the latter is more often assumed.

Because CAROM calculates the time the next collision will occur, the code is inherently stable. When a collision is detected, CAROM redistributes the linear momentum of the elements and then repeats the process. Collisions with a fixed boundary (or boundaries) are also permitted and treated in a similar way.

MODEL APPLICATION

The CAROM code was applied by Bureau researchers to simulate rock motion during bench blasting at a local quarry. A two-dimensional discrete rock model describing the 22-ft highwall is shown in figure 1. A series of 33 elements, each with a radius of 1 ft, represent the highwall. The explosive column is also indicated in the figure.

After detonation of the explosives, shock waves are generated in the surrounding rock, which not only fracture the rock, but also impart momentum causing unconfined rock to begin motion. In addition, large gaseous pressures are created within the boreholes that exert forces on the elements causing them to accelerate.

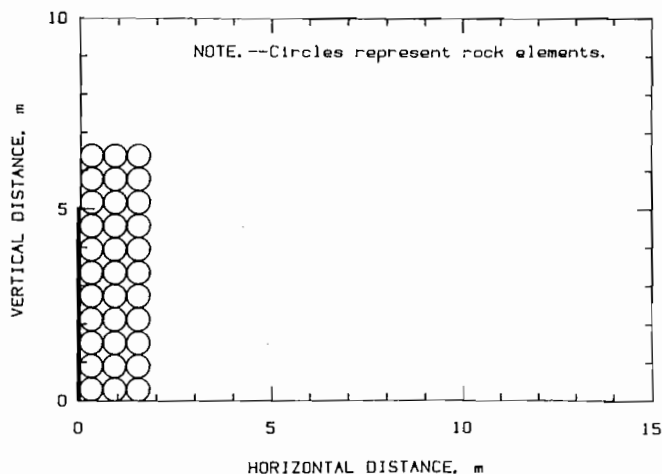


FIGURE 1.—Highwall representation by discrete rock elements. Heavy line on left indicates explosive column.

Although it is possible to include friction in the calculations, CAROM usually assumes that the coefficient of friction between the elements is zero. However, it should be noted that at the boundaries representing the pit floor, the elements are not allowed to slide. Too much sliding was deemed unrealistic, and therefore the code assumes infinite friction between the elements and the pit floor boundaries.

Research at SNL has shown that a dynamic finite-element code, DYNA2D (3), can be used to describe the explosive detonation, rock fragmentation, borehole pressures, and initial rock motion. The information calculated by DYNA2D can be used as the input necessary for the CAROM code. Both the momentum imparted to the rock fragments and the blast pressures due to the explosive gases are chosen based on DYNA2D calculations. DYNA2D cannot be used to model the entire blast because the code is a continuous representation, meaning it cannot describe material that breaks apart.

Beside the initial conditions, further assumptions were necessary prior to running the CAROM code. For the quarry tests, the size of each element was fixed and not allowed to change. This meant that elements were not permitted to break up into smaller fragments. Also, to simplify calculations, the elements were not allowed to roll and therefore, could not possess any angular momentum. The coefficient of restitution for collisions between elements was chosen to be a constant for all of the elements. A value of 0.01 was assumed based on research at SNL. This implies that all of the collisions are treated as nearly perfectly inelastic. One final assumption deals with the collision criterion. Based on work at SNL, a collision is defined to occur when the distance between the centers of two elements equals 1.9 ft.

The quarry tests were conducted as single-row bench blasts to study the effect

of delay timing on fragmentation. Four blastholes were fired in each of six full-scale shots. Because CAROM is at the moment only a two-dimensional code, it cannot be used to predict the effects

of delay timing between holes in a single row. However, once a three-dimensional code becomes available, these studies can be done.

RESULTS

All of the test shots were monitored with two high-speed cameras operating at 500 frames per second to record the actual rock motion. Analysis of the resulting films yielded data providing a basis for confirmation of the CAROM code. Of primary interest was the initial outward velocity of the bench face material. This was observed in all tests to be about 15 m/s.

The output from the CAROM code is shown in figures 2-11 for the first 2 s of rock motion at 200-ms intervals. The average velocity for any element can easily be calculated from the figures. The element representing the leading edge of the blasted rock face, as shown in figure 2, has an average velocity determined to be 16.5 m/s, which is in excellent agreement with the cinematographic data.

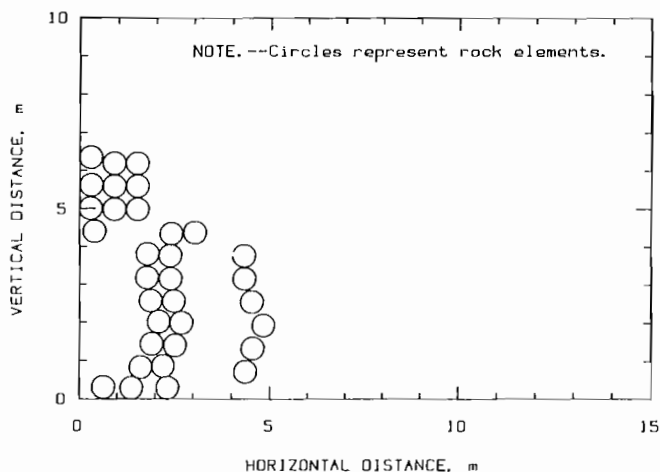


FIGURE 2.—Predicted rock motion at time equals 0.2 s.

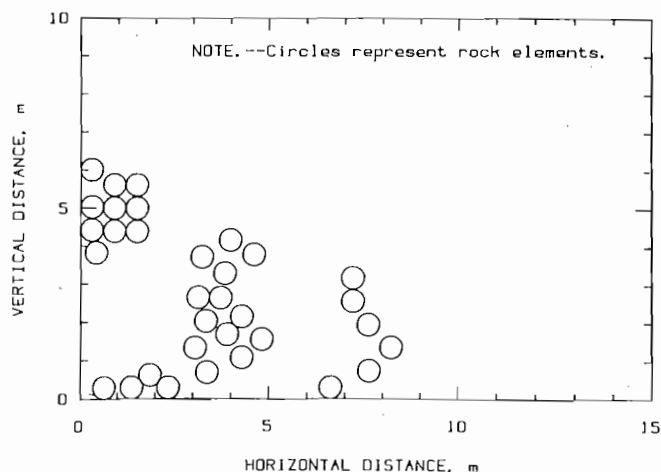


FIGURE 3.—Predicted rock motion at time equals 0.4 s.

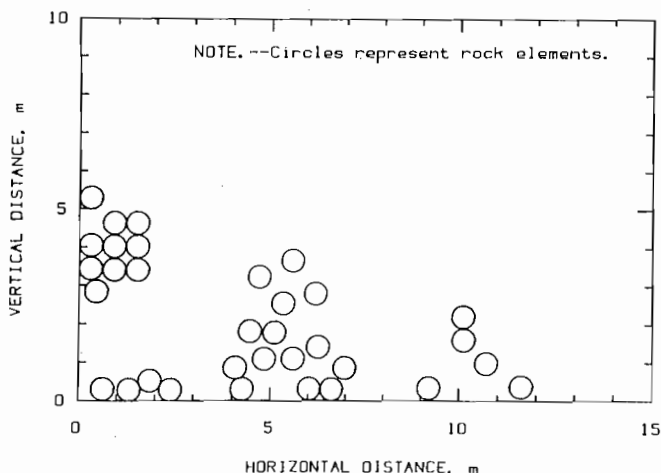


FIGURE 4.—Predicted rock motion at time equals 0.6 s.

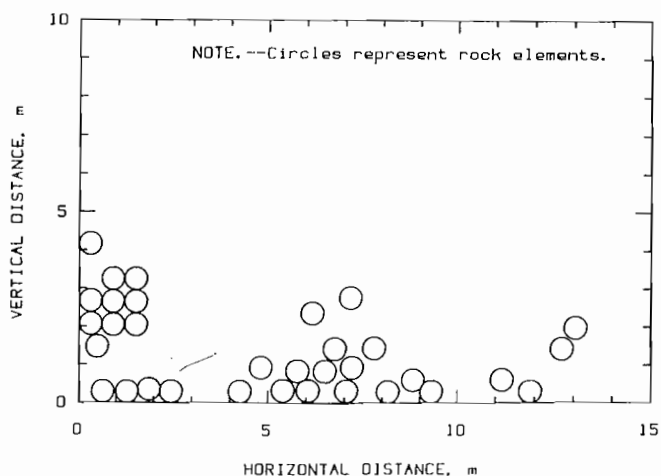


FIGURE 5.—Predicted rock motion at time equals 0.8 s.

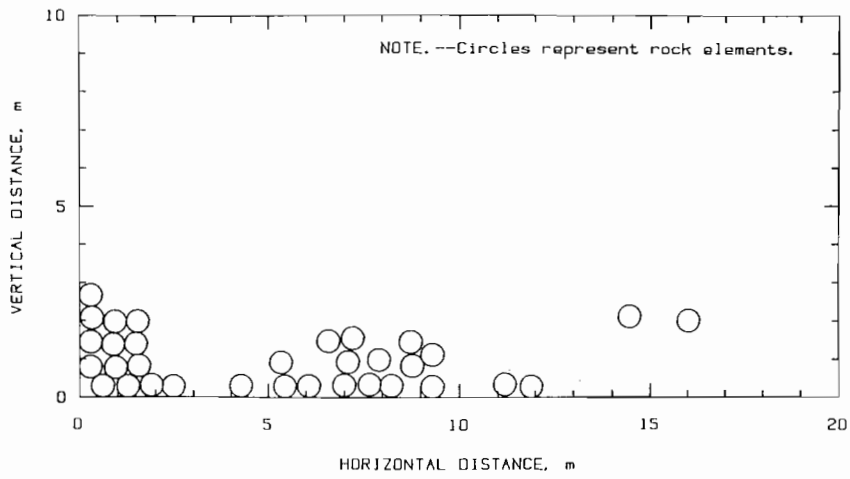


FIGURE 6.—Predicted rock motion at time equals 1.0 s.

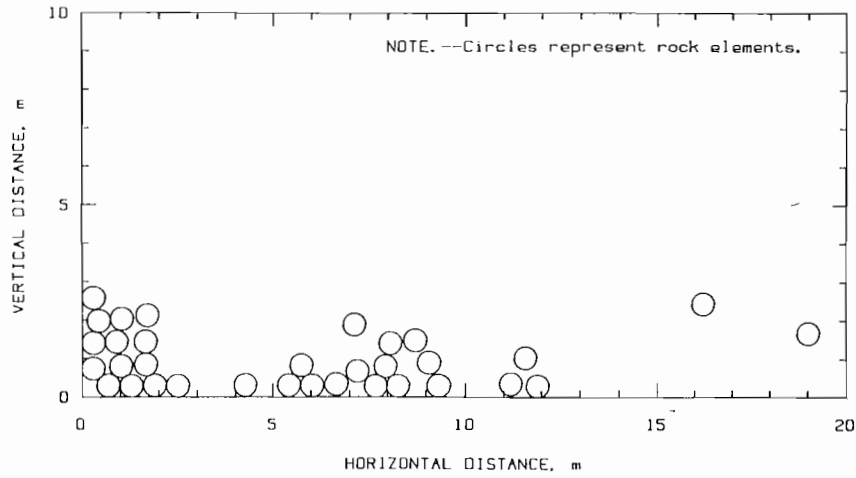


FIGURE 7.—Predicted rock motion at time equals 1.2 s.

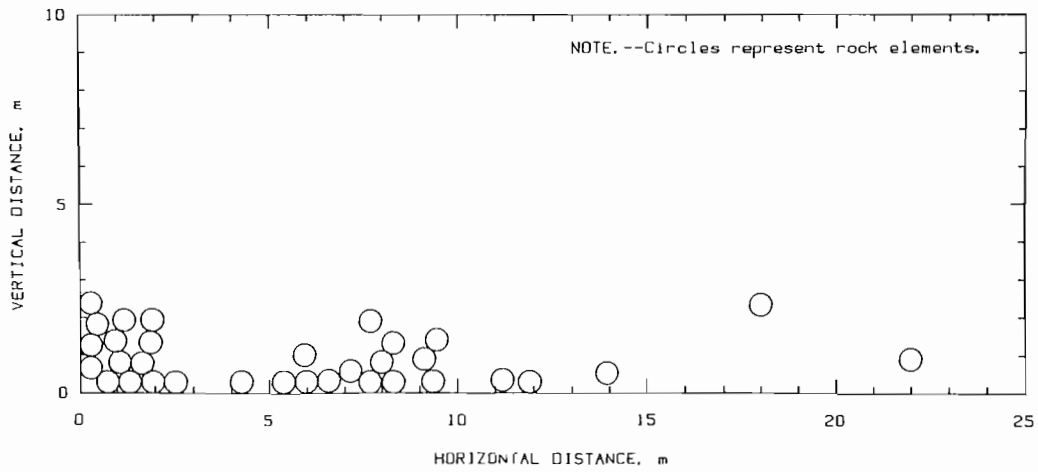


FIGURE 8.—Predicted rock motion at time equals 1.4 s.

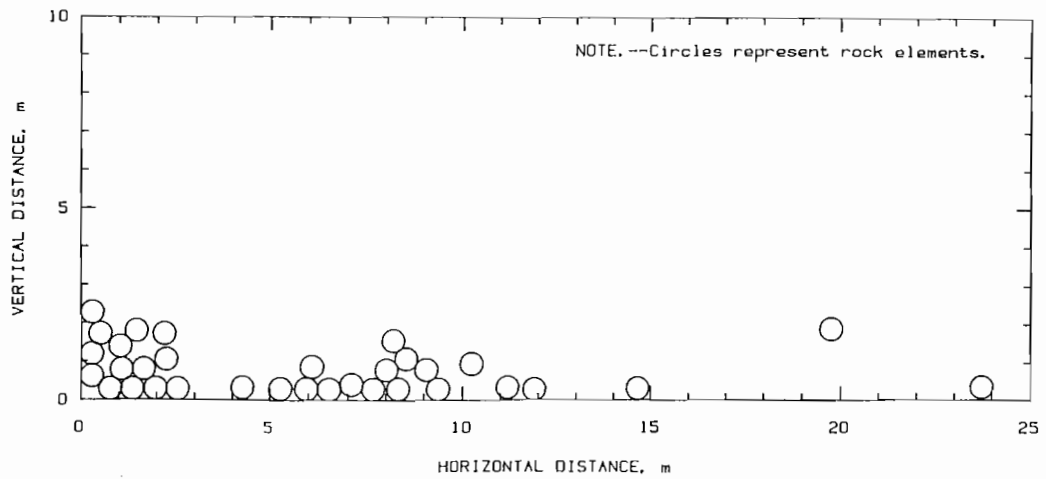


FIGURE 9.—Predicted rock motion at time equals 1.6 s.

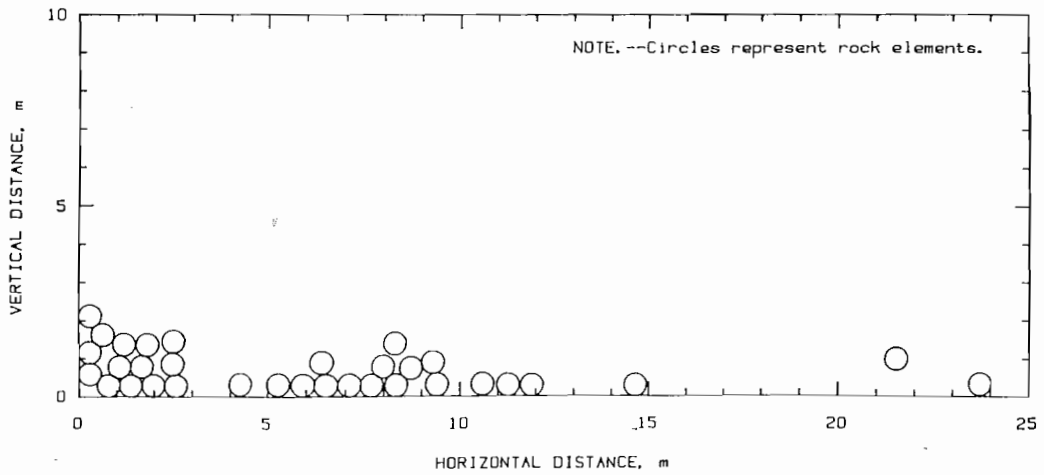


FIGURE 10.—Predicted rock motion at time equals 1.8 s.

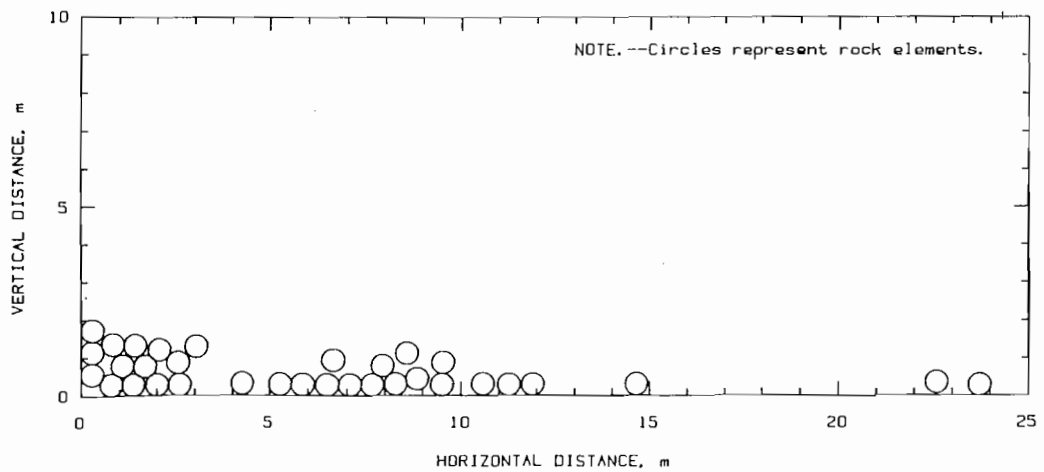


FIGURE 11.—Predicted rock motion at time equals 2.0 s.

In the fragmentation test shot utilizing 4.0 ms/ft of burden as the delay timing, the maximum rock throw was measured to be 30 m. The CAROM code predicted, as shown in figure 9, at least one element thrown to about 24 m after 1.6 s of time passage in the calculations. CAROM is also capable of predicting vertical motion of the bench top. However, in the series of results presented here no upward motion is observed. It is possible that CAROM still did indicate such motion, but that it occurred prior to 200 ms (fig. 2). At this time CAROM shows the surface material falling downward under the influence of gravity. The high-speed films confirmed that there was

little or no vertical rock motion in the field tests.

One of the deficiencies of the code appears to be its inability to predict the profile of the rock motion of the bench face. As shown in figure 2, CAROM predicts almost the entire face to break off as a slab. It is likely that this is due to improper balancing of the input parameters. That is, there is too much emphasis on the imparted momentum due to the shock wave impulse, and not enough emphasis on the accelerations due to the forces created by the pressures of the explosives gases. Research continues to develop a step function approximation for the high-pressure gases.

CONCLUSIONS

The computer code CAROM, which models rock throw in blasting, has been briefly discussed. The code was used to simulate rock motion at a limestone quarry, and the results of the analysis have been presented. The code accurately predicted the velocity of burden movement and the

maximum throw of the rock fragments. CAROM is still being developed, and many improvements are being implemented. The objective continues to be to provide mine operators with accurate predictions of rock motion and final muckpile distribution.

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INFLUENCE OF BLAST DELAY TIME ON ROCK FRAGMENTATION: ONE-TENTH-SCALE TESTS

By Mark S. Stagg¹ and Michael J. Nutting²

ABSTRACT

The Bureau of Mines is studying blast delay timing influences on rock fragmentation in a series of tests that started in 3-ft concrete blocks and includes reduced-scale and full-scale bench blasts. This paper reports on the reduced-scale tests. In a 45-in-high dolomite bench, 18 single-row blasts were fired with 15-in burdens. Spacings were 21 and 30 in. Delay intervals ranged from 0 to 45 ms, equivalent to 0 to 36 ms/ft of burden. Each shot was instrumented for strain and pressure for both in situ dynamics and interactions between blastholes. All fragmented rock was screened.

The finest fragmentation occurred at blasthole delay intervals of 1 to 17

ms/ft of burden. In this range, stress-wave-induced strains interacted with longer lasting gas-pressure strains from earlier holes. Coarse fragmentation resulted from short delays (<1 ms/ft), where breakage approached presplit conditions with a major fracture between blastholes and large blocks in the burden region. Coarse fragmentation also resulted from long delays (>17 ms/ft), with explosive charges acting independently. The broad acceptable range provides blast design tools for a variety of purposes, including optimum muckpile displacement and vibration control.

INTRODUCTION

The explosives industry is developing and testing delay blasting caps of improved accuracy. Precise delays have been cited as factors in controlling blast vibration amplitude and frequency and improving fragmentation (1-2).³ However, data on complete fragmentation assessment of shots initiated with precise delays is limited. As part of its blasting research program, the Bureau of Mines is examining the influence of timing intervals by completely screening the blast-induced rock. Three- and four-hole shots have been conducted in concrete blocks in the laboratory and at reduced (45-in bench) and full (22-ft bench) scale in the field. Tests thus far have mostly been concerned with the effect of delay time on fragmentation and on the

interaction between shotholes. Initial testing in the laboratory provided an effective means for establishing a methodology of controlled experimentation. The tests at reduced scale in the field provided experience in fragmentation assessment techniques and results that could be used to optimize the expensive full-scale field tests. The full-scale field tests are currently in progress. This paper discusses the reduced-scale field tests and results.

The reduced-scale field tests were conducted at the University of Missouri's Experimental Mine in Rolla. This site was chosen because of its accessibility and geology, and the cooperation available from the University. Furthermore, the results of previous research conducted at the mine on blast design and fragmentation were reported in several theses (3-5). These studies investigated various design factors affecting fragmentation, such as coupling, initiation sequence, primer location, and airgap, and provided a comparison with the Bureau test results.

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

SITE

The experiment was conducted in a 45-in bench of dolomite, part of the Jefferson City Formation. The rock is of irregular grain size, 10 pct calcite, and thick bedded, with a specific gravity of 2.65 and compressional and shear velocities of 14,800 and 8,100 ft/s, respectively (5). Bureau researchers verified these values by in situ seismic measurements behind the blastholes, finding 14,700 ft/s (compressional) and 8,100 ft/s (shear).

The reduced-scale tests were designed to be geometrically proportional to

typical full-scale bench blasts with dimensions about 10 pct of full scale. However, at reduced scale, rock structures such as bedding and jointing are exaggerated and can have an unrealistic effect. The massive dolomite at the Rola site provided a good medium for the reduced-scale testing since only three bedding planes were in the 45-in bench and no jointing was observed within any of the test shots.

PROCEDURES

In order to prepare the pit and bench for the tests, development work was necessary to provide consistent geometry of the working faces. Horizontal holes were drilled 45 in above the pit floor to give the proper bench height, and vertical holes were drilled to obtain a vertical face. For each test shot, an open end at an angle of approximately 135° was formed, as shown in figure 1. To form the single-row pattern, three shotholes were drilled to 50-in depth, including 5 in of subdrill. The burden was held constant at 15 in, and spacings were 21 and 30 in, spacing-to-burden ratios (S/B) of 1.4 and 2.0, respectively. All faces were cleaned with a scaling bar and blown

with air to remove loose material, and the entire area was swept clean prior to each shot.

Each hole contained 144 g of 60 pct extra-high-density dynamite tamped into a 1/2-in-ID plastic tube 40 in long. Each charge was bottom-primed and initiated by an exploding bridgewire detonator (EBW). Complete coupling was assured by placing the charge into grout-filled holes. The initiation system for the EBW's consisted of a power supply, firing module, and digital delay generator with a firing accuracy of 0.0025 pct times the delay time or ± 50 ns, whichever is greater.

Most shots were instrumented with dynamic-strain and pressure gauges (figs. 2-3). The strain gauges were a six-component type, built after a design by Reed (6) as modified by Anderson (7). These gauges were grouted into the burden region at various locations between the boreholes. Pressure gauges were placed either above the strain gauges or in inclined holes in the face, which were filled with a water-revert mixture. The pressure gauges were of two types, carbon resistors dipped in liquid tape (insulating coating) and Navy-built tourmaline gauges in an oil-filled boot. Also, a fiber optic system was used to measure detonation velocities. Data were recorded on a 28-channel Wide Band I (80 kHz) recorder and a digital oscilloscope, with a 0.5- μ s response rate.

To contain flyrock and minimize secondary breakage, the entire shot was covered

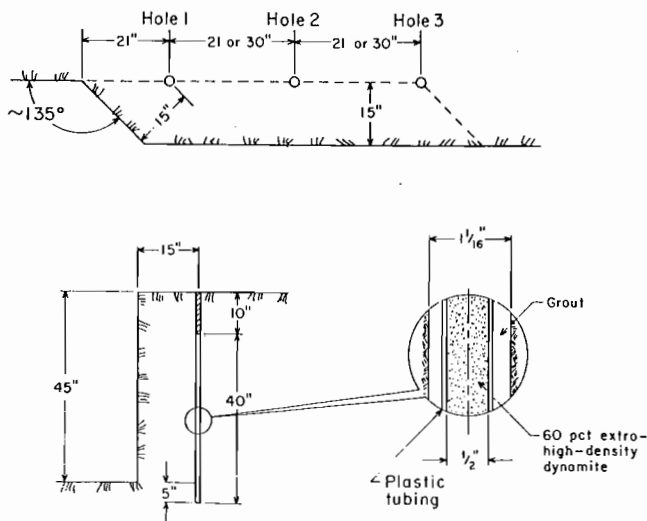


FIGURE 1.—Shot pattern and blast design for reduced-scale tests.

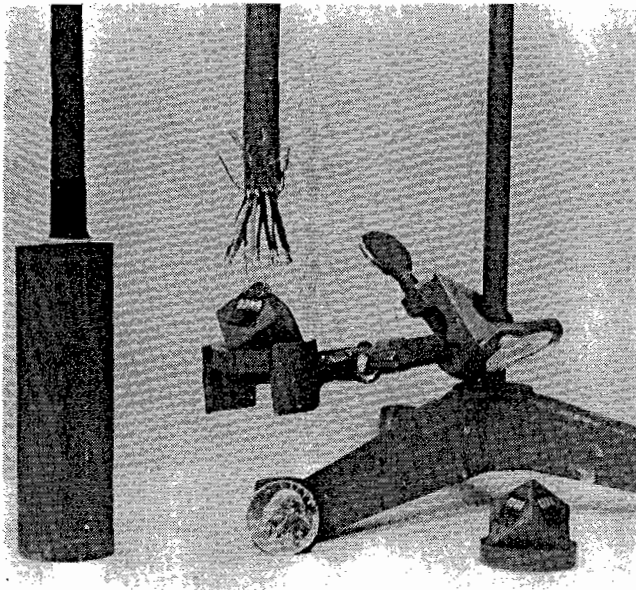


FIGURE 2.—Stages of assembly of six-component strain gauge.

with a blasting mat held in place with timbers and anchoring cables (fig. 4). The area in front of the shots was covered with a plastic sheet to aid in identifying blasted material. Flyrock went beyond the sheeting for only a few shots. This rock was identified when possible and included with the muckpile.

Screening of the muckpile began immediately after each shot. All fragments 3 in and larger were sized and weighed in the pit. The pit size fractions were minus 3, plus 3, minus 6, plus 6, minus 12, and plus 12 in. Material passing the 3-in screen was loaded into containers, removed from the pit, and mechanically shaken through screens. These size fractions were plus 1-1/2 minus 3, plus 3/4 minus 1-1/2, plus 3/8 minus 3/4, plus 3/16 minus 3/8, and minus 3/16 in. From the weight of each fraction, its percentage of the total muckpile was calculated.

ANALYSIS AND RESULTS

A total of 24 blasts were detonated. Two were development shots. Of the remainder, nine at 21-in spacing and nine at 30-in spacing were completely screened, three misfired, and one was a

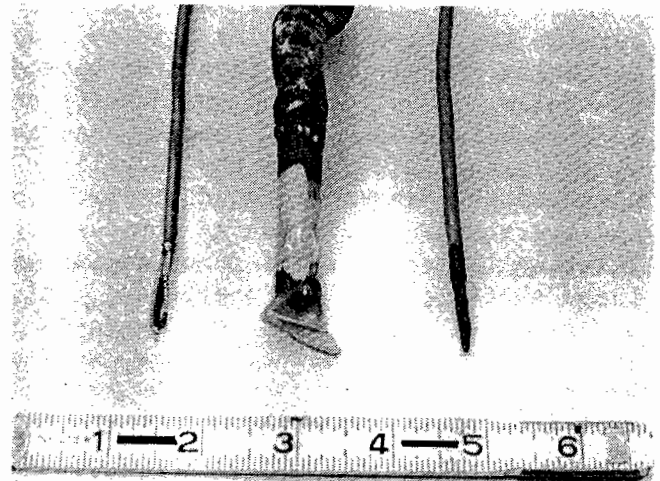


FIGURE 3.—Carbon resistor (left and right) and tourmaline (center) pressure gauges.



FIGURE 4.—Pit area with blasting mat and timbers covering test shot.

single-hole test shot. Delay intervals ranged from simultaneous to 36.0 ms/ft of burden. Table 1 lists the reduced-scale (RS) shots.

Since these tests were designed to determine the effect of delay time on fragmentation, i.e., the interaction between shotholes, it was decided to identify any overbreak from each shot and to size and weigh it separately from the muckpile. A third of the shots produced no back- or end-overbreak. There was no obvious correlation between spacing or timing and those shots producing overbreak, which was generally oversized and averaged 10 pct of the total blasted rock weight. Because the overbreak skewed the particle size distributions at the high end and

came from outside the shot pattern, it was removed from the corresponding size fraction and not included in the analysis of fragmentation. Table 2 lists the sieve weights for the shots.

The cumulative percent passing versus sieve size is plotted in figure 5 for four shots at each spacing, covering the range of delay times tested. The material that passed through the 24-in sieve would in most cases have passed through a much smaller sieve, even down to 13 in. Since the largest size piece was not measured and 24 in is too large, the 24-in

TABLE 1. - Reduced-scale (RS) test shots

Shot ¹	Spacing, in	Delay time		Shot ¹	Spacing, in	Delay time	
		Between holes, ms	ms/ft of burden			Between holes, ms	ms/ft of burden
RS-3.....	30	20.0	16.0	RS-14.....	30	1.25	1.0
RS-4.....	30	7.5	6.0	RS-15.....	30	0	0
RS-5.....	30	5.0	4.0	RS-16.....	30	45.0	36.0
RS-6.....	30	20.0	16.0	RS-17.....	30	2.5	2.0
RS-9.....	21	0	0	RS-18.....	21	21.0	16.8
RS-10.....	21	30.0	24.0	RS-19.....	21	1.75	1.4
RS-11.....	21	14.0	11.2	RS-21.....	21	.4375	.35
RD-12.....	21	3.5	2.8	RS-22.....	21	8.75	7.0
RS-13.....	30	30.0	24.0	RS-24.....	21	45.0	36.0

¹Not listed: Shots RS-1 and RS-2 (development shots), RS-7, RS-8, and RS-20 (misfires), and RS-23 (single-hole shot).

TABLE 2. - Weight of rock fragments at various screen sizes, pounds

Screen size, in...	-3/16	+3/16 -3/8	+3/8 -3/4	+3/4 -1-1/2	+1-1/2 -3	+3 -6	+6 -12	+12 -24
RS-3.....	208	243	268	535	639	1,389	1,607	241
RS-4.....	172	243	263	570	675	1,987	2,022	241
RS-5.....	215	237	258	498	523	1,628	1,941	159
RS-6.....	188	216	257	466	596	1,656	1,745	117
RS-9.....	196	182	196	356	421	1,197	1,173	1,415
RS-10.....	232	259	320	552	631	1,372	2,100	172
RS-11.....	305	318	391	587	572	1,149	800	0
RS-12.....	233	255	298	557	639	1,521	1,147	0
RS-13.....	238	238	276	452	516	1,550	1,857	520
RS-14.....	217	240	253	541	612	1,911	1,718	266
RS-15.....	167	182	208	466	521	1,657	2,092	462
RS-16.....	244	250	246	471	512	1,607	2,019	536
RS-17.....	242	247	309	522	630	1,801	1,546	446
RS-18.....	246	275	342	522	590	1,197	876	0
RS-19.....	202	234	302	548	635	1,219	911	0
RS-21.....	203	194	198	377	449	1,132	1,507	688
RS-22.....	268	235	258	454	501	1,131	1,022	143
RS-23.....	71	60	61	123	138	364	550	178
RS-24.....	295	249	270	486	541	1,060	1,110	422

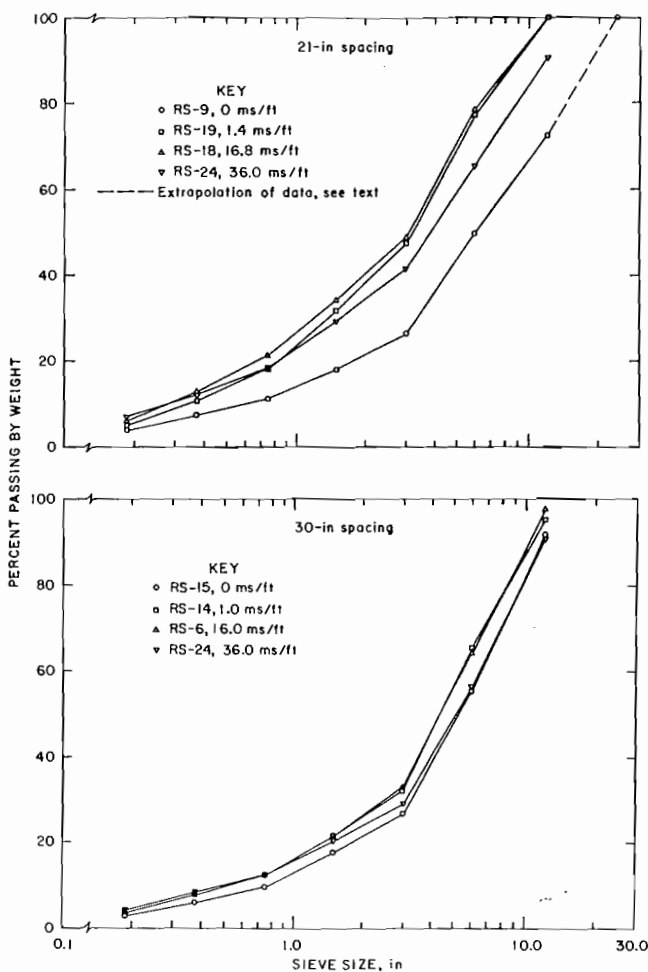


FIGURE 5.—Particle size distribution for shots at 21- and 30-in spacings.

point has been omitted from the curves, except that it was used to obtain the 80-pct-passing value for shot RS-9, as shown in figure 5. The curves in figure 5 are representative of the results, which showed that a dramatic improvement, a 20- to 50-pct reduction in average size, occurred at delays of ~ 1.0 to 17.0 ms/ft compared with simultaneous initiation. At delays longer than 17.0 ms/ft, the average size increased ~ 20 to 50 pct.

The 20-, 50-, and 80-pct passing sizes versus delay period, shown in figure 6, were determined for all the tests from percent-passing curves similar to those shown in figure 5. The delay period had little effect on the size of fragments in the 20-pct-passing fraction, except that they were slightly coarser at the simultaneous shots. At both spacings, the 50- and 80-pct-passing fractions show

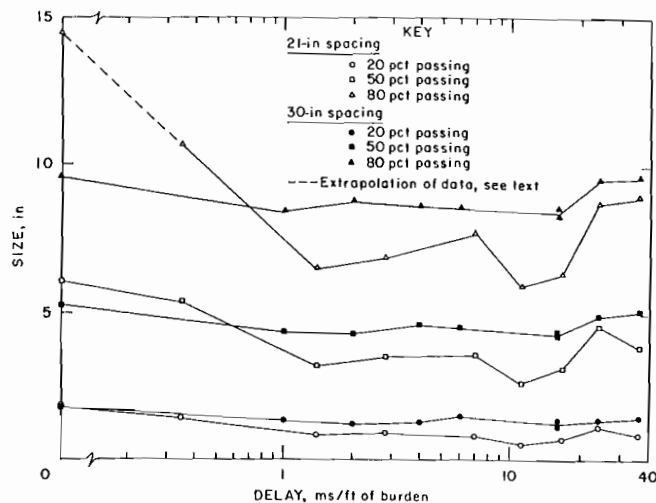


FIGURE 6.—Size at 20, 50, and 80 pct passing versus delay period for shots at 21- and 30-in spacings.

that delays of 1 to 17 ms/ft produced the finest fragmentation. Poorest or coarsest fragmentation was observed for simultaneous shots and for shots with delay intervals >24 ms/ft, although the differences at the longer delays were smaller with the 30-in spacings. The improved fragmentation obtained for most of the shots at 21-in spacings, as compared with 30-in spacings, was expected and was due in part to a higher powder factor. However, the coarsest fragmentation resulted at the very short delay times for the 21-in spacing tests, even though there was a higher powder factor.

To quantify the test results and ultimately develop an equation to predict fragmentation, it is necessary to develop a mathematical description of the cumulative percent-passing data. Previous researchers have used regression analysis to fit observed blast fragmentation data to logarithmic, power (8), Gaussian (2), and Weibull distributions (9-10). Analysis-of-variance tests can then be used to determine the statistical significance of the effect of delay time on the cumulative size distribution functions. These tests determine whether a single regression line can be used to represent the combined results of several shots. If pooling or combining the results tests positive, then there exists a statistical inference that the delay time has not significantly affected fragmentation.

Statistical tests were run on several regression line fits to the data, and the correlation coefficients (R) for the various distributions are given in table 3. The data shown in figure 6 suggest that the finer size material was less affected by delay time. An examination of the weights of the material passing the 3-in sieve found minimal variability, as shown in figure 7, except for shots at delays less than 1.0 ms/ft. Excluding these shots and an extremely high (2,170 lb) outlier value, the weight of material passing the 3-in sieve ranged from 1,716 to 1,994 lb and 1,720 to 1,950 for the 21- and 30-in spacing shots, respectively. The nearly constant weight of finer material suggests that a fractured zone extends around the borehole, which, assuming a cylindrical nature, would have a radius of 10 in. This equates to 40 explosive radii, within the range of the damage zone predicted by Siskind (11-12), Olson (13), and others in terms of charge radii (i.e., 20 to 40 radii). As shown in figure 8, the fines best fit a

log-normal distribution. Analysis-of-variance tests were run for all finer size data from delays of 1 to 36 ms/ft, and one curve could be used to represent the data at both spacings. For shots at delays of less than 1 ms/ft, the weight of fines was reduced.

The correlation coefficients in table 3 show that a Gaussian distribution for the coarser size data usually produced the best fit. The Gaussian, power, and Weibull fits to the coarse size data are shown in figure 9 for shots RS-3 and RS-13. It was observed that the 1-1/2-in sieve material also fit the Gaussian distribution, and this material was included in the regression analysis of table 3. The coarse material is assumed to be generated primarily between the boreholes outside the extended fractured zone that exists around the borehole.

An attempt to pool all of the data using a Gaussian distribution indicated (at a 95-pct-confidence level) that the delay time was indeed significant. Further analysis-of-variance tests were conducted

TABLE 3. - Correlation coefficients for Weibull, power, and Gaussian distributions

Shot	Weibull		Power		Gaussian B ²
	A ¹	B ²	A ¹	B ²	
RS-3.....	0.9923	0.9861	0.9963	0.9980	0.9999
RS-4.....	.9906	.9858	.9961	.9944	.9995
RS-5.....	.9833	.9758	.9972	.9941	.9993
RS-6.....	.9856	.9815	.9874	.9953	.9998
RS-9.....	.9961	.9936	.9978	.9942	.9837
RS-10.....	.9859	.9720	.9966	.9980	.9979
RS-11.....	.9966	.9852	.9876	.9956	.9996
RS-12.....	.9958	.9845	.9948	.9941	.9992
RS-13.....	.9893	.9847	.9973	.9946	.9988
RS-14.....	.9893	.9868	.9972	.9923	.9981
RS-15.....	.9895	.9848	.9982	.9948	.9993
RS-16.....	.9872	.9830	.9971	.9940	.9992
RS-17.....	.9832	.9906	.9970	.9928	.9945
RS-18.....	.9966	.9875	.9902	.9933	.9998
RS-19.....	.9978	.9909	.9913	.9932	1.0000
RS-21.....	.9921	.9887	.9981	.9970	.9980
RS-22.....	.9904	.9862	.9966	.9957	.9994
RS-24.....	.9956	.9921	.9968	.9980	.9964

¹All data points were used in regression analysis except the +12-, 24-in data point.

²Only data at 1-1/2-, 3-, 6-, and 12-in sizes were used; 100-pct point was also excluded.

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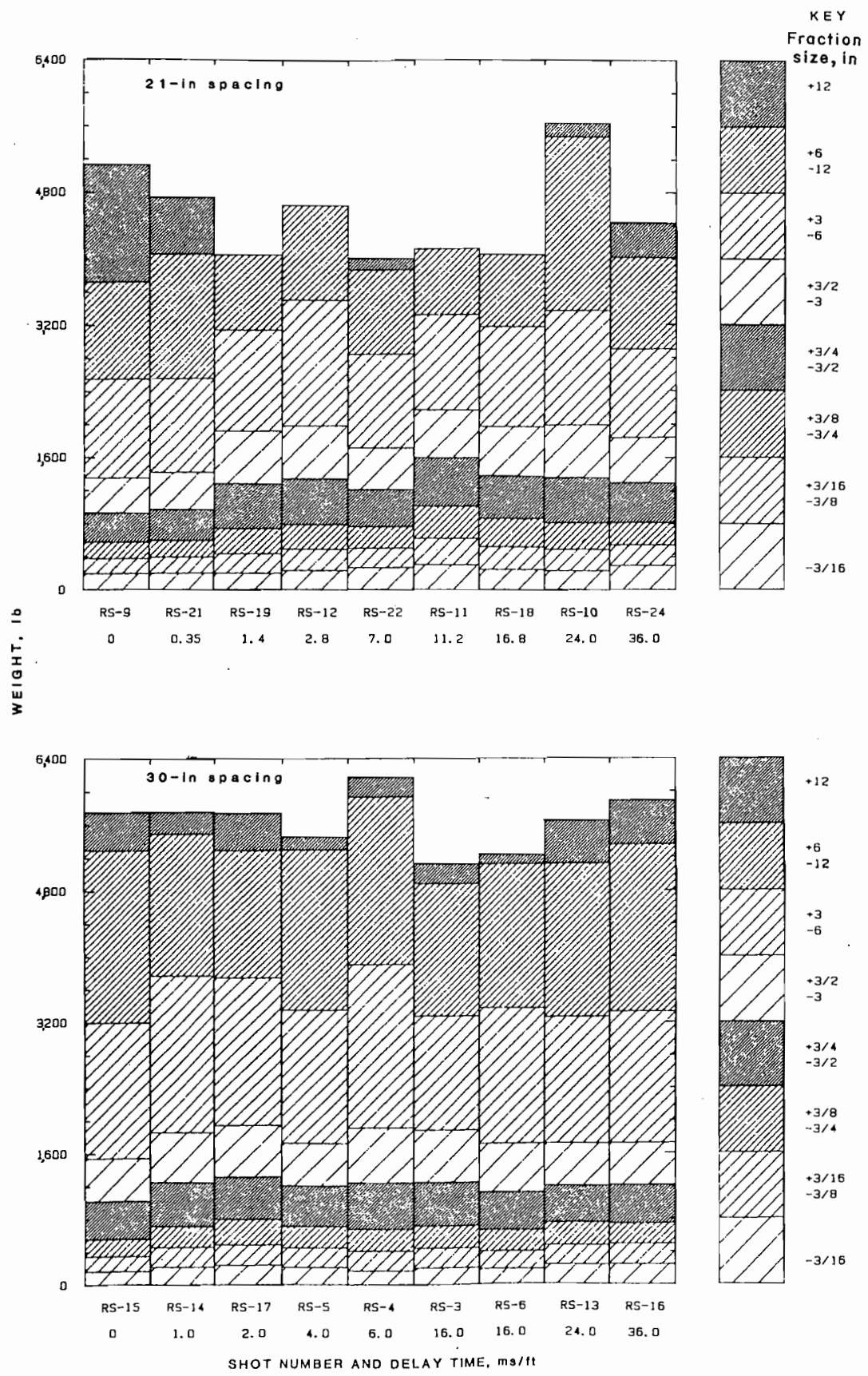


FIGURE 7.—Weight retained on each size fraction for shots at 21- and 30-in spacings.

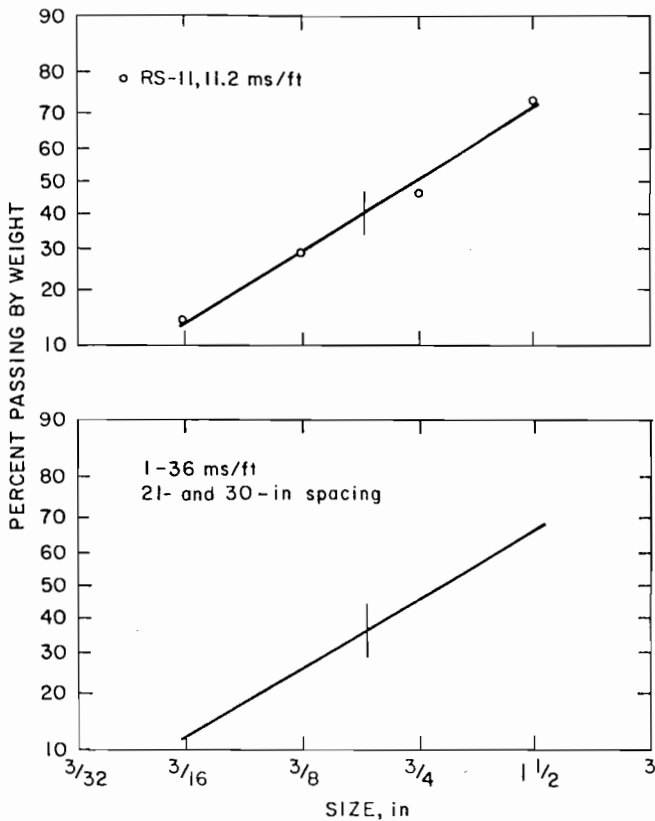


FIGURE 8.—Log-normal distribution for material passing 1 1/2-in sieve size.

to see if certain shots could be pooled to form one regression line. For example, the 21-in shots at 11.2 and 16.8 ms/ft could be combined (i.e., there was no effect due to delay time). The 30-in shots at 1, 2, 4, and 6 ms/ft could also be combined. Figure 10 is a plot of the 50- and 80-pct-passing value determined from the Gaussian distributions. A horizontal line is drawn for delays that could be pooled into one regression line.

Although not shown in the figure, the single-hole fragmentation distribution pooled with the 30-in spacing curves at delays of 24 and 36 ms/ft. Apparently, firing holes at delays of >24 ms/ft can be considered as firing single-hole shots. It is noteworthy that the distance from the corner to the first hole was 21 in (fig. 1), but the fragmentation data pooled with the 30-in spacing tests. The single-hole test reflected the 30-in results because at 21 in, the first hole's breakout angle, >135°, reduced the burden distance for subsequent holes,

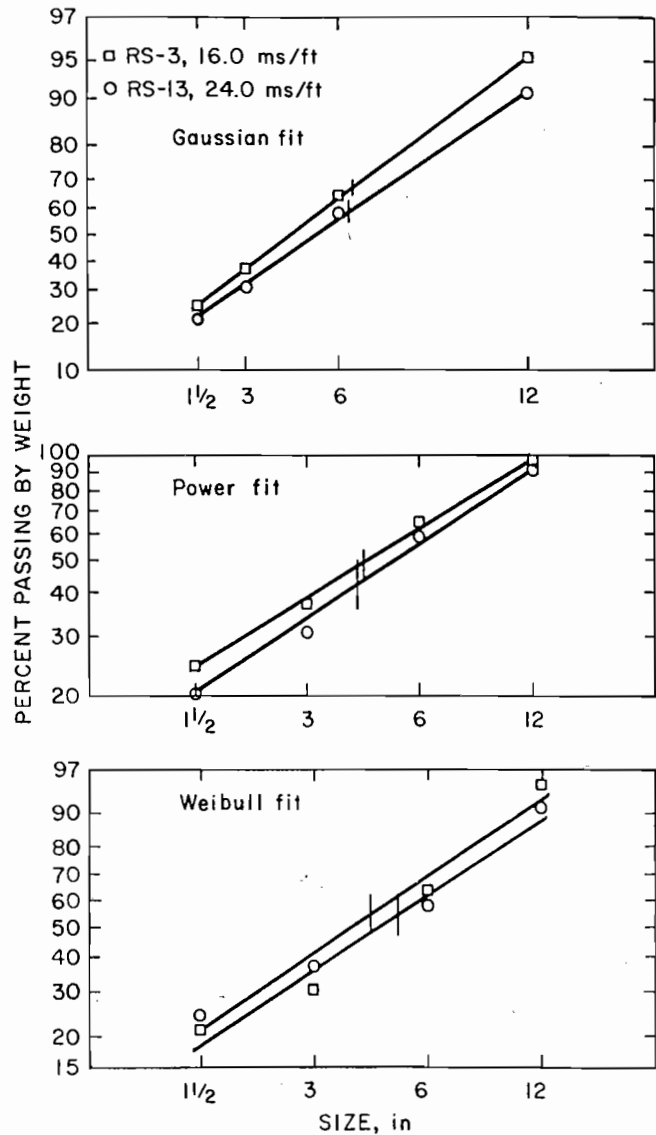


FIGURE 9.—Comparison of Gaussian, power, and Weibull distributions.

which improved fragmentation over that resulting from single-hole shot.

The 50- and 80-pct curves of figure 6 and 10 are quite similar except for the simultaneous shot at 21-in spacing, RS-9. The Gaussian distribution was not the best fit for this shot, and the regression line predicts a higher value than the data suggest.

As mentioned earlier, the University of Missouri's Experimental Mine has been used by several researchers (3-5) to conduct investigations of blast design and fragmentation. Where possible, these data have been compared with Bureau results, as shown in figure 11. Since the

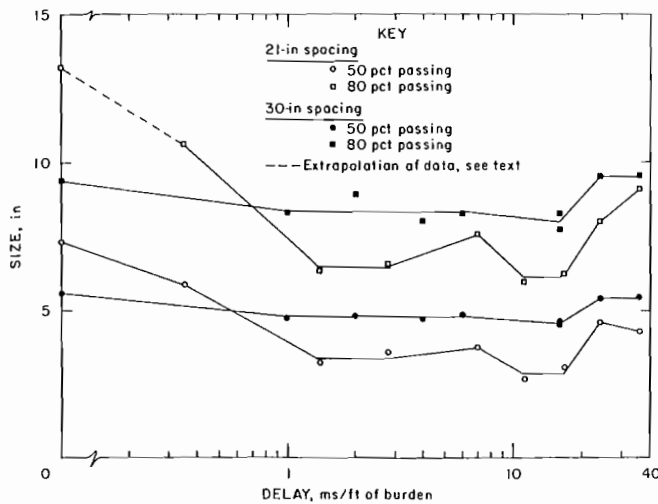


FIGURE 10.—Size at 50 and 80 pct passing versus delay period from Gaussian distribution fits to the 21- and 30-in spacing data.

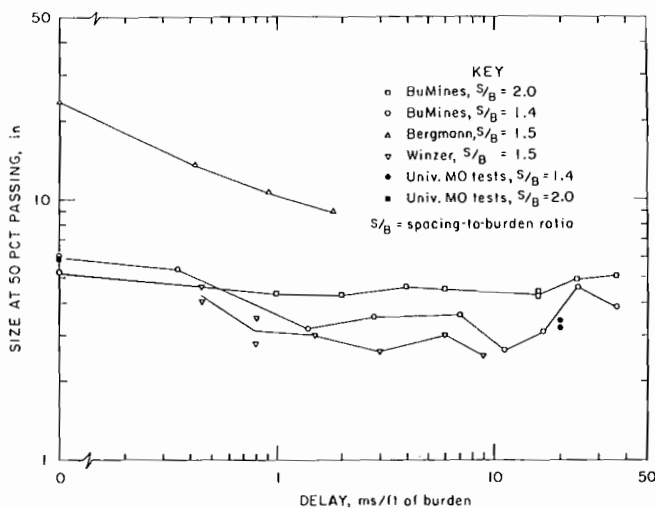


FIGURE 11.—Comparison of reduced-scale data with results obtained by other researchers.

shot designs for these tests were similar to those used by the Bureau, the fragmentation results compare very favorably. Other research, such as Bergmann's multi-hole tests in granite blocks (14), showed a similar significant improvement in fragmentation as delay times increased from simultaneous to 1 ms/ft of burden. Winzer's tests in limestone blocks and in a small bench (15) resulted in a relationship between delay time and fragmentation that correlates well with Bureau results. The character of the data in figure 11 is similar, showing substantial

improvement in fragmentation as delays increase to 1 ms/ft, slightly coarser fragmentation between 6 and 7 ms/ft, and continued improvement to 10 ms/ft.

Strain and pressure records obtained for the reduced-scale tests tend to confirm the fracture development mechanisms observed and reported by Holloway in work done under contract to the Bureau (S0245046, 1986). Initially, a fracture zone develops around the borehole because of the development of radial fractures and fracturing caused by reflected stress waves. The radial fractures propagate at speeds down to 12 pct of the P-wave velocity (16). The fractured region for these tests appeared to coincide with the finer material zone, which was generated within about 40 charge radii. From pressure gauges installed in this zone, the velocity of explosive gases penetrating fractures was found to be approximately 1,800 to 2,700 ft/s, as shown in figure 12. The large impulsive signals on the records were due to the pickup of electrical noise from the EBW initiation system and were used to confirm the delay intervals.

The P-wave velocities were determined from the arrival times and the distances from shotholes to gauges. The distance and arrival time measurements used to calculate the velocity of gas movement through the rock were adjusted to correct for the travel time of the bottom initiated explosive detonation (8,000 ft/s) to the height of the gauge.

The borehole pressure and radial crack pressurization produce stresses in the material beyond the near fracture zone, and this leads to additional failure and extension of the radial cracks to the free face. The velocity of gas penetrating fractures was estimated to be 146 ft/s for shot RS-14, as shown in figure 13. This velocity was determined by subtracting from the arrival time the travel time of the explosive to the gauge height and the time of gas penetration (1,800 ft/s) out to 10 in. The remaining distance to the gauge and the remaining time were used to determine the velocity.

Strain data from the shots were processed into resultant principal strains, octahedral shear strain and dilatation

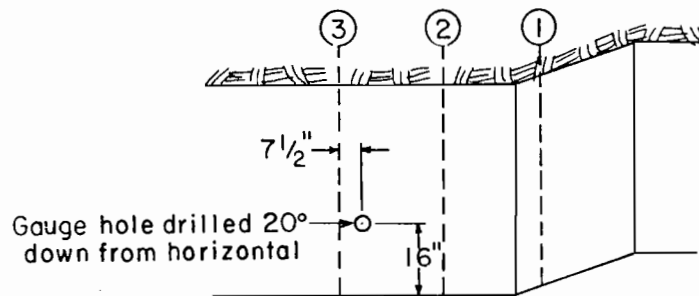
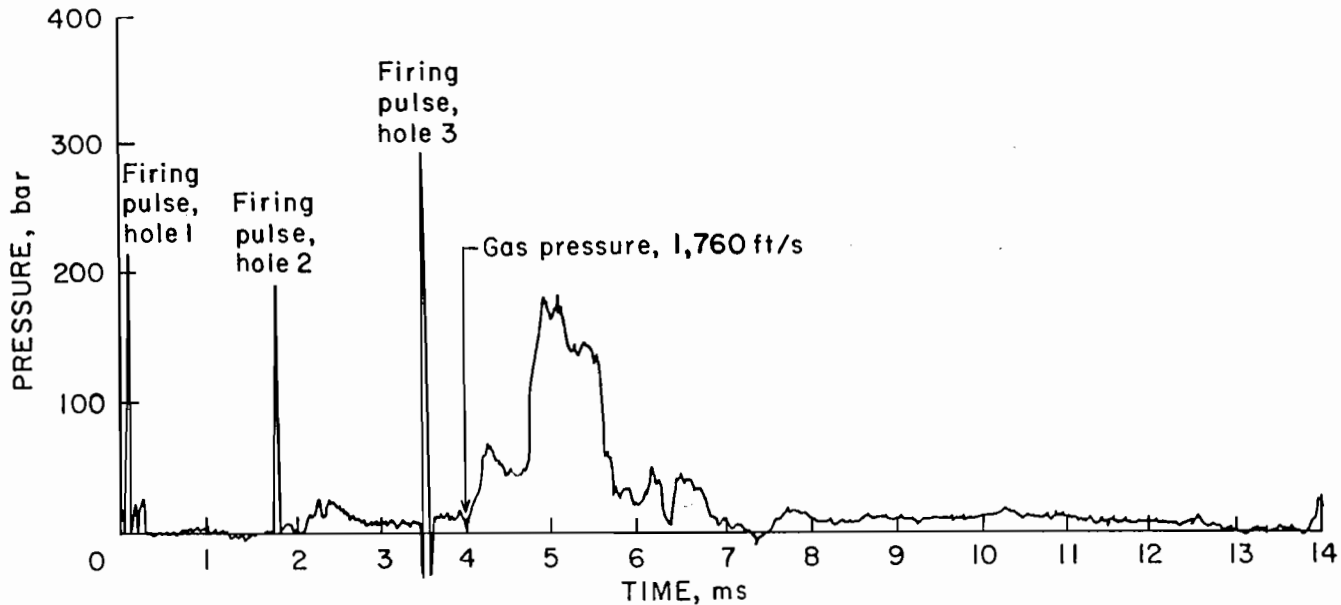


FIGURE 12.—Pressure measured for shot RS-19.

(7). Two of the principal strains are plotted in figures 14 through 17 for test shots with simultaneous and 1.4-, 2.0-, and 16.0-ms/ft delays. Shown on the records are the calculated P-wave and gas velocities determined from arrival times and shothole-to-gauge distances as discussed earlier. The strain pulses increased in amplitude with decreasing distances from shothole to gauge. These pulses correspond to the arrival of stress waves, which often arrived increasingly later in time as holes 2 and 3 were fired. The observed decrease in compressional (P) and shear (S) velocities is probably due to flaws in the rock, such as fractures and cracks produced by stress waves from an earlier shothole.

The gas velocities observed on the strain records agreed with pressure gauge gas velocity observations, except for those in shot RS-2, where it is believed

a major crack developed in line with holes 2 and 3, causing the observed gas velocity of $\sim 2,000$ ft/s. Excessive end-break was noted for this development shot.

Optimum fragmentation occurred when a hole fired such that its stress wave interacted with the stress induced by the expanding gas pressurization from the previous hole. Shots RS-2 and RS-6 show a long-term strain believed to be induced by the late-arriving gas pressure interacting with the stress wave from hole 3. Similar measurements and observations have been made at full scale (2). The interactions of strains induced by the stress waves and strains induced by gas pressure were not always observed for all shots of optimum delay (1 to 17 ms/ft), because the gauge is stressed only if near a pressurized crack and well-coupled to the rock. These interactions were also not observed for shots with delay

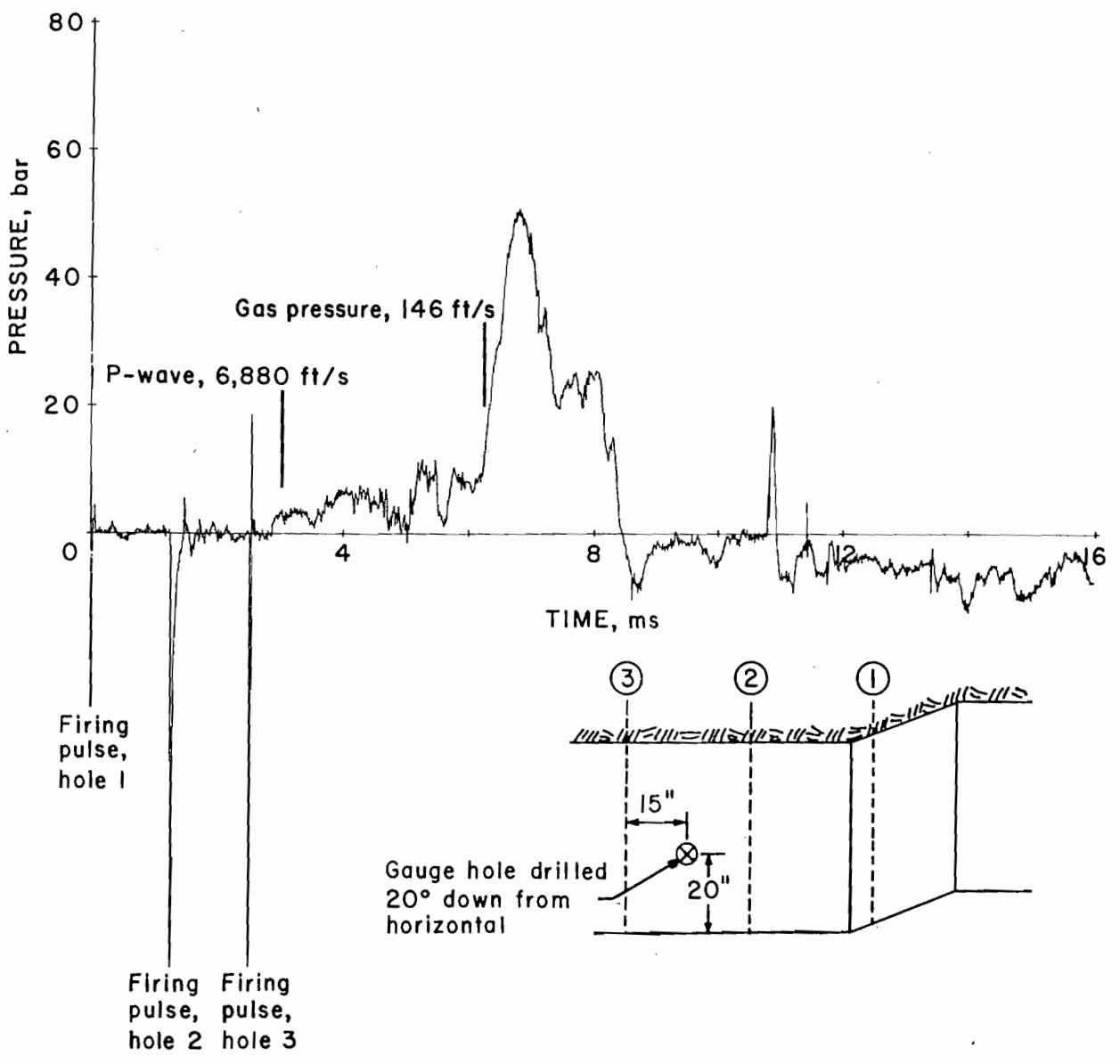


FIGURE 13.—Pressure measured for shot RS-14.

times outside the optimum range. Even though shot RS-19, shown in figure 15, does not show an interaction at the gauge location, pressure effects are still observed later in the record. Gas effects are not as apparent for the simultaneous shot, shown in figure 14.

Fragmentation results at short delays (less than 1 ms/ft of burden) suggest that fracturing in the zone around the borehole must be completed before the next hole fires. Using 1,800 ft/s as the

velocity for crack development in the 10-in zone around the borehole, plus the explosive detonation time (0.42 ms), the next hole should not be fired until after 0.9 ms or 0.7 ms/ft. Enhanced cracking appears to last as long as the gas is retained. Gas velocities through the rock suggest the process will last up to 20 ms or 16 ms/ft, based on a 45° breakout angle and gas penetration velocities of 1,800 ft/s for the first 10 in and 5 ft/s for the next 11 in.

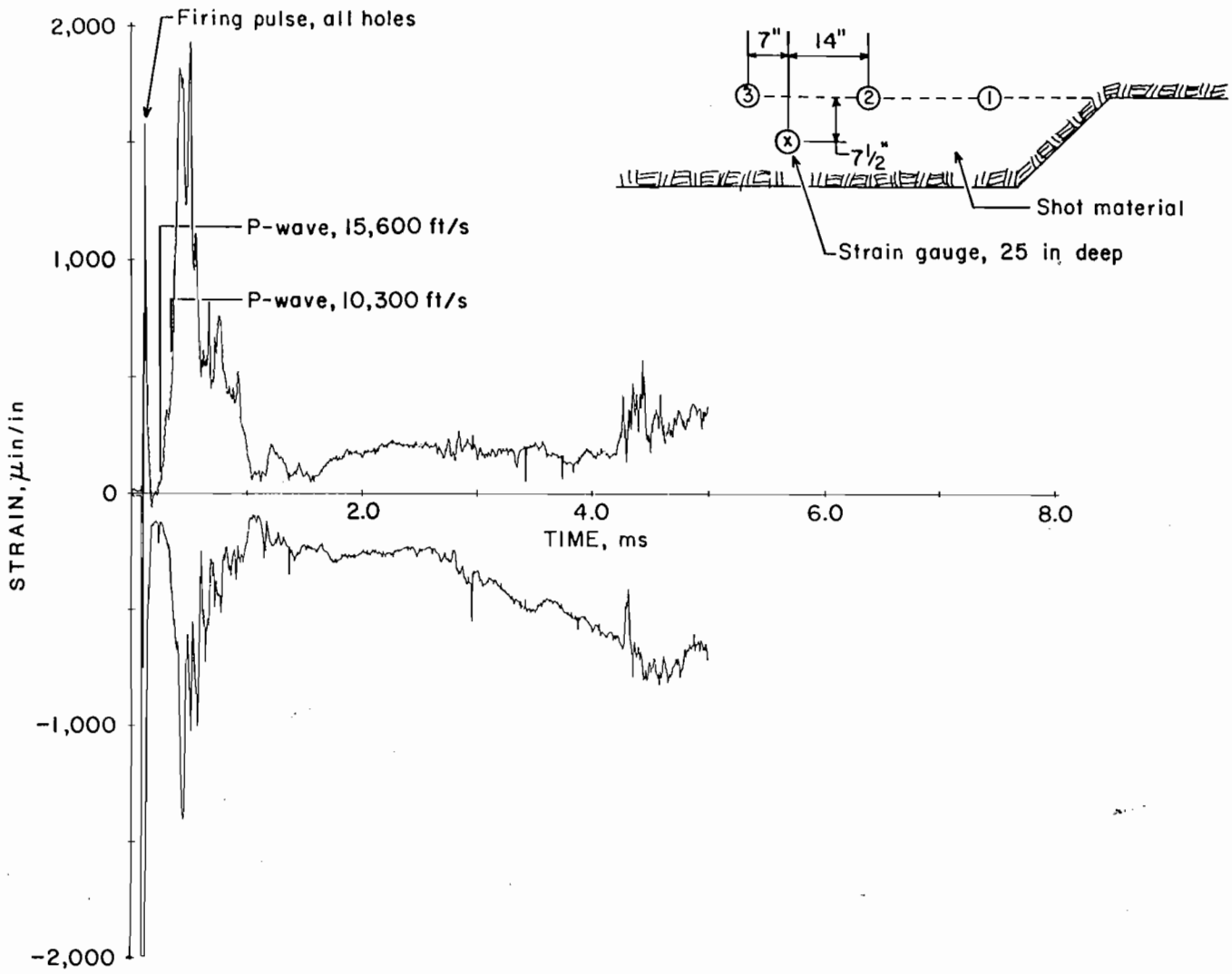


FIGURE 14.—Principal strains measured for shot RS-9. Simultaneous shot.

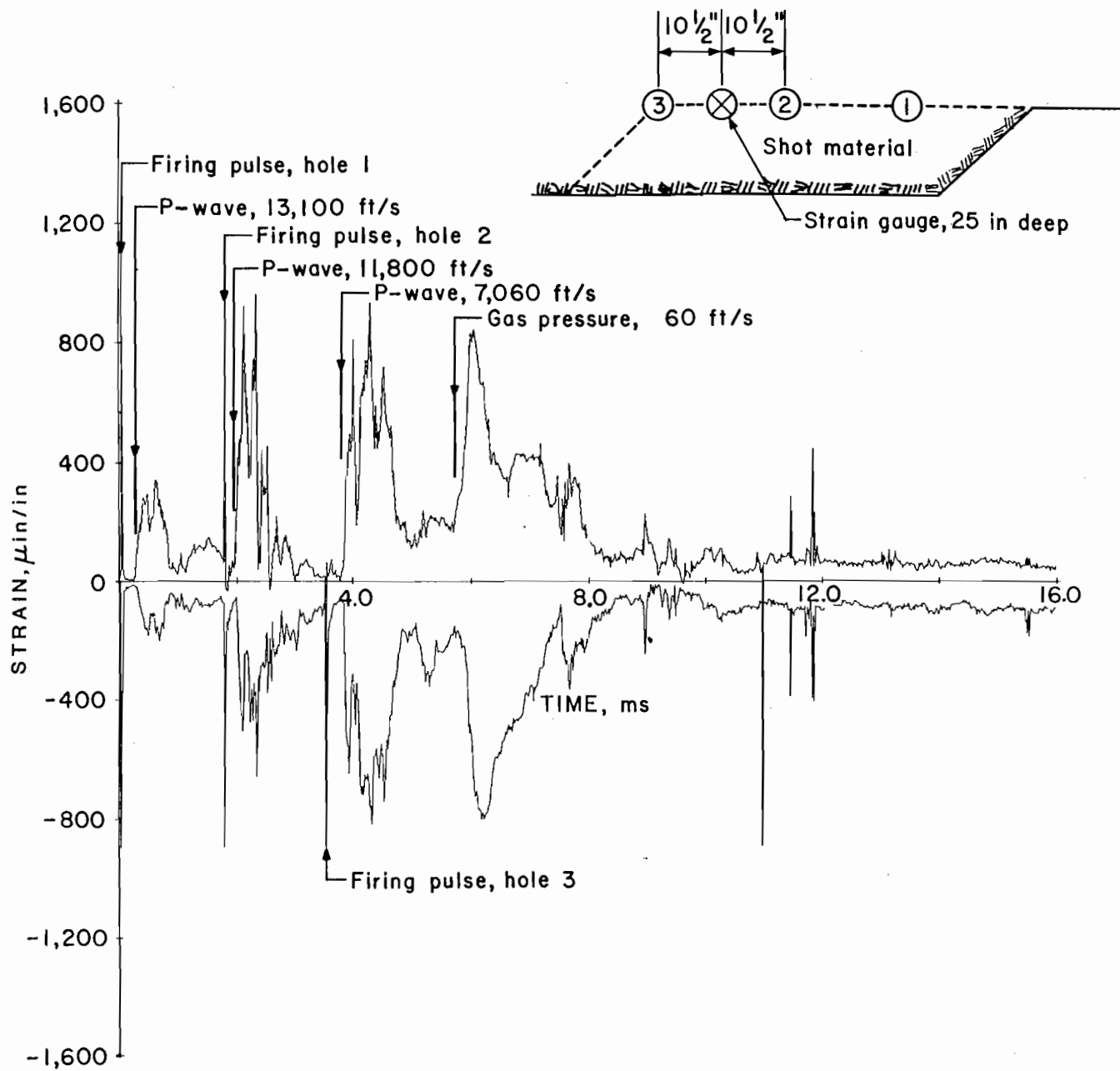


FIGURE 15.—Principal strains measured for shot RS-19. Delay time, 1.4 ms/ft.

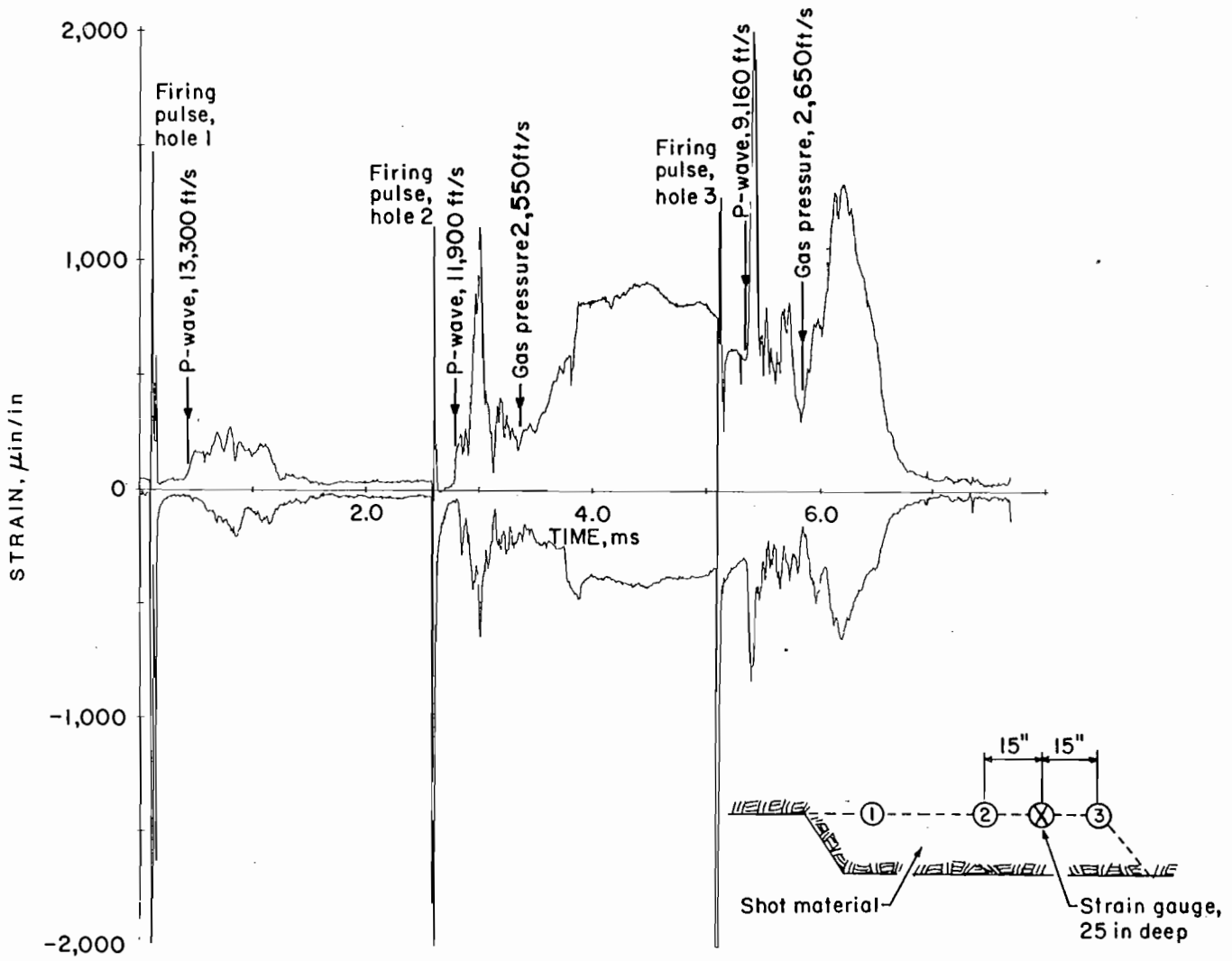


FIGURE 16.—Principal strains measured for shot RS-2. Delay time, 2.0 ms/ft.

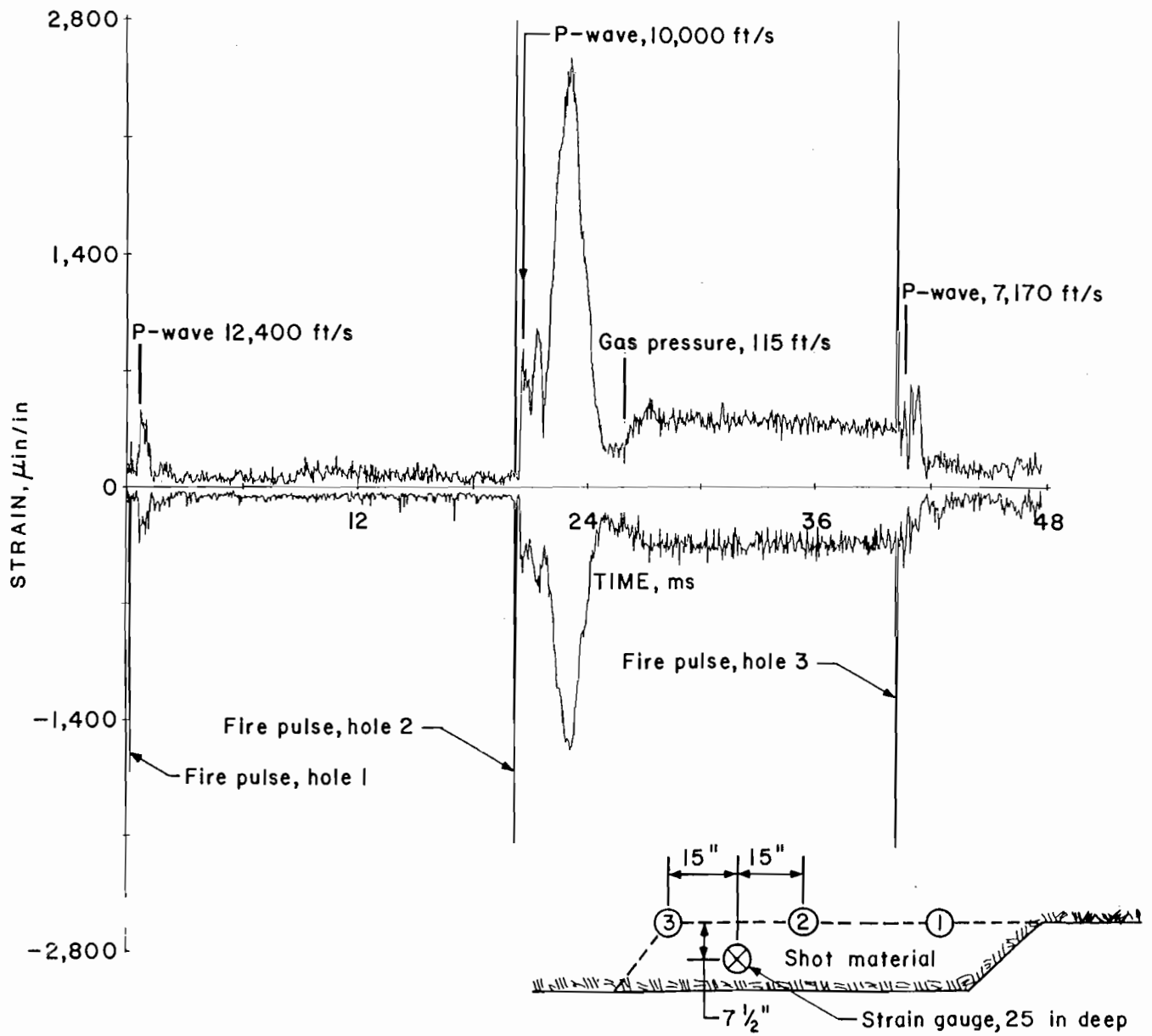


FIGURE 17.—Principal strains measured for shot RS-6. Delay time, 16.0 ms/ft.

CONCLUSIONS

An investigation of the effect of delay time on fragmentation was conducted at a reduced scale using three blastholes per shot in a 45-in bench. With the burden constant at 15 in, delay intervals were varied from 0.0 (simultaneous) to 36.0 ms/ft of burden, and the entire muckpile was screened to assess fragmentation for tests with 21- and 30-in blasthole spacings.

Attempts were made to mathematically describe the distribution of fragment sizes using power and Weibull functions, which have been used by other researchers. However, regression analysis indicated the Bureau data were best described by a Gaussian or simple normal distribution. Materials excluded from the analysis were overbreak, which came from outside the shot area, and fines, which were

determined to come from the immediate blasthole vicinity and showed little variability from shot to shot.

Analysis-of-variance tests showed that delay time did influence the distribution of fragment sizes for both spacings, but more so for the 21-in spacing. Fragmentation was coarsest for shots fired simultaneously and at delay times of 24 ms/ft of burden and greater. Better fragmentation was observed for delay times from 1 to 17 ms/ft, with the tests at 11.2 and 16.8 ms/ft resulting in the best fragmentation. Dynamic strain and pressure measurements indicated that this improved fragmentation may be the result of strains induced by stress waves constructively interacting with strains induced by gas pressure from an earlier detonated hole.

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BLASTING EFFECTS ON APPALACHIAN WATER WELLS

By David E. Siskind¹ and John W. Kopp²

ABSTRACT

The Bureau of Mines, in a contract study, examined blasting vibration impacts on low-yield domestic water wells in the Appalachian coal mining region. Researchers surveyed 36 case histories to determine if blasting was likely to have caused the claimed or observed changes, ranging from turbidity to loss of water. Following these investigations, they conducted field studies at four sites where the impacts of surface mine blasting could be directly measured on operating wells of known capacities.

Researchers found no evidence of blasting effects at the 36 well sites; instead, they observed other more likely

causes. In the field tests, researchers found no significant direct effects from the blasting. However, in three of the four cases, they did observe changes in the static water levels and specific well capacities as the excavations approached to within 300 ft. Researchers attributed these changes to mass rock movement resulting from downslope lateral stress relief in the low-yield fracture system aquifers. With sufficient recharge, static levels recovered and capacities increased, provided that the nearby mine excavations did not drain the aquifers.

INTRODUCTION

At about the time the Bureau of Mines was studying the problems of dynamic vibration response and safe levels for houses near surface mine blasting, allegations were being made that residential water wells were also being damaged by blasting. Technical experts believed that such effects were unlikely. However, there had never been a carefully designed and controlled study of this problem. Such a study appeared justified by the number of alleged cases, particularly in the Appalachia coal mining region.

The Bureau contracted with Philip R. Berger and Associates, Inc., to examine the problem of possible vibration damage to residential water wells from nearby surface mine blasting. The Berger team,

headed by Donelson Robertson, reported their research in a series of three contract final reports available for inspection at Bureau centers and for purchase through the National Technical Information Service (1-2).³ This paper summarizes the key findings, which were published in November 1980 as volume 1 of the contract final report (1).

The study consisted of three parts: (1) a background review of vibration and other impacts on water wells, such as earthquakes, earth tides, and nuclear blasts, (2) examination of 36 cases of alleged damage from blasting in Appalachia, and (3) a careful study of blasting effects on wells at four surface mine sites in Appalachia.

GENERAL REVIEW OF VIBRATION EFFECTS ON WELLS

The background review found little that was directly applicable. Observed cases of well damage were caused by permanent

ground displacement, such as land sliding, rather than vibration. The types of effects observed required vibration

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³Underlined numbers in parentheses refer to items in the list of references at the end of this paper.

levels many orders of magnitude higher than typical blasting vibrations and were listed as "casing collapse, earth displacement, pump base displacement, misalignment of pump column," etc. Cases

specifically involving mining were concerned with pit excavations and included interception with the aquifer, pumping from bit bottoms, and ground water pollution.

INVESTIGATION OF REPORTED CASES OF ALLEGED BLASTING DAMAGE TO WELLS IN APPALACHIA

Inquiries to Appalachian surface mines, regulatory agencies, explosives suppliers, coal companies, insurance companies, and trade associations identified 36 reports of blast damage to wells. Of these, 24 sites were visited for either direct well measurements or discussions with owners and/or mine operators.

In the Berger report, Robertson states:

In many cases, it was apparent that the damage claimed was caused by something other than blasting. In other cases, it was clear that there had been a general lowering of the water table, possibly as the result of unplugged flowing test holes, drainage at the high wall, or a two-to-threefold increase in the number of residences utilizing a limited supply, combined with seasonal changes.

In nearly every case, there was a lack of good benchmark data. Many residents have only a vague idea of the depth of their wells. Fewer know the depth of the casing. None of the residents interviewed knew the source of the water in their wells. About 50 pct had a vague idea of the static water level in the well when it was initially completed. Only one well had been

tested in any quantitative way. That test was inadequate and made the owner think he had a much better well than was actually the case.

Consequently, it was very difficult to confirm or deny that blast damage had occurred, but among the 36 examples, some of the well histories suggested two scenarios in which blasting might cause damage. The first is that the ground vibrations might be sufficient at times to cause loose material such as drill cuttings to slough off the uncased borehole and cause the water to become temporarily turbid, or if enough material was involved, to bury pump components at the bottom of the well. The second concerns those wells that obtain their water from flooded and abandoned deep mine workings. Ground vibrations might be sufficient at times to cause roof falls that could stir up sediment in the water or disturb an existing potable water-mine acid stratification. Of course, sloughing of the well bore and mine roof falls can occur in the absence of blasting, so these scenarios are not exclusive.

APPALACHIAN WATER WELLS

GROUND WATER OCCURRENCE IN APPALACHIA

Most ground water used for domestic supplies in Appalachian coal-bearing strata are in vertical fractures, joints, and along bedding planes. Some of these joints are tectonic in origin and have a regional pattern. However, local fracture systems exist from lateral stress

relief associated with natural topographic development. The coal seams often serve as the primary water conduit, being low in tensile strength and having extensive vertical fractures. Often, the coal is underlain by relatively fracture-free claylike rock preventing further vertical migration. In their study, Berger engineers found that local systems

TABLE 1. - Appalachian well water characteristics

Iron.....	Commonly exceeded recommended 0.3 mg/L.
Manganese.....	Often exceeded standard of 0.05 mg/L.
Sulfates.....	14 to 240 mg/L, below recommended level of 250.
Total solids:	
Suspended.....	Within acceptable ranges.
Dissolved.....	Do.
pH levels.....	Most were within 6 to 8, with the total range of 5 to 8.7.
Color.....	Within acceptable ranges.
Odor.....	Do.
Turbidity.....	Commonly exceeded standard limit of 5 units.

did not always interact, with static water levels sometimes varying between wells only 10 to 35 ft apart.

WATER CHARACTER AND QUALITY

Information on water quality was obtained from the literature and tests made on wells for this project. The general results are summarized in table 1. The only observed problem was an occasional instance of rusty or reddish-colored water seemingly unrelated to the specific iron content. In other studies, this coloration (red slime) was found to correspond to the presence of iron bacteria and was a problem when wells went unpumped for a long time. The sulfate levels suggest no serious acid mine drainage nor influence from acid rain, which is typically of pH 4.5 to 5 in the region.

TYPICAL DOMESTIC WELLS IN APPALACHIA

Residential wells in Appalachia are typically 6-in-diameter rotary or cable-tool drilled and 100 to 150 ft deep (maximum about 400 ft). Normally, only the top 20 ft or so is cased. The remainder is unprotected from sloughing off the well sidewalls, which is a common and normal occurrence. (Recently, it has

become popular to line all wells deeper than 100 ft to control sloughing problems.) Important considerations that together determine the continuous capacity of the well are the pump depth below the static water table, storage capacity of the well, natural recharge rate, and pump size.

The response to pumping these low-yield fracture water-table systems is a rapid drawdown until a near-equilibrium situation is reached. The pump must be sufficiently below the static water table (submergence) to allow this drawdown and still retain water above it. Larger pumps produce greater drawdown and require correspondingly deeper submergence, unless the flow is restricted by a valve arrangement. Additionally, rapid drawdowns from high pumping rates can cause abrupt pressure changes, turbid water, and possible "sanding up" of the pump.

As an example, Robertson calculates that an increase of pump submergence from 50 to 100 ft allows a constant pumping rate from a well with only one-fourth the specific capacity. He states that a pumping rate of 5 gal/min could be maintained by the typical Appalachia fracture-system well (specific capacity of 0.093 (gal/min)/ft of drawdown) provided it has 100 ft of pump submergence.

EXPERIMENTAL STUDY OF VIBRATION IMPACTS ON WATER WELLS

EXPERIMENTAL DESIGN

Researchers conducted an evaluation of four producing wells to specifically determine the influence of blasting on well productive capacity. At four sites,

shallow wells were drilled to obtain water from the coal measures. Additionally, deep wells drew water from the layers below the coal and were isolated from the upper measures. For each of these test wells, two or three observation wells

were used to monitor drawdown at varying distances.

Dynamic effects were measured by 10-h drawdown tests before, during, and following blasting. The static water level was monitored with float gauges. Water quality was also sampled periodically. To monitor blasting vibrations, researchers placed seismic instrumentation on the surface and at the bottom of an observation well for the 1-yr test duration.

TEST SITES

Four sites were found to be suitable for the tests, allowing sufficient monitoring time and standoff distance, and within convenient travel distance from Pittsburgh, PA, where Philip R. Berger and Associates is headquartered (fig. 1). A summary of the four test sites and

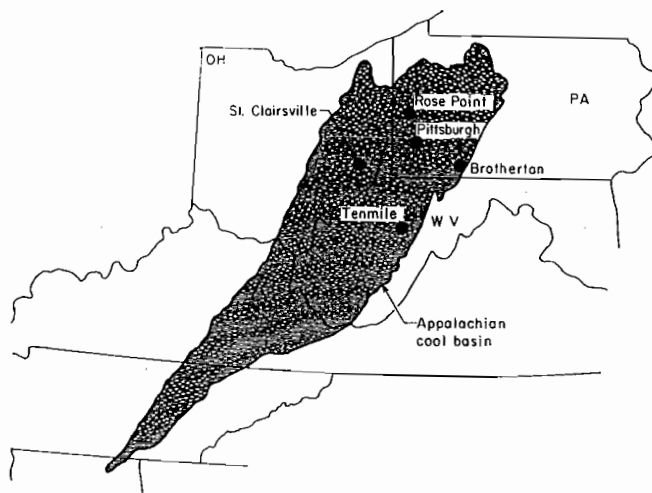


FIGURE 1.—Location of test sites.

experimental parameters is given in table 2.

RESULTS

Essentially, the same study was done at the four sites in Appalachia. A capsule summary is given in table 3.

BROTHERTON SITE

Researchers observed a correlation between the mining and well changes. The closest blast removed supporting material (e.g., toe), allowing lateral stress relief and the widening of vertical fractures. In other words, the pit excavation and not the vibrations influenced the fracture system. Changes were observed in the shallow well but not in the deep well, likely because of the more extensive vertical fractures near the surface. Significantly, the drops in static water level corresponded to increased specific capacity. The wider cracks allowed a temporary drop in static level, which would recover with sufficient rainfall. However, the cracks also provided increased flow (specific capacity).

Turbidity results were harder to interpret. Irregular fluctuations occurred during the 11-month study, and some temporary increases could have been from blasting. The nature of the tests contributed to the turbidity problem: infrequent and periodic pumping, suspended

iron and drill cuttings, and disturbance from the manual sample-collecting procedure. A continuously used well would not have some of these problems.

TENMILE SITE

Results were similar to those at Brotherton, except for smaller changes in static water levels. Specific capacities increased during the 12-month test for both shallow and deep wells. They went from initial values of about 0.065 and 0.020 (gal/min)/ft to 0.66 and 0.05, respectively. As at Brotherton, this was attributed to removal of downslope rock, resulting lateral stress relief, and the consequent widening of vertical fractures. Along with the increased capacity was the availability of increased recharge from the coal seam into the tight sandstone formation, possibly accounting for the little observed change in static water levels. Figures 2 through 5 show the site plan, profile, well arrangement, and a drawdown test record for the date June 9, 1979, when an 0.80-in/s particle velocity was recorded.

Researchers concluded that ground vibrations produced no deleterious effects on either the deep or shallow well, with

TABLE 2. - Blasting and pumping tests at four Appalachian mines

Site	Shallow well		Deep well		Blast distances, ft	Scaled distance, ft/lb ^{1/2}	Resultant vibration velocity, in/s	Notes
	Well depth, ft	Pump rate, gal/min	Well depth, ft	Pump rate, gal/min				
Brotherton, PA.	109	2.5 -7.9	169	4.0 -4.6	165- 500	12.1- 98	0.04-2.20	18 drawdown tests. Well site was 40 ft above initial mining. Pit was pumped at 500,000 gal/d.
Tenmile, WV.	146	.21-4.1	187	.27-1.0	64- 580	2.4- 34	.11-5.44	23 drawdown tests. Well site was on steep slope above blasting.
Rose Point, PA.	None	None	158	1.8 -6.8	175- 775	12.8- 60	.64-2.14	13 drawdown tests. Well site was below mining.
St. Clairsville, OH.	69	.24- .64	163	.24- .64	425-1,000	32 -153	.25- .84	14 drawdown tests. Well site was on slope above mining. Study ended before blasts reached wells.

TABLE 3. - Summary of results at four test sites

Site	Static water level	Specific capacity, (gal/min)/ft	Water chemistry	Turbidity
Brotherton, PA.	Dropped 21 ft in 11 months from pit pumping. Slight drops for closest blasts (shallow well only). Recovery related to rainfall.	Initially 0.34, steady for 11 months and then started to increase (shallow well only).	No changes.....	Possible temporary increases from blasting.
Tenmile, WV.	Minor variation. Some drop for shallow well with recovery by recharge. Little change for deep well.	Improved as test progressed by factor of 10 for shallow well and 3 for deep well. Highest value was 0.66.	...do.....	Highly variable. Possible increases from blasting within 300 ft.
Rose Point, PA.	Very minor variation except for one temporary increase.	Initially 0.33. Improved late in test except for one temporary decrease. Then recovered to initial value.	Unexplained variations and discrepancy between field and laboratory tests.	Little variation.
St. Clairsville, OH.	Incomplete data from equipment failure.	Very low at 0.007 to 0.017 except for unexplained initial high values.	Little variation.	Insufficient data.

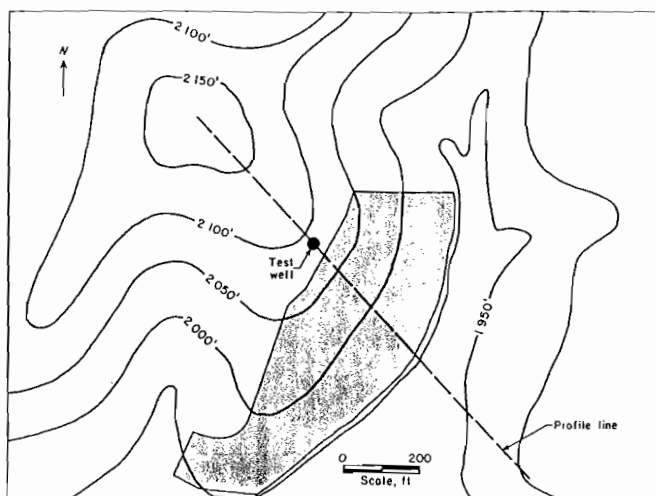


FIGURE 2.—Tenmile test site plan.

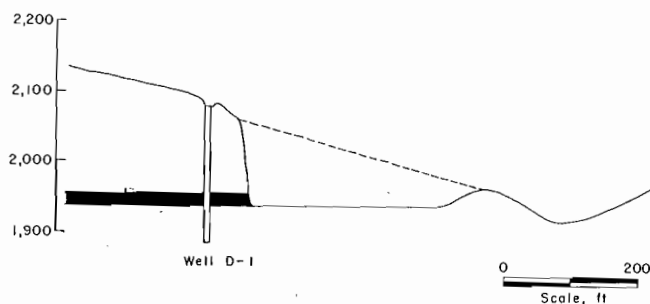


FIGURE 3.—Tenmile test site profile.

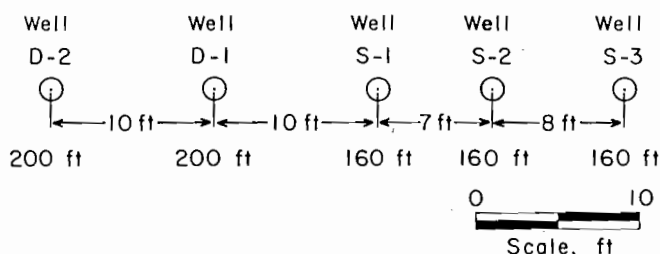


FIGURE 4.—Tenmile well layout. Deep and shallow pumped wells are D-1 and S-1. Others are observation wells.

one possible exception. A blast that went offscale at 2 in/s (distance of 85 ft) could have dislodged and caused a loose rock in the sidewall to shift into the hole, producing a partial bridge and preventing the sounding of the bottom.

Turbidity results were also similar to those from the Brotherton tests, with wide variations, difficult measurement conditions, and the suggestion of temporary increases from close-in blasts, less than 300 ft away.

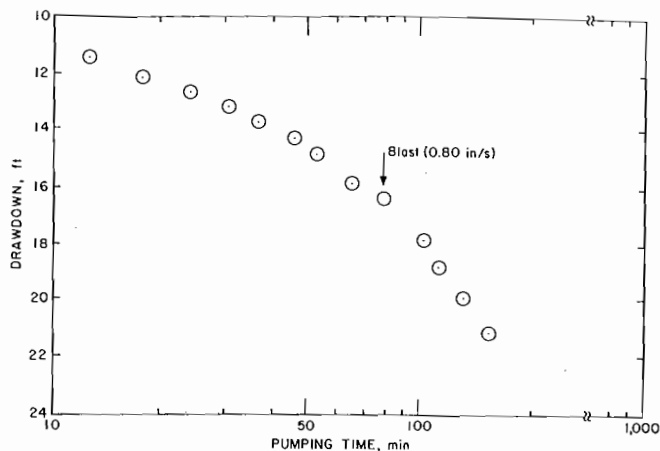


FIGURE 5.—Tenmile drawdown test for deep well (D-1).

ROSE POINT SITE

This site differed from the previous two in that the well was below the blasting. Consequently, researchers expected little stress relief effect. Indeed, little change occurred in either static water level or specific capacity until 5 months into the test, when specific capacity jumped from about 0.33 to nearly 0.60 (gal/min)/ft and later decreased. Researchers attributed the changes to nearby removal and replacement of overburden, which occurred at that time. As with previous sites, no direct effect of blasting was evident.

ST. CLAIRSVILLE SITE

This site was characterized by very low capacity wells, which were expected to be susceptible to outside disturbances. The initial drawdown test in the shallow well appeared satisfactory, with the well able to sustain pumping rates of 0.86 gal/min, equivalent to a specific capacity of 0.041 (gal/min)/ft.

The deep well was worse, and was only able to maintain 0.21 gal/min for about half the normal test period of 600 min. (For this site, as well as the others, an unperforated liner and packer were used to isolate the deeper well from recharge from the shallower coal measure being mined.)

One month after the initial drawdown test, the shallow well was again pumped,

but this time it had a specific capacity of only about one-third as much and at about one-half the pumping rate. Researchers suspected that air had been entrapped from initial tests, preventing full recharge.

In summary, researchers observed no clear blasting effects at this site. Unfortunately, work stopped short of distances found to produce effects observed at the three other sites. Some equipment failures were also experienced.

DOWN-THE-HOLE VIBRATIONS

Vibrations were monitored at the bottom of shallow observation wells as well as on the surface. Downhole vibrations were, as expected, of lower amplitudes, suggesting less risk to subsurface as opposed to surface structures. Table 4

summarizes the average relative resultant velocity amplitudes.

TABLE 4. - Reduced vibrations measured at depth

Site	Depth, ft	Relative amplitude ¹
Brotherton, PA....	149	² 0.34 ³ .68
Tenmile, WV.....	160	.44
Rose Point, PA....	168	.14
St. Clairsville, OH.....	180	~.25

¹Depth vibration divided by surface vibration.

²Blasting in poorly confined upper layers.

³Blasting in well-confined lower layers.

CONCLUSIONS

Research at four sites in Appalachia found no catastrophic effects on water wells from blasting at vibration levels up to about 2.0 in/s. At three of the sites (and possibly the fourth, had testing continued), long-term changes were observed and were attributed to the removal of confining rock.

As blasting and the pit excavation approached within 300 ft of the wells, the mechanism of lateral stress relief allowed vertical fractures to open. Because these fracture systems are typically the abode and conduit of shallow Appalachian ground water, the static water levels then dropped over a period of weeks. With sufficient rainfall, the water levels would return. Where sufficient submergence exists, such minor changes in static level would not be

noticed. Of benefit to the well user is the increased storage and flow as shown by higher observed specific capacities. Shallow wells exhibited this effect more than deep wells, consistent with expectations that more extensive fracture systems exist at shallow depths. At one site, backfilling reversed the improvement, either from clogging by fines or reintroduction of crack-closing lateral confinement.

Blasting may cause some temporary increases in turbidity, of the same order as those occurring in the absence of blasting. Results were uncertain on this because of the difficulty of sampling without causing disturbance and natural sloughing. Plastic well liners were recommended to control turbidity.

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FIBER OPTIC PROBE TO MEASURE DOWNHOLE DETONATION
VELOCITIES OF EXPLOSIVE COLUMNS

By David L. Schulz¹

ABSTRACT

Following ideas developed by researchers at the University of Maryland, the Bureau of Mines assembled a versatile, readily available, and very inexpensive

fiber optic probe for downhole measurement of explosive shock front position and velocity. The accuracy of the probe was determined in field testing.

INTRODUCTION

Detonation velocities of explosive columns are often measured to determine in situ explosive performance. Several methods are in use, ranging from simple resistance probes to very sophisticated and expensive electromagnetic resonance measuring systems.

In a recent study on blasting, under Bureau of Mines contract S0245046,

University of Maryland researchers used a fiber optic system to measure detonation velocity. Discussions with these researchers and further Bureau work led to the development of the fiber optic probe described in this paper.

SYSTEM DESCRIPTION

The fiber optic probe consists of a fiber optic cable to detect and carry the detonation-zone light emission and a sensor to detect and convert the light signal into an electrical signal. Figure 1 shows the Bureau's system.

data transmission applications, the modules cost about \$16 each and are reusable. Bright light through the system

FIBER OPTIC CABLE

The fiber cable is a DuPont Crofon² lightguide consisting of sixteen 0.010-in-diameter optic strands in a plastic tube with an overall outside diameter of 0.087 in. The lightguide used in the Bureau's tests was obtained from the Edmund Scientific Co. at a cost of about \$0.70/ft in September 1986.

LIGHT-DETECTING SENSOR

The light detector is a fiber optic data link receiver module-100 series with the trade name Optolink. Designed for

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²Reference to specific products and sources does not imply endorsement by the Bureau of Mines.

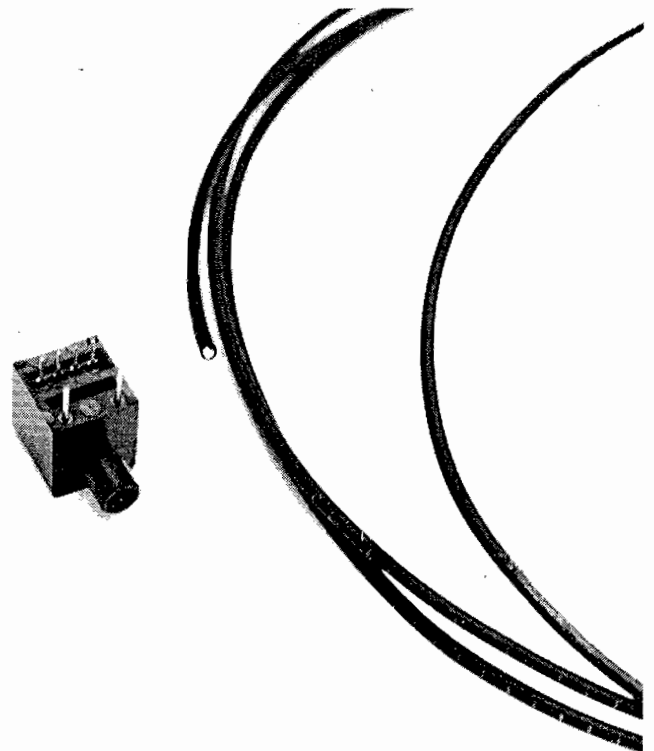


FIGURE 1.—Fiber optic detector and cable. Cable outside diameter is 0.087 in.

gives an output of about 100 to 300 mV. These particular modules by Interoptics were obtained through the Newark Electronics catalog.

The receiver module incorporates a photo diode to detect the signals, and a

receiver-integrated circuit chip to amplify the signals in one package. The overall dimensions are approximately 1/2 by 5/8 by 7/8 in.

APPLICATION NOTES

The fiber optic cables are cut to length with a sharp tool to get a good light-conducting surface, and one end is inserted into the light detector. The other end of the cable is embedded in the explosive column from the top down to the predetermined distance from the initiation point, usually the bottom of the explosive column. Intense light is given off as the detonation front moves up the explosive column. This light is picked up by the fiber optic as the detonation passes the ends of the fiber cables. Carried by the fiber optic cable to the

sensor, the light signals are converted to millivolt signals. Typical rise time is 0.01 ms. The millivolt signals are measured by an oscilloscope using a memory or hold feature to retain both the voltage spikes for each bundle of fibers and an initiation-time spike. From known insertion distances and measured times, velocities are simply calculated. A four-channel Nicolet digital oscilloscope, model 4094A, set to a time scale of 0.5 μ s per point, was used for the Bureau's tests.

FIELD TESTS

Bureau researchers used the fiber optic system for two series of field studies of explosively produced rock fragmentation. The first tests used explosive columns of 7/16-in-inside-diameter, 40-in-long plastic tubes filled with 60-pct-extra dynamite. The measured detonation velocity was 8,680 ft/s at the bottom of the column where the dynamite was densely packed. At the top where the powder was less well packed, the measured velocity was 7,350 ft/s (fig. 2). In 5/8-in-inside-diameter columns, measured velocities were 9,920 to 11,400 ft/s.

A second set of tests was run on larger charge diameters. Explosive columns were 16 ft long and consisted of 2-in sticks of 60-pct-extra dynamite. Holes for the fiber optic bundles were punched in the sticks at desired distances from the initiation point. The optic cables were then inserted and taped in place and the powder cartridges carefully loaded. Accuracy was determined by the dimensional stability of the column of powder cartridges (some compaction can occur). Researchers measured a steady-state velocity of 15,140 ft/s.

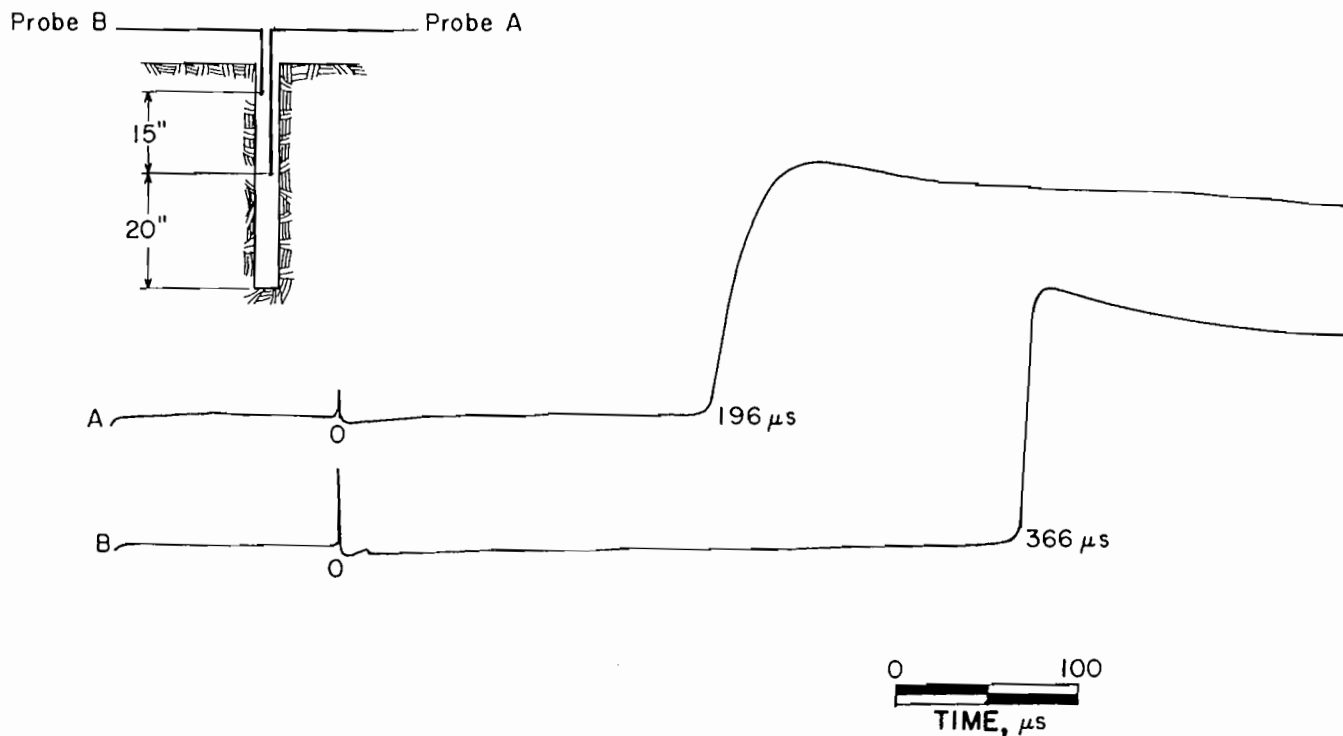


FIGURE 2.—Outputs from fiber optic probe for bottom-initiated blasthole showing measured velocity between probes of 7,353 ft/s.

CONCLUSIONS

Advances in fiber optic technology have provided an inexpensive and versatile method to measure explosive detonation velocity. An off-the-shelf system can be assembled and accurate measurements made

of detonation velocity for as little as \$10 per blast. Only a memory oscilloscope is needed, in addition to the fiber optic cable and detector module.

STEMMING EJECTION AND BURDEN MOVEMENTS
OF SMALL BOREHOLE BLASTS

by John W. Kopp¹

ABSTRACT

Stemming is used in blasting operations to help contain explosive gases as long as possible. Stemming can reduce air-blast, improve fragmentation, and reduce the chances of hot explosive gases igniting methane and dust explosions in underground mines.

The types and amounts of stemming material that are desirable in underground metal and nonmetal mine blasting to improve fragmentation while containing the hot gases are largely unknown. This Bureau of Mines research examined the effectiveness of differing lengths of stemming by measuring stemming ejection times as related to burden movement. With

properly stemmed blasts, stemming is contained until some burden movement has occurred. Test blasts at two surface limestone quarries were evaluated using high-speed photography. For the conditions of these tests, a stemming length of at least 26 charge diameters was found to prevent premature stemming ejection. In tests with stemming lengths of 16 charge diameters, the stemming was effective but there was early venting of hot gases through fractures in the rock. Further testing with other rock types, hole diameters, explosive types, and stemming materials to determine their effect on incendivity is recommended.

INTRODUCTION

Methane emissions in underground mines can present hazards, especially when ignition sources such as hot gases from explosives are present. The problems associated with blasting in underground coal mines have been addressed by use of permissible explosives and permissible procedures for their use. However, methane also occurs in some noncoal underground mines, particularly oil shale, trona, salt, potash, copper, limestone, and uranium mines. At present, such mines receive a variance from the U.S. Mine Safety and Health Administration (MSHA) blasting regulations, depending on the source of methane, associated ore body, and the method of mining. Conventional explosives and blasting agents, rather than permissible explosives, are normally used for both practical and economic reasons.

A recent Bureau of Mines contract² examined blasting practices in gassy

noncoal mines. Most of these operations use conventional explosives in standard underground blasting practices. Safety is sometimes ensured by evacuating all personnel to the surface during the blast. However, this is often not practical for large mines utilizing mining methods such as room and pillar. Some mines require 20 or more blasts per day, involving large amounts of explosives. In order to maintain production, blasts must be scheduled while personnel are working in the mine.

The contractor made a number of recommendations for blasting underground with personnel present in the mine. Important among these was use of stemming to contain the hot gases and flame from the explosion in the borehole until expansion of the burden sufficiently cooled the gases to prevent ignition of methane. The contractor made some predictions of stemming behavior, based on a simple mathematical model, and recommended that his calculations be confirmed by field studies.

This Bureau study was conducted as a followup to the previous study, to

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²Contract JO215031; Eauer, Calder & Workman, Inc.

measure the retention time of various lengths of stemming in the borehole during normal blasting and to relate stemming retention time to the burden movement caused by the expanding gases of the explosive. Tests were conducted at two surface limestone quarries, which allowed

careful control of the test blast design variables and adequate lighting for high-speed photography. While this study is directed at blasting underground, the efficient use of stemming will improve blasting in surface mines also.

EXPERIMENTAL DESIGN AND PROCEDURE

Early experimentation had shown that high-speed cinematography was the best method for measuring stemming movement. This method also allowed observation of the burden displacement and dust and smoke generated by the blast.

Field experiments were filmed with two cameras, a 16-mm rotating-prism camera capable of speeds up to 11,000 frames per second and a 16-mm registering-pin camera with filming speeds up to 500 frames per second. The registering-pin camera has better resolution than the rotating-prism camera and thus provides a much clearer picture.

The time of detonation of the explosive was recorded on film with Nonel³ shock tubing. A known length of shock tubing was attached to the explosive charge and passed through the stemming to the surface and coiled to allow the flash to be recorded on film. Detonation of the explosive initiated the tubing, which detonated at 6,000 ft/s. Thus, the time of detonation was determined by noting the flash of the coiled tubing on the film and calculating the time required for the detonation to reach the surface.

Full-scale field tests were performed at a surface limestone quarry in order to eliminate lighting problems when filming with the high-speed cameras. Twelve

cratering shots were detonated in a factorial experiment to test two types of stemming material at three lengths of stemming and with two explosive types.

A high-energy explosive and a relatively low energy explosive were used for this series of tests. The low-energy explosive was chosen to produce results similar to those from ammonium nitrate and fuel oil (ANFO). Both explosives were in 1-1/4-in-diameter cartridges. Enough explosive was used in each test to make a charge 16 in long. The volume of explosive was not varied for this test series. Properties of the explosives used are shown in table 1.

The blastholes had a 1-1/2-in diameter and ranged from 36 to 72 in deep. The holes were drilled vertically in a limestone quarry floor. This represents the worst case for blasting efficiency, allowing relief in only one direction, upward.

The stemming material consisted of crushed limestone in one of two sizes: The material was either drill cuttings screened to less than minus 10 Tyler series mesh size (0.0661 in) or limestone gravel between 3/8- and 3/16-in size. The stemming material was added above the explosive and filled the hole to the collar. Table 2 shows stemming length and type, and assignment of shot numbers. The stemming was lightly tamped and had a density of about 1.5 g/cm³.

³Reference to specific products does not imply endorsement by the Bureau of Mines.

TABLE 1. - Properties of explosives used in the test series

Explosive type	Density, g/cm ³	Detonation velocity, ft/s	Relative bulk strength ¹	Explosive temperature, K	Borehole pressure, atm
High energy	1.16	15,000	148	3,000	30,000
Low energy	1.07	10,500	115	2,870	19,000

¹ANFO = 100.

TABLE 2. - Experimental design and assignment of test numbers

Length of stemming.....in..	20	32	50-60
Fine drill cuttings:			
High-energy explosive.....	S-3	S-2	S-11
Low-energy explosive.....	S-6	S-12	S-1
Coarse crushed stone:			
High-energy explosive.....	S-4	S-8	S-10
Low-energy explosive.....	S-5	S-7	S-9

RESULTS

Twelve cratering tests were conducted according to the factorial design in table 2. All shots were monitored with the two high-speed cameras described previously, one running at 500 frames per second and the other at 1,000 to 3,000 frames per second. Analysis of the films showed that stemming was usually ejected from the shallow holes. With stemming lengths of 20 in, the stemming material was ejected as follows:

Shot	Ejection, ms
S-3.....	13
S-6.....	32
S-4.....	8.8
S-5.....	Retained

Longer stemming lengths resulted in retention of the stemming. When stemming is retained, it has done its job in terms of confining the hot explosive gases. However, when stemming is ejected, further analysis is required to determine if an adequate length of stemming was used.

Further analysis of the films indicated the motion of burden. Not only can velocities of various parts of the burden be calculated, but an estimate of the increased burden volume caused by the expanding gases can be made. Figure 1 shows stemming and burden movement for shot S-3, which was a test with 20 in of fine stemming, using the high-energy explosive, which resulted in a stemming ejection time of 13 ms. Initial movement of the stemming was obscured by dust caused by venting through cracks in the burden. The initial velocity of the stemming was calculated to be 190 ft/s.

The burden velocity was calculated to be 27 ft/s. The increase in burden volume was calculated by assuming the burden movement to be in the shape of a cone and measuring this increase on figure 1. It is apparent from the figure that the burden movement is closely approximated by a cone. The rate of volume increase was found to be 600,000 in³/s. A plot of stemming movement and burden expansion versus time elapse from initiation of detonation is shown in figure 2 for shot S-3. It is apparent from figure 2 that considerable burden movement occurred before the stemming was completely ejected after 13 ms. In this case, some cooling of the hot explosive gases had occurred because of volume expansion as the gases worked their way into the fractured burden region.

An estimate of the amount of cooling of the explosive gases can be obtained by assuming the thermodynamics of the expansion to be adiabatic. The expansion is rapid and allows little time for heat to be exchanged between the gases and the surrounding rock. From the first law of thermodynamics it can be shown that temperature and volume of the gas are related as follows:

$$T_1 \frac{1}{T-1} V_1 = T_2 \frac{1}{T-1} V_2, \tag{1}$$

which on rearranging becomes

$$T_2 = \left(\frac{V_1}{V_2}\right)^{T-1} T_1, \tag{2}$$

where T₁, T₂ are the initial and final temperatures, V₁, V₂ are the initial and

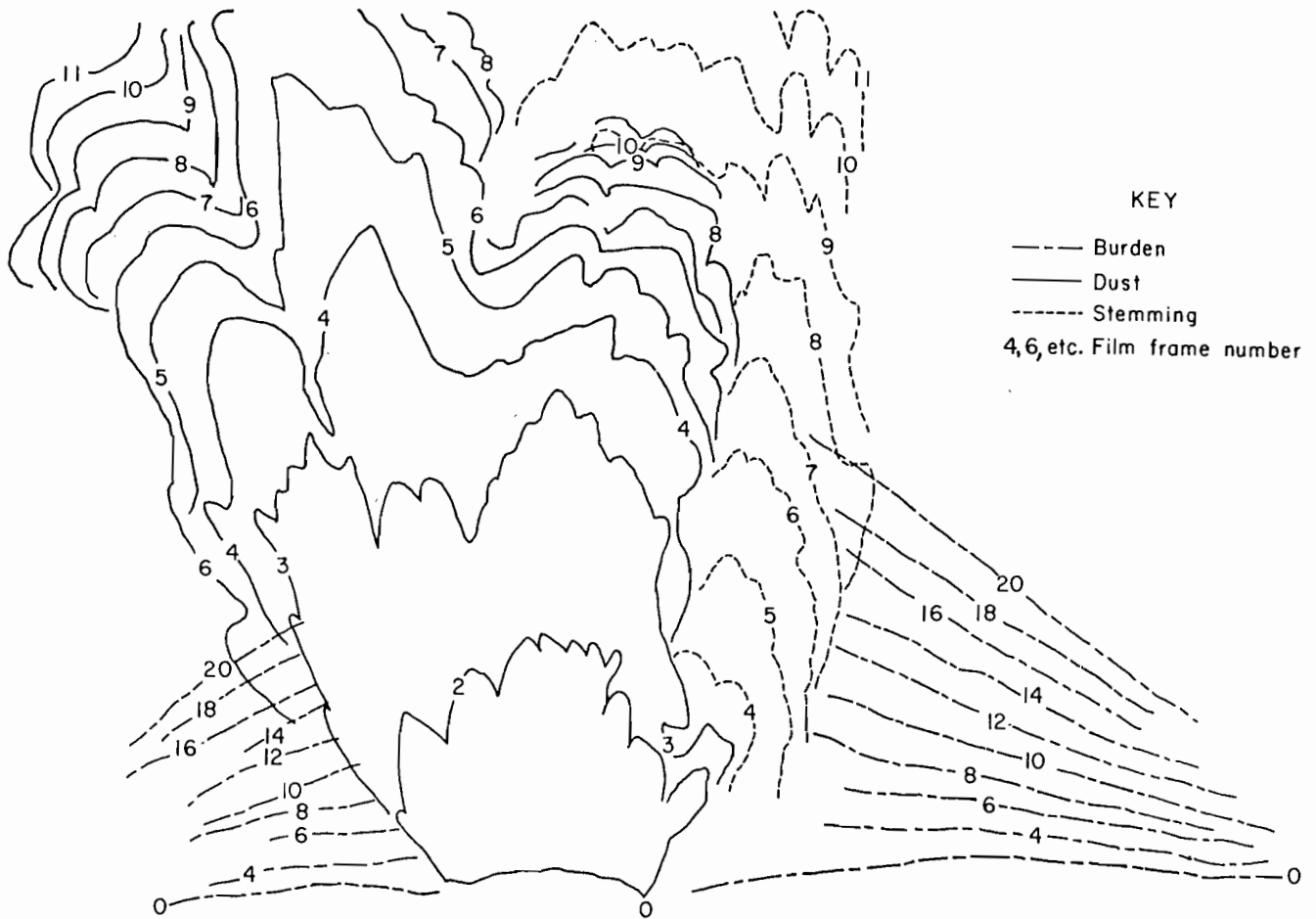


FIGURE 1.—Stemming and burden movements of shot S-3.

final volumes, and T is the ratio of the heat capacities of the expanding gases. The actual value of T depends on the molecular structure of the gases involved. Most of the gas products of the explosives used are diatomic and polyatomic gases: nitrogen, carbon dioxide, and water vapor. The average value of T for these gases is approximately 1.3.

The equation describing the gas temperature line in figure 2 becomes

$$T = \left(\frac{V_1}{V} \right)^{0.3} T_1, \quad (3)$$

where V_1 , T_1 are the initial volume and temperature of the explosive at detonation, V is the increase in volume, and T is the temperature at that volume.

It is now assumed that the gas expands into all of the new volume created by the expanding burden. Though the expanding gases may not actually fill all of the

new spaces created by fracturing of the burden, this is approximately balanced, as no allowance is made for expansion of the gases into existing voids or for porosity of the rock. The estimated temperature is thus an approximation but provides some insight into the phenomena involved.

Figure 2 also shows the predicted gas temperature decrease based on the use of equation 3 and the burden volume increase as determined from analysis of high-speed films. The stemming remained in the borehole for 13 ms, at which time the gas temperature is estimated to have cooled from the detonation temperature of 3,000 K (2,727° C) to 550 K (277° C), which would be sufficient to prevent ignition of a methane-air mixture, since the ignition temperature of methane is 905 K. However, from figure 1, it is evident that venting, probably through a major fracture, occurred before expulsion of

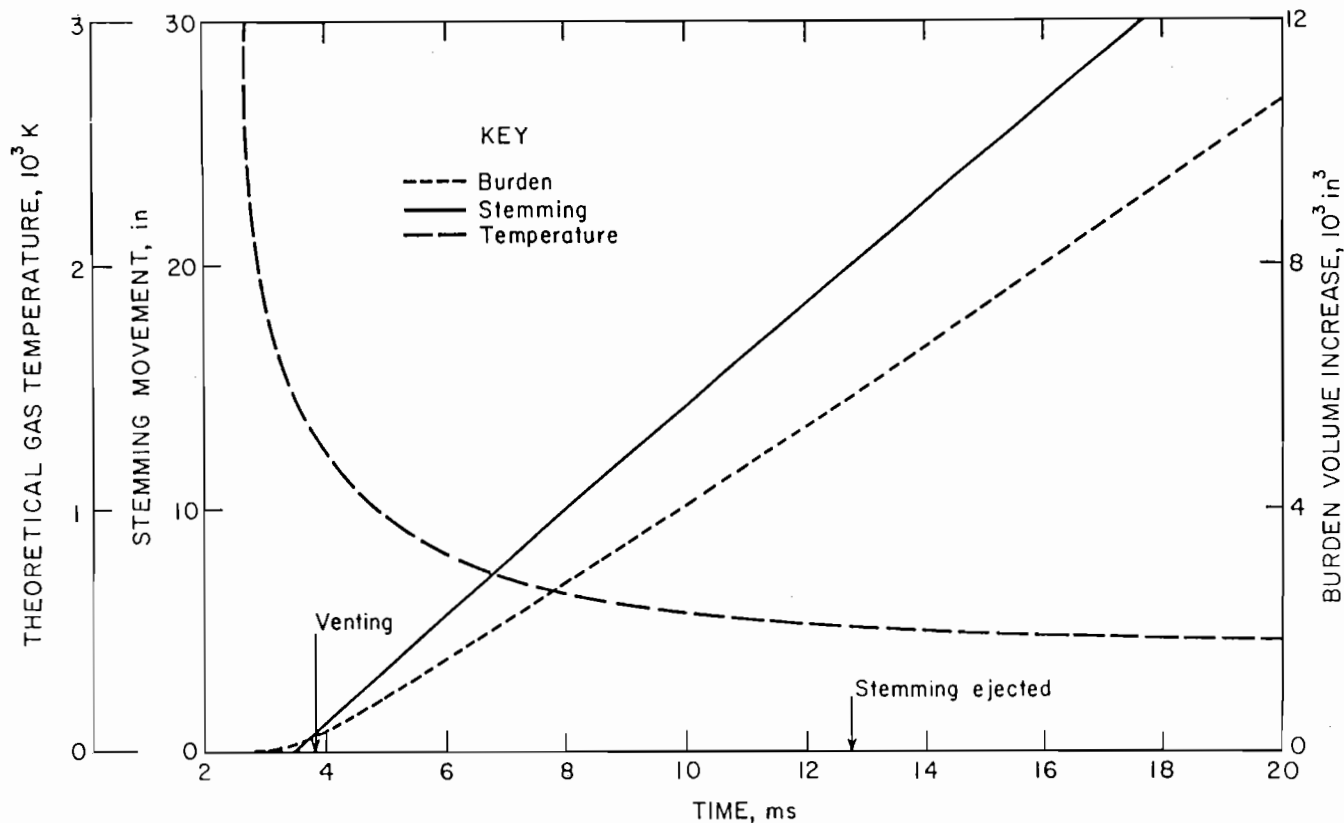


FIGURE 2.—Relative movements of stemming and burden for shot S-3 and the associated gas temperature.

the stemming, at 3.8 ms after initiation. Figure 2 shows that the explosive gases would have cooled only to 1,300 K after 3.8 ms, not sufficient to prevent methane ignition.

A similar analysis was performed for shots S-4 and S-6. Results are presented in figures 3 and 4. Figure 3 presents the analysis of shot S-4 where 20 in of coarse stemming was ejected using the high-energy explosive. At the time of stemming expulsion at 8.8 ms, the explosive gases were estimated to have cooled to 650 K, within a safe temperature range. However, venting of dust and smoke was observed at 3.5 ms after initiation. From figure 3, the estimated temperature of the gases would be 980 K, above the safe limit. Again, the venting probably occurred through existing fractures in the rock. This shot and the previously discussed shot used a high-energy explosive, but the previous shot used a different stemming size. The finer material held longer.

Shot S-6 had 20 in of fine stemming and the lower volume-energy explosive. Stemming was ejected from shot S-6, though at a slower rate than with the higher energy explosive (fig. 4). The time of ejection was 32 ms, and the estimated gas temperature at the time was 625 K. Venting of smoke or dust was also observed, starting at 4.8 ms after initiation. From figure 4, the estimated gas temperature at the beginning of venting was about 1,300 K, high enough for a methane ignition.

Four shots were fired with 32 in of stemming in each hole. The stemming remained intact for all these shots. The film analysis did not show any stemming movement. Burden movement was slower than in the previous shots using less stemming. With the exception of shot S-12, no venting of smoke or dust occurred. Also, a rubble zone of broken material was left at the surface of each hole. The zone was about 2 ft in diameter for three of these shots, smaller than the

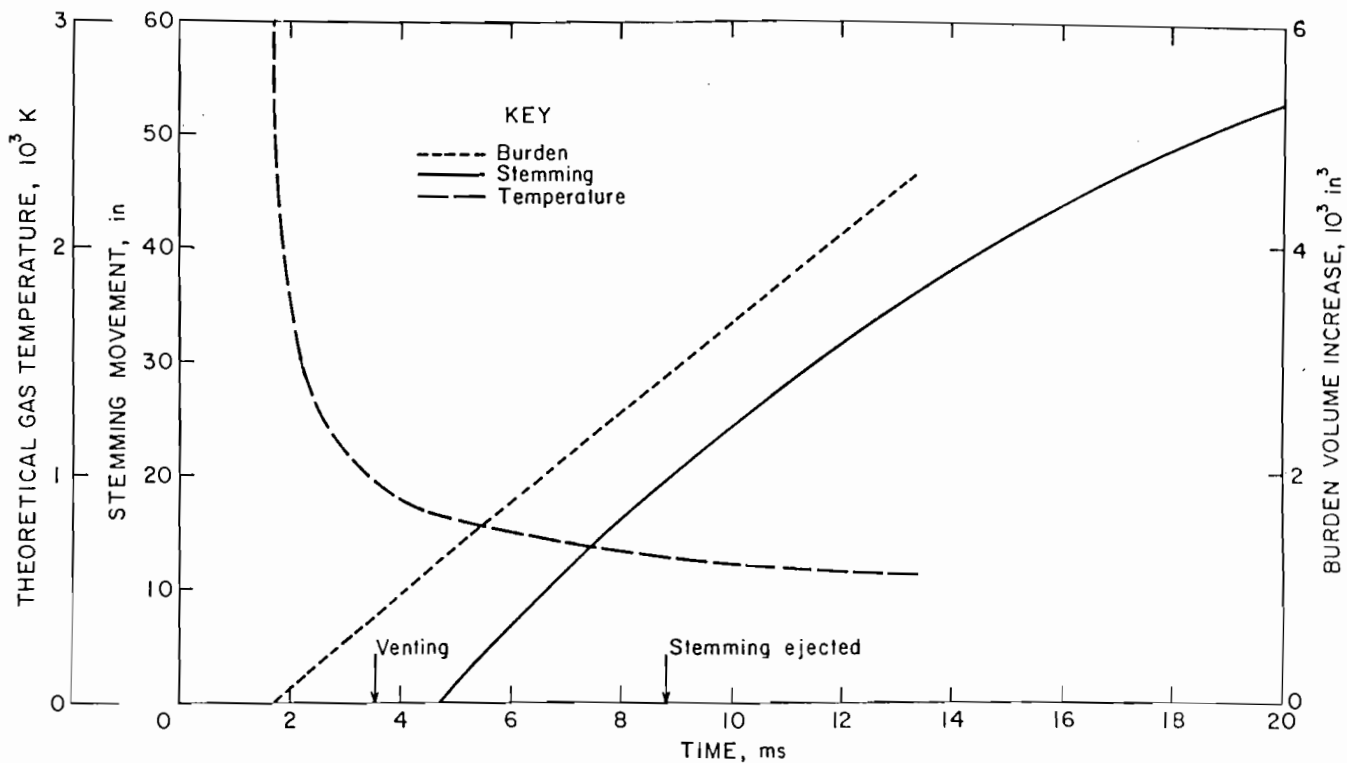


FIGURE 3.—Relative movements of stemming and burden for shot S-4 and the associated gas temperature.

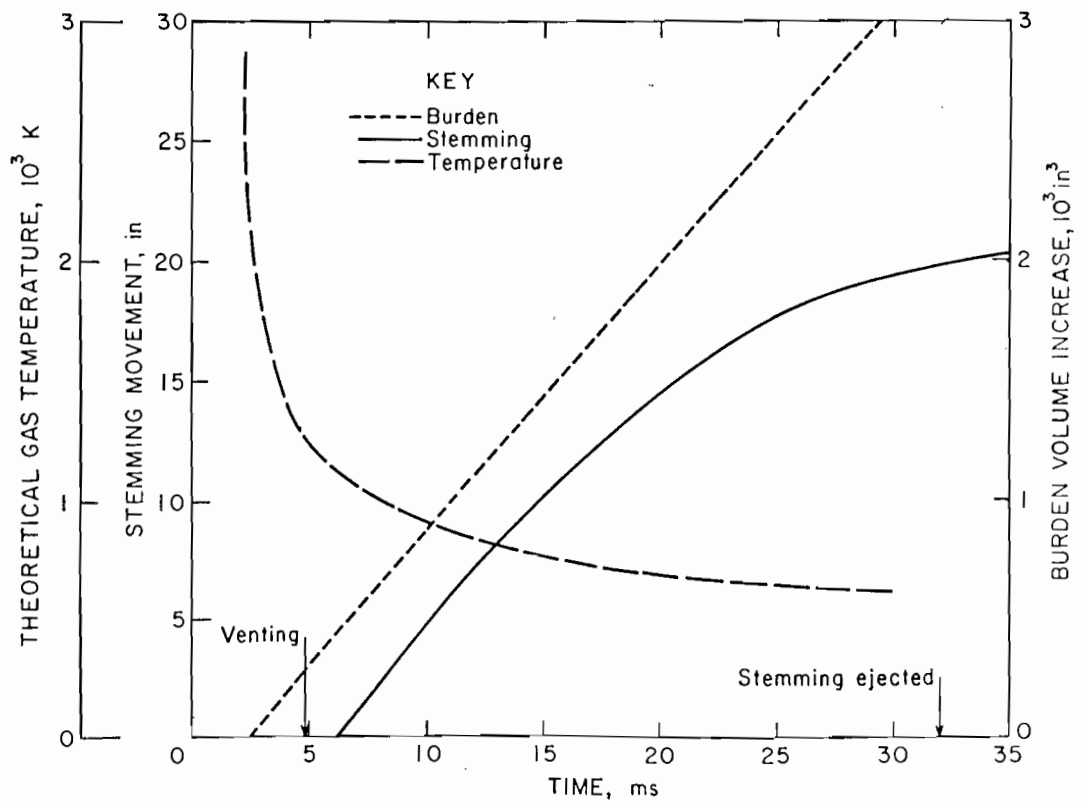


FIGURE 4.—Relative movements of stemming and burden for shot S-6 and the associated gas temperature.

2 to 5 ft for 20-in stemmed blasts. The presence of rubble zones at the surface for both the 20- and 32-in stemming cases indicates that with 32 in of stemming, the explosive is still sufficiently close to the surface to allow fragmentation of the burden.

The final four shots of this series used stemming lengths of 50 to 60 in. No stemming movement was detected for any of these shots. Burden movements were also smaller than for the previous shots. Figure 5 shows a comparison between burden velocities and the length of stemming used in each hole. The type of explosive used made a difference in burden velocity only at the two shortest stemming lengths. Differences caused by stemming type were inconclusive.

In this crater test series with 1-1/4-in charges in 1-1/2-in-diameter blast-holes, it was found that stemming was

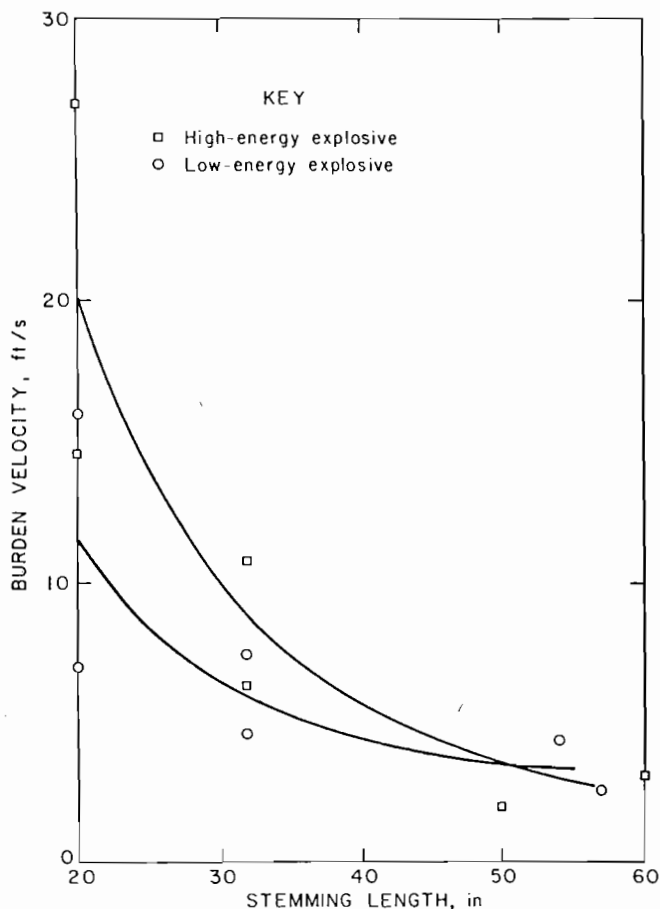


FIGURE 5.—Observed burden velocities versus length of stemming for the 1½-in-diameter shot series.

retained in all cases with stemming regions of 32 in or greater. When 20 in of stemming was used, there was stemming ejection in three of the tests after 8.8 ms or more, but venting of gases through fractures occurred before 8.8 ms. Estimates were made of explosive gas cooling associated with volume increases as the explosive gases expanded into the fractured burden. In the three tests where stemming ejection occurred, explosive gases were estimated to have cooled sufficiently to prevent ignition of methane in the time required for stemming ejection (8.8 ms) but not at the time when venting of gases through fractures occurred. There was no premature venting of gases with a 32-in stemming region. It is concluded then that for the conditions of these tests, a stemming length-to-charge-diameter ratio of 26 (32 in of stemming) was adequate to prevent ignition of methane. However, a stemming length of 16 charge diameters (20 in of stemming) could have resulted in methane ignitions because of early venting of hot gases through fractures in the limestone rock.

Bauer, Calder & Workman, Inc., suggested a simple physical model to predict the time required to eject stemming. This model depends only on inertia of the stemming material and not on frictional forces to resist movement, and thus the acceleration, a , of the stemming is given by

$$a = \frac{F}{M}, \quad (4)$$

where F is the force exerted on the stemming by explosive gases and M is the mass of the stemming.

The equation of motion is thus

$$S = V_0 t + 1/2 a t^2, \quad (5)$$

where S is distance traveled by the stemming, V_0 is initial velocity, and t = time in seconds.

Combining equations with $V_0 = 0$ gives

$$t = \sqrt{\frac{2SM}{F}}. \quad (6)$$

The force F can be estimated from the borehole pressure, P , times the cross-sectional area of the hole, A , or

$$F = PA, \quad (7)$$

and the mass of stemming equals

$$M = \rho_S A \ell, \quad (8)$$

where ρ_S is the density of stemming material, A is the cross-sectional area, and ℓ is the length of stemming. Substituting gives

$$t = \sqrt{\frac{2S \rho_S \ell}{P}}. \quad (9)$$

This prediction method yields ejection times for the shots in this investigation as shown in table 3. Also shown in table 3 is the time of first observed stemming movement and observed time to ejection.

All observed times for ejection were much longer than the calculated times. In many cases, no stemming movement was observed before the calculated time was past. This calculation method then should be used only to obtain an estimate of the minimum stemming ejection time. Improved estimates will require the addition of frictional forces in the calculations.

A clue to this late stemming movement is provided by the instrumentation used in a 6-in vertical cratering blast by Sandia National Laboratories⁴ (shot V-1). The blasthole was instrumented with the SLIFER system to monitor the detonation rate of the explosive. This instrument

⁴Shirey, D. L., J. E. Uhl, and R. L. Parrish. Atlas Cratering Tests. Ch. in Oil Shale Program Quarterly Reports, October 1984 through March 1985. Sandia Nat. Lab., Rep. SAND 85-2768, 1986, pp. 5-26.

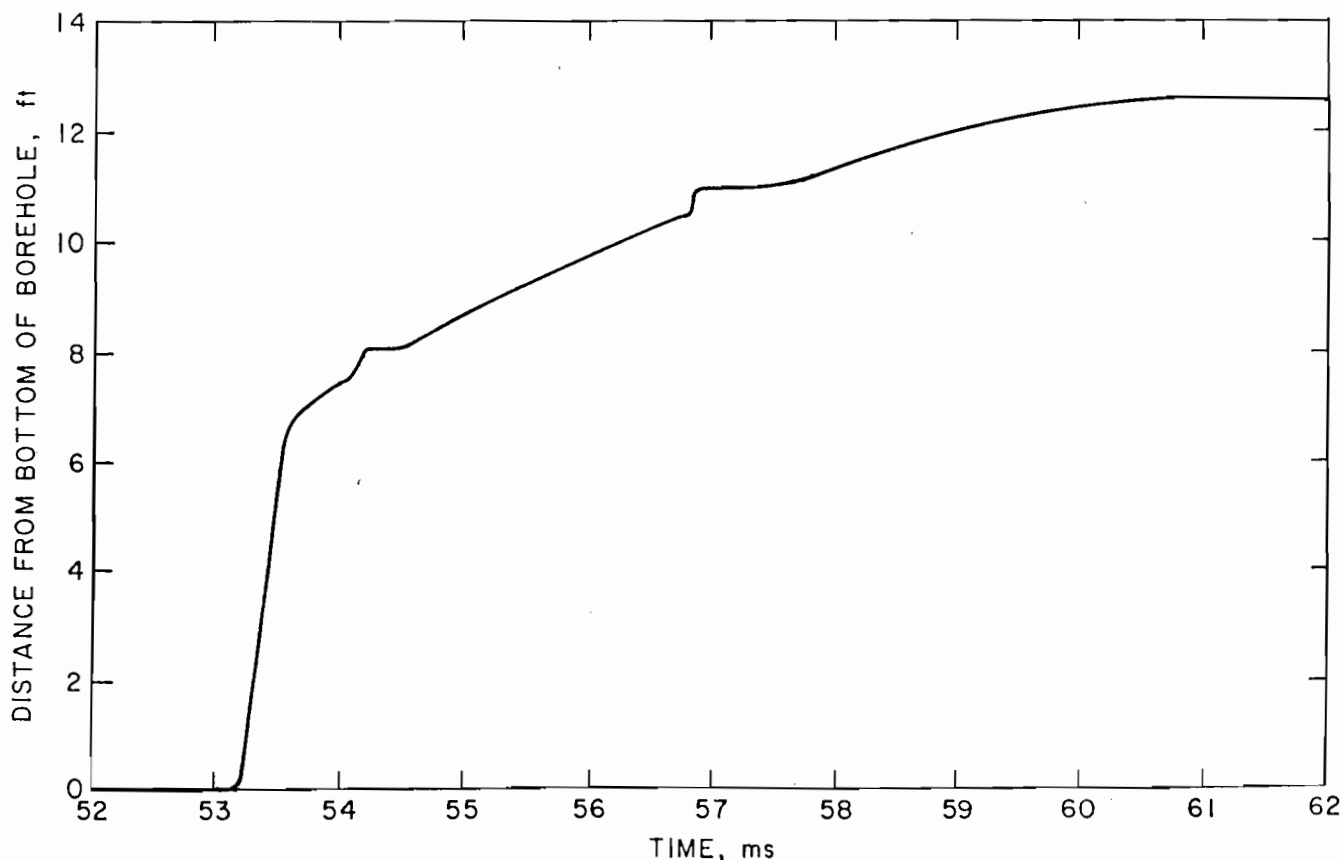


FIGURE 6.—SLIFER data from hole 3 of shot V-1 showing detonation of explosive column and crushing rate of stemming. From Sandia National Laboratories.

TABLE 3. - Calculated and actual stemming ejection times

Shot	Hole diameter, in	Stemming length, in	Calculated ejection time, ms	First stemming movement, ms	Observed ejection time, ms
S-3.....	1-1/2	20	0.5	3.4	13
S-4.....	1-1/2	20	.5	4.6	9
S-6.....	1-1/2	20	.6	6.1	32
V-1 ¹	6	108	2.1	10	Retained

¹6-in cratering shot from Sandia National Laboratories.

also observed the rate of crushing or compaction of the stemming in the borehole. The record for shot V-1 is shown in figure 6. The first 6 ft of the record shows the detonation of the explosive. The detonation velocity is approximately 19,000 ft/s. Above 6 ft, the record shows the crushing of the stemming. This crushing extends to 12-1/2 ft, or 2-1/2 ft from the hole collar, and

takes 7.5 ms to complete. The time taken for the crushing to propagate through the stemming is close to but somewhat less than the observed time for the start of stemming movement. The stemming appears to be bridging in the hole and preventing movement until it is crushed by the pressure pulse in the stemming column, and only then does it start to move.

CONCLUSIONS

High-speed films of single-hole crater test blasts in two surface limestone quarries were analyzed to evaluate the ability of stemming to contain explosive gases. Stemming ejection and burden motions were examined. When sufficient stemming was used, ejection of stemming was prevented. A ratio of length of stemming to charge diameter of 26 or more was found to prevent premature ejection of stemming and venting of gases.

If release of stemming does occur, the time required for stemming ejection may be sufficient to permit burden movement to start, with expansion of explosive gases into the fractured burden and cooling of the explosive gases. An estimate of this cooling was made using thermodynamic principles. In three tests with a ratio of stemming length to charge diameter of 16, the explosive gases cooled below the ignition temperature of methane in the time required for stemming ejection, but venting of gases through

fractures occurred before stemming ejection and the temperature of the vented gas was estimated to be above the methane ignition temperature. For the conditions of these tests, it is thus concluded that a stemming length of 16 charge diameters could have resulted in methane ignition.

Stemming ejection takes much longer and is more complex than would be predicted by a simple calculation based on inertia of the stemming material. Ejection times were three times or more longer than those predicted by a simple inertia model. Reasons for this are the additional time required for the stress wave to crush the stemming material and cause it to start to move and the subsequent decrease of borehole pressure through crushing and expansion of the borehole. A better understanding of these mechanisms requires further research with more sophisticated instrumentation, such as the SLIFER system.