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BLAST DESIGNS TO IMPROVE DRAGLINE STRIPPING RATES

Final Report — Phase I

By
A. Bauer
M. D. Brennan

April 1979

Work Performed Under Contract No. ET-77-C-01-9124

Woodward-Clyde Consultants
Denver, Colorado

and

U. S. Bureau of Mines
Washington, D. C.

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FINAL REPORT - PHASE I

Prepared for

UNITED STATES DEPARTMENT OF ENERGY
Assistant Secretary for Energy Technology
Division of Fossil Fuel Extraction
Mining Research and Development

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By

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This report represents work on a program that was originated by the Interior Department's Bureau of Mines and was transferred to the Department of Energy on October 1, 1977.

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16. Abstract

This report is in accordance with Phase I of the United States Department of Energy and Bureau of Mines Contract No. U.S.D.O.E. DE-AC01-77QQ90147. A thorough literature search was conducted and the current mining practice at nine coal stripping operations reviewed in detail. A review of the state of the art of blast design using current theory and practice is presented.

Based upon this information, a series of recommendations are presented and a detailed test program is outlined for three operations.

The recommendations to increase dragline productivity by improved blast design are directed at filling the bucket more quickly, with greater consistency and with a better fill factor.

a) overburden fragmentation, b) muckpile displacements, and c) blasting reliability. Recommended blast design concepts and operating procedures to provide the desired blast performance include:

1) Increase energy input per BCY by 40 to 100 percent and improve the effective powder distribution by modifying blast parameters such as burden and spacing, collar heights, toe burden, delay intervals and blast tie-in.
2) Introduce use of blast hole dewatering, dryliners and bulk AN/FO for wet holes.
3) Improve quality control in establishing shot patterns and charging blast holes.

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FOREWORD

This report was prepared by Woodward-Clyde Consultants of Denver, Colorado, under Department of Energy Contract No. DE-AC0I-77QQ90147 (formerly ET-77-C-01-9124 and USBM Contract J0275033). It was administered under the technical direction of the U. S. Bureau of Mines Twin Cities Mining Research Center with Mr. Richard Dick acting as Technical Project Officer. The report is a summary of work recently completed as a part of this contract during the period from October 1977 to May 1978 and was first submitted by the authors on May 19, 1978.

The authors wish to acknowledge the following contributors to this study: Lewis L. Oriard for his technical contributions to the design and report stages; Robert M. Jones and William G. Freeman of International Mining Consultants, Inc., for their assistance with the field studies and in preparation of the field study summary reports; Mr. William A. Crosby of Mining Resource Engineering, Ltd. for his assistance in the field studies; and Messrs. Clinton L. Miller and Jack H. Livingstone of Woodward-Clyde Consultants for their assistance in program direction, constructive review and program administration.

PHASE I

BLAST DESIGNS TO IMPROVE DRAGLINE STRIPPING RATES

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BLAST DESIGNS TO IMPROVE DRAGLINE STRIPPING RATES
FINAL REPORT - PHASE I
EXECUTIVE SUMMARY

Objective

This report presents results from the first phase of a proposed two-phase program for developing and demonstrating cost-effective blast designs having the potential for increasing dragline productivity and coal production in the Interior and Northern Great Plains Coal Provinces.

Background

A recent U.S. Bureau of Mines study of western coal mines indicated that 75 percent of the operations were restricted from increasing coal production due to a lack of stripping capacity. Because this study also revealed that an excess of coal loading and hauling capacity was available, it was evident that increased coal production could be achieved through increasing stripping capacity.

Stripping capacity could be increased either through purchase of more equipment or through improved productivity from presently available equipment. More than three-quarters of the western mines included in the Bureau of Mines study utilized draglines as their primary stripping tool, and it appears likely that this trend will continue. Consequently, in the interest of increased national coal production and mining efficiency, the Bureau of Mines awarded this contract (subsequently transferred to the Department of Energy) to develop and demonstrate blast designs with the potential for improving fragmentation and thereby increasing dragline productivity.

The program was to be accomplished in two phases:

- Phase I - Literature review, field studies, improved blast design development, and specific field evaluation program design.
- Phase II - Comprehensive demonstration and evaluation program of improved blast designs from Phase I at three surface coal mines; report findings with recommendations and discussion of impacts upon surface mining systems, costs and productivity.

This report summarizes work accomplished during Phase I of the program (Phase II will commence upon approval of the proposed test program by DOE). Phase I efforts were divided into four discrete tasks, as summarized below.

Task 1 - Literature Review

Recent government publications, manufacturer's data, articles from symposiums and technical journals, and blasting manuals were reviewed to determine present trends with respect to surface coal mine blasting practice. Current blasting theory and concepts were compiled. Relevant subjects are discussed in detail, including the effects of rock type, burden/spacing ratios, blasting direction, explosive charge strength, delay times, collar heights, bottom charges, priming and blast vibration and noise.

Task 2 - Field Studies

Field studies were conducted at nine surface coal mines in the western (5) and midwestern (4) United States. All operations drilled and blasted their overburden and used draglines. Detailed information was gathered pertaining to overburden stratigraphy and geological structure, drilling, explosives loading, blast tie-ins, dragline operating methods and equipment performance. Cost data were obtained where available to supplement unit costs estimated by the authors.

Limestone, massive sandstones, and occasionally shales presented most blasting difficulties. Deck charges were commonly used to break hard bands, but the success of this method was dependent on accurately locating the hard bands. Other key geologic factors affecting blasting were dip of the bedding and orientation of joints and fractures.

A variety of drilling techniques were observed. Eight mines employed vertical drilling, and one used horizontal drilling. Penetration rates were consistently high and averaged 130 ft/Op Hr at a unit drilling cost of about $.07/BCY. Holes ranged from 6-3/4 to 15 inches in diameter with depths ranging from 20 to 180 feet. Drill patterns tended to increase proportionately with hole diameter and ranged from 15 by 15 feet to 54 by 45 feet. In general, toe burdens were excessive, either due to poor design or high highwalls and flat slope angles. Field studies indicated that improved fragmentation could result at most operations if hole sizes and patterns were designed for better explosives distribution.

In general, the properties visited employed similar explosives and loading methods. AN/FO was the most commonly used explosive. Packaged AN/FO and slurry were employed only in wet-hole conditions. Even though AN/FO is less effective when wet, operators were reluctant to dewater holes or use dryliners. Powder factors ranged from .17 to 1.0 lb/BCY and averaged between .4 and .5 lb/BCY. Average unit cost for explosives was about $.06/BCY.

All three conventional tie-in methods were employed: row-on-row, diagonal and chevron. Most operations delayed and tied in the blast so that blast movement was in the direction of the muckpile from the previous shot, which prevented horizontal displacement. This confinement sometimes resulted in highwall damage and abnormal levels of vibration. Best blasting results in terms of fragmentation and diggability were achieved with the longest delay intervals possible without cut-off. Many properties surveyed were using unnecessarily short delays.

Current blasting methods were observed to loosen natural fractures rather than to induce fragmentation. Therefore, the original rock structure remained intact and resisted bucket penetration.

Data regarding dragline size, operating method, performance and cost are summarized for the western and midwestern mines visited. From the nine mines visited, data from 17 draglines were accumulated. Buckets sizes varied from 32 CY to 150 CY. Dragline productivity averaged 37.2 BCY/Op Hr/CY of bucket. Unit dragline operating costs varied from $.10 to $.23/BCY and averaged about $.16/BCY. Since 75 percent of dragline operating costs are fixed, increased productivity could have a substantial impact on cost per BCY.

Task 3 - Develop Improved Blast Design Concepts

The key to increased dragline productivity through blasting is to improve fragmentation and induce muckpile displacement so that the bucket can be filled more quickly, with greater consistency and with a better fill factor. Blast design concepts and operating procedures for improved overburden fragmentation are proposed based on the following general recommendations:

(1) Increase energy input per BCY by 40 to 100 percent and improve effectiveness of powder distribution, both horizontally and vertically, by modifying blast parameters including burden and spacing, collar heights, toe burden, delay intervals and blast tie-in.

(2) Improve quality control in establishing shot patterns and charging blast holes.

(3) Introduce use of blast hole dewatering, dryliners and bulk AN/FO for wet holes.

Where packaged AN/FO was used because of wet conditions, hole diameters could be reduced to achieve the same effect when bulk loading and dryliners are used. The savings in drilling and explosives cost should more than compensate for the dryliner cost.

The most radical change proposed is the concept of using horizontal displacement to minimize existing "tight" muckpile situations. The desired horizontal displacement could cause the dragline to sit 10 to 15 feet below the insitu highwall, requiring the construction of ramps and modifications in dragline working ranges.

Use of square drill patterns with diagonal tie-ins is recommended in lieu of rectangular or diamond patterns with row-on-row or chevron tie-ins, based on the following rationale:

(1) The square pattern is simple to lay out and drill.

(2) It gives maximum face hole frequency, which is important for muckpile displacement

(3) The diagonal tie-in will control vertical drop and will yield a uniform windrow effect along the length of the blast.

Deck charging should be minimized to establish a simple loading and delay procedure. In addition, because short delays have the greatest potential for error, delays should be as long as possible.

Task 4 - Develop Field Evaluation Program

One western and two midwestern mines were selected for demonstration programs. These mines were selected because of their present low dragline productivity, relatively high drilling and blasting costs, and anticipated high degree of cooperation.

Detailed data on costs, performance and operating parameters will be obtained for each demonstration mine so that "before-and-after" evaluations can be made. The following information and measurements will be compiled for conditions before testing and as individual tests are implemented:

(1) Costs - estimated for operation, maintenance, administration and ownership of drilling, blasting and dragline operations.

(2) Dragline Productivity - measured for each digging mode based on BCY/Op Hr and BCY/KWH.

(3) Muckpile Profile - surveyed to determine impact of blast designs on dragline reach and ramping requirements.

(4) Highwall Stability - assessed visually only.

(5) Blast Vibration and Noise - recorded for all tests.

(6) Jointing - surveyed using phototheodolite and photogrammetry techniques to determine preferred direction of blasting.

(7) Geology - determined for locating hard bands to direct placement of deck charges when necessary.

(8) High Speed Photography - used to study effects of variables such as delay interval, stemming, collar height and toe burden.

(9) Explosives Quality - monitored to assess quality of detonating cord, primers and AN/FO.

Specific test programs to improve fragmentation are outlined for the three proposed demonstration sites. The test programs involve four to nine individual tests at each mine, each designed to improve blasting results. Typical tests include evaluation of the following:

(1) Using bulk explosives instead of packaged.

(2) Using blasthole dewatering and dryliners.

(3) Implementing explosives quality control monitoring.

(4) Reducing time lag between loading and blasting.

(5) Eliminating short holes and deck charges.

(6) Increasing delays to maximum possible based on results from high speed photography.

(7) Moving face holes as close to the crest as possible.

(8) Adjusting collar heights to break hard bands without using deck charges or short holes.

(9) Using diagonal tie-ins and square drill patterns.

Detailed unit cost analyses are graphed to illustrate projected impacts of proposed tests and required productivity increases to compensate for increased blasting costs. Productivity increases of 13 to 54 percent will be required to break-even on additional blasting costs, but these are considered reasonable goals with good potential for even better improvement.

1.0 INTRODUCTION

1.1 Objective

This final report presents results from the first phase of a two-phase research program to develop and demonstrate new blast design concepts with the objective of improving fragmentation and thereby dragline productivity.

1.2 Report Structure

As described by Woodward-Clyde Consultants proposal to the Bureau of Mines, the first phase (Phase I) of the program includes the following major tasks:

Task 1 - Literature Review

Task 2 - Field Studies

Task 3 - Develop Improved Blast Design Concepts

Task 4 - Develop Field Evaluation Program

Phase II is the implementation of the Task 4 program, which will commence upon DOE approval.

The literature review (Task 1) includes relevant government publications, manufacturer's data, articles from symposia and technical journals, and blasting manuals. Much of this information provided input for Section 2.0 of this report, "Blasting Theory and Concepts." Abstracts from the literature review are included as Appendix C.

Field Studies (Task 2) were conducted at nine surface coal mines in the western (5) and midwestern (4) United States. Detailed data on current blasting and dragline operating and cost parameters were obtained from these visits, as outlined in Section 3.0, "Summary of Field Study Data."

Improved blast design concepts (Task 3) were developed as a result of the Task 1 and 2 efforts. Theory and operating experience were merged to develop the proposed changes in standard blasting practice discussed in Section 4.0, "Improved Blast Design Concepts."

Detailed field evaluation programs (Task 4) for testing proposed changes in surface coal mine blast designs are described in Sections 5.0 and 6.0, "General Field Evaluation Requirements," and "Specific Blast Test Program." A proposed cooperative agreement between mine operators and Woodward-

Clyde Consultants for conducting test programs is presented in Appendix C. References cited throughout the report are listed in Appendix A.

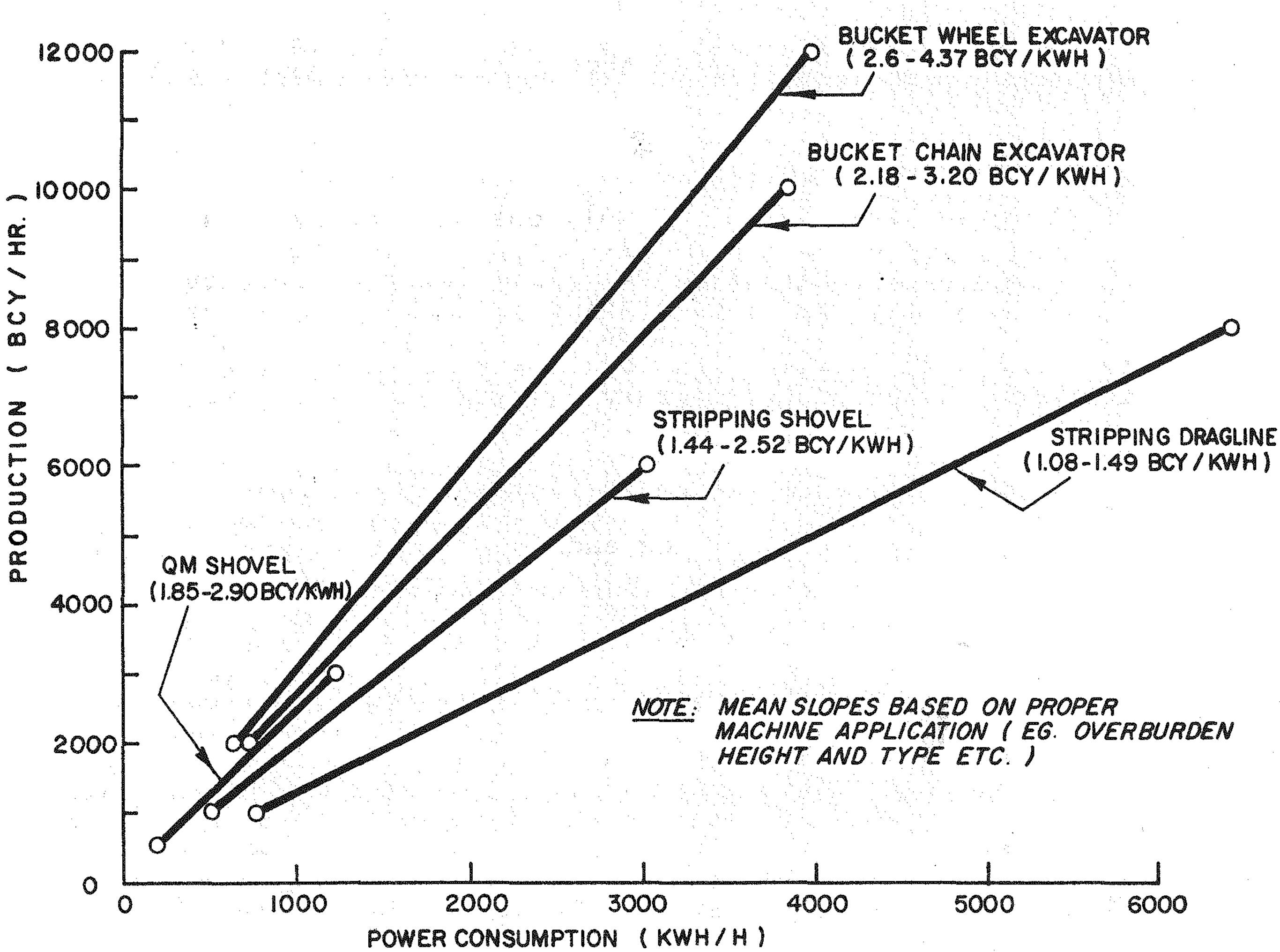
1.3 Background

Improved blast design is only one of many avenues to improving dragline productivity, but it is fundamental to all and can yield significant gains. Dragline productivity, as related to overburden fragmentation, presents a paradox because overburden fragmentation is often good, yet digging is often poor. The explanation for this problem is that the overburden, being of sedimentary origin, is usually laminated and prefractured by nature, which yields a well-fragmented material. However, present blasting practice employs very small energy input, resulting in minimal vertical or horizontal displacement. The end result is an overburden that is fractured but not displaced and thus is resistant to bucket penetration.

The impetus for this project came as the result of a recent Bureau of Mines study (No. S0144081) of western coal mines, which indicated that 75 percent of the operations were restricted from increasing coal production due to a lack of stripping capacity. Equally important, this survey also revealed that an excess of coal loading and hauling capacity was available. An important statistic relevant to this project is that 77 percent of the coal production came from operations that employed draglines as their primary stripping tool, and the forecast indicated that this operating trend would increase with time.

A brief explanation of this trend[1] is that the stripping shovel is deficient in operating radius when compared to the dragline, and this factor becomes more critical as coal operations with deeper overburden come on stream. In fact there are no new stripping shovels being built now nor have there been for the last ten years. Bucket wheel excavators of equal dragline capacity are generally lower in mechanical availability and are sensitive to overburden type.

As illustrated by Figure 1, the stripping dragline has a low ranking on the basis of current efficiency measures such as bank cubic yards (BCY) per operating hour (Op Hr) per cubic yard (CY) of bucket, or BCY per kilowatt hour (KWH) [3], primarily because these units of measurement do not fully express the work done. The quarry/mining shovel relies on the truck to complete the removal of the overburden. The stripping shovel does not displace the overburden as far horizontally or vertically as the dragline, and the bucket wheel excavator can operate only in the narrow spectrum of soft unconsolidated overburdens.



PRODUCTION VS. POWER CONSUMPTION
FOR VARIOUS TYPES OF STRIPPING MACHINES

Job No. 19089

Fig. 1

Proposed new blast design concepts will be oriented toward increasing explosive energy input per BCY and inducing muck displacement. Evaluation of the demonstration program will be in terms of increased dragline productivity expressed in terms of BCY/Op Hr and BCY/KWH. For this program to be acceptable with mine operators, it must be demonstrated that dragline productivity can be increased through improved overburden fragmentation while overall costs remain constant or are reduced. The benefits of the demonstration program are quite obvious, and, therefore, operator acceptance should be obtainable.

The greatest incentive for an operator to increase mine production is to decrease the impact of fixed costs, such as interest on capital and technical/administrative overheads. If dragline productivity cannot be improved, then the mine operator is confronted with a significant capital expenditure for purchase of a new dragline before he can increase overburden stripping capacity and raw coal production. Table I gives current purchase prices for various sized Bucyrus-Erie* draglines. From this table it can be seen that dragline capital cost is approximately $200,000 per CY of bucket capacity. More meaningful is how dragline costs translate into ownership cost per operating hour, as estimated with the following formulas:[4]

(1) Depreciation Cost/Operating Hour =

$$\frac{\left(\text{Total Purchase Price}\right) - \left(\text{Present Value of Salvage}\right)}{\left(\text{Equipment Life in Years}\right)\left(\text{Operating Hours Per Year}\right)}$$

(2) Other Cost Per Operating Hour =

$$\frac{\left(\text{\% Interest, Insurance Taxes, etc.}\right)\left(\text{50\% Total Purchase Price}\right)}{\text{Operating Hours Per Year}}$$

Note: (a) Total purchase price includes delivery, erection and auxiliary equipment such as power distribution, cable reel, etc.

* Reference to specific brand names is made for identification only and does not imply endorsement by the U.S. Department of Energy.

TABLE I

CURRENT BUCYRUS-ERIE DRAGLINE PURCHASE PRICES (MARCH 1978)

DRAGLINE NUMBER	BUCKET CAPACITY (yd^3)	BOOM LENGTH (ft)	WEIGHT (lb. x 1,000,000)	COST* ($ x 1,000,000)
380W	16 - 10	140 - 200	.9	2.1
480W	18 - 12	175 - 215	1.4	3.0
800W	27 - 18	195 - 265	2.2	5.6
1260W	40 - 24	225 - 302	3.1	6.3
1300W	45 - 29	235 - 325	3.6	7.8
1350W	48 - 37	285 - 325	5.2	10.9
1360W	53 - 46	285 - 325	5.3	10.7
1370W	64 - 51	270 - 320	5.6	11.4
1570W	80 - 57	285 - 345	6.3	13.2
2570W	96 - 115	360 - 335	11.5	23.5
3270W	175	330	16.0	35.0

*Price includes one bucket

Add 5% for erection

Add 5% for overseas orders

Add 5% for auxilliary dragline equipment

Job No. 19089

(b) Present value of salvage, based on the equipment life, becomes insignificant as compared to the purchase price and will be deleted from the calculation.

(c) Life of a dragline is estimated as 20 years.

(d) 5500 operating hours per year is estimated for a dragline, based on 80 percent mechanical availability and 85 percent utilization.

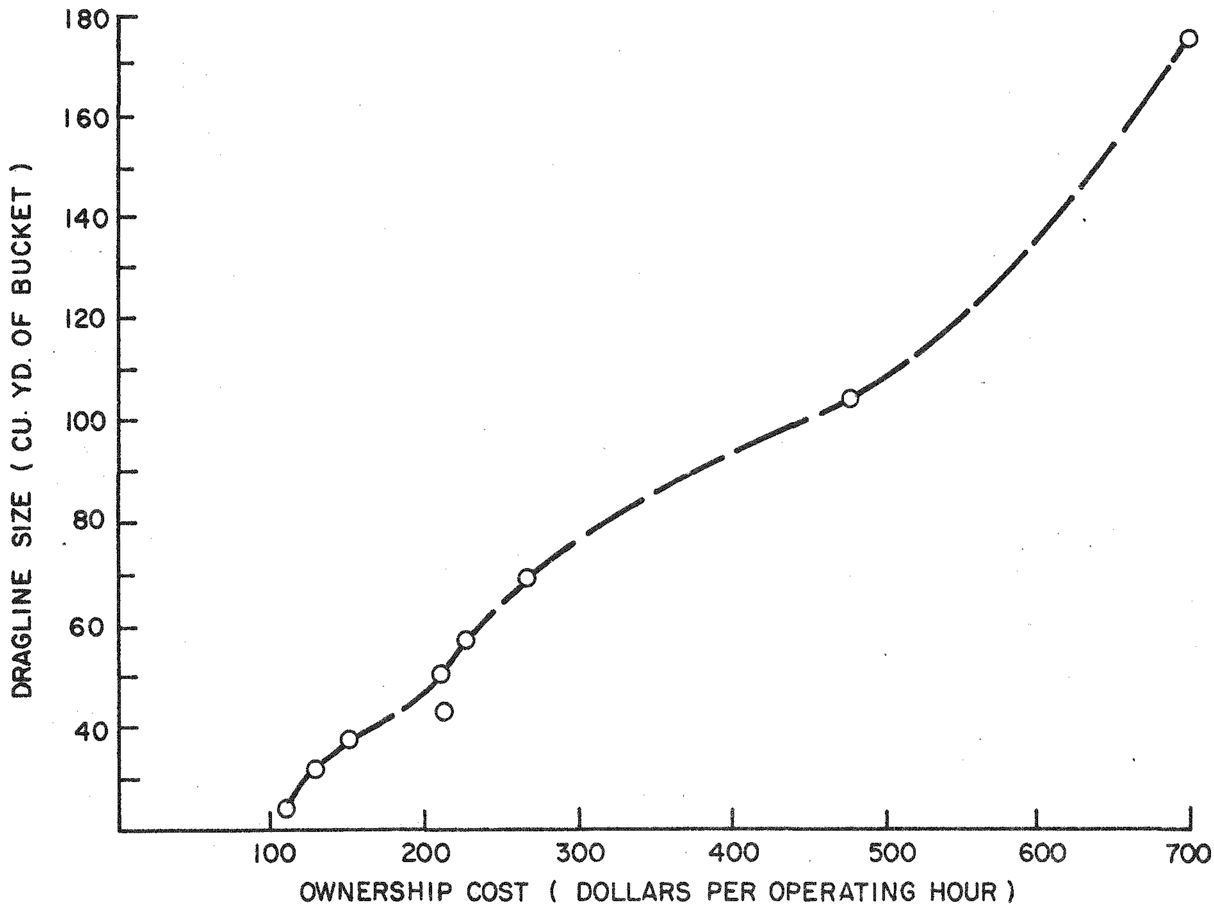
(e) Percent interest, insurance, taxes, etc., estimated to equal 10 percent.

Using Table I and the above formulas, the cost of ownership versus dragline size can be calculated as shown in Figure 2. Field studies conducted for this project indicate that a typical overburden stripping operation utilized a 75 CY dragline, a large rotary drill in the Bucyrus-Erie 61R and Gardner-Denver 120 class, and equipment to handle 5,000,000 lb per year AN/FO. This typical operation could strip approximately 15,000,000 BCY per year. Ownership costs of the drill and explosive handling equipment represent purchase prices of \$900,000 and \$100,000, respectively, and converting these into ownership cost per BCY yields the values shown in Table II.

These values, although not precise, are representative of the relative orders of magnitude among the equipment types. The most significant cost is attributable to the dragline, which supports the mine operator's reluctance to purchase additional dragline capacity and amplifies the importance of increasing the productivity of draglines presently operating.

An increase in dragline productivity could result in reduced overall costs through the ability to modify and improve operating methods. If improved overburden fragmentation increased dragline productivity, the same production could be obtained by equipping the machine with a smaller bucket. A reduction of bucket size for a given dragline permits the use of a longer boom that would effectively increase the operating radius.

Figure 3 illustrates maximum dragline suspended load with respect to machine reach factor.[5] Figure 4 shows the relationship between reach factor and operating radius. For



COST OF OWNERSHIP FOR VARIOUS
SIZES OF DRAGLINE

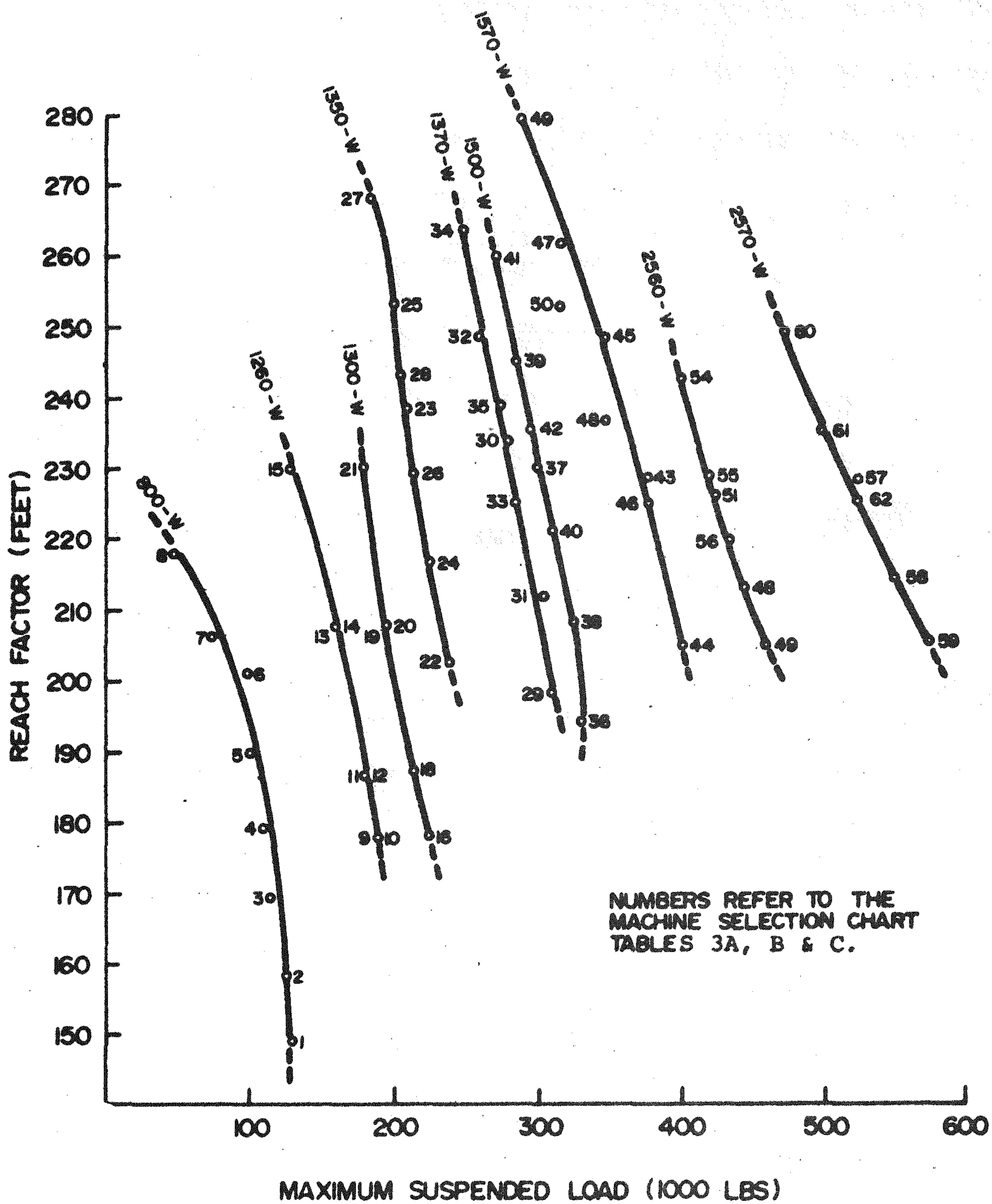
Job No. 19089 Fig. 2

TABLE II

DRAGLINE, DRILL AND EXPLOSIVE EQUIPMENT FOR A TYPICAL OVERBURDEN OPERATION (15,000,000 BCY/YEAR)

EQUIPMENT	CAPITAL COST $	USABLE LIFE (YRS.)	OPER. HRS. PER YEAR	OWNERSHIP COST/OPER. HR.	OWNERSHIP COST/BCY
75CY DRAGLINE (BE-1570W)	15,750,000	20	5500	$286	$.105
ROTARY DRILL (61R-GD120)	900,000	12	5000	24	.008
5,000,000 LB/YR ANFO SET-UP	100,000	8	2000	8.50	.001

Job No. 19089



BUCYRUS-ERIE
DRAGLINE STANDARD MACHINE SELECTION
CHART

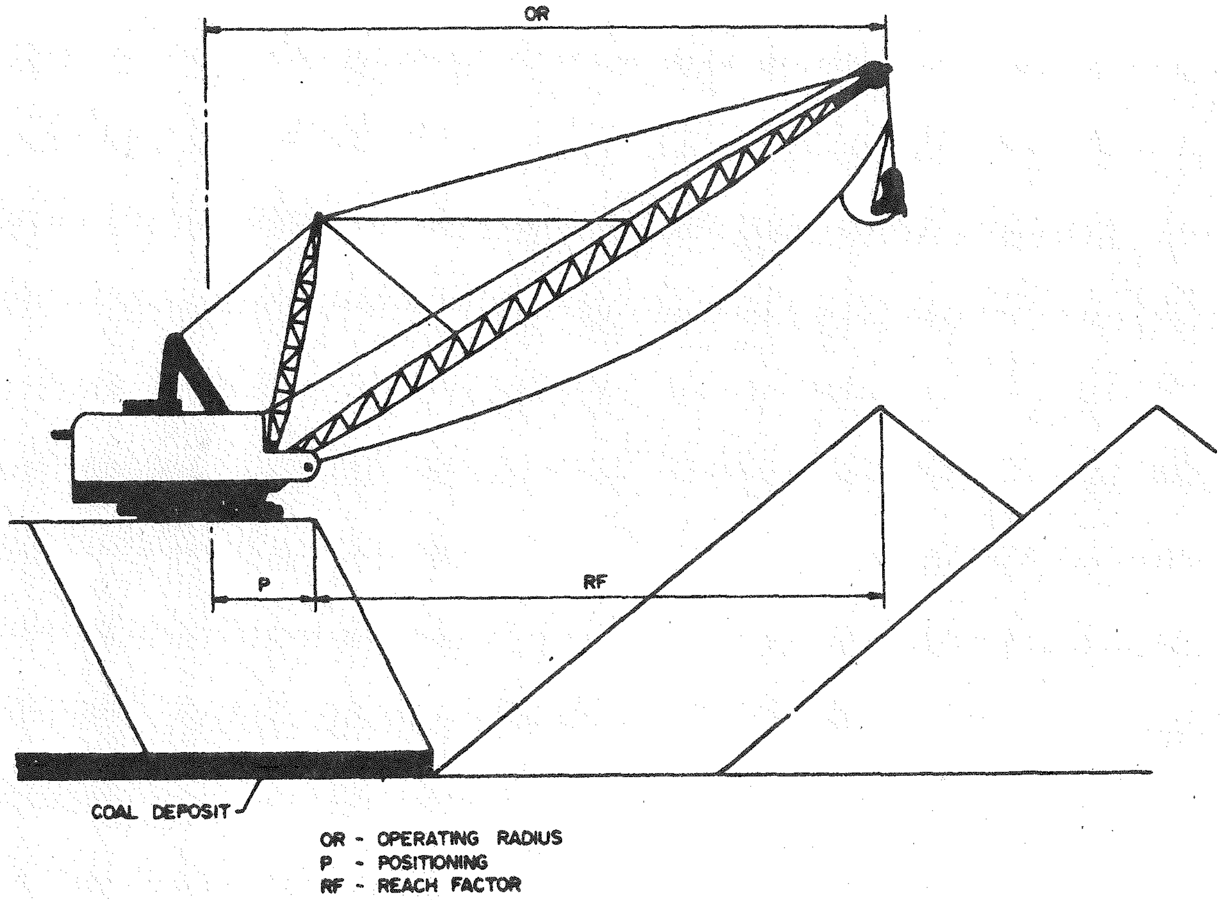


ILLUSTRATION SHOWING THE RELATIONSHIP
BETWEEN OPERATING RADIUS, REACH FACTOR AND POSITIONING

Job No. 19089

Fig. 4

example, if improved overburden fragmentation increased productivity by 30 percent on a BE-1570 dragline with an 80 CY bucket, then the same productivity should be attainable by equipping the same machine with a 57 CY bucket. The boom length could be increased accordingly to improve the reach factor from 205 feet to 280 feet (see Tables III A, B, C). As a result this may permit the operator to:

(a) eliminate benching dug in the chopdown mode

(b) reduce rehandle

(c) increase mining width

(d) remove partings that are presently being excavated with shovels and trucks.

All these items could decrease unit operating costs.

1.4 Preliminary Work

To assess the "state of the art," a literature search and field studies were conducted as Tasks 1 and 2 of this program. The literature search for coal strip mines employing draglines provided a means of gathering base information relative to drill hole depth, diameter, patterns, explosive types, powder factors and dragline operating methods. Very little relevant cost information was available, but the operating parameters served as guidelines for data collection during field studies and as a means to index mines chosen for our field studies relative to the large sampling available in the literature. In addition, the most current and widely accepted blasting theory and concepts were compiled. The blasting theory and concepts believed to be relevant to this project are discussed in the following section.

Field studies were conducted at 9 coal stripping operations, which entailed arrangement of visits, an information gathering session with management personnel and site visits to the mines. Geographically the properties visited were divided into two areas, midwestern and western mines, as listed in Table IV. Actual mine or company names are not used in order to protect confidentiality of data provided by the operators.

All operations visited drilled and blasted their overburden and used draglines. They were quite cooperative in supplying detailed operating information with regards to drilling, blasting and dragline operations; however, only three operations provided good cost data. The subsequent sections of this report summarize information obtained from

TABLE III-A

STANDARD BUCYRUS-ERIE MACHINES SELECTION TABLE

Reference Number	Boom Length (feet)	Boom Angle (deg.)	Operating Radius (feet)	Reach Factor (feet)	Max. Sus. Load (pounds)
800-W with 40-ft. tub					
1	195	35	179	149	130,000
2	195	30	188	158	125,000
3	220	35	199	169	115,000
4	220	30	209	179	110,000
5	245	35	219	189	105,000
6	245	30	231	201	100,000
7	265	35	236	206	95,000
8	265	30	248	218	90,000
1260-W with 50-ft. tub					
9	225	30	215	177.5	190,000
10	235	34	215	177.5	190,000
11	235	30	224	186.5	180,000
12	260	38	224	186.5	180,000
13	260	30	245	207.5	160,000
14	285	38	245	207.5	160,000
15	285	30	267	229.5	130,000
1300-W with 50-ft. tub					
16	235	34	215	177.5	225,000
17	235	30	224	186.5	215,000
18	260	38	224	186.5	215,000
19	260	30	245	207.5	195,000
20	285	38	245	207.5	195,000
21	285	30	267	229.5	180,000

TABLE III-B

STANDARD BUCYRUS-ERIE MACHINES SELECTION TABLE

Reference Number	Boom Length (feet)	Boom Angle (deg.)	Operating Radius (feet)	Reach Factor (feet)	Max. Sus. Load (pounds)
1350-W with 52-ft. tub					
22	267	38	241	202	240,000
23	285	30	277	238	210,000
24	285	38	255	216	225,000
25	302	30	292	253	200,000
26	302	38	268	229	215,000
27	320	30	307	268	185,000
28	320	38	282	243	205,000
1370-W with 58-ft. tub					
29	267	38	241	197.5	310,000
30	285	30	277	233.5	280,000
31	285	38	255	211.5	305,000
32	302	30	292	248.5	260,000
33	302	38	268	224.5	285,000
34	320	30	307	263.5	250,000
35	320	38	282	238.5	275,000
1500-W with 63-ft. tub					
36	267	38	241	193.75	330,000
37	285	30	277	229.75	300,000
38	285	38	255	207.75	325,000
39	302	30	292	244.75	285,000
40	302	38	268	220.75	310,000
41	320	30	307	259.75	270,000
42	320	38	282	234.75	295,000

TABLE III-C

STANDARD BUCYRUS-ERIE MACHINES SELECTION TABLE

Reference Number	Boom Length (feet)	Boom Angle (deg.)	Operating Radius (feet)	Reach Factor (feet)	Max.Sus. Load (pounds)
1570-W with 66-ft. tub					
43	285	30	277	227.5	375,000
44	285	38	254	204.5	400,000
45	310	30	298	248.5	345,000
46	310	38	274	224.5	375,000
47	325	30	311	261.5	315,000
48	325	38	286	236.5	345,000
49	345	30	329	279.5	285,000
50	345	38	302	252.5	315,000
2560-W with 65-ft. tub					
51	275	30	274	225.25	425,000
52	275	35	261	212.25	445,000
53	275	38	253	204.25	460,000
54	295	30	291	242.25	400,000
55	295	35	277	228.25	420,000
56	295	38	268	219.25	435,000
2570-W with 74-ft. tub					
57	285	30	283	227.5	525,000
58	285	35	269	213.5	550,000
59	285	38	260	204.5	575,000
60	310	30	304	248.5	475,000
61	310	35	290	234.5	500,000
62	310	38	280	224.5	525,000

TABLE IV

STRIP COAL MINES INVOLVED IN FIELD STUDY

Zone	Identification of Mine	Location
Midwestern	Midwestern A	Illinois
	Midwestern B	Indiana
	Midwestern C	Kentucky
	Midwestern D	Indiana
Western	Western A	Montana
	Western B	Wyoming
	Western C	Wyoming
	Western D	Colorado
	Western E	New Mexico

the literature review (Task 1), field study data (Task 2), proposed new blasting concepts (Task 3), and specific blast test programs proposed for each mine (Task 4).

2.0 BLASTING THEORY AND CONCEPTS

2.1 General

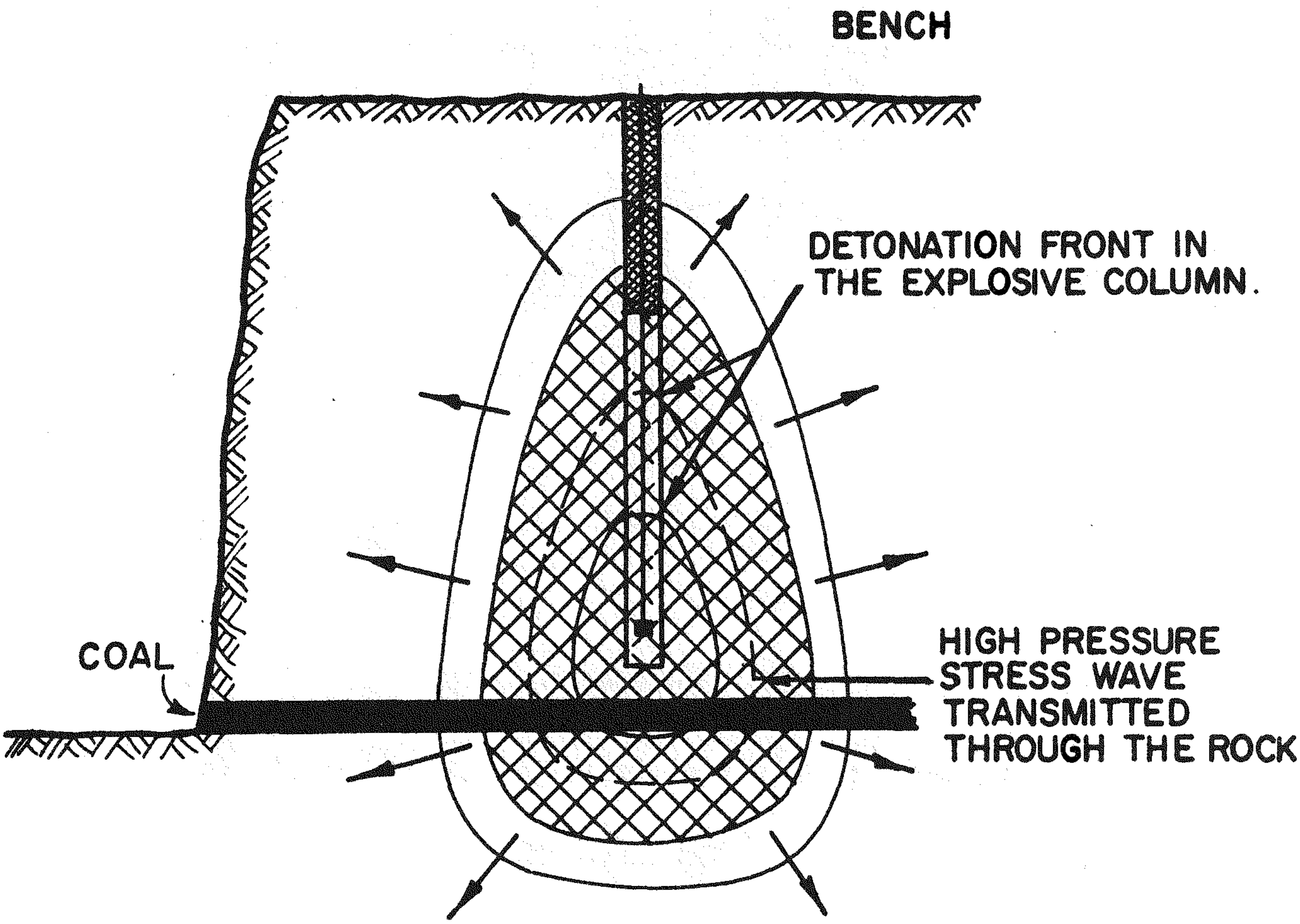
The blast evaluation and proposed designs covered in following sections of this report are based on the theory and concepts outlined in this section. These concepts have been extracted from the literature review (Task 1) and the experience of the authors.

The major factors which affect bench blasting results can be listed as follows:

1. Rock type to be blasted.
2. Charge location.
3. Geological structure, such as predominant fracture orientations and frequency, as related to drill patterns and direction of blasting.
4. Explosive charge and its useful energy expended per foot of borehole or cubic yard or ton of rock.
5. Delay interval employed between rows of holes.
6. Collar height.
7. Bottom charge and toe burden; in coal stripping there is generally no subgrade requirement and the charge is recessed from the coal to avoid excessive dilution.
8. Type, size and position of primers.
9. Blast vibration and noise.
10. Operational control in the field.

2.2 Rock Type

Figure 5 illustrates the behavior of a cylindrical charge fired in a borehole. As the detonation wave travels up the explosive column from the primer, a high-pressure stress wave travels into the rock mass. The figure indicates positions of the detonation front and the stress wave at different time



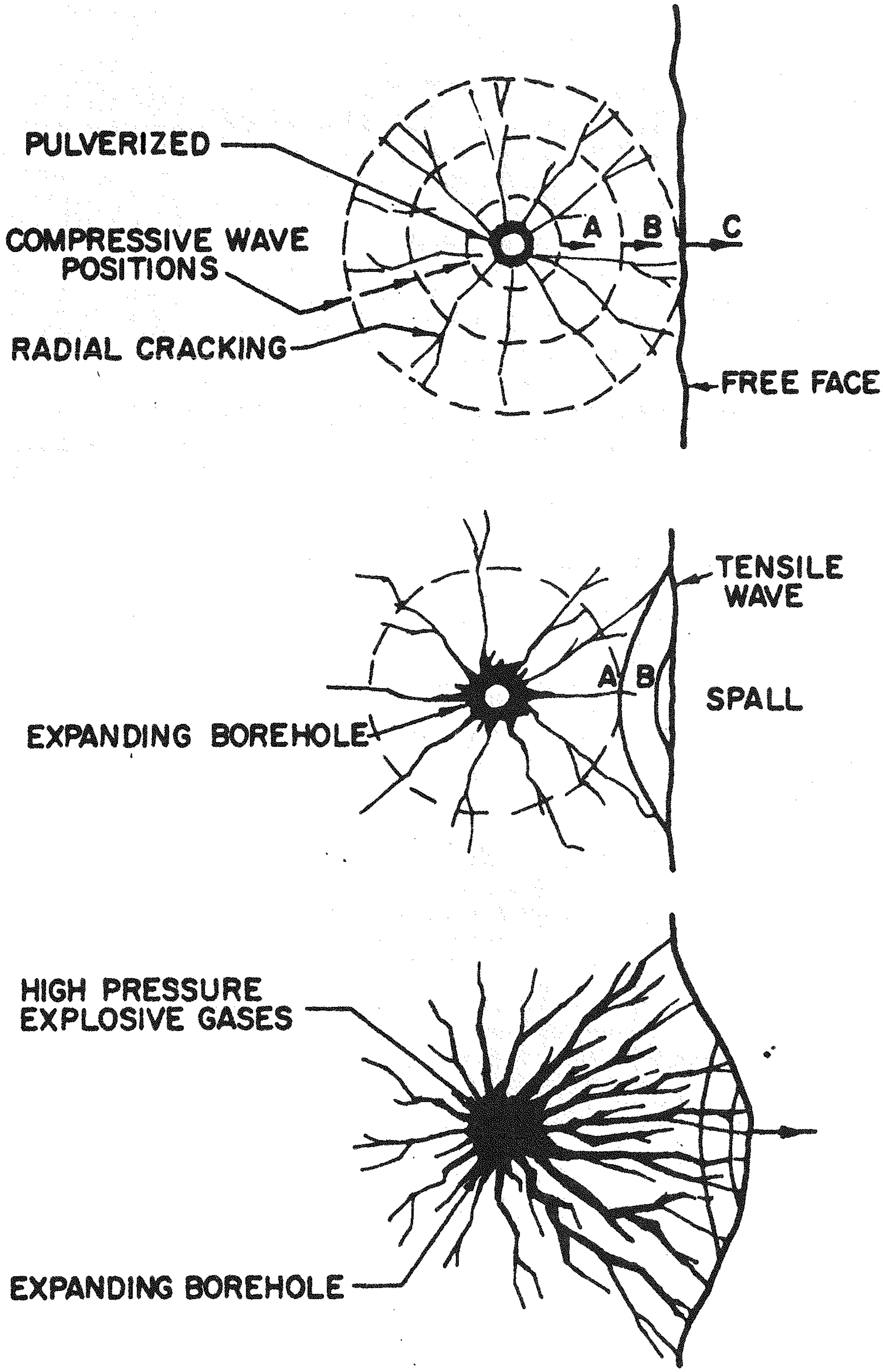
SECTION THROUGH THE FACE WHEN THE CHARGE DETONATES.

intervals. For a bottom-primed charge, the stress wave envelope is pear shaped. Close to the charge the rock fails in compression due to the high borehole pressures, which produces the hoop stress, which is illustrated in Figure 6. Figure 6 indicates a pulverized zone close to the charge with tensile radial fractures beyond this zone. Radial fractures are caused by the expanding compressive wave. Figure 6 also shows three successive positions of the expanding compressive wave front, with the radial cracking proceeding behind the compressive front at a slower speed.

At a free face, the radially expanding compressive wave is reflected as a tensile wave and travels back toward the borehole, causing a series of reflected wave tensile failures. This occurs in massive brittle rocks, and if the reflected tensile failures intersect the subsurface failures, a fragmented rock mass or crater results. In most bench blasts however, the rock has been blasted on the previous lift, resulting in inherent fractures being opened up to the degree that surface reflected tensile failure is minimal and the bulk of the failure proceeds from the borehole outwards. As the radial compressive wave proceeds outwards it produces tensile failure in a direction at right angles to the wave front. Since the radially expanding compressive wave weakens as it travels outward, larger failures occur close to the borehole where the tangential stress is high enough for failure to take place. The high-pressure explosive gases originating at the borehole wall rush into these cracks and attempt to wedge them open. If the burden on the charge is such that the compressive wave is still strong enough to produce tensile failure after reflection at the face, then failure can occur throughout the whole burden. The reflection of the compressive wave producing tensile failure results in the unloading of the burden and this permits the expanding gases to wedge open the subsurface cracks originating at the borehole and to start to expel the rock mass. This wedging action produces considerably more fracturing.

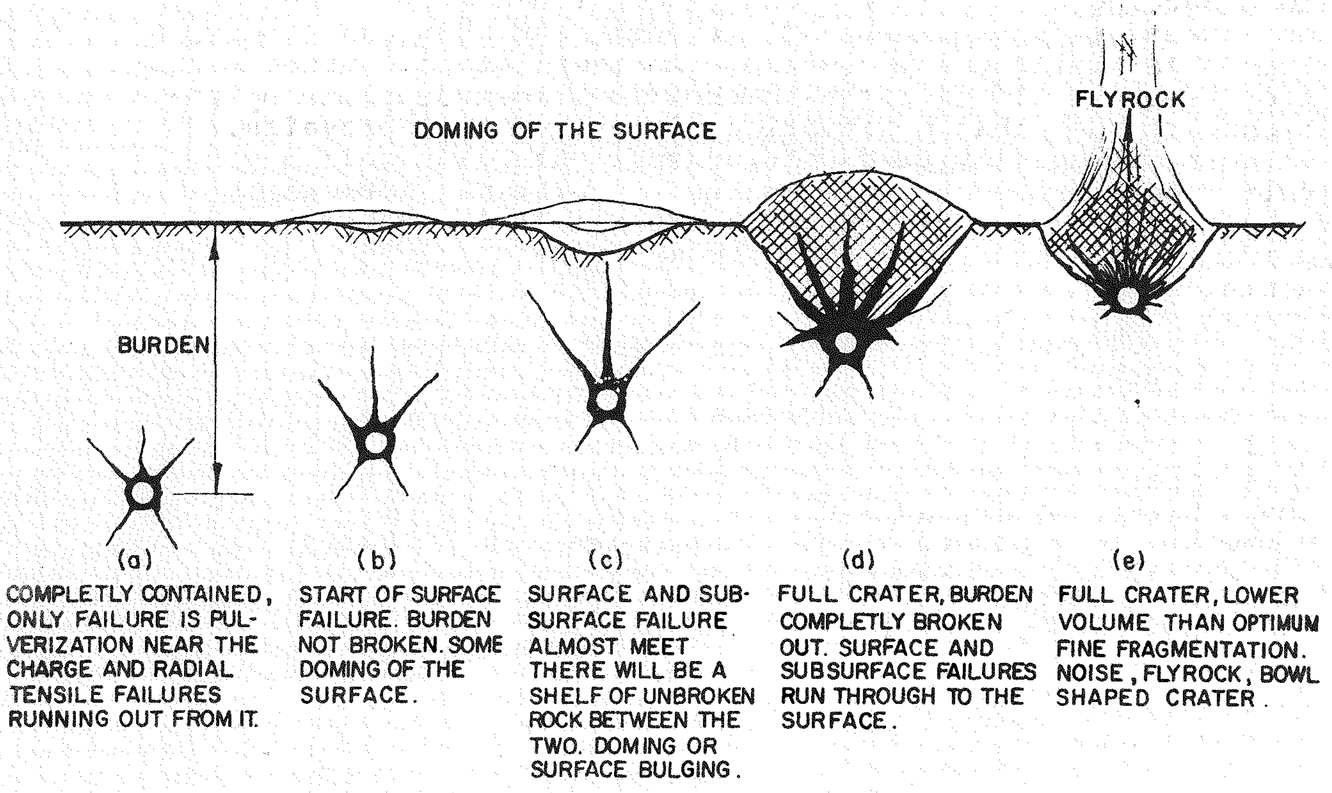
Figure 7 illustrates the effect of varying the burden on a constant charge in the same massive formation. It is a horizontal section through the vertical borehole. For a given material type and explosive charge per foot of hole, there is a maximum size of burden that can be used and still produce a full crater. In blasting rock for most loading operations, it is desirable to detach the burden and produce a full crater.

Figures 7 (a) and (b) show situations that would result in very poor digging conditions in a massive rock formation. Clearly, in order to produce a satisfactory digging condition, the rock mass should be isolated and broken back as accom-



FRACTURES OPENED UP AND PROPAGATED BY GAS EXPANSION PRODUCING AN ISOLATED FRAGMENTED ROCK MASS OR CRATER.

SEQUENCE OF EVENTS OCCURRING IN A HORIZONTAL SECTION OF THE ROCK MASS SURROUNDING A BOREHOLE IN WHICH A CHARGE IS FIRED.



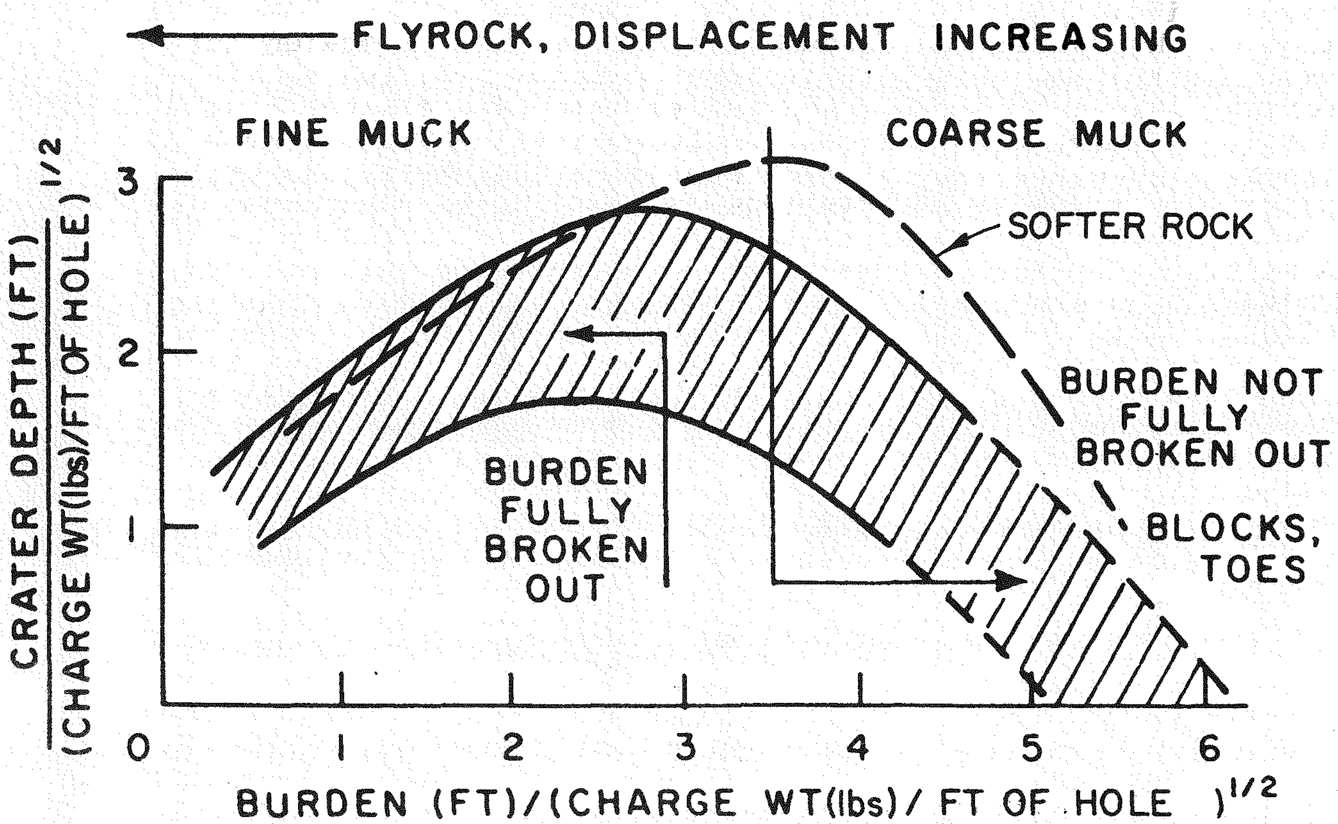
SCHEMATIC OF THE EFFECT OF DECREASING THE BURDEN ON SIMILAR CHARGES FIRED IN ROCK

Job No. 19089 | Fig. 7

plished by the charge shown in Figure 7(d). Craters produced by one hole should also overlap those produced by the next. If the burden is too shallow on the charge, then in addition to producing a full crater and fine fragmentation, there will be excessive displacement of the muck, and this can produce a muck pile having a low profile. An inefficient loading operation results. To avoid this condition, most dragline operations are such that a situation similar to 7(a) prevails. Although in heavily prefractured soft materials this may result in satisfactory performance, it will not work well in massive materials. A situation between 7(c) and 7(d) would be preferable, since there is a strong relationship between fragmentation, "diggability" and displacement.

Figure 5 also indicates three major regions of interest with regard to a column charge. The first two are at the top and bottom end of the charge where the expansion is hemispherical. The third is the central portion where the expansion effects are cylindrical. Figure 8 illustrates the effect of the sizes of cylindrical charges fired in granite. In Figure 8 the scaled depth of crater, the depth of the crater (ft) divided by the square root of explosive weight per foot (lb), is plotted versus the scaled burden (where scaled burden is the burden in feet divided by the square root of the charge weight in lb/ft of hole). In determining the charge weight it is customary to use an equivalent weight of a standard explosive charge. AN/FO is taken as this standard and all other weights are expressed relative to their equivalent AN/FO weights.

As indicated by Figure 8, scaled burdens greater than 4.0 would tend to produce coarser fragmentation, depending on jointing of the rock. This would give results in which the burden was not fully broken out, resulting in ledges and poor digging for small capacity shovels. The fragmentation in this scaled burden region is strongly influenced by the rock structure. As the burden is reduced to a scaled value of 3 or less, the burden is fully detached and fragmentation and muckpile displacement are improved. The inherent jointing and fracturing in this region strongly modify the explosive action. The position of the optimum burden in Figure 8 varies with the material type, hardness, and the geological structure, as well as the charge weight and blasthole size. For rocks softer than granite or having high fracture frequency, the optimum would move to the right, that is, to larger burdens. For harder materials, smaller burdens would be required, or explosives having more energy per foot of borehole (bulk strength) would be required to achieve the same result.



THE CHANGE IN CRATER DEPTH PRODUCED BY DIFFERENT BURDENS ON CYLINDRICAL CHARGES FIRED IN GRANITE

2.3 Burden/Spacing Ratio

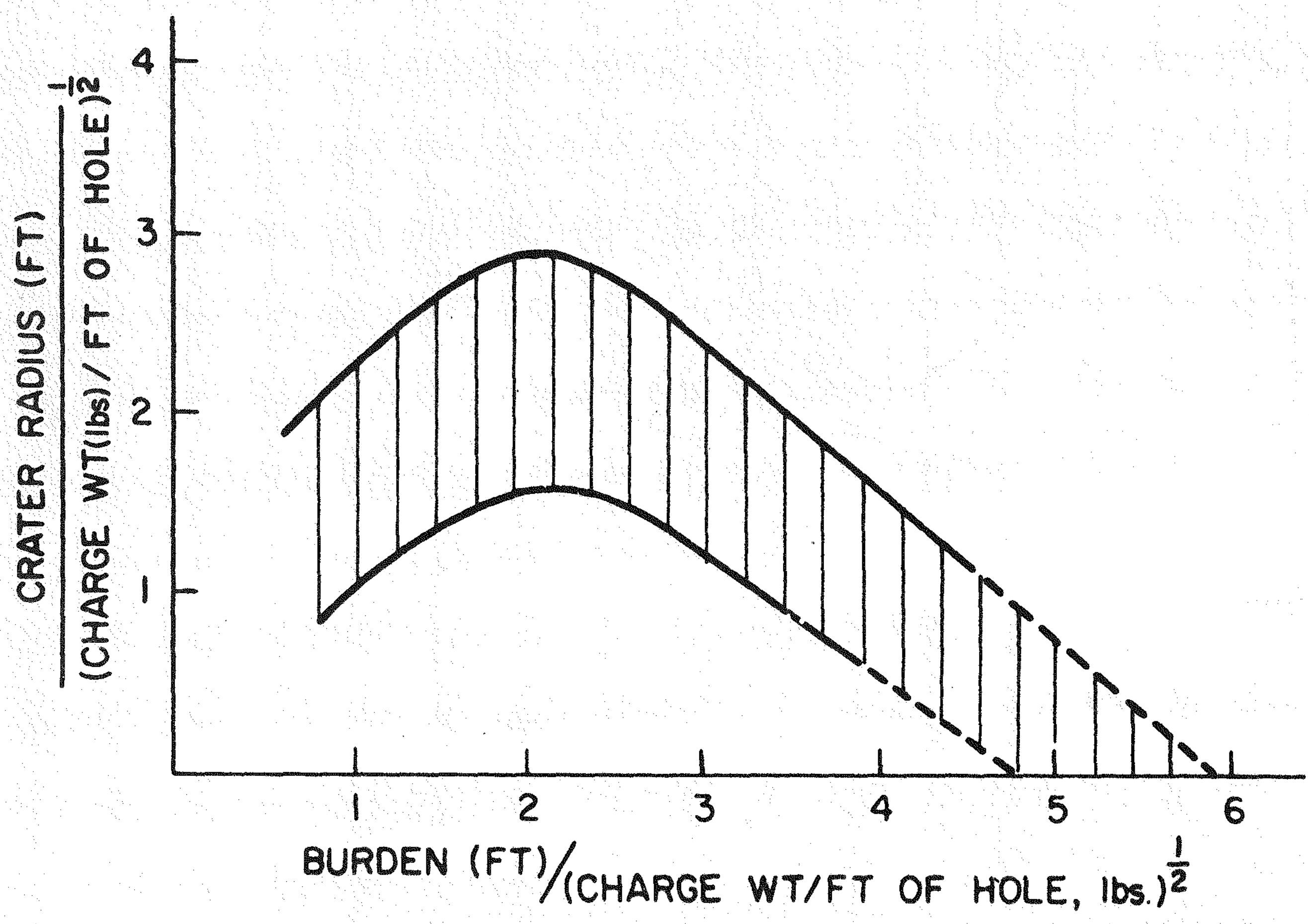
Figure 9 indicates the change in the scaled crater radius that occurs for the same scaled burden as in Figure 8. Clearly the optimum crater radius occurs at a lower value of scaled burden than that which would produce the maximum depth of crater. This is shown more strikingly in Figure 10 which is a plot of the crater radius/burden ratio versus the scaled burden. As the scaled burden is reduced below 3.0, the ratio continues to increase and the crater assumes a flatter bowl shape. In addition to this, the fragmentation becomes much finer. Therefore, for a given explosive consumption in brittle rocks, there is an advantage in blasting with burdens smaller than the spacings. Figures 8 and 9 show that a reduction of the burden to a value lower than the full crater value can be accompanied by an increase in spacing. This effect is often achieved in practice by drilling the holes in a square pattern and then delaying them in such a manner as to produce a blast "en-echelon", or along the diagonals. This gives a burden to spacing ratio of 1:2 without the disadvantage that a rectangular row-by-row blast would produce with a large spacing along the front. Row-by-row patterns also produce excessive toes or hard bottom due to face irregularity, and this is accompanied by excessive flyrock and displacement due to overloading.

The effects at the end of a column charge can be simulated by considering the six to eight diameters of charge at the end of the column as acting like a spherical charge with its center located 3 to 4 diameters into the column charge. Figure 11 shows the effect of varying the burden on the crater radius produced by spherical charges in granite. As was seen in the cylindrical charge region, there is no advantage to blasting with burdens below the optimum. It is desirable for the burdens and loading to be such that cratering is achieved uniformly throughout the column.

The subject of burden/spacing ratio has been discussed by many individuals and is well covered by Hagan (1975).[2] The basic conclusions are:

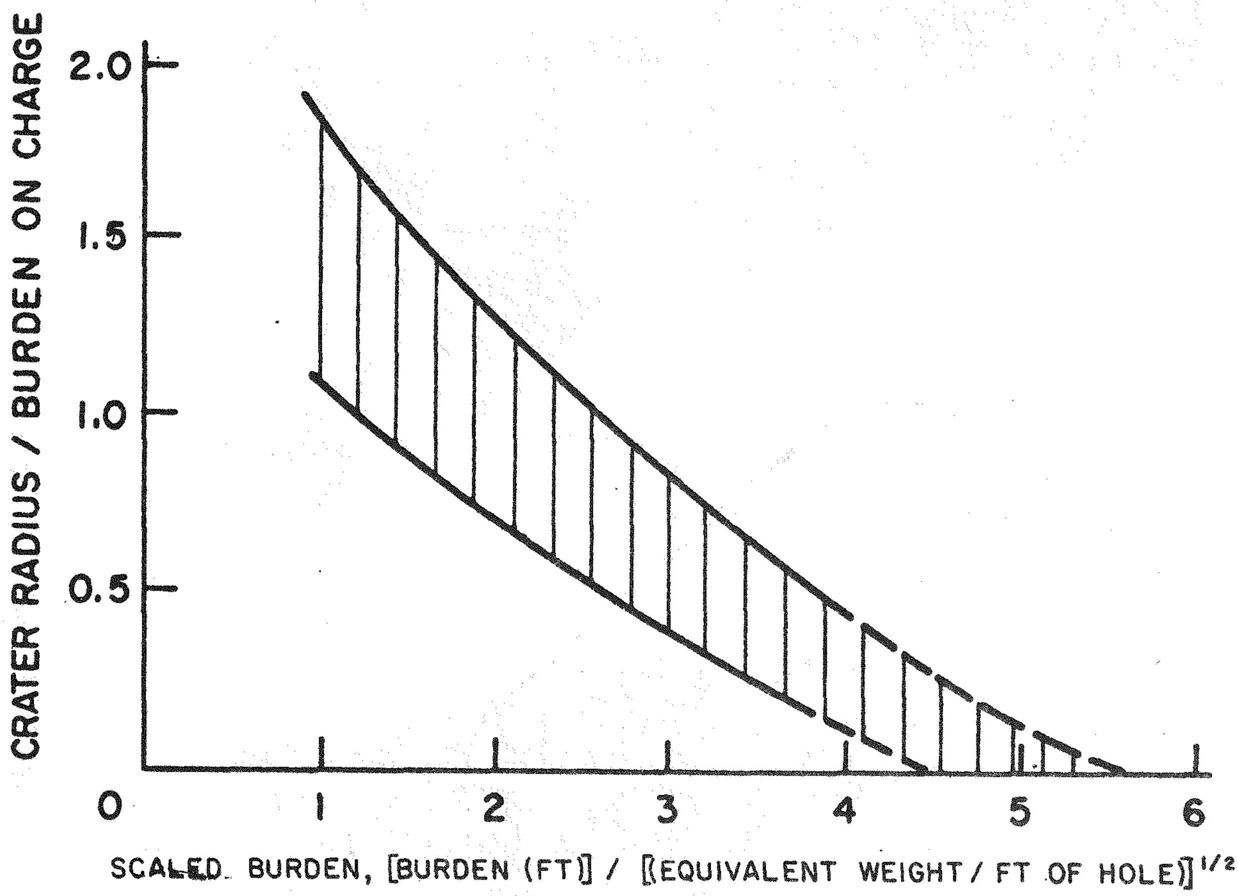
(1) Burden/Spacing ratios between 1:2 and 1:5 are best in most rock types, with the optimum occurring at approximately 1:2 to 1:3. However, this also depends on the rock structure and delay pattern.

(2) Using a square drill pattern, the best tie-in is achieved with the V_1 configuration. Tie-in is on the diagonal (45^o to the crest) with a B/S ratio of 1:2. The drill pattern is simple and there is no difficulty



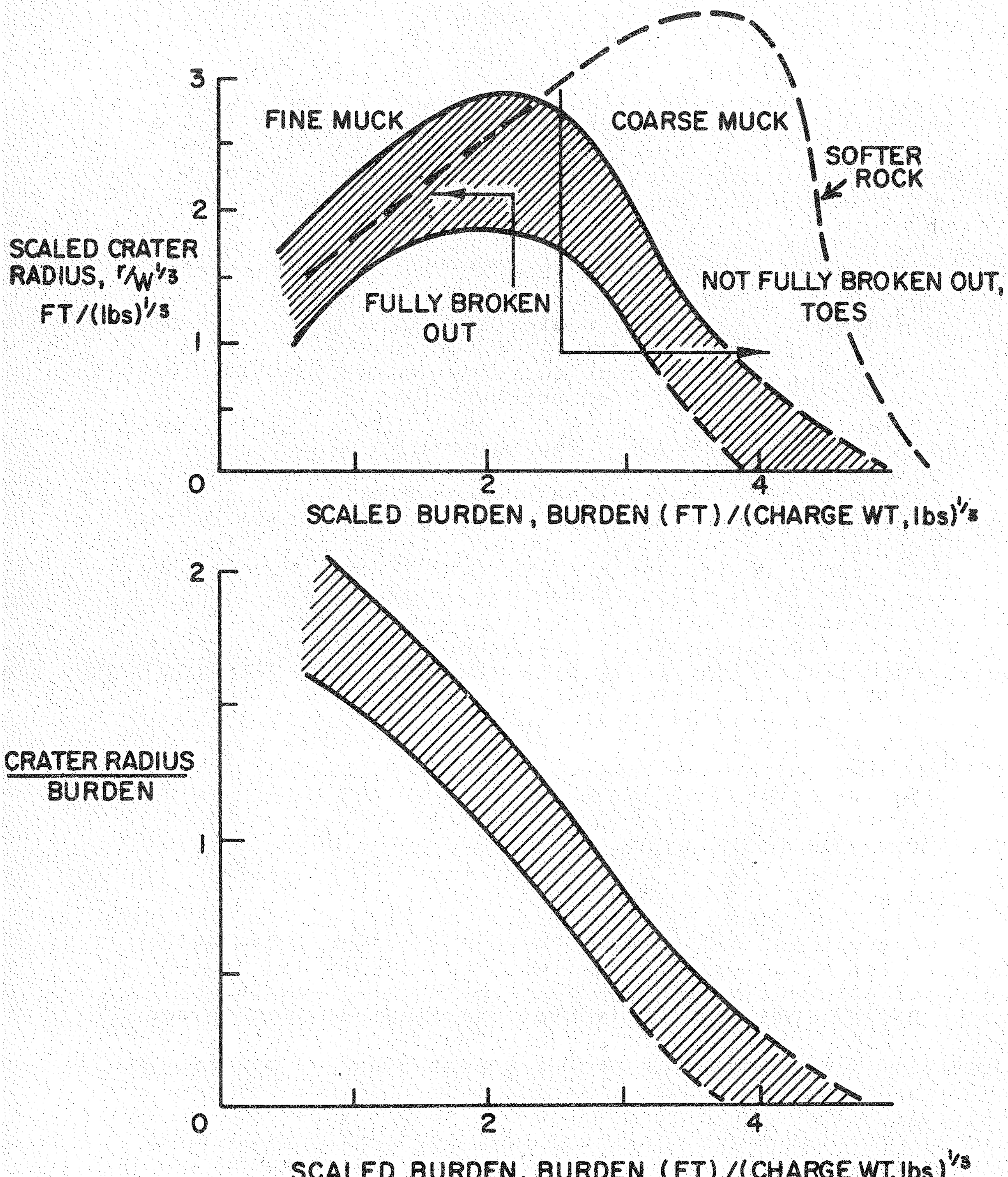
THE CHANGE IN CRATER RADIUS PRODUCED BY DIFFERENT BURDENS ON CYLINDRICAL CHARGES FIRED IN QUARTZITE.

Job No. 19089 Fig. 9



THE EFFECT PRODUCED ON THE CRATER RADIUS / BURDEN WITH DIFFERENT SCALED BURDENS ON CYLINDRICAL CHARGES FIRED IN GRANITE

Job No. 19089 Fig. 10

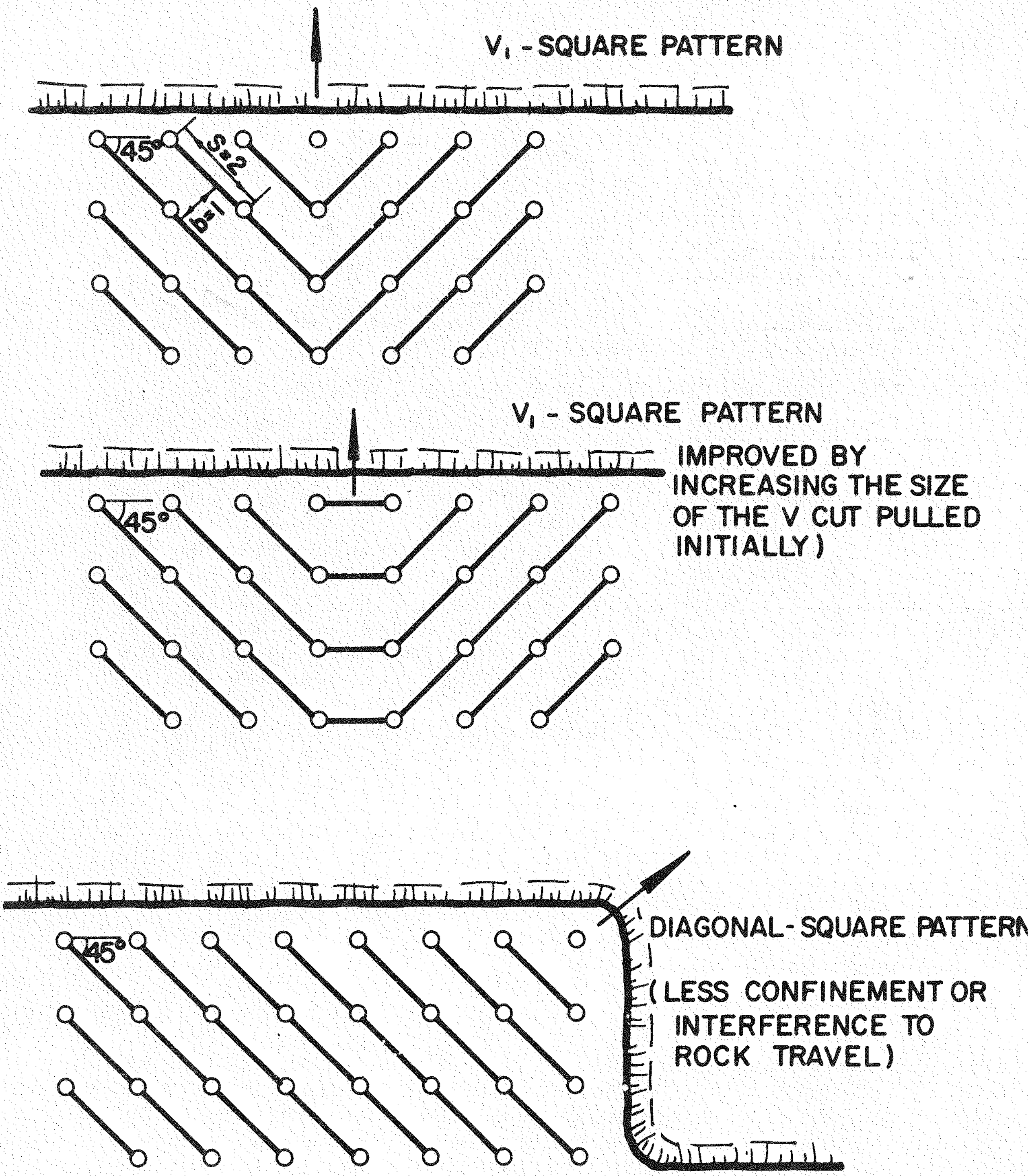


EFFECT ON THE CRATER RADIUS OF CHANGING THE BURDEN ON A SQUAT CHARGE (LENGTH / DIAMETER 6:1) IN GRANITE.

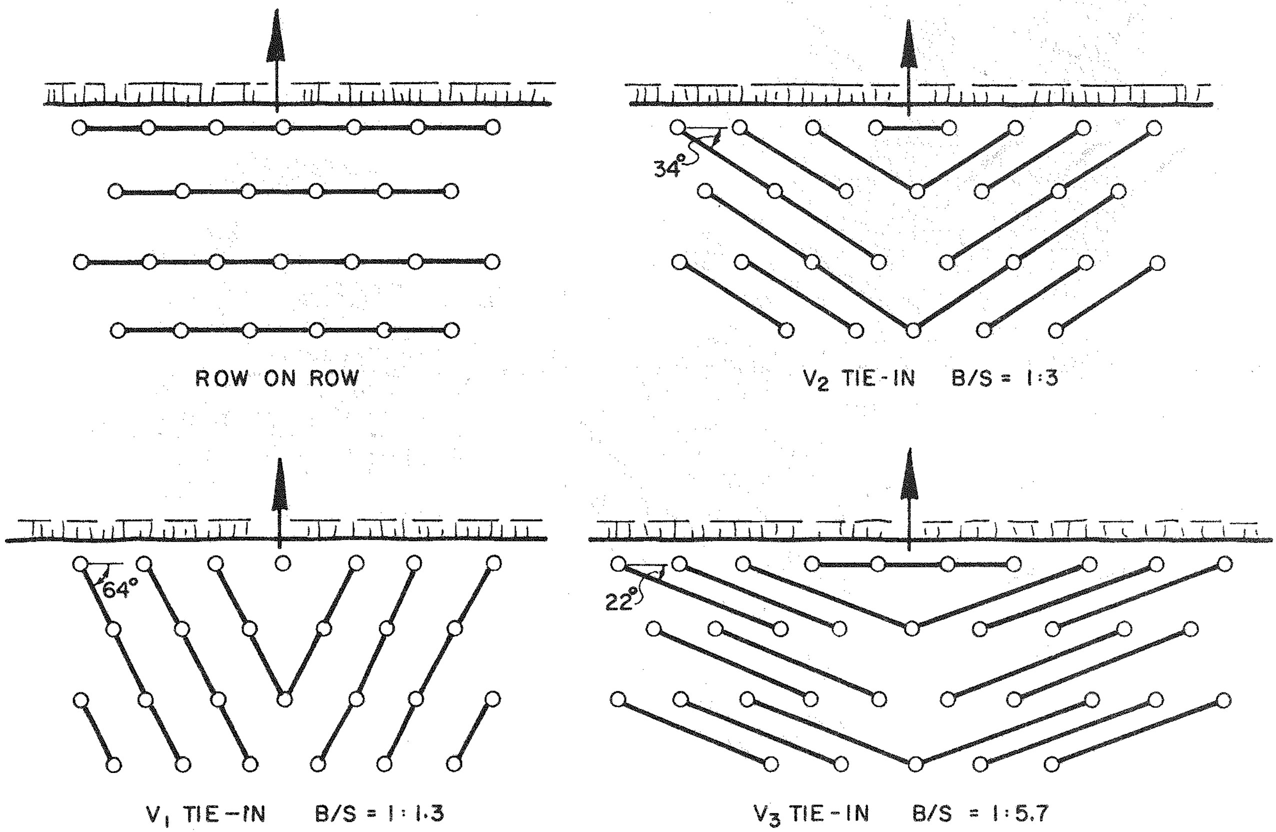
with irregularities in the new face formed. The V_2 and V_3 tie-ins give greater B/S ratios (up to 1:5) which can result in excessive horizontal displacement of the muckpile. The V_1 chevron is somewhat restrictive to horizontal displacement of the rock unless a large V cut is pulled initially or, even better, the blast is shot diagonally from one end (see Figure 12).

(3) If the blast is to be shot row on row, the alternating rows should be staggered to minimize overlap and premature arresting of the radial fractures propagated by each blasthole. This drill pattern can be tied in using a V_1, V_2 or V_3 configuration where the angle to the crest is decreased accordingly (see Figure 13). The V_1 tie-in is at 64° to the crest and the B/S ratio is approximately 1:1.3. Confinement in this case is too great. The V_2 tie-in is at 34° to the crest and the B/S ratio is 1:3. This tie seems to give the best results. The V_3 tie-in has a very shallow angle to the crest and a very large B/S ratio. It is likely that this tie-in would result in excessive displacement of the muckpile, similar to the row-on-row. Also, the probability of hard bottom pockets is considerable based on the large B/S ratio. In addition, an irregularly-shaped face is anticipated after blasting unless fill-in holes are drilled along the perimeter. Hagan expresses the opinion that the best results would be obtained with the V_2 tie-in for the staggered drill pattern. As in the case of the square pattern, a diagonal pattern shot from one end of the blast would increase the amount of free face and most probably improve the fragmentation process (see Figure 14). In addition, the V_2 and V_3 tie-in eliminate the tight V at the point of initiation.

In general, and in the absence of any unique geological structure, the V_1 and V_2 tie-ins, with the square and staggered drill patterns respectively, give the best results in terms of B/S geometry and horizontal displacement (e.g. excessive horizontal displacement is generally not desirable from a productivity viewpoint). In both cases, the amount of free face for blasting is improved by employing diagonal tie-ins starting from the free face created by the previous blast. It should be noted, however, that the staggered drill pattern is more difficult to drill out. In addition, where staggered rectangular patterns are employed, there is a loss of face-hole frequency which is important for moving the toe burden created by the previous blast.



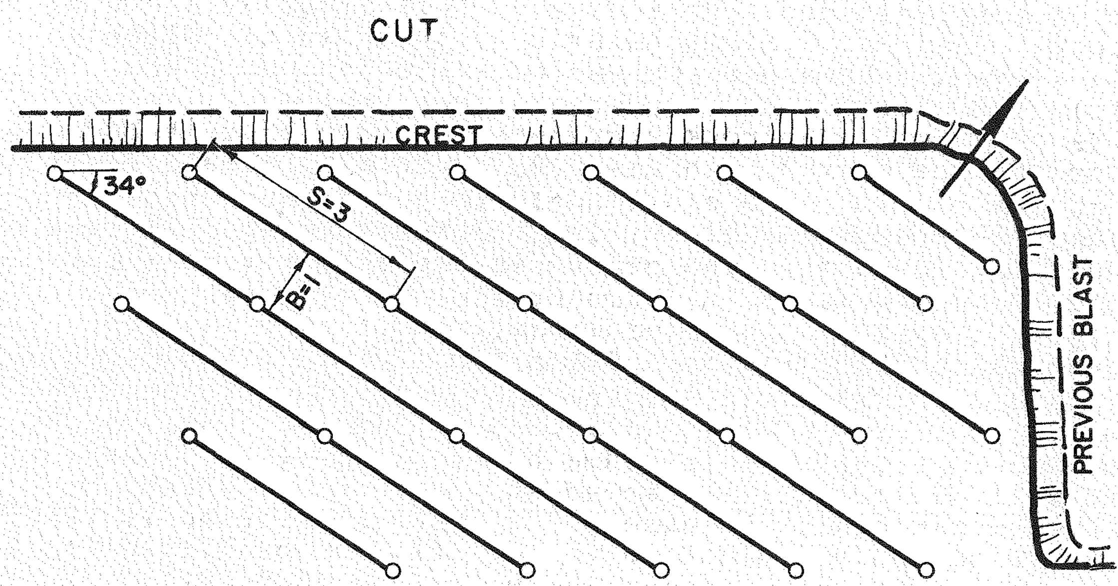
V_1 - SQUARE PATTERN WITH IMPROVEMENTS



ROW ON ROW, V_1, V_2 and V_3 TIE-INS USING A STAGGERED SQUARE DRILL PATTERN

Job No. 19089

Fig. 13



DIAGONAL TIE-IN USING V_2 CONFIGURATION FOR A STAGGERED SQUARE DRILL PATTERN.

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Fig. 14

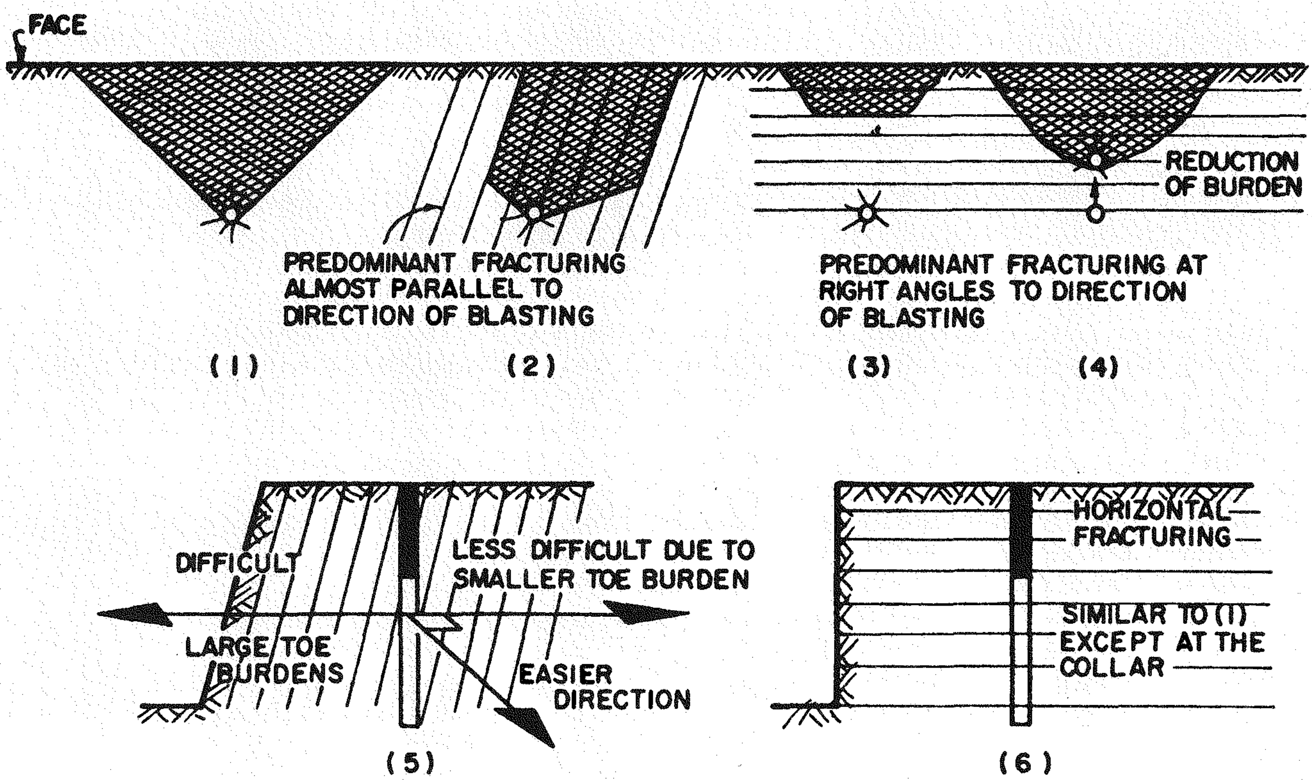
2.4 Direction of Blasting

In addition to cratering being strongly influenced by the rock type being blasted, the rock structure also has a strong influence. Figure 15 is a schematic of a horizontal slice through a charged borehole perpendicular to the rock face. In case (1), the charging and burden are such that a full crater would be produced. The rock structure was such that it had no influence on the crater. In case (2) there is a parallel set of fractures running at a small angle to the direction of blasting. The resulting crater is influenced by this pattern of structural weakness. The crater is reduced in radius and the optimum burden for a full crater would be increased slightly. In case (3) the predominant rock fracturing is at right angles to the direction of blasting. This often requires more energy to displace the rock although fragmentation is usually improved. The optimum burden is lower than before as shown in case (4). If the same burden is used as before, then it will not be fully detached and there will be sections of unblasted material. In case (5) a vertical section is run through the borehole. Blasting to the left or down dip will often be more difficult because the preceding blast will often leave a large toe burden, particularly if the dip is in the range of 40° - 60°. Blasting to the right or up dip often will not be as difficult since, with a vertical blast hole, the toe burden often will not be as large. With equal toe burdens, the reverse would be true. The easier direction to blast is along the strike, though fragmentation is usually not as fine. In case (6) a vertical section through the blast hole indicates that the blasting should be similar to case (1), which would be normal except at the collar. If large collar heights are used, the top fragmentation will be determined by the fracture interval.

In actual practice most situations are not as simple as have been indicated here. However, these examples are useful for illustrating the principles involved.

2.5 Explosive Charge Strength

In addition to such factors as water resistance, critical diameter, minimum primer weight, etc., the most important feature of an explosive used in rock blasting is its ability to do work. This can be expressed on a weight strength basis (volume of rock per weight of explosive), on a powder factor basis (weight of explosive per volume of rock) and on a volume or bulk strength basis (volume of rock per volume of explosive). It is convenient to relate explosives to a standard, and in recent years AN/FO has been selected as the standard because of its wide use. Those explosives having the same



ILLUSTRATIONS OF THE EFFECT OF ROCK
STRUCTURE ON CRATER FORMATION

bulk strength will give essentially the same results on the same blast pattern.

In Table V are listed the weight and bulk strengths relative to AN/FO at a density of 0.85 gm/cc for some commercial slurry products and aluminized AN containing various percentages of aluminum. The values for Al/AN/FO based on our calculations are slightly lower than those issued by the trade, and so both sets of values are listed.

The data in Table V can be used in several ways. For example, if a satisfactory powder factor is known for any one of the explosives listed, it is possible to obtain the powder factor of any of the other explosives required to yield the same result. If a powder factor of 1 lb/CY of AN/FO gave satisfactory results, then a powder factor of 1.16 lb/CY of Tovex 20 E would produce the same results, as would 0.81 lb/CY of a 10 percent aluminized AN/FO. However, in many cases equivalent powder factors are at a much higher explosive cost. Drilling cost savings could not justify their use in low-cost-drilling ground.

Weight strengths relative to AN/FO can be used in this same manner. Bulk strengths, on the other hand, refer to the energy per foot of borehole and therefore take the explosive density into account. As such they are more useful and indicate the relative volumes of rock per hole that can be blasted. For example, if the bulk strength of AN/FO is taken as 1.0, then Tovex 20E is 1.16 and Tovex 30E is 1.13. That is, these products can be shot on patterns yielding approximately 16 percent and 13 percent more tonnage, respectively, than AN/FO, to give the same equivalent powder factor. However, if patterns are calculated on the basis of an equal weight of Tovex 20E or 30E doing as well or better than the same weight of AN/FO, then the material will be undershot.

2.6 Delay Times

If cratering action is being obtained from each blasthole, it is important to have adequate time between successive series of holes so that there is burden relief on one row of holes before the next is fired. The time for the burden to start moving in a bench blast is proportional to the size of the burden. Put another way, it takes time for the fracturing process, followed by the gas expansion, to take place. If the delay time between successive holes is not longer than the minimum detachment time, the blast will be choked, the digging will be hard because the formation is tight, and the floor will have frequent toes. Depending on the collars along the back of the blast, there may or may not be throwback of mate-

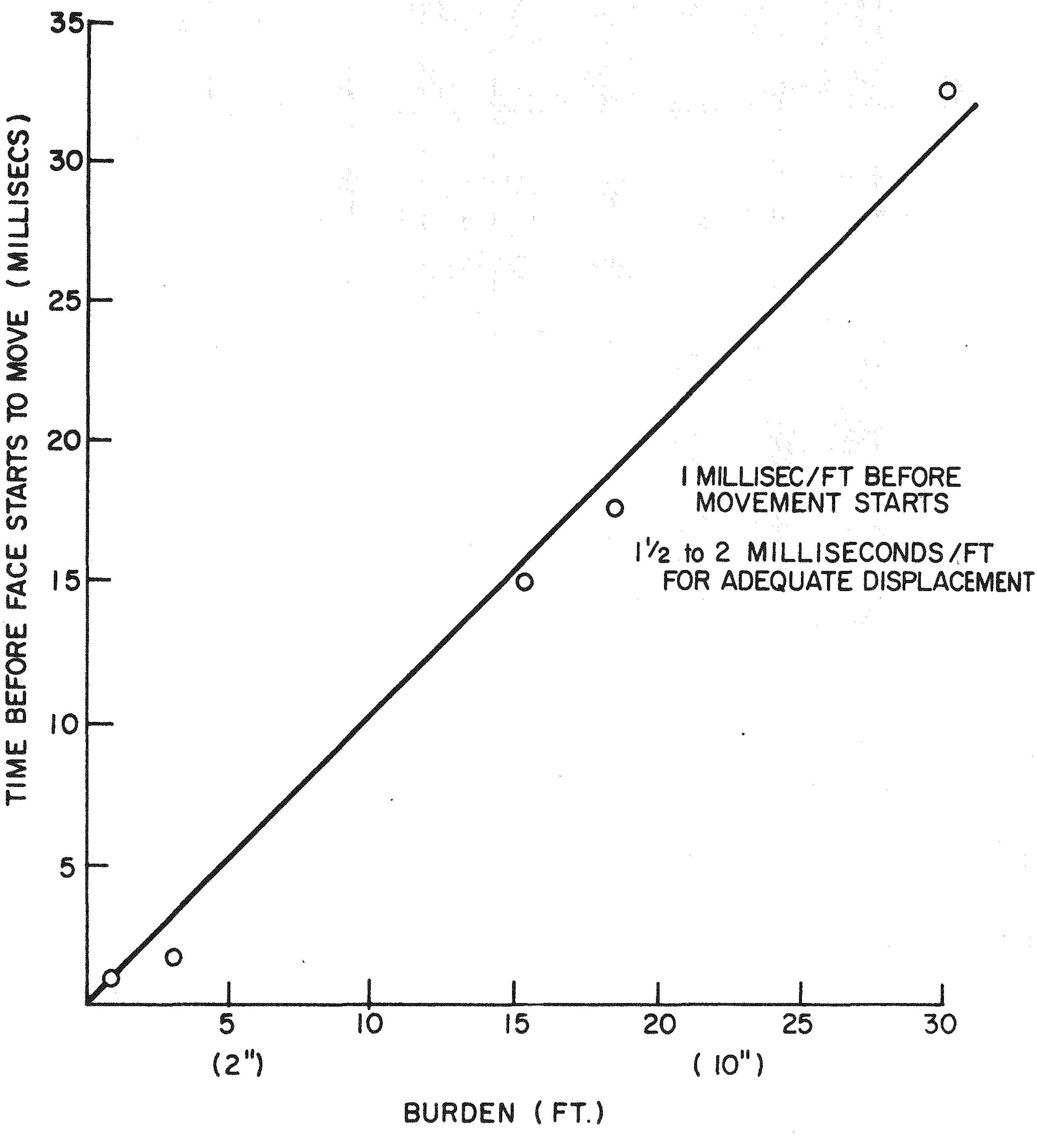
Job. No. 19089

TABLE V

STRENGTH PROPERTIES OF AN/FO COMPARED TO VARIOUS ALUMINIZED AN/FO'S AND SOME SLURRIES

EXPLOSIVE	DENSITY (gm/cc)	WEIGHT STRENGTH RELATIVE TO AN/FO	POWDER FACTOR RELATIVE TO AN/FO POWDER FACTOR	BULK STRENGTH RELATIVE TO AN/FO AT A DENSITY OF 0.85 gm/cc	TOE PULLING CAPABILITY RELATIVE TO AN/FO (relative bulk strength)$^{1/3}$	COST/LB. $	EXPLOSIVES COST/YDS^3 at an EQUIVALENT P.F.=.7#/YD^3 (ANFO) $
AN/FO	0.85	1.0	1.0	1.0	1.0	.100	.070
AN/FO/Al (89/4/7)	0.90	- (1.17)*	- (0.85)*	- (1.24)*	- (1.07)*	.142	.085
AN/FO/Al (87/3/10)	0.92	1.33(1.24)*	0.75(0.81)*	1.44(1.34)*	1.13 (1.10)*	.160	.090
AN/FO/Al (85/3/12)	0.92	1.38(1.29)*	0.72(0.78)*	1.49(1.40)*	1.14 (1.11)*	.172	.093
AN/FO/Al (82/3/15)	0.92	1.45(1.35)*	0.69(0.74)*	1.57(1.46)*	1.16 (1.13)*	.190	.099
TOVEX-20E	1.15	0.86	1.16	1.16	1.05	.170	.138
TOVEX-30E	1.15	0.84	1.19	1.13	1.04	.160	.133
NBL - 351	1.15	0.90	1.11	1.20	1.06	.200	.155
TOVEX A2E	1.15	1.12	0.89	1.51	1.15	.235	.147

* A. Bauer Values, The Others are Trade Values



ONSET OF MOVEMENT FOR BURDENS OF DIFFERENT SIZE IN ROCK BLASTING (NOMINAL HOLE SIZES)

rial due to the lack of horizontal relief. Figure 16 illustrates the minimum time in bench blasts before various sizes of burden start to move. These values were determined from high speed movies of blasts with holes ranging from 1-1/2 to 12-1/4 inches in diameter (Figure 17). It takes approximately 1 millisecond per foot of burden before the burden starts to move. To ensure adequate movement, a delay of 1-1/2 to 2 milliseconds per foot of burden would be appropriate in most instances. Even longer delay intervals could be advantageous under some circumstances, such as very deep blasts, if cut-offs can be avoided.

Another method, which has been used in the determination of the maximum delay period possible when blasting with detonating cord and surface delays, is more theoretical and is presented below and in Figure 18.

$$V_{up} = (V_{measured}) \cos \gamma$$

For a plane wave at a rock air boundary, the incident particle velocity, is one-half the measured fly-rock velocity,

$$W_i = \frac{V_{measured}}{2}$$

$$\therefore V_{measured} = 2 W_i$$

$$W_i = C_p \varepsilon$$

where C_p is the transmission velocity of compressional waves and ε is the strain.

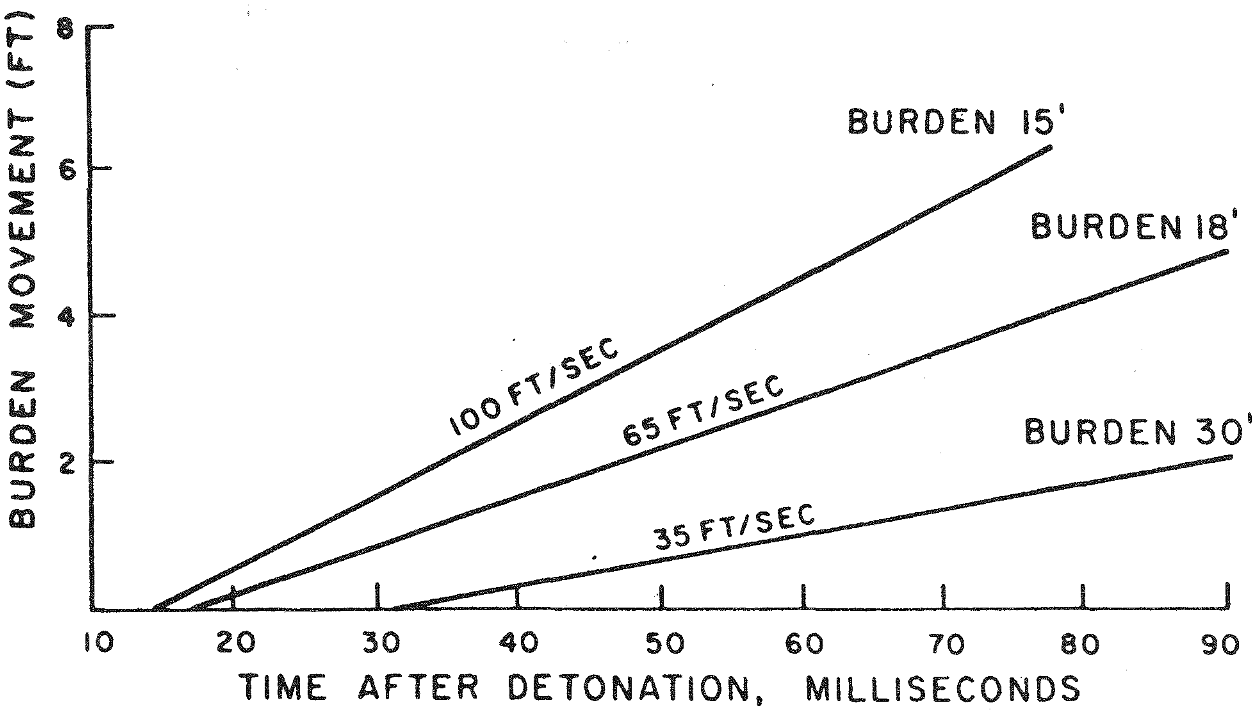
$$V_{measured} = 2 C_p \varepsilon$$

$$V_{up} = 2 C_p \varepsilon \cos \gamma$$

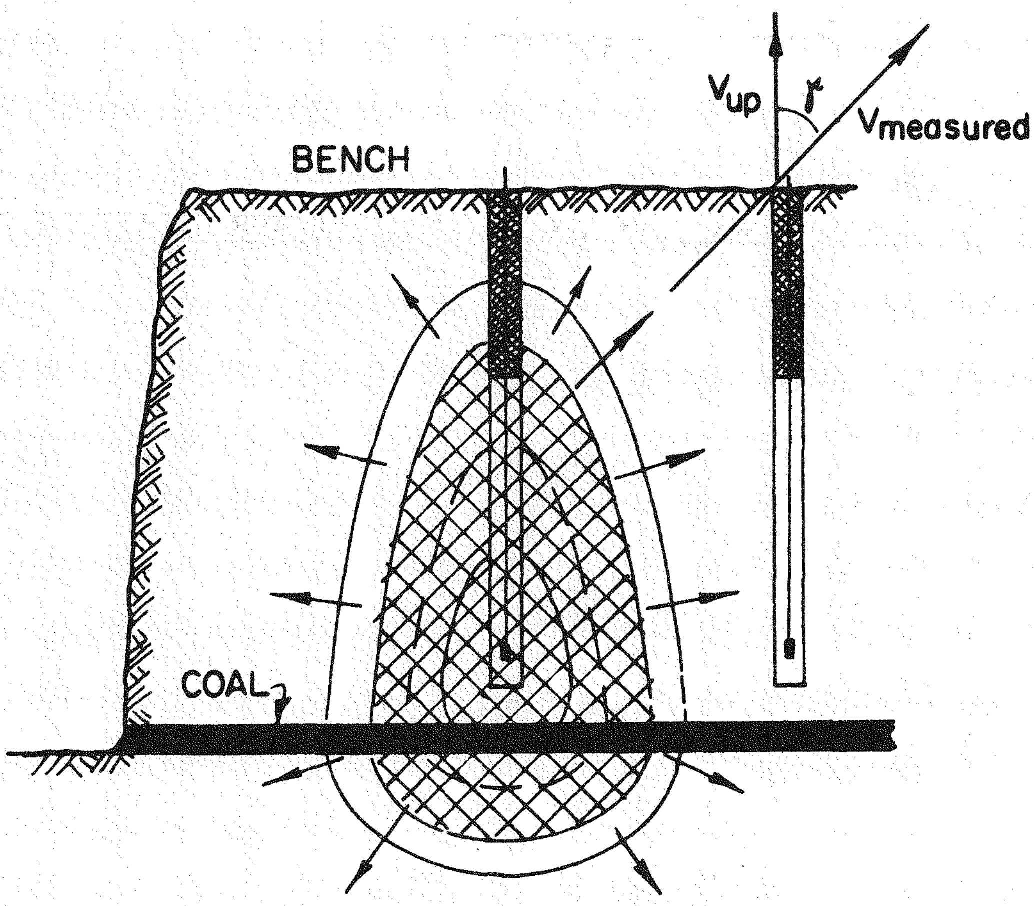
From U.S.B.M. work (20)

$$\varepsilon = \frac{ke^{-\alpha R/W^{1/3}}}{R/W^{1/3}}$$

for spherical charges where ε is the radial strain at distance R.



BURDEN MOVEMENT IN LARGE HOLE BLASTING (9 7/8") IN HARD ROCK



V measured would be a measured particle velocity at the collar of a hole designed to be fired on the next delay period. It is normal to the stress wave front. The vertical velocity is V_{up}.

CALCULATION OF V_{up}

$$V_{up} = \frac{2\ C_p\ \cos\gamma\ k e^{-\alpha\ R/W^{1/3}}}{R/W^{1/3}}$$

$$e^{-\alpha\ R/W^{1/3}} = 1 - \left(\frac{\alpha R}{W^{1/3}}\right) + \left(\frac{\alpha\ R}{W^{1/3}}\right)^2 \frac{1}{2!} - \left(\frac{\alpha\ R}{W^{1/3}}\right)^3 \frac{1}{3!}$$

The last two terms may be neglected as they are small, therefore

$$V_{up} = 2\ C_p\ \gamma\ \ k\ \left(1 - \frac{\alpha R}{W^{1/3}}\right) \frac{1}{R/W^{1/3}}$$

$$= 2\ C_p\ \cos\gamma\ k\ \left(\frac{W^{1/3}}{R} - \alpha\right)$$

From the U.S.B.M. work α = 0.03 approx.

$$V_{up} = 2\ C_p\ \cos\gamma\ k\ \left(\frac{W^{1/3}}{R} - 0.03\right)$$

Where $\quad 2\ C_p k = k^1$ Constant

$$V_{up} = k^1\ \cos\ \gamma\ \left(\frac{W^{1/3}}{R} - 0.03\right)$$

Hence if k and C_p are known, then k^1 can be calculated. Hence V_{up} can be calculated for different collars and spacings of holes if it is assumed that the top 6 to 8 diameters of the charge acts like a sphere.

If t_o is the time it takes the top to start to move then the distance moved at time t is: $V_{up}\ (t - t_o)$.

In order to determine precisely the maximum delay time possible, it is necessary to consider the maximum elongation than can be tolerated in the detonating cord prior to tensile failure. According to detonating cord suppliers, most varieties can take an elongation of 2.5 percent before failure occurs. It is normally assumed that the effective cord length between holes is 50 feet and therefore the maximum delay time is:

$$t_D = \frac{(50)\ (.025)}{V_{up}} + t_o$$

The uncertainties in the k values and the lengths of detonating cord to be stretched make it preferable to use a direct photographic method. This procedure will be discussed later in this report.

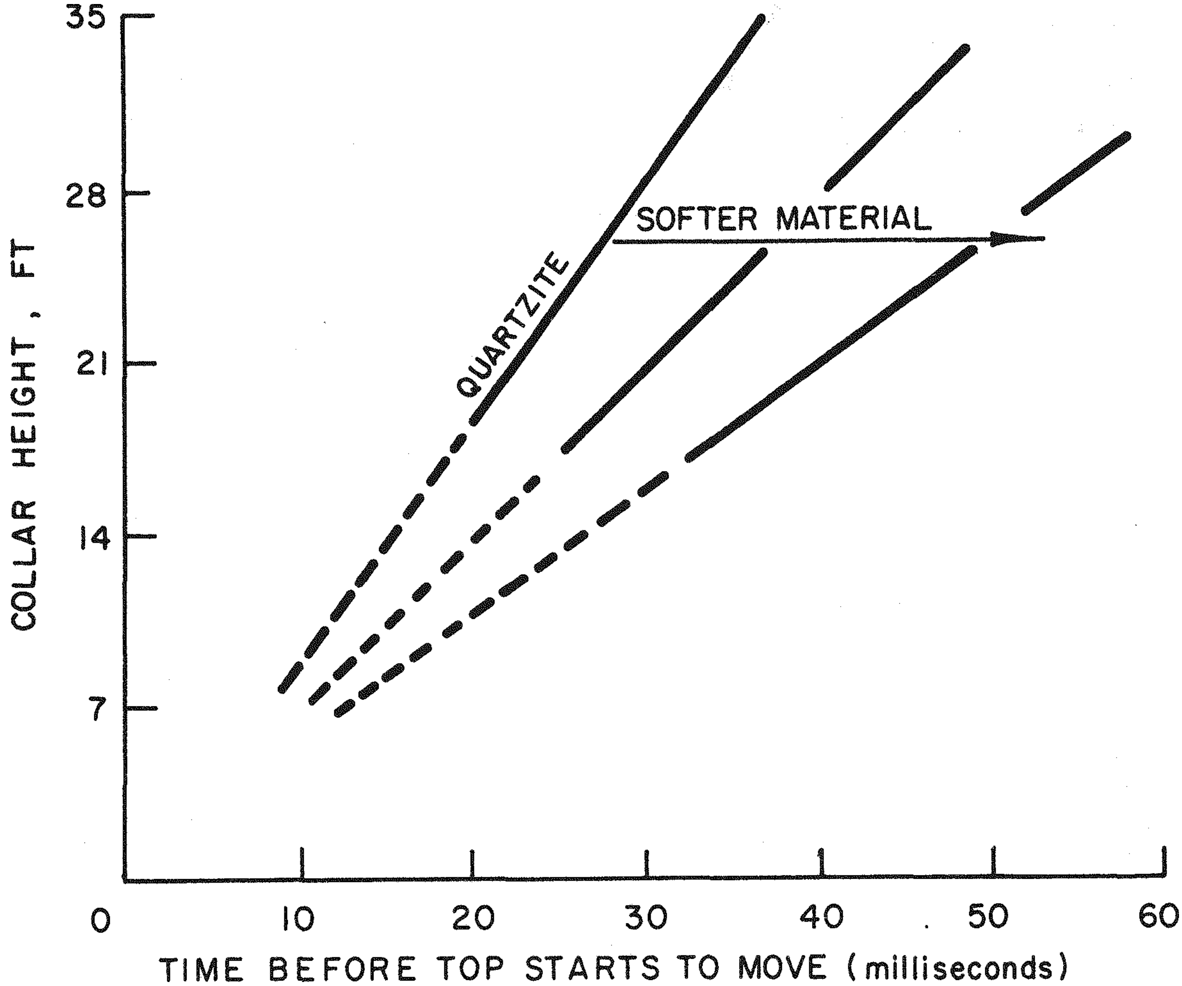
In practice it is more important to determine t_o than V_{up}, since failure criteria are highly variable within a given rock mass. At best, t_o can be estimated only roughly because actual firing times vary significantly from nominal firing times.

2.7 Collar Height

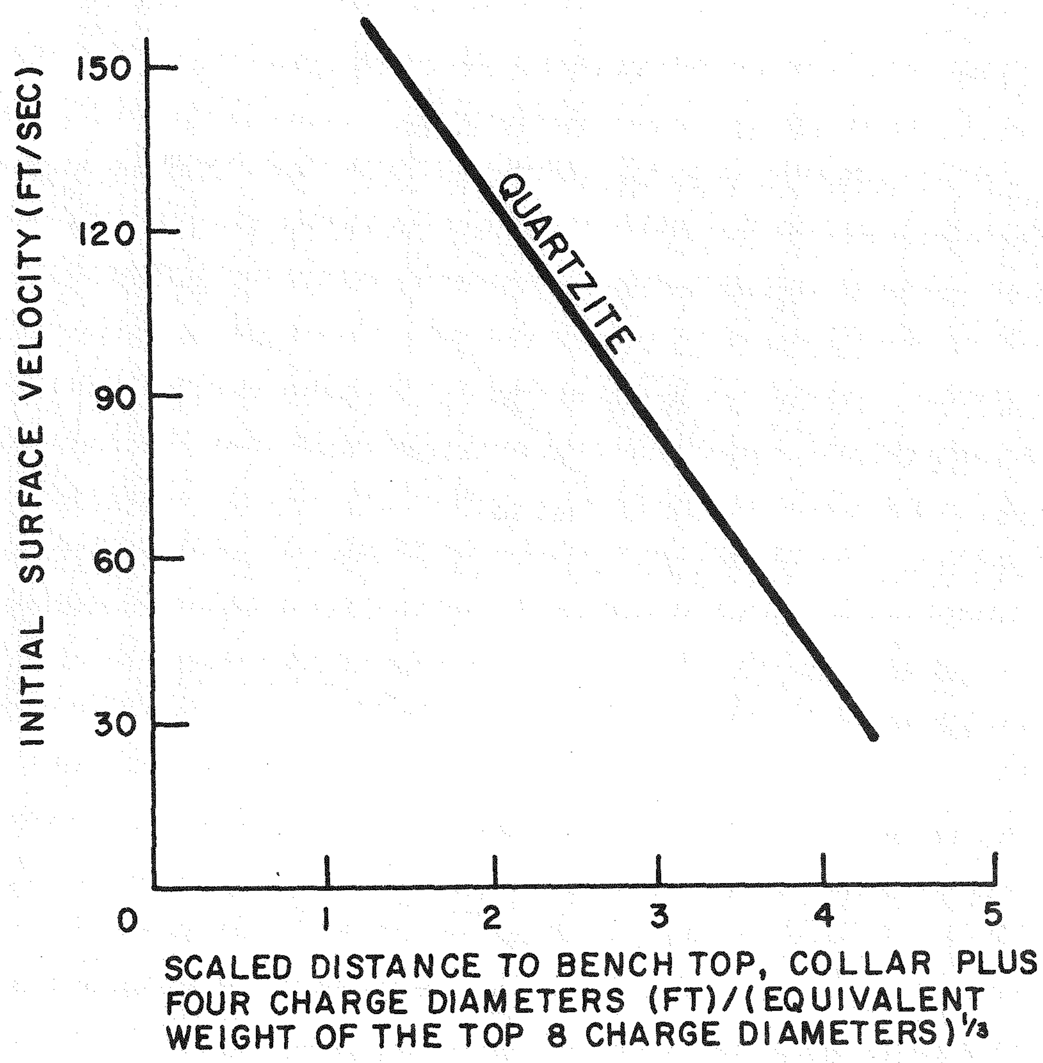
In most blast designs the delay required to produce burden movement must be balanced against the possibility of producing "cut-offs" at the next row of holes due to ground movement occurring prior to firing. To do this, knowledge of delay times prior to top movement for different sizes of collar is useful.

Figure 19 illustrates the time it takes for top movement to occur as a function of the collar height, and Figure 20 illustrates the initial surface velocity as a function of scaled burial depth of the top elements of the charge. As far as the initial surface velocity is concerned, the top of the cylindrical charge acts similarly to a sphere equivalent in weight to the top 8 charge diameters are located 4 diameters into the charge.

The delay prior to surface motion and the initial surface velocity can be useful in calculating the maximum delay time that can be tolerated prior to "cut-off" of the detonating cord from tensile stretching. In such calculations it is customary to consider (conservatively) the detonating cord as being free (not frozen in stemming) and having the ability to stretch 2.5 percent over an assumed 50-foot length prior to failure. As a rule, to produce good top fragmentation without



DELAY TIME PRIOR TO BENCH TOP MOTION FOR HOLES OF DIFFERENT DIAMETER.



INITIAL SURFACE VELOCITY AT DIFFERENT SCALED BURIAL DEPTHS IN QUARTZITE.

excessive throw, the collar height in a massive, blocky, brittle rock can be estimated from the scaled depth of burial of the top portion of the charge equal to 2.8. For softer formations, this would increase to values approaching 4.0 as shown in Figure 21. The same situation would also be true for thinly-laminated prefractured rocks. Therefore, when using larger diameter blast holes, stemming heights must be increased in order to meet blast requirements. Also, where cut-offs are a serious problem, in-hole delays may be an appropriate solution.

2.8 Bottom Charges

Figure 5 illustrates that the action of the charge can be divided into three general regions: (1) near the toe, (2) in the body of the charge and (3) the top of the charge near the collar region. The expansion at both the bottom and top of the charge is hemispherical, whereas it is approximately cylindrical in the central part. Therefore, the toe-pulling capability of any other explosive relative to AN/FO in the same diameter of blast hole can be estimated from the ratio:

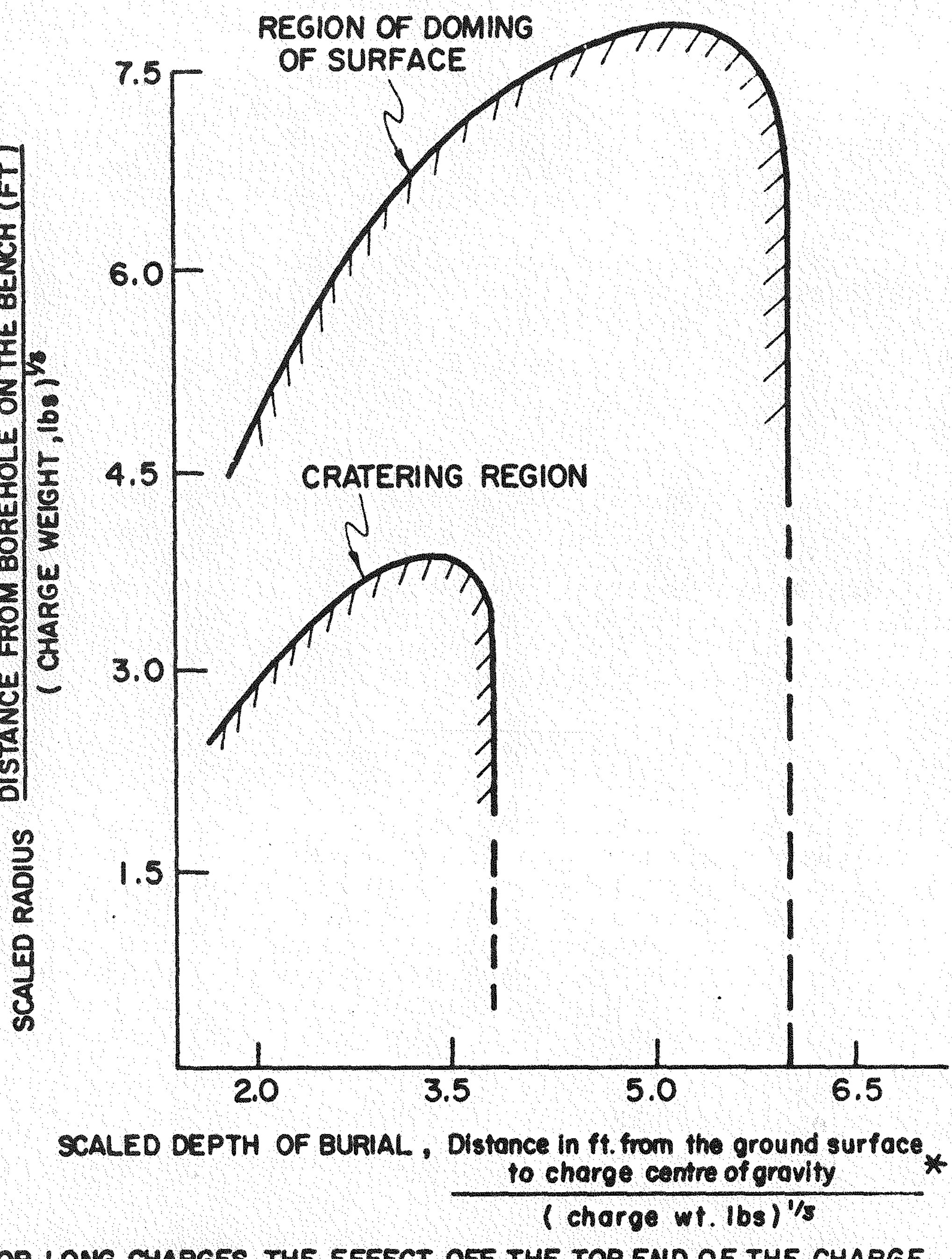
$$\frac{\text{Bulk Strength of the Explosive}}{\text{Bulk Strength of AN/FO}}^{1/3}$$

These ratios are listed in Table V. For example Tovex 20E is 5 percent stronger than AN/FO whereas a 10 percent Al/AN/FO is 10 percent stronger. Stemming heights would also be scaled in a similar manner. If 20 feet is satisfactory for AN/FO then 21 and 22 feet would give similar results if Tovex 20E or 10 percent Al/AN/FO, respectively, were used as the charge. In coal stripping, the drilling cost is normally so low that AN/FO use predominates. The economics are such that slurry use is usually precluded.

2.9 Priming

2.9.1 Primer Types and Sizes

In the past several years there has been considerable controversy over the requirements of a primer or booster for blasting agents. The role of the primer is to detonate the explosive quickly. The rate of reaction in the explosive is exponentially dependent on the temperature generated by the primer. Those primers with the highest detonation pressures generate the highest compression in the blasting agent and therefore the highest temperatures. It is quite logical, therefore, to assume that those explosives having the highest detonation pressures would be the most effective as primers. The detonation pressure, P_2, can be shown to be equal to P D W



* FOR LONG CHARGES THE EFFECT OFF THE TOP END OF THE CHARGE CAN BE APPROXIMATED BY THE WEIGHT OF THE TOP 8 DIAMETERS OF CHARGE LOCATED 4 CHARGE DIAMETERS FROM THE TOP.

SCALED CRATERING RESULTS IN SANDSTONE (LIVINGSTON) where P is the explosive density

D is the velocity of detonation

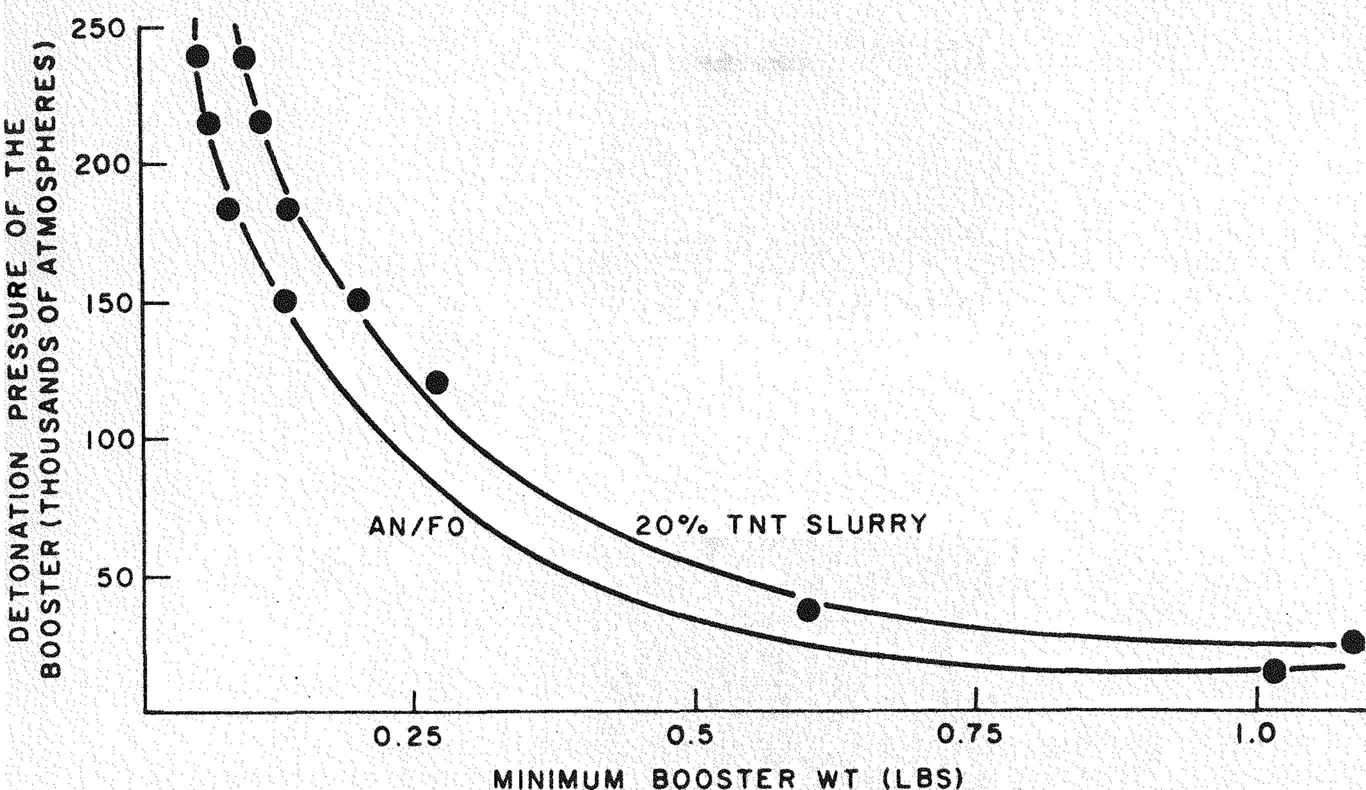
W is the particle velocity in the detonation wave and is approximately $D/4$

Cook [6] has produced evidence to support the view that explosives with the highest velocities of detonation and density are the most effective primers. Figure 22 illustrates how the minimum primer weight is strongly dependent on the detonation pressure of the primer. Therefore, primers with low detonation pressures require larger weights to prime AN/FO than those with higher detonation pressures. It is also probable that for individual explosives there will be a certain detonating pressure level below which the primer will be ineffective, and one above which the primer size will have little effect.

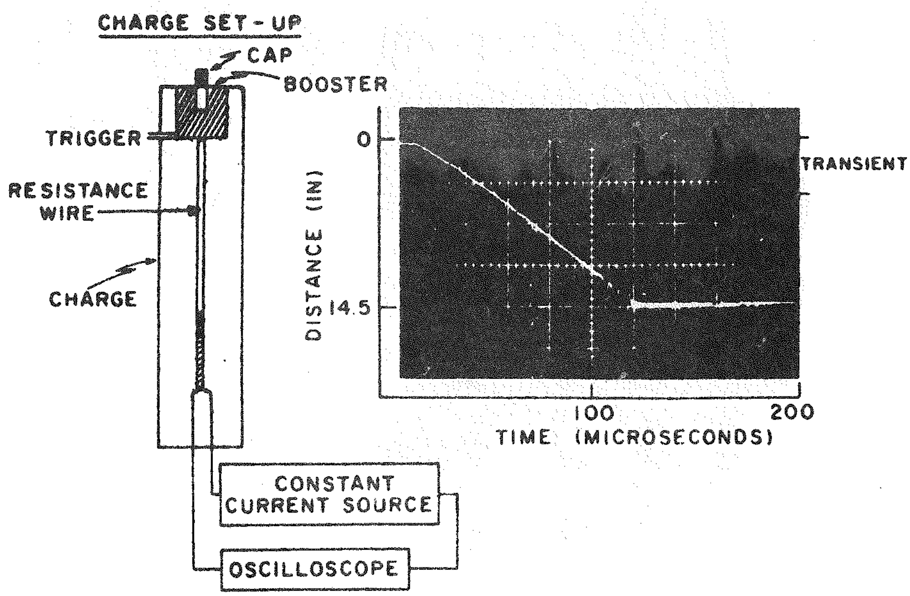
Figure 23 illustrates the experimental set-up used for measuring continuous velocities of detonation along the charge central axis, using a constant current source, a uniform resistance probe down the charge and an oscilloscope. Slopes on the voltage or distance-versus-time trace give the velocity of detonation. Typical traces for small diameter charges of AN/FO are also given. It can be seen that the initial velocity of detonation in no instance is very low. The transient velocity may change smoothly or abruptly and the final velocity is independent of the primer weight if the explosive detonates at full order.

Figure 24 shows results for crushed high- density prills/FO fired in schedule 40 Al pipe. With 0.25 lb of pentolite as the primer, the transient distance was approximately 8 inches. This dropped to almost zero with a 1 lb pentolite primer. Larger charge diameters gave similar results.

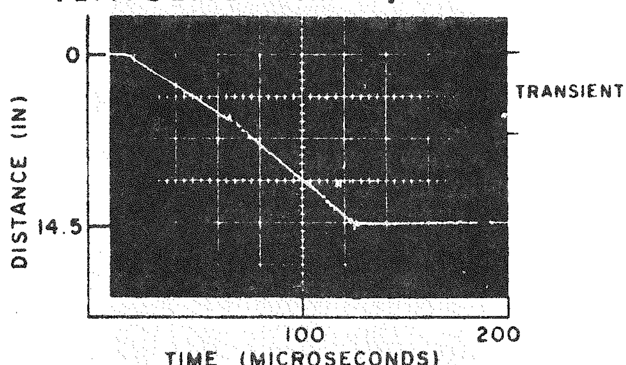
For uncrushed porous prills/FO in large diameters (10 to 12 inches), the run up distance with 1 lb pentolite primers was in the range of 18 inches, with initial velocities of approximately 8000 ft/sec. For 4-inch diameter charges this became 3 to 6 inches with 1 lb pentolite primers, 6 to 9 inches with 80-gm primers and failures with 40 gm primers. For the large-diameter charges with smaller primer weights, the run-up or transient distances increased to 18 to 24 inches.



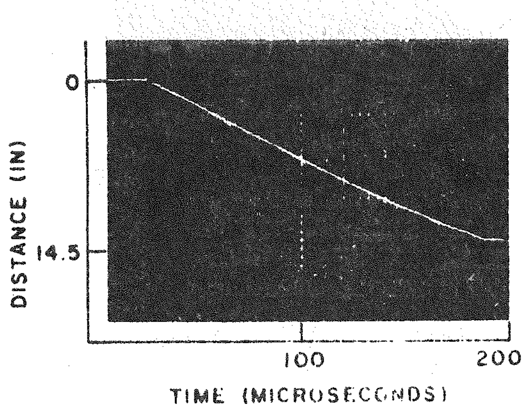
MINIMUM BOOSTER WEIGHT AS A FUNCTION OF
BOOSTER DETONATION PRESSURE (AFTER COOK)



FOUR INCH DIAMETER AN/FO, 450 gm.
PENTOLITE PRIMER.

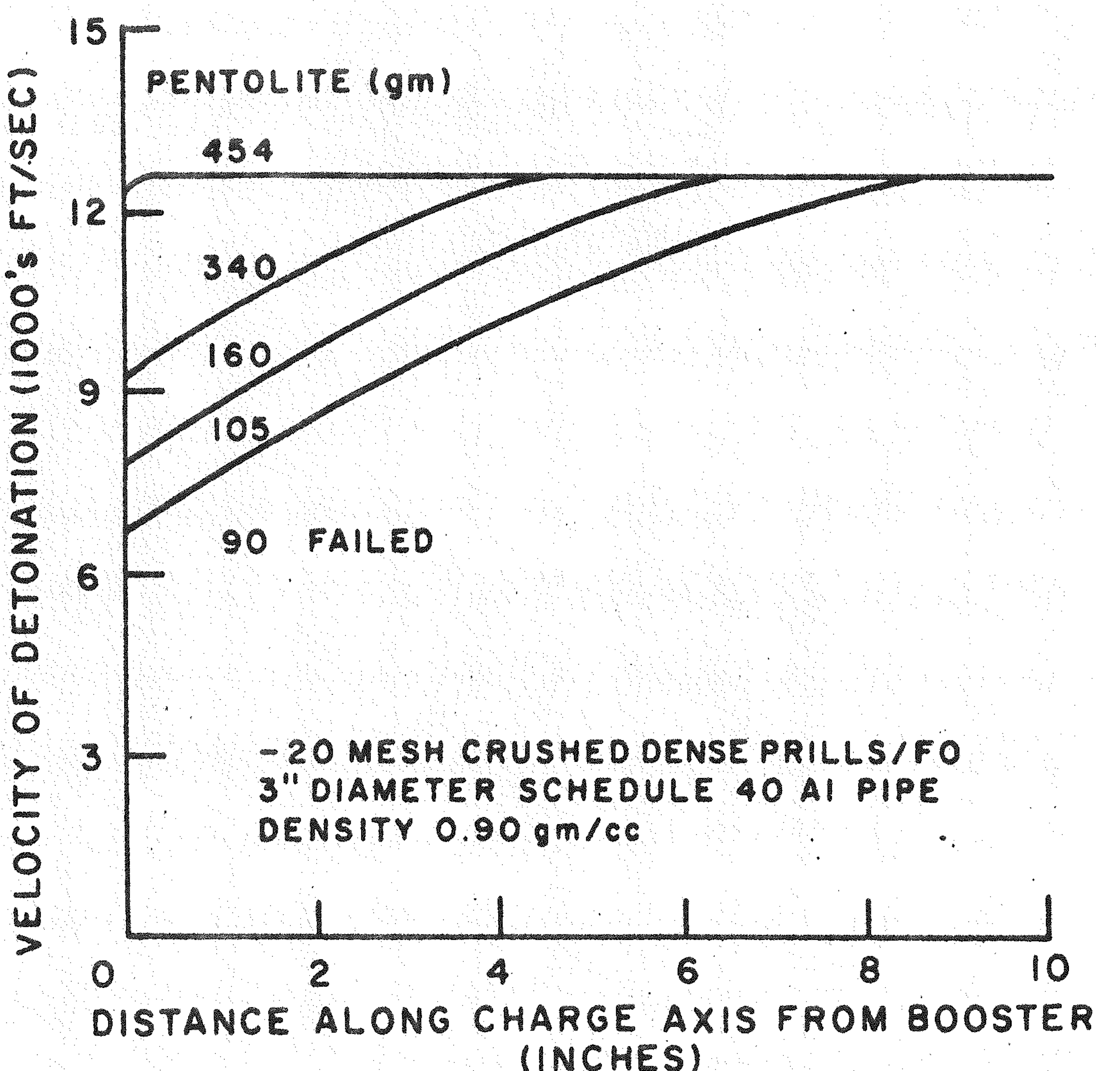


FOUR INCH DIAMETER AN/FO, 80gm.
PENTOLITE PRIMER.



FOUR INCH DIAMETER AN/FO, 40 gm.
PENTOLITE PRIMER, CHARGE FAILED.

EXPERIMENTAL SET-UP FOR MEASURING DETONATION VELOCITIES.



VELOCITY RUN-UP DISTANCES FOR -20 MESH CRUSHED DENSE PRILLS/FO FIRED IN 3-INCH DIAMETER PIPES WITH DIFFERENT BOOSTER WEIGHTS.

Therefore, when priming AN/FO with cast 1 lb pentolite primers, the run-up distances, when they occur, are small and are not a cause for concern. In large diameters, they vary from 6 to 18 inches and the total energy lost in this regard would be negligible, since the average velocity is high and the prill particles continue to react after the passage of the detonation wave front, supplying additional energy for the blasting process. The run-up distance is also a function of the primer weight, diameter of the charge, and particle size and density of the AN. It is customary, therefore, in testing to determine the minimum primer weight at a charge diameter 1-inch larger than the minimum initiating diameter. This is normally scaled by a factor of about 6 for the field size of primer to be used in large holes.

The more recent school of thought on priming has been concerned with the transient velocity of detonation up to steady state velocity. The contention is that detonation velocity (which is related to detonation pressure) will directly relate to the useful work done by an explosive. Recent tests by the Bureau of Mines [22] and by Rydlund [7] with various types and weights of primers in AN/FO loaded in 10 5/8-inch diameter drillholes indicated the following:

a) One pound cast primers resulted in the AN/FO detonation at about 65 percent of steady state for the first few feet.

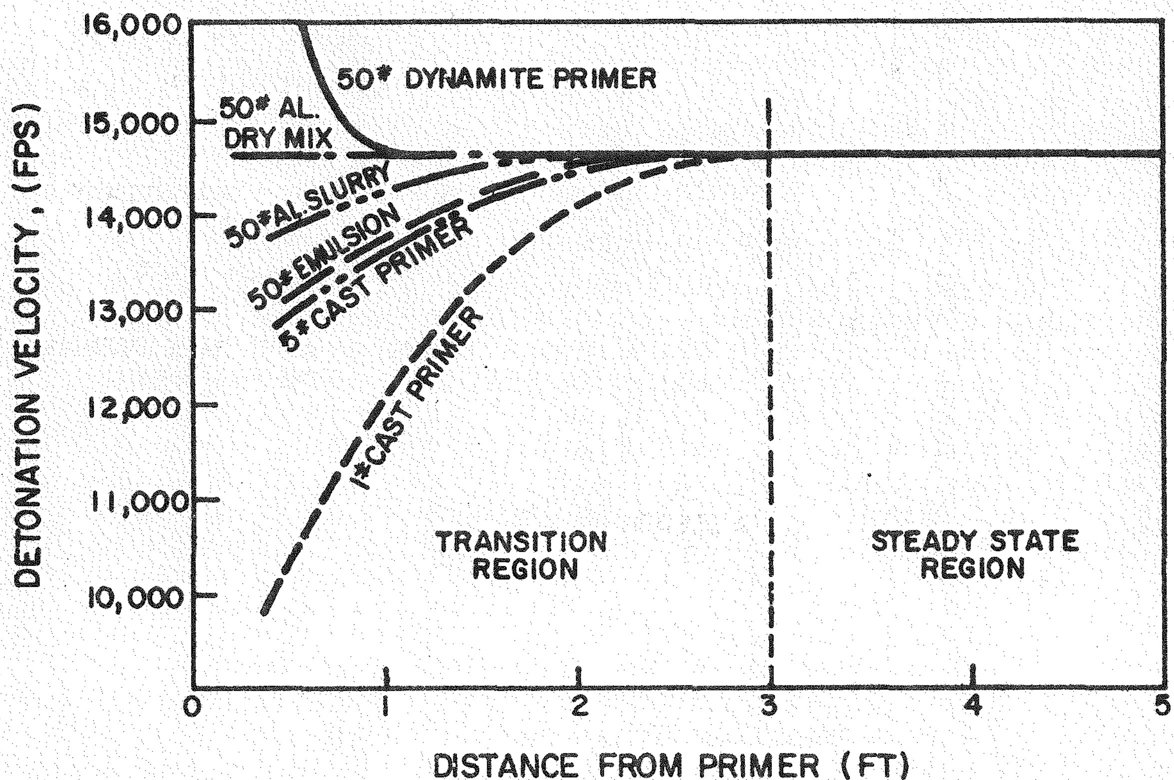
b) Five-pound cast primers resulted in an AN/FO detonation starting out at about 85 percent of steady state.

c) Fifty-pound aluminized water gels started out at near 95 percent of steady state.

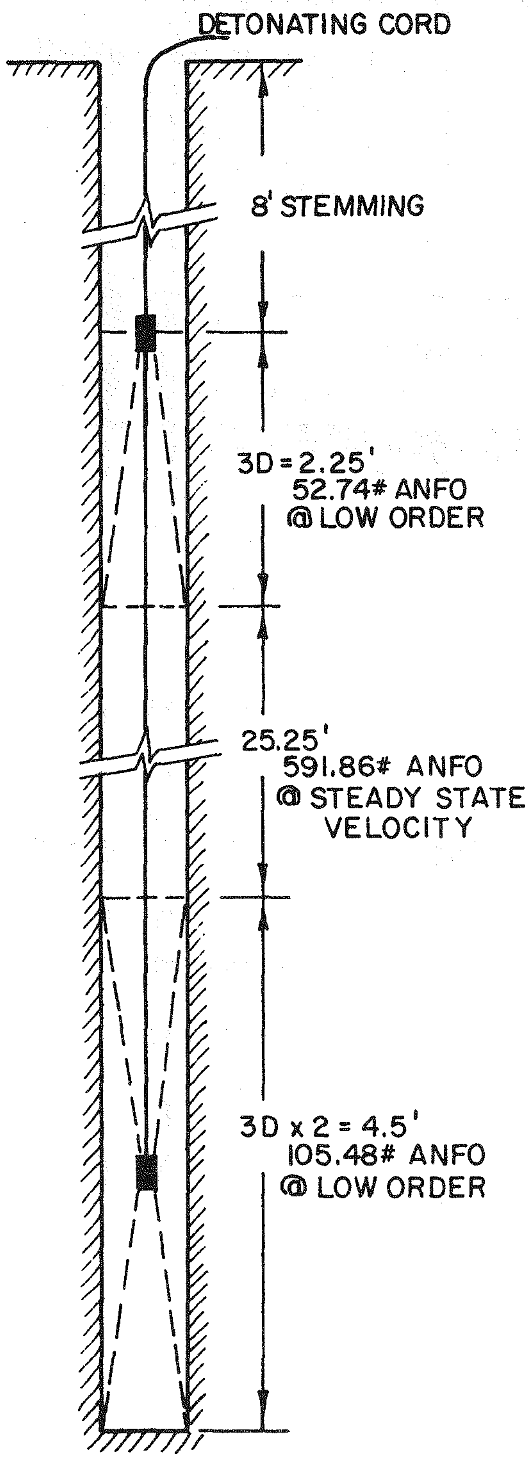
d) Fifty-pound aluminized dry mixes (15-25 percent Al) resulted in a full steady state rate immediately. These and the aluminized water gel were primed with one-pound cast primers.

e) Fifty pounds of NG gelatin of 75 percent strength or greater will overdrive the AN/FO for the first foot. However, this overdrive effect diminishes rapidly.

The results of these tests are shown in Figure 25. Figure 26[8] translates the practical implications of this concept to that of a typical hole loaded with AN/FO and primed top and bottom with one-pound cast primers. In this case, 79 percent of the AN/FO detonates at full steady state velocity



DETONATION VELOCITIES IN
10-5/8 INCH BLASTHOLE



DIAMETER OF HOLE = 9"
DEPTH OF HOLE = 40'
STEMMING = 8'

ANFO IN 9" DIA. HOLE
@ 0.85 DENSITY = 23.44 lbs./FT.

32 x 23.44 = 750# TOTAL ANFO

158.22# @ LESS THAN STEADY STATE VELOCITY

591.78# @ STEADY STATE VELOCITY

78.9 % OF ANFO IS DELIVERING OPTIMUM ENERGY.

PERCENTAGE OF AN/FO AFFECTED BY
TRANSIENT V.O.D. IN TYPICAL BLASTHOLE

and the remainder at 85 percent of steady state. Note that as the column charge length increases or decreases, this effect will change accordingly.

The two schools of thought on priming conflict. One group claims any energy lost in this regard would be negligible, since the average velocity is high (e.g. borehole pressures remain sufficiently large to affect fragmentation), and the prill particles continue to react following passage of the detonation wave front, thus supplying additional energy for the blasting process (e.g. rate of work is reduced but total work done remains fairly constant). The second group contends that the reduction of detonation velocity over the transient length results in a corresponding reduction in fragmentation at that location. Until this conflict is resolved, both methods appear acceptable.

2.9.2 Position of the Primer

If the explosive is sensitive to detonating cord, top priming is essential because detonating cord running through a sensitive mixture will cause deflagration and poor blasting performance. Ground or crushed AN/FO, hot NCN slurry or NCN slurry in warm boreholes are all sensitive to detonating cord. The sensitivity of NCN slurries decreases in long boreholes due to "deadpacking", that is, the development of excessive pressures due to gravity.

Discounting the effects of detonating cord sensitivity, little field evidence exists that shows differences in results from top or bottom-primed blasts. One current opinion is that bottom priming results in better breakage in the toe area and that it is good practice to prime in the zone of rock most difficult to break. On the other hand, from stress versus time records in the literature [9,10], the stress levels in a borehole are considerably higher at the opposite end of the charge from the primer. In this case, top priming would be preferred. It is not possible to conclude at this time which of these factors is of greatest importance. At present the general basis for top or bottom priming is the sensitivity of the explosive column.

If many boosters are being used in a hole, a problem may exist. Either the explosive is beneath its critical diameter and not propagating properly or the boosters are of inadequate size. If the charge consists of a combination of slurry and AN/FO in a hole, then both should be primed because AN/FO may not initiate some of the slurries. If time has elapsed between the slurry and AN/FO loading, dirt can fall in the hole. A 1-inch layer of dirt will stop the propagation of detonation from the slurry to the AN/FO. Water could also get into the AN/FO from the top of the slurry and this would desensitize a small region next to the slurry. Wet holes using packaged AN/FO require additional primers to protect against faulty wet-hole bags and possible separation due to unforeseen hang-ups in the hole.[8]

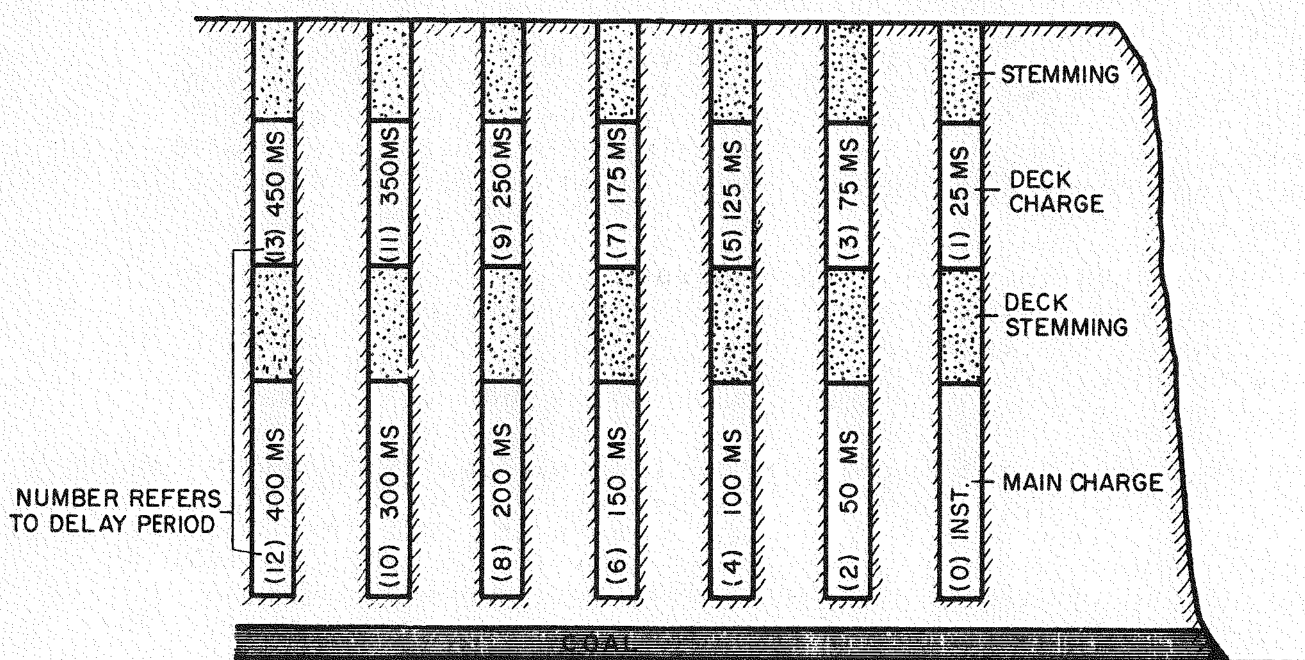
Extensive tests have shown that the AN/FO detonation rate is constant after the steady state is attained, assuming there are no adverse hole conditions. After two or three feet in large diameter holes (12-1/4 to 15 inches), the AN/FO detonation rate does not build up or slow down. Therefore it is unnecessary to add extra primers or boosters up the column of bulk ANFO to keep it detonating at steady state. Where explosive sensitivity to detonating cord is not a factor, it is common practice to prime top and bottom to safeguard the drilling and blasting investment in the hole. Where detonating cord sensitivity is a problem, electric blasting caps or low-energy detonating cord may be used.

Where deck charges are employed, each deck requires a separate primer. The primers may be strung on separate or common downlines depending on local experience with misfires. If the decks are delayed for vibration control, it is recommended that initiation be from bottom to top to avoid weakening the upper part of the hole before the lower explosive column is generating its full velocity. Upper fracturing creates an escape route for the gases which should remain confined to accomplish the work of breaking and fragmenting. Normally, bottom-hole detonation is desirable, but the criterion of a successful shot is how the bottom pulls. This can best be accomplished by maintaining complete confinement of the gases as long as possible. Figure 27 shows a typical bottom-delaying arrangement for a set of blastholes with deck charges. The only acceptable method for delaying deck charges from top to bottom would be where each deck charge was designed by cratering principles with regards to burden and collar height.

2.10 Blast Vibration and Noise

2.10.1 Types of Blast Vibration Waves

When an explosive is detonated in a blasthole, high gas and shock pressures are developed. Following the crushing of the material immediately surrounding the borehole, the pressure quickly decreases, with distance from the charge, to values below the compressive strength of the confining medium. At this point the remaining energy travels into unbroken material in the form of a pressure wave, traveling at approxi-



TYPICAL BOTTOM DELAY ARRANGEMENT
FOR A SET OF BLASTHOLES WITH DECK CHARGES

Job No. 19089 Fig. 27

mately the speed of sound in the surrounding material without breaking it in compression.[11] This wave causes tensile radial or hoop stresses to be formed in the rock surrounding the borehole, which may cause tensile failure if the tensile strength of the rock is exceeded. In addition, a tensile wave is formed when the compressive wave is reflected at media boundaries. This wave can also cause tensile failure.

This strain pulse attenuates into an oscillatory wave in which the ground particles move along cyclically repeating orbits. From this stage on, the energy radiating from the explosion produces particle movements in the surrounding rock which are within the rock's elastic limits. Therefore, the material completely recovers to its original shape and volume after the energy has passed. These waves are called elastic waves.

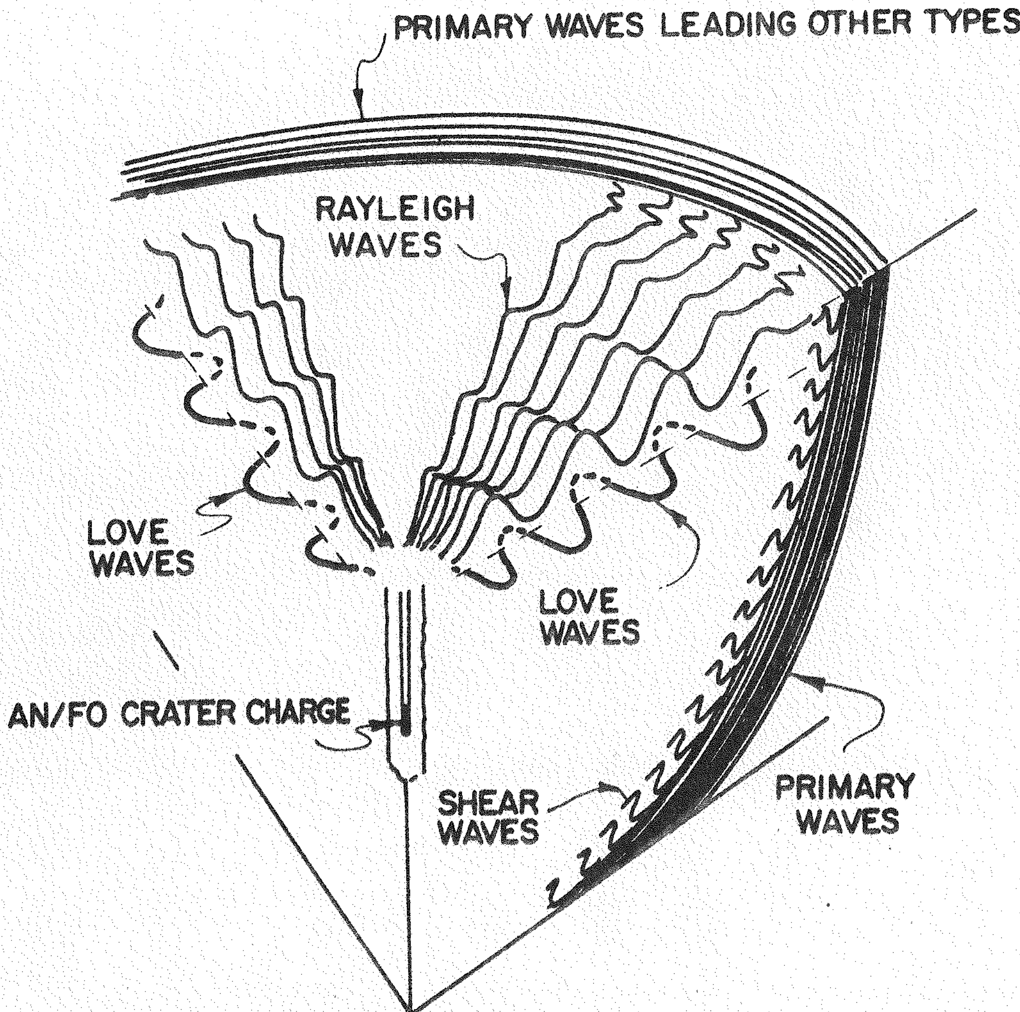
Figure 28 is a schematic representation of a single cratering charge a few milliseconds following detonation. In this figure, the principal surface waves have been indicated, these being the Rayleigh and Love waves, and all amplitudes have been greatly exaggerated.

Scaling relationships are modified when delayed blasts are to be considered. The explosive weight should, ideally, be the explosive weight per instant of time. This is usually taken as the explosive weight fired on a single delay period. The subject of delays and their relation to blast vibration levels is discussed in more detail in a later section.

2.10.2 Vibration Effects at Relatively Large Distances

Vibration from blasting rock is a real problem to mine operators and contractors in populated areas, both as a cause of potential structural damage and as an annoyance to local residents. Generally, two approaches to minimizing the problem are utilized. First, attempts have been made to predict vibration levels from blasts and to correlate these with damage or annoyance, thereby permitting the avoidance of conditions that may lead to problems. The second approach has been an investigation of methods whereby blast vibration levels may be reduced.

These problems have led to legislation concerning blasting in proximity to property such as dwellings or plants not owned by the mine or quarry operator. Damage criteria have been discussed by a number of authors. Extensive statistical reviews have been made of published vibration data to determine which of the blast induced parameters, such as particle displacement, velocity or acceleration best measures the dam-



SCHEMATIC PATTERN OF ELASTIC WAVES AROUND A SINGLE CRATER CHARGE FIRED IN OVERBURDEN.

aging effects of a vibrational wave on common structures. A few researchers have studied the responses of different types of structures to ground waves and air blast over-pressures.

The most common conclusion is that damage to typical residences can best be correlated with the peak particle velocity of the ground motion. Some investigators use the resultant of the three components of particle velocity for this correlation. Historically, several empirical methods have proved useful from a practical viewpoint, such as those developed by Morris (1953) (12) and Edwards (1960). (15)

2.10.3 Bureau of Mines Blast Vibration Studies

As a result of further vibration studies (13)(14) in 1971, the Bureau of Mines (Nicholls, et.al., 1971) (15) presented a re-evaluation of all of its past vibration records, which were considerable, to determine if peak particle velocity gave an acceptable correlation with vibration damage to residences. Figure 29 shows their data presentation on a plot of log displacement versus log frequency in which they statistically fitted the limit lines of the onset of plaster cracking, minor damage and, finally, major damage. Each of these limit lines were lines of constant particle velocity.

Figure 30 illustrates limit lines for the Bureau of Mines data with no emphasis on any particular component of motion. As a result of this study, Nicholls, et.al., recommend that for the worst possible case there will be no chance of exceeding 2 inch/sec if $d/W^{1/2}$ is kept greater than or equal to 50. This proves to be a handy rule of thumb. It fits quite well with the earlier results. However, it is the worst case and in many instances, as will be seen later, much smaller values can be employed.

Several studies have been completed with respect to the additive effects of vibrations from delayed blasts. Bauer (16)(17) has shown that, from field experience and simulation studies, additive effects can produce as much as twice the peak particle velocity for multiple delayed blasts as compared with the vibration from a single detonation of the maximum weight per delay. Bureau of Mines experience, though not definitive, indicates that no additive effects will occur with delays of 9 msec or greater. Bauer (16)(17), however, states that 15 msec should be classed as the minimum value of delay time because of the erratic firing times of shorter delays (25 percent of 7 msec delays shot instantaneously) and the fact that the particle velocity peak usually occurs sometime into the particle velocity time record. It should be recognized,

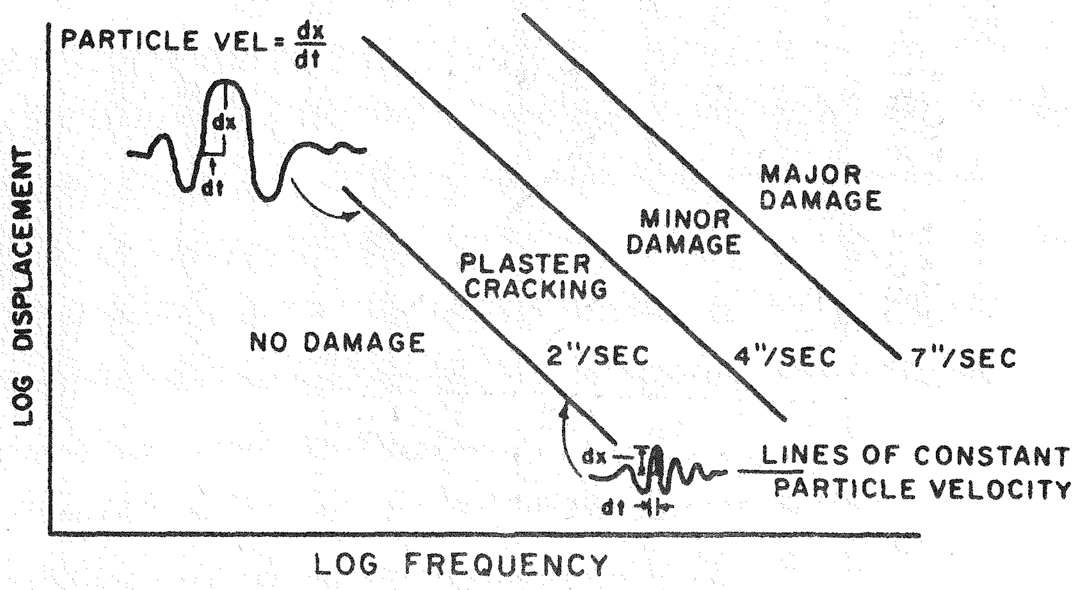
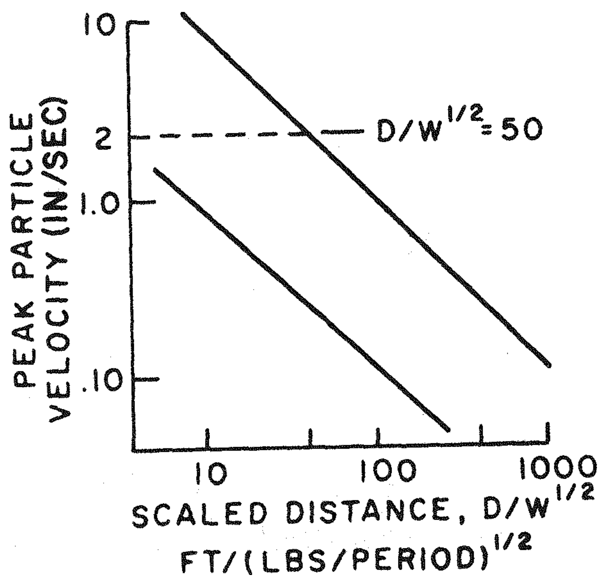


ILLUSTRATION OF THE U.S.B.M. FINDINGS



LIMIT LINES FOR U.S.B.M. PARTICLE
VELOCITY DATA FROM QUARRY BLASTS

however, that additive effects will be dependent on the site conditions and blasting parameters.

2.10.4 Large-Scale Open-Pit Blasts

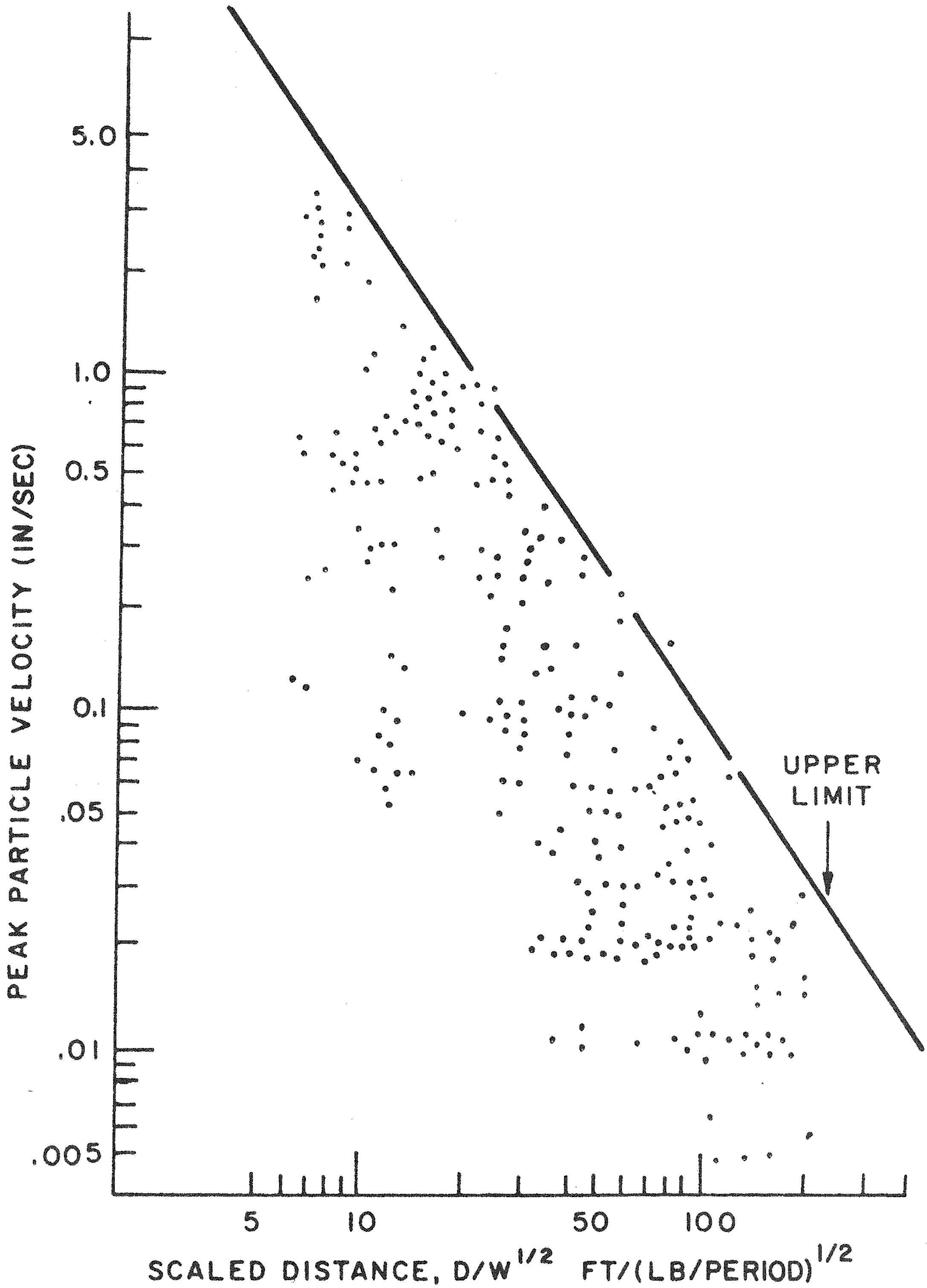
Figure 31 is a plot of peak particle velocity for large-scale open pit and strip-mine blasts. The maximum charge per delay period was 60,000 lb, and, in this instance, there were 6 delay periods. Any additive effects from multiple delay periods are included in this plot. The maximum total charge weight in a blast was in excess of 700,000 lb. Hole diameters ranged from 9 to 12-1/4 inches. No predominant direction produced a noticeably larger particle velocity. The figure is a representative sample of 1500 recordings from different mines, and the composite upper limit line is found to be extremely useful in laying out blasts in critical situations, if used in conjunction with observations on damage thresholds.

Figure 32 indicates values of peak particle velocity thresholds at which certain types of damage start to occur. In Figure 33 the peak particle velocity damage threshold values have been superimposed on the upper limit line of the pit blast peak particle velocity relationship. From these, useful guides may be obtained with regard to safe scaled distances. This line is conservative, since there is considerable scatter in the data even at a single site. The other scatter in the data is due to material variation and degree of water saturation which, when it increases, leads to higher values. The data scatter often makes it more convenient to work with such a composite plot until sufficient records are obtained at a property, rather than using sparse data which could lead to erroneous conclusions.

2.10.5 The Influence of Short Period Delays on Blast Vibrations

The use of short period delays can transform a large blast into a succession of smaller blasts of one or more holes each. However, some additive effects will sometimes be experienced which can increase the peak vibration levels above those for a single hole or above those for a given delay level except for the very long delays of 300-400 msec. This occurs because vibrations from even a single hole can have a duration of up to 250-300 msec for large charges in most materials.

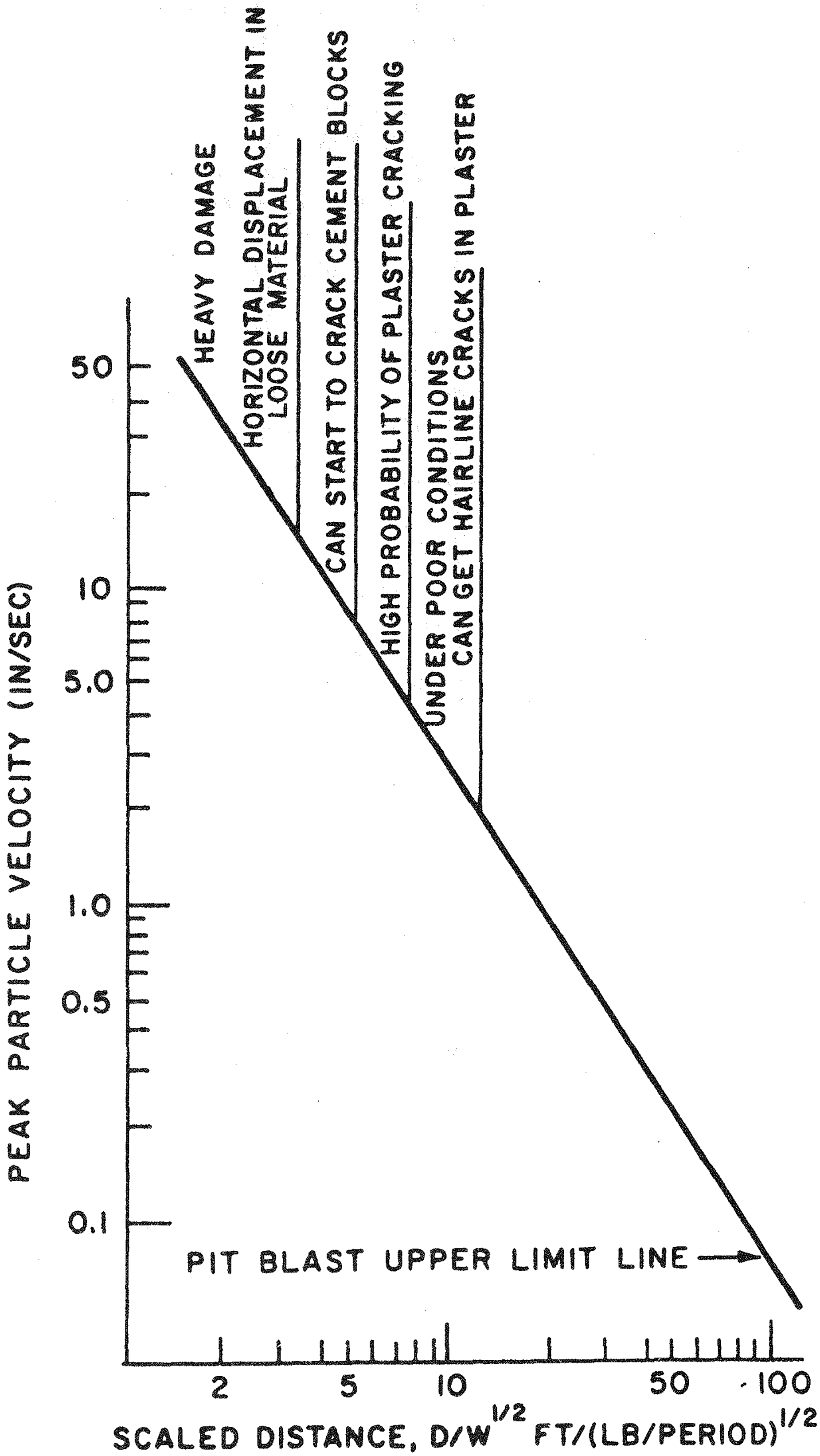
Morris and Westwater (1953)[12] found additive effects of up to 50 percent of the charge weight per delay for short period delays. This increased the peak particle displacements by up to 25 percent.



TYPICAL VIBRATION DATA FROM MULTIPERIOD
DELAY BLASTS IN OPEN PITS AND ROCK STRIP MINES

TYPE OF STRUCTURE	TYPE OF DAMAGE	PEAK PARTICLE VELOCITY THRESHOLD AT WHICH DAMAGE STARTS(IN/SEC)
RIGIDLY MOUNTED MERCURY SWITCHES	TRIP OUT	0.1 – 0.5 (HAS STRONG FREQUENCY DEPENDENCE)
HOUSES	PLASTER CRACKING	2
CONCRETE BLOCK AS IN A NEW HOUSE	CRACKS IN BLOCKS	8
CASED DRILL HOLES RETAINING WALLS, LOOSE GROUND	HORIZONTAL OFFSET	15
MECHANICAL EQUIPMENT - PUMPS, COMPRESSORS	SHAFTS MISALIGNED	40 BEYOND 10 IN/SEC MAJOR DAMAGE STARTS, SUCH AS POSSIBLE CRACKING OF CEMENT BLOCK.
PREFABRICATED METAL BUILDING ON CONCRETE PADS	CRACKED PADS, BUILDING TWISTED AND DISTORTED	60

TYPE OF DAMAGE RELATED TO THE PEAK PARTICLE VELOCITY IN THE GROUND WAVES FROM BLASTS



STRUCTURAL RESPONSE AT DIFFERENT
SCALED DISTANCES FROM PIT BLASTS

Nicholls, et. al. (1971),[15] in the Bureau of Mines studies showed no additive effects for delays greater than 15 msec. Gustafsson (1973)[18] indicates that, with correct delay time selection, the vibrations can be made to cancel out. This is possible theoretically and most vibration recordings show some constructive interference. However we regard this as a technique that is impractical for field application.

2.10.6 Delay Additive Effects for Blasts in Tar Sand

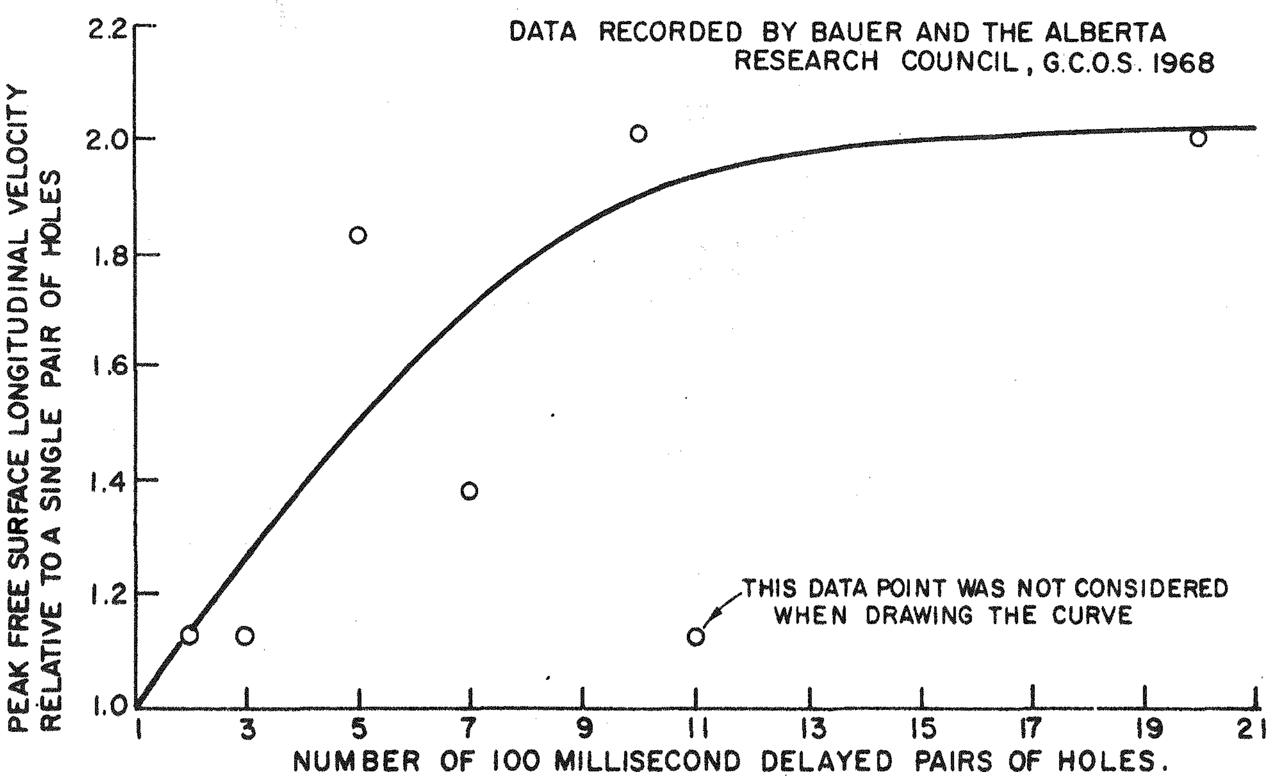
Bauer (1975)[17] reports peak free surface longitudinal velocity data for groups of test blasts fired with various numbers of recorded delayed holes containing constant explosive weights. Initial shots were instantaneously fired to obtain a base reading upon which the relative increase in vibration level could be determined.

Figure 34 illustrates these data as a plot of additional pairs of holes delayed by 100 msec versus relative increase in vibration level above that of an instantaneous firing. As can be seen, disregarding one data point, a fairly good relationship has been established. A maximum increase in velocity of twice the value for an instantaneous shot is indicated at greater than about 12 delay periods.

2.10.7 Blast Vibration Reduction

Blast vibration reduction can be achieved by the use of delays. As a rule blasts are fired along the long axis of a staggered pattern or along the diagonal of a square pattern. The size of the delay is chosen to obtain isolation of the burden before the next row of holes detonates (i.e. in the range of 1 to 3 msec/ft of burden). For a multirow blast where it is considered necessary to detonate each hole on a separate delay, the delay time between holes in successive rows can be excessive, leading to the possibility of "cut-off" holes. To overcome this, down-the-hole non-electric delays can be used, such as the Anodet, Toe Det or Primadet systems.

The Anodet or Toe Det system was developed originally by Canadian Industries, Ltd., to avoid cut-offs in shear zone areas of iron ore mines. These alternative delay schemes call for all of the delays to be down the hole and the use of no surface delay elements. This could be done by choosing appropriate successive or alternate delay periods in the Toe Det or Anodet series for in-the-hole priming of subsequent rows of holes. The first period is selected to be of the order of 100-150 msec (50 msec may be adequate) in order that several rows of surface detonating cord fire before the first holes shoot. A difficulty with this scheme is that the blast has to



THE EFFECT OF ADDITIONAL PAIRS OF HOLES DELAYED
BY PERIODS OF 100 MILLISECOND VS.
RELATED INCREASE IN PEAK SURFACE LONGITUDINAL VELOCITY

be fired as loaded. There is no flexibility in the system which would permit a different hook-up depending on the conditions of the face of the blast. It may permit the use of very large delays between rows of holes without the fear of cut-offs. Furthermore, if it is necessary to deck charge holes on account of the vibration produced, successive periods can be put into each of the deck charges with a separate downline for each and surface delays used between holes (Figure 35). The scheme employs constant 100 msec and 128 msec delays in the hole and surface delays between rows of holes. This gives flexibility with regard to the blast tie-up. Alternatively, a second scheme which does not have this flexibility could be employed where different delays are employed in each hole. If noise is a problem, then the surface initiation can be achieved with delay electric detonators. This is easier and safer than inserting the electric detonators into each of the deck charges. Using 25 msec delays between decks, it has been found that stemming columns as short as 8 feet are adequate between decks in 9-inch diameter holes.

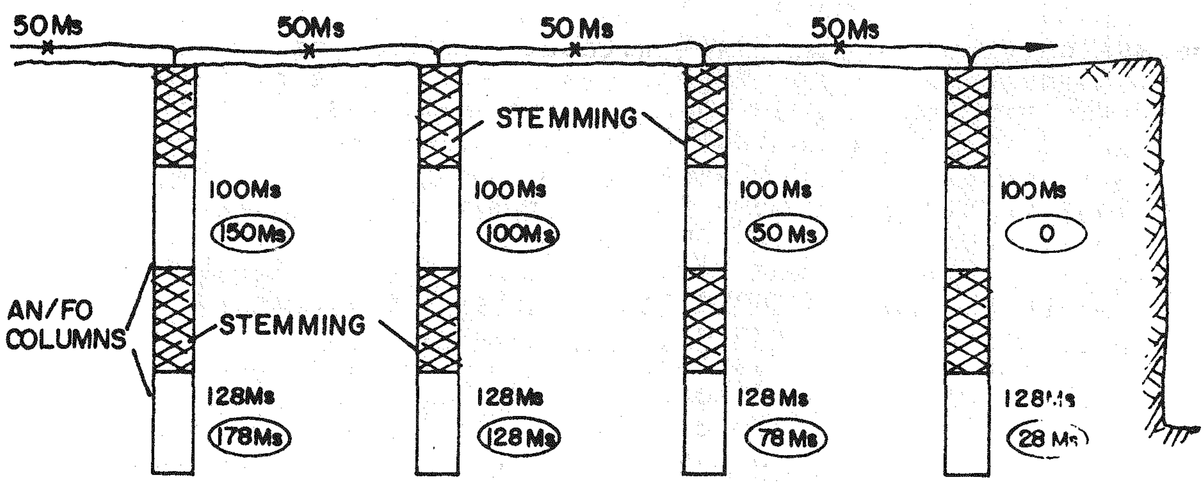
Figures 35, 36, and 37 show typical delay sequences using combinations of surface and down-the-hole delays to achieve the desired charge weights per period. Also shown in Figures 36 and 37 is an electric initiation and delay scheme (Chironis, 1974)[19] using a sequential timer to initiate delay caps attached to the downline at the collar of each blast hole. Both of these methods are effective in reducing the weights per delay period and hence the vibrations from the blast.

2.11 Operational Control

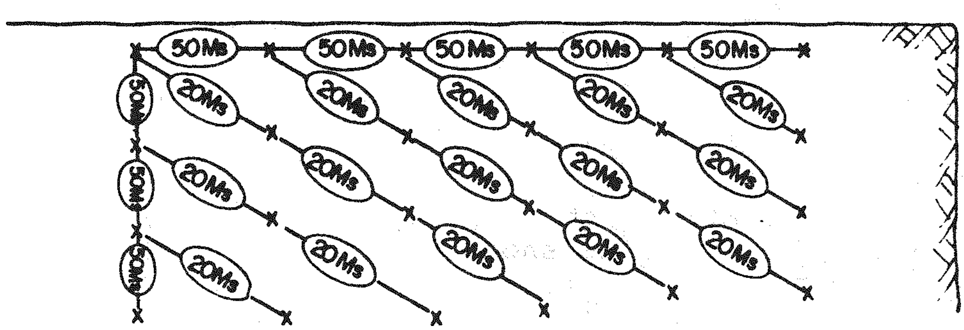
It is essential to have good control on the blast hole layouts. Large diameter holes should be within 1 foot of the designed location. To reduce projection into the pit, the front row of holes should be loaded according to the amount of backbreak experienced. It is usual to increase the stemming height by reducing the explosive column length in the front row of holes. This produces a much better muck pile profile with considerably less "tail" to the blast.

If attention is paid to the geology, direction of blasting, spacing to burden ratio, explosive charging, stemming heights, subgrade drilling and delay selection and sequencing, then fragmentation and loading characteristics can be controlled. This requires some experimentation.

SURFACE DELAYS or ELECTRIC DETONATORS
PERIODS 1,2,3 etc. AT SUCCESSIVE HOLES

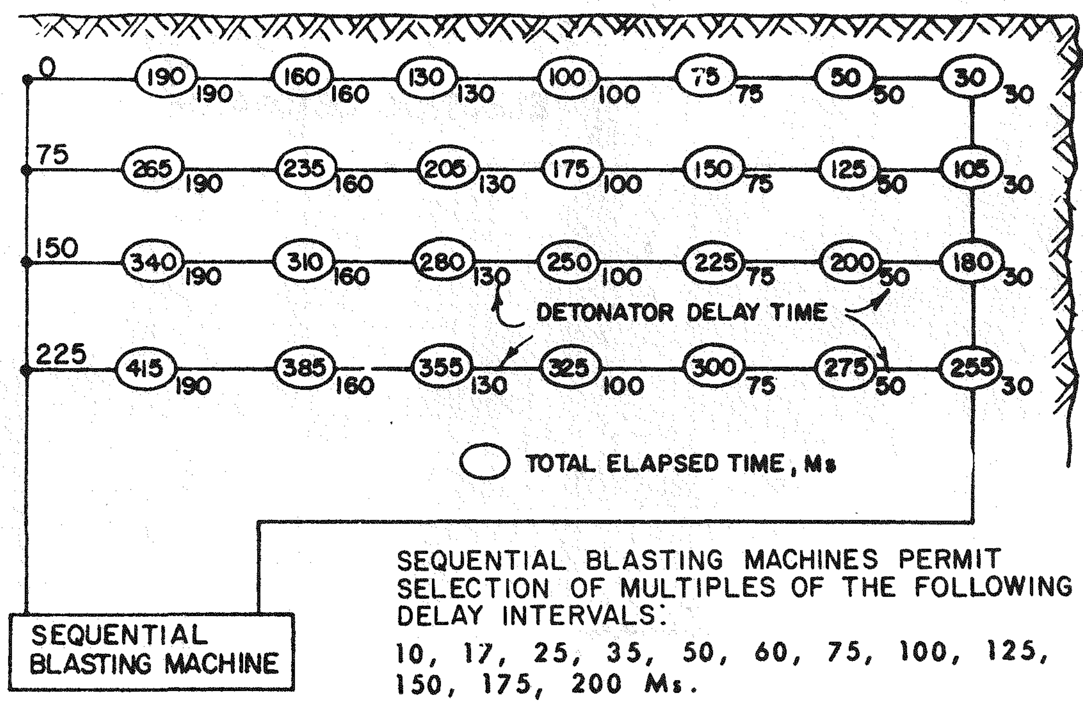


1. 100 Ms AND 128 Ms ARE DOWN THE HOLE DELAYS
2. THE CIRCLED DELAY PERIODS ARE ELAPSED TIME VALUES FOR EACH COLUMN COMMENCING AT ZERO TIME

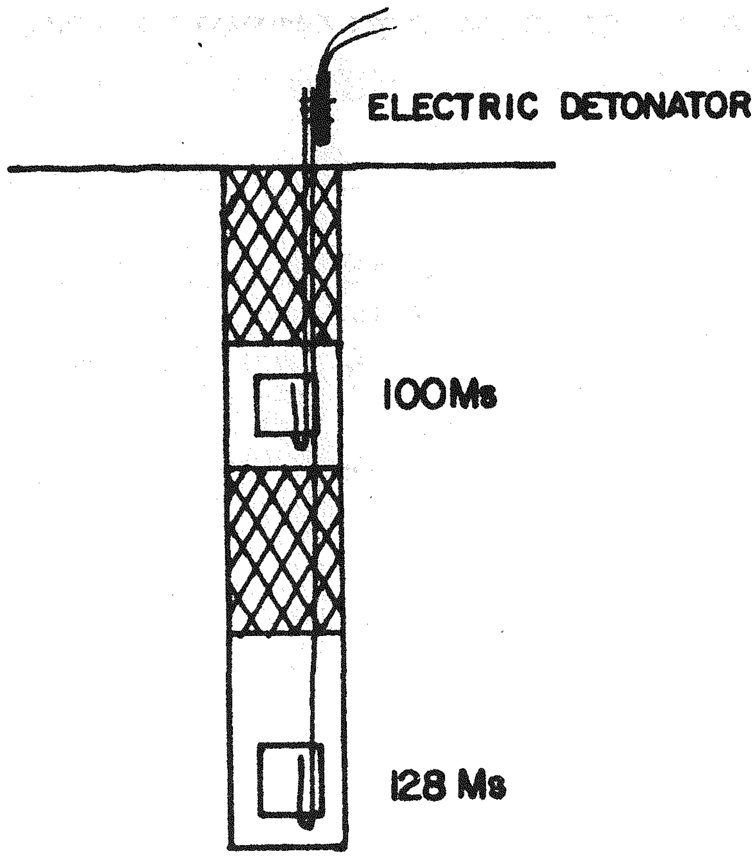


TYPICAL SURFACE DELAY SYSTEM SO EACH DECK CHARGE FIRES SEPARATELY.

DELAY SYSTEMS USING COMBINATIONS OF SURFACE AND DOWN THE HOLE DELAYS TO REDUCE THE CHARGE WEIGHTS PER DELAY PERIOD



ELECTRIC DELAY SYSTEM USING SEQUENTIAL BLASTING MACHINE TO REDUCE VIBRATION



TYPICAL INDIVIDUAL BLAST HOLE USED
WITH SEQUENTIAL BLASTING MACHINE

3.0 SUMMARY AND APPRAISAL OF FIELD STUDY DATA

3.1 General

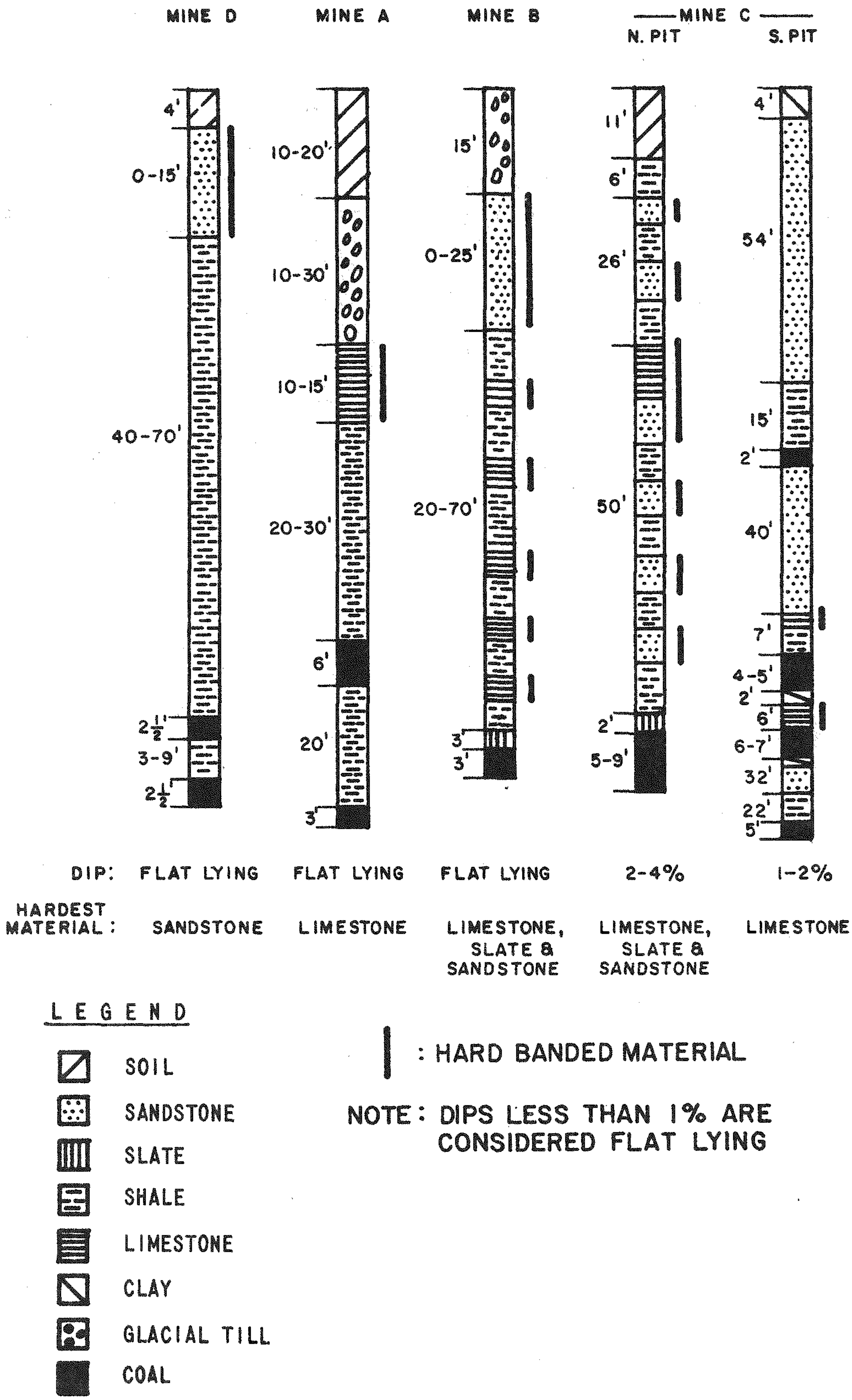
To make the field study data as comprehensive as possible, detailed information was gathered pertaining to overburden stratigraphy and geological structure, drilling, explosives loading, blast tie-ins, dragline operating methods and equipment performance. This information has been summarized in tabular form for both the midwestern and western properties. Where operating and cost data were lacking, estimates were made from the information available. All unit cost information indicated in this report should be assumed to have been estimated by the authors. Estimates have been based on limited actual cost data and on actual operating parameters and performance data obtained during the field studies.

3.2 Geology

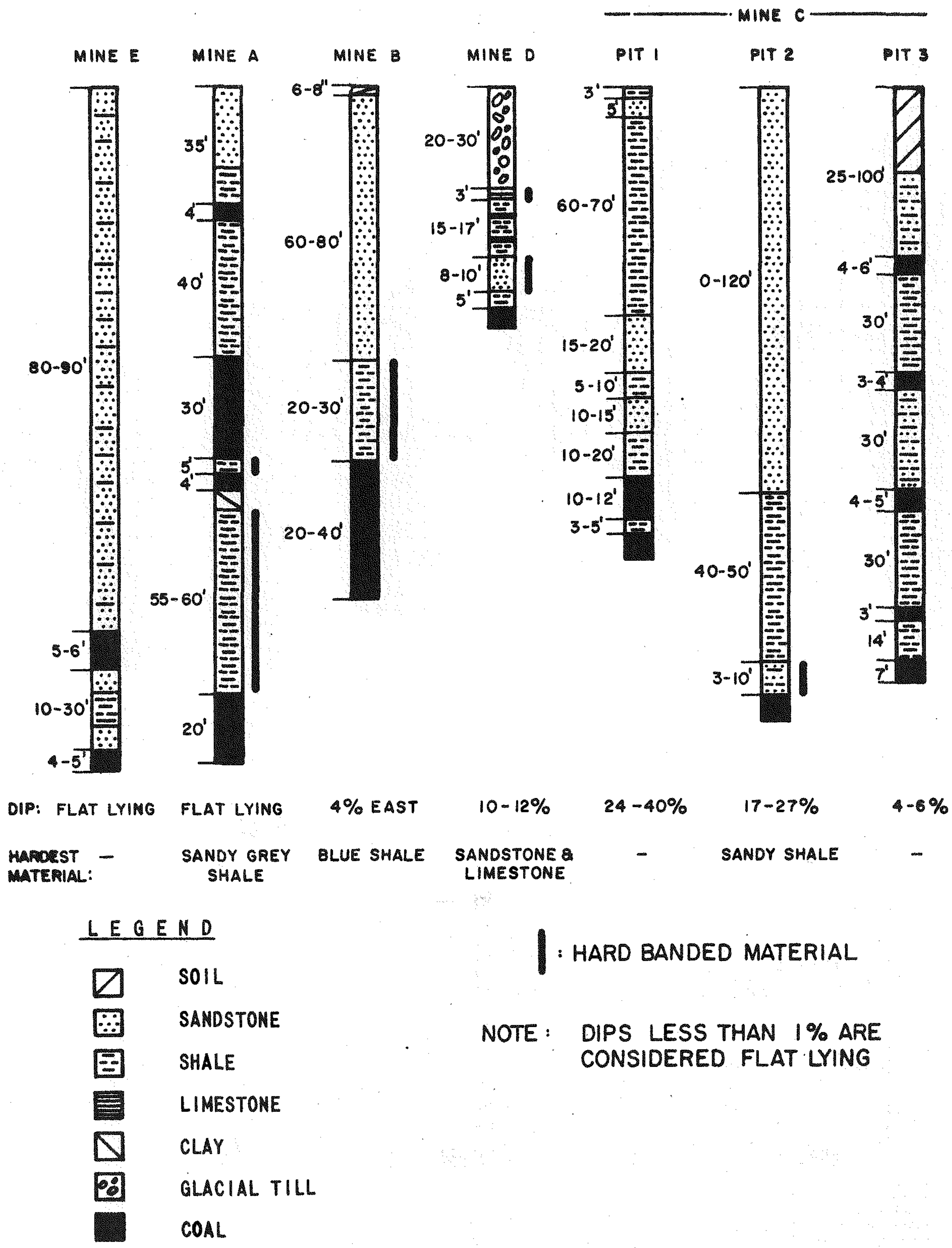
Figures 38 and 39 schematically illustrate the stratigraphic sequences found at the various properties. These show hard bands with respect to blasting and dip of the structure. In general, the sedimentary members forming the overburden and parting are very well jointed and fractured. The stratigraphic sequence varies considerably between properties, but the types of members present are similar and are most commonly those listed below:

soils
glacial till
clay
sandstone
shale (or "slate")
limestone

The limestone, massive sandstones and occasionally the blue-black shale (locally called slate) present the greatest blasting difficulties. Usually these hard members are recognized, and some special blasting effort is made to improve fragmentation. The special techniques consist of drilling additional short holes within the main drill pattern when the hard formation is a single and upper member of the stratigraphy. This method is employed at Midwestern Mine A and is illustrated in Figure 40. Where the overburden consists of multiple hard bands, the common practice is to deck charge the bands with AN/FO and slurry as undertaken at Midwestern Mines B and D and illustrated in Figure 41. In both cases the concepts are good but their success is highly dependent on defining the limits of the hard bands. The drillers report or log appears to be the accepted method of correlation, but mine operators indiate that recording accuracy is often suspect. Relating rock hardness as an inverse function of penetration

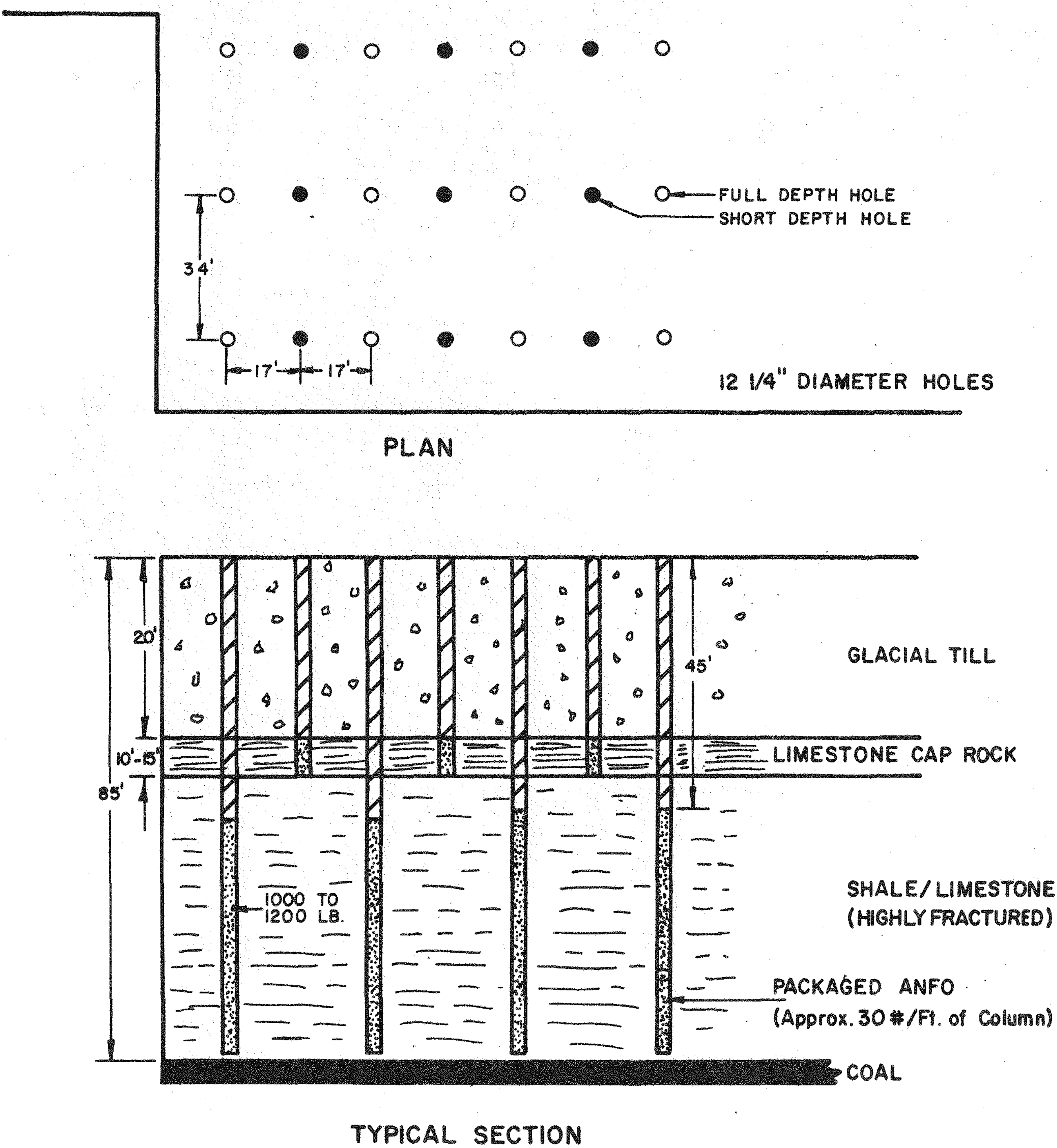


MIDWESTERN COAL STRIPPING FIELD STUDIES
GEOLOGICAL SUMMARY (OVERBURDEN)



WESTERN COAL STRIPPING FIELD STUDIES
GEOLOGICAL SUMMARY (OVERBURDEN)

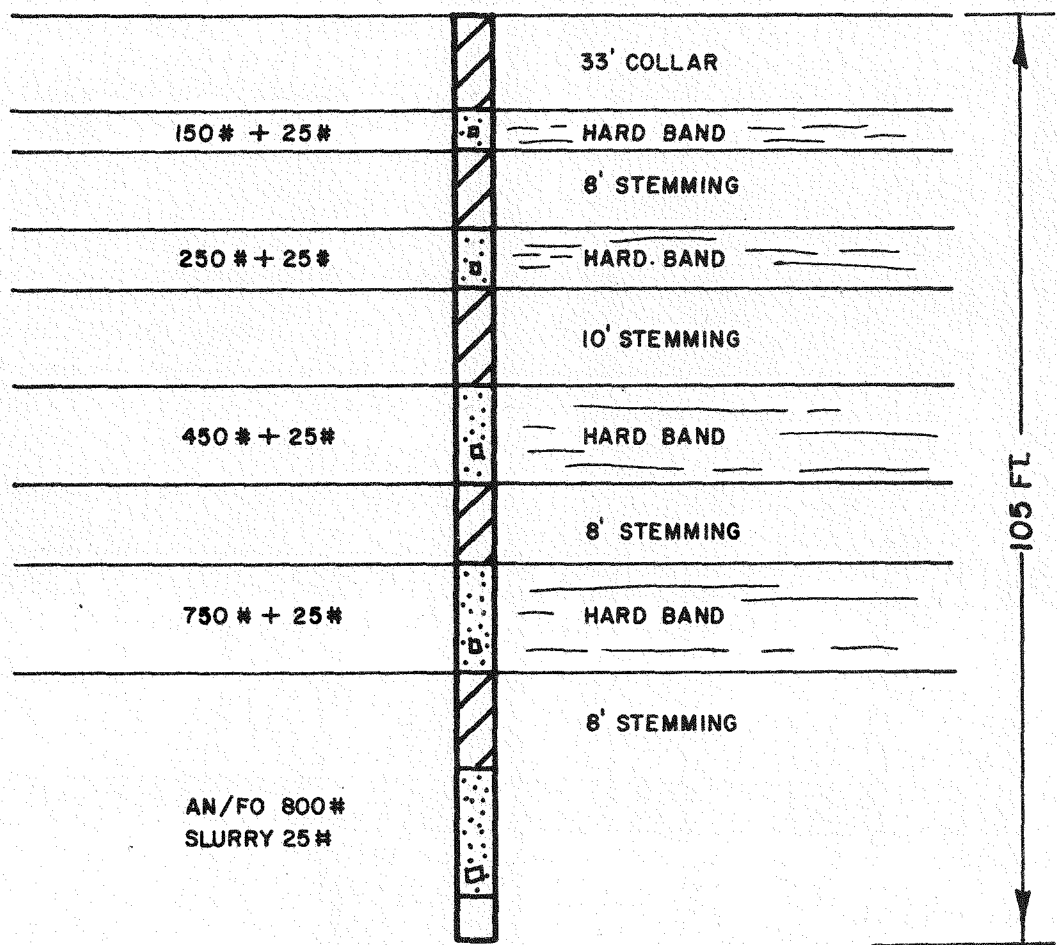
Fig. 39



DRILLING AND LOADING AT MIDWESTERN MINE A

Job No. 19089

Fig. 40



TYPICAL HOLE LOAD

DECK CHARGING OF HARD BANDS

Job No. 19089

Fig. 41

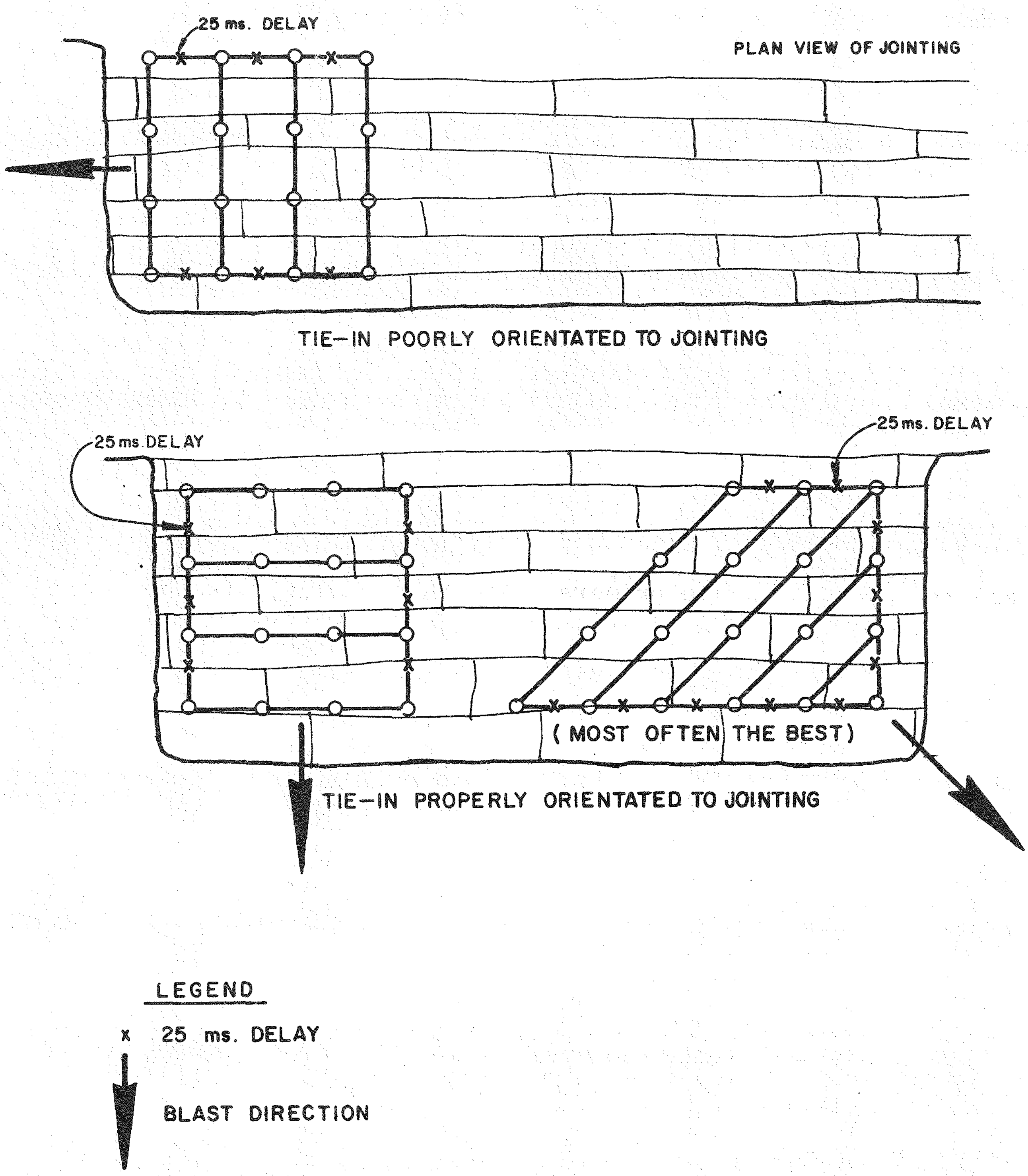
rate is complicated by other factors such as pulldown pressure, rotary rpm and bailing velocity. Depth counters, where employed and well calibrated, are quite accurate, but where proper maintenance facilities are not available, depth estimation has an accuracy of only about 3 feet. Explosive and stemming slumping in the blast hole is quite common, especially where holes are unlined or the explosive is relatively viscous and the ground is highly jointed and fractured. Slumping values of 5 to 10 feet are quite common under those conditions. Therefore, decking of small hard bands (4 to 8 feet) would have a poor success factor unless the hard band was at the bottom of the hole.

Key geologic factors that affect blasting are dip of the bedding and spacing and orientation of the joints and fractures. At the properties where the overburden is steeply dipping, such as Midwestern Mine C, Western Mine D, and Western Mine C, blasting up-dip is avoided, which eliminates dip as a blasting problem. However, the orientation of blast tie-ins relative to dominant sets of joints and fractures is not considered. If proper orientation is being achieved it would appear to be by coincidence rather than design. A proper study of this condition is warranted at the properties selected for the test program. Figure 42 is a simple illustration of good and poor orientation of blast tie-in to jointing.

3.3 Drilling

The drilling parameters, performance and cost are summarized in Tables VI and VII for the midwestern and western properties, respectively. The drilling, perhaps more than any other item, expresses the individuality of the various mines. Major differences and similarities noted are as follows:

(1) Drilling Direction: Vertical drilling of the overburden appears to be the standard, with the exception of Midwestern Mine A which employs a 9 inch twin horizontal drill at the end of the cut where rock overburden is shallow (maximum burden to cap rock is 30 ft). The horizontal drilling takes advantage of the stratigraphic sequence where the upper overburden members are unconsolidated materials that do not require blasting. Angle drilling was not observed at any property.



ORIENTATION OF THE BLAST TIE-IN TO THE JOINTING

Fig. 42

TABLE VI

MIDWESTERN COAL STRIPPING FIELD STUDIES
DRILLING SUMMARY (OVERBURDEN)

Mine	Hole Diameter (in.)	Hole Depth (ft)	Crest Burden (ft)	Toe Burden (ft)	Drill Pattern ft x ft	Drill Type	Penetration Rate ft/op hr	Drilling Yield BCY/ft	Cost Per Foot $	Cost ($/BCY)	Remarks
MINE D	15	+50 +40 +20 -20	30	65 39	36 X 36 34 X 34 30 X 30 24 X 24	61R	120	48 21	3.15	.065 .15	Average hole depth of 60 ft
MINE A	12 1/4 10 5/8 9 (H)	60-90 20 100	40 20 50	75 20 --	34 X 34 20 X 24 28	61R 50R Horizontal	100-125	35* 15 31-52	3.50 2.30 2.00	.10 .15 .07-.04	Horizontal Drilling up to 50' of overburden depth.* Short holes used between full depth holes.
MINE B	15	100+ 80 50 30	18	50	54 X 45 42 X 36 36 X 32 30 X 27	61R	150	90 56 43 30	2.40	.03 .04 .06 .08	Average hole depth 75-80 ft. Horizontal drilling for shovels only.
MINE C	10 5/8	90 82	10 10	55 50	27 X 33 27 X 30	60R &M4	125	28* 30*	1.12 1.50	.04 .05	*Use 25 ft short holes interlaced with 27' X 33' full depth holes.

H - Horizontal Drilling

NOTE: All cost information is estimated by authors based on available data.

TABLE VII

WESTERN COAL STRIPPING FIELD STUDIES
DRILLING SUMMARY (OVERBURDEN)

Mine	Hole Diameter (in.)	Hole Depth (ft)	Crest Burden (ft)	Toe Burden (ft)	Drill Pattern ft x ft	Drill Type	Penetration Rate ft/op hr	Drilling Yield BCY/ft	Cost Per Foot $	Cost ($/BCY)	Remarks
MINE A	10 5/8 9	70 70	60 60	75 75	30 x 30 25 x 25	45R IR	131 131	33 23	1.95 1.75	.06 .075	Avg. hole depth 70'. Cost data fictitious due to conglomerate relationship.
MINE B	10 5/8	45 -130	10 -15	35 -70	26 x 26	45R	110	25	2.25	.09	Hard to dig toe, some 11" holes
MINE D	7 3/8	50	30	55	21 x 30	40R	180	23	1.15	.05	
MINE C PIT 1	9 12 1/4	75 75	Varied	Varied	25 x 25 33 x 33	40R 60R	100 160	23 40	2.07 2.80	.09 .07	Wet holes
MINE C PIT 2	10 5/8 9 7/8 6 3/4	40 -180 ↓	As Close as Possi- ble	+77	30 x 30 25 x 25 15 x 15	45R (2) RR11 RSK	180 | 300	33 23 8	1.60 1.45 .95	.05 .06 .11	Wind blows drill cuttings back into hole - do not clean holes
MINE C PIT 3	12 1/4	35 -70	20	60	24 x 24	60R	110	21	2.50	.12	
MINE E	9 7/8 11	40-100	2-15	45-50	25 X 30 30 X 35	60R, 45R, M4, GD100	125-140	28 39	1.80 1.90	.065 .05	Very little 11" drilling

NOTE: All cost information is estimated by authors based on available data.

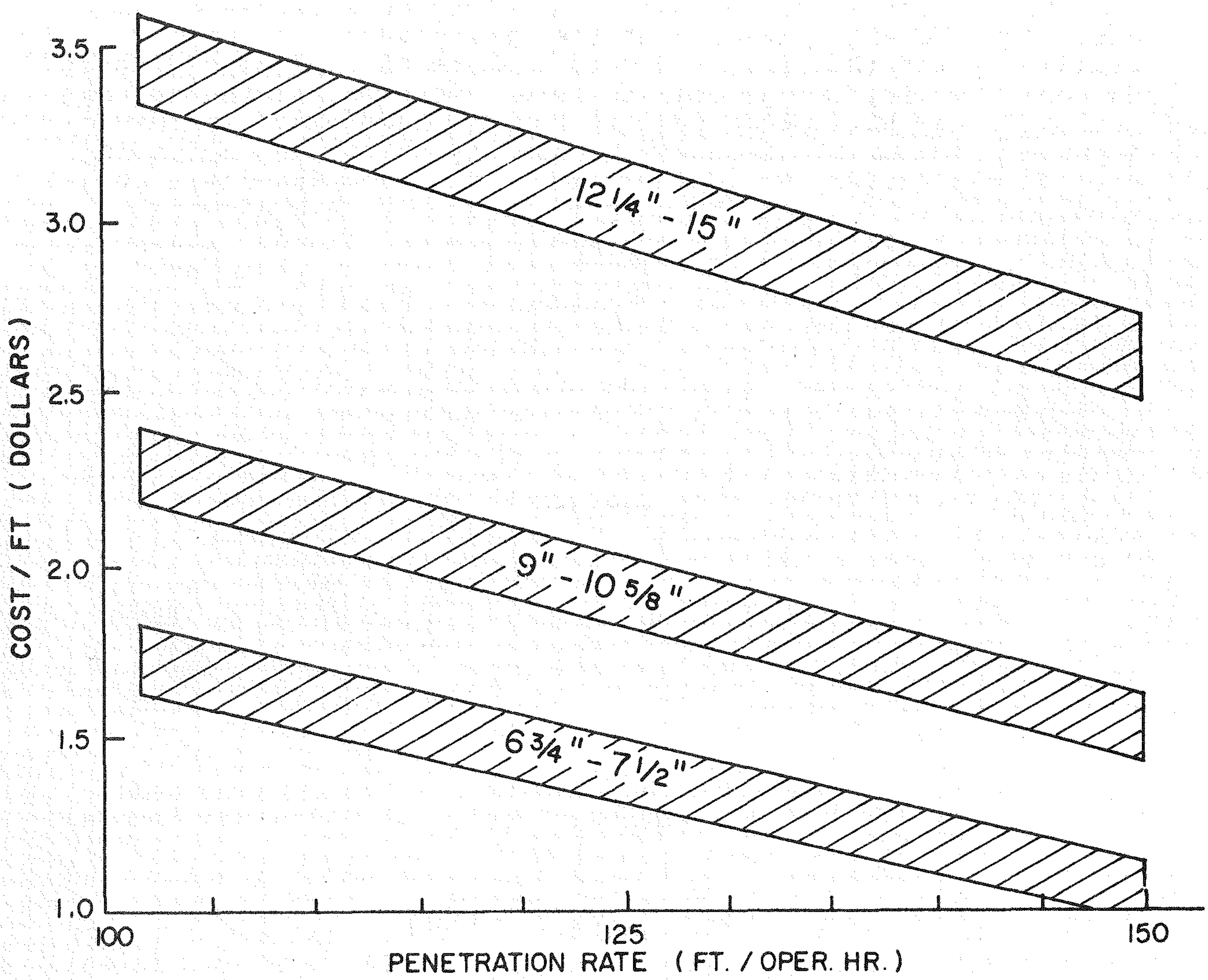
(2) Penetration Rate: Penetration rates were consistently high regardless of hole diameter and averaged approximately 130 ft/Op Hr. The bits most commonly employed were tri-cone tooth, tungsten carbide insert, and drag types. The high penetration rates and low bit cost per foot ($.10 - $.20) were reflected in a low unit operating cost ranging from $.03 to $.13/BCY and averaging $.06 to $.07/BCY. Based on actual costs and drilling parameters provided during the field studies, Figure 43 has been constructed as a means of estimating drilling cost for those properties where cost information was not available. Drill unit cost ($/BCY) was generally lowest with large diameter holes.

(3) Redrills: Redrills as a result of bad ground was not a major problem for most operations, but Midwestern Mine C had re-drills as high as 5 percent as a result of water. Wet holes were redrilled rather than dewatered. Western Mine C had difficulty with cuttings that were blown back into holes by wind.

(4) Hole Size: Various hole sizes are used. Holes ranged from 6 3/4-inch to 15-inch rotary on hole depths of 20 to 180 feet.

(5) Drill Patterns: In general the drill patterns increase proportionally with the hole diameter and range from 15 by 15 feet on the smallest hole size to 54 by 45 feet on the largest hole size. A practice noted at several properties, especially at Midwestern Mine B, is to increase hole spacing as hole depth increases for the same size hole. This practice is common and can be justified where hole depths are shallow (e.g. less than 25 feet). However, in the range of 30 to 100 feet one mine has effectively tripled the pattern size (30 by 27 feet to 54 by 45 feet) with essentially the same powder factor, geological members and hole diameter. This approach is not consistent with good blast design rationale.

(6) Face Hole Burdens: No consistent relationship exists between crest and toe burden and hole diameter and depth. In general, toe burdens appear excessive either as a result of faulty design or highwall height and slope angle. The impact of this will be explained further in the blasting section.



DRILLING COST ESTIMATE BASED ON
PENETRATION RATE AND HOLE DIAMETER

The character of the overburden fragmentation is dependent upon the effectiveness of the explosive distribution. Drilling yield (BCY/ft) is a good measure of explosive distribution; that is, the greater the yield, the poorer the distribution in the horizontal plane. For the same explosive energy input per ton or CY of rock, the quality of fragmentation decreases as drilling yield increases. In fact, the field studies indicated that the three properties with the highest dragline productivity had relatively low drilling yields (e.g. less than 30 BCY/ft).

3.4 Explosives Loading

Explosives loading methods and techniques have been summarized in Tables VIII and IX for the midwestern and western coal properties, respectively. In general the various properties have similar loading methods, except that three of the nine operations employ deck charging to counteract specific hard-banded formations or to achieve better vertical powder distribution for a given hole size and powder factor.

AN/FO is the most commonly used explosive, and packaged AN/FO and slurry are employed only in wet-hole conditions. A known characteristic of AN/FO is its degradation when wet, yet there seems to be a complete reluctance to dewater holes or use dryliners (plastic hole liners used to keep AN/FO dry). In addition, packaged AN/FO is dropped down the hole rather than lowered. This leaves the effective water proofing suspect, since most of these bags will split when dropped from a height of 6 or 7 feet.

Cast primers of 3/4 to 1 lb are most commonly used to initiate the explosive and should be adequate for the range of hole diameters employed and the use of AN/FO. Most operations use two primers for a continuous column of explosive and locate one in the top and one in the bottom of the column. Where packaged AN/FO or slurry is used, it is common to use a primer for each 50-lb bag of explosive. The operations that employ deck charges prime each deck on a separate down-line and locate the primer in the bottom 2 feet of each deck. Often a 25 or 50 lb bag of slurry is inserted in the deck beside the primer, but it is uncertain whether this is to serve as a priming booster or to improve the bulk strength characteristics of the deck charge within the hard-banded formation.

Powder factors range from .17 to 1.0 lb/BCY and average .4 to .5 lb/BCY. As a result, average blasting unit cost is in the order of \$.05 to \$.06/BCY depending on the price of AN/FO. This price varies from \$.07/lb in bulk to \$.12/lb packaged.

TABLE VIII

MIDWESTERN COAL STRIPPING FIELD STUDIES
EXPLOSIVE LOADING SUMMARY (OVERBURDEN)

Mine	Explosive Type	Toe Charge	Priming		Collar Heights	No. of Decks	Down Lines	Power Factor lb./BCY	Blasting* Cost/BCY	Remarks
			Type	Location						
MINE D	ANFO(B) ANFO(P) Slurry (P)	-	3/4# Pent. +25# slurry	2 ft. from bottom of each deck	23-35'	1-6	40 grain	.5	.06-.07	No dewatering or dryliners Packaged ANFO and NCN Slurry used in wet holes
MINE A	ANFO(P)	-	1# Pent.	Top & Bottom	30' Top of Limestone	-	50 grain	.4-.5	.06-.07	No dewatering or dryliners Packaged ANFO - $.12/lb
MINE B	ANFO(B) Slurry (P) ANFO(P)		1# Pent. +25# slurry	2 ft. from bottom of each deck	33'	4-6	40-50 grain	.3-.4	.03-.05	No dewatering or dryliners Packaged ANFO & Slurry in wet holes. Bulk ANFO $.10-.11/lb
MINE C	ANFO(B) ANFO(P) Slurry(P)		2# Pent.	Bottom of each deck	21-36'	1-4 as requir-ed	40 grain	.33	.03-.04	Dryliners used. Packaged ANFO and Slurry used in wet holes. Bulk ANFO $.07-.08/lb

B - Bulk
P - Packaged
*$.01/BCY added for labor and miscellaneous blasting supplies

NOTE: All cost information is estimated by authors based on available data.

Job No. 19089

TABLE IX

WESTERN COAL STRIPPING FIELD STUDIES
EXPLOSIVE LOADING SUMMARY (OVERBURDEN)

Mine	Explosive Type	Toe Charge	Priming		Collar Heights	No. of Decks	Down Lines	Powder Factor lb./BCY	Blasting* Cost/BCY	Remarks
			Type	Location						
MINE A	ANFO(P&B)	-	1#Cast	1 with bulk 1 with pack-aged	30'	None	N/A	0.5	.05-.06	Packaged explosive used in wet holes only. Minimum 30' stemming to prevent flyrock.
MINE B	ANFO(B) Water gel Slurry	-	1#Cast	4-5 bulk 1 above bottom bag 1 below top bag with gel	+20	None	Special 30 or Special 50 (1)	0.17-0.19	.02-.03	Use 3 trunk lines. Average time between loading and firing is 24 hrs. In key area P.F.=0.3
MINE D	ANFO(B&P) Water gel Slurry	-	1#Cast	1-middle	20-25	Not Used Now	(1)-40 grain	0.3-0.4	.04-.05	Tried decking; did not seem to help. Use 20-25' stemming to control flyrock.
MINE C PIT 1	ANFO(B&P) Slurry (P)	-	1#Cast	1-bulk, bottom 1-touching each bag	varied dependent on hole depth & cuttings No mini-mum	None	1-2 strip mine special	0.5	.07	Average time between loading and shooting 3 days. Use high density material in key cut.
MINE C PIT 2	ANFO(B&P)	-	1#Cast wet holes	1-bottom in shallow holes 2-center & bottom for deep holes	30' in holes over 70' Varied in others	None	1 ea. 25 gr. with dynamite 45 gr. with cast.	0.8	.11	Explosives loaded by column rise. Time between loading and firing 1-14 days.
MINE C PIT 3	ANFO(B) Some(P)	-	1#Cast 3 x 8 detagel	1-bottom sometimes add 1 in center	21" Avg.	None	40 gr. 1 ea.	0.5-1.0	.05-.10	Blasting results thought to be good. Time between loading and firing 2-14 days. Some water problems
MINE E	ANFO(B) Slurry (P)		1 stick of 1 7/8" gelatin 70%	Bottom	25'	-	25 grain	.35	.03-.04	Packaged slurry used in wet areas. No dryliners or de-watering

B - Bulk ANFO
P - Packaged
*Blasting cost per BCY when estimated includes 1¢/BCY for labor etc.
NOTE: All cost information is estimated by authors based on available data.

Midwestern Mine A uses packaged AN/FO exclusively due to the high moisture content of the overburden. Field observations indicate that packaged explosives typically give a column rise of approximately 30 lb/ft of 12 1/4-inch diameter hole.

Fragmentation difficulties were observed in both the toe and collar areas of the overburden. The collar heights employed are consistently 5 to 15 feet greater than the scaling laws of Section 2.3 would dictate. When the collar material is unconsolidated, such as Midwestern Mine A, no problem should occur, but where the rock is very near surface, such as at Midwestern Mines B and D, a problem exists with fragmentation. In addition, the toe burden and drill patterns listed in Tables X and XI are generally too large when compared to the values generated by the scaling laws of Section 2.0.

3.5 Blast Tie-In

Blast tie-in data also have been summarized in Tables X and XI for the midwestern and western coal properties, respectively. The main information of interest is as follows:

(1) Tie-in Configuration: All three conventional tie-in methods are employed: row-on-row, diagonal and chevron. Figure 44 illustrates typical examples of all three. It should be noted that the row-on-row method can imply rows parallel or perpendicular to the crest.

(2) Direction of Blasting: The majority of the operations delay and tie in the blast in such a manner that blast movement is in the direction of the muckpile from the previous shot. In effect this is buffer blasting because horizontal displacement of the blast is prevented. In fact, with a substantial toe burden toward the cut and overburden on the other two sides, there is virtually no opportunity for horizontal displacement in any direction. With this confinement, it is difficult to understand what function the delays serve other than to reduce vibration. In addition, this excessive confinement can lead to damage of the highwall and abnormal levels of vibration.

(3) Burden to Spacing Ratio: Unless a unique geological structure exists, it has been found that a burden-to-spacing (B/S) ratio of 1:2 to 1:5 gives best results. This B/S ratio applies to the tie-in configuration and not the drill pattern. Approx-

TABLE X

MIDWESTERN COAL STRIPPING FIELD STUDIES
BLAST TIE-IN SUMMARY (OVERBURDEN)

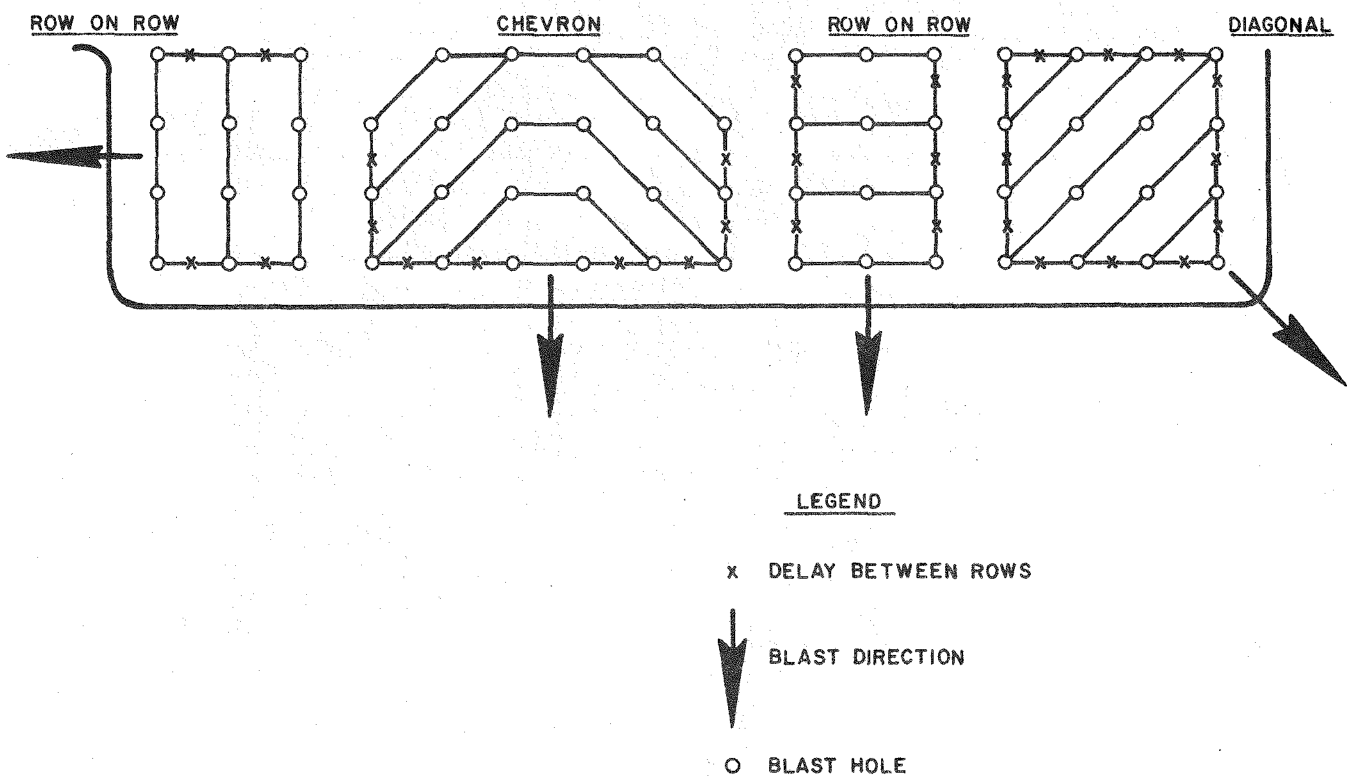
Mine	Tie-in Configuration	Direction Of Blasting	Burden to Spacing Ratio	Method of Initiation	Type of Delay	Delay Between Rows	Delay Between Holes	Delays in the hole Between Decks	Trunkline	Remarks
MINE D	Diagonal	Previous Shot	1:2	Cap *	Surface Delays	42 ms	17 ms	-	nonel	Just being introduced
	Diagonal	Previous Shot	1:2	E.B.**	EBC	50-70	10-17 m	25 ms	electric	Sequential timer used.
MINE A	Row on Row	Cut	1:1	Cap	Surface Delays	9 ms	9 ms	-	40 grain	Rows shot from opposite ends
MINE B	Row on Row	Previous Shot	1:.8-.9	Cap	Surface & Down-the-Hole Delays	17-25 ms	17 ms	25 ms	nonel & 40 grain	The drilled spacing and burden get reversed by the blasting direction.
	Diagonal	Previous Shot				42 ms	17 ms	25 ms		
MINE C	Diagonal	More towards Cut than Previous Shot	1:2	E.B.	EBC	60 ms	50 ms	-	electric	Good orientation to structure
	Row on Row	Previous Shot	1:1	Cap	Surface Delays	17 ms	9 ms	-	40 grain	

* Cap - Electric or non-electric cap affixed to detonating cord lead line.
**E.B. - Multi-cap electric blasting machine connected to electrical lead, trunk and downlines.

TABLE XI

WESTERN COAL STRIPPING FIELD STUDIES
BLAST TIE-IN SUMMARY (OVERBURDEN)

Mine	Tie-In Configuration	Direction of Blasting	Burden to Spacing Ratio	Method of Initiation	Type of Delay	Delay Between Rows	Delay Between Holes	Delays in the hole Between Decks	Trunkline	Remarks
MINE A	Chevron & Diagonal rows	Shoot into Buffer @60' spacing	1:2 1:1	Elect. cap & Primacord	Surface & down-hole with primacord	N/A	N/A	None	N/A	Experimenting with nonel delays Use double trunk-line.
MINE B	Chevron	Towards Previous Shot	1:2	Reynolds Detonating Bridge wire	Surface Delays with Primacord	25 ms	None	None	Special 50 & 30 grain	Have 5 test shots ahead of D/L. Each done by a different powder company.
MINE D	Single row minimal double row	Shoot parallel to dip-towards free face	3:2:1	Electric Cap	Surface Delays with Primacord	50 ms	25 ms 2nd & 4th hole	None	30 grain	Typical blast of 1-row
MINE C PIT 1	Varied	Varied	1:1 1:2	Electric Cap	Surface Delays with Primacord	25 ms	25 ms	None	Deta cord E-cord 40 grain	Appears to be no set tie-in or direction of blasts dependent on situation
MINE C PIT 2	Rows	Parallel to Strike	1:1	Electric Cap	Surface Delays with Primacord	25 ms	None	None	25 grain & 40 grain	6-10 rows per shot Use 1 EBC/1000' of primacord
MINE C PIT 3	Diagonal rows	Towards Previous Blast	1:2	Electric Cap	Surface Delays with Primacord	25 ms	None	None	40 grain	Blast parallel to strike. Have mud zones; they do not place explosives in these areas.
MINE E	Row on Row	Previous Shot	1:1.2	Cap	Surface Delays with Primacord	25 ms	None	None	18 grain	



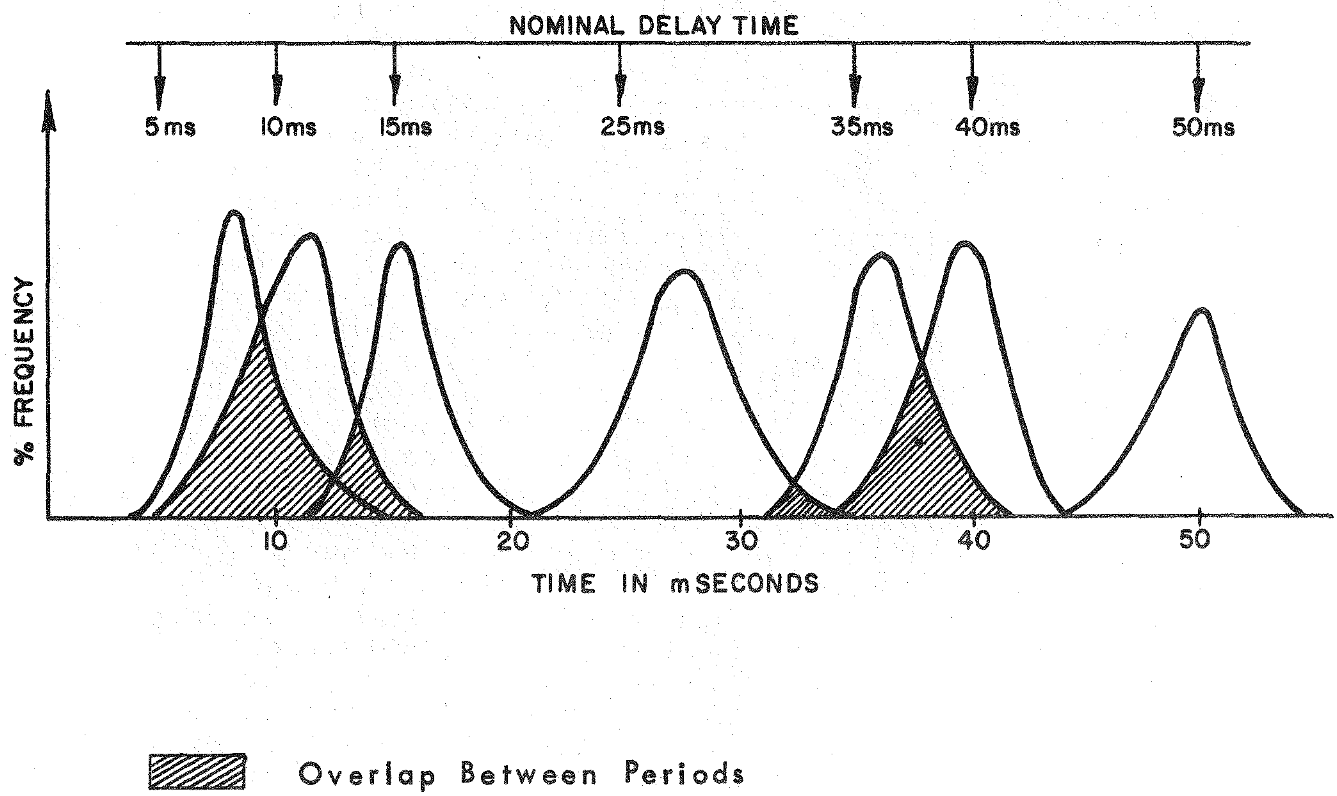
TYPICAL TIE-IN CONFIGURATION
FROM FIELD STUDIES

imately half the properties use a 1:2 B/S ratio. The other properties have a method of tie-in that often results in a burden greater than the spacing. In general, a square drill pattern is drilled where the tie-in is to be diagonal or chevron and a rectangular or staggered-rectangular pattern is drilled where a row-on-row tie-in is planned. The V_1, V_2 and V_3 tie-in methods (Figures 12 and 13) on squared or staggered drill patterns were not being employed at any of the properties visited.

(4) Delays: From experience and test work it has been shown that a delay interval between adjacent holes should not be less than 1 msec per foot of burden and delays of up to 1 1/2 to 2 msec for surface delays and detonating cord are considered safe for avoiding cut-offs. Best blasting results in terms of fragmentation and diggability are achieved with the longest delay intervals possible without cut-off. For the collar heights, explosive type, rock type and drill patterns observed, a 30 to 50 msec delay interval between holes would be conservative with surface delays and detonating cord. Sixty to 100 msec delays are feasible for down-the-hole delays (e.g. Midwestern Mine C has 110 msec between adjacent holes). Many of the properties surveyed were using the extremely short periods of 9 and 17 msec. These delays are not only very short but have the greatest percentage error value (160 percent max. error for 5 msec delay). As the delay interval increases, the percentage decreases (14 percent max. error for 50 msec delay). Figure 45 shows the deviation about the mean for various periods of delays tested. Only two properties, Midwestern Mines B and D, delayed between decks. In both cases, the delays were timed to relieve from top to bottom, which is a questionable practice in view of confining the energy.

3.6 Blast Assessment

It is felt that overburden fragmentation to date is more a function of the natural prefractured state of the rock than the drilling and blasting techniques employed. Current blasting methods tend to loosen natural fractures rather than induce fragmentation. When the dragline exposes the muckpile the original structure is normally intact and resists bucket penetration. Tables XII and XIII summarize blast assessments of the midwestern and western coal operations.



TYPICAL DELAY TIME VARIATION FOR
DETONATING DELAYS
(SAMPLE + 4000 DELAYS)

TABLE XII

MIDWESTERN COAL STRIPPING FIELD STUDIES
BLAST ASSESSMENT (OVERBURDEN)

Mine	Oversize	Flyrock	Horiz. Displ.	Vert. Displ.	Fragmentation	Diggability Characteristics	Noise Vibration
MINE D	Primarily at top in sandstone which is benched.	Negligible in highwall 800 ft. in parting	Negligible	3'-4'	3' to 4 max. except in bench which can be quite hard	Muckpile difficult to penetrate with bucket.	No problems
MINE A	Mainly in 10-15 ft. limestone band	200 ft.	Negligible	4'-5'	Fairly good overall	Muckpile difficult to penetrate with bucket.	No problems
MINE B	Top portion & in various hard bands if deck charges are not well matched to drill reports.	Nil	Negligible	4'-5'	4'-8' max. 1' average	Chop down in bench portion is difficult Remainder of high-wall resists bucket penetration.	Vibration & noise closely monitored in some portions of the pit.
MINE C	Associated with hard bands.	300'-400'	20-30 ft. in areas.	4'-5'	4'x4' common Good overall	Muckpile difficult to penetrate with bucket.	No problems

TABLE XIII

WESTERN COAL STRIPPING FIELD STUDIES
BLAST ASSESSMENT (OVERBURDEN)

Mine	Oversize	Flyrock	Horiz. Displ.	Vert. Displ.	Fragmentation	Diggability Characteristics	Noise Vibration
MINE A	Minimal	Very little; if any	Negligible	±6'	Medium	Better than average	No problems - remote location
MINE B	Negligible	Almost none	Minimal	Minimal	Fine - little back break	Good in top section but poor digging in blue shale above coal seam.	No problems - remote location
MINE D	Some in inter-bedded layers	None - controlled by increased stemming	Minimal	Minimal	Fractures in place	Better than average	No problems
MINE C PIT 1	Minimal	Some up to 200' dis-tance	Minimal	0-5'	Good to excellent	Tight Muckpile	No problems - remote area
MINE C PIT 2	Minimal	Very little controlled by stemming	Minimal	10' typical	Good	Tight Muckpile	No problems - remote location
MINE C PIT 3	Minimal	Some up to 250' travel. Short holes a problem.	Not much	6'-7'	Good - 12" or less	Tight Muckpile	No problems - remote location
MINE E	Negligible	90 ft.	Negligible	4'-5'	Excellent	Good digging. Material is highly prefractured.	No problems

Tight Muckpile - implies basic overburden structure is intact after blasting and bucket penetration in the bank is difficult

3.7 Dragline Operation

Data regarding dragline size, operating method, performance and cost have been summarized in Tables XIV and XV for midwestern and western coal properties, respectively. The prime objective of this study is to improve dragline productivity and, therefore, a great deal of weight has been placed on current dragline productivity in deciding the blasting parameters which could benefit most from a test program. Overburden fragmentation is not the only factor affecting dragline productivity and so the digging mode has also been shown. Figures 46, 47, 48 and 49 illustrate typical dragline casting, parting removal, bench chopdown and extended bench rehandle, respectively.

Data for 17 draglines were collected from the nine properties visited. Dragline sizes at these properties varied from 32 CY to 150 CY. To better relate productivity between the different size machines the BCY/Op Hr/CY of bucket capacity has been shown for each machine. The average value for this factor is 37.2 BCY/Op Hr/CY of bucket. In relation to this average, the values at three properties draw immediate attention. Midwestern Mines A and B were 25 percent below the average. Very little explanation is indicated by the digging mode, but these two properties did exhibit questionable practices in their drilling and blasting methods. Western Mine E, on the other hand, is 40 percent above the average dragline productivity, and it was noted during this field study that fragmentation was excellent. A test program would be of little benefit at this operation, but the relationship between dragline productivity and excellent fragmentation gives insight to the potential of a test program.

Dragline productivity versus size has been plotted for a number of other properties prior to this study (Figure 50). These data show that 40 BCY/Op Hr/CY of bucket is typical for the industry.

Actual costs were available for approximately half the draglines. From this information a graph showing cost per operating hour versus dragline size was constructed for estimating costs at other operations (Figure 51). The dragline unit cost varies from \$.10 to \$.23/BCY and averages approximately \$.16/BCY. A cost distribution at Midwestern Mine B indicates that approximately 75 percent of the costs are fixed and 25 percent are variable. This is typical and indicates that an increase in productivity would have a substantial impact on unit cost.

TABLE XIV

MIDWESTERN COAL STRIPPING FIELD STUDIES
DRAGLINE OPERATION SUMMARY (OVERBURDEN)

Mine	Machine Type	Bucket CY	Digging Mode (%)				Mech. Avail.%	BCY/* Op Hr	BCY/Op Hr per CY of Bucket	Cost/** Op Hr	Cost/ BCY	Remarks
			Casting	Parting	Chop down	Rehandle						
MINE D	M-8950	150	77%	8%	15%	-	72%	4,830	32.2	$743	.154	310' boom
MINE A	B.E.-2570	110	80%	-	20%	22.5%	70%	3,009	27.4	$540	.180	310' boom
	B.E.-1550	50	60%	20%	20%	10%	78%	1,464	29.3	$333	.227	322' boom
MINE B	B.E.-2570	110	75%	-	25%	20%	81%	3,655	33.3	$595	.163	310' boom
	B.E.-1370	58	75%	-	25%	20%	81%	1,670	28.9	$290	.175	310' boom
	B.E.-1370	58	75%	-	25%	20%	77%	1,691	29.2	$277	.165	310' boom
MINE C	B.E.-2550	75	82%	-	18%	28%	70%	2,690	35.9	$363	.135	300' boom
	B.E.-1450	60	75%	-	25%	32%	82%	2,280	38.1	$228	.100	250' boom

Rehandle - % of total overburden
* BCY/Op Hr includes rehandle
**Cost of ownership excluded
NOTE: All cost information is estimated by authors based on available data.

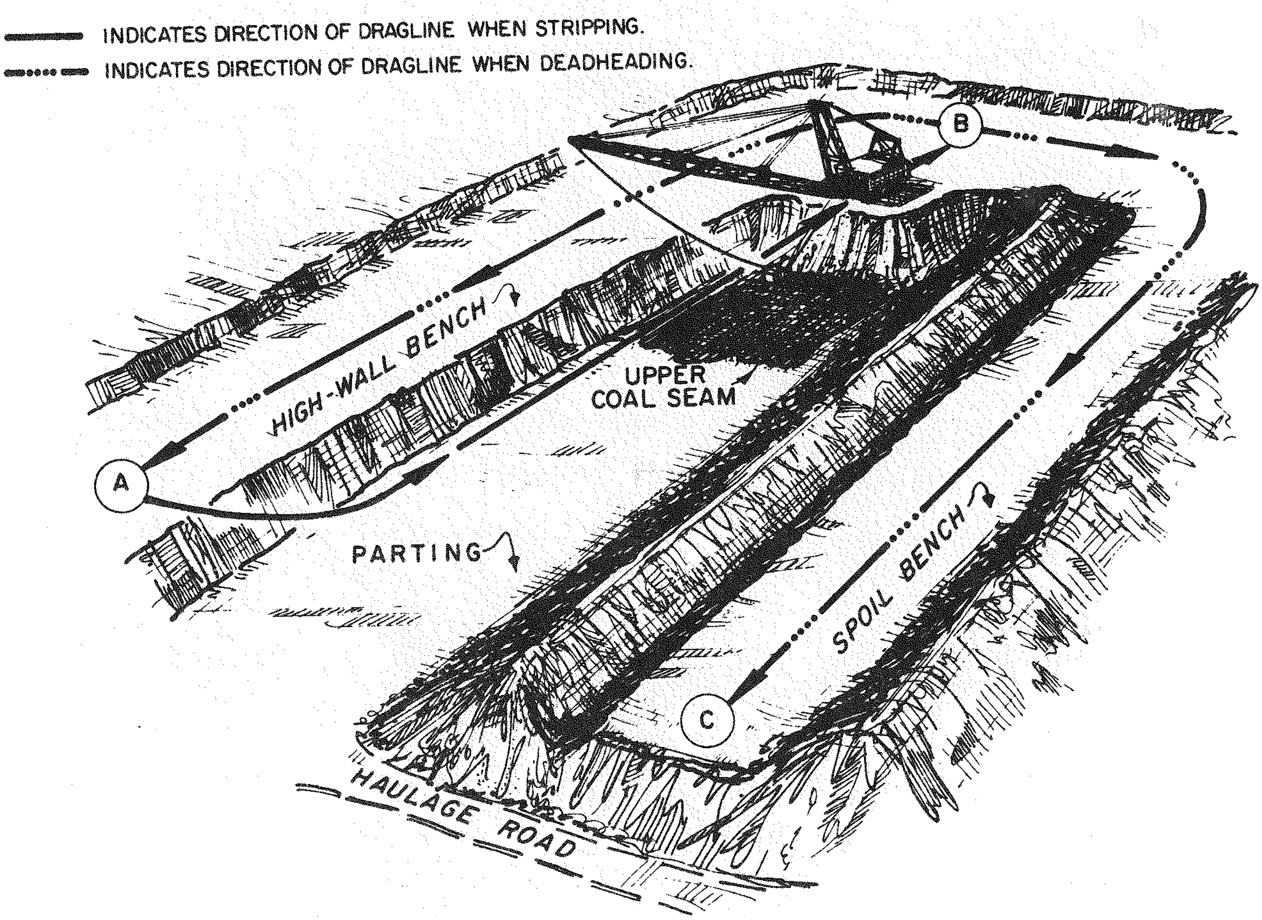
TABLE XV

WESTERN COAL STRIPPING FIELD STUDIES
DRAGLINE OPERATION SUMMARY (OVERBURDEN)

Mine	Machine Type	Bucket CY	Digging Mode (%)				Mech. Avail.%	BCY/* Op Hr	BCY/Op Hr per CY of Bucket	Cost/** Op Hr	Cost/ BCY	Remarks
			Casting	Parting	Chop-down	Rehandle						
MINE A	M-8200	75	54%	43%	-	6%	83-84%	3-4000	46.7	400	.11	325' boom 5' interburden removed by scrapers
MINE B	P-752	39	95-100%	-	0-5%	10-15%	90%	12.50	32.1	236	.19	
MINE D	B.E.-1260	40	97-100%	-	0-3%	8-15%	70-75%	1627	40.1	241	.15	
MINE C PIT 1	M-8000	62	100%	-	N/A	4-24%	61-75%	2130	34.3	327	.15	No records kept on chopdown. Parting taken by loader & trucks.
MINE C PIT 2	M-7820 P-752	32 32	100%	-	-	25%	77%	1200	37.5	204	.17	Zero chopdown. Average data for all 3 D/L's
MINE C PIT 3	B.E.-1570W P-736	78 18	67%	33%	-	0-12%	70%	2550	32.7	414	.16	Partings vary
MINE E	B.E.-1350	50	75%	25%	-	8%	91%	2440	48.7	285	.117	290' boom
	M-7900	50	75%	25%	-	8%	80%	2730	54.5	285	.105	260' boom
	M-7920	45	75%	25%	-	8%	87%	2350	52.2	260	.111	300' boom

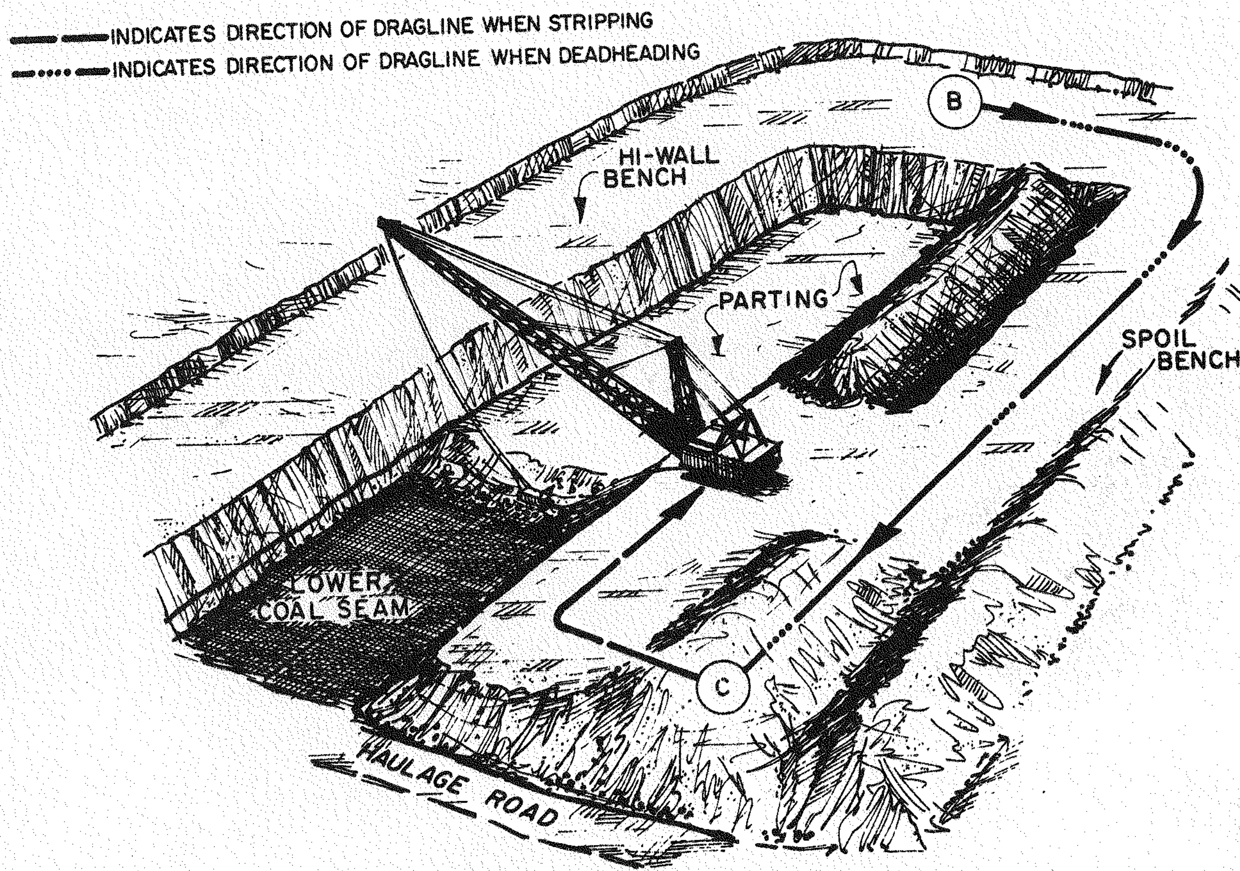
Rehandle - % of overall overburden
* BCY/hr. includes rehandle
** Cost of ownership excluded

NOTE: All cost information is estimated by authors based on available data.



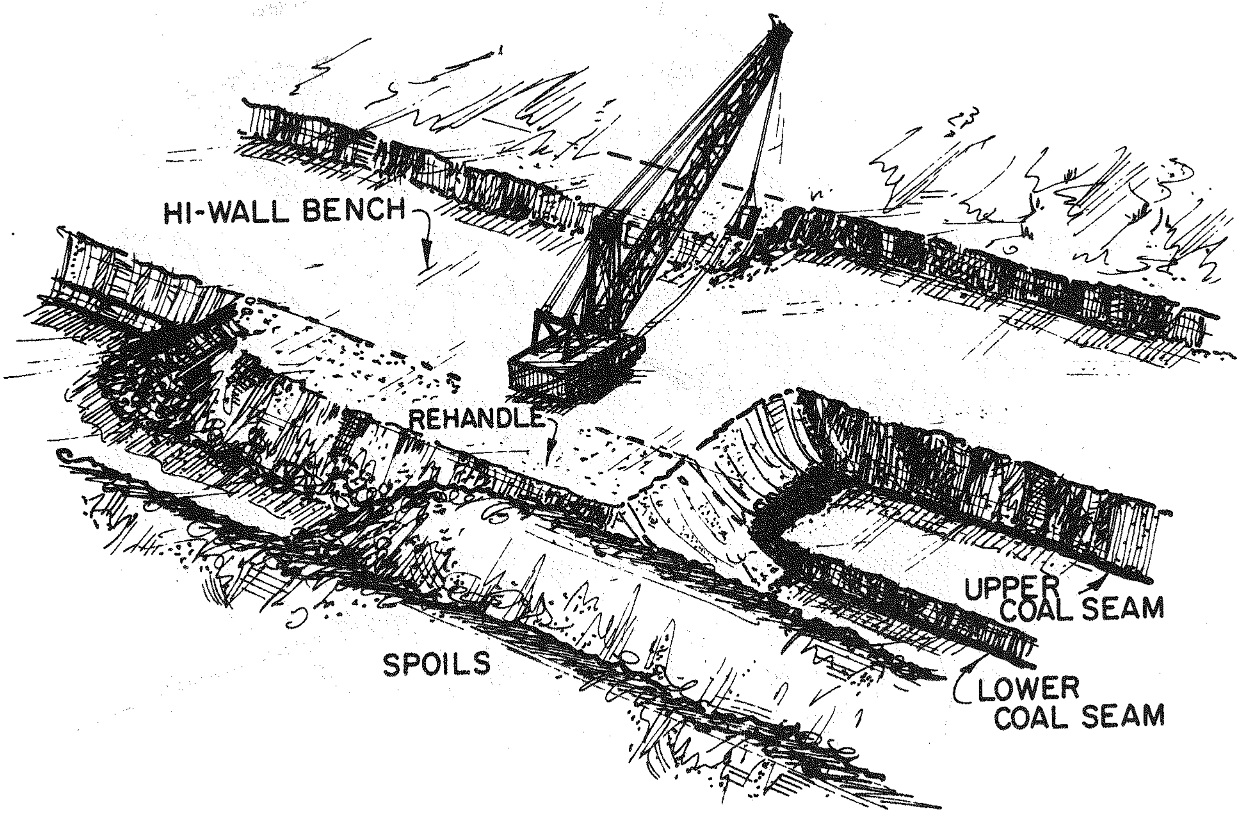
TYPICAL DRAGLINE CASTING OPERATION

Job No. 19089 Fig. 46

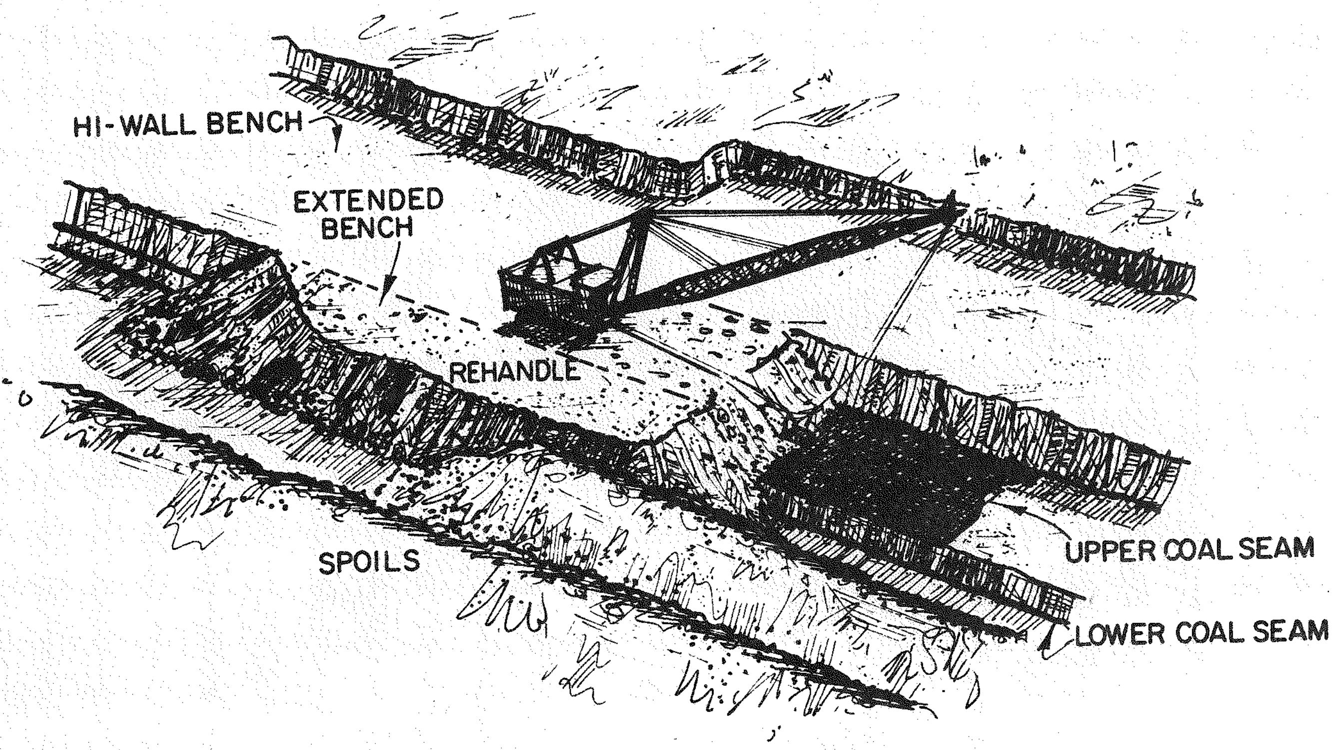


TYPICAL DRAGLINE PARTING REMOVAL OPERATION

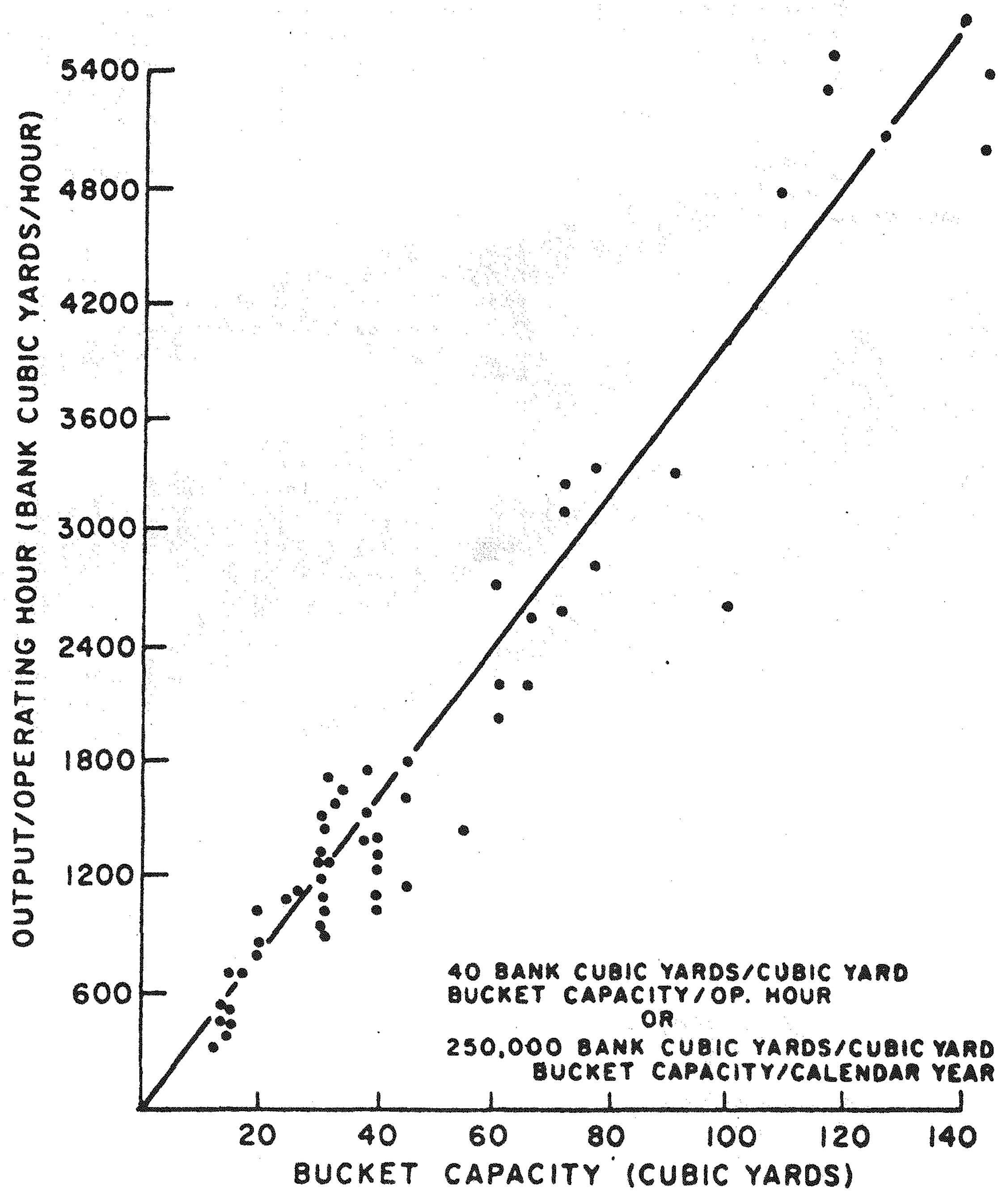
Job No. 19089 Fig. 47



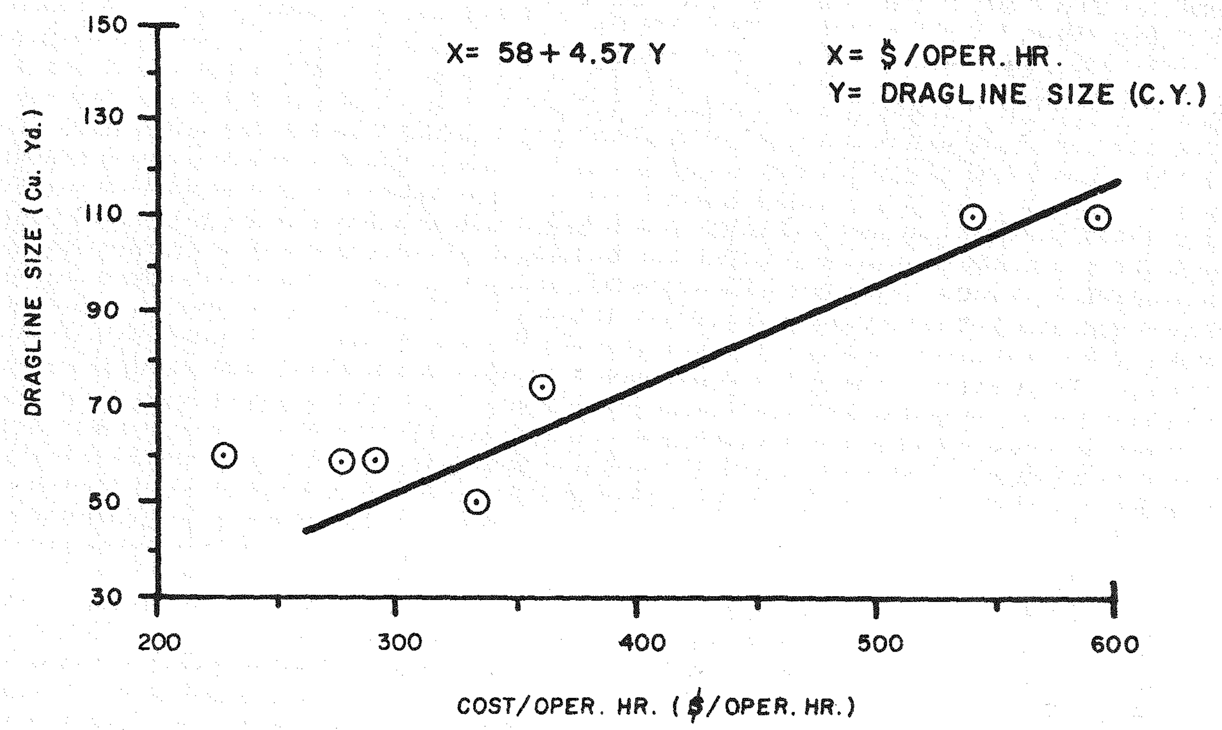
TYPICAL DRAGLINE BENCH CHOPDOWN OPERATION



TYPICAL DRAGLINE EXTENDED BENCH
REHANDLE OPERATION



BANK YARDAGE MOVED PER OPERATING HOUR FOR DRAGLINES OF VARIOUS CAPACITY



OPERATION AND MAINTENANCE COST
VERSUS DRAGLINE SIZE

Job No. 19089

Fig. 51

4.0 IMPROVED BLAST DESIGN CONCEPTS

Review of blasting theory and recent literature, as well as observations at the nine mines visited, indicates that blasting practices could be improved at most operations. However, much of this improvement could be achieved through implementation of more sophisticated standard procedures rather than any "new" concepts. Therefore, among the improved blast design concepts resulting from this study is the concept that more sophisticated standard blasting designs and procedures, although more expensive, could improve dragline productivity and decrease total costs.

In addition to the above, recommended improvements include the following key elements to improve overburden fragmentation, induce horizontal muckpile displacements and increase the blasting reliability:

1. Increase energy input per BCY by 40 to 100 percent and improve effectiveness of powder distribution both horizontally and vertically by modifying blast parameters including burden and spacing, collar heights, toe burden, delay intervals and blast tie-in.

2. Improve quality control in establishing shot patterns and in charging blast holes.

3. Induce blasthole dewatering, dryliners and bulk AN/FO usage for wet holes.

The use of blasthole dewatering, dryliners and bulk AN/FO is recommended over packaged explosives to improve blasting results in wet holes. In addition to higher cost ($.01 - $.05/lb), the disadvantages of packaged explosives are:

1. Excessive handling.

2. Decoupling effect in the hole.

3. Failure to use the full potential of a given hole diameter.

4. Questionable water proofing after the bags are dropped down the hole.

Item 3 implies that, for a given blasting pattern, smaller diameter holes charged with bulk AN/FO can be equivalent to larger holes charged with packaged explosives. For example Midwestern Mine A achieved a column rise of 30 lb/ft using packaged AN/FO in a 12 1/4-inch diameter hole. If the same hole were loaded with a bulk explosive, a column rise of 42 lb/ft would be expected. In effect a 9 7/8-inch hole, bulk loaded with AN/FO, would give the same column rise. The 9 7/8-inch hole would reduce drilling cost and eliminate the obvious decoupling effect. Decoupling will reduce the borehole pressure, which is related to fragmentation, by the following ratio:[23]

$$\text{Percent Reduction, (R)} = 100 - \left(\frac{\text{Dia of explosive}}{\text{Dia of the hole}}\right)^{2.6} \times 100$$

If the packaged explosive is considered to have an effective diameter of 10 inches and the hole diameter is 12 1/4 inches then the percent reduction in borehole pressure

$$R = 100 - \left(\frac{10}{12.25}\right)^{2.6} \times 100$$

$= 41\%$ (Note: Water in the hole will reduce the decoupling effect)

The use of collar casings is recommended where cuttings may be blown back in holes by the wind. (Experience indicates that such practice is an appropriate solution.)

More effective drill patterns represent the primary means of improving powder distribution. The use of square drill patterns with diagonal tie-ins is recommended in lieu of rectangular or diamond patterns with row-on-row or chevron tie-ins based on the following rationale:

1. The square pattern is simple to lay out in the field and presents a simple set-up sequence for the drill.

2. The square pattern gives maximum face hole frequency, which is important for muckpile displacement.

3. The diagonal tie-in will induce a certain degree of confinement to control vertical drop and will also yield a uniform windrow effect along the length of the blast, which is contrary to the results from row-on-row and chevron blasting. In addition, the chevron blast requires much more elaborate delay sequencing to reduce the number of holes fired per delay interval. This can be important where blast vibration and noise level are critical.

It is recommended that deck charging be eliminated or reduced to establish a simple loading and delay technique; however, experience during the test program may show that decking is performing an efficient function. Deck charge performance is highly dependent upon an accurate location of hard-to-blast strata in the overburden. The accuracy should be improved by meticulously logging and analyzing all drill parameters or by employing an automatic drill recorder. The hole depths should be taped when hard bands are encountered or a regular program of depth-counter calibration should be used. Slumping should be offset by lining the holes and using good stemming material such as 1/2-inch crushed stone.

By decreasing the excessive collar heights, consistently observed at the mines visited, the difficulties with fragmenting the upper strata should be alleviated. A decrease in collar height will increase the probability of flyrock and noise, but applying scaled distance laws that were developed for cratering theory (see Sections 2.2 and 2.3) should maintain these effects at acceptable levels. For laminated and prefractured rock, a scaled depth of burial of 4 should give satisfactory results (see Figure 8).

$$\frac{D}{(W)^{1/3}} = 4$$

D = distance to center of gravity of top 8 diameters

W = weight of explosive in top 8 diameters

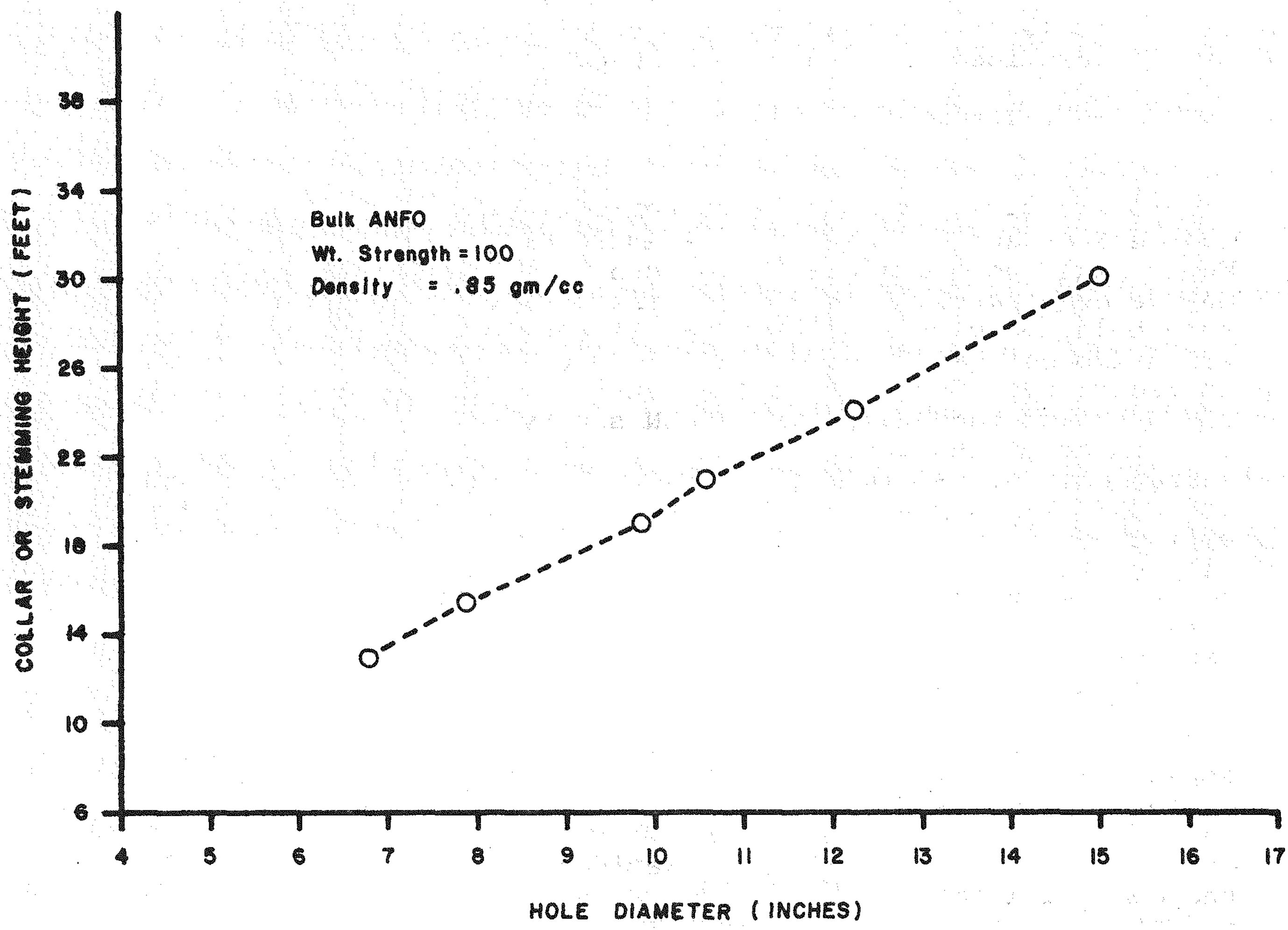
$$\text{Collar Height} = 4\,(W)^{1/3} - \frac{4\ (\text{Hole Diam.})}{12}$$

Collar Height in Feet
Hole Diameter in Inches

Collar heights have been plotted in Figure 52 for various hole diameters using bulk AN/FO.

At present, most operations visited have a toe burden well beyond the breaking capacity of the face holes. Such toe burdens act as the greatest deterrent to horizontal displacement. Crest holes should be drilled closer to the face or high density slurry toe charges be employed.

For example, if 15-inch diameter holes were used and the hole depth was 75 feet (slope angle 65°), then face holes should be drilled as close to the crest as possible if AN/FO is used.



COLLAR HEIGHT VS. HOLE DIAMETER
IN LAMINATED & PREFRACTURED ROCK

e.g. Toe Burden = Cot 65° (Height)
= 35 feet

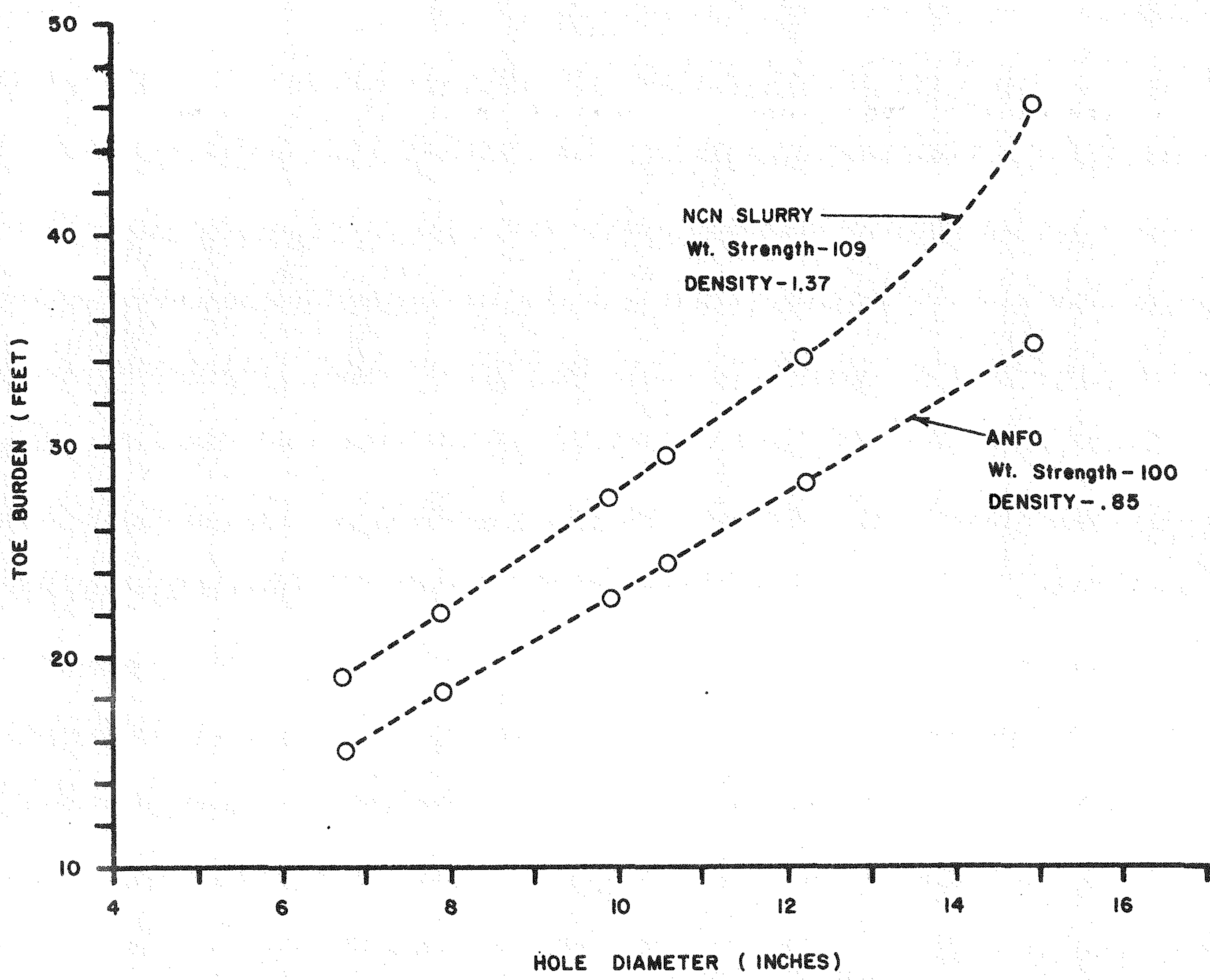
or 11 to 12 feet from the crest if slurry such as described in Figure 53 is used.

e.g. Toe Burden = 35 + 11
= 46 feet

In general, it is more economic to move the holes closer to the face and use AN/FO or Al/AN/FO rather than slurry. Where small hole diameters are used on large face heights, it may be necessary to use a high density slurry even with the face holes as close to the crest as possible. Scaling laws will also be used to determine toe burden and have been shown in Figure 53. Two curves have been developed, one for the use of AN/FO and the other for an NCN slurry with a density of 1.37 gm/cu cm and a weight strength of 109. The greatest toe burden encountered in a blast is that on the face holes. Once the blast is in progress, the drill pattern and tie-in used will dictate the burden on the remainder of the holes. It will be the intent of this program to modify the face hole geometry so that the vertical drop and "feather edge" of the blast is not excessive. To achieve this end, it will be necessary to reduce the toe burden in gradual increments until a satisfactory muckpile is obtained.

As noted in the previous section, the drilling yield appears to be a good measure of the effectiveness of the explosive distribution, i.e., the greater the yield, the poorer the distribution. The cost relationship between drilling and blasting will be examined for each property and reduced drill patterns developed with a corresponding scaled reduction in hole diameter so the powder factor remains constant. The advantages of improved fragmentation with this approach will be weighed against the required changes in drill string size and existing equipment capabilities. The reduction in drilling yield may also present a drill capacity deficiency. It should be noted however, based on drill scheduling and utilization, that a surplus capacity appeared to be available at most operations.

The most radical blasting concept proposed is that of horizontal muckpile displacement. Horizontal displacement is required before blasting can effectively disturb the natural stratified structure of the overburden to improve bucket penetration. Desired displacement may cause the dragline to sit 10 to 15 feet below the insitu highwall. Consequently, the dragline or dozers will be required to level off the peak of



TOE BURDEN VS. HOLE DIAMETER
IN LAMINATED & PREFRACTURED ROCK

the muckpile, ramps will be needed when deadheading, and minor modifications to the dragline working range will be indicated.

Figures 54 and 55 illustrate by sectional and plan view the required sequence for a dragline when deadheading for a new cut. In this case, the dragline does its own levelling of the blasted overburden. Figures 56 and 57 show the same situation, but in this case the levelling of the blasted overburden is done by dozer.

Discussions with strip mine operators indicated that none of these items should pose a major difficulty and the inconvenience is likely to be offset by the extra overburden spoiled (moved to spoil bank) by the displacement. Because spoiling of overburden by the blast could significantly reduce the volume of stripping required, the cost benefits may more than make up for such inconveniences as extra flyrock, longer equipment moves, ramp construction and muckpile preparation.

5.0 GENERAL FIELD EVALUATION REQUIREMENTS

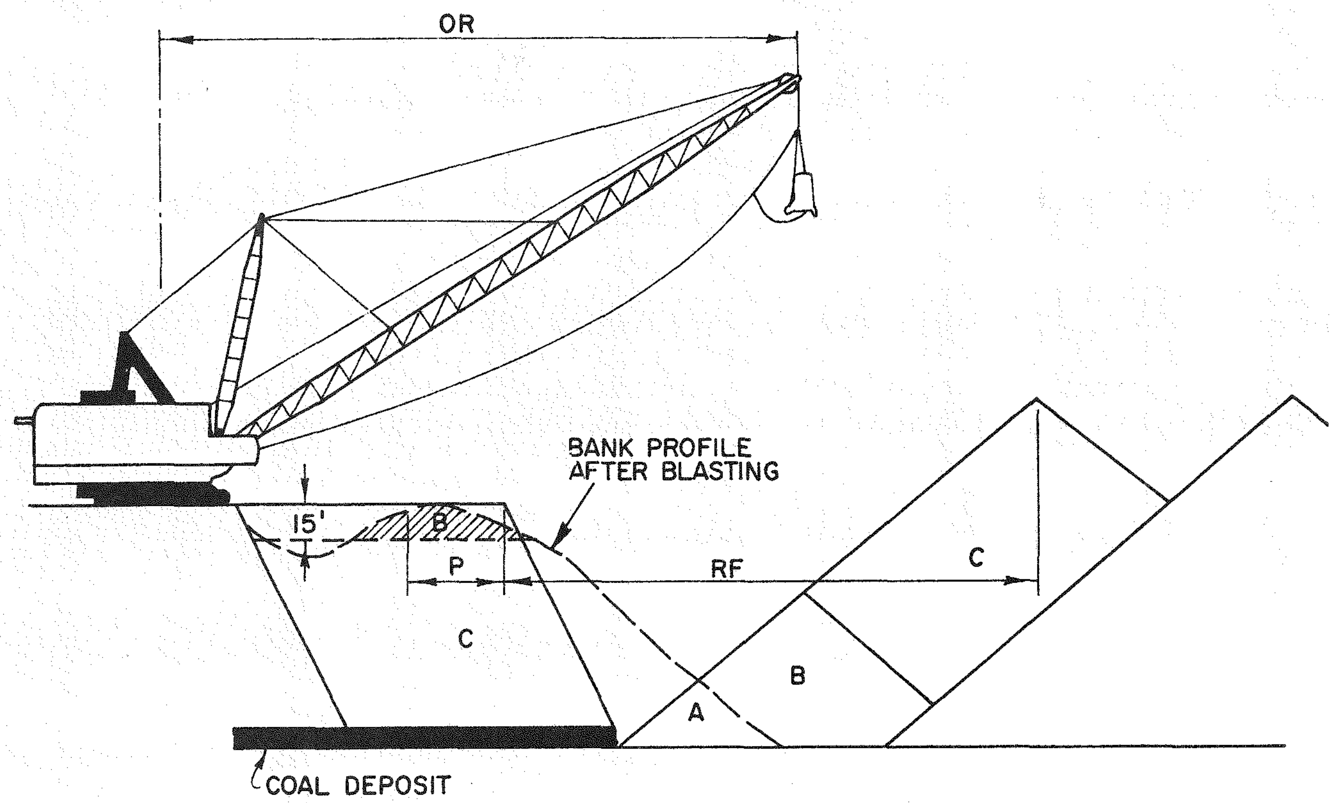
5.1 General

After the test program has been accepted by the operators, comprehensive programs of analysis, measurement and calculation can be conducted. Each property is to be revisited, and detailed cost, operating parameters, and performance data are to be compiled and analyzed in such a manner that they can be appraised in a "before-and-after" manner.

5.2 Selection of Test Mines

The mine operations shown in Table XVI were listed in order of their priority for conducting blasting test programs. This ranking was based on the operations' relative dragline productivity. In addition, an appraisal was made of the improvement potential of this productivity if drilling and blasting-related problems were corrected.

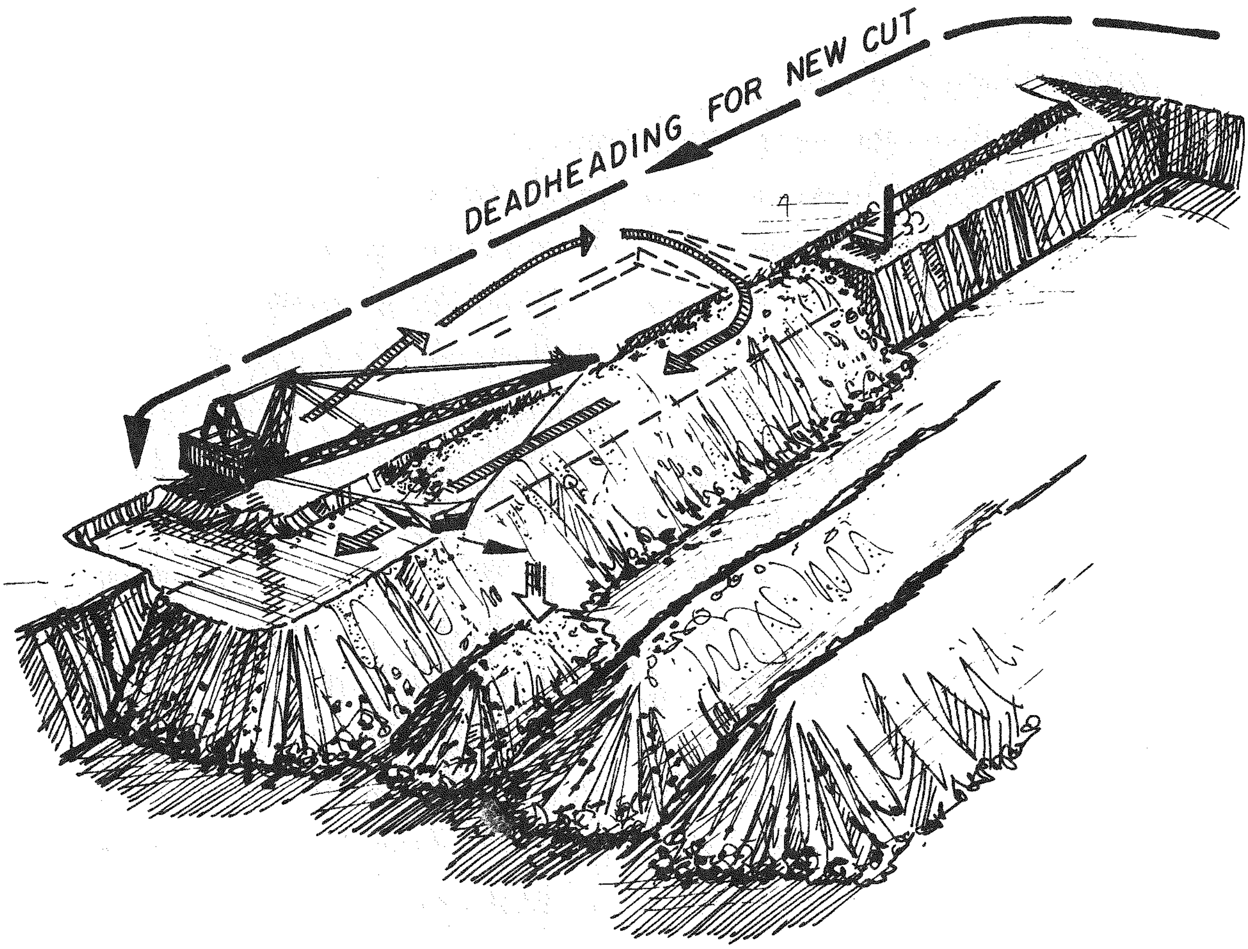
The cost distribution (including ownership) among drilling, blasting and the dragline operation were also important in deciding the order of priority. Where the cost was substantial the greatest potential existed to modify drilling and blasting methods to improve dragline productivity and lower costs.



LEGEND

OR = OPERATING RADIUS
P = POSITIONING
RF = REACH FACTOR
A = MATERIAL SPOILED BY BLAST MOVEMENT
B = MATERIAL SPOILED TO LEVEL TOP OF OB
C = REMAINDER OF OB TO BE SPOILED IN NORMAL DRAGLINE CASTING

DRAGLINE LEVELLING TOP OF BLASTED OVERBURDEN



DRAGLINE DEADHEADING SEQUENCE FOR NEW CUT

Job No. 19089

Fig. 55

D-9

PRODUCTION = 800 CY/HR (MAX PROD x CORRECTION FACTORS)

ASSUME 10% OF OVERBURDEN MUST BE PUSHED

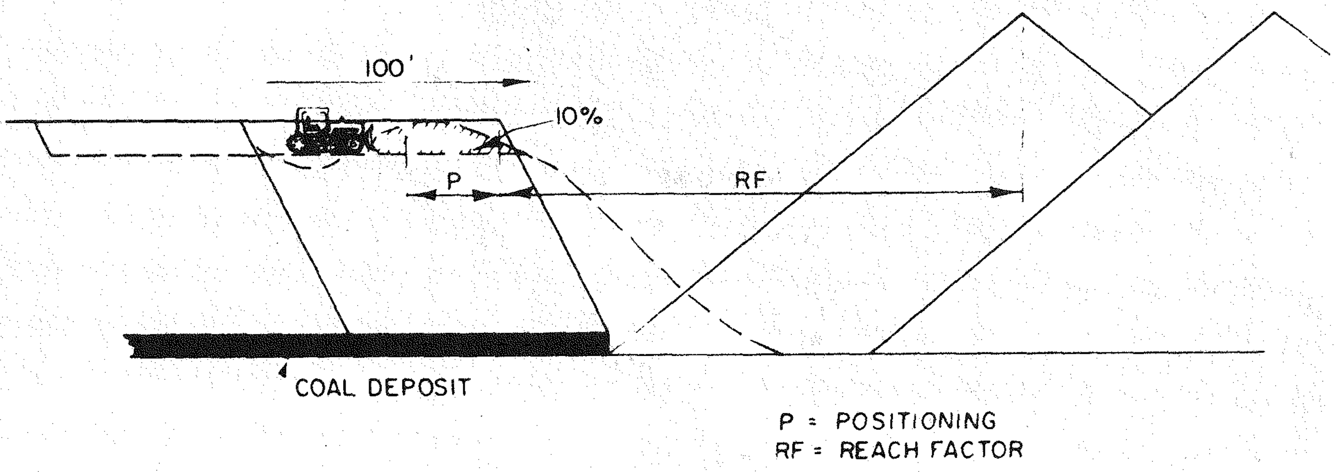
EXAMPLE; 100,000 CY/1,000,000 CY OVERBURDEN

OPER HR. 100,000/800 = 125

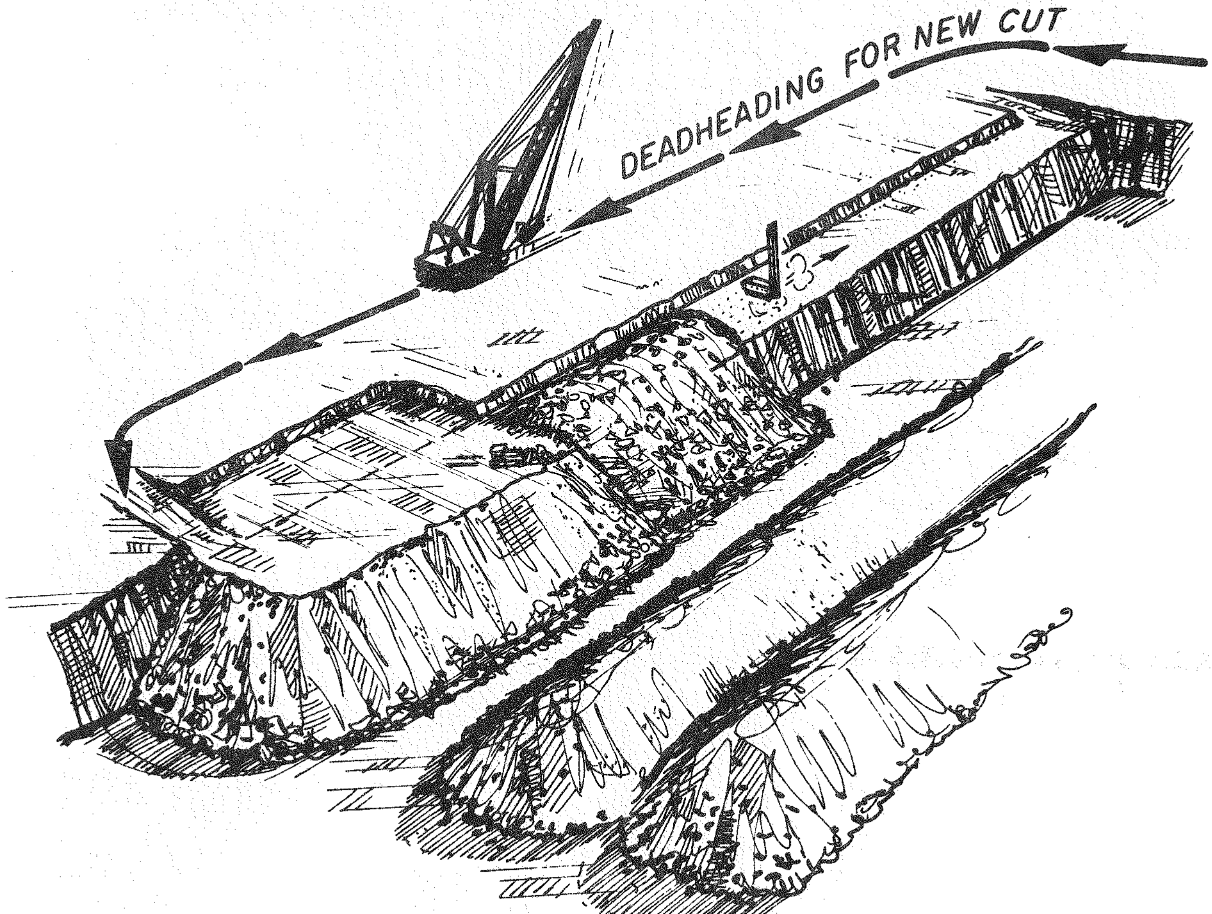
OPER & OWNERSHIP COST ≈ \$70/OPER HR

TOTAL COST = 125 x \$70 = \$8750

$$\text{UNIT COST (TOTAL OVERBURDEN)} = \frac{8750}{1,000,000} = \$.009/\text{BCY}$$



LEVELLING TOP OF BLASTED OVERBURDEN WITH TRACTOR



DRAGLINE DEADHEADING SEQUENCE WHEN
TRACTOR USED TO LEVEL BLASTED OVERBURDEN

Job No. 19089

TABLE XVI

SELECTION OF MINES FOR TEST PROGRAM

Priority	Mine	Dragline Prod BCY/OP HR/CY of Bucket	Drilling	Explosive Loading	Tie-In & Delays	Comments
1.	Midwestern Mine A	28	- Toe burden - Crest burden	- All packaged ANFO - Powder factor - Decoupling - Water	- Short delays - B/S ratio	Receptive to a presentation of a test program.
2.	Midwestern Mine B	30	- Patterns too large - Toe and crest burden	- Collar height - Decking correlation - Powder factor	- Short delays - B/S ratio - Shooting direction	Receptive to test program and good cooperation anticipated
3.	Western Mine C	32	- Poor control - Small hole diameter - Toe and crest burden	- Hard bands over coal - Powder factor	- Structure orientation - Short delays - Shooting direction	Receptive to test program and good cooperation anticipated
4.	Midwestern Mine D	32	- Toe and crest burden	- Collar heights - Powder factor - Correlation of deck charges	- Shooting direction	Receptive to test program and good cooperation anticipated
5.	Western Mine B	32	- Toe and crest burden	- Hard material over coal - Powder factor	- Shooting direction - Short delays	1 1/2 years overburden reserve. Receptive to test program and good cooperation anticipated
6.	Western Mine A	46			- Buffer blasting	Good fragmentation - Possible problem for test program - dual ownership and management.
7.	Western Mine E	52			- B/S ratio - Short delays	Excellent fragmentation
8.	Western Mine D	41	- Toe and crest burden	- Powder factor	- Structure - B/S ratio	Good fragmentation at present (Not interested in test program.)
9.	Midwestern Mine C	37		- Wet holes	- Some blasts have short delays	Not interested in test program.

5.3 Cooperative Agreement

The mine management's acceptance of a test program and the degree of cooperation anticipated were also considered as key to the success of the test program. The proposed cooperative agreement between the mine operators and Woodward-Clyde Consultants for conducting the test programs is presented in Appendix B.

5.4 Information to be Compiled

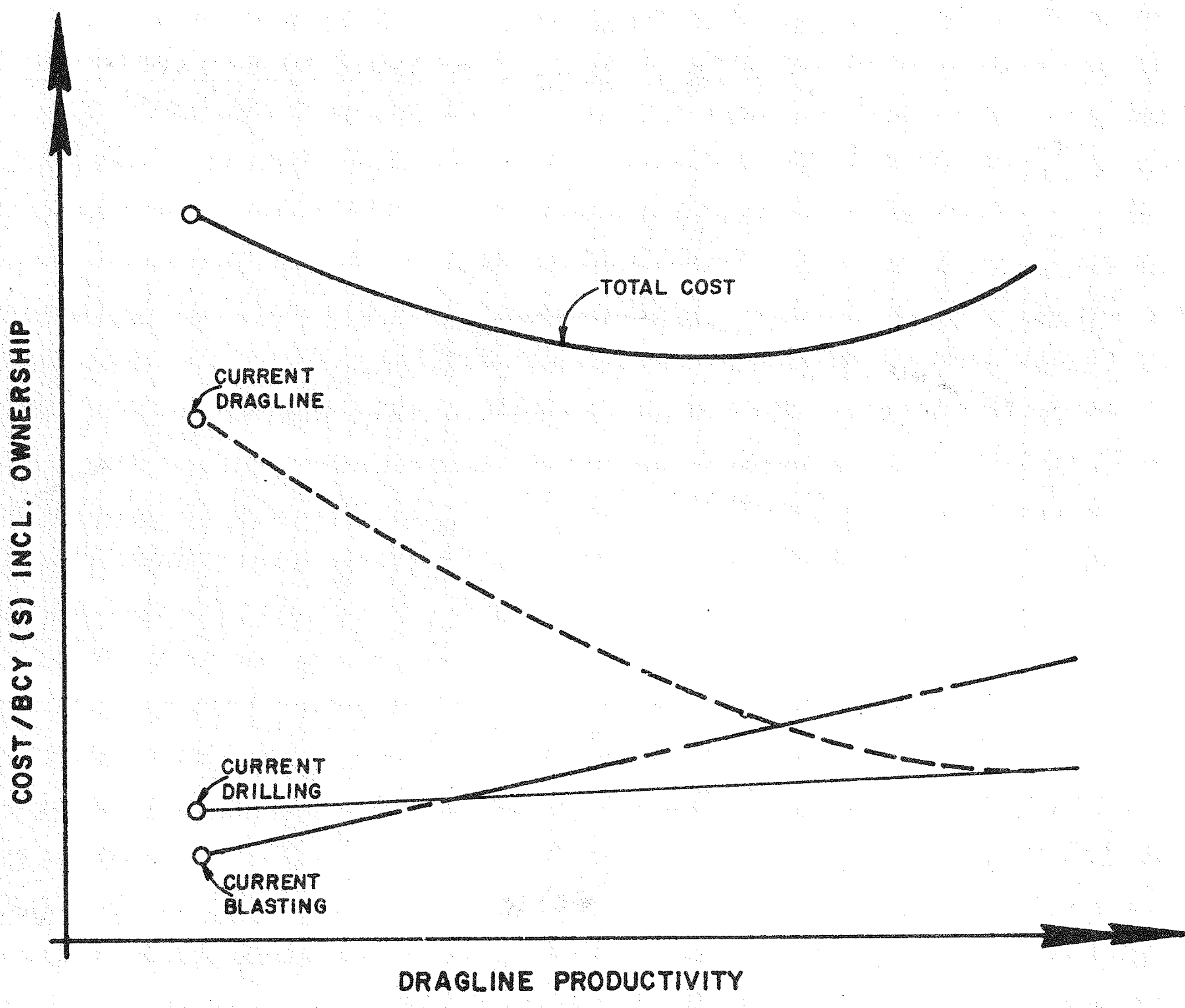
Since the objective of the test is to relate dragline productivity and stripping costs to overburden fragmentation, the following information is considered necessary:

(1) Costs: The operating, maintenance and administrative costs, with a reasonable estimate of ownership cost, must be determined for drilling, blasting and dragline operations. A graph of unit cost versus dragline productivity should be prepared with an estimated projected trend (Figure 58). As the program progresses, new cost data should be plotted.

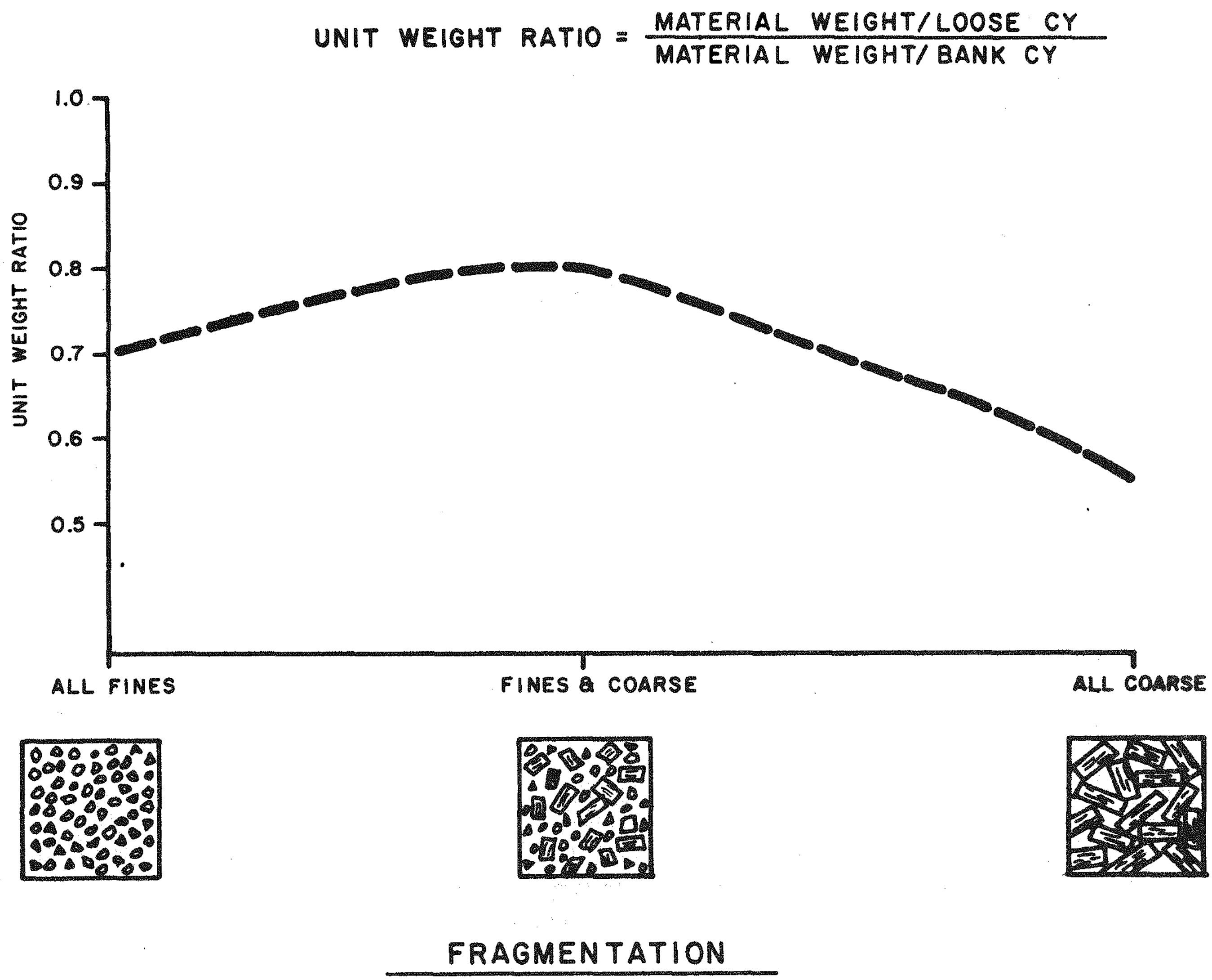
(2) Dragline Productivity: Dragline productivity should be measured for each of the digging modes, such as casting, bench chopdown, and parting. The actual measurements should be based on surveyed BCY as a function of dragline operating hours and KWH as compiled by the onboard meter. Improved overburden fragmentation should increase dragline productivity and capacity in four ways as listed below:

a) Digging time: The basic elements of a dragline work cycle are dig, swing to spoil, dump and swing to bank. The last three elements will not be improved by fragmentation, but the digging element will, and this represents 25 percent of a typical work cycle. Time studies will determine improvements made in this area.

b) Unit Weight Ratio: Blasted sandstone, shale and limestone normally have unit weight ratios ranging from .55 to .80 depending on the size distribution of fine and coarse material. Either extreme, all fines or all coarse, does not give the maximum ratio. The maximum lies between the extremes depending on size and shape. Figure 59 illustrates an idealized unit weight ratio range as a function of rock-size distribution. At present, the ratio approaches



TYPICAL UNIT COST VS. DRAGLINE
PRODUCTIVITY PROJECTION



TYPICAL UNIT WEIGHT RATIOS IN ROCK

FOR VARIOUS DEGREES OF FRAGMENTATION

Job No. 19089

Fig. 59

the "all coarse" value. A minimum increase in the volume of rock when broken is desirable, which occurs when the distribution of fine and course material results in the maximum ratio. An improvement in this ratio should result in an increase of capacity in BCY per work cycle and reduce the height of spoil piles.

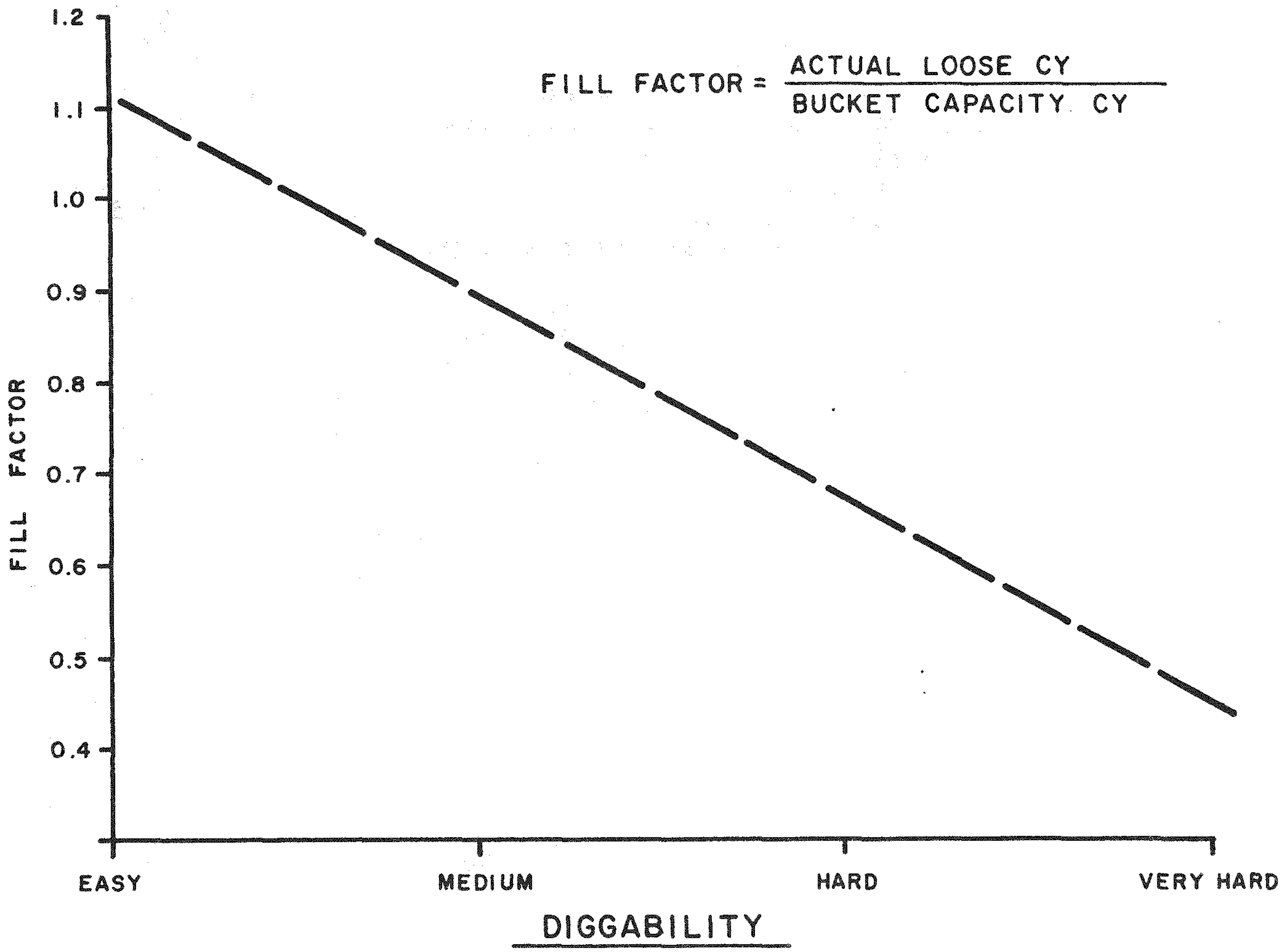
c) Fill Factor: This item will probably cause the greatest improvement in dragline productivity. The fill factor is the ratio of actual loose CY in the bucket versus the bucket rating in CY, and can vary from 0.4 to 1.10, depending on the ease with which material will flow into the bucket. The deeper the bucket teeth penetrate the bank, the greater the fill factor. Figure 60 gives a typical curve of fill factor versus "diggability" for a dragline operating in rock.

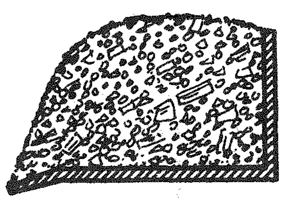
d) Horizontal Blast Displacement: If improved fragmentation is to be achieved, a reasonable amount of horizontal displacement of the blasted overburden is anticipated, as illustrated in Figure 61. Dragline productivity would not be improved due to the displacement, but the annual capacity for spoiling overburden would increase because of the spoiling effect caused by the displacement. This value is best calculated by field survey after blasting.

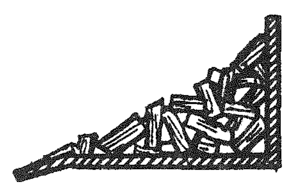
3. Other Items of Measurement:

a) Blast Displacement of Overburden: Increased displacement of blasted overburden will result in the dragline working on an elevation slightly lower than the insitu highwall (estimated at 10 to 15 ft). For this reason, a typical muckpile profile should be surveyed and new dragline range diagrams constructed (Figure 62). In addition, a provision for ramping up to the insitu highwall should be made at the end of each cut.

b) Highwall Stability: Highwall instability should not result when the explosive energy input per BCY is increased if adequate delays and proper direction of shooting are utilized. However, a survey of highwall slope angle and a visual stability assessment should be made of present highwalls and of the highwalls after each phase of the test program.







TYPICAL DRAGLINE FILL FACTORS VERSUS
OVERBURDEN DIGGABILITY

Job No. 19089 Fig. 60

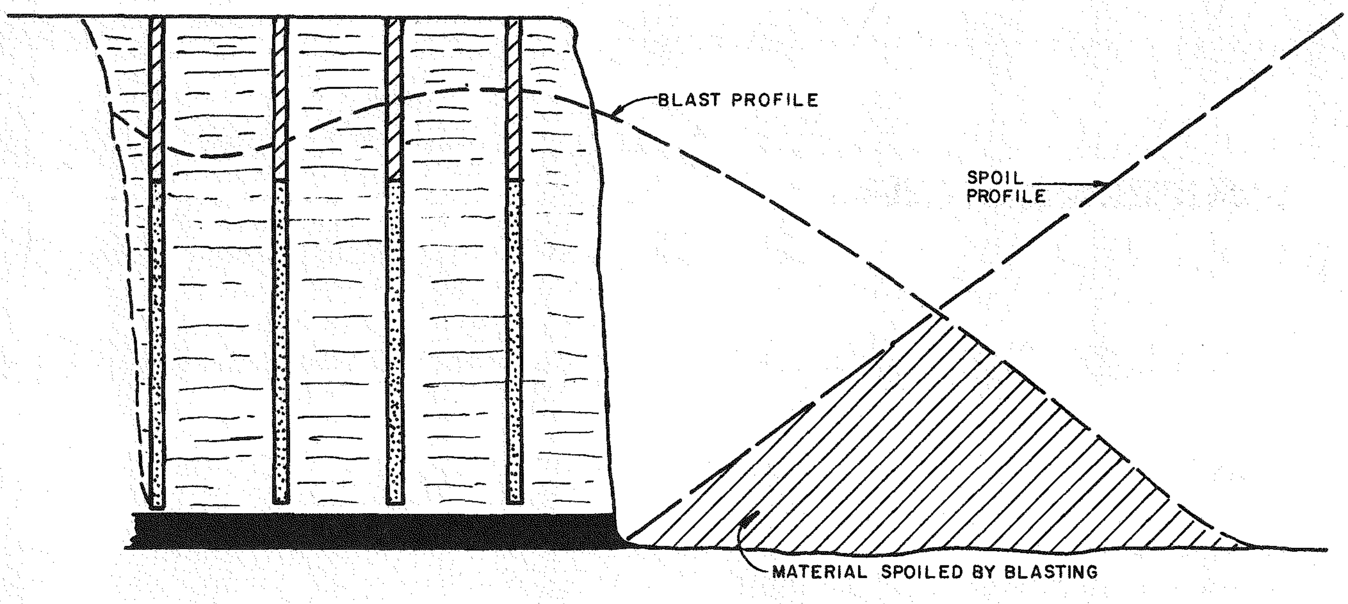
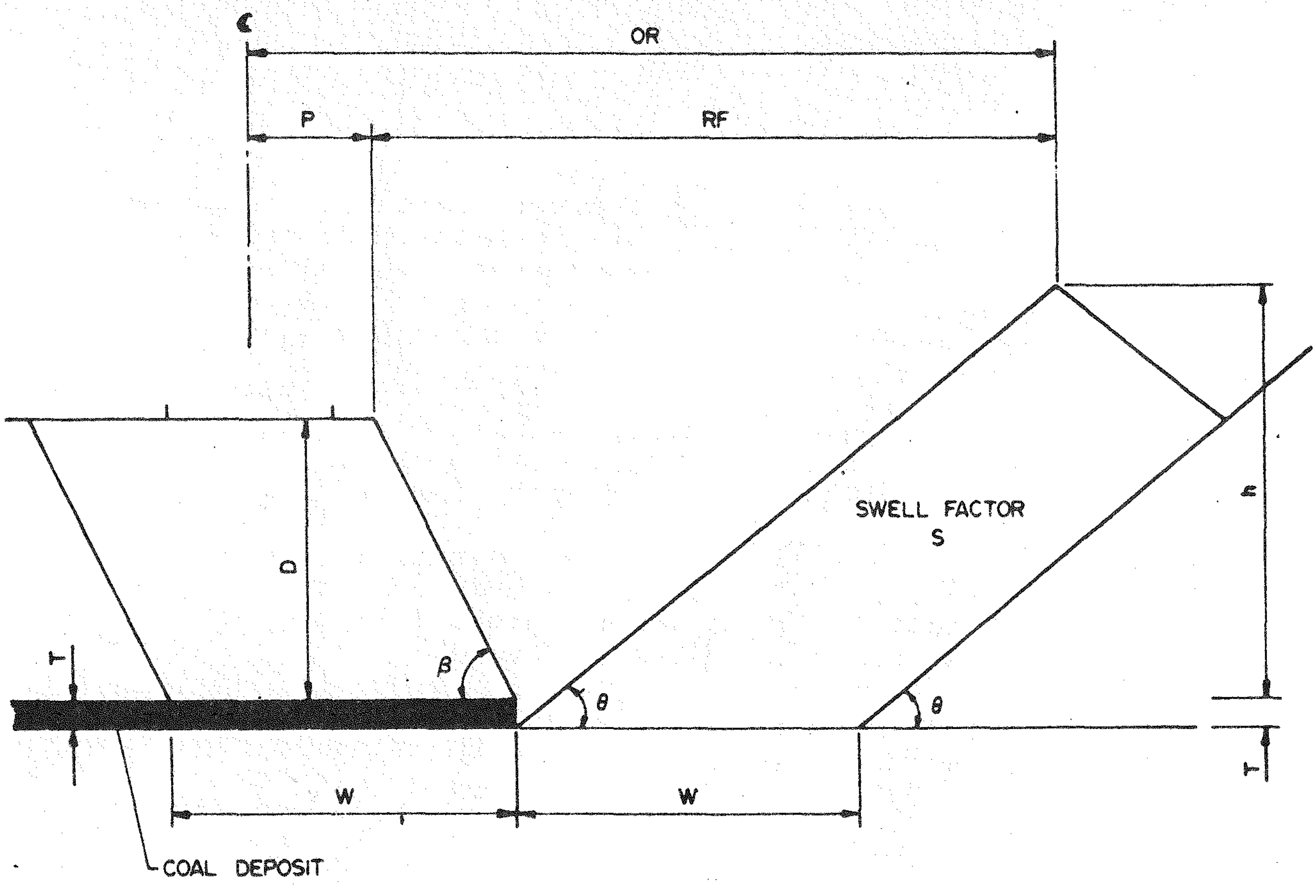


ILLUSTRATION OF OVERBURDEN
SPOILED BY BLASTING ACTION



LEGEND

P = POSITION
OR = OPERATING RADIUS
RF = REACH FACTOR
h = SPOIL PILE HEIGHT TO PEAK
θ = SPOIL PILE ANGLE OF REPOSE
β = HIGHWALL ANGLE OF REPOSE
D = DEPTH OF OVERBURDEN
W = WIDTH OF CUT
T = COAL THICKNESS

SIMPLE SIDE CASTING RANGE DIAGRAM SHOWING RELEVANT GEOMETRICAL PARAMETERS

Job No. 19089

Fig. 62

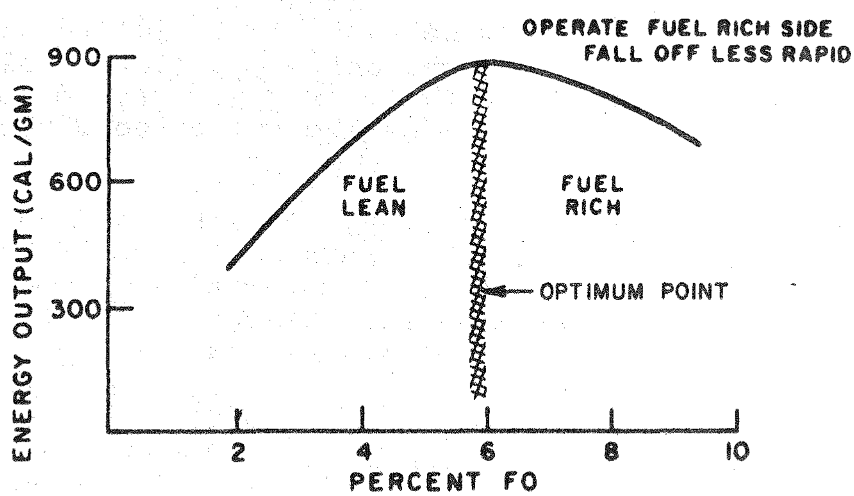
c) Correlation of Drill Report: Where drill reports or logs are used to guide placement of deck charges, depths to "hard" strata should be measured and correlated with drill logs. In many cases penetration rate can be constant over the depths of the hole, even though rock hardness varies (e.g. in soft ground, penetration rate is limited by bailing velocity, and, in hard ground, penetration rate is maintained by increasing pull down and rpm).

d) Blast Vibration and Noise: Measurements of blast vibration and noise levels should be recorded. All test blasts should be precalculated for acceptable levels and monitored.

e) Jointing: A complete survey of joint orientations and frequency, using a phototheodolite plus photogrammetry techniques, should be employed to determine the preferred direction of blasting.

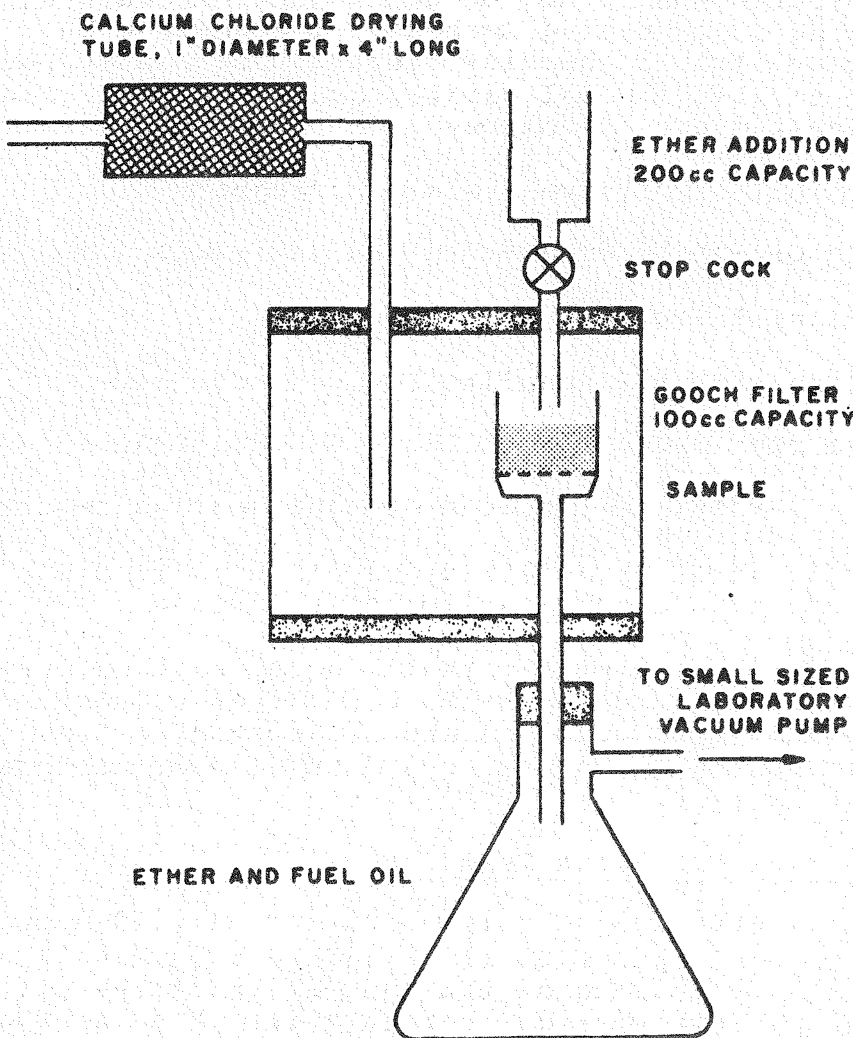
f) High Speed Photography: A high-speed 16mm camera (300 to 500 frames/sec), should be used to determine the delay interval (minimum and maximum), stemming efficiency, collar height and toe burden. These data can materially aid the blast designs.

g) Explosives Quality Control: In addition to periodic checks on the reliability of detonating cord and primers, it is very important to test the percent of fuel oil in the AN/FO. Figure 63 shows the effect on the energy output of different percentages of fuel oil added to AN. On both sides of the oxygen balanced mixture a "fall off" occurs in the energy output. This "fall off" is more pronounced on the fuel-lean side. Because of this, most operators run the mixture slightly fuel-rich. Figure 64 shows the type of extraction apparatus used for fuel oil determination. Samples are taken on a random basis from the bulk mix truck and diethyl ether is used to extract the fuel oil. The weight of the sample before and after the extraction gives the weight and percentage of fuel oil. The drying tube is used so that the AN will not absorb moisture from the air, as large quantities of air are pumped through the apparatus to eliminate residual ether.



ENERGY OUTPUT VS. PERCENT FUEL OIL
ADDED TO AMMONIUM NITRATE

Job No. 19089 Fig. 63



SCHEMATIC OF APPARATUS FOR FUEL
OIL DETERMINATION

Job No. 19089

Fig. 64

6.0 SPECIFIC BLAST TEST PROGRAM

6.1 General

The test program should not be contingent on changing existing equipment or operating methods. Changes can be suggested and presented to the mine operator for his approval, but they should involve minimal capital cost. Typical changes that may be suggested are as follows:

1. Compare bulk explosives versus packaged;
2. Utilize blasthole dewatering and dryliners;
3. Change drill string;
4. Establish explosive quality control tests;
5. Improve cost accounting methods as required;
6. Improve performance records of equipment as required;
7. Improve control of drilling and blasting such as drill-pattern layout and explosive loading.

The personnel conducting the test program must be prepared to justify all changes by technical and/or economic advantage. One change in drilling and blasting at a time should be tested, and each change should be assessed before proceeding with the next.

6.2 Midwestern Mine A - Blast Test Program

Two draglines are used at this operation, a 2570-W and a 1550-W. They are equippped with 110CY and 50CY buckets, respectively. The overburden at the 1550-W end of the pit is drilled by a 12 1/4-inch vertical rotary drill at an average hole depth of 75 to 85 feet to the first coal seam. Drilling at the 2570-W end of the pit is accomplished with horizontal holes approximately 90 feet in depth. Both draglines bench 15 to 25 feet of soil, using a chopdown mode of digging, and work off the top of an unconsolidated glacial till which is 20 to 30 feet deep. Beneath this glacial till lies a 10 to 15 feet thick cap rock of limestone which should be considered as the collar horizon for blasting purposes. A well-fractured shaley limestone lies between the cap rock and upper coal seam and varies in thickness from 20 feet at the 2570-W end of the pit to 40 to 50 feet at the 1550-W end.

Considerable moisture is present in the overburden and for this reason packaged AN/FO is used. This raises the first point in the blast test program. Supervisors at the operation note that incomplete detonation of the explosive is occurring as evidenced by substantial amounts of yellow-brown fumes. This indicates that bags of AN/FO are breaking when they are dropped into the vertical holes or when they are rammed into the horizontal holes, which allows moisture to infiltrate the AN/FO. This situation is further compounded by a 1- to 3-month duration between loading and initiation.

6.2.1 Reduce Time Lag Between Loading And Initiation or Use Dryliners And Bulk AN/FO.

Dryliners are preferred, since the cost difference is negligible. In this regard, it is also important to note that packaged AN/FO reduces the weight per foot of hole by 30 percent so a 12 1/4-inch hole with packaged AN/FO is equivalent to a 9 7/8-inch hole that is bulk loaded. This represents an estimated $.01/BCY saving on bits, stem stabilizers, and other consumables. Since the two drilling methods are so different, a separate test program for each is required.

6.2.2 Blast Design for 12 1/4-Inch Diameter Vertical Holes

At present, a 34 by 34 foot pattern is drilled to full depth and the same pattern of interlaced short holes is drilled to the bottom of the limestone cap rock. This results in an overall drill yield of 35 BCY per foot of hole. These short holes can be eliminated by decreasing the pattern for the full-depth holes to 32 by 32 feet, resulting in a drilling yield of 38 BCY per ft. The decrease in drilling cost will be approximately $.008/BCY.

STEP 1 - ELIMINATE SHORT HOLES AND DRILL 32 BY 32 FOOT FULL DEPTH PATTERN

This pattern is arrived at by using scaled distance laws for a cylindrical charge and a scaled burden of 3.5, as outlined in the blast theory section of this report.

$$\frac{\text{Burden, B (ft)}}{(\text{Weight of Explosive, W, lb/ft of Hole})^{1/2}} = 3.5$$

$$B = 3.5\ (42)^{1/2}$$

$$= 22.7 \text{ ft (Blasting Burden)}$$

For a square pattern tied-in on the diagonal (B/S=1:2), then the drill pattern (x) is given by

$$X^2 = \frac{(2 \times B)^2}{2}$$

$$X = \frac{(45.4)}{2} = 32.107, \text{ say } 32 \text{ feet}$$

The drill pattern = 32 by 32 feet

In addition to the diagonal tie-in of square pattern (V_1 configuration), a staggered square pattern tied in on the diagonal (V_2 configuration) can be used. Still another approach is to drill a rectangular or staggered-rectangular pattern with a 22 1/2-foot burden and 45-foot spacing. The two former patterns are preferred, based on the rationale outlined in the blasting theory section. Figure 65 illustrates all four of these patterns and tie-ins.

STEP 2 - BULK LOAD ALL HOLES WITH AN/FO TO A DISTANCE OF 15 1/2 FEET BELOW THE TOP OF THE LIMESTONE CAP ROCK

Collar or stemming heights can be calculated assuming the top 8 diameters as a spherical charge and using scaled distance laws for cratering and a scaled depth of 2.8 where the limestone cap rock is the hard stratum, e.g:

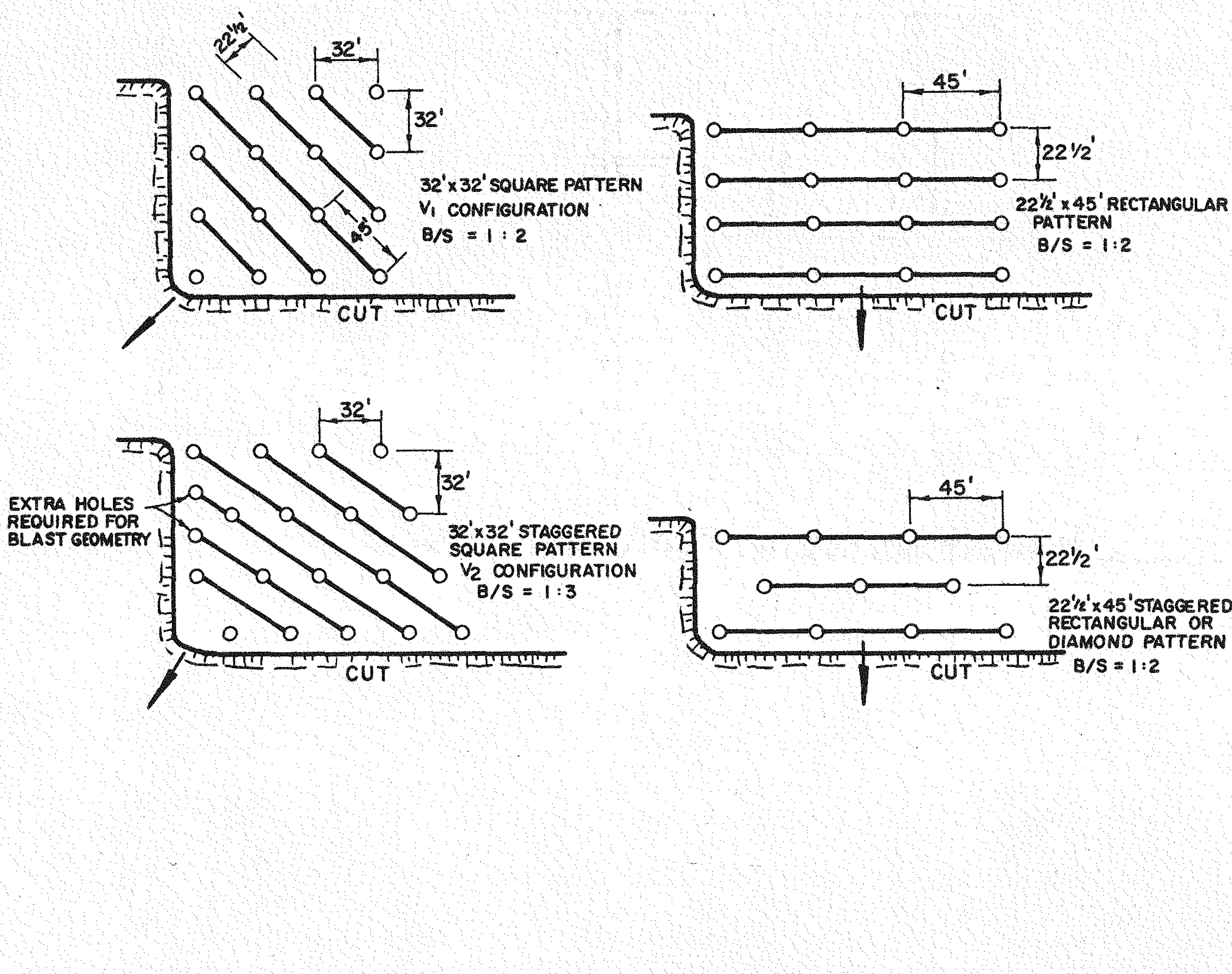
$$\frac{\text{Depth (ft)}}{(\text{Weight of Top 8 Dia. of Explosive, lb})^{1/3}} = 2.8$$

Where Depth = depth of burial to center of gravity of top 8 diameters in feet, and

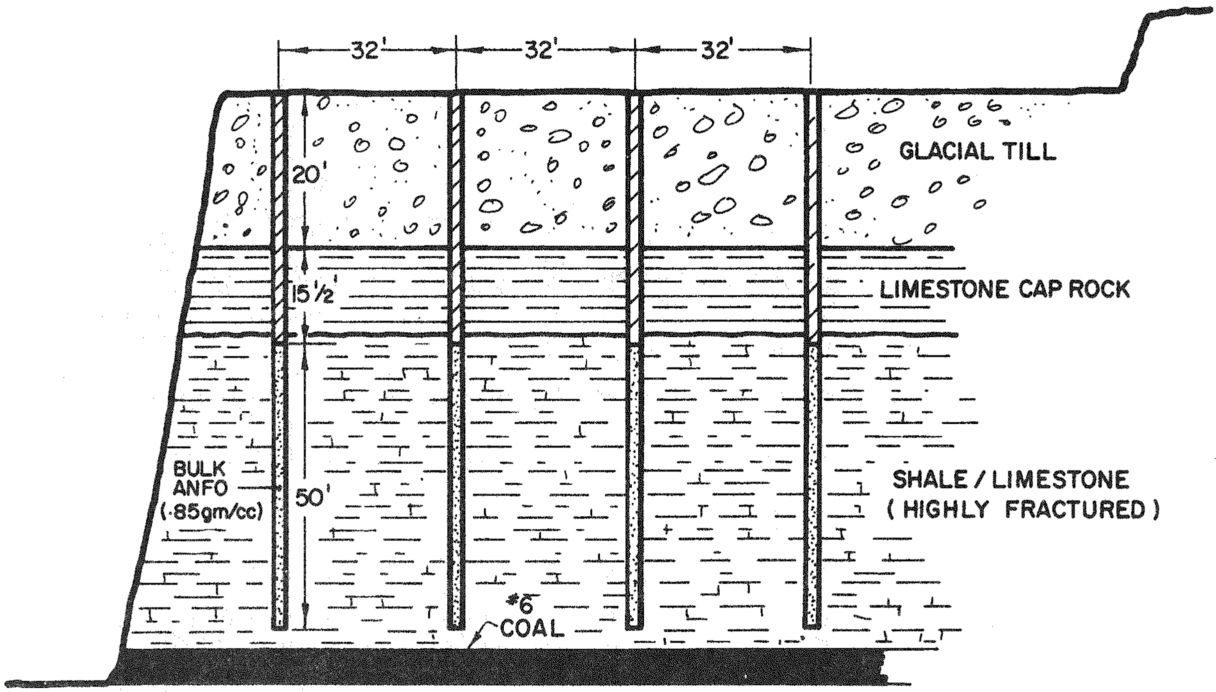
$$\text{Collar Height} = 2.8\ (8 \times 43)^{1/3} - \frac{4(12.25)}{12}$$

$$= 15.5 \text{ feet}$$

Figure 66 is a sectional view through a blast illustrating the proposed pattern and explosive loading. The present powder factor used is .4 lb/BCY, but this proposed pattern and loading will result in a .6 lb/BCY. Based on a bulk ANFO cost of $.10/lb (versus packaged at $.12/lb) and dryliners at $15 per hole, an increased blasting unit cost of $.022/BCY and an increased energy input per BCY of 50 percent are realized. This percentage may be a minimum, since overall powder distribution has been improved and decoupling eliminated.



DRILL PATTERNS FOR 12 1/4" DIA. VERTICAL HOLES AT
MIDWESTERN MINE A



PROPOSED DRILL PATTERN AND EXPLOSIVE LOADING AT
MIDWESTERN MINE A

Job No. 19089 Fig. 66

Elimination of the short holes has been suggested to simplify drilling, loading and tie-in, but, if unsatisfactory fragmentation of the limestone occurs (due to absorption of blast energy by the underlying soft shale), then it will be necessary to load the limestone and use intermediate short holes. This alternate method of loading and drilling has been shown in Figure 67 for a 15-foot thickness of limestone.

This method essentially requires separate loading of the limestone cap rock on a smaller drill pattern. The pattern for the underlying shale and limestone remains the same, but the collars are adjusted to reflect the softer material. Figure 68 illustrates the calculations for dimensioning the loading of the limestone cap rock and the larger collars in the shale. This method would result in an increased drilling cost of about \$.066/BCY and an increased blasting cost of about \$.012/BCY.

STEP 3 - INCREASE DELAYS TO 40 MSEC AND USE HIGH SPEED PHOTOGRAPHY TO DETERMINE MAXIMUM SURFACE DELAY POSSIBLE

At present, 9 msec delays are being used between holes. A minimum of 1 1/2 to 2 msec per foot of burden should be used (e.g. 35 to 45 msec). This is based on collar heights, explosive type, rock type and pattern. Even 50 to 100 msec may be acceptable, but this can best be determined by high speed photography.

STEP 4 - MOVE FACE HOLES AS CLOSE TO THE CREST AS POSSIBLE

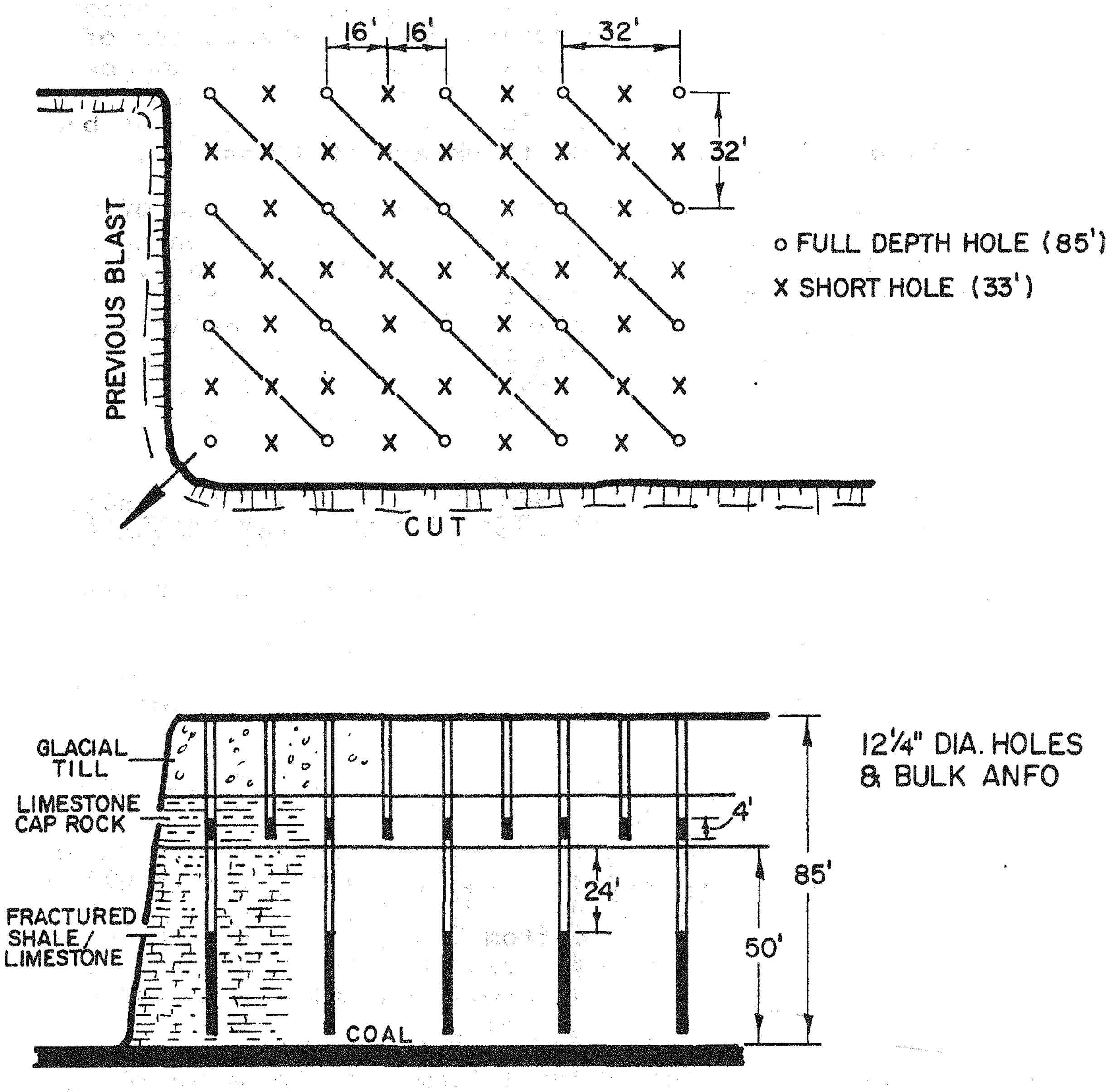
At present, face holes are drilled 30 to 50 feet from the crest and this, combined with a highwall slope angle of 65° over a 75-foot face, results in toe burdens of from of 65 to 80 feet. Again, assuming the bottom 8 diameters of explosive as a spherical charge and using scaled distance laws for cratering, a scaled burden of 4 for laminated and prefractured rock gives the following toe burden:

$$\text{Toe burden} = 4\,(\text{Wt. of bottom 8 diam. of explosive})^{1/3}$$

$$= 4\,(8 \times 43)^{1/3}$$

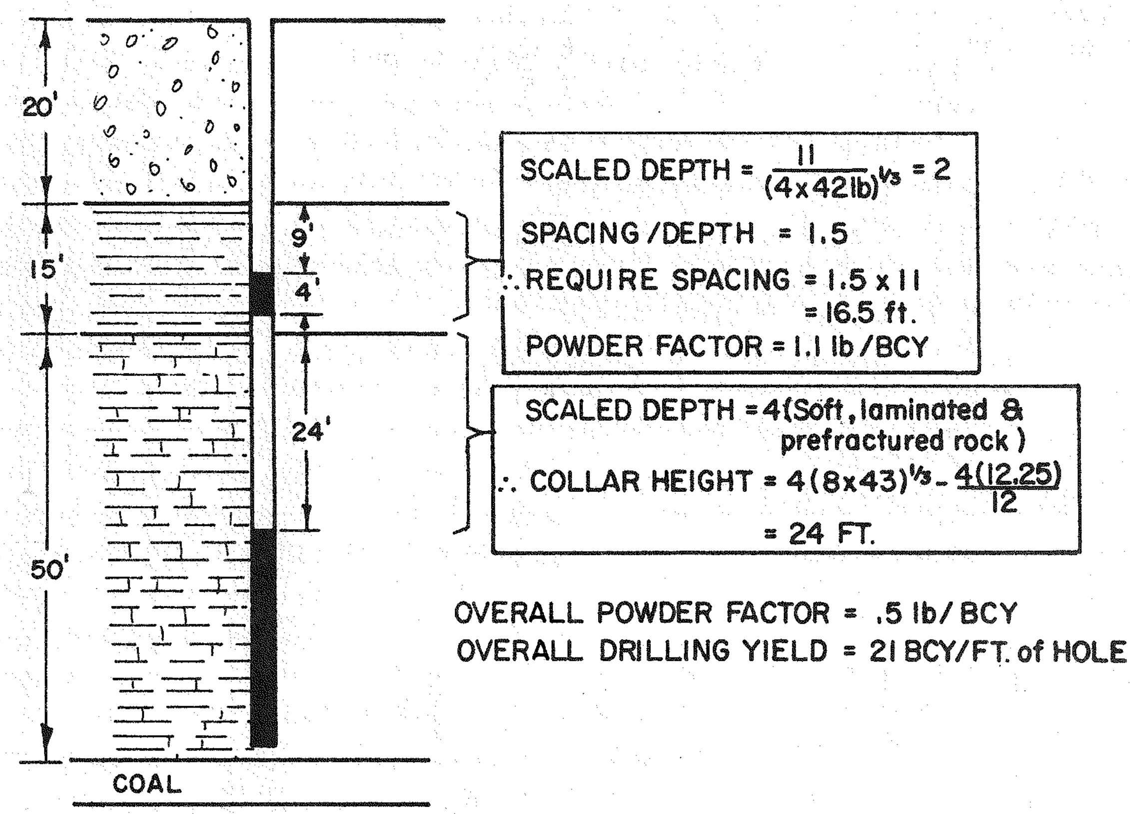
$$= 28 \text{ feet}$$

This implies that the toe burden will not be kicked out properly even if face holes are drilled as close to the crest as possible (e.g. minimum toe burden = 35 feet). A high density slurry could be employed to achieve the desired effect, but it is felt that moving the face holes as close to the crest as possible would be a good initial test.



PROPOSED DRILL PATTERN AND LOADING AT
MIDWESTERN MINE A

Job No. 19089 Fig. 67



CALCULATIONS FOR LOADING OF LIMESTONE CAP
ROCK AND SHALE AT MIDWESTERN MINE A

Job No. 19089 Fig. 68

The last step of the 12 1/4-inch diameter vertical hole program would entail establishing quality control standards and tests for explosives, primers, detonating cord and delays.

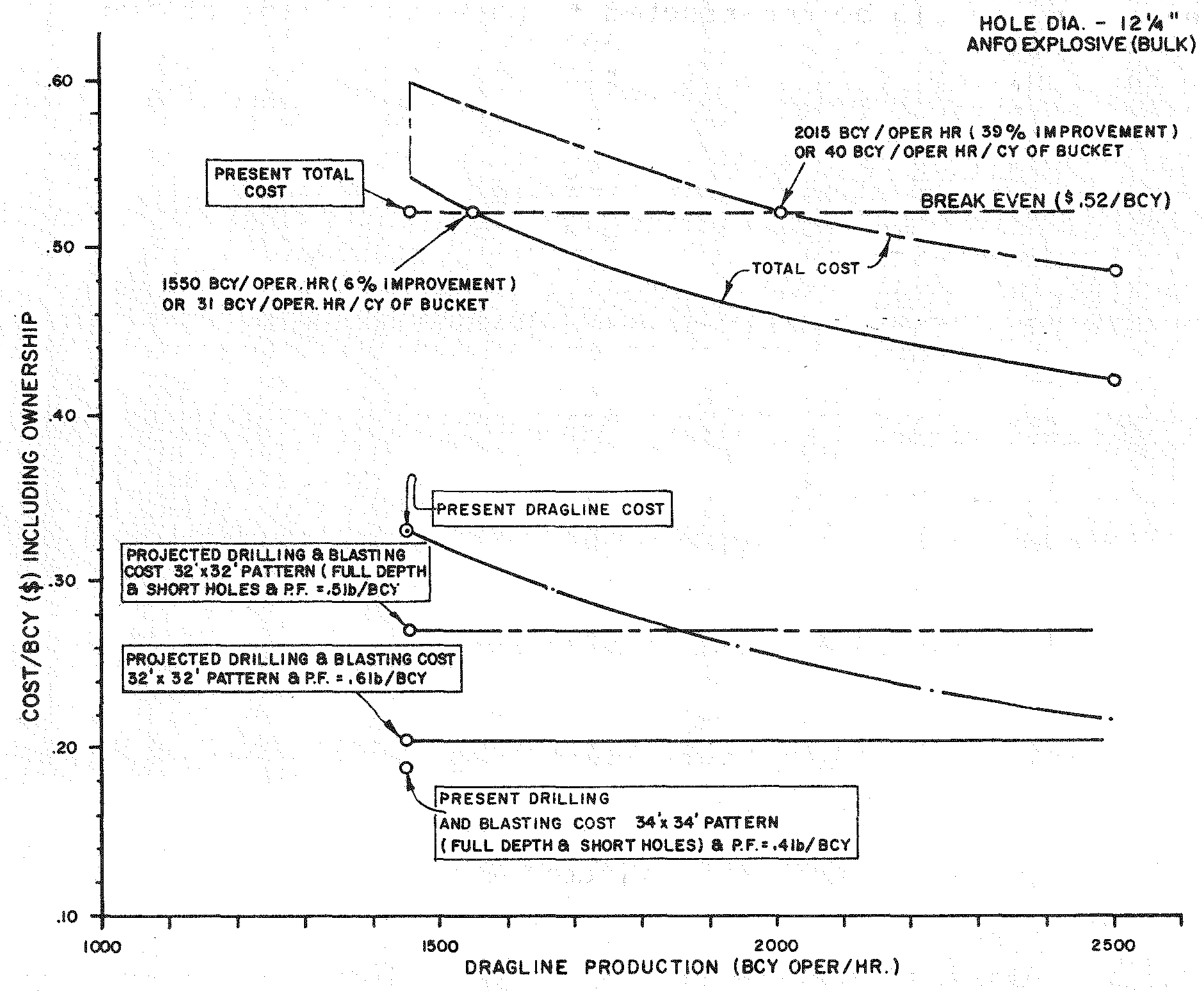
If all these program steps are carried out (50 percent increase in energy input, elimination of decoupling, improved powder distribution, increase of delay interval and the introduction of horizontal displacement to eliminate the "tight" muckpile situation), a substantial improvement in fragmentation should result. A productivity equivalent to the industry average (e.g. 40 BCY/Op Hr/CY of Bucket) is realistic.

Figure 69 is a projection of drilling, blasting and dragline unit costs as a function of increased dragline productivity. The cost of dragline ownership is based on a purchase price of $8,000,000 which results in operating costs of approximately $150/Op Hr or $.102/BCY at the current level of productivity. It is assumed that 25 percent of the operating cost will vary directly with the increase in production. Drill and explosive-handling ownership cost has been estimated at $.01 and $.001/BCY respectively. The impact on total unit cost for both proposed methods of drilling and blasting have been projected, based on dragline productivity increasing from the present level of 1460 BCY/Op Hr to the optimistic end of the scale at 2500 BCY/Op Hr. The latter figure is not necessarily the objective but merely an indication of the potential.

The main emphasis of Figure 69 is to illustrate the minimum improvement in productivity required such that either of these blasting programs can be conducted with no increase in total cost to the mine operation (it won't change the break-even point). From this graph it can be seen that the break-even point for the use of full depth holes requires a productivity increase of 6 percent (1460 to 1550 BCY/Op Hr). The alternate proposal of deck charging the limestone cap rock and using a pattern of 32 by 32 feet for both full depth and short holes requires a more substantial productivity increase of 39 percent to achieve a break-even on total cost. This required increase would only bring this dragline up to the industry average of 40 BCY/Op Hr/CY of bucket.

6.2.3 Blast Designs for 9-Inch Diameter Horizontal Holes

The horizontal holes are drilled to a depth of 90 feet and have 30-foot collars. The average burden to the top of the limestone cap rock is 20 to 30 feet. Packaged AN/FO is loaded in the holes, using a hydraulic ram. Holes are drilled approximately 3 feet above the coal seam at 28-foot centers. No delays are used between holes. Approximately 1100 pounds of explosive are placed in the hole or 18.3 lb/ft of column.



DRILLING, BLASTING & DRAGLINE COST
VS. DRAGLINE PRODUCTIVITY FOR 1550-W
AT MIDWESTERN MINE A

Using the same scaled distance laws for cratering as explained for the 12 1/4-inch diameter vertical holes, the collars should be 18 feet, the spacing 30 feet, and the burden should not exceed 15 feet.

If the burden to the top of limestone exceeds 20 feet, then a ramp should be constructed to permit drilling of two rows of horizontal holes or the 50R drill, which is used on the parting, should be used to drill the overburden with conventional vertical holes.

STEP 1 - DECREASE COLLAR HEIGHTS TO 18 FT

STEP 2 - USE 2 ROWS OF HOLES WHEN BURDEN IS GREATER THAN 20 FT

STEP 3 - USE DELAYS BETWEEN HOLES (e.g., 25 MSEC). HIGH SPEED PHOTOGRAPHY SHOULD BE EMPLOYED TO MAKE EXACT DETERMINATION

STEP 4 - TEST BLAST 10 5/8-INCH DIAMETER VERTICAL HOLES

The drilling and blasting parameters to be used are as follows:

a) Bulk AN/FO and dryliners.

b) 28 by 28-foot square pattern.

NOTE: Use one short hole centered between full depth holes to improve fragmentation of cap rock. Figure 70 illustrates the proposed drill pattern and loading.

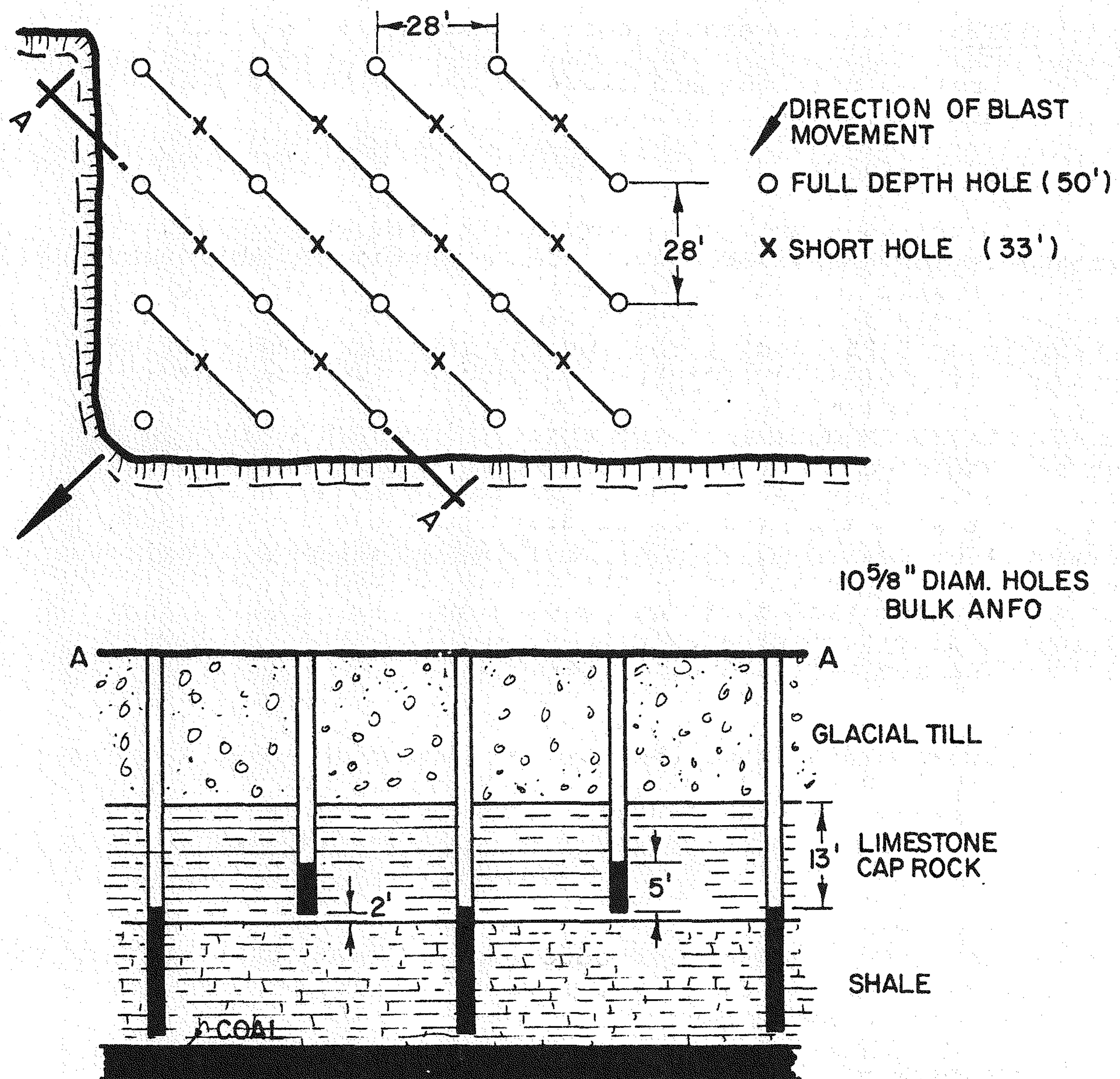
c) Tie-in diagonally (V_1 Configuration).

d) Use 30 to 40 msec delays between rows. The estimated drilling and blasting cost for this test is:

Drilling Cost = Cost/Ft/Yield

= \$2.30/Ft/21 BCY/Ft
= \$.11/BCY

Blasting Cost = Present Cost + Cost for Increase in P.F. + Dryliner Cost.
= .07 + .005 + .01
= \$.085/BCY
= Total drilling and blasting cost = \$.195/BCY



POWDER FACTOR = .45 lb/BCY
DRILL YIELD = 21 BCY / FT. of HOLE

DRILL PATTERN AND LOADING FOR 2570-W DRAGLINE AT MIDWESTERN MINE A

Job No. 19089 Fig. 70

NOTE: Drill ownership cost estimated at $.01/BCY

The present drilling and blasting cost with horizontal holes is approximately $.12/BCY.

6.2.4 Cost Analysis

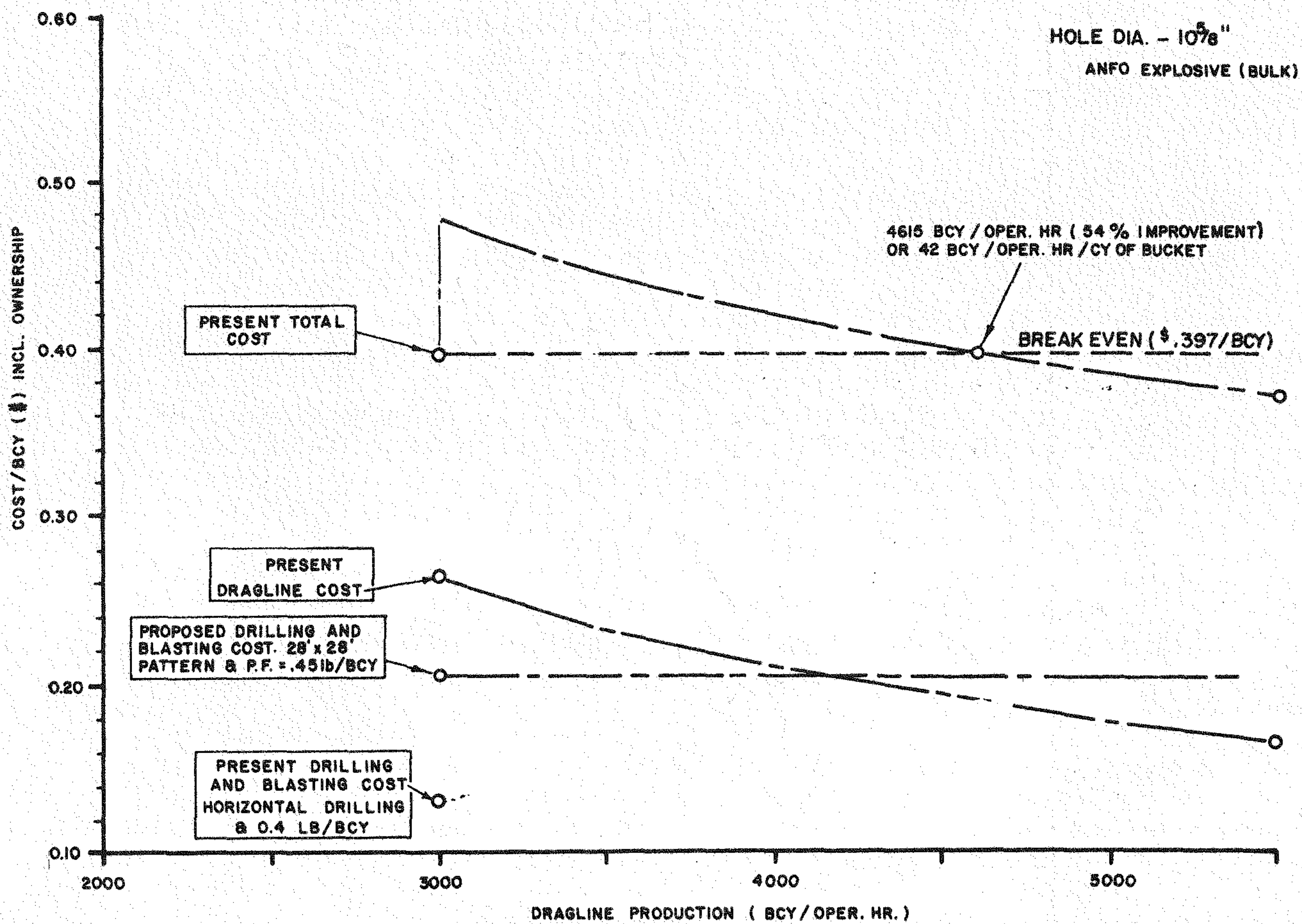
Using the same unit cost versus dragline productivity approach as for the 1550W, Figure 71 is developed, which shows the relationship for the 2570W dragline. Ownership cost was based on a 1972 purchase price of about $13,000,000, which is $260/Op Hr or $.087/BCY at the current level of productivity. A productivity of 50 BCY/Op Hr/CY of bucket was again projected as the optimistic end of the scale. This represents an 83 percent improvement over the current level of production, however, and should not be considered as the immediate objective. Energy input per BCY would be increased by only 13 percent but with improved powder distribution and reasonable horizontal displacement, a considerable improvement is anticipated in the fragmentation of the limestone and overall digging characteristics of the overburden. In addition, the horizontal displacement should be especially beneficial with the extended bench method of operation.

It should be noted with regards to cost, that if new draglines were being used, the cost of ownership would be in the order of $.10 to $.12/BCY, while operating costs would remain the same. This would imply a great deal more energy input could be used and still show an appreciable overall cost saving.

Based on Figure 71, the break-even point occurs at a 54 percent increase in dragline productivity. This value is high primarily due to the underblasting with current drilling and explosive loading techniques.

6.3 MIDWESTERN MINE B - BLAST TEST PROGRAM

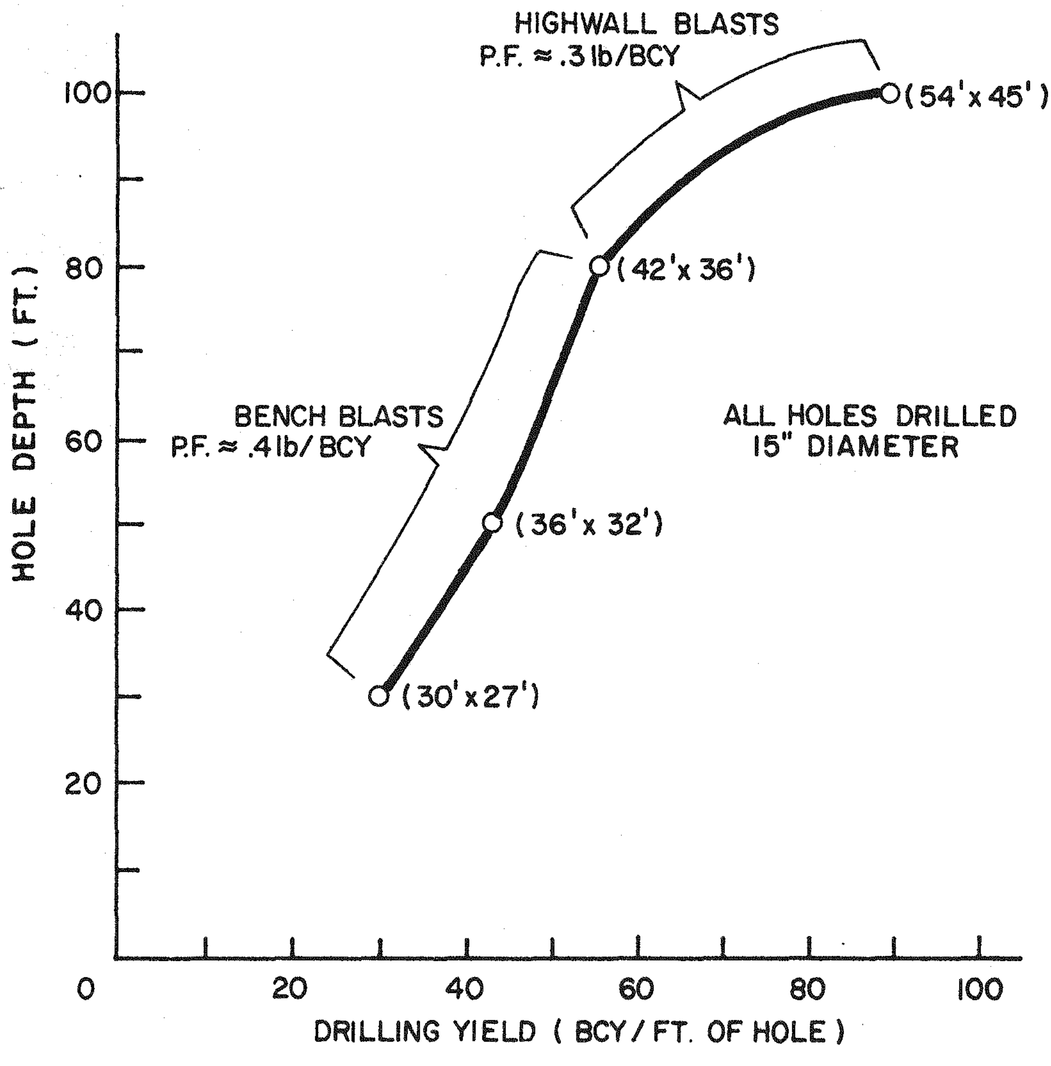
Midwestern Mine B operates a single coal seam operation with overburden depths in excess of 100 feet. The top of the overburden is 40 to 50 feet thick and is benched due to undulating surface topography. The bulk of the bench is sandstone overlain by glacial till. The majority of the highwall overburden is shale with some hard bands of limestone. The sandstone and limestone create the greatest fragmentation difficulties. As a result, a system of deck charging (AN/FO +25 lb slurry) is used to improve fragmentation in these zones. All drilling is done with 15-inch diameter holes and patterns vary with hole depth (see Figure 72). Drill patterns in bench work vary from 27 by 30 feet for 30-foot depths up to 36 by 42 feet



DRILLING, BLASTING & DRAGLINE
COST VS. DRAGLINE PRODUCTIVITY FOR 2570-W
AT MIDWESTERN MINE A

Job No. 19089

Fig. 71



DRILL PATTERN VS. HOLE DEPTH PRACTICE AT
MIDWESTERN MINE B

Job No. 19089 Fig. 72

for 80-foot depths. Collar heights are usually 33 feet but are only 25 feet when a 150-pound deck charge is used. The latter loading configuration gives a calculated scaled depth of burial of 5, which would imply total confinement. In many cases the sandstone in the bench work extends to the surface, which often results in blocky fragmentation and oversize. This type of fragmentation, combined with the dragline digging in the chopdown mode, results in very poor productivity (30-50 percent of capability). It is estimated that bench fragmentation could be improved considerably by using one pattern and decreasing collar heights with respect to the top of the sandstone.

6.3.1. Bench Blast Designs

STEP 1 - ADJUST COLLAR HEIGHTS TO 20 FEET FROM TOP OF HARD SANDSTONE.

The collar heights can be calculated as before, using the top 8 diameters of explosive as a spherical charge and using scaled distance laws for cratering. A scaled depth of burial of 2.8 is used since the sandstone is a massive competent rock.

$$\text{Collar height} = 2.8\,(670)^{1/3} - \frac{4(15)}{12}$$

$$= 20 \text{ feet}$$

STEP 2 - FOR BENCH WORK IN EXCESS OF 30 FEET IN SANDSTONE, USE A 34 BY 34-FCOT PATTERN WITH TIE-IN ON THE DIAGONAL.

For bench heights greater than 30 feet (e.g. 20 feet collar + 8 diameters of explosive), the pattern can be calculated assuming a cylindrical charge and a scaled burden of 3.

$$\frac{\text{Burden}}{(\text{Wt/Ft of hole})^{1/2}} = 3$$

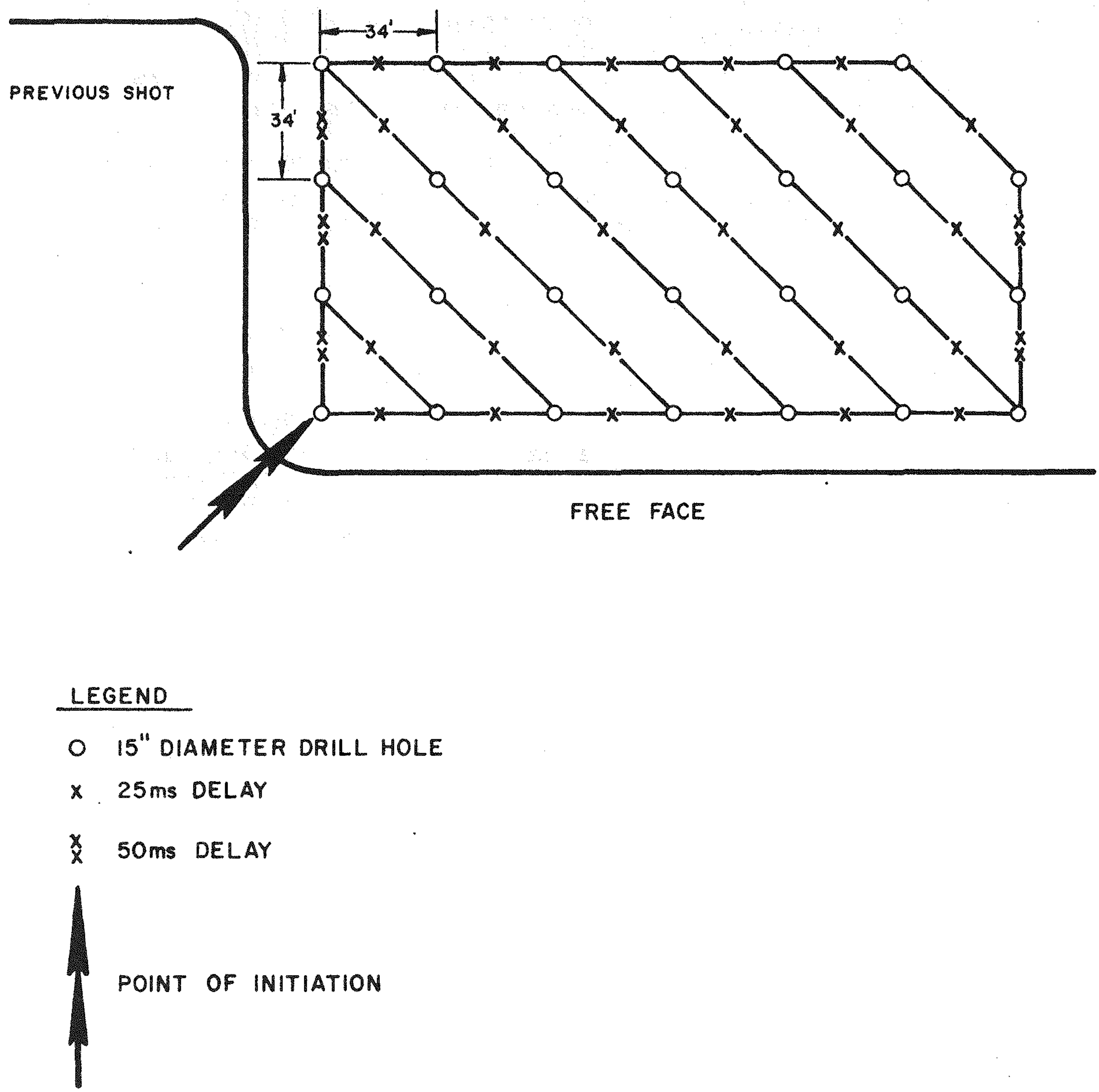
$$\text{Burden} = 3 \times (67)^{1/2} = 24.5 \text{ feet}$$

For a square drill pattern and tie-in on the diagonal (V_1 Configuration):

$$\text{Drill Pattern} = \frac{(2 \times 24.5)}{2} = 34 \text{ by } 34 \text{ feet}$$

STEP 3 - INCREASE DELAY INTERVAL BETWEEN HOLES

At present, delays varying from 17 to 42 msec are being used. This can be increased to 40 to 50 msec (1-1/2 to 2 msec per foot of burden) by using 25 msec between rows and holes (see Figure 73).



TIE-IN AND DELAY
SEQUENCE FOR BENCH
BLASTING AT MIDWESTERN MINE B

Job No. 19089 Fig. 73

6.3.2 Highwall Blast Designs

The overburden in the highwall is commonly over 100 feet deep. The bench work removes most of the hard sandstone, leaving well-fractured shale with occasional bands of limestone. The limestone is deck-charged and decks are delayed from top to bottom in 25 msec periods. A great deal of effort goes into correlating drill records to locate the deck charges, but, as discussed earlier, the system is not always successful.

STEP 1 - CHANGE DRILL PATTERN AND LOADING OF HIGHWALL BLASTS

As a test, the deck charging will be abandoned, the pattern decreased, and the powder factor increased. A moderate increase in energy per BCY will be sufficient to give satisfactory fragmentation in the hard bands and an overall improvement in fragmentation. For this test the following drilling and blasting parameters are suggested:

Collar Height	$= 4\,(670)^{1/3} - \frac{4(15)}{12}$ $= 30$ feet	(Scaled distance of assuming laminated and prefractured rock)
Burden	$= 3.5\,(67)^{1/2}$ $= 28.6$ feet	(Scaled distance of 3.5 for soft rock)
Pattern	$= \frac{57}{2} = 40$ by 40 feet	(Square pattern, diagonal tie-in)
Delays - 50 msec		(25 msec between rows and holes. This can probably be increased to 75 or 100 msec from high speed photography analysis)
Toe Burden	$= 4\,(670)^{1/3}$ $= 35$ feet	(Move face holes as close to crest as possible)

6.3.3 Cost Analysis

At present, a powder factor of .35 lb/BCY is used with a drilling yield of approximately 70 BCY/ft of hole. Based on information provided, this represents a drilling and blasting unit cost of $.087/BCY. The 2570-W dragline is currently operating at 3655 BCY/Op Hr with an estimated total cost including ownership of \$766/Op Hr or \$.21/BCY. The 2570-W dragline is equipped with a 110 CY bucket which gives a productivity of 33.3 BCY/Op Hr/CY of bucket.

The decrease of drill pattern spacing and increase powder factor will increase costs as follows:

Drilling yield $= \dfrac{40 \times 40}{27} = 59$ BCY/ft of hole

Cost/ft = \$2.40

Drilling unit cost = 2.40/59 = \$.041/BCY

Powder Factor $= \dfrac{(50 \times 67)}{59 \times 80} = .70$ lb/BCY

(80 ft hole)

Bulk AN/FO = \$.12/lb

Blasting cost = (.70 x .12) + \$.01/BCY (Misc. supplies)

= \$0.94/BCY

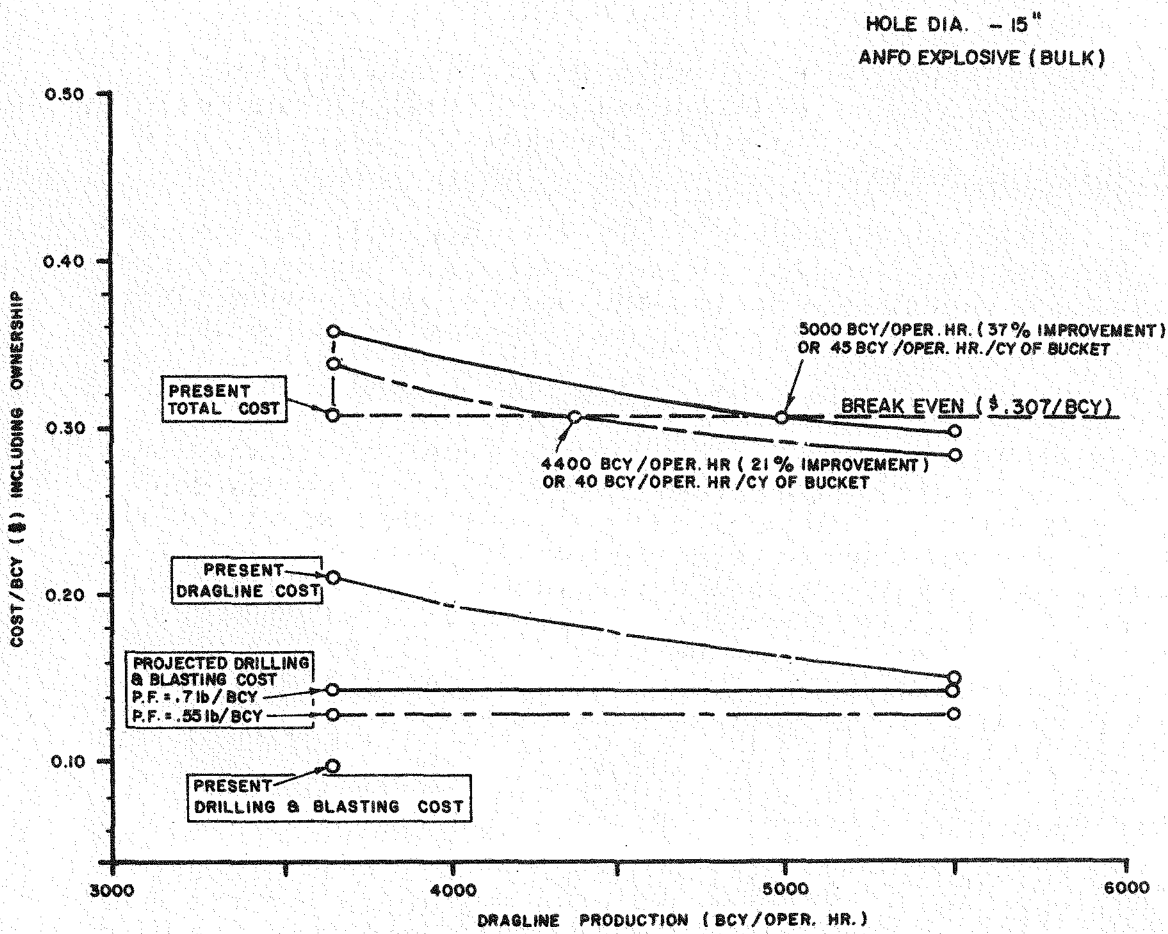
Increase in energy input $= \dfrac{(.70 - .35)}{.35} \times 100$

$= 100\%$

At present, dryliners are not being used, although 95 percent of all explosive loaded is bulk AN/FO. Moisture problems are sometimes overcome by loading and blasting every day. However, the value of the delay time versus cost of dryliners should be examined.

Figure 74 depicts present and projected drilling, blasting and dragline costs versus dragline productivity. As before, 25 percent of the dragline operating cost varies with production. A production rate of 5500 BCY/Op Hr would result in an overall cost saving of \$.01/BCY. The proposed drill pattern and powder factor gives an economic break-even point at 5000 BCY/Op Hr or 37 percent improvement. A continuous column of powder that gives an overall powder factor of .55 to .60 lb/BCY may yield the best results.

For 80-foot holes drilled on 40 by 40-foot pattern, the loading configuration would be as follows:



DRILLING, BLASTING & DRAGLINE
COST VS. DRAGLINE PRODUCTIVITY FOR 2570-W DRAGLINE
AT MIDWESTERN MINE B

Job No. 19089 Fig. 74

Bottom AN/FO column------------------------31 ft
Stemming deck------------------------------11 ft
AN/FO deck charge-------------------------- 8 ft
Stemming to surface------------------------30 ft
Total Explosive Weight = 2610 lb
BCY per hole = 4740
Overall Powder Factor = .55 lb/BCY
Estimated Blasting Cost = (.55 x $.12/lb) + .01/BCY (Misc. supplies)
= $.076/BCY

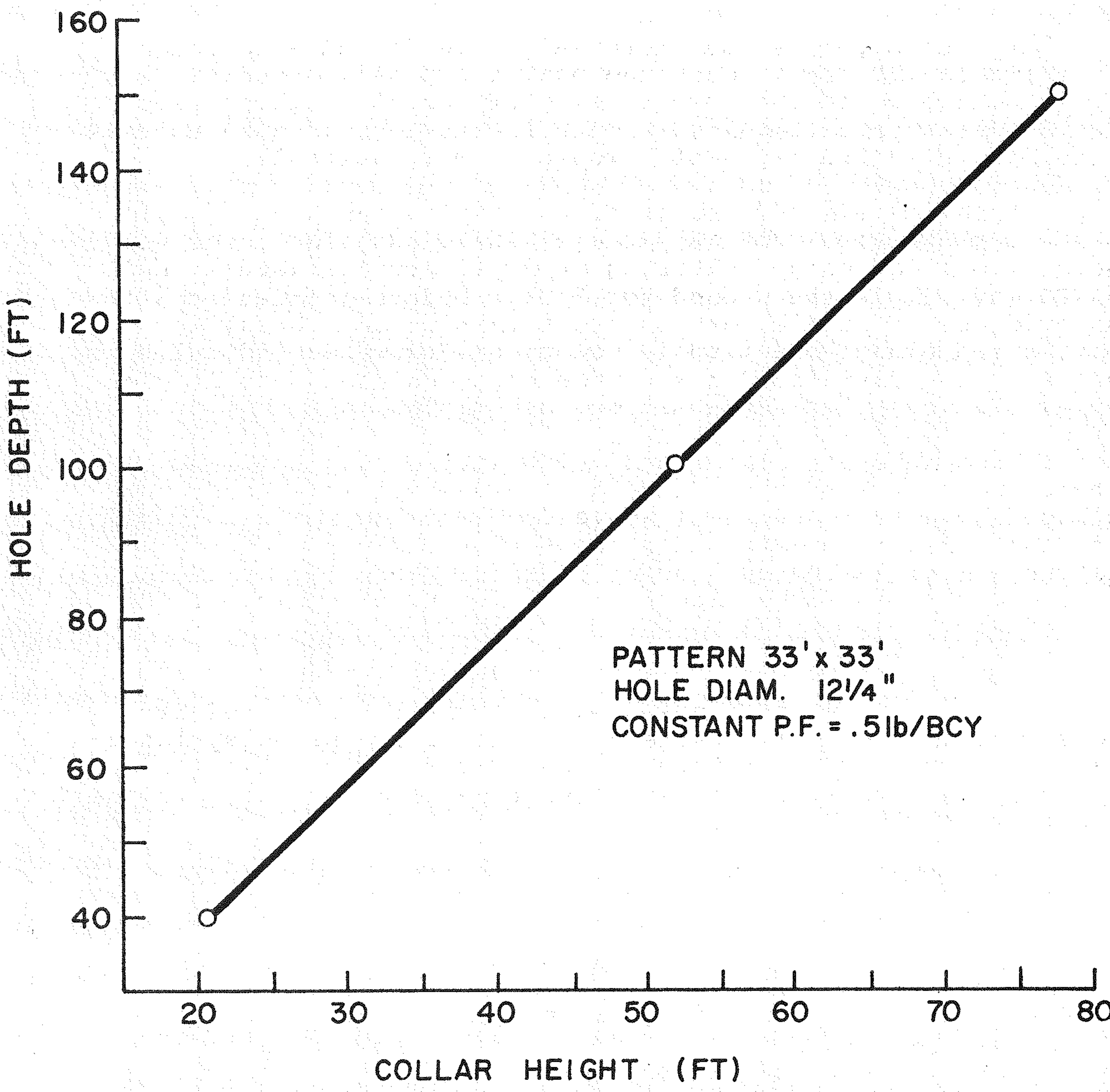
The break-even point for this drilling and loading arrangement would require an increase in dragline productivity of 21 percent. This would equal the industry average of 40 BCY/Op Hr/CY of bucket.

The 2570-W was purchased in 1972 at one-half of the 1978 price. Therefore, the cost of ownership plus current operating costs are relatively small, and only a small productivity increase would be necessary to reach the break-even point.

6.4 Western Mine C, Pit 1 Blast Test Program

Western Mine C, Pit 1, is a single seam coal operation. The seam dips at 24 to 40 percent, resulting in overburden ranging from zero at the coal outcrop to a maximum of 150 feet down dip. The cuts are across the strike, and, as each cut moves down dip the overburden depth increases. The shallow overburden is usually removed by trucks with a shovel or a backhoe loading from within the pit. A 62 CY, Marion M-8000 dragline removes 80 percent of the overburden and operates primarily in overburden that is over 40 feet deep. At present the M-8000 averages only 2130 BCY/Op Hr or 34.3 BCY/Op Hr/CY of bucket. However, a larger dragline is being erected and will be ready for operation in 1979.

Both 9-inch and 12 1/4-inch diameter holes are rotary drilled. The nine-inch holes are used primarily for the shovel-truck operation and the 12 1/4-inch holes are used for the dragline operation. The 12 1/4-inch diameter holes have a 33 by 33 foot drill pattern, and blasting is done with a .5 lb/BCY powder factor. Consequently collar heights are not fixed. Figure 75 illustrates the relationship between collar height and hole depth increase. From this figure it can be seen that over a 40 to 150-foot hole depth range, collar heights vary from 21 to 78 feet, respectively. However, collar height should be about 24 feet for the properly scaled burial depth.



RELATIONSHIP BETWEEN COLLAR HEIGHT AND HOLE DEPTH FOR A CONSTANT POWDER FACTOR

At present, in softer material, hole depths also range from 100 to 150 feet. Here the scaled explosives burial is 8 to 22, which would indicate total confinement of the explosive charge. For soft laminated and prefractured rock such as this material, a scaled depth of 4 would be more suitable.

Approximately 80 percent of the explosives used is bulk AN/FO and the remainder is packaged AN/FO and slurry for wet holes (no blast hole dewatering or dryliners are used). Bulk AN/FO may sit in the ground for 3 days before initiation with no evidence of partial or low order initiation. Dragline overburden blasts are tied in row-on-row (row perpendicular to crest). They are fired toward the previous shot with 25 msec delays between rows and sometimes between holes.

The overburden consists of alternating bands of sandy shale and soft sandstone, and, although there is a hard band of sandstone (6 to 10 feet) above the coal, no special loading is used. Blast fragmentation depends on the prefractured nature of the overburden and blast displacement is negligible, which results in the typical "tight" muckpile condition.

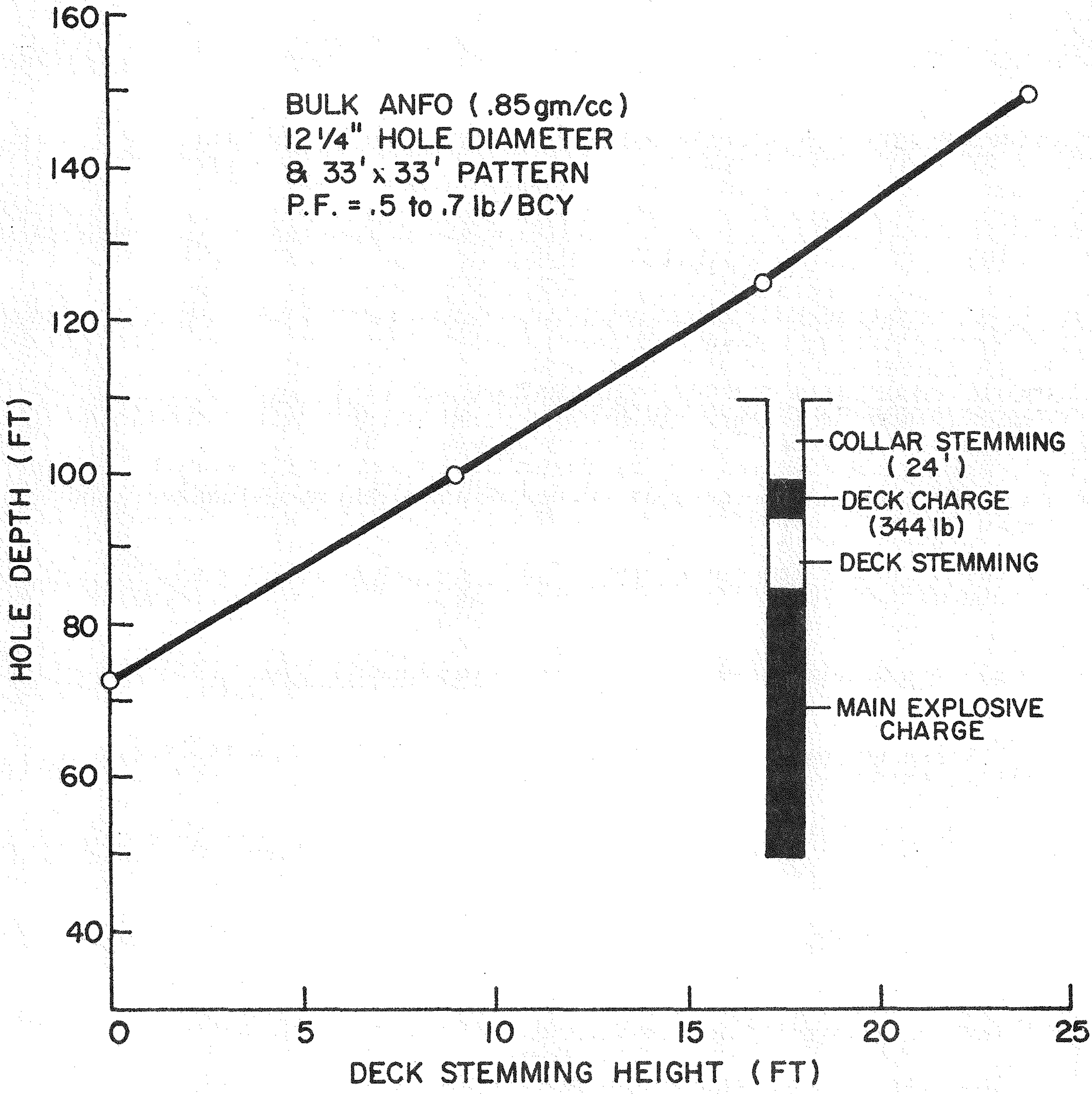
6.4.1 Blast Designs for 12 1/4-Inch Vertical Holes

The test program proposed will decrease collar heights, leave the present drill patterns intact, increase delay intervals, direct the shot toward the cut with diagonal tie-in, and minimize the toe burden by drilling face holes as close to the crest as possible.

STEP 1 - REDUCE COLLAR HEIGHTS TO A STANDARD 24 FEET FROM TOP OF HOLE.

If collar heights are held constant as hole depth is increased, the powder factor would reach a maximum of .875 lb/BCY at a 150-foot depth. This powder factor is excessive. Therefore, the explosive column will be split with deck stemming so that the powder factor does not exceed .7 lb/BCY. By design, the deck charge would never be less than 8 diameters in length, collared 24 feet from surface. Also, stemming between the deck column and bottom column charge would not exceed 24 feet. Figure 76 indicates a convenient way of determining the deck stemming height for varying hole depths such that the powder factor does not exceed .7 lb/BCY.

The calculation to determine the collar height is based on scaled depth of burial equal to 4.



DETERMINATION OF DECK STEMMING HEIGHT TO MAINTAIN MAXIMUM POWDER FACTOR OF .7 lb/BCY.

Job No. 19089

Fig.76

$$\text{Collar Height} = 4.0\ (\text{Explosive, lb, in top 8 dia.})^{1/3} - \frac{4(\text{Hole Dia.})}{12}$$

$$= 4.0\ (344)^{1/3} - \frac{4(12.25)}{12}$$

$$= 24 \text{ feet}$$

STEP 2 - MAINTAIN PRESENT DRILL PATTERN OF 33 x 33 FEET (SQUARE)

The drill pattern may be determined by scaled distance laws for cratering and using a scaled burden equal to 3.5 (soft rock).

$$\frac{\text{Burden (ft)}}{(\text{Explosive, lb/ft of hole})^{1/2}} = 3.5$$

$$\therefore \text{Blasting Burden} = 3.5(42)^{1/2} = 22.7 \text{ feet}$$

For a square drill pattern tied in on the diagonal (V_1 Configuration):

$$\text{Pattern} = \frac{2\ (\text{Blasting Burden})}{2}$$

$$= \frac{2(22.7)}{2}$$

$$= 32 \times 32 \text{ feet}$$

NOTE: This design pattern is close enough to that presently being employed to warrant no change.

STEP 3 - TIE IN DIAGONALLY (V_1 CONFIGURATION) AND USE 25 MSEC DELAYS BETWEEN ROWS AND HOLES TO GIVE AN EFFECTIVE DELAY OF 50 MSEC.

The tie-in indicated in Figure 73 would be suitable for this operation. As outlined in the theory section of this report, a delay interval equivalent to 1-1/2 to 2 msec per foot of burden would be safe for surface delays and detonating cord. Exact determination of maximum safe delay interval can be determined with high speed photography.

STEP 4 - POSITION FACE HOLES SUCH THAT TOE BURDEN DOES NOT EXCEED 28 FEET; WHERE BANK HEIGHT EXCEEDS 60 FEET MOVE FACE HOLES AS CLOSE TO CREST AS POSSIBLE.

The toe burden breakout characteristics of ANFO for 12 1/4-inch diameter holes can be examined by assuming that the bottom 8 diameters of explosive act as a spherical charge. By applying scaled distance laws for cratering and using a scaled burden of 4 (soft, laminated and prefractured rocks), a value of toe burden pulling capability can be calculated as follows:

$$\frac{\text{Toe Burden (ft)}}{(\text{Explosive, lb, in bottom 8 diam.})^{1/3}} = 4$$

$$\begin{aligned}\text{Toe Burden} &= 4\ (344)^{1/3} \\ &= 28\text{ feet}\end{aligned}$$

For a highwall slope angle of 65°, the toe burden will exceed 28 feet for bank heights greater than 60 feet. For extreme bank heights of 100 to 150 feet, it would be necessary to use explosives with greater weight strengths such as Al/AN/FO or greater bulk strength, such as high-density slurry. Angle drilling is possibly the only solution presently available for extreme overburden heights.

6.4.2 Cost Analysis

Present Unit Costs

Present dragline operating cost is estimated at $454/Op Hr. Of this, $127/Op Hr represents ownership cost based on a machine purchase price of $7,000,000. The current level of production, 2130 BCY/Op Hr, yields a unit operating cost of $.214/BCY.

Drilling is currently at 40 BCY/ft of hole with an estimated cost of $3.20/ft, which results in a unit cost of $.08/BCY. As in previous cases a drill ownership cost of $.01/BCY will be used.

Currently a powder factor of .5 lb/BCY is being used. A bulk AN/FO cost of $.07/lb plus labor and supplies gives an estimated blasting cost of $.07/BCY.

The present total unit cost for the dragline drilling and blasting is estimated at $.376/BCY, including ownership.

Projected Unit Cost for New Blast Design

Dragline unit cost at Western Mine C has been reduced by projecting greater production rates and assuming that 75 percent of operating costs are fixed while the remainder are variable and in direct proportion to the increased production (see Figure 77). Again, production has been projected to the very optimistic end of the scale at 50 BCY/Op Hr/CY of bucket or 3100 BCY/Op Hr.

No drilling changes are proposed, so projected and present cost are the same at $.08/BCY plus $.01/BCY for ownership.

The plan will increase the powder factor from .5 to .7 lb/BCY and use dryliners in wet holes. It is estimated that this will increase the blasting unit cost by $.02/BCY to $.09/BCY.

The inceased powder factor effectively raises the energy input per BCY by 42 percent. This figure is minimal with regards to fragmentation since improved delaying and horizontal displacement should have a major impact on diggability (elimination of "tight" muckpile).

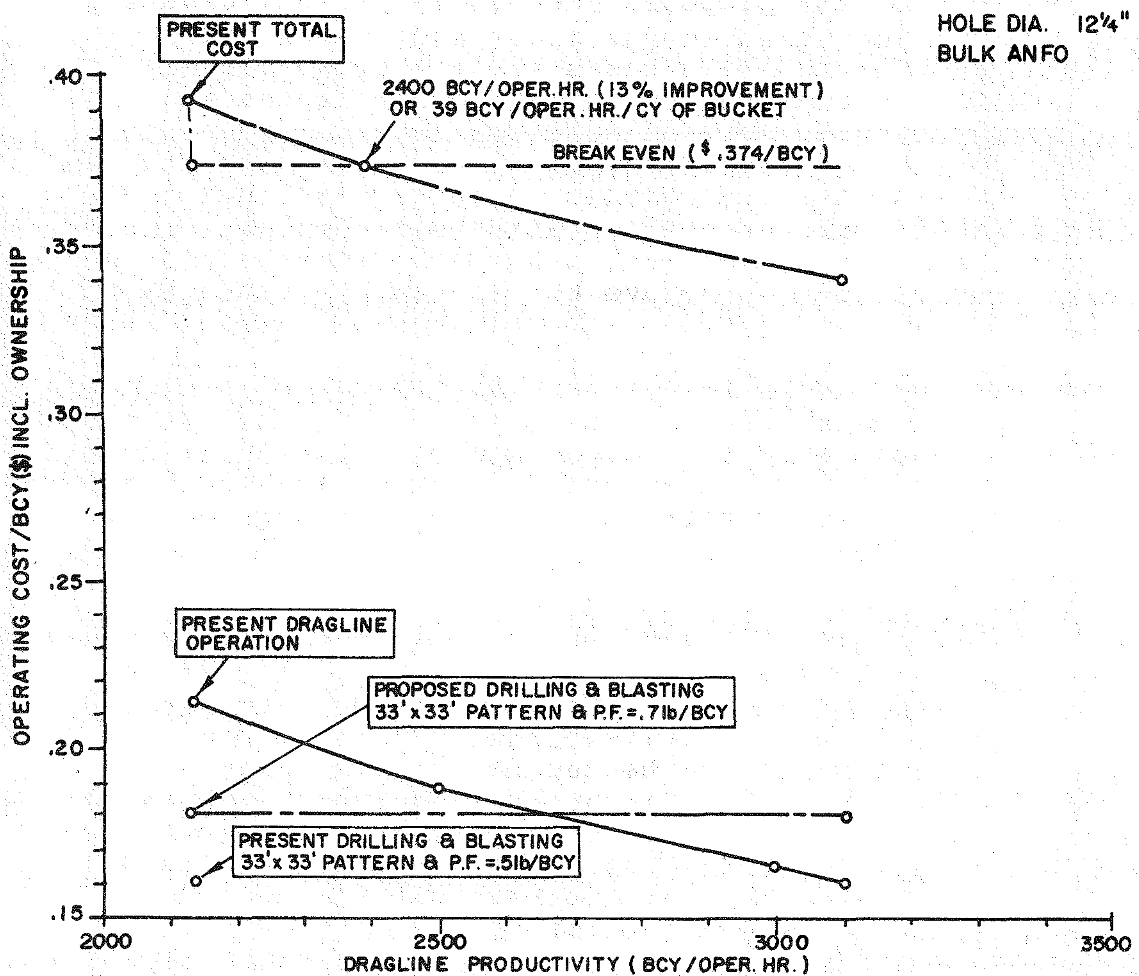
Figure 77 shows the unit cost of operations versus dragline productivity. It should be noted that new dragline ownership costs will be considerably higher as outlined by purchase prices in Table I.

The break-even point for this test program occurs at 2400 BCY/Op Hr, 13 percent improvement over present practice. The required increase in productivity is small due to the low price of bulk AN/FO in Wyoming (e.g. $.07/lb). Packaged AN/FO at the same site sells for $.12/lb which makes the use of dryliners economically favorable. In addition, dryliners would reduce the frequency of blasting (twice a week). A minor problem was the wind-drifting of drill cuttings back into the blasthole resulting in a 3 to 5-foot loss of hole depth. This problem can be remedied with the use of 3 or 4-foot cardboard hole casings.

7.0 CONCLUSIONS

7.1 General

Review of literature, blasting theory and field studies at nine surface coal mines has resulted in the development of "new" blast design concepts for surface coal mines. Although



DRILLING, BLASTING & DRAGLINE
COST VS. DRAGLINE PRODUCTIVITY FOR M-8000
AT WESTERN MINE C, PIT 1

Job No. 19089 Fig. 77

many of the recommended concepts are not new to blasting as it is practiced in hardrock mining and construction, they are new to surface coal mining. Prudent application of the procedures proposed in this report, whether they are new or simply more sophisticated applications of standard practice, should result in improved fragmentation and better dragline productivity in surface coal mines.

Because many of the proposed changes to present blasting practice, such as increased energy input and use of dryliners, will result in higher blasting costs for the operator, improved fragmentation must result in sufficient improvement in productivity and dragline operating costs to maintain or lower total mining costs. Operator acceptance of these concepts can only be justified if the test program demonstrates that the ultimate impact of improved fragmentation will be lower total mining costs. Therefore, the test programs proposed in this report have been designed to prove the economic advantages of greater emphasis on the blasting requirements of surface coal mining.

Although outside the objectives of this program, field observations indicate that the potential for substantial improvement in dragline efficiency exists through the implementation of better operating methods, planning and sequencing.

7.2 Recommendations

Based on observations at the nine mines, dragline productivity could be increased cost-effectively at surface coal mines by implementing the following general recommendations for improving blasting performance. These recommendations are directed at filling the bucket more quickly, with greater consistency and with a better fill factor. They would achieve these objectives by improved (a) overburden fragmentation, (b) horizontal muckpile displacement and (c) blasting reliability. The recommended improvements in blast design concepts and operating procedures include:

1) Increase energy input per BCY by 40 to 100 percent and improve the effective powder distribution by optimizing the following blast parameters:

   - burden and spacing
   - collar height
   - toe burden
   - delay intervals
   - blast tie-in

2) Introduce use of blast hole dewatering, dryliners and bulk AN/FO for wet holes.

3) Improve quality control in establishing shot patterns and charging blast holes.

The most radical of these recommendations relate to increased horizontal displacement of the muckpile. Horizontal displacement will allow the structure of the stratified overburden to be disturbed, resulting in more desirable fragmentation. The effects of this displacement on the dragline stripping system must be evaluated carefully because of potential negative factors, such as a lower muckpile and more flyrock. It is anticipated that these factors will be more than compensated for by improved productivity and overburden spoiled by the blast.

Use of square drill patterns with diagonal tie-ins is recommended in lieu of rectangular or diamond patterns with row-on-row or chevron tie-ins, based on the following rationale:

1) The square pattern is simple to lay out and drill.

2) Square patterns give maximum face hole frequency, which is important for muckpile displacement.

3) Diagonal tie-ins control vertical drop, yielding a uniform windrow effect along the length of the blast.

In addition to changes in drilling and blasting layout, proposed changes include the use of blast hole dwatering, dryliners and bulk AN/FO. Also, deck charging should be minimized to simplify loading and delay procedures.

7.3 Test Program

One western and two midwestern surface coal mines are recommended for test programs to evaluate proposed changes in blasting practice. Specific test programs have been individually designed for each test mine with the objective of improving fragmentation through implementation of the above recommendations. Typical tests include the following evaluations:

1) Use of bulk explosives instead of packaged.

2) Use of blast hole dewatering and dryliners.

3) Implementation of explosives quality control monitoring.

4) Reduction of time lag between loading and blasting.

5) Elimination of short holes and deck charging.

6) Increasing delays to maximum possible based on results from high speed photography.

7) Moving face holes as close to the crest as possible.

8) Adjusting collar heights to break hard bands.

9) Using diagonal tie-ins and square patterns rather than more complex methods.

Detailed cost analyses will be utilized to evaluate economic impacts of blasting changes with respect to drilling and blasting costs, dragline operating costs, equipment maintentance costs, supervisory costs, etc. It is anticipated that dragline productivity increases of from 13 to 54 percent will be required to break even on additional blasting costs. Dragline productivity increases of this magnitude are considered reasonable, and the potential of even better improvement is considered good.

APPENDIX A

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APPENDIX B

PROPOSED AGREEMENT FORM

A G R E E M E N T

THIS AGREEMENT, entered into this _____ day of ___________, 1977 by and between WOODWARD-CLYDE CONSULTANTS, including all of its managers, supervisors, employees, agents and subcontractors, (herein called "Contractor") and ____ of ____, including all of its managers, supervisors, employees, agents and subcontractors (herein called "Owner");

W I T N E S S E T H:

WHEREAS, Contractor has a contract with the U. S. Department of Energy to develop new blast designs to increase dragline stripping rates in western and midwestern United States surface coal mines; and

WHEREAS, Owner is willing to participate in a program of increasing dragline stripping rates by trying new blasting programs designed by Contractor on the terms and conditions hereinafter set forth.

NOW, THEREFORE, in consideration of the premises and the mutual covenants hereinafter set forth, the parties agree as follows:

1. Contractor agrees to characterize blasting parameters at owner's property by first studying the geology of the overburden and parting seams, the depths of the overburden and parting seams and the highwall stability factors. From this information, contractor shall design a blasting program which will include but will not be limited to general hole layout and spacing, diameter of blast holes, stemming depth, point of detonation initiation (top or bottom of hole), type of priming, attitude of hole (vertical, inclined, or horizontal) powder factor, and other factors deemed pertinent by the Contractor. Depending on results of test blasts, Contractor may have to alter blasting design parameters in succeeding blasts until the most efficient design is found for the owner's ground.

2. Owner agrees to comply with Contractor's blasting program to the best of his ability; however, owner has a right to review all blasting programs of Contractor and to suggest changes that, in the owner's experience and because of equipment limitations may not be applicable to his property. Owner has final veto over any blast design which he feels will not be applicable to his peculiar ground condition.

3. Owner agrees to use his regular mining equipment on this project and agrees to pay for the following costs on this project:

1) Ownership costs on his equipment

2) All taxes accruing on the mine or equipment

3) All drilling and blasting costs and supplies

4) All other ordinary mining costs and supplies

5) Equipment maintenance and supply costs

6) All owner's employee costs who are or may be involved in this project.

Contractor agrees to provide and pay for the following costs on this project:

1) To provide and maintain ground vibration and air blast measuring equipment

2) All necessary vehicles for Contractor's use

3) All contractor's employee costs who are involved in this project.

4. Contractor shall at all times be acting as an independent contractor and shall in no way be considered an employee or agent of Owner.

5. Contractor shall not engage in any work involving drilling, explosive handling, explosive emplacement, or in any owner's operations. The Contractor is to design, advise and record the details of the drilling and blasting program. Owner is to do all mining work including, but not limited to, drilling, blowing holes, loading holes, detonating blast holes, and all phases of dragline operation.

6. Owner agrees to allow Contractor access to test blasted areas and spoil piles so that Contractor can study the fragmentation parameters and the highwall stability in the test area. Contractor agrees to comply with all owner's rules and regulations and will only go in these areas with owner's prior permission and knowledge.

7. Owner agrees to allow Contractor to ride on dragline during working shifts so that Contractor may observe and study dragline cycle time, bucket-fill factors, dragline power consumption, and any peculiarities of dragline operation that may be caused by Contractor's blasting program. Contractor agrees to be an observer only and will in no way interfere with Owner in carrying out his normal duties.

8. Owner agrees to let Contractor have access to all maintenance records and costs on drills and draglines one year prior to this study and during the entire study period so that Contractor can compare maintenance costs. Owner agrees that when equipment is down for inspection, Contractor is allowed to visually inspect equipment for any indication of increased or decreased maintenance because of Contractor's blast programs. Contractor agrees that it will in no way interfere with the maintenance cycle or time used by the Owner.

9. Owner agrees to let Contractor have access to all drilling and blasting costs one year prior to the study and during the entire study period so that Contractor can compare drilling and blasting costs. Owner will either provide means whereby the drilling and blasting costs can be kept separate in the Contractor's blasting program test area, or the Contractor can set up an accounting system to keep track of these costs in the test area.

10. Contractor agrees to record ground vibrations and air blasts on all experimental blasts with its own equipment. Contractor shall select the areas to position this equipment but agrees that its selection shall not interfere with the operation of the mine by the Owner. If the Owner finds that the location of this equipment interferes with normal mining activity, Contractor agrees to move it to a location where it will not interfere with the normal mine operations.

11. Owner shall give Contractor access to survey information in Contractor's Blasting Program test area so that Contractor can more easily calculate earth volumes in the test area. In the event that Owner has not surveyed the area, Contractor may conduct his own surveys to acquire the necessary information. Contractor agrees not to interfere with mining operations and agrees to give owner all information on any surveying that he does.

12. Contractor agrees to give Owner the results of its blast program findings as soon as they become available and after this contract terminates, Owner may use any of the improvements found in its operations with no obligation to the Contractor.

13. Contractor agrees to indemnify and save harmless Owner against any and all loss and expense, including attorneys' fees and other

legal expenses, by reason of liability imposed or claimed to be imposed by law upon Owner for damage because of bodily injury, including death at any time resulting therefrom, or on account of damage to property sustained by any person or persons arising out of or in consequence of the performance of the work, whether or not such bodily injuries or damage to property are due or claimed to be due to any negligence or acts, including violation of any duty imposed by a statute, ordinance or regulation of Contractor.

14. The Contractor will obtain and continue in force, during the term of this agreement all policies required by the Department of Energy under the terms of its agreement with the Contractor. The following is a copy of the agreement between the Department of Energy and the Contractor. (Note: the contracting officer referred to in the following section is an agent of the United States Department of Energy and is the liaison between the Department and the Contractor.)

> "(a) The Contractor shall procure and thereafter maintain workmen's compensation, employer's liability, comprehensive general liability (bodily injury), and comprehensive automobile liability (bodily injury and property damage) insurance, with respect to performance under this contract, and such other insurance as the Contracting Officer may from time to time require with respect to performance under this contract: Provided, That the Contractor may with the approval of the Contracting Officer maintain a self-insurance program: And provided further, That with respect to workmen's compensation the Contractor is qualified pursuant to statutory authority. All insurance required pursuant to the provisions of this paragraph shall be in such form, in such amounts, and for such periods of time as the Contracting Officer may from time to time require or approve, and with insurers approved by the Contracting Officer.
>
> "(b) The Contractor agrees, to the extent and in the manner required by the Contracting Officer, to submit for the approval of the Contracting Officer any other insurance maintained by the Contractor in connection with the performance of this contract and for which the Contractor seeks reimbursement hereunder."

15. Contractor has subcontractors working on this project which aid the Contractor with some details of this work. Contractor may appoint one of these subcontractors as his agent on any specific tests. The Owner will be notified of all such appointments. Such subcontractor will be subject to all of the terms of this agreement but will act for and in place of the Contractor.

16. The terms of this agreement shall commence on __________ __________, 19__ and shall continue until __________________, 19___. This contract may be terminated by either party prior to the expiration

of its term, upon the happening of any one of the following events:

a) Failure of either party to well and truly perform each and all of the covenants herein provided as his part to be performed.

b) Failure of either party to comply with applicable work rules and regulations of both federal and state agencies or to take reasonable safety measures for the protection of persons or property.

c) Discovery by either party that the other party has committed waste and that the acts constituting such waste are either wilful or the result of gross negligence or incompetence on the part of the other party, agent, associate or employee of said party.

The aggrieved party shall give the other party twenty (20) days previous written notice of any of the foregoing grounds for forfeiture or termination hereof; and unless the other party shall cure such breach within said twenty days, forfeiture and termination of the other party's rights shall follow as a matter of course.

17. This agreement may not be assigned by either party without the prior written consent of the other party. The appointment of subcontractors as the Contractor's agent as outlined in section 15 does not violate this clause.

18. All notices and other communications required or permitted hereunder shall be effective when deposited, postage prepaid and certified in the United States mail addressed as follows:

If to Contractor:

Woodward-Clyde Consultants
Rocky Mountain Region
2909 W. 7th Avenue
Denver, Colorado 80204

If to Owner:

____
____
____
____

Either party may, by notice given as aforesaid, change its address for the purpose of this paragraph.

IN WITNESS WHEREOF, the parties have executed this instrument on the day and year first above written.

WOODWARD-CLYDE CONSULTANTS

by ______________________________
(title)

____

by ______________________________
(title)

APPENDIX C

LITERATURE SEARCH AND ABSTRACTS

1. Andrews, A.B.; Air Blast and Ground Vibration in Open Pit Mining; Min. Cong. Journ., May 1975, pp. 20-25.

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25. Porter, D.D.; Use of Rock Fragmentation to Evaluate Explosives for Blasting; Min. Cong. Journ., Jan. 1974, pp. 41-43.

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27. Turner, T.D.; Overburden Preparation Moura Kianga Coal Mines, Bowen Basin; Aust. IMM Conference, Southern and Central Queensland, July 1974, pp. 315-321.

ANDREWS, A.B., *Air Blast and Ground Vibration in Open Pit Mining*

Millisecond delays substantially reduce ground vibration and improve fragmentation. Recommends 1 ms/ft of burden as delay for good fragmentation. Experimental procedure is described and conclusions drawn as follows: Peak particle velocity and frequency of ground motion were made ineffective by changes in timing; blasts of short duration produced vibration of shorter duration and energy but increased burden displacement. Simultaneous firing of opening holes should be avoided to minimize air blast.

ANON, *Special Report, Surface Mine Productivity, - Regional Aspects Affect Planning of Surface Mining Operations*

Methods of surface mining are discussed for four regions:

1. E. Ky., W. Va., Va., Tn. Stripping using drilling and blasting (7 to 10 inch holes + ANFO) and dozers.
2. Pa., Md., Ala., S.Ohio. Stripping using draglines or shovel/trucks. No D and B described.
3. W.Ky., Ill., Ind., Ohio, Ks., Iowa, Ark. Stripping using draglines. No D and B described.
4. Az., Colo., Mont., N.Mex., N.Dak. Various Methods used.

ANON, *Special Report, Surface Mine Productivity, - Draglines - High Producers With Maneuverability*

The operating advantages of draglines as stripping tools are discussed. Types and specifications of walking and crawler draglines are tabulated. Discussions of electrical and mechanical operation and methods of increasing productivity using data loggers, optimizing maintenance and availability.

ANON, *New Company Finds the Road Work Know-How Ideal for Surface Mining*

Small coal operation in mountainous area is described. Rural Mining Co., Hurley, Va. Pit mining involves cutting and maintaining a 60-ft bench with minimum of overside spillage. Thirty-five feet of dense sandstone overburden prepared using Robbins and Damco blasthole drills and ANFO (1 - 1.5 lb/cu yd). Overburden stripping by shovel/grader operation. Twelve-foot parting removed by dozers, coal loaded by F.E. loaders. Reclamation process is described.

ANON, Surface Mining, - Drilling and Blasting - Removing Overburden

The aim of overburden preparation is to get maximum fragmentation with the least drilling and the most effective use of explosives. The benefits of the seismic method of overburden analysis are described. Drilling operating costs (1961-1969) and methods for 6 operations are described as are explosives used and methods of loading and handling. Dragline, shovel and bulldozer stripping methods and methods of improving their efficiency are discussed.

ANON, How To Get More Bang From Your Blasting Buck

Trend to large diameter holes for overburden breakage is described. Breakage ability of explosive not dependent on VOD but the effect of a primers det. pressure to initiate an ANFO column. Noise and vibration problems are also discussed.

ANON, Priming Large Diameter ANFO Charges

During the past 20 years, single, axial, multiple-point, thermal and shock priming were all considered optimum. Primer size has effect on overdrive or underdrive to steady state VOD. Ultimate VOD unaffected by primer type or size. VOD not important parameter in rock breakage, work by Swedish Detonic Foundation and Ash quoted. Primer locations, size and type are considered. Primer should be in area of burden giving most problems - i.e., the toe or hard zone of burden. High energy booster with primer will give max. blasting action prior to loss of confinement due to burden movement. To prevent coal damage by the priming, charge higher in column may be preferable. With good ANFO mixture, multiple priming is not necessary, but multiple primers may be necessary for decked charges and may be necessary for burdens with slip planes or burdens with more than one hard band. Axial priming is described; advantage of no loss of confinement due to burden movement. The use of proper detonating cord downlines, with possibly higher costs, is considered important.

ANON, Dragline Productivity - It's Important To Get It Right In the Planning Stage

The dragline has become the prime stripping machine due to its flexibility on digging and placement of overburden and its capability of operating in variable overburden depths. Methods of increasing dragline productivity are considered; these include: increasing bucket load, increasing hoist and drag speeds. Thirty percent boost of dragline horsepower would result in only 5% return in productivity.

Alternatives to draglines are also considered and each shows advantages in specific conditions, these include: stripping shovels, hydraulic excavators, haulback techniques.

ANON, Peabody Coal Co. Model Mining Issue

The operations of the Peabody Coal Co. are described. In areas where hard materials overlie the seam, horizontal drilling is used. Horizontal drilling gives better fragmentation at shallow depths. Angled drilling gives better highwall stability. ANFO used as basic explosive. Delays used to better fragmentation, especially in shale/sandstone/limestone laminar formations. Detonating cord is being phased out due to noise-vibration problems. Regular studies to review blasting efficiency are carried out. A bucket loading time of 15 sec. for a dragline is considered to indicate proper fragmentation and highwall preparation. Selected loading and hauling methods are described and a complete list of mining methods by operation given.

ANON, Open Pit Coal Extraction at Westfield in Scotland

The open pit mine at Westfield is described, together with its geology and history. Monthly production is approximately 800,000 yd^3 of coal and rock. Box cut 2,500 ft long x 1000 ft wide x 500 ft deep is made using 25-ft benches in hard abrasive sandstone. Forty-foot wide benches left every 75 feet. Drilling of overburden with Joy 58BH rotary machines, GD RDC 16 rotary drills or Housherr HB 20 KHY machines. Hole size of 5 1/8 - 6 1/4-inch. Twenty-five-foot hole plus a 3-foot subgrade pattern normally 18 x 18 feet though down to 9 x 9 feet in hard dolerite. ANFO used in 1/3 of blasting (in favorable conditions), primed with gelignite and initiated by Cordtex detonating cord. In wet conditions - poor fragmentation even with dryliners. PF of up to 3.3 lb/yd^3 in the worst conditions. N.G. sensitized slurry manufactured on site (Cosminex slurry) now used. Loading by 150-RB shovels into 50-ton trucks. The service, dispatch and other operational services are then described.

BERGMANN, O.R., WU, F.C., and EDI, J.W., Model Rock Blasting Measures Effect of Delays and Hole Patterns on Rock Fragmentation

Details of model blasting tests in 15-foot granite blocks are given. Interaction between primary stress waves from adjacent holes did not noticeably affect fragmentation. Short delays cause poor fragmentation. Best fragmentation with 1-2 ms delay per foot of burden. With rectangular patterns with spacings larger than burdens, better fragmentation was obtained than with square patterns.

BERGMANN, O.R., RIGGLE, J.W., and WU, F.C., Model Rock Blasting - Effect of Explosives Properties and Other Variables on Blasting Results.

Results of instrumented model blasting experiments in homogeneous granite and limestone showed that rock fragmentation is not controlled by a single explosive property but by a combination of several properties. Empirical fragmentation equations were developed for granite, limestone and sandstone. From these relationships an expression was developed that may be used to rate fragmentation performance of different explosives. The results also indicated that bore-hole pressure can be related to fragmentation for PETN based explosives provided the proper corrections are applied to the measured pressure values.

BHANDARI, S., BUDAVARI, S., VUTUKURI, VLS., A Laboratory Study of The Effect of Burden and Spacing Parameters on Rock Fragmentation in Blasting

During recent years many investigators have suggested large spacing to burden ratios to impove the fragmentation obtained in blasting. In spite of these suggestions, most operators still use a conventional ratio of 1:2. From single hole longitudinal charge tests in the laboratory, it is shown that burden controls the fragmentation. The burden for obtaining uniform fragmentation is smaller than the burden to obtain maximum breakage. Tests on two or three holes show that a larger spacing burden ratio can be used for small burdens, but it is not possible for the larger burdens chosen to obtain maximum breakage. A finite-element study is used to justify this evidence.

DAVIS, H., Balanced Draglines Dig Deep Cover

The Jeddo-Highland Coal Co.'s Hazleton operation is described. Anthracite deposit in synclinal form with steep dips, previously maximum depth of extraction was a 75 feet down dip. Working by two back-to-back pits, each with three draglines, leapfrogging action between pits. For each pit a relay rehandle system is used: a 15-yd^3 dragline prepares overburden for blasting and dumps into pit, an 85-yd^3 dragline, operating in the pit, removes remaining overburden and stacks spoil. Spoil then restacked by 25-yd^3 dragline. Overburden preparation by 2 BE 61R machines using 14-inch holes 220 feet deep to the coal. Shock subs used to improve penetration rates. Blasting using ANFO detonated with 60% gelatin dynamite primers through detonating cord. Coal loaded by FE loaders into twenty 65-yd^3 haul trucks. Coal preparation is briefly discussed as are future production plans.

FERKO, M.R., <u>An Analysis of Strip Mining Methods and Equipment Selection</u>

A review of strip mining practice is given for several unnamed U.S. mines. Reviews of mining equipment are given and new advances envisaged. The following is a summary of overburden preparation data for the mines listed which utilize draglines.

<u>Eastern Mine 1</u>:

Overburden of sand, rock, and shale to depths of 50 feet.
Drilling by Davy drill 6-inch holes on square grid of 15 to 25 feet.
300 pounds ANFO per hole with dryliners fired electrically.
Dragline - 4600 Manitowoc 7 yd^3 with 120-foot boom.

<u>Mideastern Mine 3</u>:

Overburden of shales, mudstones, lime rock and sandstone to 155 feet.
Drilling by BE 61R on 30 x 30-foot grid.
1 1/2 - 5 1/2 tons of ANFO per hole PF 1 pound/yd^3.
Dragline BE 4250W 220 yd^3.

<u>Central Mine 4</u>:

Overburden consists of clays, sandstones and shales of 60-70 feet. Then parting of 20-30 feet shale.
Drilling by BE 61R on a 30 x 34-foot grid for overburden and Marion MR II drill on 30 x 27-foot pattern for parting.
Blasting by ANFO.
Dragline 8900 Marion with 145 yd^3 and 250-foot boom.

<u>Central Mine 5</u>:

Overburden of clays, sand, gravel and shales to 85 feet.
Drilling by Robbins R.R.10 with 10 5/8-inch holes - 38 to 40 feet deep on a 30-foot square grid.
Blasting by 100-pound ANFO per hole, primed by 1-pound cast primer and initiated by primacord PF = 0.075 pound/yd^3.
Stripping by BWE and Marion 7800 30 yd^3 dragline.

<u>Western Mine 8</u>:

Overburden of 50-90 feet, parting of 20-30 feet.
Drilling by BE 50R 10 5/8-inch drill on 30 x 30-foot grid in both parting and o.b.
ANFO used as blasting agent.
Dragline Marion 7400 14 yd^3 with 175-foot boom.

FISH, R., Fordings Dual Operations are Unique in Canada

Two surface coal mines in B.C. are cited, covering the dragline operation at one pit and a truck-shovel operation at another pit. Drilling utilizes BE 45R and 60R drills with 9 7/8 and 12 1/4-inch holes, respectively. Depth is 40 feet on a 24-foot square pattern. Slurry exp. Nitrex is generally used with a load factor of 1.85 pound/yd^3. In dry months AL/ANFO or ANFO is used with a load factor of 1.75 pound/yd^3. Stripping is either by Marion 8400 dragline or a fleet of 15 yd^3 shovels and trucks. Hydraulic mining plans, reclamation and the coal washing plant are also described.

HAGAN, T.N., <u>Blasting Physics - What the Operator Can Use in 1975</u>

Blasting effectiveness must be guaged by total production costs. ANFO charges made up in polythene "sausages" are not as profitable as is indicated by the price of ANFO. All-slurry charges are warranted only where blastholes cannot be dewatered effectively, or where drilling costs are very high.

An increase in blasthole diameter leads to coarser fragmentation. This effect is most pronounced in tough and massive or blocky strata, or in ground which consists of unfissured boulders in a softer matrix. When ANFO and some slurry-type charges of small diameters are used in plastic-acting ground, charges may fail to detonate through precompression by charges fired on an earlier delay. The optimum blasthole length is rarely less than four times the burden distance. For best result, blastholes should be inclined to the vertical and parallel to the face. Nominal burdens and spacings are often altered radically by the initiation sequence. For best fragmentation, all blastholes must have good effective faces, and should be effectively staggered with an actual spacing:burden ratio in the approximate range 2.0 - 5.0.

Relatively long subgrades may be necessary in some blastholes in the front row. Greater attention should be paid to the effects of "fallback" on (nominal) subgrade. If subgrade is too great, drilling and blasting costs are squandered, and drilling the bench beneath may become more difficult.

Fragmentation of rock alongside the stemming column can be improved by using "pocket" charges. Where the burden is densely fissured, long stemming columns allow reductions in total production costs. Longer stemming columns are profitable in the back row of multi-row blasts.

Except where considerable (bonus) backbreak production is achieved, multi-row blasts give lower production costs than single-row shots. The optimum width:length ratio of the blast area is usually less than 0.5 for firing to an open face, and less than unity when firing to a free end.

Fragmentation improves when the inter-blasthole delay allows the crack system around each blasthole to develop fully before the charge in the next blasthole detonates. The reliability and quality of blasting results are very dependent upon the amount of attention given to initiation timing.

HARRIES, G., and MERGER, J.K., The Science of Blasting and Its Use to Minimize Costs

This paper gives a description of the research undertaken by an explosives manufacturer to give better technical advice to the Mineral Industry.

Blasting Physics is defined as giving a systematic and coherent account of the initiation and subsequent effects of high explosives upon the surrounding medium. As a result of this work a mathematical model based on the generally accepted mechanism of blasting has been developed. The mechanism and the resulting model are briefly outlined. From this model the effects of burden, spacing, blasthole diameter, rock and explosive properties upon fragmentation and heave can be calculated.

Using these calculations it is then possible to see how variations in fragmentation and heave can affect the overall production costs of a mine whether it is surface or underground. The overall costs are made up of a number of individual costs - drilling, blasting, loading, hauling and crushing. Each of these operations influences or is influenced by fragmentation and heave. As these costs are interrelated, the effects of all the variables determining fragmentation upon the costs of the individual operations have to be taken into account so that rock breaking costs can be optimized.

JACKSON, D., Modern Equipment, Town, Reclamation Give Western Energy Bright Future

Western Energy's mine at Colstrip, Montana, is described. Twin seam sub-bituminous, flat lying deposit. Until 1968, the operation used old equipment. Dragline enables higher highwalls than previously used shovels. Topsoil removed by scrapers. Overburden of 97 feet drilled with BE 45R on a 27 x 30-foot pattern using 11-inch holes. Dewatering of 75 % of holes and dryliners fitted. Charge of ANFO prills with 25-pound bag of Monsanto slurry as booster. Primer of one stick 85% gelatin dynamite with detonating cord. Powder factor of 0.33 pound/yd^3 for good fragmentation of sandstone. First 40 feet of overburden removed by tractor scrapers and remainder by dragline or shovel.

Twenty-six feet of coal blasted using 6-inch holes on 18 to 20-foot centers. ANFO with gelatin primers in drylined holes gives PF of 0.33 pound/ton of coal. Loading by 17 yd^3 shovel into 100 ton or 120 ton trucks. The coal plant, reclamation and the town are then described.

JACKSON, D., Montana-based Westmoreland Resources Mines Crow Indian-owned Coal at Absaloka Mine

Development of new mine is described. Four seams of total thickness 58 feet. Up to 5 feet of top soil removed and stored for later reclamation. Overburden drilled with BE 45R using 11-inch holes on a 30 x 30 or 31 x 31 foot pattern. Loaded with ANFO. Stripped using a 75 yd^3 dragline. Coal prepared by GD drill, 6-inch holes charged with ANFO. Loading using FE loaders into 115-ton tractor/trailer coal haulers. Interburden ripped with dozers as is the 2nd seam coal prior to loading. Second Interburden (60 feet thick) is drilled and blasted as per the overburden. The coal preparation plant and training departments are then discussed.

JUNK, N.M., Overburden Blasting Takes On New Dimensions

Studies by Atlas Chemicals showed that initiation of large diameter blasts with ANFO were not as effective as small diameter. Results show primer should match hole diameter, have sufficient length and high detonation pressure. Common practice to use 1-pound primers and sand decking to get good powder factor for limited breakage. Use of slurries as initiators recommended - referred to as combination priming or slurry boostering. Studies show 15% cost saving using such a system and 30-40% increased breakage.

LEARMONT, T., <u>Productivity Improvement in Large Stripping Machines</u>

The emergence of the dragline as the dominant stripping tool is described and reasons for this are noted. Brief comparisons are made with stripping shovels and wheel excavators. Representative output and machine availability figures for draglines are presented. The effect of changes in machine parameters such as boom length, swing power, and hoist power on machine productivity are discussed. An analysis of machine availability and the causes of machine downtime is made. Measures which are being taken to improve machine availability are discussed in detail. These include (1) improved methods of structural analysis, standardization of welded joints and welding methods to increase structural reliability and to develop more efficient structural designs, (2) standardization of gearing and mechanical componentry to permit the use of proven designs, (3) review of fairlead design and bucket hardware to improve rope life and reduce downtime, and (4) modifications to reduce maintenance time. The General Electric Data Logging system is described and consideration given to its use as a tool for improving productivity. Areas for future improvements in productivity are discussed.

LIVINGSTONE, G.K., <u>Surface Mining of Coal at Sparwood, B.C.</u>

At Sparwood, B.C. Kaiser Resources Ltd. obtains 85% of its raw coal production from surface mining. The strip mining procedures involved utilize some of the largest equipment available. The holes are 12 1/4-inch in diameter, and are drilled on a 30 x 32-foot pattern. The 9 7/8 inch holes are drilled on a 26-foot square pattern. Hole depth is 50 feet and subgrade is 10 feet. Six to ten rows are blasted per blast; ANFO is the prime explosive, with a slurry toe load in hard conditions. Dryliners are used, and 25 ms delays are used between rows. Loading of broken material is by 25 yd electric shovel and 200 ton trucks. Equipment maintenance, production methods, quality control, reclamation and waste disposal are also considered.

MELNIKOV, N.N., The Soviet Union - Recent and Future Developments in Surface Coal Mining

The general characteristics of the Soviet coal deposits are described and brief statistics are given on surface operations. Drilling and blasting is required of 85% of overburden. Normal hole size is 8-10 inches for rotary drilling. Both slurries and ANFO used. Millisecond-delay multiple row blasts. Air gaps in charge decrease the initial gas pressure, thus preventing excessive crushing around the charge and increasing time of energy application to the rock. This results in uniform fragmentation, reduced PF and reduced vibration. Dragline and shovel equipment, and transport systems are described. Dragline applications in open pit coal mines, especially in relatively steeply inclined seams (17-45°) are outlined.

MERRITT, S.E., Cimarron Strip; Providing Coal for Energy Markets

Multi seam operation at Volunteer Mine, Madisonville, KY, is described. Robbins RR 105 and RR-10 are used for overburden drilling. Nine-inch holes in sandstone and limestone. Penetration rates of 71 feet/hr, 70 feet of overburden. Holes loaded with ANFO (dryliners used) and high velocity primers. Drill cuttings used as stemming. Stripping used 45 and 7 1/2 CY draglines. For sandstone PF is 0.5 pound/CY. For shale 27 x 24-foot pattern with PF of 0.32 pound/CY. Coal removed by ripping, loaded by 61B and 71B BE shovels. Seven-foot hard limestone parting is removed by 9-inch holes on 9 x 9-foot pattern loaded with 12 pounds ANFO per hole - PF of 0.57 pound/CY. Both the overburden and parting are fired 16 holes at a time with 1/2 sec delays. The haulage, preparation and train loading facilities are also described.

PORTER, D.D., Use of Rock Fragmentation to Evaluate Explosives For Blasting

Fragmentation is the paramount blast factor that can be used as a basis for evaluating explosives performance. Experiments carried out on large blocks of varying types. A mathematical relationship between fragmentation effectiveness of explosives and such parameters as VOD, explosive energy, density and rock sonic velocity is drawn. This is also correlated with the coupling ratio and burden.

PORTER, W.E., Multiple Seam Strip Mining, A Survey and Economic Feasibility Model

A survey of multiple seam strip mine operation is given together with operational details. A feasibility model is given which carries out sensitivity of operation to DCF rate of return, operating costs, depth of overburden, etc.

TURNER, T.D., Overburden Preparation Moura Kianga Coal Mines, Bower Basin

The Peabody operation at Moura Mine is described. The specs of the BE 61R are given for 15-inch holes. Fragmentation influenced by nature of rock, drill patterns and delays, stemming, hole size, PF, types of explosive and hole wetness. Rock types are discussed. Detailed description is given to drill patterns and delays. Ten ms delays in jointed ground, 17 ms in stronger ground. Blast tied in echelon technique. Ten to 18° holes from vertical. Ten percent backfill of holes use drill cuttings. Decked charges - diagrams given. Wet holes frequently encountered - slurries used. Details of priming system are given.

APPENDIX D

BIBLIOGRAPHY

NOTE:

The Bibliography is incomplete before 1965. Because of the tremendous technological change since this period, many of the early papers were out-dated, and therefore, not included.

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