EXPLOSIVE FRAGMENTATION OF OIL SHALE:
RESULTS FROM COLONY AND ANVIL POINTS MINES, COLORADO

by

Richard D. Dick and William L. Fournery
Department of Mechanical Engineering
University of Maryland
College Park, MD 20742

and

Chapman Young, III
Sunburst Recovery, Inc.
Steamboat Springs, CO 80477

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ABSTRACT

From 1978 through 1983, numerous oil shale fragmentation tests were conducted at the Colony and Anvil Points Mines, Colorado. These experiments were part of an investigation to determine factors required for the adequate fragmentation of oil shale and to evaluate the feasibility of using the vertical modified in situ retort (VMIS) method for recovery of kerogen from oil shale. The objective of this research was to support the design of a large volume (10^4 m^3) rubble bed for in situ processing. In addition, this rubble bed was to be formed in a large single-blast event which included decked charges, time delays, and multiple boreholes.

During 1978 and 1979, numerous tests were conducted at Colony Mine to provide basic data about cratering and fragmentation. Then, in 1981 and 1982, an extensive field testing program was sponsored by a Consortium of oil companies and managed and directed by Science Applications, Inc. The field tests were conducted at Anvil Points Mine in Rifle, Colorado, and focused on the creation of an in situ retort. In the 1983 and 1984 period, Los Alamos National Laboratory and Sandia National Laboratories conducted additional tests to provide data in a few areas of oil shale blasting that needed attention as identified in the Consortium program of 1981 and 1982.

A large body of data was obtained from the 80 tests conducted in oil shale at the two mines. These tests ranged from very small, very simple configurations (single charges of 10 kg) to large, very complex blast designs (32 charges of 90 kg). The data obtained included rock mass motion, crater features, explosive engineering factors, particle size distributions, and much more.

Some basic conclusions from the many tests are that oil shale is a very difficult rock to fragment into a uniform size, deep rock motion is small during the first 50-100 ms, several explosive engineering problems need solutions before VMIS retorting is feasible, and horizontal and vertical spacing and timing effects on fragmentation need additional research. Additionally, the assay of the oil shale is of minor importance in a complex blast array, but the blast parameters (spacing and timing) and stemming performance in complex geometry are very important. For the single-borehole cratering geometry, the geological influences (joints and layering), and charge burial depths are important. The generally accepted scaling rules are adequate for describing the cratering results. At present, the technology is not available to produce a rubble bed for in situ retorting by means of a single large blasting scheme. With the Department of Energy's interest in fragmentation research declining, there appears to be little hope of developing an in situ retort capability in the near future. The hope is that

future fragmentation programs on oil shale will have the benefit of the results from the Colony and Anvil Points Mines experience as reported here.

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CHAPTER 1 - INTRODUCTION

A. Research Programs

The primary motivation for the fragmentation of oil shale is to provide broken rock in a suitable size for retorting to economically convert the kerogen to oil and remove this oil from the shale. To date, most oil shale retorting has been done on the surface with the oil shale being mined, taken to the surface, run through a crusher to produce a uniform particle size, retorted, and the spent shale placed in piles on the surface. In situ retorting involves making a rubble bed by explosive fragmentation, retorting underground, and pumping the oil and gases to the surface. This reduces the mining, minimizes haulage to the surface, and the spent shale remains underground. Several methods for forming a rubble bed are vertical and horizontal free-face blasting, modified in situ (MIS) [Ricketts, 1984], Geokinetics horizontal in situ retorts [Britton and Edl, 1985], and single-blast MIS retorts. A variation of the MIS method called vertical modified in situ retort (VMIS) is the basis for the fragmentation research described in this report. This technique involves a room mined above and below the column of rock to be fragmented and retorted to provide a 20-30% void volume for the oil shale to occupy after the blast event and involves creating the final rubble bed in a single blast event using decked charges and sequenced time delays.

Since adequate fragmentation is essential for rubble bed preparation in the VMIS method, the research program was aimed at (1) determining the important parameters involved in fragmenting oil shale, (2) identifying geometric and geologic effects important in fragmentation, and (3) investigating explosive engineering factors that control consistent and predictable fragmentation. The Consortium and Department of Energy (DOE) programs included laboratory tests for rock characterization, numerical modeling to develop a predictive capability, and extensive field testing, all directed toward oil shale fragmentation. Only the results from the field test program will be discussed in this report. The field tests were conducted in the Parachute member of the Green River formation that covers northwestern Colorado, northeastern Utah, and southwestern Wyoming. Figure 1.1 is a map illustrating the extent of the oil shale deposits in the United States, and Figure 1.2 shows the deposits in the Colorado-Utah-Wyoming region. The DOE sponsored the involvement of Los Alamos National Laboratory in this program as part of the DOE-sponsored research in rock fragmentation.

This report is one of many describing the experiments conducted at Anvil Points and Colony Mines. Other reports cover specific tests, summaries, and evaluations of the fragmentation programs. Some are referenced at the end of this report.

Figure 1.1 Oil shale deposits in the United States.

OIL SHALE IS A CONSIDERABLE RESOURCE

ESTIMATED 3 TRILLION BARRELS OF OIL IN WESTERN OIL SHALE DEPOSITS
PLUS NAHCOLITE AND DAWSONITE

Figure 1.2 Oil shale resources in the Colorado, Utah, and Wyoming area.

B. History and Field Test Programs

Los Alamos began fragmentation field tests in the ARCO/TOSCO (later EXXON/TOSCO) Colony Mine located near Parachute, Colorado. The location is shown in Figure 1.3. During 1978 and 1979, 19 single- and multiple-borehole tests were conducted using ANFO explosives (ammonium nitrate and fuel oil mixture). These tests were designed to provide fundamental data on fragmentation and cratering, particle-size distribution, cratering features, applicability of scaling relations, and effects of multiple-borehole blasting. In parallel with the field programs, a computer program was being developed to simulate rock blasting [Adams et al., 1985]. The field tests provided the data to guide and calibrate the computer calculations. Specifically, the single-borehole tests provided data on optimum depth-of-burial (burial depth of the explosive as measured from the surface to the charge center for which the largest crater volume is obtained), critical depth-of-burial (burial depth of the explosive charge at which no surface crater is formed), particle-size distribution and average particle size, and crater dimensions and volume. The four-borehole tests provided data on the extent of fragmentation, effects of horizontal spacing of the explosive on fragmentation, depth-of-pull, and particle size distribution. In addition, the influence of common geological factors such as joints, bedding planes, and vugs on cratering and fragmentation were documented. The results from this series of tests were utilized in the next test program. An outline of the test programs is shown in Table 1.1. Figure 1.4 is a map of Colony Mine showing the location of the tests. Preliminary results from this research have been published by Harper and Ray, 1981.

In 1981 and 1982, Los Alamos participated in an extensive fragmentation research program with a Consortium of oil companies (Cities Service Co., Getty Oil Co., Mobil Research and Development Co., Phillips Petroleum Corp., Sohio Shale Co., and Sunoco Energy Development Co.) with Science Applications, Inc. (SAI) managing and directing the project. The field program was conducted at the Anvil Points Research Mine which was operated by Development Engineering, Inc., a subsidiary of Paraho Corp., under contract to the DOE. The Anvil Points Mine is located near Rifle, Colorado, twenty miles east of Colony Mine. Fifty-seven field experiments were conducted at the Anvil Points Mine over a one-year period. The experiments ranged from small-scale tests using less than 0.3 kg of explosive, to single-borehole tests using 30 kg of explosive, to 16-borehole, multiple-level mini-retorts using up to 3000 kg of explosive. The test series consisted of five sets of 45 large-scale experiments designated the A-, B-, C-, D-, and Special-Series and 12 small-scale tests designated CT- and K- (for charge timing and kerfing) series. An outline of the test series is listed in Table 1.1. The Special-Series tests were added to the original research plan to address specific problems that were encountered as the program

Table 1.1
Test Plan and Major Objectives for the Colony
and Anvil Points Mines Field Experiments.

Colony Mine 1978-1979 Los Alamos/ARCO Program

- o Single-Borehole Series (16 Tests)
 - determine optimum and critical depth-of-burial.
 - establish crater parameters such as powder factor, volume, and dimensions.
 - determine the particle size distribution as a function of depth-of-burial.
 - evaluate scaling relations.
 - investigate geological influences on fragmentation.
 - investigate in situ rubble bed.
- o Multiple-Borehole Series (3 Tests)
 - compare cratering features with single borehole results.
 - evaluate effects of horizontal spacing on crater formation and fragmentation.
 - compare particle size distribution with single borehole results.
 - investigate geological influences on fragmentation.

Anvil Points Mine 1981-1982 Consortium Program

- o A-Series: Single-Hole (6 Tests)
 - Compare the effects of TNT and ANFO prills on fragmentation.
 - Establish particle size distribution, crater dimensions, and depth-of-pull.
 - Observe the effects of joint spacing and distribution, shale grade, and limited free face on fragmentation and crater dimensions and shape.
 - Provide an estimate of the borehole spacing for the B Series tests.
 - Replicate Shot 79-10 conducted at Colony Mine to compare cratering and fragmentation characterization for two oil shale mines.
 - Rock motion and displacement data at the surface and at depth by means of accelerometer and velocity gauges.

Table 1.1
(continued)

B-Series: 5-Borehole, Single-Deck Pattern (4 Tests)

- Evaluate horizontal spacing and timing on crater formation and fragmentation using TNT prills.
- Measure particle size distribution, crater dimensions, and depth-of-pull in a multiple-borehole geometry.
- Evaluate changes in depth-of-burial and charge weight on particle size distribution and crater formation.
- Measure rock mass motion using accelerometer and velocity gauges mounted on the free face and embedded in the rock.
- Establish horizontal spacing for 8- and 16-borehole tests in the C and D series tests.

o C-Series: Vertical Multiple-Level in a Multiple-Borehole Arrangement (15 Tests)

- Study and confirm the explosive and stem performance in decked boreholes prior to conducting the C Series multiple-borehole fragmentation tests.
- Obtain data on effects of horizontal spacing and timing, vertical spacing and timing, staggered decking, and charge overlapping prior to conducting the D series tests.
- Evaluate multiple-level interaction effects for 8, 16, and 24 charges.
- Investigate explosive deadpressing and sympathetic detonation for several explosive types.
- Evaluate electric blasting cap performance in decked charge geometry.
- Fragmentation results from multiple-borehole, multiple-level charge placement in a decked pattern.
- Measurement of rock mass motion from accelerometer and velocity gauges and framing camera coverage.

o D-Series: Small-Scale Retorts (3 Tests)

- Evaluate fragmentation from a 16-borehole staggered pattern of 24 and 32 charges in 3- and 4-level configurations.
- Establish reproducibility of mini-retort designs.
- Conduct reverse-pull fragmentation study.
- Evaluate performance of stem designs, electric blasting caps, and IRECO 1205C ANFO slurry.
- Evaluate effects of geologic features on fragmentation in a complex geometry.
- Rock mass motion measurements within the 16-borehole array and outside the blast location.

Table 1.1
(continued)

- o Special Series: Single- and Multiple-Borehole (17 Tests)
 - Study special problem areas that affect fragmentation of oil shale such as stem performance, crater formation mechanism, crater profile produced by reverse plane configuration, fragmentation of rich oil shale beds using different explosive slurries, deadpressing in single-borehole and decked arrangements
- o CT Series: Small-Scale Charge Timing (6 Tests)
 - Study effects of charge detonation timing on fragmentation.
 - Effects of delayed timing in a five-borehole array.
 - Limited void volume in fragmentation.
- o K Series: Small-Scale Kerfing (6 Tests)
 - Kerfing control of fracture formation.
 - Effectiveness of borehole notching on driving fractures.
 - Effects of spacing and delayed timing on fragmentation using notched boreholes.

Anvil Points Mine 1983 Los Alamos and Sandia Program

- o Single-Borehole Series (7 Tests)
 - evaluate stemming designs for use in two extensively instrumented tests.
 - investigate relative effects of gas pressure and shock waves on fragmentation by comparing a stemmed borehole test with an unstemmed (open) borehole test.
 - provide rock mass motion data for computer code calibration.
 - investigate crater formation mechanisms by means of a confined volume experiment.
 - rubble size distribution in lean and rich oil shale.

Figure 1.3 Map of northwestern Colorado showing the location of the Colony and Anvil Points Mines.

Figure 1.4 Colony Mine layout and locations of the fragmentation tests.

developed, such as stem performance and crater formation. Ideally, each succeeding test series was to build on the results of the previous series, starting with uncomplicated single-borehole tests with a single charge, to five-borehole tests each with a single charge, to more complex multiple-borehole tests with one and two charge columns in each blasthole, and ending with mini-retort test design. In a general sense, the execution of the test program went as planned, but many problems arose as the project progressed necessitating additional investigation of these problems. Several experimental problems were not solved [Fourney and Dick, 1985].

Figure 1.5 is a map of part of the Anvil Points Mine showing the location of the 45 large-scale tests. Figure 1.6 shows the range of kerogen content over which the tests were conducted. The small-scale tests were conducted early in the project to study pre-splitting, fracture control blasting, and charge timing. The results of these tests are not included in this report. Nine fragmentation and cratering tests (four by Sandia [Parrish, 1985] and five by Los Alamos) were performed at Anvil Points Mine in 1983. These focused on two general areas identified from the Consortium project as important areas to study. They were gas pressure versus stress wave effects on fragmentation and crater formation mechanisms. All experiments were single-borehole geometry and included diagnostics for evaluating the blast event. Also, many small-scale tests using a few grams of explosive were performed to investigate scaling parameters and geologic influences useful for empirically describing the cratering and fragmentation process. These results from these scaling studies have been published elsewhere [Edwards et al., 1983].

A large amount of data exists on fragmentation from the 80 tests conducted in oil shale over the seven years of testing. These tests range from small, simple configurations to large-scale, very complex multiple-borehole, multiple-level designs and cover explosive engineering factors, horizontal and vertical spacing and timing effects, and geological influences. Most of the tests were instrumented and monitored by a variety of techniques to determine explosive performance, stem behavior, and rock motion. Some of the test sites were mapped in detail to document the joint pattern and frequency. The sites were surveyed to provide locations of the boreholes and instrumentation and to identify the shale grade along the length of the borehole. Where appropriate the craters were excavated, the rubble screened to determine the particle size distribution, and the crater volumes and profiles determined. These data were compared to previous tests to evaluate the effectiveness of the blast configuration.

C. Report Outline

This report covers four areas of the fragmentation research: instrumentation, fragmentation phenomena, stem behavior and performance, and explosive engineering considerations. Most of the data reported here was obtained during the

Figure 1.6 Shale grade for the Consortium tests.

Consortium-sponsored program at the Anvil Points Mine. However, the data from Colony Mine and the 1983 test program at the Anvil Points Mine are included (especially the fragmentation results).

Under the heading of instrumentation, rock mass motion measurements at the Anvil Points Mine are discussed including the signal recording system, recording electronics, transducers, and installation of the gauges. Explosive performance and initiation sequence measurements are explained in detail and an interpretation of the results are presented.

Fragmentation phenomena from tests from the three research program periods are presented. This includes a discussion of the crater profiles, particle size distributions, and correlations with other data and measurements.

Stem behavior and performance were monitored and became more important as the Consortium program progressed. The instrumentation data proved very useful, especially when correlating the stem performance with field observations and the extent of fragmentation. Proper stem performance is very important in blast designs that incorporate multiple-borehole, multiple-deck configurations with long delay times between levels. An analysis of the performance data has resulted in the formulation of a computer model that compares well with experiment.

There are several explosive engineering considerations that require attention and, in some cases, a solution [Fourney et al., 1985] before single-blast VMIS rubble beds can be produced. Some of these considerations are discussed in this report in detail. These include detonator, explosive and stem performance, charge timing effects, and reverse plane difficulties.

CHAPTER 2 - INSTRUMENTATION AND RESULTS

A. Rationale

Instrumental and diagnostic coverage of the fragmentation tests were included on most of the experiments to evaluate quantitatively the effectiveness of the shot configuration, since very little prior data existed on the response of oil shale during blasting. These data, along with pre- and post-shot information, provided a more complete picture of the fragmentation process. This knowledge is essential in understanding the performance and effectiveness of a blast in order to design new tests. For example, the measurement of the burn speed of the explosive in the test geometry was important in determining the performance of the explosive compared to "ideal" conditions because the degree of fragmentation is directly related to proper explosive performance. In a similar fashion, rock mass motion at a given time is needed to choose the time delays of the charges to properly sequence a complex blast geometry.

The objectives of the instrumentation were to collect data on initiation time of the explosive, the explosive detonation velocity (which is a function of the explosive performance), stem behavior over long periods of time, stress wave propagation velocities in the shale, acceleration and velocity history of the blast event, and stress-time history in the shale. Pre- and post-shot data are collected to relate the gauge data to the crater and fragmentation results. Preshot preparation included mapping the joint patterns and surface topography of the site; surveying the site to locate the explosive borehole and boreholes relative to the shale grade to be fragmented; measuring the location of the gauges with reference to the borehole; and determining the specifications of the test, such as explosive and stem column lengths, weight of explosive in each wellbore, and details of the stem column. Postshot data were crater dimensions after excavation, rubble sizes from screening, bulking and heave of the free face, and unusual features that affected the character of the crater such as joints.

In the following sections, details of the computer-controlled timing and firing systems (T/F), signal recording, rock mass motion instrumentation, and the explosive performance and stem behavior measurements are discussed. In addition, sample data records are shown and then results and interpretation of the results are presented from both the Colony and Anvil Points Mines tests.

B. Timing and Firing System (T/F) and Signal Recording

The CORTEX (Continuous Reflectometry for Radius versus Time Experiments) technique was used at both Colony and Anvil Points Mines to obtain explosive performance signal data. However, the

electronic systems used to conduct the tests were very different between the Colony and Anvil Points Mines, as were the signal recording methods. At the Colony Mine, the shots were fired manually and the gauge signals were recorded on high-quality tape recorders. At the Anvil Points Mine, the shots were fired using a computer-controlled T/F system and state-of-the-technology transient waveform recorders were used to digitally record and store the gauge signals. The digital data was later stored on diskettes for processing by computer at a later time.

For each test at the Anvil Points Mine, the T/F events were programmed into the computer through a special code to provide triggers in the form of closure voltages to the various types of instrumentation recording units and countdown sequence.

At both mines, the data acquisition systems were located in mobile trailers parked inside the mine approximately 300 m from the experiment sites. The equipment used at Colony Mine was mounted in a single trailer, while at the Anvil Points Mine the computer system and the digital recorder equipment were placed in one trailer and the analog recorders and CORRTX units were located in a second trailer. The real-time instrumentation provided three basic types of information: (1) rock mass motion, (2) explosive and initiator performance, and (3) stem column behavior. A variety of instruments and gauges were used in the tests and included the CORRTX technique [Virchow et al., 1980; Deupree et al., 1981; Storey et al., 1982; Schmitt and Dick, 1985; Dick and Parrish, 1986] to monitor explosive and detonator performance, ionization pins [Dick and Parrish, 1986] for backing CORRTX measurements, accelerometers and velocity gauges to measure rock mass motion and stem behavior, gauges to determine the stress wave amplitude at Colony, and framing and video cameras to observe ground movement.

1. Rock Mass Motion Measurements at the Colony Mine. The measurements at Colony Mine consisted of lithium niobate and ytterbium stress gauges embedded in the oil shale at several slant distances from the explosive charge, velocity gauges mounted on the surface and buried within the oil shale served as time-of-arrival (TOA) monitors of the stress wave as well as rock motion, and framing camera pictures taken at 100 frames per second. The locations for the tests are shown in Figure 1.4. Table 2.1 is a directory of the tests [Harper and Oliver, 1980] listing the experiment design information and the type of instrumentation, and Table 2.2 is a list of the shot data. As indicated, only three tests were instrumented for rock mass motion. CORRTX, however, was used to measure explosive detonation velocity on each test.

a. Instrumentation and Data Acquisition. The gauge signals from the velocity and stress gauges and the TOA monitors were recorded on Ampex magnetic tape recorders (FR-600) mounted in the instrumentation trailer. Figure 2.1 is a schematic of the analog recording system, and Figure 2.2 outlines the specifications for

Table 2.1. Colony Experiment Design Information

Expt no.	Place	Borehole	Detonator Booster Explosive	Stemming	Results	Instru
78-1	floor	1	#6 cap Atlas G ANFOprill	shale fines	crater	none
78-2	floor	1	#6 cap Atlas G ANFOprill	shale fines	crater	none
78-3	rib	1	#6 cap Atlas G ANFOprill	shale fines	crater	none
79-1	floor	1	#6 cap Atlas G ANFOprill	shale fines; grout	no crater surface fractures	none
79-2	floor	1	#6 cap Atlas G ANFOprill	grout	no crater surface fractures	stress velocity CORRTEX
79-3	rib	1	#0 cap Atlas G ANFOprill	grout	crater	stress velocity CORRTEX
79-4	floor	1	#0 cap Atlas G ANFOprill	grout	crater	CORRTEX
79-5	floor	1	#0 cap Atlas G ANFOprill	grout	misfire	CORRTEX
79-6	floor	1	#6 cap Atlas G ANFOprill	grout	misfire	CORRTEX
79-7	floor	1	EBW Atlas G ANFOprill	shale fines	no crater	CORRTEX

Table 2.1. Colony Experiment Design Information (Continued)

Expt no.	Place	Borehole	Detonator Booster Explosive	Stemming	Results	Instru
79-8	floor	1	EBW Atlas G ANFOprill	shale fines	crater	CORRTEX
79-9	floor	1	EBW Atlas G ANFOprill	shale fines	contained volume	CORRTEX
79-10	floor	1	EBW Atlas G ANFOprill	shale fines	crater	CORRTEX
79-11	rib	1	EBW gel&RDX ANFOprill	shale fines	crater low dens charge	CORRTEX
79-12	floor	4	EBW Atlas G ANFOprill	shale fines	crater	CORRTEX
79-13	floor	1	EBW Atlas G ANFOprill	shale fines	cables survived	CORRTEX
79-14	floor	1	EBW Atlas G ANFOprill	shale fines	cables survived	CORRTEX
79-15	rib	4	EBW Atlas G ANFOprill	shale fines	crater	CORRTEX
79-16	floor	4	EBW Atlas G ANFOprill	shale fines	crater hole #2 misfire	stress velocity CORRTEX

Table 2.2. Colony Experiment Design Parameters

Single Borehole Configuration

Expt. No.	Borehole Diam (m)	ANFO Diam (m)	Borehole Depth (m)	Explosive Length (m)	Explosive Mass (kg)
78-1	0.108	0.108	1.98	0.65	5.22
78-2	0.108	0.108	2.55	0.64	5.22
78-3	0.108	0.108	2.59	0.66	5.22
79-1	0.152	0.108	3.78	0.66	5.22
79-2	0.152	0.152	8.64	5.49	90.72
79-3	0.152	0.152	8.99	5.81	90.72
79-4	0.152	0.152	2.59	0.61	9.66
79-5	0.152	0.152	3.35	2.31	25.67
79-6	0.152	0.152	4.13	0.73	5.94
79-7	0.152	0.152	4.57	0.99	14.51
79-8	0.152	0.152	3.31	0.99	14.51
79-9	0.152	0.152	2.59	0.67	9.66
79-10	0.152	0.152	3.31	1.73	24.68
79-11	0.162	0.162	3.48	1.54	14.51
79-13	0.127	0.127	4.11	0.91	7.35
79-14	0.127	0.127	2.54	0.81	7.35

Four Borehole Configuration

Expt.No.	Borehole Diam (m)	ANFO Diam(m)	Borehole Depth(m)	Explosive Length(m)	Explosive Mass (kg)	Borehole No.	Spacing (m)
79-12	0.152	0.152	3.81	1.87	27.25	1	(1-2) 3.23
79-12	0.152	0.152	3.81	1.88	27.25	2	(2-3) 3.05
79-12	0.152	0.152	3.59	1.92	27.25	3	(3-4) 3.12
79-12	0.152	0.152	3.51	1.76	27.25	4	(4-1) 3.25
average	0.152	0.152	3.66	1.86	27.25	eff.	ave. 3.16
79-15	0.108	0.108	3.86	1.93	14.55	1	(1-2) 2.13
79-15	0.108	0.108	3.86	1.93	14.55	2	(2-3) 2.13
79-15	0.108	0.108	3.86	1.93	14.55	3	(3-4) 2.13
79-15	0.108	0.108	3.86	1.93	14.55	4	(4-1) 2.13
average	0.108	0.108	3.86	1.93	14.55	eff.	ave. 2.13
79-16	0.108	0.108	3.09	1.58	11.9	1	(1-2) 2.14
79-16	0.108	0.108	2.99	1.58	11.9	2	(2-3) 2.17
79-16	0.108	0.108	3.02	1.56	11.9	3	(3-4) 2.14
79-16	0.108	0.108	3.08	1.59	11.9	4	(4-1) 2.19
average	0.108	0.108	3.05	1.58	11.9	eff.	ave. 2.16

ANALOG RECORDING TRAILER

Figure 2.1 Signal recording system at the Colony Mine.

INSTRUMENTATION ANALOG RECORDING

PRESENT SYSTEM

➡ MEDIUM SPEED AMPEX RECORDERS (FR-600)

- 60 TO $1\frac{7}{8}$ IPS TRANSPORT
- 30 MINUTES RECORDING TIME AT 60 IPS

*** 1 FM RECORD/REPRODUCE**

- 26 SIGNAL AND 2 TIME CODE CHANNELS
- DC TO 20 kHz FREQUENCY RESPONSE
- 0.25 TO 1.4 VOLTS INPUT

*** 2 DIRECT RECORD/REPRODUCE**

- 13 SIGNAL AND 1 TIME CODE CHANNELS
- 100 Hz TO 100 kHz FREQUENCY RESPONSE
- 1.0 V RMS NOMINAL INPUT

➡ SIGNAL CONDITIONER

- AMPLIFIER / FILTER

Figure 2.2 Analog recording equipment specifications.

the FR-600 recorders. Three recorders, two direct analog recorders, and one frequency modulated (FM) recorder were used. Signal processing by conditioners and amplifiers preceded recording on tape. Long multiple-conductor cables connected the gauges at the test site to the instrument trailer. The velocity gauges, the TOA gauges, and the seismometers were connected to the direct tape machines to provide the maximum frequency response while the stress gauge signals were recorded on the FM tape machine to record DC voltages as well as higher frequency signals. A current probe on the output wires of the fire set furnished a timing pulse to each recorder and a trigger pulse to the CORRTEX units. The fire set was located inside the trailer and connected to the detonator wire from the explosive borehole by means of a long twisted pair wires.

Sandia provided the stress gauges and reduced the signal records to standard stress units. The techniques and problems associated with the use of stress gauges and the interpretation of stress-gauge records have been discussed by Reed and Boade, 1979; and Reed, 1979, has presented an account of the operation of the gauge and calibration techniques. The gauges used were either single-element lithium niobate or ytterbium gauges.

As indicated above and in Table 2.1, three tests (two single-borehole tests and one four-borehole test) were extensively instrumented with stress gauges, velocity gauges, and CORRTEX. The layout for the placement of stress gauges in these tests generally follows that shown in Figure 2.3 (this layout was used on Test 79-16). The stress gauges were placed in separate instrument holes to record the stress wave loading and subsequent response of the rock. A rock-matching grout coupled the gauge package to the surrounding rock to minimize wave reflections from the borehole wall. Lithium niobate [Graham and Jacobson, 1973] is piezoelectric material and, hence, produces an electric charge proportional to the amplitude of the stress wave. Ytterbium, however, is a piezoresistive material that changes resistance from an applied stress [Grady and Ginsberg, 1977]. Both gauge types respond to low-stress amplitudes (less than 1 GPa) regimes expected away from the explosive borehole in a fragmentation experiment.

Tests 79-2 and 79-3 were single-borehole geometry in which the explosive hole and instrument holes were drilled perpendicular to the free surface. Test 79-2 was fired in the floor of the mine and Test 79-3 in the rib. Test 79-16 consisted of four boreholes drilled perpendicular to the free face and bedding planes in a square pattern providing a three-dimensional configuration. Instrument holes were drilled according to the pattern shown in Figure 2.3. The purpose of the experiment was to compare fragmentation from a multiple-borehole geometry to a single-borehole arrangement and to study the interaction of the stress waves from the four holes. In addition, the data from the three instrumented tests were used to calibrate and compare to computer simulation of the blast.

Figure 2.3 Layout of Colony Mine experiment, 79-16.

The velocity gauges were standard Model L-11D and Model L-22 moving coil geophones manufactured by Mack Products, U.S. Inc., Houston, TX. The purpose of these gauges was to determine the arrival times of the stress wave and the rock mass velocity. The output voltage was connected by means of long signal cables to signal conditioner and amplifier electronics and finally to the analog tape recorder.

The gauges were mounted on the surface of the oil shale and also were embedded in the rock at meter distances from the explosive borehole. For the surface mounting, the gauges were oriented to record vertical motion and firmly attached with grout. The embedded gauges were first positioned in a short length of PVC pipe with the signal leads exiting from the top and then filled with epoxy. This unit was then placed in the instrument borehole at a known slant distance from the explosive charge and coupled to the rock with a rock matching grout. The gauges were oriented in the instrument borehole to record the stress in the vertical direction.

Because of low sensitivity, insufficient frequency response, and a small maximum displacement range, very little useful data were obtained from these gauges from any of the three instrumented tests.

2. Rock Motion Measurements at the Anvil Points Mine.

Nearly every test was instrumented to obtain information on the motion of the oil shale rock during a blast. The measurements consisted mainly of accelerometers, velocity gauges, CORTEX sensor cables, video and framing camera coverage and, in addition, an assortment of gauges fielded by Sandia. In this report, the Sandia gauge results are not discussed. The Los Alamos motion gauges were mounted on the surface and embedded in the rock in addition to being placed in the stem columns to provide TOA data as well as motion data. Cameras provided a time history of the surface motion until dust and smoke obscured the scene. Table 2.3 is a list of the basic shot designs and the type of instrumentation used. Figure 1.5 shows the locations for the experiments. The following sections describe the instrumentation and recording methods used at the Anvil Points Mine.

a. Data Acquisition and Instrumentation. A state-of-the-technology data acquisition system was used to provide the timing and firing sequence to allow remote control of apparatus at the experiment site, to record and store gauge signals on digital transient wave recorders, and to accommodate mass storage and retrieval of the digital data for later analysis. The system consisted of a computer-controlled T/F system and a CAMAC (Computer Automated Measurement and Control) - based transient wave digitizer recording method. Peripheral electronics to provide an adequate voltage to the recorders included signal conditioners and power supplies. In addition,

Table 2.3.
Brief Description of the Fragmentation Tests Conducted at the Anvil Points Mine.

Test I.D.	Number Explosive Holes Charges		Explosive Type	Total Explosive Mass (kg)	Average Explosive Length (m) ^a	Average Depth of Burial (m) ^b	Average Stem Length (m)	Instrumentation
A-1	1	1	ANFO Pellets	29.0	1.80	2.52	1.62	A/C/T/TV
A-2	1	1	TNT Pellets	33.1	1.80	2.54	1.64	A/C/T/TV
A-3	1	1	TNT Pellets	29.4	1.65	2.56	1.74	A/V/C/P/T/TV
A-4	1	1	TNT Pellets	30.3	1.65	2.16	1.33	A/P/T
A-5	1	1	TNT Pellets	33.8	1.65	2.50	1.67	A/P/T
A-6	1	1	ANFO Pellets	26.5	1.65	2.41	1.58	A/V/P/T/TV
A-7	1	1	TNT Pellets	50.0	3.35	4.98	3.30	A/V/C/P/T
A-8	1	1	1185C ANFO Slurry	71.0	3.05	6.10	4.57	A/P/T
A-9	1	1	1205L ANFO Slurry	56.8	2.44	0.99	2.63	A/V/C/T
A-10	1	1	1115 ANFO Slurry	56.1	2.41	0.98	2.56	A/V/C/T
A-11	1	1	TNT Pellets	49.5	2.53	3.70	2.43	-
A-12	1	1	TNT Pellets	35.8	1.83	2.74	1.82	-
B-1	5	5	TNT Pellets	147.5	1.65	2.47	1.65	A/V/C/P/T/TV
B-2	5	5	TNT Pellets	150.0	1.68	2.70	1.86	A/V/C/P/T/TV
B-3	5	5	TNT Pellets	299.5	1.98	2.73	1.74	A/V/C/P/T
B-4	5	5	TNT Pellets	299.5	3.35	3.42	1.74	A/V/C/P/T
C-1	8	8	TNT Pellets	229.6	2.29	3.97	2.58	A/C/T/TV
C-2	16	16	TNT Pellets	913.6	2.81	8.53L1; 5.36L2	662L1; 3.42L2	A/C/P/F/TV/T
C-3a	1	2	TNT Pellets	102.7	3.05	4.42U; 5.94L	2.89U; 4.41L	A/C/P/T
C-3b	1	1	TNT Pellets	49.9	1.30	-	3.51	A/C/P/V/T
C-3c	1	1	TNT Pellets	59.9	3.35	-	3.04	A/C/P/T
C-3d	1	1	TNT Pellets	59.9	3.35	5.33	3.66	A/C/P/T
C-3e	1	2	TNT Pellets	119.8	3.13	4.75	3.41	A/C/P/T
C-3f	1	1	TNT Pellets	24.6	1.22	5.79	5.18	A/C/P/T
C-3g	1	1	1185 ANFO Slurry	47.1	1.83	5.49	4.57	A/C/P/T
C-3h	1	2	1115 ANFO Slurry	99.1	1.22U; 2.74L	1.52U; 4.72L	0.91U; 3.22L	A/C/P/T
C-3i	1	2	1205C ANFO Slurry	94.2	1.22U; 2.44L	2.74U; 3.85L	2.13U; 2.89L	A/C/T
C-4	3	3	TNT Pellets	90.0	1.65	6.70	1.65	A/V/C/P/T
C-5	16	24	1115 ANFO Slurry	1,797.6	3.08	-	-	A/C/P/S/T
C-6a	3	3	1205C ANFO Slurry	23.8	3.08	-	-	V/C/T
C-6b	3	3	1205C ANFO Slurry	175.8	2.63	-	-	A/V/C/T

Table 2.3.
(Continued)

Test I.D.	Number Explosive Holes	Charges	Explosive Type	Total Explosive Mass (kg)	Average Explosive Length (m) ^a	Average Depth of Burial (m) ^b	Average Stem Length (m)	Instrumentation
CF-1	1	1	1205C ANFO Slurry	66.7	2.59	4.04	2.73	V/C/T
CF-2	2	2	1205C ANFO Slurry	137.4	2.67	3.93	2.58	V/C/T
D-1	16	24	1205C ANFO Slurry	1972.4	2.40-5.20	-	2.79-5.20	A/C/T/P
D-2	16	32	1205C ANFO Slurry	3020.8	2.50-4.50	-	2.70-6.25	A/V/C/T/P
D-3	16	32	1205C ANFO Slurry	2896.6	2.74-4.57	-	1.68-5.64	A/V/C/T
DD-1	1	2	1205C ANFO Slurry	176.6	2.74U;4.12L	3.96U;7.24L	2.89U;5.02L	A/C/T
DD-2	1	2	1205C ANFO Slurry	180.5	2.74U;4.27L	3.35U;5.94L	2.89 ;3.80L	A/C/T
RP-1	6	6	1205C ANFO Slurry	454.8	2.13E;3.760	3.81	3.19E;1.980	C
S-1	1	1	TNT Pellets	57.1	2.74	14.42	13.04	A
S-2	1	1	TNT Pellets	57.1	2.74	14.42	13.03	A
S-3	1	1	1205C ANFO Slurry	72.9	2.83	6.48	5.06	A/C/T
S-4	1	1	1205C ANFO Slurry	71.3	2.77	6.58	5.19	A/C/T
S-5	1	1	1205C ANFO Slurry	54.6	2.65	4.10	2.77	V/C/T
S-6	1	1	1205C ANFO Slurry	83.9	3.26	5.68	4.04	A/V/T/TV
83-A	1	1	1175U ANFO Slurry	56.9	2.49	3.86	2.62	A/C/T/TV
83-B	1	1	1175U ANFO Slurry	58.1	2.50	3.83	2.58	A/C/T/TV
83-C	1	1	1175U ANFO Slurry	31.4	1.32	2.44	1.78	C/TV
83-D	1	1	1175U ANFO Slurry	63.2	2.54	4.01	None	C/TV
CV-1	1	1	1175U ANFO Slurry	25.1	1.03	1.69	1.17	C/TV
SB-1	1	1	1175U ANFO Slurry		2.35	3.67	2.50	
SB-2	1	1	1175U ANFO Slurry		2.35	3.67	None	

^a U after the number indicates upper charge and L indicates lower charge

^b L1 and L2 after number indicate level 1 and level 2 charges

^c A - accelerometers; V - velocity gauges; C - CORRTEX; T - TOA; P - pin probes;
F - framing camera; TV - video camera

Ampex FR-600 tape machines were available to record the signals in analog form. These recorders were not directly controlled by the computer, but were activated manually. Figure 2.4 is a block diagram of the data acquisition system showing the basic connecting paths to the various devices. The system is designed around a LSI-11/02 computer with a dual floppy disk unit (8 in. diskettes) for mass data storage, keyboard graphics terminal, and a graphics hardcopy unit. The computer drives 32 ports which, in turn, provide closures to peripheral equipment such as the fire set. Closures were also furnished to line drives power supplies for start and stop triggers to CORTEX, stop triggers to the LeCroy digitizers, start triggers for the Biomation recorders, and other devices. Figure 2.5 is a schematic of the T/F system illustrating the external equipment controlled by means of the computer actuated closures. A variety of signal lines and control cables linked the trailer complex with the experiment sites. Table 2.4 is a list of the cables used during the experiments. The coaxial signal cables were primarily used for high-frequency signals (greater than 1 MHz) and the twisted-shielded pairs were used to provide the excitation voltage and signal return for the ENDEVCO accelerometers. This wire can transmit signals of 1.0 MHz frequency. RG-214 coaxial cable is a high-quality, low-attenuation coaxial cable required to connect the CORTEX recording units to the sensor cables for applications for which fast pulse rates were necessary. Figure 1.5 shows the location of the instrumentation trailers and the route of the signal cables. All the cables ended at a junction-box (J-box) in the 100 Drift as shown in Figure 1.5. An appropriate number of long cable jumpers were run from the J-box to the experiment sites as needed. For Test A-1, the cables terminated at the J-box near the site. For Tests A-3, A-6, S-5, and S-6, the cables and the J-box pulled back were to those sites and then returned to the 100 Drift after conducting the experiments. At the location of the J-box in the 100 Drift, a raise was mined to allow access to the upper and lower levels of that position of the experimental mine and to provide improved ventilation. Cables were run up or down to the test site as needed. In addition to the two instrument trailers, an office and stock trailer was located at the trailer park. Later in the project, Sandia parked an instrument trailer and supply sheds near the Los Alamos trailers.

A diagram of the signal map of the 50-ohm system is shown in Figure 2.6. This system was used only for the first two tests, A-1 and A-2, and then abandoned in favor of the systems shown in Figure 2.7. This second signal recording arrangement was easy to use, versatile, and simple. The hardest part was lugging cables to the experiment site and connecting to the gauges. RG-223 coaxial cable was chosen because of the double shield to reduce cross-talk and 50-ohm impedance. The twisted-shielded wire pair with a ground wire was used with the ENDEVCO gauge/Ectron signal recording to reduce cross-talk, wire resistance, and ground loops.

Table 2.4.
Signal Cables and Control Lines Used at Anvil Points

<u>Lines</u>	<u>Number</u>	<u>Length</u>	<u>Application</u>
RG-223 (coaxial)	32	300 m	PCB accelerometers, velocity gauges, ionization pins, CORRTEX jumpers.
RG-58 (coaxial)	12	300 m	PCB accelerometers, velocity gauges, ionization pins.
RG-214 (coaxial)	6	300 m	CORRTEX jumper.
C-cable (coaxial)	6	300 m	Control lines for FS-10 fire set to fire the EBW detonators and REO blasting box.
Twisted/ Shielded Pair #22 Wire	24 pairs	300 m	Endevco accelerometers, strain gauges.
Twisted Pair	20 pairs	300 m	Control wires for video camera, flood lights, warning horn, mine phone.

DATA ACQUISITION SYSTEM AT ANVIL POINTS

EXPERIMENT TYPE

DIGITAL RECORDING TRAILER

FIG. 2.4 Data acquisition system.

TIME AND FIRING SYSTEM INPUT

TIME	EVENT
-5 m	VIDEO TAPE RECORDER (ON)
-5 s	SANDIA EQUIPMENT TRIG
-2 ms	CORRTEX SYSTEM (ON)
0	FIRING BOXES (DECK 1)
+5 ms	STOP TRIGGER DIGITIZERS
+20 ms	FIRING BOXES (DECK 2)
+20 ms	BIOMATIONS (ON)
2 m	VIDEO TAPE RECORDER(OFF)

Figure 2.5 T/F system and controls to external devices.

Figure 2.6 Low impedance recording system.

As stated previously, the accelerometer gauges for Tests A-1 and A-2 were connected according to the schematic diagram of Figure 2.6. The ENDEVCO gauges were connected to J-box I via shield/twisted pairs. Within J-box I, the appropriate pairs were connected to the excitation voltage supply and the line driver transmitter. The output signals from the line driver transmitters were then connected to the line driver receivers in the trailer via RG-223 signal lines. These signals were filtered and amplified before being recorded by the LeCroy transient digitizers or the analog tape units or both units. The signal paths for the PCB accelerometers were as shown in Figure 2.6 in which the gauges at the site were connected to matching resistance boxes and then to the RG-223 jumper lines and signal cables. The jumper cables were connected to a PCB power supply and signal conditioner with unity gain mounted in the digital trailer and, finally, the signals were connected to the input jacks of the LeCroy transient digitizers for recording.

After Tests A-1 and A-2, the PCB gauges continued to be connected as described above, but the electrical path for the ENDEVCO gauges were accomplished as shown in Figure 2.7. The excitation voltage and signal pairs from the gauge were connected to Ectron signal conditioners mounted in the recording trailer via twisted/shielded cables from the test site. A jumper cable connected the Ectron unit to the LeCroy transient digitizers, or the tape units, or both. As indicated in Figure 2.7, the ionization pin gauges and the velocity gauges were connected to the transient digitizers via jumper and signal cables. The signal from the velocity gauge was amplified when recorded by Biomation transient recorders.

The LeCroy multi-channel waveform digitizer was the primary device for recording and storing signals from the rock mass motion and ionization pin gauges. The Model 2264H digitizer (100 kilo-ohm impedance model) is an analog-to-digital converter packaged according to the international CAMAC standard which standardizes the method for transmission of data and control between instrumentation modules (LeCroy digitizer) and the controlling computer (LSI-11/02). The main features are eight input channels, 8-bit resolution, 100 kHz analog bandwidth, 500 kHz to 4 MHz (single-channel mode) sampling rate, and 10.24 volt range. A companion memory module, Model 8800/8, was used to provide 32K, 8-bit words total memory per unit. Two CAMAC crates with a crate controller each accommodated five LeCroy digitizers with memory modules for a total recording capacity of 80 channels. The system digitizes continuously, allowing retention of the waveform data in memory before the "stop digitizing" trigger from the computer is received. By activating a "read" program in the computer, the data in memory of the digitizers is transferred to a 8-in. diskette by means of a computer-controlled disk unit. This system was reliable, almost maintenance free, and simple to use. Table 2.5 summarizes the computer-controlled digital recording system.

Table 2.5.
Instrumentation Digital Recording

LSI 11/02 Computer (DEC)

- o Textronics CTR 4010-1
- o Floppy Disk Recording System
- o Integral Data Systems Hard Copy Unit

CAMAC Digitizers

- o CAMAC Crate with Controller
- o LeCroy Multichannel Waveform Digitizers (2264H)
 and Memory Module (8800/8)
 - 1, 2, 4, or 8 input channels
 - 0.25, 0.5, 2.5, 5.0, and 25 s/sample
 (4 MHz to 40 kHz frequency response)
 - 0 to 10.24 V or ± 5.12 V input voltage
 - 100 k Ω input impedance
 - 32.768 k memory words per channel maximum
 - 8 bits amplitude resolution
 - computer readable

Signal Conditioners

- o ECTRON (HPSCU 776A) - 14 Channels for Piezoresistive Gauges Forming Part
 or All of Wheatstone Bridge
 - 180 kHz frequency response
 - X1000 voltage gain in steps
 - Provides excitation voltage, balance, amplifier, and filter
 - Applicable to accelerometers, stress gauges, and time of arrival
 (TOA)
- o PCB Power Supply - 12 Channels for PCB Piezoelectric Gauges with
 Built-In Charge Amplifier
 - Unity gain
- o Los Alamos Design - 32 Channels for Piezoresistive Gauges with a
 Completed Wheatstone Bridge Circuit
 - 50 kHz frequency response
 - 50 Ω impedance system
 - Can drive long coaxial cable lengths
 - Provides excitation voltage, amplifier, and filters

Figure 2.7 Signal path for accelerometers, velocity gauges, and ionization pins.

In addition to the use of LeCroy waveform digitizers, Biomation signal recorders were used to provide eight channels for higher frequency signals. Each channel of these recorders was equipped with an amplifier section. The signal paths to the Biomations were as shown in Figure 2.7. The Biomations, Model 2805M (master) and 2805S (slave), were not interfaced with the LSI-11/02 computer as were the LeCroy digitizers except for the "stop digitizing" trigger generated by a T/F program. The main features of the Biomations are 2048 words per channel (8 bit words), 0.5 kHz - 5 Mhz sampling rates, and 50 mV to 50 V input voltage range. Biomations were used primarily to record signals from PCB accelerometers, velocity gauges, and ionization pins.

Conditioning of the signals from the gauges on Tests A-1 and A-2 was accomplished by a filter and variable amplifier section. The amplification of each channel was adjusted to provide a one volt signal based on the expected signal output from the gauge. In addition, each channel was calibrated by measuring the voltage after conditioning from a known input voltage at the gauge location. The excitation voltage for the piezoresistive accelerometers was from a power supply in J-box I. A constant current source for the piezoresistive accelerometers with a built-in amplifier was also located in J-box I, placed near the experiment site.

After Tests A-1 and A-2, Ectron signal conditioners were used with the piezoresistive transducers and a PCB power unit for the piezoelectric transducers. The Ectron instrument used was Model 776B HPSCU (high performance signal conditioning unit) that has the following main features: gain range of 1 to 1000 in steps, high impedance input, +10 volts maximum output, constant voltage gauge excitation of 5 to 30 volts, bridge completion and balancing circuitry, filters, and a 100 kHz bandwidth. The Ectron units were connected in the signal paths as shown in Figure 2.7, and the 14 units were mounted in two racks in the digital recording trailer. The PCB transducers used to monitor rock motion were powered by a 12-channel rack mounted unit, Model 483A. This unit is needed to power low-impedance piezoelectric transducers with built-in amplifiers over the signal lines and for coupling to recording instruments. The main features of the 483A power supply are: 12 channels, transducer excitation, unity gain, and ability to drive long cables. Figure 2.7 shows the connections to the gauge and signal recorders. Table 2.5 summarizes the three types of signal conditioning used with the gauges.

The transducers most used in monitoring the tests at Anvil Points were ENDEVCO piezoresistive accelerometers, PCB piezoelectric accelerometers, and electromagnetic velocity gauges. In this application, single-component accelerometer gauges designed for high-amplitude shock wave conditions were placed in instrument holes to record the G-load at the surface and within the oil shale. In addition, accelerometers were placed in the stem column on many tests to obtain data on stem

performance and behavior. The electromagnetic velocity gauges were used to measure rock motion in a similar manner as with the accelerometers. The PCB accelerometers were also used as time-of-arrival (TOA) gauges in the explosive column, the oil shale, and stem column.

Two types of accelerometer transducers were used. One was the ENDEVCO piezoresistive miniature shock accelerometer, Model 2264A (manufactured by ENDEVCO Corporation, San Juan Capistrano, CA) rated for +5,000, +20,000, and +50,000 G's. The second type was the PCB piezoelectric high-shock accelerometer Model 305A (manufactured by PCB Piezotronics, Inc., DePew, NY) rated for +50,000 G's. These gauges were chosen for their small size, low mass, sensitivity, and high-frequency response. The ENDEVCO 2264A incorporates piezoresistive elements in two arms of a Wheatstone bridge circuit and internal fixed resistance in the other two arms of the bridge. The bridge unbalance due to an applied G-load is the signal source. Occasionally, ENDEVCO 2262-2000 damped accelerometers were placed on a test. These gauges have larger mass and greater sensitivity than the 2264A model. Figures 2.6 and 2.7 show the four-arm full bridge arrangement of the gauge and connections to the signal conditioner and the 10.0 volt DC excitation supply.

The PCB 305A accelerometer employs a quartz transducer element with a built-in ICP (integrated circuit piezoelectric) that includes a miniature IC (integrated circuit) voltage amplifier and impedance converter. This results in a very simple two-wire system that has low noise, high-frequency response, good sensitivity, and eliminates the need for an external charge amplifier. The specific designation for the gauge used in the tests is PCB 305A02. Table 2.6 summarizes the main features of the accelerometers fielded on the fragmentation experiments. The combination of low output impedance of the PCB 305A gauge and the driving of long RG-223 signal lines, as was done in the tests, required that the cable impedance be matched closely. This was accomplished by connecting a variable resistance impedance matching module (PCB Piezotronics Model 073A) in the signal line as close to the gauge as possible. The effect was to prevent clipping of the output voltage.

The operation of an electromagnetic velocity gauge is a straightforward application of Faraday's law of induction. Figure 2.8 illustrates the technique [Fowles, 1972] as used in the laboratory. A wire or foil is embedded in the sample in a plane oriented parallel to the shock front. A steady magnetic field is applied parallel to the plane of the shock front and perpendicular to the wire. When the advancing shock front arrives at the wire, motion is induced, and motion of the wire through the magnetic field generates a voltage proportional to its velocity. Assuming the wire motion moves at the mass velocity, the induction voltage is proportional to the particle velocity of the shocked material. In principal, no calibration is required because the velocity is computed directly from the

Table 2.6.
Accelerometer and Velocity Gauge Specifications.

<u>Specifications</u>	<u>ENDEVCO</u>				<u>PCB</u>	<u>Velocity</u>
	<u>2264A-5K-R</u>	<u>2264A-20K-R</u>	<u>2264A-50K-R</u>	<u>2262-2000</u>	<u>305A0Z</u>	<u>Gauge</u>
Range	$\pm 5,000 \text{ g}$	$\pm 20,000 \text{ g}$	$\pm 50,000 \text{ g}$	$\pm 2,000 \text{ g}$	$\pm 50,000\text{g}$	200 mm
Nominal Sensitivity (mV/g)	0.250	0.025	0.010	0.25	0.100	0.047 ^a
Resonant Frequency (Hz)	50,000	100,000	180,000	12,000	60,000	unknown
Transverse Sensitivity (%)	5	5	5	3	5	unknown
Weight (grams)	1.5	1.5	1.5	28	4.5	100
Excitation (V)	10.00	10.00	10.00	10.00	24.0	none
Sensing Element	Piezite P-16	Piezite P-16	Piezite P-16	Piezite P-9	Quartz	Wire Coil
Output Impedence (Ω)	500	500	500	350	100	1

^a Units are V/m/s.

Figure 2.8 Electromagnetic velocity gauge technique.

measured voltage and known values for the magnetic field and the length of the active element.

For blast experiments in rock, the generation of an external magnetic field is impractical. This difficulty led to the development of an electromagnetic velocity gauge [Young et al., 1983] used for the Anvil Points Mine tests. In this design, the active gauge element is a helical coil wound on a lexan tube with a bar magnet placed inside the tube. The tube and magnet are placed in an instrument hole and grouted to the rock mass. In this configuration the coil and tube move with the rock; and the bar magnet, which is free to slide inside the tube, is the inertial mass and the magnetic field. Shock wave-induced motion of the gauge causes the coil to move through the flux lines of the bar magnet. This results in the generation of a voltage directly proportional to the velocity. Figure 2.9 is a diagram of the construction of the velocity gauge and Table 2.6 is a list of the gauge specifications.

Gauge calibration was accomplished by two methods, one of which allowed for verification of calibration in actual field tests. The bar magnets employed were standard Alnico V magnets, 0.95-cm diameter by 15.24-cm long. There were slight variations in the magnetic strength of each magnet and each gauge had to be calibrated. Since the gauge output would depend upon the specific coil/magnet interaction of each gauge, it was desirable to calibrate each gauge in terms of its velocity dependent output rather than to calibrate each magnet for field strength. For each coil/magnet combination, gauge calibration was effected by moving the magnet through the coil at a known velocity and then measuring the coil output voltage. The velocity of the moving magnet was determined by letting the magnet accelerate in free fall through a vertical tube aligned with the gauge tube. It was then possible to calculate the velocity of the magnet at the time it was passing through the coil. Measurements made of the time interval for the magnet to pass two reference points near the entrance to the coil served to verify the magnet velocity. The second and preferred calibration method involved suspending the magnet in the gauge as discussed below and then rapidly arresting the gauge after it had been accelerated (e.g., free fall dropped) to some velocity. The sudden arrest of the gauge caused the magnet to move through the coil much as it would were the coil to be accelerated by explosively generated shocks and rock motion. For this calibration method, the entire gauge output from the start of differential motion between the magnet and coil to the arresting of this differential motion when the magnet struck the plug in the gauge tube was digitally recorded. The numerical integration of this record would then provide an effective displacement for the magnet. The correct gauge factor would be that which would make the effective displacement equal to the displacement mechanically available to the gauge. This calibration factor would be an average factor for the entire travel allowed for the gauge. Acceptable gauge linearity was realized over a 10-cm interval, with a 2.5-cm minimum distance

Figure 2.9 Velocity gauge design used at Anvil Points Mine.

being allowed between the ends of the coil and magnet. This calibration method often allowed for the verification of gauge calibration in the field tests. Many of the fielded gauges traveled their full 10-cm calibrated distance before gauge failure. In these cases the gauge displacement, as integrated from the digitally recorded velocity output, could be verified against the pre-shot measured gauge displacement from the bar magnet hold position to the stop plug.

b. Gauge Installation. Accelerometer gauge packages, some with as many as three transducers, were fabricated by encapsulating the ENDEVCO and PCB transducers in a canister molded from an epoxy material that approximated the acoustic impedance of the oil shale. Within each canister, the gauges were oriented to monitor acceleration in one or two and sometimes three coordinate directions. The gauge packages, which were about 4-cm diameter by 7-cm long, were positioned in an instrument hole, oriented with respect to the charge column, and coupled to the shale with a rock matching grout. The location of each gauge was carefully determined relative to the collar of the explosive borehole. Moreover, care was taken to align each package so that the orientation of the gauges in the canister matched the acceleration direction to be measured--vertical or axial (z), radial (r), and tangential (t) directions. Deep rock motion was measured by orienting the gauges in the canister along the axial and radial components and, occasionally, a third gauge was included to monitor tangential acceleration. Surface motion of the shale was accomplished by a single transducer in a canister oriented along the vertical direction. Figure 2.10 is a schematic of a single-borehole test showing the placement of gauge packages, borehole dimensions, and satellite instrument holes for CORRTX. For multiple-borehole tests, such as Tests C-1, C-2, C-5, D-1, and D-3, several accelerometers were embedded in a single-instrument hole drilled outside the array of explosive boreholes and on the surface within the array. Figure 2.11 shows locations of accelerometers and velocity gauges for Test D-3 in plan view and in cross-section. Added details are available in the report by Dick and Craig, 1986.

The velocity gauges were emplaced within the rock to be blasted by drilling small-diameter (3.8 cm) holes into the rock with either a jackleg-type percussive drill or a small rotary jumbo drill. The gauges were made water tight by wrapping them completely with 1.5-cm-wide plastic electrical tape. It was possible to determine the position of the bar magnet within the sealed gauge package and to thus be certain that the gauge was functioning properly prior to gauge emplacement. The sealed gauge package would then be placed in the small diameter hole, positioned to the desired depth and then grouted in place using a thin slurry of Hydrostone or Duracal grout. As the Armorflex shielding over the coaxial cable exiting from each gauge was of only 1.0-cm diameter, it was possible to install more than one gauge in each hole, by emplacing the deeper gauge(s) first.

Figure 2.10 Test A - 3 instrumentation.

Figure 2.11a Test D-3 plan view of explosive boreholes and gauge location.

Figure 2.11b Test D-3 cross section showing instrument hole.

Figure 2.11c Test D-3 showing velocity gauge locations.

C. Instrumentation for Explosive Performance and Stem Behavior

1. **CORRTEX Technique.** CORRTEX was the primary technique for monitoring several blast parameters such as detonation velocity of the explosive column, detonator times, and the behavior of the stem column confining the explosive at both the Colony and Anvil Points Mines. The CORRTEX data were also used to deduce the occurrence of deadpressing and incomplete or low order detonation of the explosive. CORRTEX occasionally was used with limited success to monitor rock mass motion at a range of 1 - 3 m from the explosive charge.

The CORRTEX system is a passive method that employs time-domain reflectometry (TDR) to interrogate the two-way transit time of a 50-ohm coaxial cable. Included in the system are digital data acquisition electronics, a minicomputer, TDR circuitry, and data reduction programs. The microprocessor controlled unit periodically samples a 50-ohm sensor cable placed in the explosive column and stores this data as three digital words representing the two-way transit times and, in addition, stores one real-time clock-data word. Resolution of the two-way transit time is 125 ps, corresponding to a resolution of 30 mm from a total cable length of several hundred meters. Pulsed TDR consists of a series of pulses transmitted along the signal and sensor cables. At the start of each pulse, a clock is started in the unit. When the pulse encounters a change in impedance of the sensor cable due to crushing, the pulse reflects and returns to the unit. The return pulse is detected, clock-stop pulse is issued, and the microprocessor computes the elapsed time and stores this value in long-life memory. The sequence is repeated for each pulse and the data sorted. The latest CORRTEX equipment is capable of interrogating the sensor 4000 times with pulse rates of 5 - 90 ms. This provides 20 - 360 ms of recording time. The sampling rate depends upon the two-way transit time of the pulse of the combined length of sensor and signal cable. The entire system is portable and can be easily carried and deployed by one person. Figure 2.12 is a schematic diagram of the CORRTEX technique.

At the Anvil Points Mine, the CORRTEX units were triggered by a 5-volt trigger pulse generated by the computer-controlled T/F system. Generally, the units were started a few milliseconds prior to the firing of the detonators to provide a baseline and initiation time data. The CORRTEX units employed on the cratering tests at the Colony Mine were triggered from the voltage generated by a pickup coil around the firing cable to the detonator.

After an experiment, the two-way transit time data stored in computer memory and the known propagation speed in the sensor cable were used to compute from a resident computer code to calculate time-cable length data. From this, the detonation and shock velocities and initiation times were determined.

Figure 2.12 CORRETEX technique schematic.

The key element in this velocity measurement technique is the "crushing" (i.e., mechanical deformation, electrical shorting, or large change in electrical impedance) of a 50-ohm coaxial sensor cable as the detonation front or shock wave sweeps along the cable. Several different types of coaxial cables were used on the cratering tests at the Colony and Anvil Points Mines. The commonly used solid dielectric cables were RG-174, RG-214, and RG-223. A specially configured cable with a solid metallic sheath and a foam dielectric, designated FSJ1-50, and a small diameter coaxial cable with a foam dielectric, designated RG-174F, were also used extensively. These cables represent a range of crush strengths in which high crush strength cables were used to monitor the explosive and the low crush strength cables were used to measure the shock wave in the stem column. Table 2.7 is a list of 50-ohm coaxial cables used for sensor cables.

The CORRTEx units were located in the analog recording trailer and were self-contained, including an independent power supply. Two trigger voltages were supplied to the CORRTEx units by means of a trigger fan-out at specified times relative to the known shot time. This is illustrated in Figure 2.5.

a. Installation of Simple Experiments. In the single-borehole and simpler multiple-borehole experiments, the cables were all installed in the following fashion. The cable was attached to the booster usually with 1 m of length wrapped or folded over the booster to provide initiation times. The CORRTEx cable was then used to lower the booster detonator combination into the hole. The CORRTEx cable was centered in the hole and tension applied so that the cable was straight in the hole. The meter excess at the booster allowed a more precise determination of first point of data. This could be of particular importance in applications with a high media sound speed combined with explosives of slow burn speed. The installation of a typical simple experiment is illustrated in Figure 2.10.

b. Complex Blast Installations. For multiple-borehole shots, and shots with more than one charge in a borehole, or shots that combined both, the number of measurements sometimes exceeded the number of CORRTEx units available. In this case, a single long sensor cable was placed through more than one explosive with increasing delay times. An example of this type of installation is shown in Figure 2.13. In addition to the detonation velocity information, the detonation time of the explosive was also determined to the precision of the pulse period. Even in cases where the geometry made the detonation velocity measurement impossible, it was often still possible to obtain detonation timing.

In time-staged shots with multiple-timed blasts some protection was needed at the collars of the holes. This was particularly true when grout was used as the stemming. The impedance discontinuity at the grout interface caused the shock

Table 2.7.
50 Ω Coaxial Sensor Cables Used with CORRTEX.

<u>Cable Type</u>	<u>Dielectric</u>	<u>Diam. (mm)</u>	<u>Crush (GPa)</u>	<u>Application</u>
RG-174F	Foam	2.5	0.005	Stem, Grout
RG-174	Polyethylene	3.0	0.01	Explosive, Stem, Grout
FSJI-50	Foam	6.4	0.1	Explosive, Stem, Grout
RG-223	Polyethylene	6.4	0.3	Explosive, Stem, Grout, Rock
RG-214	Polyethylene	12.7	2.0	Rock, Crater
RF-214	Foam	12.7	2.0	Rock, Crater

Figure 2.13 CORRTEx installation of a test containing multiple-boreholes and time delays.

wave to reflect, and the associated motion to cut the cable. These phenomena were observed on several occasions. Protective schemes used to prevent this included Armorflex, high-pressure hose, pipe, and neoprene sheeting. Any of the schemes will work as long as the expected pressures are accurately estimated and allowed for.

2. Pin Probes. The pin-probe technique involves the shorting by ionization or mechanical crushing of a coaxial cable electrically charged by a pulse-forming network. Figure 2.14 shows the charging circuit and the pulse-forming network. The cable is an open circuit that is configured as a switch. When the end of the cable is crushed or ionized by the detonation front, a short-circuit occurs and current flows in the pulse-forming network. A pulse is then generated in the signal line which in turn is recorded by a fast oscilloscope or transient digital recorder. This pulse represents the arrival time of detonation at the cable end. By placing several cables at known positions along the explosive charge, the arrival times of the detonation front at each of the pins or cable ends are determined. The slope from the plot of time of arrival versus position of the cable end is the detonation velocity.

At Anvil Points, RG-174 coaxial cable was used typically as the pin-probe wire. Figure 2.15 shows a single pin-probe record containing six RG-174 ionization pin cables positioned along a TNT explosive column at 0.3-m intervals. Each fast-rising pulse is an electrical short that occurs at the time the detonation front arrived at the sensor location. In the test, each pin was provided with a negative electrical charge of 24 V. When the pin is shorted, the pin probe discharges through an R-C circuit with a time constant of about 100 ms. In the particular measurement of Figure 2.16, a detonation velocity of 4.78 km/s was computed from a measurement of the signal record and the known interval distance based on the record of Figure 2.15. A CORRTEX measurement in the same explosive column was 4.84 km/s for comparison.

3. Stem Motion Instrumentation. Motion and shock wave TOA data for the stem column were obtained from accelerometers, velocity gauges, and CORRTEX. Test C-3i, shown in Figure 2.17, illustrates a typical test configuration. PCB 305A02 accelerometers were preferred in the stem measurements because of their two-wire connection and easy mounting. Usually, gauges were screw mounted in phenolic disks that were positioned in the gravel stem or hydrostone grout plug at a known location relative to the explosive charge columns. The remainder of the stem material was then added to the borehole after the gauge was in position. The velocity gauges described previously were emplaced in a manner similar to the accelerometers, except they were not mounted on a phenolic disk before placement in the borehole. The gauges also were placed in the grout plugs comprising some of the stem designs.

Figure 2.14 Ionization pin pulse network.

Figure 2.15 Typical pin signals from explosive burn measurement.

Figure 2.16 Time-distance plot from detonation velocity measurement from Figure 2.15.

Figure 2.17 Shot diagram of a stem motion measurement.

CORRTEX records from the Anvil Points and Colony Mines were also used to obtain data on stem movement. These signals provided a continuous record of sensor cable crush length versus time for several tens of milliseconds during a blast. In this way, a measure of the decline of the shock wave velocity as a function of stem length was obtained for several stemming materials. A CORRTEX record illustrating wave propagating through the stem is shown in Figure 2.18. The record contains five parts: (1) a 2.5-ms baseline record, (2) explosive initiation at 0.0 ms, (3) TNT burn with a velocity of 4.95 km/s for approximately 0.7-ms duration, (4) a large change in slope of the curve at the explosive-stem interface, and (5) a monotonically decreasing crush length with time that showed an average sensor cable crush speed of 0.59 km/s. This is interpreted as the shock wave propagation speed in the gravel stem.

D. Data Presentation and Results of Colony and Anvil Points Tests

1. General Discussion. Many of the gauges did not produce readable peak amplitude signals, but some did provide decipherable TOA signals. The readable gauge signals including signal time relative to shot time, peak amplitude from accelerometers, peak amplitude from the velocity gauges, calculated peak mass velocity from an integration of the acceleration record one, and slant distance of the gauge from the explosive charge have been recorded in Table 2.8. The slant distance for each gauge, defined as the direct line distance the stress wave traveled between the source location and the gauge, was calculated. For the complex shots, the signal records for the accelerometer gauges contain much structure that represent the response from explosive charges with later detonation times and different slant distances. In some tests, gauges were located within the explosive array of the large tests making the analysis somewhat simple--at least for the first signals. Some of the gauges were located in the stem columns in an attempt to determine the behavior of the stem materials due to the action of the shock wave propagating through the column. Both confining stems and inter-charge stem columns were instrumented in some of the complex tests.

In Table 2.8, the gauge identification code is as follows. The first letter designates the organization that fielded the gauge: "l" indicates Los Alamos and "s" indicates SAI. The second letter symbolizes the location of the gauge: "s" means surface-mounted and "e" defines an embedded gauge. The third character is the gauge number in the set of gauges for a particular test number. Lastly, the /z, /r, and /t designation identifies the coordinate or direction of motion the gauge was intended to measure with respect to a given charge. A "z" indicates motion in the vertical or axial direction, "r" indicates motion in the radial direction, and "t" measures motion in the tangential direction perpendicular to both the axial and radial directions. All surface gauges were mounted to measure

Table 2.8. Accelerometer and Velocity Gauge Data

Single and Multiple Borehole Data Set in Shale

Test No.	Gauge ID	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
A2	ls4	0.706	2.78	30.2	3	9	shale
A2	ls5	1.067	4.2	8.3	1.1	9	shale
A2	ls6	1.386	5.2	4.6	2.2	9	shale
A2	ls7	1.669	6	2.4	1.6	9	shale
A2	le2/z	0.347	1.28	43	2.1	9	shale
A2	le4/z	0.771	2.91	10.5	1.19	9	shale
A2	le6/z	0.993	3.65	12.1	2.54	9	shale
A3	ls3	0.867	2.96	20	3.3	31	shale
A3	ls5	0.954	3.6	32	3.5	31	shale
A3	se1/z	0.492	1.6		2.5	31	shale
A3	se2/z	0.85	2.95		2.4	31	shale
A3	se3/z	1.25	3.66		2	31	shale
A5	ss1	0.421	1.65	28.8		12	shale
A5	ss2	0.471	1.88	27	12	12	shale
A5	ss3	0.712	2.66	15	7.8	12	shale
A5	ss4	0.684	2.64	25.8		12	shale
A6	ls1	1.347	4.26	11.5	8.67	32	shale
A6	ls3	1.894	6.28	0.75	1.85	32	shale
A6	ls4	1.941	6.28	5	10	32	shale
A6	ls5	2.55	8.04	0.4	3.4	32	shale
A6	le3/z	1.409	4.23	1.7	0.9	32	shale
A6	ss6	1.7	5.45	2	4	32	shale
A6	se1/z	1.258	2.79	30.8	9	32	shale
A6	se2/z	0.674	2.2		1.3	32	shale
B1	se1/z	1.225	5.39	2.5	8.5	16	shale
B1	se2/z	0.952	4.21	4	15	16	shale
B1	ss1	0.859	3.63	4.2	13.5	16	shale
B1	ss3	0.738	3.17	4	7.5	16	shale
B2	se1/z	0.7	2.92	10.6	4	16	shale
B2	se2/z	0.571	2.21	42	6	16	shale
B2	se3/z	0.91	3.57	10	3	16	shale
B2	se4/z	0.825	3.21		3.5	16	shale
B2	se5/z	0.621	2.21		10	16	shale
B2	se6/z	0.965	3.57		3.2	16	shale
B3	se1/z	0.738	2.94	24	5.5	10	shale
B3	se2/z	0.625	2.52	28	13	10	shale
B3	se3/z	0.518	2.03	38	10	10	shale
B3	se4/z	0.975	3.51	20	0.75	10	shale
B3	se5/z	0.875	3.26		3.4	10	shale
B3	se6/z	0.734	2.82		9	10	shale
B3	se7/z	0.558	2.03		7.5	10	shale
B3	se8/z	0.975	3.51		2	10	shale

Table 2.8. Accelerometer and Velocity Gauge Data (Continued)

Single and Multiple Borehole Data Set in Shale

Test No.	Gauge ID	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
B4	ls1	0.896	3.2	24	11	8	shale
B4	ls2	0.602	2.1	17.5	12.5	8	shale
B4	ls3	0.717	2.44	24	30	8	shale
B4	se1/z	0.663	2.56	35	18	8	shale
B4	se2/z	0.501	2.05	33	12	8	shale
B4	se3/z	0.397	1.58	30	10	8	shale
B4	se5/z	0.703	2.53		5.2	8	shale
B4	se5/z	0.53	2.04		5.5	8	shale
B4	se7/z	0.543	1.79		5.5	8	shale
B4	se8/z	0.769	2.65		16	8	shale
C1	le1/z	2.544	8.34	1.8	1.5	30	shale
C1	le2/r	2.544	8.34	1.5	1.6	30	shale
C1	le3/z	2.424	8.3	0.7	0.4	30	shale
C1	le4/r	2.387	8.3	3.4	5	30	shale
C1	le6/r	2.544	8.85	3.2	5	30	shale
C1	ss1	1.325	2.9	7.5	6	30	shale
C3A	ls1	2.513	9.49	31.1	14.5	30	shale
C30	ls3	1.427	3.78	7.5	4	12	shale
C30	ls4	1.264	3.95	7.5	5.5	12	shale
C5	ls2	5.52	11.95	0.17		33	shale
C5	ls2'	4.69	11.75	0.33		33	shale
C5	ls4	2.93	6.01	24.3		33	shale
CF1	se1/z	0.375	1.22		21.01	30	shale
CF1	se2/z	0.305	1.23		23.5	30	shale
CF2	se1/z	0.55	2.22		14	30	shale
CF2	se2/z	0.57	2.3		26	30	shale
CF2	se3/z	0.576	2.24		24	30	shale
S5	ss1	1.133	2.77		3.8	32	shale
S5	ss1'	7.745	2.77		1.9	32	shale
S6	ss1	0.274	2.61	0.99	0.49	32	shale

Stemming Gauges Data Set

Test No.	Gauge I.D.	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
C1	se1/z	0.475	0.91	15	20	30	stem/rock
C1	se2/z	4.784	2.04	55	37	30	stem
C1	se3/z	0.605	0.87	55	50	30	stem/rock
C1	se4/z	4.184	2.01	50	50	30	stem

Stemming Gauges Data Set (Continued)

Test No.	Gauge ID	Time (ms)	Dist. (m)	Accel. '0 (1000 g)	Velocity (m/s)	Grade (gpt)	Material
C3B	se1/z	6.159	2.43	50	45	12	stem
C3B	se2/z	16.322	3.5	17	42	12	stem
C3C	se1/z	14.875	3.14	5	50	12	stem
C3D	se1/z	0.205	0.15	39	1	12	stem/rock
C3D	se2/z	5.08	1.9	37	41.5	12	stem
C3E	se1/z	2.75	1.83	50.1	45.1	30	stem
C3E	se2/z	8.75	3.5	50.1	27.1	30	stem
C3E	se3/z	15.45	5.1	27.1	28.1	30	tnt
C3E	se4/z	22.45	6.59	7.5	29.1	30	boost
C3H	se1/z	7.9	2.44	41.1	42.1	26	stem
C3H	se2/z	15.5	3.66	2.5	3.1	26	stem
C3I	se1/z	1.7	1.52	10.1	9.01	33	stem
C3I	se1/z'	3.61	1.52	20.1	48.1	33	stem
C5	se1/z	4.6	2.39	75.1	45.1	33	stem
C5	se2/z	5.86	2.39	75.1	50.1	33	stem
C5	se3/z	4.8	2.39	75.1	30.1	33	stem
C5	se4/z	5.15	2.39	75.1	40.1	33	stem
C5	se5/z	5.4	2.39	75.1	50.1	33	stem
C5	se6/z	4.9	2.39	75.1	50.1	33	stem
C6A	ss1	1.001	3.29		7.5	30	rockshock
C6A	ss1'	5.1	3.29		7	30	stem
C6A	ss2	1.32	3.05		5	30	rockshock
C6B	ss2	1.177	2.9		6.5	30	rockshock
C6B	ss2'	6.438	2.9		44.5	30	stem
C6B	ss3	0.754	2.4		14.4	30	rockshock
C6B	ss3'	3.736	2.4		47.1	30	stem
CF1	ss1	0.846	2.74		12.5	30	rockshock
CF2	ss1	0.903	2.71		3.6	30	rockshock
CF2	ss1'	6.281	2.71		7.2	30	stem
CF2	ss2	0.851	2.46		3.6	30	rockshock
CF2	ss2'	6.456	2.46		5.6	30	stem
DP1	se1/z	0.685	1.52	10	10	30	rockshock
DP1	se1/z'	2.884	1.52	21	55	30	stem
DP2	se2/z	9.53	3.04	1.5	3.8	30	stem
S3	se1/z	6.102	2.38	13.5	35	30	stem
S4	se1/z	8.604	1.96	12	47.5	30	stem
S5	se1/z	0.723	2.16		10	32	rockshock
S5	se2/z	0.625	1.8		6	32	rockshock
S6	se1/z	0.509	1.36	55		32	rockshock
S6	se1/z'	3.985	1.36	14.4		32	stem
S6	se2/z	0.652	1.33		6	32	rockshock
S6	se2/z'	0.838	1.33		6	32	rockshock
83A	le1/z	1.09	0.79	108	24	9	stem
83A	le2/z	3.15	1.33	24	33	9	stem
83B	le3/z	7.02	1.91	5	18	9	stem
83C	le1/z	0.96	0.66	50	20	9	stem
83C	le2/z	2.031	1.17	50	90	9	stem
83C	le3/z	4.941	1.73	2.5	13	9	stem

Surface Gauges on Shale

Test No.	Gauge I.D.	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
A2	1s4	0.706	2.78	30.2	3	9	shale
A2	1s5	1.067	4.2	8.3	1.1	9	shale
A2	1s6	1.386	5.2	4.6	2.2	9	shale
A2	1s7	1.669	6	2.4	1.6	9	shale
A3	1s3	0.867	2.96	20	3.3	31	shale
A3	1s5	0.954	3.6	32	3.5	31	shale
A5	ss1	0.421	1.65	28.8		12	shale
A5	ss2	0.471	1.88	27	12	12	shale
A5	ss3	0.712	2.66	15	7.8	12	shale
A5	ss4	0.684	2.64	25.8		12	shale
A6	1s1	1.347	4.26	11.5	8.67	32	shale
A6	1s3	1.894	6.28	0.75	1.85	32	shale
A6	1s4	1.941	6.28	5	10	32	shale
A6	1s5	2.55	8.04	0.4	3.4	32	shale
A6	ss6	1.7	5.45	2	4	32	shale
B1	ss1	0.859	3.63	4.2	13.5	16	shale
B1	ss3	0.738	3.17	4	7.5	16	shale
B4	1s1	0.896	3.2	24	11	8	shale
B4	1s2	0.602	2.1	17.5	12.5	8	shale
B4	1s3	0.717	2.44	24	30	8	shale
C1	ss1	1.325	2.9	7.5	6	30	shale
C3A	1s1	2.513	9.49	31.1	14.5	30	shale
C3D	1s3	1.427	3.78	7.5	4	12	shale
C3D	1s4	1.264	3.95	7.5	5.5	12	shale
C5	1s2	5.52	11.95	0.17		33	shale
C5	1s2'	4.69	11.75	0.33		33	shale
C5	1s4	2.93	6.01	24.3		33	shale
S5	ss1	1.133	2.77		3.8	32	shale
S5	ss1'	7.745	2.77		1.9	32	shale
S6	ss1	0.274	2.61	0.99	0.49	32	shale

Embedded Gauges in Shale

Test No.	Gauge I.D.	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
A2	1e2/z	0.347	1.28	43	2.1	9	shale
A2	1e4/z	0.771	2.91	10.5	1.19	9	shale
A2	1e6/z	0.993	3.65	12.1	2.54	9	shale
A3	se1/z	0.492	1.6		2.5	31	shale
A3	se2/z	0.85	2.95		2.4	31	shale
A3	se3/z	1.25	3.66		2	31	shale
A6	1e3/z	1.409	4.23	1.7	0.9	32	shale
A6	se1/z	1.258	2.79	30.8	9	32	shale
A6	se2/z	0.674	2.2		1.3	32	shale

Embedded Gauges in Shale (Continued)

Test No.	Gauge I.D.	Time (ms)	Dist. (m)	Accel. (1000 g)	Velocity (m/s)	Grade (gpt)	Material
B1	se1/z	1.225	5.39	2.5	8.5	16	shale
B1	se2/z	0.952	4.21	4	15	16	shale
B2	se1/z	0.7	2.92	10.6	4	16	shale
B2	se2/z	0.571	2.21	42	6	16	shale
B2	se3/z	0.91	3.57	10	3	16	shale
B2	se4/z	0.825	3.21		3.5	16	shale
B2	se5/z	0.621	2.21		10	16	shale
B2	se6/z	0.965	3.57		3.2	16	shale
B3	se1/z	0.738	2.94	24	5.5	10	shale
B3	se2/z	0.625	2.52	28	13	10	shale
B3	se3/z	0.518	2.03	38	10	10	shale
B3	se4/z	0.975	3.51	20	0.75	10	shale
B3	se5/z	0.875	3.26		3.4	10	shale
B3	se6/z	0.734	2.82		9	10	shale
B3	se7/z	0.558	2.03		7.5	10	shale
B3	se8/z	0.975	3.51		2	10	shale
B4	se1/z	0.663	2.56	35	18	8	shale
B4	se2/z	0.501	2.05	33	12	8	shale
B4	se3/z	0.397	1.58	30	10	8	shale
B4	se5/z	0.703	2.53		5.2	8	shale
B4	se6/z	0.53	2.04		5.5	8	shale
B4	se7/z	0.543	1.79		5.5	8	shale
B4	se8/z	0.769	2.65		16	8	shale
C1	le1/z	2.544	8.34	1.8	1.5	30	shale
C1	le2/r	2.544	8.34	1.5	1.6	30	shale
C1	le3/z	2.424	8.3	0.7	0.4	30	shale
C1	le4/r	2.387	8.3	3.4	5	30	shale
C1	le6/r	2.544	8.85	3.2	5	30	shale
CF1	se1/z	0.375	1.22		21.01	30	shale
CF1	se2/z	0.305	1.23		23.5	30	shale
CF2	se1/z	0.55	2.22		14	30	shale
CF2	se2/z	0.57	2.3		26	30	shale
CF2	se3/z	0.576	2.24		24	30	shale

Figure 2.18 Cortex signal from explosive burn and stemming crushing.

only in the axial direction. Under the gauge entries, an apostrophe is used to designate a second signal peak that was measured and subsequently included in the table. An example is gauge designation ssl and ssl' of Test S-5.

There are an enormous number of signal records from the accelerometers and velocity gauges fielded on the Anvil Points Mine test series. As a consequence, considerable time was spent evaluating the signals, measuring peak amplitudes and signal arrival times, converting voltage amplitudes to physical units, computing the signal times relative to a common reference time, calculating the slant distance of the gauge to the source point on the explosive column, and tabulating the results. For the Colony Mine test series, only the stress gauge records are of value and these results are discussed in a later section of this report.

In addition to the stem motion data from the accelerometers and velocity gauges, there is also an enormous amount of information relevant to stem behavior in the CORRTEx signal records. With some exceptions, these data have not been analyzed in detail and only a few records are included in this report. The data, however, are available in a Los Alamos internal report that resides in Group EES-3; in addition, all the original CORRTEx signals are stored on diskettes in Group P-15.

2. Oil Shale Motion Results of Anvil Points Mine Tests.

Stress wave propagation speeds were obtainable from the slant distance-time data. The slant distances between the stress wave source location along the explosive charge and the gauge location were determined as shown in Figure 2.19. This figure shows a composite wave front at 350 and 580 ms after detonation of a 1.8-m-long charge of TNT initiated at the bottom for Test A-2. In drawing this figure, the value of 5100 m/s for the detonation velocity as determined from the CORRTEx signal records and a value of 3600 m/s for the P-wave speed in oil shale were used. The source for the spherical front, labeled "shock front," originated at the bottom of the charge at the instant of detonation. The spherical front propagating through the stem column and outward in the shale originated at the top of the charge. The conical portions of the wave front positions at the two times represents the tangent line connecting spherical waves originating from a continuum of locations along the charge column. Hence, the wave front is composed of an upper spherical portion tangent to a conical section which in turn is tangent to yet another spherical portion at the bottom. As time advances, this composite wave activates the gauges. At the free surface, the surface gauges respond to an upward movement of the shale free face. The slant distance, D_s , is the distance as measured along a line perpendicular to the wave front connecting the gauge and source point on the charge column. As indicated in Figure 2.19, the gauges on the surface nearest the borehole were subjected to a wave originating from the top of the explosive while the embedded gauges were subjected to a wave which originated at points along the length of the cylindrical charge.

Figure 2.19 Slant distance and gauge location diagram.

The time-of-arrival of the stress wave measured from the gauge signal records were relative to the time of initiation of the explosive. The gauge source transit time, T_s , is computed from the following relation,

$$T_s = TOA - D_e/D \quad (1)$$

where TOA = time-of-arrival read from the signal records, D_e = length of explosive from the point of initiation to the source point, and D = detonation velocity measured from CORRTX or pin probe records. The wave propagation speed in the shale, V_s , is then computed from

$$V_s = D_s/T_s \quad (2)$$

where D_s is the slant distance from the charge center to the gauge. In practice, the wave propagation speed was determined by an iteration process. The first iteration utilized the measured detonation of the explosive and an assumed P-wave speed of 3600 m/s. From these values and the known shot dimensions, the wave speed at each gauge location was calculated. The second iteration utilized the P-wave speed computed in the first iteration to compute a new P-wave speed. A third iteration was performed if necessary, but usually only two iterations were necessary for the wave speed to converge to a consistent value.

The zero time reference for the rock motion gauges is defined as the initiation time for the detonator or electric cap measured from the CORRTX or pin probe records. For EBW detonators, the zero reference was 0.0 ms. To obtain the signal time, the digitizer trace start time, D_t , for the gauge signals was calculated from known recorder settings in the following manner,

$$D_t = T_d - 32768 R/N - 1024 S_{pt} \quad (3)$$

where T_d = programmed digitizer stop-trigger time from the computer in milliseconds, R = signal sample rate (0.25, 0.50, 2.5, 5.0, or 25.0 ms sample), S_{pt} = post-trigger sample setting (1 through 8), and N = number of signal input channels used (1, 2, 4, or 8). The signal TOA value was then calculated from the formula

$$TOA = D_t + \Delta T \quad (4)$$

where D_t is defined by Equation 3 and ΔT = time difference between the signal start and trace start measured from the signal record. Most of the parameters in Equation 3 were input parameters for the T/F computer system code provided by the experimenter for each test.

The voltage signals from the accelerometer gauges were converted to physical units in terms of Earth gravity values (G)

using the gain of the amplifiers and the sensitivity of each gauge according to the formula,

$$A(G) = S V_{pk}/A_m \quad (5)$$

where S = gauge sensitivity in millivolts per G (mV/G), V_{pk} = peak signal amplitude in millivolts (mV), A_m = gain of the amplifier, and $A(G)$ = acceleration measured in terms of one Earth-G. This value of "A(G)" represents the acceleration imparted to the rock by the action of the stress wave and the high pressure explosive gas products acting on the fractured oil shale. A velocity, V , is calculated by integrating the accelerometer record in the following manner

$$V = 9.8 A(G) D_t \quad (6)$$

where A = acceleration value from Equation 5, D_t = time interval for the integration, and the constant in front of the integral converts "A(G)" from units of gravity to units of distance and time. Equation 6 can be integrated to obtain displacement. The velocity from Equation 6 is interpreted to be the mass velocity or particle velocity of the rock.

a. Typical Accelerometer and Velocity Gauge Records. Only a very few examples of the signal records obtained from the acceleration and velocity measurement of the shale and stemming can be conveniently shown. The signals from the ENDEVCO and PCB accelerometers were similar in character, and Figure 2.20 shows the signals converted to physical units from the two types of gauges mounted in the same canister. Gauge a2-1e6/z was oriented to measure motion in the z-direction and a2-1e7/r measured radial motion. The two frames below the time versus acceleration frames are the velocity and displacement after integration. The example test was A-2 and consisted of a single-borehole geometry and TNT explosive in lean shale. The time scale was referenced to the detonator time. This is a typical record showing a fast-rising pulse representing the compressional wave, followed by a release wave, then a second sharp compressional pulse arriving about 0.1 ms later, followed by a tension tail. Finally, high-frequency low-amplitude peaks appeared as a result of exciting the resonant frequency of the gauge and gauge package. In the succeeding analysis, only the first peak amplitudes were determined.

Figure 2.21 is the record from an accelerometer in Test C-2 which contained 16 boreholes and an equal number of TNT explosive charges. The charges were grouped to be initiated at 0.0, 10.0, 20.0, and 30.0 ms which resulted in four groups of signals. Amplitudes of each successive groups of peaks increased monotonically from 500 to 4,000 G's. Only the peak amplitude of the first pulse was determined. Time was referenced to the computer start trigger for the fire set.

More explosive charges produced even more complicated signal

Fig. 2.20. Accelerometer records from two gauges in Test A-2.

Fig. 2.21. Accelerometer record from Test C-2 which contained sixteen charges detonated at four distinct times.

records. This is illustrated in Figure 2.22 which is a reproduction of the signals from Test D-3. This shot had 16 boreholes, 32 charges, and seven separate delay times for explosive initiation. Figure 2.22 is the signal from an accelerometer oriented to measure tangential motion within the rock. The signal is complicated, but six and possibly seven energy bursts are observable from the record that generally coincides with the delay times. Time was referenced to the time of the trigger to the REO blasting box. These complicated records were not analyzed in very much detail.

Examples of velocity gauge signals are shown in Figures 2.23, 2.24, and 2.25. The solid lines are the velocity measurement and the dashed lines are displacement curves after integration of the signal. Figures 2.23 and 2.24 are records from velocity gauges located in the same instrument hole, in which one gauge was 2.44-m deep and the other was 1.22-m deep. The instrument hole was placed in the center of one set of four boreholes of the 16-borehole blast configuration in Test D-3. Both gauges survived the motion for approximately 30 ms. Figure 2.25 is a signal record converted to physical units of a velocity gauge in the stem column of Test CF-2. The early signal is from the shock wave traveling along the borehole wall to the gauge, and the second arrival is the shock wave propagating through the stem material. In this example, the gauge survived about 10.5 ms.

b. Gauge Results. The source distance versus source travel time, source distance versus acceleration, and source distance versus particle velocity were plotted for single-borehole and multiple-borehole tests. These plots are presented in Figures 2.26 through 2.34. Surface-mounted gauge data are presented in Figures 2.35 through 2.37, and embedded gauge data are shown in Figures 2.38 through 2.40. As observed in the plots, there is much scatter of the data.

The slant distance versus wave transit time data listed in Table 2.8 were fit by a linear least squares method to the following two equations:

$$D_s = p + q T_s + r T_{2s} \quad (7)$$

and

$$D_s = k_1 + k_2 T_s \quad (8)$$

where D_s = slant distance, T_s = wave transit time, and p , q , r , k_1 and k_2 are the coefficients of the fit. The coefficient k_2 in Equation 8 is the propagation speed of the wave in the shale. Table 2.9 is a listing of the coefficients of the fits to the two equations.

In general, the curves that fit the data are either linear or quadratic in T_s . Except for the curve matching the multiple-borehole test data, the shape of the curves are concave

Table 2.9.
Curve Fitting for the Time-Distance Data
from Oil Shale and Stem Measurements.

Oil Shale Instrumentation^a

Single Borehole	$S = 0.029 + 3.809t - 0.274t^2; R^2 = 0.9847$ $0 < S < 8.0 \text{ m}; 0 < t < 2.6 \text{ ms}$ $S = 0.025 + 3.727t; R^2 = 0.9765$ $0 < t < 1.0 \text{ ms}$
Multiple Borehole	$S = -0.004 + 3.672t + 0.027t^2; R^2 = 0.9365$ $0 < S < 6.0 \text{ m}; 0 < t < 1.4 \text{ ms}$ $S = -0.021 + 3.846t; R^2 = 0.9364$ $0 < t < 1.0 \text{ ms}$
All Data Curve I	$S = -0.003 + 3.787t - 0.004t^2; R^2 = 0.9811$ $0 < S < 10.0 \text{ m}; 0 < t < 3.0 \text{ ms}$
All Data Curve II	$S = -2.825 + 5.837t - 0.574t^2; R^2 = 0.9173$ $2 < S < 9.0 \text{ m}; 1.0 < t < 2.5 \text{ ms}$
Surface Gauges	$S = 0.067 + 3.882t - 0.308t^2; R^2 = 0.9921$ $0 < S < 12.0 \text{ m}; 0 < t < 6.0 \text{ ms}$ $S = 0.048 + 3.716t; R^2 = 0.9490$ $0 < t < 1.0 \text{ ms}$
Embedded Gauges	$S = 0.139 + 3.647t - 0.121t^2; R^2 = 0.9907$ $0 < S < 9.0 \text{ m}; 0 < t < 2.6 \text{ ms}$ $S = 0.019 + 3.753t; R^2 = 0.9907$ $0 < t < 1.0 \text{ ms}$
Stem I (early time)	$S = 0.288 + 2.931t; R^2 = 0.7657$ $0 < S < 3.5 \text{ m}; 0 < t < 1.0 \text{ m}$
Stem II ^b	$S = 1.006 + 0.783 \ln t; R^2 = 0.7665$ $0 < S < 5.0 \text{ m}; 0 < t < 24.0 \text{ ms}$

^a S = slant distance (m), t = wave transit time (ms); and R^2 = correlation coefficient.

^b Data delineated with brackets and gauges C-3e se 3/z and se 4/z were not included in the curve fitting.

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Fig. 2.22. Tangential accelerometer gauge record from Test D-3.

Fig. 2.23. A typical velocity gauge record of deep rock motion in Test D-3.

Fig. 2.24. Companion velocity gauge record to Fig. 2.23 of rock motion in Test D-3.

Fig. 2.25. Velocity gauge record of stem motion in Test CF-2.

Source Distance vs Wave Transit Time

All Single Borehole Data

Fig. 2.26. Slant distance versus travel time for gauges on single borehole tests. A multiple linear regression was fit to the data.

Slant Distance vs Acceleration

All Single Borehole Data

Fig. 2.27. Slant distance versus acceleration data for single borehole tests. Data were fit to an exponential and power curves.

Slant Distance vs Velocity

All Single Borehole Data

Fig. 2.28. Slant distance versus rock mass velocity data from integrated accelerometer signals and velocity gauge signals.

Slant Distance vs Wave Travel Time

All Multiple Borehole Data Thru B4

Fig. 2.29. Slant distance versus travel data for multiple borehole tests through Test B-4. A multiple linear regression was fit to the data.

Slant Distance vs Acceleration

All Multiple Borehole Data Thru B4

Fig. 2.30. Slant distance versus acceleration levels for multiple borehole tests through Test B-4. The data were fit to exponential and power curves.

Slant Distance vs Velocity

All Multiple Borehole Data Thru B4

Fig. 2.31. Slant distance versus rock mass motion data from integrated accelerometer records and velocity gauge signals.

Slant Distance vs Wave Travel Time

All Data

Fig. 2.32. Slant distance versus travel time data from all gauges (single and multiple borehole, surface and embedded, accelerometers and velocity gauges). A linear regression was fit to the data

Slant Distance vs Acceleration

All Data

Fig. 2.33. Slant distance versus acceleration from all gauges (single and multiple borehole, surface and embedded, accelerometer and velocity gauges). The data were fit exponential and power curves.

Slant Distance vs Velocity

All Data

Fig. 2.34. Slant distance versus rock mass velocity from all gauges (single and multiple borehole, surface and embedded, accelerometers and velocity gauges).

Slant Distance vs Wave Transit Time

All Surface Gauge Data

Fig. 2.35. Slant distance versus travel time data from all surface mounted gauges. A linear regression was fit to this data.

Slant Distance vs Acceleration

All Surface Gauge Data

Fig. 2.36 Slant distance versus acceleration data from all surface mounted gauges. The data were fit to exponential and power curves.

Slant Distance vs Velocity

All Surface Gauge Data

Fig. 2.37. Slant distance versus rock mass velocity data from surface mounted accelerometer and velocity gauges.

Slant Distance vs Wave Transit Time

All Embedded Gauge Data

Fig. 2.38. Slant distance versus travel time data for all gauges mounted within the oil shale. A multiple linear regression fit is shown.

Slant Distance vs Acceleration

All Embedded Gauge Data

Fig. 2.39. Slant distance versus acceleration data from all gauges mounted within the oil shale. Exponential and power curves are shown.

Slant Distance vs Velocity

All Embedded Gauge Data

Fig. 2.40. Slant distance versus rock mass velocity data from accelerometer and velocity gauges mounted within the oil shale.

downward suggesting a decrease in wave speed as the slant distance increases. Figures 2.26, 2.29, 2.32, 2.35, and 2.38 show the curves. Table 2.9 includes a correlation coefficient, R^2 , which is an indication of the quality of the fit. If R^2 equals 1, the fit to the data is exact; and if R^2 is zero, the points are randomly distributed. As indicated in Table 2.9, the R^2 values for most of the fits are good.

A second grouping of plots of the data is slant distance versus acceleration featured in Figures 2.27, 2.30, 2.33, 2.36, and 2.39. The data were fit by exponential and power curves. The curves are expressed by

$$A(G) = a \exp[-(b D_s)] \quad (9)$$

and

$$A(G) = a D_s^b \quad (10)$$

where $A(G)$ = the acceleration in units of gravity, D_s = the slant distance of the gauge, and a and b are the regression coefficients. Table 2.10 is a listing of the coefficients of the curve fitting as well as the coefficient R^2 . The data are scattered to a considerable extent, as indicated by the values for R^2 of 0.6 to 0.8 for the combinations shown in the figures.

The slant distance versus rock mass velocity data are shown in Figures 2.28, 2.31, 2.34, 2.37, and 2.40. These data include both the integrated values for velocity and direct velocity measurements from the velocity gauges. No attempt was made to fit the data to an analytical equation because the scatter was so large. The data set for the multiple-borehole gauge measurements indicates a nearly random distribution, while the other data sets are distributed such that the velocity decreases as distance from the explosive charges increases. At this writing, the reasons for the large scatter are not known.

All the rock mass motion data presented above were obtained from the Consortium program. No usable rock mass motion data were obtained from accelerometer and velocity gauges from the Colony Mine test series nor from the Los Alamos test series conducted at Anvil Points Mine during the Los Alamos/Sandia program in 1983.

3. Stem Motion Data and Results. Data relating to stem column behavior including motion were obtained from accelerometer and velocity gauges and from the CORRTEX signal records. In the Colony Mine tests, CORRTEX signals provided the only information on stem performance. However, both CORRTEX signals and gauges were used in the Consortium program and the Anvil Points Mine tests that were conducted in the Los Alamos/Sandia program. Several hundred CORRTEX signals were obtained from the fragmentation shots at both oil shale mines, but only a limited number are presented in this section.

Accelerometers and velocity gauges were used a little later

Table 2.10.
Curve Fitting of the Acceleration Versus Slant Distance Data
and Stress Versus Slant Distance Data.

Single Borehole ^a	$a = 117.1 \exp(-0.68 S);$ $a = 166.9 S^{-2.40};$	$R^2 = 0.8342$ $R^2 = 0.7574$
Multiple Borehole	$a = 161.9 \exp(-0.80 S);$ $a = 179.2 S^{-2.38};$	$R^2 = 0.6775$ $R^2 = 0.6358$
All Data	$a = 62.2 \exp(-0.49 S);$ $a = 172.0 S^{-2.36};$	$R^2 = 0.7996$ $R^2 = 0.6793$
Surface Gauges	$a = 57.7 \exp(-0.50 S);$ $a = 214.3 S^{-2.63};$	$R^2 = 0.8482$ $R^2 = 0.8379$
Embedded Gauges	$a = 64.0 \exp(-0.45 S);$ $a = 129.1 S^{-2.03};$	$R^2 = 0.7549$ $R^2 = 0.7973$
Observed I	$\sigma = 84.84 - 16.25 S;$	$R^2 = 0.6346$
Observed II	$\sigma = 216.4 - 47.00 S;$	$R^2 = 0.6346$
Calculated I	$\sigma = 76.50 - 12.22 S;$	$R^2 = 0.5974$

^a a = acceleration ($\times 1000g$), S = slant distance (m), R^2 = correlation coefficient, and σ = stress (MPa).

in the Consortium program to evaluate stem behavior and performance when it was realized this information was needed. The tests that did not include stem motion gauges were Tests A-1 through A-6, Tests B-1 through B-4, and Tests C-1, C-2, and C-4. However, these tests utilized CORRTEx from which wave propagation data for the stem was contained in the signal. The specifics about the stem instrumentation are available by Dick and Craig, 1986. Gauges were placed in the crushed gravel confining the explosive grout plugs within the stem column and at the collar of the borehole and in grout plugs and gravel columns in the intermediate stem between explosive columns in the same borehole.

The gauge signal records were processed similar to the records from gauges measuring rock motion described in the previous two sections. Figures 2.41, 2.42, and 2.43 are plots of all the data: source distance versus wave travel time, source distance versus acceleration, and source distance versus mass velocity; the data from which the plots were made are listed in Table 2.8. Two curves appear to describe the data. The early signals observed on some of the records correspond to a shock wave traveling along the borehole to the gauge location at a speed of 3000 ms, while the later signals represent the shock wave through the stem material. The two empirical expressions that fit the data are

$$D_s = 0.288 + 2.931 T_s \quad (R^2 = 0.7657) \quad (11)$$

and

$$D_s = 1.006 + 0.783 \ln(T_s) \quad (R^2 = 0.7665) \quad (12)$$

where D_s = distance, T_s = time, and R^2 is the correlation coefficient. In Figure 2.43, the data delineated by brackets and the gauge signals from C-3e se3/z and se4/z were not included in the logarithmic curve fit since they appeared not to fit the main body of points. The correlation coefficients indicate a poor fit to the data.

Of the four curves (linear, power, exponential, and logarithmic) used to fit the data, the logarithmic form provided the best fit for later time response and the linear curve fit the early time data. The slope of 2930 m/s for the linear curve probably represents the rock shock. This value is lower than the measured oil shale shock wave speed of 3760 m/s. The logarithmic curve approaches zero slope after the shock wave travels about 3.5 m, which corresponds very closely with the results from the stem performance of Test C-3d in which the stem was observed to have held completely. In this test, an accelerometer located at the top of the 3.6-m stem column indicated zero acceleration. A post-test inspection of the site revealed floor heave, but the gauge was still in place at the collar of the hole. In addition, two CORRTEx sensor cables running through the stem were crushed to within 0.6 m of the collar and undamaged from there to the collar. It appears that a 3.6-m length of stem provided complete bridging. The acceleration levels measured near the explosive charge frequently

Distance vs Wave Transit Time

All Stemming Data

Fig. 2.41. Plot of time-distance data from Table 9.

Distance vs Acceleration

All Stemming Data

Fig. 2.42. Distance versus accelerometer gauge data for all the stemming measurements.

Distance vs Velocity

All Stemming Data

Fig. 2.43. Distance versus mass velocity data from all the stemming measurements.

were in excess of 50,000 G's (see Figure 2.42), while stem lengths of 3.5 m or greater attenuated the shock strength to values less than 5,000 G's. (These numbers must be multiplied by 9.8 m/s^2 to convert to physical units.) There is considerable scatter in the data but the data include measurements of stems with different composition, different explosives, different charge weights, and different explosive initiation sites. Scatter of the data could probably be reduced by considering the details of the stem column.

Using the plot in Figure 2.41 and the curve fitting form Equation 12, the instantaneous wave speed was calculated in the stem at four distances beyond the end of the explosive column. These values are listed in Table 2.9. For stem lengths less than one meter, there was insufficient gauge data from which to determine the wave speed.

An inspection of CORRTEX data indicated that sensor cable crushing in the stemming occurred in a similar fashion as in the explosive and the sequence of events that produced a signal were: (1) at the explosive-stem interface, a dynamic pressure wave of a few tens of kilobars, depending on the explosive was transmitted into the stem material; (2) the wave speed slowed considerably after entering the stem column; (3) as the wave traveled through the stem column, the amplitude attenuated significantly; and (4) when the pressure wave decreased to a value at or below the threshold crush strength of the cable, or when the wave reached the collar of the borehole, the CORRTEX signal no longer indicated crushing.

Several CORRTEX signals were analyzed to provide propagation speed of the pressure wave in the stem for several different explosives (ANFO and TNT pellets and IRECO slurry 1205C). Figure 2.18 is an example of a CORRTEX record showing the crush of the cable in the stem. TNT and 1205C are considered more brisant explosives than ANFO pellets. In the tests using ANFO, propagation speed versus distance of the wave in the stem was about 2200 m/s at 0.1 m from the explosive-stem interface. At 1.0-m distance from the interface, the wave speed had decreased to about 100 m/s. For the tests using the higher brisant explosives, the wave in the stem initially propagated at about 2700 m/s at the 0.1 m position from the interface. The speed decreased to 100 to 150 m/s after a run of 1.0 m, similar to ANFO. The backward wave propagation speed (the wave traveling in a direction opposite to the detonation front in the stemming materials below the initiation point) began at about 1000 m/s, which is much lower than in the stem column above the charge. After 0.5-m run the wave speed had decreased to about 300 m/s, and at 1.0 m the wave was moving at about 120 m/s. This backward wave appeared to decay in a similar manner to that in the forward direction.

In many tests, a 0.15-m thickness of clay was placed between the top of the explosive and the column of -10 or -19-mm-sized

crushed gravel to contain the explosive gases more effectively. Preliminary analysis of the CORRTEx records from tests containing clay indicated that even this small thickness mitigated the shock wave noticeably. In quantitative terms, the propagation speed of the shock wave in the crushed gravel at the clay-gravel interface was approximately 1100 m/s and decreased to about 220 m/s after traveling 0.3 m. This was a significant reduction in the initial propagation speed compared to tests in which clay was not used. A grout plug was used in the center or at the top of the stem column in a few tests to evaluate an intermediate stem design. The grout plug material was a mixture of hydrostone and crushed stone aggregate. The dynamic response was monitored by a CORRTEx sensor cable placed through the stem column. A typical CORRTEx record with a grout plug as a stem ingredient is shown in Figure 2.44.

The CORRTEx records from a few tests incorporating the grout plug in a single-borehole geometry showed the normal explosive burn curve followed by a length-time curve typical of shock wave propagation in the gravel stem column. At the gravel-grout interface, the signal indicated constant length. This feature suggested that the shock wave did not propagate beyond the bottom of the plug. We concluded that the plug effectively mitigated the shock wave in a stem column.

In Test C-3d, the stem column composed of 0.15 m of clay and 3.5 m of -10-mm crushed gravel completely confined a 3.4-m column of TNT. CORRTEx sensor cables were used to monitor both the explosive and stem performance. The CORRTEx signal indicated normal explosive burn and an initial wave speed in the stem of 1000 m/s, which decayed to 150 m/s after running 1.9-m distance. Using the wave speed of 150 m/s measured from CORRTEx and the particle velocity of 40 m/s determined from an accelerometer gauge and assuming an initial density of 1.75 g/cm³ for the stem, the shock pressure was calculated to be approximately 11 MPa (1620 psi) at the center of the stem column. This illustrates the usage of the CORRTEx and gauge data to determine physical quantities to evaluate the stem performance.

4. Stress Gauge Data and Results from Colony Mine Tests.

Stress measurements were performed on three Colony Mine oil shale fragmentation tests to record the development of the stress loading and response of the rock during the event. Sandia provided the gauges and reduced the gauge records to standard stress units for the Colony Mine Tests 79-01, 79-03, and 79-16. The results from these experiments are discussed below. The outcome of the Colony Mine tests have been reported previously but, for completeness, the analysis is reproduced in part in this report (Adams, 1979; Boade and Reed, 1980).

Table 2.11 is a listing of the data from the three Colony Mine Tests 79-02, 79-03, and 79-16, and includes the axial and radial coordinates and the slant distance from the source for each stress gauge, the observed peak stress measurement, and the

Table 2.11. Colony Mine Stress Gauge Data (79-2, 79-3, and 79-16)

Test Num.	Gauge I.D.	Axial Distance (m)	Radial Distance (m)	Slant Distance (m)	Observed Stress (MPa)	Predicted Stress (MPa)	Gauge Type
79-02	1AL	7.090	1.244	2.030	17.3	57.9	LiNb
79-02	1BL	5.133	1.229	1.790	130.0	88.3	LiNb
79-02	1CL	2.625	1.210	1.760	130.0	65.7	LiNb
79-02	1DL	0.626	1.195	1.350	79.0	45.0	LiNb
79-02	2BY-1	5.131	1.873	2.730	45.3	52.1	Ytterbium
79-02	2BY-2	5.131	1.873	2.730	32.8	52.1	Ytterbium
79-02	2BY-3	5.131	1.873	2.730	45.8	52.1	Ytterbium
79-02	3AL	7.045	2.536	3.690	16.5	34.7	LiNb
79-02	3BL	5.065	2.540	3.700	48.0	32.8	LiNb
79-02	4BL	5.105	3.196	4.650	9.5	22.3	LiNb
79-02	5BL	5.004	3.096	4.510	23.2	23.4	LiNb
79-03	1AY	7.615	1.377	2.270	55.5	52.0	Ytterbium
79-03	1BY	5.647	1.597	2.230	105.2	66.2	Ytterbium
79-03	1CY	3.172	1.876	2.620	64.5	39.2	Ytterbium
79-03	1DY	1.191	2.099	2.410	19.6	24.9	Ytterbium
79-03	2AL	7.281	1.744	2.430	107.0	52.2	LiNb
79-03	2BL	5.310	1.162	1.620	48.0	63.0	LiNb
79-03	3AL	7.396	2.604	3.630	11.5	33.2	LiNb
79-03	3BL	5.425	2.676	3.730	21.8	30.4	LiNb
79-03	4BL	5.346	3.001	4.190	19.0	24.5	LiNb
79-03	5BL	5.655	3.052	4.260	12.5	24.6	LiNb
79-03	6FL	-0.477	2.701	2.730	92.0	9.6	LiNb
79-16	5	0.914	1.067	1.410	78.5	36.7	Ytterbium
79-16	6	0.914	1.067	1.410	59.6	36.7	Ytterbium
79-16	7	0.914	0.686	1.070	73.5	59.8	Ytterbium
79-16	8	0.914	0.478	0.750	185.0	89.8	Ytterbium
79-16	9	0.914	0.808	1.220	74.0	49.8	Ytterbium
79-16	10	0.914	1.067	1.410	51.8	36.7	Ytterbium

Fig. 2.44. CORRTEX record from a stem containing a grout plug.

corresponding calculated peak mean stress value. A typical stress history is shown in Figure 2.45 along with the calculated peak mean stress value for the ytterbium gauge at Location 5 in Test 79-16 (see Table 2.11 for dimensions). Each gauge record has similar structure--that is, the signal rose in response to a compressive stress wave. As the wave passes the gauge, the trailing refraction portion of the wave causes the signal to decline at about the same rate as the rise time. Reflections of the waves from the free surface and other discontinuities within the rock mass were manifest as a series of oscillating signals following the first signal.

A slant distance was determined for each gauge in a manner similar to that used for the accelerometers and velocity gauges described in Section D.2 of this chapter. To perform these calculations, the wave speed in the oil shale and the detonation velocity of the ANFO explosive was needed. The burn velocities were measured by CORRTEX sensor cables and the values were 3620 m/s for Test 79-02, 3940 m/s for Test 79-03, and 3200 m/s for Hole 4 of Test 79-16. A speed of 3760 m/s was used for wave speed in shale that was determined from the Anvil Points Mine tests (see Section D.2).

Figure 2.46 is a plot of peak stress versus slant distance from the three tests and includes both observed and calculated peak stress (Adams, 1979). There appears to be two sets of data. Most of the data belong to the group labeled "observed set 1" which are generally for stresses less than 90 MPa. A second group of data labeled "observed set 2" are for peak stresses greater than 90 MPa. The data groupings including the calculated stresses fit linear curves. As indicated in the plot, the calculations are in reasonable agreement with the observations.

5. Explosive and Cap Performance from the Probes.

a. Explosive Results. The so-called burn velocity of the explosive is a direct measure of the performance of the explosive--that is, if the measured velocity corresponds to the ideal value, then the explosive has performed normally. To compare fragmentation tests, the explosive must behave the same from shot-to-shot so that the performance is not an experimental parameter. Hence, the detonation velocity was measured for each column on each shot and compared to the accepted value. The CORRTEX technique was used for both the Colony and Anvil Points Mines fragmentation test series. In addition, pin probes were used in some of the Anvil Points Mine tests as a backup to CORRTEX, the primary diagnostic. CORTEX signal records also provided accurate data on initiation time of the explosive, a measurement of actual time delays in complex blast designs, and electric blasting cap performance. The CORRTEX and pin probe data, the gauge data, and visual observations provide the basis from which to evaluate the effectiveness of the fragmentation experiment.

There are hundreds of CORRTEX signal records from the tests

Fig. 2.45. Measured stress history from a ytterbium stress gauge in test 79-16.

Slant Distance vs Stress

Observed & Predicted 79-02 & 79-03

Fig. 2.46 Slant distance versus peak stress data from three Colony

conducted at Colony and Anvil Points Mines, but only a few are discussed. However, an attempt is made to summarize the most important results such as average detonation velocities of several explosives and electric blasting cap performance.

Figures 2.18 and 2.44 present the important features of a CORRTEx record for measuring the time of initiation and detonation velocity of the explosive. In addition, the record of Figure 2.44 illustrates a typical response from a sensor cable placed in a borehole containing two charges and two stemming columns.

Figures 2.15 and 2.16 are signals and results from a pin probe measurement of the detonation velocity in a column of ANFO prills. In this example, the detonation velocity of 3810 m/s from the pin measurements agreed well with the CORRTEx measurements in ANFO. The good agreement between CORRTEx and pin probe measurements of the detonation velocity for the various explosives was the general result obtained throughout the fragmentation tests series. In some cases, CORRTEx was not available so pin probes were substituted as the primary explosive performance monitor.

A total of five commercial blasting agents were used in the Colony and Anvil Points Mines fragmentation tests and detonation velocities were measured for most of them. Table 2.12 is a listing of the average burn velocity for each explosive. ANFO and TNT prills and IRECO 1205C have the best statistics of the five explosives. For ANFO and TNT prills, a 50% and 33% increase in diameter of the borehole, respectively, causes a 7% increase in detonation velocity. The standard deviation for a single CORRTEx measurement of detonation velocity is typically ± 0.06 km/s as determined from a linear least squares fit to the digital time-distance data.

Another use for the CORRTEx records was demonstrated in two multiple-borehole tests. Colony Mine Test 79-16 was a four-borehole shot containing 1.5 m of ANFO prills and 1.5 m of stemming. The CORRTEx signals indicated that the explosive in Borehole 2 did not detonate, while the explosive in the other three holes detonated properly. Post-test excavation of the crater confirmed this conclusion. Test B-4 at the Anvil Points Mine was a five-borehole geometry, boreholes at the corners of a square array and a borehole in the center. Each hole contained 2.5 m of TNT prills and 2.5 m of stemming. Figure 2.47 shows the geometry and sensor locations. Pin probes were used in addition to the FSJ1-50 and RG-174 coaxial sensor cables for CORRTEx. The time-distance plot of Figure 2.47 is a composite of the records from a sensor cable in each borehole, and also shown is a pin probe time and location. Proper detonation of the explosive in the corner holes was indicated, but the center-hole explosive did not detonate. Again, this was confirmed during the post-test excavation of the crater.

Fig. 2.47. Test B-4 results showing an undetonated explosive column(K1).

b. Caps Detonator Results. These results are discussed in Chapter 5.

c. Rock Motion Results from CORRTEX. On several tests at Anvil Points Mine, CORRTEX sensor cables were grouted in satellite holes one or two meters from the explosive holes to measure crack propagation. Figure 2.48 is a signal record from an experiment, and Figure 2.49 is a plot of sensor cable crush versus time data from two separate tests (Tests CF-2 and S-5). Two lines with slopes of 360 and 270 m/s fit the data reasonably well, but the significance of this result is not understood. This number may represent the rate at which the crater was formed during the blast event or it may indicate a propagation velocity (this value is about 10% of the wave propagation speed in oil shale) of cracks in the rock. However, these explanations are not at all satisfactory. This particular application of CORRTEX did not provide much insight into the cratering process or crack propagation speed during the event.

E. Interpretation Of Results

1. Stress Measurement at Colony Mine. To interpret the results from the stress gauge measurements, more must be known about the triaxial nature of the transient stress field and the directional sensitivity of the gauges. Directional sensitivity became a factor because each gauge was oriented with the planar surface normal to the radius vector from the axis of the explosive borehole. Uncertainty, in directional sensitivity caused difficulty in separating the components of the stress field (Boade and Reed, 1980). Generally, the largest principle stress component is normal to the stress wave propagation direction. Near the axis of the explosive column, the axial component of stress, S_{zz} , is larger than the other two components in terms of which component dominates the measurement. Farther from the axis the radial component of stress, S_{rr} , dominates the measurement. Figure 2.45 shows the data and the fit to the data and the calculated mean stress data corresponding to experiment and Figure 2.3 shows the design for Test 79-16. For stresses less than 90 MPa, the experimental and the calculated values generally agree. The computed stresses are usually less than the measured stresses, possibly because the observed stress was not the mean stress as was calculated. Again, the directional sensitivity and gauge orientation factors also contributed to the differences in observed and computed stress.

From a plot of the observed data the first inclination is to describe the data by two separate straight lines, one above 90 MPa and another below 90 MPa. There is no physical model that would explain this phenomenon. The explanation is probably connected to the directional sensitivity of the gauges and the orientation relative to the borehole axis. The amplitude of the stress wave decreases linearly at 16 MPa per meter from 1 - 5-m slant distance.

Fig. 2.48. CORRTEX record of a sensor cable grouted in a satellite Instrument hole located next to the explosive hole in Test CF-2. See Fig. 2.49.

Fig. 2.49. CORRTX cable crush data for sensors grouted in oil shale rock.

In conclusion, these data need much more analysis to be able to provide a consistent picture. In addition, close-in (less than one meter) measurements are needed to formulate early time behavior.

2. Accelerometer and Velocity Gauge Measurements at Anvil Points Mine. The accelerometer and velocity gauge data provided wave speed, rock mass motion, and attenuation information on blasting in oil shale. Source distance (or slant distance) versus wave transit time data were consistent for single-borehole, multiple-borehole embedded gauges, and surface gauges. Considering the variety of tests, instrumentation methods, and rock condition, the scatter in the points was remarkably small. Most of the time-distance data fit a quadratic curve with time the independent variable and the coefficient for the t^2 term negative. This indicated a slowing of the wave speed with increasing distance, as expected. The scatter in the data and the decrease in wave speed with distance is probably caused by the dispersive nature of oil shale due to the stratigraphy, natural joints, and shale grade. In many instances, the gauge signal rise time was observed to increase with distance which is also indicative of a dispersive material. The average wave speed in oil shale from all measurements was calculated to be 3760 m/s covering a wide range of shale grade, rock competence, and rock strength.

Acceleration versus slant distance plots show considerable scatter in the data. Generally, a power curve provided a fit to the data for slant distances close to the explosive while an exponential curve provided a fit to the data for more distant range values. This change from a power curve to an exponential curve occurred at approximately 2.5 m. However, either form may adequately describe the data, considering the scatter. At close range, the acceleration values were near 50,000 G's but declined rapidly to 1000 G's at distances of 6 - 10 m. Rock motion calculations were done for Test A-2 of the Anvil Points Mine program. Figure 2.50 (Adams et al., 1983) shows radial distance versus acceleration and velocity from surface gauge data and from calculations. The observed peak acceleration was larger than was calculated, but the peak velocity determined from an integration of the accelerometer signals indicates very good agreement with calculations. Overall, there is good agreement which suggests that the computational model is reasonably close and the gauges are correctly measuring the rock motion.

Velocity gauge data obtained from Test D-3 and others have provided a quantitative measure of deep rock mass motion (see Figures 2.23 and 2.24). These records indicate that rock mass velocities were surprisingly low at depths of two to three meters below the surface. The shape and time interval of the velocity curves suggests that sustained rock mass velocities are due to the heaving action of the explosive gases as the gases penetrate the fracture network created by the shock wave. In Figure 2.24, the peak rock mass velocity at the 1.22-m depth was approximately

Fig. 2.50. Comparison of observed and calculated acceleration and velocity response from Test A-2.

twice the rock velocity at 2.44-m depth shown in Figure 2.23. To create a free face of 0.1 m at a depth of two or three meters for the next lower level of charges would require about 50 ms.

Rock mass velocity measurements from surface gauges were slightly less than measurements from embedded gauges (see Figures 2.37 and 2.40). Many of the velocity values were the result of integrating the acceleration signal and, hence, the value for velocity depends on the rise and fall time of the signal, the persistence of the motion, noise, baseline shifting, and the effects of reflections from unknown discontinuities on the signal structure.

3. CORTEX Data Interpretation. CORTEX was very valuable in diagnosing the detonator operation, explosive performance, and stem behavior throughout the fragmentation program. The exploding bridgewire detonator (EBW) functioned as specified by Reynolds Industries, Inc., but the standard electric caps performed erratically. The blasting cap firing precision is satisfactory for normal blasting applications, but it was insufficient for the oil shale fragmentation blasting needed for the Consortium program. With CORTEX, some statistical data were accumulated from several manufacturers' caps in an ideal environment and in the harsh environment of the complex blast patterns used for the mini-retort series at Anvil Points Mine. These data are presented in Tables 2.12, 2.13, and 2.14. If the multiple charges in the complex designs had been initiated with a single cap, the variance would have been much larger. The firing times and corresponding variances resulted from two, three, or four caps connected together for initiation of the explosive.

Even after conducting the Consortium series of tests at Anvil Points, the precision with which the caps must operate was not decided. Wiring a shot with four caps per charge and a backup initiation system was very cumbersome and time-consuming. Hence, caps that are reliable, rugged (resistant to the high stresses), and initiate accurately (1.0 ms) to be developed for in situ retort fragmentation blasting.

Most of the explosive were monitored by CORTEX to determine the performance. Five blasting agents were monitored and the average detonation velocity for each one is listed in Table 2.12. As observed, detonation velocity for ANFO and TNT prills increased with an increase in borehole diameter. The table also shows higher burn velocities for IRECO 1115 and 1175U slurry than for IRECO 1205C slurry in a 152-mm diameter borehole. Table 2.12 lists the average measured values and the manufacturer's specifications. Some of the multiple-borehole tests were instrumented with CORTEX sensor cables to evaluate sympathetic detonation. None of these tests exhibited pre-detonation of the charge from the detonation of nearby charges. However, deadpressing or desensitization of the ANFO slurry explosives was observed in Test D-2 and possibly in D-1 and C-5. In Test D-2, post shot examination revealed undetonated 1205C explosive at the

Table 2.12.
Some Average Values for Explosives and Electric Caps.

<u>Explosive</u>	<u>CORRTEX D(km/s)</u>	<u>Expected D(km/s)</u>
ANFO Prills		
101 mm diameter borehole	3.60	3-4
151 mm diameter borehole	3.88	3-4
TNT Prills		
152 mm diameter borehole	4.93	5.0
203 mm diameter borehole	5.21	5.3
IRECO 1115 ANFO Slurry		
152 mm diameter borehole	5.70	6.0
IRECO 1175U ANFO Slurry		
152 mm diameter borehole	5.41	5.5
IRECO 1205C ANFO Slurry		
152 mm diameter borehole	4.78	5.0

<u>Caps</u>	<u>Nominal Delay (ms)</u>	<u>Average Delay (ms)</u>	<u>Standard Deviation (ms)</u>	<u>Caps Tested</u>
Standard DuPont	50	53.1	4.8	6
New Technology DuPont	75	65.0	4.0	28
Hercules	50	46.7	8.0	6
Atlas				
Before Test C-2	50	56.2	3.9	14
After Test C-2	50	62.2	6.8	15
Period 0	0	3.7	0.7	10
Period 3	75	70.4	1.6	11
Period 4	100	104.9	9.4	8

Table 2.13.
Comparative Data for the Actual Delay
Firing Times for Test C-2

Charge	Firing Times (ms)		Average & Variance Actual	Delay (ms)
	Design	Actual		
6	0	4.9	3.9 $\pm$ 1.0	0
10	0	2.9		
2	10	15.3	14.2 $\pm$ 5.6	10.3
4	10	20.4		
8	10	9.3		
12	10	13.2		
14	10	15.0		
16	10	12.0		
7	20	28.3	27.5 $\pm$ 0.7	23.7
11	20	26.8		
1	30	37.4	36.9 $\pm$ 7.1	33.0
3	30	34.2		
5	30	29.9		
9	30	41.6		
13	30	34.1		
15	30	44.1		

Table 2.14.
Comparative Data for the Actual Delay Firing Times for Test C-5

Charge	Firing Times (ms)		Average & Variance Actual	Delay (ms)
	Design	Actual		
6L	0	80.1	79.8 $\pm$ 0.1	0.4
10L	0	79.4		
2L	10	92.5	92.1 $\pm$ 0.8	12.7
4L	10	90.8		
8L	10	93.5		
12L	10	92.1		
14L	10	92.5		
16L	10	91.2		
7	20	99.4	101.0 $\pm$ 2.4	21.6
11	20	102.5		
1	30	112.9	110.7 $\pm$ 1.5	31.3
3	30	—		
5	30	109.8		
9	30	110.3		
13	30	109.5		
15	30	110.8		
6U	40	113.0	111.8 $\pm$ 1.6	32.4
10U	40	110.5		
2U	50	127.0	132.7 $\pm$ 20.2	53.3
4U	50	—		
8U	50	—		
12U	50	—		
14U	50	138.0		
16U	50	133.0		

U = Upper charge column

L = Lower charge column

third level hanging from several undamaged boreholes. In addition, there was poor and incomplete fragmentation. A microscopic inspection of the undetonated explosive indicated the glass beads, needed for sensitization of the 1205C, were probably crushed prior to the detonator firing. Therefore, it was concluded that the explosive was deadpressed by the action of the stress waves originating from previously detonated explosives in adjacent boreholes. To study this problem in a complicated blast geometry, Tests DP-1 and DP-2 were conducted in which two charges in a single borehole were detonated with 30 and 60-ms time delays, respectively. The results from the CORRTX measurements indicated both explosive columns detonated properly with no evidence of deadpressing in this test arrangement. Perhaps movement of the intermediate stem columns in Test D-2 induced sufficient pressure to cause deadpressing of the third level of explosives. In these applications, CORRTX instrumentation provided data for understanding the fragmentation results in complex blast designs.

Wave propagation was easily and accurately monitored in the various stem columns using CORRTX sensor cables. The measured wave speeds ranged from 2500 m/s near the explosive to 200 m/s three meters from the charge. Figure 2.18 shows the three basic measurements derived from a typical CORRTX signal record. They are the detonator time (0.0 ms in this case), detonation front position versus time, and shock front position versus time for the stem column.

The pin probe technique was very reliable and proved to be an excellent method to backup CORRTX for measuring the detonation velocity of the explosive. On the occasions when CORRTX and pin probes were used to monitor the explosive, the detonation velocity measured by both methods agreed within 5 %. In this research project, the use of ionization pin probe was limited to measuring the detonation velocity. The probes were not used to monitor the shock wave in the stem column. In principal, however, pins could be used for this purpose. The pin probe technique is simple, inexpensive, and accurate.

F. Summary

Instrumenting the fragmentation tests at the Colony and Anvil Points Mines provided much data on rock mass motion, detonator and explosive performance, and stem behavior in a wide range of experimental test configurations. In addition, a versatile and smoothly functioning computer-controlled time and firing system and a signal acquisition system was perfected during the Consortium Program and later fragmentation programs. This system greatly simplified controlling and activating peripheral equipment, providing detonator trigger at the desired time and collecting, storing, and retrieving gauge signals in digital form.

The rock mass measurements from the single-borehole and

uncomplicated multiple-borehole tests indicated that the particle velocity within the rock mass ranged from 5 to 20 m/s at a depth of three meters from the surface, as illustrated in Figures 2.23 and 2.24. This suggests that a vertical time delay of 10 or 20 ms is sufficient to produce an adequate free face for succeeding explosives. However, in the D-series tests the deep rock mass velocity ranged from 2 to 5 m/s. These measurements suggest the timing delays should be about 50 ms between successive explosive levels to provide an adequate free face.

The major conclusions from the rock mass response measurements are as follows:

1. The average wave propagation speed resulting from all the data analyzed was 3760 m/s.
2. The wave speed decreased as the source distance increased.
3. Acceleration versus distance plots showed that acceleration declined very steeply as the slant distance increased.
4. Acceleration versus slant distance data appeared to fit a power curve for distances less than 2.5 m and an exponential curve for distances greater than 2.5 m, subject to scatter of the data.
5. The large scatter of the velocity-slant distance data preclude curve fitting but, generally, velocity decreased with increased slant distance.
6. In Test A-2, the agreement between the measured peak surface velocity from integrated accelerometer signals and the calculated vertical velocity was very good (see Figure 2.57).
7. Peak stress versus slant distance plots indicated the data were fit by two straight lines, one for stresses greater than 90 MPa and another for stresses less than 90 MPa. The directional characteristics of the gauge and associated calibration difficulties need to be incorporated in this analysis.
8. The peak stress gradient was about 17.5 MPa/m.
9. Rock motion and stress measurements appeared to be insensitive to shale grade, explosive type, stemming composition, and geologic influences regardless of whether the gauges were mounted on the surface or embedded in the rock.

The fidelity of the rock motion gauges is believed to be good. This conclusion is based on two facts: (1) the acceleration measurements agreed with calculations (Adams et al., 1983), and (2) the rock velocity values determined from integration of the acceleration record generally agreed with direct measurements from the velocity gauges. The principal concern is that the accelerometer and velocity gauges do not have adequate bandwidth and sensitivity for measurements in close

proximity to the explosive charge. Therefore, the gauges should probably not be used closer than a two meter slant distance. The dynamic stress field created by a blast is complicated even in a single-borehole geometry. Hence, stress gauge must be carefully located and oriented to be certain of the stress component measured.

CORRTEX and pin probe measurements of explosive performance and stem behavior were invaluable for correlating the effectiveness of fragmentation among tests. These measurements eliminated the need to question explosive performance as a parameter in evaluating the resulting fragmentation. A direct comparison of the measured detonation velocity of an explosive by CORRTEX with the expected value provided almost instantaneous verification of performance. Detonation velocity measurements obtained by the pin-probe method agreed very well with the CORRTEX technique.

CHAPTER 3

A. Introduction

1. **Fragmentation Phenomena.** The successful utilization of the VMIS method of recovery of kerogen from oil shale requires very special control of the fragmentation process. The initial void which is located in the pre-mined cavities must be fairly uniformly distributed throughout the fragmented retort after the blasting is completed. In addition, the resulting fragment size distribution must also be carefully controlled. Extremes on the particle sizes must be minimized. That is, large fragment sizes and extremely small sizes must be eliminated. The large pieces will not be completely retorted and recovery efficiencies will decrease to unacceptably low values. The dust and extremely small sizes will plug the air flow through the retort causing uneven flame advance which will also decrease recovery efficiencies.

Figure 3.1 depicts one proposed design for the VMIS technique and the one of interest in the Anvil Points test program. In this scheme two rooms are mined by conventional techniques utilizing drill and blast--one at the bottom of the intended retort and one at the top. This shale is transported to the surface to be crushed and retorted. The void created by this mining is normally 20 to 25 percent of the total retort volume. The proposed dimensions for such a retort on a commercial basis would be 120-m deep with a cross sectional area that is 45 m by 45 m. For the test program at Anvil Points the mini-retorts (D-series tests) were 24-m deep including the pre-mined rooms with a cross section that was 9 m by 9 m.

After the voids are mined into the retort, the remaining shale must be fragmented and jumbled to completely fill the retort. Some programs have utilized the vertical crater retreat (VCR) blasting method to fragment the remaining shale. In the VCR technique, all the void is placed at the bottom of the retort and the shale is fragmented level by level, moving up from the bottom until the fragmentation is complete. For the full sized retort this would normally be accomplished with a large number of separate blasting operations with the loading and stemming of the next level to be shot being accomplished after the blasting operation is completed for the preceding level. The VMIS technique being investigated at Anvil Points called for the shale to be fragmented in levels but with all the loading of explosives and stemming to occur at one time and the levels to be removed in the proper sequence by use of delay timing between detonations. This places much more restraint on the blasting operation. In this situation, as stated earlier, the void is initially created both below and above the retort since only the retort volume is drilled and boreholes from the surface are not required. This drilling operation also normally requires a slightly larger void at the top of the retort to facilitate drill clearance in the top

Figure 3.1. Underground retort schematic for VMIS.

room.

In the case of the D-series mini-retort tests it was anticipated that four levels of explosive charges would be required to fragment the 18-m section of remaining shale. Three of these levels were to be thrown upward and one downward.

The process evaluated at Anvil Points therefore calls for very precise blasting techniques. The D-series tests called for sixteen boreholes to be drilled in the 18-meter shale interval between rooms. Each borehole was to contain two column charges with stemming above and below the two explosive columns and an intermediate stem between the two charges. The stem between the two column charges is especially important since it must provide protection for the charge to be detonated last. It must also insure that crushing and movement of the charge does not occur as well as insure that the proper timing for the latter charge to detonation is not compromised. This particular problem is addressed in the stemming and explosive engineering section and will not be discussed further in this section.

The type of blasting described is unique to VMIS retorting. It is similar in some respects to tunnel blasting in that the free face is perpendicular to the boreholes being used, but it is dissimilar to tunnel blasting from the requirement of fragmenting into a confined volume and the requirement for very close control of resulting particle size. As with tunnel blasting, the first step is the development of an initial crater that will result in free face oriented more or less parallel to the remaining boreholes. In tunnel blasting, this initial stage is called an "opening cut". The timing between holes in the same level and between holes on different levels is very critical, since the delay must be long enough to permit free face to form but short enough to take advantage of the synergetic effects between charges and insure that the stemming between charges is still effective.

As stated above, the first step in the blasting operation is formation of a crater. In this section, the effectiveness of cratering with single charges as investigated in the oil shale program at the Anvil Points and Colony Mines will be addressed. In addition, results from a number of multiple-borehole tests will be reviewed from the standpoint of determining synergetic effects from the overlapping of detonations with a delayed timing and from the standpoint of determining the proper borehole spacing to be utilized in the mini-retort testing series.

B. Crater Profiles.

For many of the tests conducted within the programs at the Anvil Points and Colony mines, it was standard procedure to dig the resulting muck out of the craters and to weigh and sieve the fragments to obtain a distribution of the particle sizes. The test sites were surveyed prior to the test, after the test prior

to mucking out the craters and once again after the craters were dug out. Some of the craters were mucked by hand - section by section - to determine the distribution of fragment sizes in relation to location - while others were mucked by means of a backhoe. Table 3.1 is a listing of tests that will be discussed in this section from the standpoint of crater profiles.

Of the twenty-nine tests listed in Table 3.1, nineteen are single hole tests consisting of sixteen tests conducted in the floor of rooms or drifts in the two mines and three are tests conducted in ribs of the drifts (Tests 78-3, 79-3, and 79-11). The nomenclature used is as follows: "78" and "79" signifies tests conducted in 1978 and 1979 by LANL in the Colony Mine. The

A-series were done in the Anvil Points Mine during the consortium project. Tests CV1, 83D and 83C were done in the Anvil Points Mine by LANL while Tests SB1 and SB2 were also done in the Anvil Points Mine but were conducted by SNL (Sandia). From the standpoint of crater profiles the data obtained for these single hole tests is quite good. These were mucked out completely (by hand or with a backhoe) and later surveyed to obtain crater profiles. The single-borehole tests ranged in size from charges of a little more than 5 kg of explosive (Tests 78-1, 78-2, and 78-3) to charge sizes of more than 90 kg (Test 79-3). The bulk of the data, however, is for charge sizes on the order of 30 kg. Four different explosive types are represented in the data -ANFO, TNT prills, and Ireco 1175 and Ireco 1115 (two water gel explosives). Some tests were unstemmed (Tests 83D and SB2), some used a grout stem (Tests 79-3 and 79-4), some used oil shale fines for stemming and others a gravel stem. The ratio of charge length to charge diameter ranged from 4 to nearly 40. The borehole diameters utilized were either 11 cm or 15 cm. The ratio of the length of stemming (or open hole) to the length of charge varied from 0.54 to 3.25. Although the actual kerogen content of the shale in which the tests were conducted ranged from 5 to 80 gal/ton, the average content through the test beds ranged from 9 to 43 gal/ton. Two types of constraints were imposed on the tests within the series. Some tests were constrained by placing broken oil shale on the free surface (CV1), while others were conducted in rooms which were restricted in size (Test A-5 and to some extent Test A-4). All of this, as well as geological factors, changed from test to test.

The factors affecting the various single-borehole tests were therefore quite numerous and an attempt will be made in a later section to address the effect of some of these parameters. This task is difficult, however, since in most cases more than one parameter was changed from shot to shot.

The last ten tests listed in Table 3.1 were multiple-borehole tests and, as pointed out in earlier sections, many contained more than one charge per borehole (that is, they were multiple-level tests). In the case of these multiple-borehole tests, the data available from the standpoint

Table 3.1.
Shot Description and Test Data.

Test	1/lc	Location	Stem	Description
78-1	2.05	Colony	Shale fines	Single hole - floor
78-2	2.98	Colony	Shale fines	Single hole - floor
78-3R	2.92	Colony	Shale fines	Single hole - rib
79-3R	0.54	Colony	Gravel fines	Single hole - rib
79-4	3.25	Colony	Gravel fines	Single hole - floor
79-8	2.33	Colony	Shale fines	Single hole - floor
79-10	0.91	Colony	Shale fines	Single hole - floor
79-11R	1.26	Colony	Shale fines	Single hole - rib
A-1	0.93	Anvil Points	Gravel	Single hole - floor
A-2	0.92	Anvil Points	Gravel	Single hole - floor
A-3	1.05	Anvil Points	Gravel	Single hole - floor
A-4	0.85	Anvil Points	Gravel	Single hole - floor
A-5	0.98	Anvil Points	Gravel	Single hole - floor
A-6	0.96	Anvil Points	Gravel	Single hole - floor
CV1RP	1.14	Anvil Points	Gravel	Single hole - floor
83C	1.37	Anvil Points	Gravel	Single hole - floor
83DUS	1.08	Anvil Points	Gravel	Single hole - floor
SB1	1.00	Anvil Points	Gravel	Single hole - floor
SB2US	1.00	Anvil Points	Gravel	Single hole - floor
B-1		Anvil Points	Gravel	Five hole - floor
B-2		Anvil Points	Gravel	Five hole - floor
B-3		Anvil Points	Gravel	Five hole - floor
B-4		Anvil Points	Gravel	Five hole - floor
C-1		Anvil Points	Gravel	16 hole - back
C-2		Anvil Points	Gravel	16 hole - two level
C-5		Anvil Points	Gravel	16 hole - three level
79-12		Colony	Shale fines	Four hole - floor
79-15		Colony	Shale fines	Four hole - rib
79-16X		Colony	Shale fines	Four hole - floor

US - not stemmed

X - one hole failed to detonate

RP - rubble pack

R - rib

of crater profiles is at best sketchy. In the B-series tests, fairly accurate profiles were obtained for part of the resulting craters but only partial excavations were obtained as a result of the very large resulting craters. In some cases where the crater depth was large it was not feasible to excavate to the bottom of the resulting craters. In the case of the C-series tests which were conducted in the back of the mine, it was impossible to get into the rooms with surveying instruments so only crude approximations of the craters could be made. This was also the situation with the D-series mini-retort tests. Hence in general, the data available from the standpoint of crater profiles for the multi-borehole multi-level tests is not extremely accurate. This will be evident from the crater profiles to be displayed and discussed in the following section.

C. Single Hole Tests.

Table 3.2 gives a listing of the single-borehole tests for which enough data was generated to discuss crater profiles and cratering efficiency. Pertinent information is presented on emplacement such as borehole length and diameter, charge type, charge weight and size, as well as location of charge relative to the free face. In addition, average shale grade and average dimensions of the resulting crater are also presented for each test. Average shale grade represents an average of the shale grades as determined from shale densities from the free face to the bottom of the borehole. The crater dimensions represent averages as determined from survey data taken after the test sites were mucked. The volume of rock removed is also given and represents either a numerical integration performed on the survey data or an estimation of the volume of muck removed considering void content of the resulting muck. Both of these methods then used the density of the shale to estimate the weight of the shale in place removed from the crater. In many cases, the resulting muck was weighed and sieved for size distribution and, when available, these two values of shale removed were utilized to arrive at the volume of rock removed reported in Table 3.2.

Figure 3.2 is a collection of the crater profiles as drawn from the survey data taken after the test sites were mucked out. In general, the profiles were drawn for two perpendicular cross sections through the resulting crater. Contour maps were not available for any of the tests conducted at Colony and only a few representative samples are presented for the tests conducted at Anvil Points. These profiles and contour maps were used to assess to some extent abnormalities in the test results. It is evident, for example, by viewing the resulting profiles for Tests 78-2 and 78-3 that the geology in the shot vicinity had an effect on the test. In Test 78-2, note that an apparently weak joint plane or planes resulted in a very asymmetrical crater profile. In the "3-4" profile, no breakage occurred past the joint very close to the borehole in the "4" direction. This was also true to a lesser extent in the "1" direction. The same phenomena occurred in Test 78-3 which was a test conducted in a rib at

Table 3.2.
Borehole, Explosive, and Shale Grade Data
for the Single-Borehole Fragmentation Tests.

Test	Hole diam (m)	Charge length (m)	Charge weight (kg)	Charge type	Shale grade (gpt)	Stem length (m)
78-1	0.11	0.65	5.22	A	30	1.33
78-2	0.11	0.64	5.22	A	30	1.91
78-3R	0.11	0.66	5.22	A	30	1.93
79-3R	0.15	5.81	90.72	A	43	3.18
79-4	0.15	0.61	9.66	A	30	1.98
79-8	0.15	0.99	14.51	A	30	2.31
79-10	0.15	1.73	24.68	A	30	1.57
79.11R	0.16	1.54	14.51	A	30	1.94
A-1	0.16	1.80	29.0	A	31	1.68
A-2	0.16	1.80	33.1	T	9	1.65
A-3	0.16	1.65	29.3	T	31	1.74
A-4	0.16	1.65	30.3	T	13	1.40
A-5	0.16	1.65	33.8	T	12	1.62
A-6	0.16	1.65	26.4	A	32	1.58
CV1	0.16	1.03	25.1	I	31	1.17
83C	0.16	1.31	31.3	I	9	1.79
83DUS	0.16	2.54	63.2	I	13	2.74
SB1	0.16	2.50	60.0	I	20	2.50
SB2US	0.16	2.50	60.0	I	20	2.50

R - rib

US - unstemmed

A - ANFO

TNT - TNT prills

I - IRECO 1175 U

Table 3.2 (cont.).

Test	Volume of rock (m ³)	Depth of pull-p (m)	Radius-R R/p	
78-1	5.57	1.46	2.03	1.4
78-2	6.82	2.39	1.83	0.8
78-3R	9.73	2.13	2.58	1.2
79-3R	118.1	4.62	5.41	1.2
79-4	21.4	1.98	2.78	1.4
79-8	10.0	1.22	2.27	1.9
79-10	34.2	2.35	4.64	2.0
79-11R	43.3	2.7	3.68	1.4
A-1	31.1	1.91	3.80	2.0
A-2	54.0	3.62	4.39	1.2
A-3	49.0	3.33	3.69	1.1
A-4	27.4	2.94	2.98	1.0
A-5	10.0	2.90	1.95	0.7
A-6	47.0	3.40	4.38	1.3
CV1	33.0	2.34	5.26	2.2
83C	52.0	3.30	3.25	1.0
83DUS	83.5	4.18	4.38	1.0
SB1	43.4	1.82	3.89	2.1
SB2US	40.8	1.82	3.72	2.0
Average				1.4

R - rib

US - not stemmed

(78.01)

(78.03)

(79.03)

Figure 3.2a. Crater profiles for single borehole Tests 78-1, 78-2, 78-3, and 79-3.

Figure 3.2b. Crater profiles for single borehole Tests 79-4, 79-8, 79-10, and 79-11.

[illegible]

Contour Map After Excavation for Test A-2

Figure 3.2c. Crater profiles for single borehole Tests A-1 and A-2.

North-South Section Through Bore Hole for Test A-2

East-West Section Through Bore Hole for Test A-2

Contour Map After Excavation for Test A-3

HORIZONTAL
VERTICAL
0
1
METERS

----- PRESHOT PROFILES
————— POSTSHOT PROFILES

* DEBRIS BEYOND END POINTS ON PROFILES

North-South and East-West Sections Through Bore Hole for Test A-3

Figure 3.2d. Crater profiles for single borehole Tests A-2 and A-3.

North-South Section Through Bore hole for Test A-4

SCALE: 1" = 4'

East-West Section Through Bore hole for Test A-4

Contour Map After Excavation for Test A-5

Figure 3.2e. Crater profiles for single borehole Tests A-4 and A-5.

Figure 3.2f. Crater profiles for single borehole Tests A-5 and A-6.

North-South Section Through Bore hole for Test A-6

East-West Section Through Bore Hole for Test A-6

Profiles of the craters for tests 83-C and 83-D.

Profiles of the crater for test CV-1.

Figure 3.2g. Crater profiles for single borehole Tests A-6, 83C, 83D, and CV1.

Figure 3.2h. Crater profiles for single borehole Tests SB1 and SB2.

Colony. In this test, a bedding plane in the "3" direction greatly altered the symmetry of the resulting crater. The presence of this apparently weak bedding plane caused asymmetric breakage in the vertical direction. It is assumed that these weak planes (perhaps even open joints) prevented stress wave preconditioning of the rock past the joints and redirected most of the energy from the explosive in the other directions. The close proximity of these relatively open joints would immediately transform compression waves into tensile waves traveling in the opposite direction. These very high magnitude tension waves could be very effective when following closely behind the outgoing compressive waves in the direction away from the open joint and the borehole.

The same behavior, but to a much lesser extent, is evident in the profiles from Test A-3. In this case, the contour of the resulting crater is also presented. Note the crater is wider in the NE/SW direction than in the NW/SE direction. In this case, however, it is felt that the joint in the NW direction was far enough removed from the borehole so as to not have had too large an effect on the test results. As will be discussed later, the presence of such planes actually appears to have resulted in larger volumes of shale being broken in the test than if the open joints had not existed. Another feature of interest pointed out by viewing the information presented in Figure 3.2 is seen for Test A-5. In this case, the test was conducted in a small room mined into the side of the drift. The purpose of the test was to observe the effect that confining ribs might have on the resulting crater. By observing the information in Figure 3.2, it is evident that restrictions were imposed by the close ribs and that a very asymmetrical crater resulted with no evident breakage occurring back into the ribs. The crater was much elongated in the direction toward the open drift. (See the contour map presented.) In addition, it appears to have pulled quite deeply compared to the extent of breakage in the radial direction.

Test CV1 was intended to determine what effect would be observed if a rubble mass was placed above the free face to hold the shale broken by the explosive in place. At the test site, spacers were placed on the free face and then covered with plywood sheets and broken oil shale to provide confinement to hold the broken rock in place but yet permit a free surface for wave reflections to occur. The height of the muck pile above the free surface was nearly equal to the intended depth of the crater. The rubble mound was not entirely successful in holding the broken shale in place, but it did provide a great degree of confinement. This is evident from viewing the resulting crater. In this case, the radius of the crater extent was quite large in comparison to the depth-of-pull. In other words, the "cone angle" of the resulting crater was larger than for tests with no confinement. Specific results for these abnormal tests will be discussed in greater detail in a later section.

D. Multiple-Borehole Tests.

Figure 3.3 presents crater profiles for the multiple-borehole tests listed in Table 3.1, as well as for the three D-series tests which were mini-retort shots. Unfortunately the data for the multiple borehole testing is not as good as that obtained in the single hole series. The reasons are as follows: First, much more muck removal was required for the multi-hole tests which made the complete mucking more difficult. Second, many of these tests created craters of nearly the same dimensions as the mined rooms in which they were conducted making it impossible to get the backhoe in position to remove all of the muck. Thirdly, for those tests conducted in locations where maneuvering room was not a problem the depth of pull tended to be greater than for single borehole testing and the backhoe arm was not long enough to reach completely to the bottom of the crater. Finally, many of the larger tests were conducted in the back (or roof) of the mine. Although this made for easier muck removal by means of an LHD vehicle, the LHD could not, for safety reasons, go under the shot site for fear of loosened shale falling on the machine and/or operator.

Therefore, the profiles of craters given in Figure 3.3 are in some cases crude sketches made from a few rough measurements and from visual approximations of resulting shapes. This is especially true for the C- and D-series tests.

The profiles presented in Figure 3.3 for Tests 79-12, 79-15 and 79-16 are quite accurate and show that the charges pulled nearly to the full length. The profiles for the B-series tests are somewhat less accurate due to limited maneuvering space for the backhoe and/or limited reach on the backhoe arm. In the B-series, none of the tests pulled to the borehole bottoms, especially between the boreholes. The C-series tests were conducted in the back and, unlike the B-series, appeared to have pulled for the most part to the full extent of the charges. The D-series tests were the mini-retort tests and were intended to completely rubblize the interval between the upper and lower mined rooms. As can be observed from the crater profiles presented in Figure 3.3 for this series, an unbroken shale ledge was left in all three tests. This was felt to be due to: a) poor intermediate stem performance resulting in timing errors, poor explosive performance and physical displacement of the explosive charges; and (b) inadequacies of the test design mainly for the requirement of a plane reversal. That is, the requirement for some rock to be moved upward and some to be moved downward.

A summary of the pertinent loading details for the multiple-borehole tests is given in Table 3.3, as well as estimates on crater dimensions and estimates of volume of shale removed.

In the section that follows, fragment size distributions for single- and multiple-borehole tests will be presented.

Table 3.3.
Multiple Borehole Tests at Colony and Anvil Points Mines.

Shot	No. Holes	Charges per Hole	Hole Diam (m)	Hole Spacing (m)	Stem Length (m)	Middle Stem Length (m)
79-12	4	1	0.15	3.16	1.82	none
79-15	4	1	0.11	2.13	1.93	none
79-16X	4	1	0.11	2.19	1.49	none
B-1	5	1	0.16	3.81	1.64	none
B-2	5	1	0.16	3.81	1.86	none
B-3	5	1	0.16	3.81	1.73	none
B-4	5	1	0.16	2.44	1.73	none
C-1	8	1	0.16	3.23	2.29	none
C-2	16	1	0.16	2.42	3.20	none
C-5	16	1o/2e	0.16	2.42	6.1o/3.05e	3.05
D-1	16	1o/2e	0.16	2.42	4.4o/3.0ea	4.22a
D-2	16	2	0.16	2.42	2.9oa/ 2.7ea	2.88oa/ 2.7ea
D-3X	16	2	0.16	2.42	2.7oa/ 1.7ea	2.88oa/ 2.7ea

X - hole failed to detonate

o - odd numbered holes

e - even numbered holes

In D-series even numbered holes always lower level

Table 3.3 (cont.).

Test	Weight/ Charge (kg)	Length/ Charge (m)	Type Charge
79-12	27.25	1.86	A
79-15	14.55	1.93	A
79-16	11.90	1.57	A
B-1	29.50	1.65	T
B-2	30.00	1.68	T
B-3	60.00	1.98	T
B-4	60.00	3.35	T
C-1	28.70	2.29	T
C-2	48.40	2.74	T
C-5	74.90	3.10	I
D-1	105.10a/63.1ea	4.350a/2.61ea	I1
D-2	99.30a/74.9ea	4.110a/3.10ea	I1
D-3	108.72I1a/50.0Ta/74.9ea	4.50I1a/4.00I1a/3.1ea	I1/T*

A - ANFO

T - TNT (prills)

I - Ireco 1115

I1 - Ireco 1205C

o - odd numbered holes

e - even numbered holes

a - average values

* - TNT used in third level charges

In D-series even numbered holes always lower level

-145-
PROFILE 1

PROFILE 2

0 100 200
CENTIMETERS

Crater profiles 1 and 2 for the intermediate scale experiment 79.12.

PROFILE 3

PROFILE 4

0 100 200
CENTIMETERS

Figure 3.3a. Crater profiles for multiple borehole Test 79-12.

Figure 3.3b. Crater profiles for multiple borehole Test 79-15.

Figure 3.3c. Crater profiles for multiple borehole Test 79-16.

Figure 3.3d. Crater profiles for multiple borehole Tests B-1 and B-2.

Figure 3.3e. Crater profiles for multiple borehole Tests B-3 and B-4.

Figure 3.3f. Crater profiles for multiple borehole Tests C-1 and C-2.

Figure 3.3g. Crater profiles for multiple borehole Tests D-1 and D-2.

Figure 3.3h. Crater profiles for multiple borehole Tests C-5 and D-3.

E. Fragment Size Distribution.

Due to the importance of controlling the fragment size of oil shale broken by the explosive in VMIS retort formation, the muck was carefully examined for size distribution. The normal procedure was to transport the resulting fragmented shale utilizing an LHD vehicle outside the mine where it was run through a commercial power sieve and separated into various size ranges. After the muck was separated into sizes, the muck was either weighed or the volume of each pile was estimated. Data was recorded either by percent weight or percent volume of the muck falling in the several selected size ranges.

No size data is available from the 1978 series of tests conducted by LANL. For the 1979 series of tests the fragments were separated into piles which contained particles greater than 18 inches (that is ones which would not pass through an 18 inch screen), those between 12 and 18 inches, between 8 and 12 inches, between 6 and 8 inches, between 4 and 6 inches, between 2 and 4 inches, and finally less than 2 inches. In this case the largest of these screens were hand operated.

For the 1981 and 1982 Anvil Points tests all screens were power operated. This device separated the fragments as follows:

- greater than 12 in.
- between 6 and 12 in.
- between 4 and 6 in.
- between 2 and 4 in.
- less than 2 in.

OR

- greater than 14 in.
- between 8 and 14 in.
- between 6 and 8 in.
- between 4 and 6 in.
- between 2 and 4 in.
- less than 2 in.

All the Anvil Points results (1981, 1982, and 1983) were based on weight rather than volume. In the Los Alamos 1979 tests, the figures presented were corrected for a void content in the muck of 50 percent. That is, when the total broken volume values were reported, the estimated muck volume was assumed to contain 50 percent void. This number was arrived at by filling a container with muck, then pouring in water until the container was full. The volume of water added was then compared to the total volume of the container.

Table 3.4 presents the fragment size distribution from the various tests for which data is available. This data has been converted into percent passing a given sieve size in Table 3.5. Also included in Table 3.5 is the average shale grade within the

Table 3.4.
Fragment Size Distributions.

Test	< 50 mm %	50-100 mm %	100-150 mm %	150-200 mm %	200-300 mm %	300-500 mm %	> 500 mm %
79-3	15.2	6.6	15.3	15.3	20.3	21.0	6.4
79-4	16.4	9.3	3.3	12.1	10.3	8.9	39.7
79-8	18.8	11.0	10.0	7.0	5.0	16.0	33.0
79-10	41.2	8.5	5.3	7.0	9.4	10.5	18.1
79-11	16.4	12.0	12.0	9.2	11.8	12.2	26.1
79-12	17.1	13.5	7.0	7.9	17.1	16.8	19.3
79-15	23.4	18.6	7.1	9.6	10.8	9.4	21.1
79-16	20.0	12.3	9.6	6.6	15.6	20.8	15.3

Test	< 50 mm %	50-100 mm %	100-150 mm %	150-300 mm %	> 300 mm %
A-1	38.0	19.0	13.0	22.0	8.0
A-2	31.0	29.0	13.5	21.0	5.0
A-3	18.0	20.0	11.0	25.0	26.0
A-4	33.0	24.0	14.0	21.0	8.0
A-5	48.0	26.0	11.0	10.0	5.0
A-6	19.0	18.0	14.0	22.0	27.0
B-1	29.0	22.0	16.0	13.0	20.0
B-2	34.0	24.0	15.0	18.0	9.0
B-3	63.0	17.0	10.0	8.0	2.0
B-4	46.0	29.0	12.0	10.0	3.0
C-1	15.0	19.0	13.0	25.0	28.0
C-2	27.0	30.0	15.0	21.0	7.0
C-5	27.0	24.0	13.0	21.0	15.0

Test	< 50 mm %	50-100 mm %	100-150 mm %	150-200 mm %	200-350 mm %	> 350 mm %
SB1	19.0	15.7	11.7	13.3	15.0	25.4
SB2	19.8	14.9	10.7	12.4	12.5	29.8
CV1	23.2	18.3	13.4	15.0	11.1	19.0
83C	38.9	27.0	13.0	11.1	7.6	2.4
83D	35.2	25.8	13.4	11.3	8.7	5.6

Table 3.5.
Percent Passing of Oil Shale Rubble.

Test	50 mm %	100 mm %	150 mm %	200 mm %	300 mm %	500 mm %	Grade (gal/ton)
79-3	15.2	21.8	37.1	52.3	72.6	93.6	43
79-4	16.4	25.7	29.0	41.1	51.5	60.3	30
79-8	18.0	29.0	39.0	46.0	51.0	67.0	30
79-10	41.2	49.7	55.0	62.0	71.4	81.9	30
79-11	16.4	28.4	40.4	49.6	61.4	73.6	30
79-12	17.1	30.6	37.6	45.5	62.6	79.4	30
79-15	23.4	42.0	49.1	58.7	69.5	78.9	30
79-16	20.0	32.3	41.9	48.5	64.1	84.9	28

Test	50 mm %	100 mm %	150 mm %	300 mm %	Grade (gal/ton)
A-1	38.0	57.0	70.0	92.0	31
A-2	31.0	60.0	73.5	94.5	9
A-3	18.0	38.0	49.0	74.0	31
A-4	33.0	57.0	71.0	92.0	13
A-5h	48.0	74.0	85.0	95.0	12
A-6	19.0	37.0	51.0	73.0	32
B-1	29.0	51.0	67.0	80.0	16
B-2	34.0	58.0	73.0	91.0	16
B-3h	63.0	80.0	90.0	98.0	10
B-4h	46.0	75.0	87.0	97.0	8
C-1	15.0	34.0	47.0	72.0	31
C-2	27.0	57.0	72.0	93.0	27
C-5	27.0	51.0	64.0	85.0	34

Test	50 mm %	100 mm %	150 mm %	200 mm %	350 mm %	Grade (gal/ton)
SB1	19.0	34.7	46.3	59.6	74.6	20
SB2	19.8	34.7	45.3	57.7	70.3	20
CV1	23.2	41.5	54.9	70.0	81.0	31
83C	38.9	65.9	78.9	90.0	97.6	9
83D	35.22	61.0	74.4	85.7	94.4	13

Table 3.5. (cont.).

Test	Average Size (mm)
79-3	194
79-4	292
79-8	282
79-10	105
79-11	208
SB1	166
SB2	172
CV1	133
83C	72
83D	80
A-1	81
A-2	85
A-3	159
A-4	85
A-5	53
A-6	147
C-1	173
C-2	90
C-5	99
B-1	99
B-2	85
B-3	39
B-4	57
79-12	230
79-15	158
79-16	213

test bed for each test. Figure 3.4 is a graph of the percent passing information for the various tests. It is graphed in six separate figures: three for the single-borehole tests and three for the multiple-borehole tests. Figure 3.4a presents results from Tests A-5, 83C, 83D, A-2, A-4, and A-1. All of these tests had an average particle size (size at the 50 percent passing point) between 50 mm and 90 mm. Figure 3.4b presents results from Tests A-3, A-6, SB1, SB2, CV-1, and 79-10. These tests resulted in an average particle size between 100 mm and 173 mm. Likewise Figure 3.4c gives percent passing results for Tests 79-3, 79-4, 79-8, and 79-11 where the average particle size ranged from 193 mm to 292 mm. The results for the multiple-borehole shots were grouped in a similar fashion. Figure 3.4d is for Tests B-3 and B-4 with an average size between 37 mm and 50 mm. Figure 3.4e is for Tests C-2, C-5, B-1, and B-2 which had average particle sizes between 84 mm and 99 mm. Finally, Figure 3.4f gives information on Tests C-1, 79-12, 79-15, and 79-16 where the average particle size ranged from 157 mm to 230 mm. It should be recognized that the accuracy of the data presented in Tables 3.4 and 3.5 and Figure 3.4 is felt to vary. This is mainly due to the sample used to determine fragment size. In the 1979 series as well as the single-borehole tests at Anvil Points (with the exception of Test CV1) as much muck as possible was used to determine the distributions. That is, the craters were completely dug out and all the resulting muck sieved and weighed or its volume estimated. In the B- and C-series however, only representative samples were used to determine the distributions. In the B- and C-tests all of the muck was not removed and of that removed only 20 to 25 percent was sieved and weighed. In Test A-2, for example, nearly 136,000 kg of shale was sieved and weighed. By contrast, in Test C-5, a test with approximately 24 times the charge and an assumed crater twenty-one times the volume of Test A-2, only about 40,000 kg of oil shale was weighed and sieved. It is therefore felt that the single hole data is more accurate than the multiple hole data. Of the multiple hole data the B-series and C-1 are felt to be more accurate than Test C-2 and C-5, the two largest tests for which fragment size distributions were obtained.

In the sections that follow the data presented earlier for the single hole and multiple hole tests pertaining to crater profiles and fragmentation will be further analyzed and discussed. The single hole data will be considered in the next section with the multiple hole data analyzed in the section following the single hole data.

F. DATA ANALYSIS

1. Single-Borehole Cratering Results. The primary objective of the testing programs at Colony and Anvil Points was to understand the mechanism of breakage of oil shale by explosives well enough to predict the breakage that would result from any given charge configuration. The results from any single-borehole test by itself is not of importance since retort formation

Figure 3.4a. Sieve data for several single - borehole tests.

Percent Passing vs Sieve Size

Figure 3.4b. Sieve data for several single - borehole tests.

Percent Passing vs Sieve Size

Figure 3.4d. Sieve data for two multiple - borehole tests.

Figure 3.4e. Sieve data for four multiple - borehole tests.

Percent Passing vs Sieve Size

Figure 3.4f. Sieve data for four multiple - borehole tests.

requires many holes to complete the blasting process. The single-borehole results are, however, important since they can be used to help design successful multiple-borehole patterns.

2. Philosophy Behind Retort Design. The results from single-borehole cratering tests can be used to characterize the properties of the material being blasted and the parameters obtained from the single-borehole tests can be utilized to design the multiple-borehole design. Redpath [Redpath, 1972] has presented equations for use in the design of both quarrying-type shot arrays and in situations where the blasting is done to a free face perpendicular to a multiple array of holes (trenching).

The suggested procedure in the design of an array of holes for a VMIS situation starts with a series of single-borehole tests that would involve determining a scaled volume removed versus scaled depth-of-burial (DOB) curve for the material in question. Ideally, this would be accomplished by using a given size spherical charge and emplacing it in a series of tests at different distances below the surface. For each of the tests conducted, the resulting crater would be mucked out and the volume of rock removed by the explosive determined. The customary procedure, although still controversial, would involve scaling the volume removed by the charge weight and the DOB of the charge with the cube root of the charge size. That is

$$SVOL = VOL/WC \quad (1)$$

and
$$SDOB = DOB/WC^{1/3}. \quad (2)$$

When plotted, the SVOL versus SDOB data should result in a curve such as shown in Figure 3.5. The curve shown is from the results of [Duvall and Atchison, 1957] from testing that the U.S. Bureau of Mines conducted in the mid 1950 time frame at the Anvil Points Mine (within the experimental mine which was located near the site of the 1980 tests). The maximum SVOL point on the curve defines a SDOB which indicates for the material being blasted the depth at which to locate a charge to remove the maximum amount of rock. This is called the optimum scaled DOB.

For the VMIS situation the many charges used can be taken to represent a plane explosive source as described below.

In this case the charge weight used for design purposes is the charge weight per unit surface area (WC/S^2 , where S is the charge spacing). The SDOB for use in a relationship with scaled volume removed is then defined as the DOB divided by the charge weight per unit area

$$SDOBPL = DOB/W/S^2. \quad (3)$$

The SDOB of the plane charge array has units of L^3/W (that is $L/W/L^2$). The SDOB of the point charge has dimensions $(L^3/W)^{1/3}$ (that is $L/W^{1/3}$).

Marlstone Semigelatin Type A Explosive

Figure 3.5. Scaled crater volume versus scaled charge depth in marlstone (Duvall and Atchison).

The powder factor for blasting gives the amount of explosive in kilograms required per volume of rock removed. That is $P_f = WC/\text{volume}$. Its dimensions are W/L^3 .

The implication given by [Redpath, 1972] and verified by empirical data is that the power factors and SDOB are related. The equation given by Redpath is that the

$$[SDOB_{pt}]^3 = 1/P_f \quad (4)$$

where $SDOB_{pt}$ is the scaled depth of burial of a point charge or stated differently, the powder factor for a point charge can be obtained by cubing the inverse of the scaled DOB. Point and plane charges will have the same effect if they have the same powder factors according to this model. Therefore,

$$[SDOB_{pt}]^3 = DOB/W/S^2 = SDOBPL \quad (5)$$

where SDOBPL is the scaled depth of burial of a planar charge.

Hence, if the optimum SDOB is determined from point charge investigations, then for a given charge size and spacing the optimum depth can be determined for the plane charge array. In other words the optimum SDOB from a point charge characterization of the material gives a relationship among DOB, charge size, and spacing for a plane array.

In practice, cylindrical charges are normally used in commercial blasting operations instead of spherical ones. In this case, the proper way to develop a SVOL versus SDOB curve might be to define the cylindrical configuration for the charge, i.e. a given diameter and length, and then bury the charge at various increasing depths until an optimum DOB was found. Using the same charge configurations, the above equation would be used to determine the proper spacing to be used at a given DOB. This was not done in the oil shale testing being described.

Redpath suggests using cylindrical charges with length to diameter that are small--less than five. With the series of tests being described, the aspect ratios of the charges varied from 4 to 37, and only one test had a ratio less than 5. With such long charges, it is hard to justify arguments based upon spherical charges. In order to determine the DOB for cylindrical charges it is common practice to take the midpoint of the charge length as the appropriate value. For a charge with a fixed diameter the SDOB could remain unchanged as the charge size increases. For example if the charge size is doubled and the depth of burial of the center of the charge is increased by 26 percent the SDOB will be unchanged. At the same time, to reproduce the same value for SVOL it would be required for the charge that was twice as large to remove twice as much rock. With cylindrical charges experience has shown that at a certain point no more volume of rock will be broken with added explosive.

If defined in the normal way, the SVOL versus SDOB relationship will cease to have the same meaning.

The variety of different charge aspect ratios used in the test series being described would likely tend to add to the scatter of data in the SVOL versus SDOB curve, as well as other pertinent results. This scatter would be in addition to scatter normally encountered due to differences in shale grades, explosive type, and geological variations from test site to test site.

The procedure used in the design of the mini retorts at Anvil Points was more empirical in nature than outlined above. Basically, the single-borehole tests were used to find an approximate value for the correct DOB, and the crater size developed was used as a first estimate of multiple-borehole spacings. These values were then refined in five-hole tests. The best estimates of charge size, DOB, and hole spacing were then used in single- and dual-level testing prior to the retort tests.

G. Cratering Results.

Figure 3.6 presents a curve of the volume of rock broken as a function of charge weight used. It should be pointed out that no attempt at scaling is made in the figure. It is presented just to observe the amount of rock broken as a function of charge weight. There is considerable scatter in the data but that is to be expected since burial depths, charges loading density, grade, and other important parameters are not accounted for. The general trend is an increasing value of volume with charge size.

Figure 3.7 presents the scaled volume broken as a function of SDOB of the charge. As stated above this is the typical curve used in the literature to present single-borehole cratering data. In the figure, the volume is scaled with charge weight and the DOB with charge weight to the one-third power. The DOB used here is the standard one which is defined as the distance from the free face to the center of the charge. As stated above most of the charges used in the determination of the data presented in Figure 3.7 had aspect ratios considerably greater than 5. The charges used by Duvall and Atchison in defining Figure 3.5 were all less than five except for one which had an aspect ratio of seven. The charge which had an aspect ratio of seven in Duvall and Atchison's results resulted in no crater formation. Details of some of the tests results presented in Figure 3.7 are given below.

H. Test Orientations

Of the results presented, three were conducted in ribs of the drifts with the borehole parallel to the bedding planes and all others were in the floor with boreholes perpendicular to bedding. Test 78-2 was a floor test while Test 78-3 was an

Figure 3.6. Volume removed for several single-borehole tests.

Figure 3.7. Scale depth of burial versus scale volume for several single-borehole tests.

identical rib test. The same is true for Test 79-8 (a floor test) and Test 79-11 (a rib test). Table 3.6 presents a comparison of these tests with one another. Table 3.6 shows that for identical charge weights and nearly identical depths of burial the rib shot (Test 78-3) resulted a volume of material removed that was more than forty percent greater than the volume removed by the floor test (Test 78-2). Similar results are shown in Table 3.6 for Tests 79-8 and 79-11. There, the rib test (Test 79-11) shows a much larger volume removed than the floor test (Test 79-8). Both tests used the same amount of explosive but unfortunately the loading density of explosive was not the same nor was the depth of burial. The rib test was less deeply buried and less densely loaded and resulted in a crater that was more than four times greater than the floor test. The third rib test used 90 kg of explosive and was the largest single hole test conducted-both from the standpoint of the amount of explosive used and the amount of material removed. There is no question that the performance of the explosive was better for the three rib tests than for the floor shots on the basis of material removed per weight of explosive. In Figure 3.6 the three rib tests have been indicated with an "r" beside the data points. All three points are on the high side of the data but equal scatter exists in the floor data. The average fragment size for the rib test (Test 79-11) was smaller than for the corresponding floor tests (Test 79-8). The size for rib test was 208 mm compared to 282 mm for the floor test. No sieving was done for Tests 78-3 and 78-2. The size of the average fragment for Test 79-3 (the third rib test) was 194 mm. The rib tests in general appear to have broken more rock per kg of explosive than corresponding floor tests and resulted in average particle sizes that fell in the intermediate range of the results for all of the tests conducted in the program.

It is felt therefore that the results of the rib test may be somewhat different from the results obtained for the floor tests.

I. Test Confinement

As stated earlier, tests were conducted within series to determine the effect of two different types of confinement on the resulting craters. Tests 79-4 and CV1 were conducted with mounds of oil shale rubble piled on top of, but not contacting, the "free" surface that was being blasted. A plywood cover was placed above the floor of the test bed on spacers to leave a surface for reflection of stress waves but yet to contain as much as possible of the fragmented rock. Test 79-4 was not successful in that the rubble pack failed to fully contain the fragmented shale. Test CV1 was more successful but still did not totally contain the crater material. The effect of the rubble pack appears to have been the formation of a crater with an unusually large radius when compared to the depth of burial of the charge. The ratio of radius to depth of burial in CV1 was greater than three - by far the largest value of this ratio for any of the

Table 3.6.
Test Data for Some Floor and Rib Shots at Colony Mine.

	78-2 - Floor	78-3 - Rib
Charge	5.22 kg	5.22kg
DOB	2.22 m	2.26 m
Rock Volume	6.82 m ³	9.73 m ³
Loading Density	856 kg/m ³	829 kg/m ³
Charge Volume	0.0061 m ³	0.0063 m ³
SVOL	1.21 m ³ /kg	1.86 m ³ /kg
SDOB	1.29 m/kg ^{1/3}	1.30 m/kg ^{1/3}
SVOLR/SVOLF	1.44	1.44
	78-8 - Floor	79-11 - Rib
Charge	14.51 kg	14.51
DOB	2.81 m	2.71 m
Rock Volume	10.00 m ³	43.3 m ³
Loading Density	829 kg/m ³	484 kg/m ³
Charge Volume	0.0175 m ³	0.030 m ³
SVOL	0.689 m ³ /kg	2.984 m ³ /kg
SDOB	1.15 m/kg ^{1/3}	1.11 m/kg ^{1/3}
SVOLR/SVOLF	4.33	4.33

tests conducted.

Test A-5 was conducted in a small room mined out of a drift for the expressed purpose of determining the effect of limited room size on fragmentation results. The room in which Test A-5 was conducted was approximately 3 by 3 m as opposed to the rooms of at least 10 by 10 m for the other cratering tests. As can be observed from looking at the results obtained for crater profiles and fragmentation in Figure 3.2, this confinement created a very deep elliptical crater elongated in the direction of the open drift. It also resulted in a much smaller volume of rock broken and a very small average fragment size.

J. Effects of Geology

Throughout the series, the test sites were closely examined for major geologic abnormalities - open joints, vugs, dipping planes, etc. Very few of the resulting craters showed that geological features present played an important role. By looking at the crater profiles (Figure 3.2), it appears only three tests were affected in a major way. Tests 78-2 and 78-3 appear to have been the most affected. Test A-3 shows a minor effect. The crater profiles for both Tests 78-2 and 78-3 appear to have been defined by unbroken joint and bedding planes. There was a visible effect in Test A-3 from an unbroken joint, but this was not as pronounced as was the case for Test 78-2 - probably since its joint was located further from the borehole than in Test 78-2.

K. Effects of Shale Grade

In viewing the results of the tests with regard to crater volume no effect of oil shale grade on volume removed is evident. There is however an effect of shale grade on fragment size. Figure 3.8 gives the percent passing various sieve sizes as a function of oil shale grade. As is evident from viewing the figures for all sieve sizes the percent passing decreases as the shale grade increases. Figure 3.8a presents single borehole results and Figure 3.8b gives the data for multiple borehole tests. Figure 3.9a and 3.9b present the average fragment size as a function of shale grade. Although there is considerable scatter in both the percent passing curves and the average fragment size curves it is apparent that the higher the oil grade the larger the fragment size.

L. Effect of Stemming

Tests SB1 and SB2 were conducted to examine the effect of stemming versus the presence of no stemming. A third test, Test 83D was identical to Test SB2 except for shale grade. The cratering results for all were confusing.

Test 83D was conducted in a 9 gal/ton grade shale and Tests SB1 and SB2 were conducted in 20 gal/ton shale. All three tests

% Passing 2 inch Sieve

vs Shale Grade-Single

Figure 3.8a. % passing versus shale grade for single hole tests.

% Passing 4 inch Sieve

vs Shale Grade-Single

Figure 3.8a. % passing versus shale grade for single hole tests.

% Passing 6 inch Sieve
vs Shale Grade—Single

Figure 3.8a. % passing versus shale grade for single hole tests.

% Passing 8 inch Sieve
vs Shale Grade—Single

Figure 3.8a. % passing versus shale grade for single hole tests.

Figure 3.8a. % passing versus shale grade for single hole tests.

Figure 3.8b. % passing versus shale grade for multiple hole tests.

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% Passing 4 inch Sieve vs Shale Grade—Multiple

Figure 3.8b. % passing versus shale grade for multiple hole tests.

% Passing 6 inch Sieve vs Shale Grade—Multiple

Figure 3.8b. % passing versus shale grade for multiple hole tests.

% Passing 8 inch Sieve

vs Shale Grade—Multiple

Figure 3.8b. % passing versus shale grade for multiple hole tests.

% Passing 12 inch Sieve

vs Shale Grade—Multiple

Figure 3.8b. % passing versus shale grade for multiple hole tests.

Average Fragment Size vs Shale Grade—Multiple

Figure 3.9a. Average size vs shale grade for multiple hole tests.

Average Fragment Size vs Shale Grade—Single

Figure 3.9b. Average size vs shale grade for single hole tests.

were conducted in the same size borehole - both diameter and depth - and utilized the same weight of the same explosive. The distance from the top of the charge to the free surface was nearly the same for all three tests. Tests SB2 and 83D were shot in an open hole - not stemmed while Test SB1 was stemmed. The results from Tests SB1 and SB2 were almost identical, while Test 83D fragmented nearly twice the volume of rock as either Test SB1 or SB2, pulled more than twice as deep and all three tests created craters with nearly equal radii. The average fragment size for Tests SB1 and SB2 was nearly twice the average fragment size for Test 83D. The results from Test 83D are far better than Tests SB1 and SB2 no matter what criteria is used to judge. In fact the results of both Tests SB1 and SB2 would both be judged to be disappointing compared to the results of all other tests conducted. The results of Test SB1 when compared to Test SB2 would seem to indicate that stemming has no effect on the results of crater blasting. When compared to the other tests however both Tests SB1 and SB2 seem to be abnormal.

M. Effect of Explosive Type

The explosive type used in the various tests appears to have had no major effect on crater volume. Figure 3.10 presents volume removed as a function of charge weight and includes multiple and single borehole testing. Figure 3.10a presents the results for all tests conducted including the D series tests. Figure 3.10b presents data only for the tests which used less than 280 kg of explosive since details on the type of explosive used cannot be determined in that range in Figure 3.10a due to the large amount of data in that region. The type of explosive used in each of the tests is indicated on the graph. As can be seen from viewing the graphs presented there does not appear to be a trend with regard to type of explosive of explosive used.

N. Effect of Burial Depth

Figure 3.11 presents data which shows the effect of burial depth (in terms of charge length) on the average particle size. Most of the tests were conducted with geometries which has the length of stem (or open borehole) nearly equal to the length of charge use. All but two tests had stem to charge ratios which varied from 0.85 to 1.37. The stem length in Test 79-4 was 3.2 charge lengths and in Test 79-8 the stem length was 2.3 times the charge length. The relatively great burial depth appears to have resulted in average fragment sizes which were much larger than those obtained when the stemming and the charge were nearly equal in length. This very large burial depth did not appear to affect the size of craters produced.

O. Scaled Depth of Burial

The range of scaled depths of burial investigated in this series of tests was from $0.7 \text{ m/kg}^{1/3}$ to $1.9 \text{ m/kg}^{1/3}$. The SVOL vs SDOB curve shown in Figure 3.5 as developed by [Duvall and

Charge Weight vs Volume Removed

Figure 3.10a. Charge weight vs volume - three explosives.

Charge Weight vs Volume Removed

Figure 3.10b. Charge weight vs volume - three explosives all tests.

Figure 3.11. Average fragment size vs stem-to-charge length ratio.

Atchison, 1957] covers the range of scaled depth of burial from zero to $1.6 \text{ m/kg}^{1/3}$ and in fact the optimum scaled depth of burial was $0.91 \text{ m/kg}^{1/3}$. The values of SDOB therefore are similar. The large amount of scatter present in the results of the current test series which was not obtained in the earlier study could be attributed to two reasons. First, the charges used by Duvall and Atchison were much smaller than the ones used in this study and did not vary much (.7 to 6.1 kg). In the current study the smallest charge was 5.22 kg and charges as large as 90.72 kg were used. Secondly, the charge geometry used in the current study could not be classified as spherical. Cylindrical charges were used in order to prevent the necessity of drilling large diameter holes. The crater blasting theories upon which the SVOL vs SDOB characterization curve is based depend upon spherical or near spherical charges being used.

Blasting results can perhaps better be predicted when non spherical charges are used by utilizing an unscaled predictive curve. Dick and coworkers [Dick et. al., 1990] have proposed such a scheme. There it is found that such a curve can predict expected results well from charges of only a few milligrams to charges to charges which include large nuclear explosions. In the case of this study a curve of Log Yield vs Log Volume is recommended. Log yield is the log of the charge size while log vol is the log of the volume of material removed. This curve is presented as Figure 3.12. Even though some scatter is evident in this curve - which shows results for all of the tests - it appears to be a better predictor than any of the other schemes attempted. This of course implies that the power of the explosive masks any other of the normal parameters felt to be of importance - including explosive, type depth of burial, etc.. Bear in mind however that most of the tests conducted in this series were in the vicinity of what was felt to be the optimal depth of burial.

P. Multiple-Borehole Cratering Results

Figure 3.13 shows a plot of volume removed versus charge weight from the multiple-borehole tests conducted during the test program. With the exception of Test C-1 the results seem to fit the trend curve established by the single hole results (See Figure 3.10a). It appears Test C-1 broke nearly twice as much volume as would have been predicted by using the trend curve.

Table 3.7 gives the powder factor--weight in kg of explosive required to break a cubic meter of shale for the multiple-borehole tests. These values ranged from a low of 0.52 for Test C-1 (the most efficient) to 1.49 kg/m^3 for Test 79-12 (the least efficient). Test 79-12 was conducted in a small room. Postshot examination revealed that some of the pillars at the room edge were actually undercut and highly fractured. It did not, however, pull unusually deep as did the single-borehole Test A-5 that was confined in a similar manner.

Table 3.7.
Powder Factor Statistics.

Test	Powder factor (kg/m ³)
79-12	1.49
79-15	0.72
79-16	0.98
B-1	1.18
B-2	0.92
B-3	1.30
B-4	1.04
C-1	0.54
C-2	1.20
C-5	1.37

Log Yield vs. Log Vol

Figure 3.12. Log-log plot of charge weight vs volume removed.

Volume Removed vs Charge Weight

Figure 3.13. Charge weight versus volume removed.

It should be pointed out that in two of the multiple borehole tests (Tests 79-16 and B-4) one hole in the array did not detonate. Since they were later detonated by the addition of another booster and some small amount of additional explosives, this was not considered in the analysis of the results. In Test 79-16, in fact, there was an indication that the explosive in the other three holes did not detonate properly and this could also help account for the high powder factor obtained. The objective of Test B-2 was to determine if a time delay between holes was beneficial. Test B-2 was identical to Test B-1 except that in Test B-1 all holes detonated simultaneously while in Test B-2 a delay of 10 milliseconds was used between the center hole and the outer four holes. The results obtained showed a slight improvement in blasting efficiency and hence delays were used in all subsequent tests.

Test B-3 used a larger borehole diameter and hence a greater amount of explosive than Test B-2. As can be seen from Table 3.7, this resulted in a less efficient result. Test B-4 used a longer charge (and hence more explosive) resulting in a deeper DOB. As shown from the powder factors in Table 3.7 this resulted in a slightly lower powder factor and a slightly more efficient result. However, the improvement was not felt to justify the added expense of the deeper drilling required.

Q. Conclusions

The test series permits several conclusions to be drawn concerning crater profiles (volume removed) and fragmentation (particle size distributions).

Oil shale appears to be a difficult rock in which to accurately predict results. The amount of scatter in the data obtained was larger than would have normally been expected. Even though the series utilized various explosives ranging in energy potential from ANFO through TNT, it was very difficult to determine any major effect on either volume removed or particle size.

The variation in oil content was great even within any single test bed. In the interval in which the series of tests were conducted, the grade ranged from 5 gal/ton to 80 gal/ton. The average grades within the test series ranged from 9 gal/ton to 43 gal/ton. There was no discernable correlation of crater profile formed with oil grade. The shale grade, however, did seem to affect the size distribution of the resulting muck. The higher the grade, the larger the fragments obtained. The opposite can be said for amount of explosives used. For example, Tests B-3 and B-4 used twice the explosive as Tests B-1 and B-2 and the resulting average particle size was much smaller.

As the DOB increases (in terms of the length of the charge), a larger average fragment size can be expected but no effect was observed on volume removed.

Although design curves were generated for SVOL versus SDOB there is considerable scatter in the results obtained. Figure 3.14 shows results for all single and multiple hole tests from the standpoint of Log Yield versus Log Volume removed. The line shown is a least square fit of all data and includes the three mini retort tests as well as the smallest tests (Tests 78-1, 78-2, and 78-3). This is the curve recommended for predicting blasting results. The slope of the best fit is 1 indicating a powder factor of 1 kg/cubic meter of shale fragmented appears to be valid for the range of charge sizes investigated.

One result of the program that is apparent is that blasting with horizontal boreholes yields somewhat different results than blasting with vertical holes. Hence, a series of tests providing additional information on rib tests should be conducted.

Confinement of a blast by too small a room appears to cause a deeper depth-of-pull and result in smaller fragments. Confinement with rubble piled on top of the test site results in a crater with an abnormally large radius and tends to increase the volume removed at the expense of depth of pull.

Finally, with the exception of a few isolated cases, there appears to be no major effect of geology on resulting crater profiles. It is certain that it has an effect, but this effect unfortunately appears to be hidden in the scatter of data resulting from the other various parameters.

CHAPTER 4 - STEM BEHAVIOR AND PERFORMANCE

A. Stemming Requirements

Proper stemming of the explosively loaded shotholes in oil shale blasting is required for a variety of reasons. In terms of rock fragmentation, successful stemming will serve to contain the explosive reaction products (gases) so that this gas energy might be better utilized to fragment and move the rock. In terms of blast performance, successful stemming could be required to protect delayed charges in a decked-charge, multiple-borehole shot design. Considerable effort in the Anvil Points' portion of the oil shale fragmentation program was directed toward evaluating and improving stemming designs. These efforts involved the measurement and analysis of stem performance in explosive tests designed to specifically study stem performance, as well as in many tests designed primarily to study rock fragmentation. Both existing and specifically developed instrumentation techniques were utilized to monitor stem performance.

B. Instrumentation

A variety of instrumentation techniques were developed and/or used to make measurements of actual stem performance in explosive fragmentation tests. These techniques have been discussed in some detail in Chapter 2 on instrumentation. The techniques allowed for the measurement of stem failure by recording the propagation of a "failure wave" through the stem column with the CORTEX time-domain reflectometry system or the measurement of stem motion with embedded accelerometers and velocity gauges. The CORTEX data generally served to confirm stem bridging and to give an indication of the length of stem subject to failure and crushing prior to bridging. The accelerometer and velocity gauge data provided a more accurate measure of stem accelerations, velocities, and displacements thus serving to quantify stem motion. The CORTEX system was employed in many multiple-borehole tests thus serving to confirm stem performance when data recording limitations precluded using accelerometers or velocity gauges on all the shot holes. The accelerometers and velocity gauges were used more often in single-borehole and simple special tests designed to evaluate particular stem designs.

Commercially available piezoelectric and piezoresistive type accelerometers, manufactured by ENDEVCO and PCB Piezotronics, were placed in gravel stems of shot holes at distances as close as one meter to the explosive charges. As both types of accelerometers were subject to high-frequency noise and/or significant baseline shift in the adverse environment in which they were employed, efforts were also directed toward developing an electromagnetic velocity gauge technique suitable for measuring rock mass motion and stem performance in the immediate vicinity of detonating explosive charges.

The type of electromagnetic velocity gauge found to be most effective is depicted in Figure 2.9 and described in detail by Young, et al. (1983). This gauge functions simply by the motion of a bar magnet through a helically-wound pickup coil. As indicated in Figure 2.9, the bar magnet, which is free to slide within the tube wrapped with the pickup coil, is positioned prior to the explosive event at a location such that the expected rock mass or stem motion will cause the pickup coil to move past the magnet. The magnets were typically held in place by their magnetic adherence to a small bolt positioned in the tube assembly as illustrated in Figure 2.9. The arrival of any explosive-induced shock or motion would dislodge the magnet for this supporting bolt, and during the few milliseconds of valid recording, the magnet would remain essentially stationary in space by nature of its inertia while the pickup coil tube assembly moved past it. By performing a variety of drop test calibrations on the electromagnetic velocity gauge assembly, it was possible to calibrate each gauge for the voltage generated as a function of tube velocity past the magnet. The gauge outputs were typically accurate to within 5 % over the 8- to 10-cm travel built into the gauges. As each gauge had a known travel or displacement limit, it was possible to re-verify the gauge calibration factors during an explosive test by comparing the gauge displacement obtained by integrating the velocity record with the mechanically defined displacement limit.

The electromagnetic velocity gauges had the disadvantage that they had a greater impedance mismatch with the oil shale and the grout in which they were embedded than did the piezoelectric and piezoresistive accelerometers, and that special precautions had to be taken during gauge installation to ensure that the magnet was not dislodged or mislocated prior to the execution of the explosive test. The electromagnetic gauges had the advantages that they provided a large amplitude noise free voltage output which was directly proportional to gauge velocity and that they could provide long (up to 30 ms) records of ground or stem motion.

As described in Chapter 2, the CORRTX technique was utilized to determine with great accuracy the detonation times and detonation velocities of the explosive charges in the tests. As illustrated in Figures 2.13 and 2.19, the CORRTX system gave excellent records of detonation velocities. As may also be seen in these figures, a distinct cable shortening, but at a much lower velocity, continues to occur past the end of the explosive charge. This shortening is due to the crushing of the cable by the stemming material in which it is embedded and is a measure of the propagation of the shock wave propagating through the stemming column. The stem behavior data portion of the CORRTX records often gave valuable information on the performance of the various stem materials and geometries tested and employed in the Anvil Points tests.

C. Field Test Program

Twenty-seven explosive tests involving thirty-two stem designs were conducted within the Anvil Points program in which instrumentation provided direct data on stem performance. In addition, inferred information was obtained on fifteen tests, including nine multiple-borehole, multiple-charge tests. All of the tests from which stem behavior data was obtained, along with basic test parameters, are summarized in Table 4.1.

The majority of stem performance tests were classified as part of the Special-Series Tests which included small-scale pre-splitting and charge timing tests, as well as large-scale tests designed primarily to evaluate stemming performance and/or multiple-borehole cratering phenomena. All of the large-scale tests which provided stem performance data (14 in the Special-Series, 12 in the C-series, and one in the B-Series) were conducted with nominal 15.8-cm-diameter boreholes loaded with between 35 and 95 kg of explosive charge.

Two of the larger Special-Series are representative of the types of test conducted within this series to evaluate stemming performance. The test geometry utilized in the Special-Series Tests DP-1 and DP-2 is illustrated in Figure 4.1. These tests were designed to evaluate the effectiveness of an intermediate stem between decked charges in a single-borehole, in terms of delayed detonator performance and dead-pressing of a slurry explosive charge. Thirty- and sixty-millisecond delays were utilized between the detonation of the lower charges comprising 90 kg of IRECO's 1205C emulsified ANFO slurry and the upper charges comprising 50 kg of the same explosive. All charges were initiated by an experimental electronic initiating device provided by Atlas Powder Company. These detonators were placed in 1.4 kg NG boosters which were in turn embedded in approximately 30 cm (5 kg) of prilled TNT within the wellbore. As indicated in Figure 4.1, the intermediate stem between the two delayed charges in each test had a composite construction comprising a clay powder layer, a 19-mm gravel section, an intermediate grout plug, and a final 19-mm gravel section. The clay layer was intended to restrict the flow of the lower charge explosive gases into the lower 19-mm gravel section, thus increasing the possibility for successful bridging. The grout plug was intended to provide an initial frictional and inertial bridging of the wellbore so as to further improve the bridging of the gravel stem immediately below the plug. The possible benefits of such an inertial grout plug were investigated further by means of a numerical model as discussed in the modeling section below.

As noted in Figure 4.1, 50,000 G piezoelectric accelerometers, manufactured by PCB Piezotronics, were placed in the intermediate grout plugs; and 20,000 G piezoresistive accelerometers, manufactured by Endevco, were placed near the

Table 4.1.
Summary of the Stem Performance Tests Conducted at Anvil Points Mine.

TEST NO.	GAUGE NO.	H.E. kg	DET *	SP #	DIST m	TOA msec	ACCEL Mm/s ²	VEL m/s	SVEL m/s	STRESS MPa
Tests with Prilled TNT Explosive										
A-7	PCB1	50	B	F	1.01	0.65	0.49	55	1547	128
	PCB2				2.01	5.35	0.49	50	376	28
B-3	PCB5		B	F	1.74	2.15	0.11	37	808	45
C-1	PCB1	35	B	F	0.91	0.48	0.27	40	1925	116
	PCB2				2.04	4.78	0.49	35	427	22
	PCB3				0.85	0.61	0.49	50	1411	106
	PCB4				1.98	4.18	0.49	50	474	36
C-3b	PCB1	50	T	F	2.44	6.16	0.44	42	396	25
	PCB2				3.66	16.3	0.17	52	224	17
C-3c	PCB1	60	T	F	3.35	14.9	0.05	44	225	15
C-3d	PCB1	60	B	H	0.37	0.21	0.49		1784	
	PCB2				1.83	5.08	0.36	41	360	22
	PCB3				3.35	99	0.00	0	34	0
C-3e	LPCB1	60	T	F	1.37	2.75	0.69		499	
	PCB2				3.51	8.75	0.69		401	
	PCB3				5.18	15.5	0.49	40	335	20
	PCB4				6.43	22.5	0.08	29	286	12
C-3f	LPCB1	50	T	H	3.35	99	0.00	0	34	0
C-3g	PCB1	50	T	H	3.35	99	0.00	0	34	0
S-1	PCB1	60	T	H?	1.95	5.6	0.78	60	350	32
	PCB2				3.69	13.8	0.05	35	267	14
S-2	PCB1	60	T	H	2.16	11.9	0.78	75	183	21
	PCB2				4.18	19.7	0.06	18	213	6
Tests with IRECO 1205C Emulsified Slurry										
A-9T	VG1	55	T	H	3.66	2.20		5.3	1664	13
A-9B	PCB1		B	H?	2.59	2.54	0.54	30	1020	46
C-3i	PCB1	68	T	H	1.52	3.60	0.20	50	423	32
	PCB2				3.05	99	0.00	0	31	0
C-6a	VG1	68	B	F	3.29	5.10		14	645	14
C-6b	VG1	61	B	F	1.89	6.44		46	293	20
C-6b	VG2	68	B	F	2.83	3.74		52	758	59
C-6b	PCB1	68	B	H	2.56	99	0.00	0	26	0
CF-1	VG1	55		H?	2.59	8.70		16	298	7
CF-2A	VG4	55	B	H?	2.71	6.28		39	432	25
CF-2B	VG5	68	B	H?	2.47	6.46		40	382	23
DP-1	PCL1	95		H?	1.52	2.88	0.07	46	529	37
DP-2	END1	95		H	2.90	8.70	0.01	8	333	4
S-3	PCB1	68	B	H?	2.35	6.10	0.14	35	385	20
S-4	PCB1	68	B	H?	2.04	3.00	0.12	49	681	50
S-5	VG1	68	B	H?	2.83	7.70		7.6	368	4
S-6	PCB1	75	B	H	1.37	4.05	0.14	9.5	339	5

Tests with IRECO 1185 Emulsified Slurry

A-8	PCB1	68	T	F	3.05	6.13	0.54	40	497	30
C-3hI	PCB1	68	T	H	2.44	7.80	0.41	42	313	20
	PCB2				3.66	15.40	0.03	4	238	1

Tests with IRECO 1115 Emulsified Slurry

A-1OT	VG1	55	T	H	3.66	12.70		7.3	288	3
A-1OB	PCB1		B	H	2.59	8.75	0.02	7.7	296	3
C-5	PCB1	68	T	F	2.29	5.23	0.74	50	437	33

Key: * Detonator location with respect to free surface, B = Bottom, T = Top, # Stem Performance, F = Failed, H? = Possibly Held, H = Held.

Fig. 4.1. Single-hole stem test geometry for Tests DP-1 and DP-2 showing typical test configuration and instrumentation.

interface between the upper portion of the gravel stem and the upper explosive charge. These accelerometers were designed to measure the shock wave propagation through the stems and to establish whether or not the stems bridged sufficiently to protect the upper delayed charges. As illustrated in Figure 4.1, several CORRTEX cables were placed within the explosive charges and two special holes in each test were drilled to carry special CORRTEX cables. The 108-mm diameter hole drilled 2.3 m from the main explosive charge hole was intended to measure any fracturing of the rock at that range, through the cutting and consequent impedance change in the CORRTEX cable through it. The inclined hole leading from the upper level mine floor into the upper explosive charge was designed to provide access for a short length of CORRTEX cable to measure the detonation time and detonation velocity of the upper charge. It would not be possible to exit a CORRTEX cable capable of measuring upper charge detonation velocities through the through the previously detonated lower charge and to the lower mine level.

The output from the piezoelectric and piezoresistive accelerometers utilized in these tests is illustrated in Figure 4.2. The record for the PCB accelerometer located in the intermediate grout plug of test DP-1 shows the distinctive two shock response characteristics of a majority of the stem performance measurements. An initial shock at a time of 0.7 ms corresponds to the arrival time of the initial shock wave information through the rock surrounding the wellbore. The larger amplitude arrival at 3.0 ms corresponds to the shock transmitted through the granular stemming material between the lower charge and the intermediate grout plug. As noted in Figure 4.2, the velocity obtained by integrating the acceleration record and displacement obtained by integrating the velocity record both indicate that the intermediate grout plug was not subject to a continual acceleration and velocity increase and that this stem most probably bridged. That the stem held is further attested by the record obtained from the piezoresistive accelerometer at the interface between the upper gravel stem and the upper explosive charge in Test DP-2, with a 60 ms delay between charge detonation. This accelerometer record, also shown in Figure 4.2, shows a much reduced acceleration peak and only a very small, 2.0 m/s, velocity for the interface. That the stem functioned adequately was further attested to by the proper detonation of the upper delayed charge at 60 ms.

Data from electromagnetic velocity gauges embedded in the grout plug at the collars of explosive holes are illustrated in Figure 4.3. Both of these tests employed a composite gravel stem with a clay layer directly adjacent to the explosive charge. Both tests also utilized an approximately 2.5-m-long (55 kg) explosive charge of IRECO 1205C slurry. In Test A-9 the stem comprised 0.1 m of clay, 1.13 m of pea gravel, and 2.07 m of 19 mm gravel. The stem was capped with a 0.30 m grout plug at the collar of the hole and contained an electromagnetic velocity gauge. The velocity record obtained from Test A-9, and shown in

Fig. 4.2. PCB and Endevco accelerometer data from Test DP - 1 and DP - 2. Solid line is acceleration, dotted line is velocity, and dashed line is displacement.

VELOCITY GAGE DATA TEST A-9

VELOCITY GAGE DATA TEST CF-2

Fig. 4.3. Electromagnetic velocity gauge data from tests A-9 and CF-2.

successful stems. The peak velocity of just over 5.0 m/s was caused by the rock shock which propagated through the oil shale surrounding the shot hole. The sustained, 5.0 m/s, velocity is most probably due to the spall of the mine floor in which the velocity gauge was grouted. That there are no increases in this velocity out to 24 ms when the gauge met its limit of travel indicates that the gravel stem bridged in the shot hole.

In Test CF-2 the stem was much shorter with 0.15 m of clay, 1.01 m of pea gravel, and 1.25 m of 19-mm gravel. Again, an electromagnetic gauge was grouted into a 0.30 grout plug at the collar of the hole. In contrast to Test A-9, the velocity record for Test CF-2, as shown in Figure 4.3, illustrates the response typical of stem failure. The Test CF-2 data show the characteristic rock shock at 1.3 ms followed by a much larger stem shock at 6.8 ms. The very rapid drop in gauge output occurring just at 8 ms is due to the gauge meeting the mechanical limit of its travel. The 110-mm displacement obtained by integrating the velocity record up to the sharp velocity drop served to reconfirm the gauge calibration factor as noted earlier. The 30 m/s velocity of the stem prior to the meeting of the gauge displacement limit indicates that this stem is probably undergoing failure and in the process of being rifled from the borehole.

D. Data Analysis and Correlation

All of the gauge data relevant to stem performance obtained from the tests at Anvil Points are tabulated in Table 4.1. The location detonator (T for top detonated, B for bottom detonated) for explosive charge initiation with respect to the stem performance measurements are indicated in this table. The actual performance of each stem in terms of bridging is indicated in the table. The stem performance factors--F for failed, H? for probably held, and H for definitely held--are based upon gauge data, CORRTX data, and post shot observation. Accelerometers and/or velocity gauges were emplaced in the stem column in some thirty tests. The stem shock arrival time, the peak acceleration, for the accelerometers, and peak velocity, either measured directly for the velocity gauges or integrated for the accelerometer data, are tabulated for all 46 successfully recorded gauges in Table 4.1. The acceleration and/or velocity records obtained from all the gauges which were recorded for stem performance are available in the A-, B-, C-, D-, and S-Series Summary Reports by SAI. The type of CORRTX cable in the stems are also indicated in Table 4.1. Records of the CORRTX data providing useful information on stem performance are given in [Schmitt, 1984]. Finally, Table 4.1 includes the lengths of various stemming materials used to make up each stem. The stem geometry data of Table 4.2 have been sorted according to possible stem behavior (failing, possibly bridging, and bridging).

The data presented in Tables 4.1 and 4.2 have been incorporated into a Lotus 123 spreadsheet and subjected to

Table 4.2.
Details of Stem Construction Geometry for Failing, Possibly
Bridging, and Bridging Stems.

Test No.			STEM GEOMETRY (m)							
			CLAY	PEA	GROUT	3/4	GROUT	OTHER		
Failing Stems										
A-7	TNT	B				3.05				
A-8	1185	T		1.52		3.05				
C-1	TNT	B		1.14		1.14	0.30			
C-3b	TNT	T	0.15		0.46		0.30	1.50	Clay/Pea	
C-3c	TNT	T	0.15		0.46			1.50	Pea	
C-6aA	1205C	B	0.15	0.64		2.04	0.30			
C-3eL	TNT	T	0.15	3.40				3.20	TNT	
C-3eU	TNT	T		2.90						
C-4-1	TNT	B				1.68				
C-5	1185	T		1.52		1.74				
C-6bA	1205C	B	0.10	0.50		0.90	0.30			
C-6bC	1205C	B	0.10	0.70		1.50	0.55			
Possibly Bridging Stems										
A-9B	1205C	B	0.15	0.76		1.37	0.46			
C-3iU	1205C	B				1.80	0.30			
CF-1	1205C	B	0.15			2.29	0.30			
CF-2A	1205C	B	0.15	1.01		1.25	0.30			
CF=2B	1205C	B	0.15	0.79		1.22	0.30			
DP-1	1205C		0.15			1.22	0.30	1.22	3/4"	
S-1	TNT	T		1.96		2.80				
S-3	1205C	B	0.15	0.79		1.74				
S-4	1205C	B	0.15	0.43		1.13	0.49			
S-5	1205C	B	0.15			2.29	0.30			
Bridging Stems										
A-9T	1205C	T	0.15	1.13		2.07	0.30			
A-10T	1185	T	0.15	0.91		2.32	0.30			
A-10B	1185	B	0.15	0.61		1.04	0.76			
C-3d	TNT	B	0.15	3.50						
C-3fL	TNT	T		1.50		1.80				
C-3iI	1205 C	T	0.15	1.20	0.30	1.20				
C-3hI	1185	T		1.20		2.40				
C-6bB	1205C	B	0.10	0.80		1.60				
C-3gI	TNT	T		1.50		1.80				
DP-2	1205C	T	0.15			1.22	0.30	1.22	3/4"	
S-2	TNT	T				5.24				
S-6	1205C	B	0.15	0.91	0.30	1.25				

incorporated into a Lotus 123 spreadsheet and subjected to extensive analysis. Utilizing Lotus 123, numerous comparisons and correlations between the data have been made. These comparisons have revealed several aspects of the data which may be correlated with stem performance. The comparisons and correlations, as discussed below, include stem-shock arrival times, stem-shock velocities, acceleration, particle velocity and computed stress. In addition, several of these parameters have been scaled for distance from the explosive charge in attempts to further elucidate correlations.

The most direct stem performance data is the arrival time of the stem shock at an accelerometer or velocity gauge as a function of distance from the explosive charge. Gauge arrival times versus gauge distance for all the accelerometer and velocity gauge data obtained are illustrated in Figure 4.4. The data points in Figure 4.4 are sorted according to the type of explosive used in each test. As seen in the figure, the data indicate an expected drop in shock wave velocity with increasing distance from the explosive charge and no discernable dependence upon explosive type. With the exception of one TNT data point (at 2 m and 12 ms) from Test S-2 which bridged quite successfully, all of the shock arrival times fall within a consistent window.

The time of arrival data of Figure 4.4 has been replotted in terms of stem performance in Figure 4.5. Figure 4.5 does show some correlation of gauge arrival times with stem behavior. Generally, the stems believed to have bridged show a later shock arrival time at a given distance than do the stems which failed. The consistency of the shock arrival data is illustrated in Figure 4.5 by the three CORTEX records which have been superimposed upon the data. For Tests C-3e and C-5, the gauge arrival times correspond very closely with the CORTEX arrival times. For Test C-3b, however, the gauge data indicate significant shock arrivals well beyond the distance of measured CORTEX cable crushing. This discrepancy is due, most probably, to the fact that the stem failure resulted in stem motion without the stresses and crushing required to effect a CORTEX measurement.

The shock velocity, as computed from gauge distances and arrival times, is plotted versus distance in Figure 4.6. The points are keyed to indicate stem performance for bridged, possibly bridged, and failed stems. Figure 4.6 indicates a better correlation between shock velocities (distance divided by arrival time) and gauge distance than appears to be the case for arrival time alone. At any given distance the stems which failed gave consistently higher shock velocity than do the stems which held with the being intermediate between these two. Due to the marked drop in average shock velocity with distance as seen in Figure 4.6, the shock velocities were scaled with distance (e.g., scaled velocity equals the measured velocity times distance) and plotted in Figure 4.7. The effect of this scaling is to make the

STEM PERFORMANCE

Fig. 4.4. Plot of stem-shock arrival time versus gauge distance for all instrumented stemming tests.

STEM PERFORMANCE

Figure 4.5. Data of Figure 4.2 plotted to show relations to stem performance and to CORTEX data.

STEM PERFORMANCE

Fig. 4.6. Plot of stem-shock velocity versus gauge distance for all Instrumented stemming tests.

STEM PERFORMANCE

Fig. 4.7. Plot of scaled stem-shock velocity data for all instrumented stemming tests.

correlation between shock velocity and stem performance more apparent. Also the fact that the scaled velocity is independent of distance in Figure 4.7 indicates that true velocity drops inversely with gauge distance.

The peak acceleration as measured with the various accelerometers (PCB piezoelectric and ENDEVCO piezoresistive) are plotted versus gauge distance for all valid gauge records in Figure 4.8. Accelerations as might be obtained by differentiating the velocity are not included. As for the shock velocity data, the accelerations show some correlation with stem performance. A very high acceleration (Mm/s^2) at 2.2 m for Test S-2 was recorded for a successfully bridging stem. In contrast, a very low acceleration (0.27 Mm/s^2) at only 0.9 m in Test C-1 was recorded for a failing stem. Accelerations near the explosive charge are not very indicative of stem performance. Conversely, accelerations at the three to four meter range are very indicative of stem performance. If stem bridging has not progressed to the point that stem accelerations were quite low (if not zero) in this range, the stems always failed. Attempts to scale the peak accelerations with distance did not yield data presentations that were more informative than Figure 4.8.

Peak mass or particle velocities as measured by electromagnetic velocity gauges and as calculated by integrating valid accelerometer records are plotted versus gauge distance in Figure 4.9. In general, the peak velocities display a similar correlation with stem performance as do peak accelerations. Velocities near the explosive charges are not very indicative of eventual stem bridging, but high velocities measured beyond 3.0 m always indicate stem failure. Attempts to scale the velocities with distance did not serve to improve the data presentation nor to elucidate any better correlations.

A final form of the accelerometer and velocity gauge data was evaluated for correlations with stem performance. An effective stem shock stress was calculated by applying the jump condition equation for the conservation of momentum to the gauge data. This equation simply defines the shock wave stress as

$$\sigma = \rho U_p U_s$$

where ρ is the initial density, U_p is the particle velocity, and U_s is the wave velocity. Taking an initial stem density of 1500 kg/m^3 and shock and particle velocities from the gauge data, effective shock stresses were calculated from all valid gauge records. This effective peak stress is plotted versus gauge distance in Figure 4.10. As would be expected, the stresses show a comparable correlation with stem performance as did the particle velocities and shock wave velocities from which they are calculated. Due to the rapid (and expected) drop in peak stress with distance as seen in Figure 4.10, scaling the stress with distance does provide a clearer presentation of the data and correlations as shown in Figure 4.11. As the stress values are a

STEM PERFORMANCE

Fig. 4.8. Plot of stem accelerations versus gauge distance as measured by PCB and Endevco accelerometers.

STEM PERFORMANCE

FIG. 4.9. Plot of peak stem velocities versus gauge distance as obtained from electromagnetic velocity gauges and integrated accelerometer records.

STEM PERFORMANCE

STEM PERFORMANCE

Fig. 4.11. Plot of scaled stem stresses calculated from particle and shock velocity data.

product of both particle and wave velocities, each of which display fair correlation with stem performance, the stress data might be expected to show the best correlation with stem behavior. This improved correlation is apparent in Figure 4.11 where a distinct boundary exists between the data for bridging and failing stems and the data for probably holding stems lies intermediate between them.

The construction details of the stems evaluated in 34 tests are summarized in Table 4.2 in terms of borehole lengths of materials used. The stem geometries have been sorted according to probable stem performance, with designations of stem failure, possibly bridging, and definitely bridging. As seen in Table 4.2, a great variety of stem lengths and compositions were tested within the Anvil Points program. The tabulated data do not clearly reveal any correlations between stem geometry and performance. The stem geometries summarized in Table 4.2 are illustrated schematically in Figures 4.12, 4.13, and 4.14. Again, the stem geometries are sorted according to probable performance.

The majority of the failing stems, as illustrated in Figure 4.12, were less than 3.0-m long and were of a fairly simple design. Only the stem in Test C-3b contained an intermediate grout plug, and since this stem was only 2.4-m long, it was not expected to hold. The 4.5-m-long stem in Test A-8 probably failed since this stem included a long length of Armor-Flex cable shielding around the upper PCB accelerometer cable. Since the Armor-Flex was of soft foam-rubber and was 38 mm in diameter, it could well have precluded the development of the radial stresses needed for successful bridging along the upper 1.5-m length of stem where it was placed.

The stems which have been classified as possibly holding and illustrated in Figure 4.13 are characterized as being nearly 3.0 m long and as having a capping grout plug at the top or collar of the stem. With the exception of Tests C-3iU, CF-1 and S-5, all of these stems had a composite geometry with two types of gravel being utilized. Only the stem in Test DP-1 contained an intermediate grout plug. In general, the only significant difference between the failing stems (Figure 4.12) and the possibly bridging stems (Figure 4.13) is that the possibly bridging stems were either longer or had a more composite construction.

The stems which most probably held are illustrated in Figure 4.14. With the exception of the stem in Test S-2, which was unusually long at 5.3 m, all of the bridging stems had composite constructions and lengths greater than 2.5 m. The probable performance of the stems illustrated in Figures 4.12 to 4.14 is not clearly related to stem design. Certainly stems less than 2.5-m long would be expected to fail despite special construction geometries. Stems greater than 3.0 m with composite constructions could be expected to hold in single-borehole tests. Simple stems might require lengths greater than 4.0 m to have a

STEM GEOMETRY

FAILING STEMS

Fig. 4.12. Geometry of twelve stems which clearly failed to bridge in Anvil Points stem performance tests tests.

STEM GEOMETRY

POSSIBLY BRIDGING STEMS

Fig. 4.13. Geometry of ten stems which possibly bridged (transitional)
In Anvil Points tests.

STEM GEOMETRY

POSITIVE BRIDGING STEMS

Fig. 4.14. Geometry of twelve stems which definitely bridged in Anvil Points stem performance tests.

reasonable probability of holding.

E. Numerical Simulation

Numerous explicit finite difference calculations of stem performance have been conducted. The many calculations done during the course of program development and verification served to indicate the relative importance of the various factors influencing stem behavior. Once the computer program was considered correct in terms of physical models and numerical performance, a few select calculations were performed to illustrate the functioning of the model and the sensitivity of stem performance to various parameters. It is beyond the scope of this report to document all calculations performed or to illustrate every aspect of stem performance dependent upon stem physics or phenomenology.

To illustrate the model and its predicted stem performance sensitivity, a matrix of calculations was performed. This calculational matrix included three distinct stem geometries and several variations in stem material properties or model physics. A "Base Case" calculation was defined to illustrate the applicability of the model to actual Anvil Points field tests and the validity of the most probable physical properties and processes. Employing the simple "one-dimensional" geometry illustrated in Figure 4.15 with 30 stem zones each 0.10 m long, a 3.0 m long gravel stem was simulated for the "Base Case" calculation. The explosive pressure left boundary condition was tailored from a pressure time history for prilled TNT calculated by Los Alamos National Laboratory (Adams, 1986) in a two-dimensional simulation of oil shale fragmentation. As these 2-D calculations did not include escape of the explosive gases into a developing fracture system nor a plastic yield model which would provide greater wellbore expansion and thus explosive pressure drop, this pressure profile probably represents a most severe case. The gravel compaction model involved a constitutive relation for axial stress as a function of stem compression (compaction), u ,

$$u = 1/V_r - 1,$$

where V_r is relative volume, V/V_0 . The "Base Case" compaction curve gave an initial compression modulus of 0.1 GPa and a modulus of 4.1 GPa at $u = 1.0$, corresponding to 50 percent compaction. All of the material properties included in the "Base Case" calculation, as well as in the other calculations discussed here, are tabulated in Table 4.3. The various properties, along with the models to appropriately incorporate them into the calculations, included a cohesive strength and coefficient of internal friction (a Mohr-Coulomb) yield model, a parabolic compaction equation, the coefficient of friction between the stem and the borehole wall, and a variable explosive pressure left boundary condition. In addition, the model included both quadratic and linear (Newtonian) viscosity for numerical

Table 4.3.
Summary of Explicit Finite-Difference Calculations Performed to
Evaluate Stem Behavior Parameters Sensitivities.

Parameter Units	Code	BASE	HEDF	COEF	SHORT	SOFT	DP-1
Variable	CASE	1.0	0.08	NZN=25	CMR=2	DP-2	
Stem Length m	LEN	3.0	3.0	3.0	2.5	3.0	2.9
Zone Length m	ZON	0.1	0.1	0.1	0.1	0.1	0.1
Number of Zones	NZN	30	30	30	25	30	29
Bottom 1st Grout Plug m	BP1						1.4
Top 1st Grout Plug m	TP1						1.7
Bottom 2nd Grout Plug m	BP2						2.7
Top 2nd Grout Plug m	TP2						2.9
Explosive Properties							
Peak Pressure GPa	HEP	4.0	4.0	4.0	4.0	4.0	4.0
Rise Time Factor ms	TREF	0.10	0.10	0.10	0.10	0.10	0.10
Decay Factor	HEDF	0.65	1.00	0.65	0.65	0.65	0.65
Stem Gravel Properties							
Poisson-Dilatation	PDRG	0.3	0.3	0.3	0.3	0.3	0.3
Compact Modulus GPa	ALPHA	0.1	0.1	0.1	0.1	0.1	0.1
Compact Modulus GPa	BETA	2.0	2.0	2.0	2.0	2.0	2.0
Compact Modulus Ratio	CMR	4	4	4	4	2	4
Cohesive Strength GPa	COHG	0.01	0.01	0.01	0.01	0.01	0.01
Internal Friction	TANG	0.60	0.60	0.60	0.60	0.60	0.60
Spall Strength GPa	SPLG	-0.1	-0.01	-0.01	-0.01	-0.01	-0.01
Initial Density Kg/m3	ROG	1500	1500	1500	1500	1500	1500
Grout Plug Properties							
Poisson-Dilatation	PDRP						0.3
Compact Modulus GPa	MODP						5.0
Cohesive Strength GPa	COHP						0.10
Internal Friction	TANP						0.60
Spall Strength GPa	SPLP						-0.01
Init Density Kg/m3	RHOP						2500
Wall Coef Friction	COFP						0.2
Computational Results							
Number of Cycles	CYC	600	600	600	600	600	600
Time Step ms	STEP	0.15	0.15	0.15	0.15	0.15	0.15
Simulated Time ms	TIME	9.00	9.00	9.00	9.00	9.00	9.00
Stem Performance	Trans	Held	Fail	Fail	Trans	Trans	
Reference Figures							
		4.16-	4.19-	4.21-	4.23-	4.25-	4.27-
		4.18	4.20	4.22	4.24	4.26	4.30

Fig. 4.15. One dimensional geometry for the numerical simulation of stem behavior under explosive loading.

stability and, in the case of the latter, hysteretic yielding, and a compaction (u) variation in the cohesive strength term of the yield model. The compaction variation in cohesive strength was designed to simulate the increase in the shear strength of the gravel stem as it went from a loose gravel, at $u = 0$, to a dense compact concrete like material with compaction.

In general the most vivid presentation of stem performance was found to be velocity profiles of a stem at different times during a numerical simulation. By plotting stem mass velocities as a function of location in the stem and by using the Eulerian coordinate of the stem as it was displaced along the wellbore, a clear presentation of both stem velocities and total displacements was obtained. Such velocity profiles for 50 cycle (0.75 ms) increments up to 300 cycles (4.5 ms) of a 600 cycle calculation are shown in Figure 4.16. For the simulation of Figure 4.16, a "Base Case" geometry with a 3.0-m long gravel stem was used. The stem geometry and material properties for the "Base Case" calculation are given in Table 4.3. The velocity profiles show the very large mass velocities and stem displacements occurring near the "explosive charge" and the rapid decay or attenuation of these with both time and distance from the charge. The velocity profiles in Figure 4.16 indicate that stem bridging with zero velocities began at the charge end of the stem at a time between 3.0 and 3.75 ms. At a time of 4.5 ms (the sixth and last profile), only the last 0.5 m of the stem has any discernible velocity with a peak velocity of some 20 m/s at the rear stem free surface. This rear surface reflected wave is just beginning to propagate back into the stem and conceivably could result in the eventual increase of stem velocities to the point of stem failure. The continuation of the "Base Case" calculation to 600 cycles (9.0 ms) does indicate that the "Base Case" 3.0-m gravel stem successfully held or bridged, but only marginally, as was observed in the actual field tests.

The velocity time histories for the "Base Case" calculation are illustrated in Figure 4.17. Velocity-time histories are readily obtained experimentally with the electromagnetic velocity gauges or by the integration of accelerometer records. The calculated velocity time histories may be compared to actual experimental data to establish further the validity of the models and to compare the effects of theoretical and actual variations in stem geometry and material properties. The stem velocity history predicted for a "gauge" at 2.0 m into a gravel stem compares well with actual gauge records in similar stems such as in Test C-3d.

A third means of viewing stem performance as predicted by the finite difference calculations is with compaction profiles such as shown in Figure 4.18 for the "Base Case" calculation. These profiles show the extreme compaction $u = 1.0$, corresponding to 50 percent compaction) near the explosive charge and the rapid drop in compaction along the stem. As the compaction, u , is strongly dependent upon peak stress, the compaction profiles

Fig. 4.16. Plot of stem velocity profiles at 50 cycle (0.75 ms) increments for Base Case 30 zone (3.0 m) gravel stem.

Fig. 4.17. Plot of stem velocity versus time for zones at 1.0, 1.5 and 2.0 m Into Base Case, with 3.0 m long gravel stem.

Fig. 4.18. Plot of stem compaction, MU, at 50 cycle (0.75 ms) increments for Base Case gravel stem.

serve to indicate the distribution of peak axial stress in the stem. The strong compaction gradient of Figure 4.18 is thus indicative of a strong peak stress gradient and indicates that stem wall friction, even when the stem is in motion, serves to rapidly attenuate the axial stresses. Without the combined effects of compaction and wall friction the peak stresses would show very little attenuation along the stem.

The "Base Case", 3.0-m long gravel stem, provides a probably bridging (transitional) case against which calculations employing different geometries properties might be compared. One of the several cases summarized in the calculational matrix of Table 4.3 involved utilizing a pressure time history with a more rapid decay factor than used in the base case. As seen in Figure 4.19, the low case explosive pressures dropped more rapidly than did the "Base Case" pressures. This low case pressure time history is believed to be more representative of real conditions because of the effects of gas flow into fractures, wellbore expansion and explosive gas quenching.

Stem velocity profiles for a 3.0-m long gravel stem driven by the low explosive pressure history of Figure 4.19 are shown in Figure 4.20. As seen in Table 4.3, all other stem properties are identical to the "Base Case". The peak stem velocities are significantly lower for the low explosive pressure case compared to the "Base Case". As for the "Base Case", however, stem bridging began near the explosive charge. Comparable lengths of stem had bridged with zero velocity at comparable time increments. With the more rapid drop in explosive pressures, stem bridging was initiated at a much earlier time and the stem may be considered as definitely bridging.

A third stem scenario is illustrated in Figure 4.21 where the effect of a lower stem wall coefficient of friction is modeled. In comparison to the velocity profiles for the "Base Case" and the low explosive pressure case, reducing the coefficient of friction by 20 % significantly reduces stem bridging. The large material velocity of 40 m/s at the rear (top) end of the stem with stress wave reflection indicates that this stem will most probably fail. No stem bridging (with zero velocity) has occurred near the explosive charge at the time of wave reflection off of the rear end. As seen in Figure 4.21, the reflected wave rapidly increases overall stem velocity to around 30 m/s by the end of the calculation at 9.0 ms. Calculated velocities versus time for two locations (at 1.5 and 2.0 m) into the stem in the low friction case are illustrated in Figure 4.22. Both velocity records show a drop in velocity to nearly zero at 4.5 ms (as occurs when a stem bridges) but a marked increase in velocity to around 30 m/s as the reflected wave propagates back into the stem. This calculation indicates that stem bridging is quite sensitive to the effective coefficient of friction between the stemming material(s) and the wellbore wall. This is not an unexpected result and indicates that field efforts to improve stem performance would be well directed towards increasing this

Fig. 4.19. Plot of explosive pressure versus time simulating TNT pressure time history obtained in LANL 2-D calculations with HEDF=0.65 and for a more rapid pressure decay with HEDF=1.0.

Fig. 4.20. Plot of stem velocity profiles at 50 cycle (0.75 ms) increments for 3.0 m gravel stem with more rapid explosive pressure decay of Fig. 4.19.

Fig. 4.21. Plot of stem velocity profiles at 50 cycle (0.75 ms) increments for 30 zone (3.0 m) gravel stem with a wall coefficient of friction of 0.08.

Fig. 4.22. Plot of stem velocity versus time for zones at 1.5 and 2.0 m into a 3.0 m long gravel stem with a wall coefficient of friction of 0.08.

coefficient of friction. This could be accomplished by using hard angular stemming materials and/or by drilling a rougher wellbore wall.

The effect of using a shorter stem is illustrated in Figures 4.23 and 4.24. Stem velocity profiles for a 2.5-m gravel stem with the base case material properties and explosive pressure loading are shown in Figure 4.23. Due to the shorter stem length, stress wave reflection from the rear of the stem occurs much earlier and before much bridging has occurred towards the charge. In Figure 4.23 it may be seen that nearly all of the stem has attained a fairly uniform velocity of 40 m/s by the end of the calculation at 9.0 ms. This stem is clearly failing to bridge and has already rifled 0.25 m up the hole at 9.0 ms.

Stem velocity time histories at three locations within the 2.5-m gravel stem are shown in Figure 4.24. Comparison of these velocity histories with those for the "Base Case" in Figure 4.17 shows the strong effect of the earlier rear surface wave reflection. While the velocity histories at 1.0, 1.5 and 2.0 m for the two cases are identical up to about 3.0 ms, the short stem case gives a late time increase in stem velocity to around 40 m/s. The fact that all three gage locations have the same 40 m/s velocity after a time of about 5.0 ms attests to stem failure.

One of the many parameters investigated for its influence upon stem performance was the modulus of the compacted stem material. Due to the severe compaction that stemming gravel would undergo during the dynamic loading by explosive detonation, the stem material model included the provision for markedly increasing stem modulus as a function of the peak compaction, or u , of each zone of the modeled stem. As indicated in Table 4.3, a "Base Case" compacted modulus was taken to be 4.0 times greater than the slope of the stress versus compaction curve at peak compaction (see Figure 4.30). As revealed in Figures 4.16, 4.17, and 4.21, this relatively high modulus served to give the entire stem a fairly uniform velocity. That is, the stem tended to move as a rigid plug. A calculation identical to the "Base Case" calculation in all respects except that the compacted modulus was defined to be only 2.0 times the peak compaction curve modulus, as indicated in Table 4.3, illustrates the importance of compacted modulus upon stem performance. Both the stem velocity profiles shown in Figure 4.25 and the velocity histories shown in Figure 4.26 indicate that a lower modulus (softer) stem will show much greater velocity divergence. This divergence is illustrated in Figure 4.25 by the greater variation in velocity along the stem as compared to the "Base Case" as shown in Figure 4.16. The divergence is also well illustrated in Figure 4.26 by the lack of coincidence for the three velocity versus time histories as compared to the "Base Case" velocities of Figure 4.17. While the lower modulus case gave a higher rear surface reflection velocity, the stem appeared to have about the same bridging potential as the "Base Case" stem.

Fig. 4.23. PLOT of stem velocity profiles at 50 cycle (0.75 ms) increments for 25 zone (2.5 m) gravel stem with base case material properties.

Fig. 4.24. Plot of stem velocities versus time for zones at 1.0, 1.5 and 2.5 m long gravel stem with base case properties.

Fig. 4.25. Plot of stem velocity profiles at 50 cycle (0.75 ms) increments for 30 zone (3.0 m) gravel stem with a compacted modulus ratio of 2.0 rather than the base case value of 4.0.

Fig. 4.26. Plot of stem velocities versus time for zones at 1.0 and 2.0 m into a 3.0 m long gravel stem with a compacted modulus

As the use of an intermediate grout plug in the Anvil Points tests appeared to give improved stem performance, several calculations were performed to study and evaluate the effects and benefits of intermediate grout plugs. A calculation to model as closely as possible the experiments for Tests DP-1 and DP-2 is illustrated in Figures 4.27 through 4.30. As summarized in Table 4.3, this calculation employed the "Base Case" explosive pressure time boundary condition and "Base Case" properties for the gravel portion of the stem. As for Tests DP-1 and DP-2 the total stem length was 2.9 m and an intermediate grout plug was modeled between 1.5 and 1.7 m. The properties for the grout plug, as tabulated in Table 4.3, included a non-compacting material with a linear deformation modulus of 5.0 GPa, a cohesive strength for the Mohr-Coulomb yield criteria of 0.1 GPa and a wall coefficient of friction of 0.2.

The stem velocity profiles for six 0.75 ms time increments for the grout plug case are shown in Figure 4.28. The primary effects of the intermediate grout plug, as may be seen by comparing Figures 4.16 and 4.25, are to reduce the peak velocity and spread out the length of stem with finite velocities. As expected, the velocity profiles for the portion of the stem ahead of the grout plug are reduced due to the reflection of a compressive wave from the plug back into the gravel. The velocity profiles for material behind the plug are noticeably increased, however, with the total velocity impulse being higher in the grout plug case than in the "Base Case".

Velocity time histories for the 1.0, 1.5 and 2.0-m locations in the grout plug case are shown in Figure 4.28. Comparing these histories with those for the solid gravel stem case as shown in Figure 4.17 shows the redistribution of stem velocity as caused by the grout plug. While the grout plug is seen to significantly reduce the velocity impulse of the stem at 1.0 m, the velocity impulse (displacement) seen at both 1.5 and 2.0 m is greater in the grout plug case. By 4.5 ms the velocity at the two closer "gauge" locations has dropped to zero while the gravel beyond the grout plug still has a small velocity. Continuing the grout plug case calculation up to a simulated time of 9.0 ms indicated that the entire stem would drop to zero velocity and that the stem could be considered as probably holding.

The presence of the grout plug is clearly seen in profiles of compaction, u , in Figure 4.29. The grout plug proper, with a linear and non hysteretic deformation modulus, shows very little compaction in the 1.5 to 1.7-m interval of the stem. The gravel portion of the stem, however, shows increased compaction, as compared to the "Base Case" shown in Figure 4.18, ahead of the plug and reduced compaction behind the plug due to the reflection of wave energy off of the plug. Note that the compaction and velocity profiles at early times, before the stem shock has interacted with the grout plug, are identical in the solid gravel and grout plug cases.

Fig. 4.27. Plot of stem velocity profiles at 50 cycle (0.75 ms) increments for 2.9 m stem with 0.3 m grout plug at 1.5 m into stem (DP-1 and DP-2 geometry).

Fig. 4.28. Plot of stem velocity versus time for zones at 1.0, 1.5 and 2.0 m into stem with an Intermediate grout plug (DP-1 and DP-2 geometry).

Fig. 4.29. Plot of stem compaction at 100 cycle (1.50 ms) increments for 2.9 m stem with an Intermediate grout plug (DP-1 and

Fig. 4.30. Plot of axial stress (SIGX) versus compaction (MU) in a 2.9 m stem with an intermediate grout plug, showing characteristic gravel and grout plug behavior.

An important feature of the stem model, the irreversible compaction of gravel stem material, is illustrated in Figure 4.30. The two stress versus compression, σ , curves shown in Figure 4.30 are from the grout plug case illustrated in Figures 4.27 through 4.29 and are for zone 13, within the gravel ahead of the grout plug and zone 14, one of three grout zones. The stress versus compression curve for the gravel zone displays the very low modulus for this material prior to compaction, the expected increase in modulus with compression or compaction and the hysteretic behavior of the material with unloading. The knee or change in slope of the unloading curve near the end of unloading is caused by the onset of reverse yielding of the compacted gravel according to the Mohr-Coulomb yield model incorporated within the calculational model. The compacted modulus ratio, CMR, discussed above in reference to Figures 4.25 and 4.26, defines the slope of the unloading (and reloading curve) for the gravel stem when the stress drops below its peak value. The unloading curve shown for zone 13 in Figure 4.30 is for a CMR of 4.0. A CMR of 2.0 as used for the calculation of Figures 4.25 and 4.26 would give a much flatter unloading curve, lying midway between the loading and unloading curves of zone 13.

The experimental data on stem performance obtained during the course of the oil shale fragmentation programs complemented by the analysis of stem performance with the finite difference model served to indicate those parameters of stem design which are most important for good stem performance. Both the field experimental and numerical modeling results strongly indicate that stem length is a very important if not critical factor. Both experimental and modeling data show that stems much shorter than 3.0 m in a 0.15-m diameter shot hole cannot generally be expected to bridge properly. Conversely, stems much longer than 3.0 m could be expected to bridge consistently provided other stem bridging factors are not adversely compromised. Of the numerous parameters evaluated for their influence upon stem performance, it appears from the numerical modeling results that stem-wall friction is the most important parameter after stem length. Unfortunately, efforts to control and evaluate stem-wall friction in the experimental program were not attempted. The numerical results strongly indicate that optimum performance stemming gravel should be composed of hard angular material. Crushed carbonate and quartzitic rocks would thus be preferred while rounded stream gravels and drill cuttings would be less desirable.

The use and evaluation of an intermediate grout plug to improve stem performance in numerous Anvil Points tests (eg. Tests DP-1 and DP-2) indicated that a grout plug through a combination of impedance mismatch wave reflection and possible wellbore locking of the plug proper could significantly improve stem performance. While the numerical results did show that an intermediate grout plug would initiate some stem bridging immediately before the plug (due to stress wave reflection) this

bridging did not necessarily indicate nor contribute to long-term stem holding. In general, the numerical results indicate that a stem with an intermediate grout plug would be expected to perform about the same as a simple gravel stem of comparable length. Possibly, the better performance of stems with intermediate grout plugs in the anvil points test was due to a locking of the grout plugs into the wellbores with an effective coefficient of friction much higher than the 0.2 employed for the grout plugs in the numerical simulations.

F. Summary of Stem Performance

The data obtained with the accelerometers and velocity gauges placed within stem columns in explosive field tests provided much useful information on stem performance. These data were valuable both for interpreting stem performance and for developing and validating the explicit finite-difference stem model. Data obtained near the explosive charges is not a good predictor of eventual stem behavior. Rather, stem bridging appears to be dependent upon phenomena occurring in the two to three meter range of stems in 0.15-m diameter shot holes. If a stem begins to bridge in this interval it will lock into the borehole wall and hold indefinitely. The field tests indicated that efforts to keep explosive gases from entering them stem and thereby reducing effective stem-wall friction improved stem performance. The field tests also indicated that intermediate grout plugs improved stem bridging, either by the effects of shock wave reflection off of the plugs and/or the higher effective friction of the plugs in the shot holes.

The explicit finite-difference model of stem behavior was useful in establishing the relative effect and importance of various stem features and properties upon bridging performance. Model results indicate that stem-wall friction is probably the most important parameter as might be expected. The impedance mismatch aspect of grout plug contributions to stem performance was not found to be particularly important in the numerical simulations. Stem performance was only moderately influenced by changes in stem material yield strength and compaction behavior models. Both the field and modeling results suggest that efforts to optimize stem performance should be focused upon maximizing stem-wall friction. This could be accomplished with the use of hard angular gravels, gas impermeable plugs to preclude explosive gas penetration into the stem and grout plugs which bond or lock into the shothole wall. Even with efforts to maximize wall friction, a minimum stem length is required. For the 0.15 m in diameter shot holes evaluated in both the field and models, stem lengths approaching 3.0 m (20 times hole diameter) are required for consistent stem bridging in single hole tests. The shock wave interactions between holes in multiple hole blasts appears to seriously degrade stem performance as discussed in the following section on explosives engineering. Data to establish stem design requirements for multiple hole blasts was not obtained within the scope of the field program.

CHAPTER 5 - EXPLOSIVE ENGINEERING PROBLEMS

A. Introduction.

Several parameters were thought to be of importance in VMIS blasting prior to the start of the testing programs at the Colony and Anvil Points Mines. These were incorporated into the program for evaluation and for the most part were not found to be of particular importance.

Geological factors were one such parameter. Extreme care was taken to geologically map the area surrounding the test areas. Any abnormal features were noted (such as open joints, vugs, etc.) and the location of joints and bedding planes recorded. The presence of these features were taken into account in the data analysis and post-test area examination. As described earlier, with only a few exceptions, geological factors played a minor role in influencing the fragmentation results. Explosive type is another factor originally felt to be of importance whose effects proved to be buried within the scatter of the data obtained.

The most important factors affecting success or failure of forming VMIS retorts were not anticipated before the program was planned and, although some testing was added to the program under the Special-Series tests, most remain unanswered at this time. Some of these problems could be classified under the broad category of "explosive engineering problems" and include delayed blasting cap performance, explosive dead-pressing, and explosive displacement. The explosive displacement problem is associated with poor stem performance which has already been addressed in Chapter 4. In addition, there were design defects in the program planning--mainly in the form of a requirement for a direction reversal in the throw of the broken shale. In the following sections, the main obstacles to successful VMIS retort formation will be discussed in detail. Some have been introduced already as part of preceding discussions but will be reiterated here. These include (a) blasting cap performance, (b) stemming performance, (c) explosive performance, and (d) reversal of throw direction.

B. Blasting Cap Performance

At the beginning of the test program, the precision required for a blasting cap to be successfully utilized in VMIS blasting was not known. The time delays needed between charges in holes at the same level and between different levels was thought to be smaller than that required for typical quarry blasting where 2.0 and 4.0 ms per meter of burden is used. Delay caps were needed because of the requirement to detonate the more deeply buried charges at a later time than those nearer the surface. If detonator wire cutoffs were to be avoided, it was necessary to use delayed caps which would receive their initiation current well before any explosive in the shot sequence detonated. It was

felt that delays as short as 10 ms might be required. Samples of caps from Hercules, DuPont, and Atlas were purchased and tested on the mine floor (after reaching an equilibrium temperature). Results from those tests are presented in Table 5.1. Notice that both conventional DuPont caps and the new technology (NT) DuPont caps were tested. The difference between the two types was the much more rigid quality control applied to the NT caps. Therefore, a more precise operation was expected. However, expected performance determined for all the caps tested was inadequate and the practice of using multiple caps per charge was used to reduce scatter in subsequent testing, that is, two to four caps of the same type and lot were used per charge. This procedure appeared to be partially successful as demonstrated in Table 5.2 which shows detonation times obtained for Test C-2, a 16 charge, two level test which used three Period 3 (75 ms) Atlas caps per charge. As can be seen from the table, the firing times of the caps were not perfect but the precision was improved compared to single cap performance. Of the six holes that were to detonate 10 ms after Holes 6 and 10 (at zero), the earliest firing time was 4.4 ms and the latest was 15.6 ms, both after Hole 6. In a similar fashion, of the two-level charges that were designed to fire 10 ms after Holes 7 and 11, the earliest was 1.6 ms and the latest was 15.8 ms, both after Hole 7. This performance was disappointing, but since the shot had performed well--that is, good rock breakage and good depth-of-pull were achieved--the same concept was used for Test C-5 (the next test). However, DuPont NT caps with four caps per charge were used instead of the Atlas caps.

Test C-5 was a three-level test with 24 charges. The even numbered blastholes had two charges per hole separated by 3.5 m of stem. The resulting charge detonation times are presented in Table 5.3. Note that firing times were not obtained for four charges: 3, 4U, 9U, and 12U. The CORTEX sensor cables for these charges were cut at the surface at about 80 ms. Based upon post-test examination, Hole 3 did not fire. Note the excellent cap performance achieved for the first two levels (lower deck of the even and odd charge numbers). The largest variance was only 2.4 ms. Of the three charges in the last level to fire for which data was obtained, however, the performance was totally unacceptable. The difference in firing for the three caps was 11 ms out of a required delay of 10 ms from Holes 6U and 10U. In this particular test, an emulsified ANFO-based slurry explosive was used for the first time. It is speculated that the explosives in the third level experienced large dynamic pressures from the detonation of charges in surrounding holes and within the same borehole. Since the explosive was a slurry rather than a prill, this pressure was probably transmitted directly to the caps which adversely affected the precision. (According to the manufacturers, when a delay cap is subjected to pressure the cap can rupture and not detonate at all or detonate late; this may have occurred in Hole 3. Or, if the pressure does not rupture the cap shell, it can fire early.)

Table 5.1.
Summary of Blasting Cap Delays for DuPont, Hercules, and
Atlas Blasting Caps.

Caps	Nominal delay (ms)	Average delay (ms)	Standard Deviation (ms)	S/t x100	Caps tested
Std. Dupont	50	53.1	4.8	9.0	6
NT Dupont	75	65.0	4.0	6.1	23
Hercules	50	46.7	8.0	17.0	6
Atlas before Test C-2	50	56.2	3.9	6.9	14
after Test C-2	50	62.2	6.8	10.9	15
Period 0	0	3.7	0.7	20.2	10
Period 3	75	70.4	1.6	2.3	11
Period 4	100	104.9	9.4	9.0	8

Table 5.2.
Comparative Data for the Actual Delay Firing Times for Test C-2.

Charge	Firing Times (ms) Design	Actual	Actual Times (ms) Average	Variance	Delay (ms)
6	0	4.9	3.9	1.0	0
10	0	2.9			
2	10	15.3	14.2	5.6	10.3
4		20.4			
8		9.3			
12		13.2			
14		15.0			
16		12.0			
7	20	28.3	27.5	0.7	23.7
11		26.8			
1	30	37.4	36.9	7.1	33.0
3		34.2			
5		29.9			
9		41.6			
13		34.1			
15		44.1			

Table 5.3.
Comparative Data for the Actual Delay Firing Times for Test C-5.

Charge	Firing Times (ms)		Actual Times (ms)		Delay (ms)
	Design	Actual	Average	Variance	
6L	0	80.1	79.8	0.4	0.4
10L		79.4			
2L	10	92.5	92.1	12.7	12.7
4L		90.8			
8L		93.5			
12L		92.1			
14L		92.5			
16L		91.2			
7	20	99.4	101.0	2.4	21.6
11		102.5			
1	30	112.9	110.0	1.5	31.3
3		-----			
5		109.8			
9		110.3			
13		109.5			
15		110.8			
6U	40	113.0	111.8	1.6	32.4
10U	50	110.5	132.7	20.2	53.3
2U		127.0			
4U		-----			
8U		-----			
12U		-----			
14U		138.0			
16U		133.0			

U - upper charge column
L - lower charge column

If VMIS fragmentation is to become a reality, and if proper delay times are found to be similar to those used in the Anvil Points program, then better delay caps must be developed. These caps, if of the conventional type, must be more accurate than presently available and must also be made to withstand the high pressures resulting from the charges detonating in the same and adjacent holes. Some cap manufacturers are currently working on electronic caps which are capable of being very accurate, but they also must be able to withstand large shock pressures. Perhaps a cap could be devised that could be armed from the shock wave produced by the detonation of nearest charges. The data from the Anvil Points program has shown that the shock wave is very predictable both in time and magnitude.

C. Stemming and Explosive Performance

At the start of the testing program, it was felt that fragmentation by explosives was due to a combination of the early fracture development by stress wave propagation through the rock mass and, later, gas action within these fractures. The relative roles of the stress waves versus the gas pressure has been a controversial subject for the past 25 years in all types of rock fracture by explosives. Evidence has been very convincing in situations involving oil and gas stimulation [Young, et.al., 1986] and quarry blasting that proper stemming greatly enhances rock breakage. For this reason, considerable attention was given to the determination of optimum stems. As mentioned in Chapter 4, a stem recipe was selected and tested in the single-borehole and Special-Series tests. This recipe consisted of a small layer of clay near the charge (to keep gases from entering the main stem) followed by a length of 9.5 mm dirty angular gravel and an additional length of 19-mm angular gravel. Also, an intermediate grout plug between the two gravel columns was judged to be beneficial. It was felt that this combination would prevent gas penetration into the stem and would result in a bridging action whereby the gravel stem would increase in diameter, gouge into the borehole wall, and remain in place. Exactly how long the stem was required to hold was never determined, but it was postulated that if the stem stayed in place until the rock around it was broken and removed the gases generated by the explosive detonation would achieve their maximum effectiveness.

Throughout the test series the effect of a good stem was never fully demonstrated. In one particular test, Test C-3d, the stem (3.6 m in length) completely bridged and CORTEX and accelerometer instrumentation revealed that no movement of the stem occurred at the free face. The shot was very deeply buried, and although not expected to crater, the shot actually mounded to a height of approximately 2 m. The crater was not mucked but the general consensus was that the stem column bridged and resulted in optimum breakage.

The selection of proper stemming length is almost a "Catch 22" situation. That is, the stem must be of a certain minimum

length to insure bridging; if too long, the charges will be too deeply buried and poor breakage (if any) will result. Hence, either a too short or a too long stem length will run the risk of insufficient free face being formed and the next lower level of charges will not perform properly.

In the later stages of the program, four tests were conducted to determine the true effect of stemming on fragmentation. Two tests, Tests 83C and 83D, were conducted by Los Alamos - one stemmed and one unstemmed. Two tests, Tests SB1 and SB2, were also conducted by Sandia - also one stemmed and one unstemmed. The performance of Test 83D was judged to be better than Test 83C, that is, Test 83D pulled deeper - the bottom of the charge - and removed a larger volume (on a percentage basis) of rock than did Test 83C. Its performance was, however, not greatly different from Test 83C. On the other hand, Tests SB1 and SB2 were identical in all respects - except for stemming. The results for Tests SB1 and SB2 were nearly identical in volume removed and crater size, indicating no benefit at all from stemming. Unfortunately, both Tests SB1 and SB2 performed far poorer than any other single-borehole tests within the program; and as discussed in Chapter 3, the two tests did not match the data from the other tests. Hence, the answer to the importance of gas action on fragmentation still remains largely unanswered, although the crater blasting configuration results to date indicate stemming between the charge and the face being blasted is not nearly important as believed at the beginning of the program.

The stemming between charges in a common borehole, on the other hand, was shown to be extremely important. The design for this intermediate stem was changed slightly from the stem design used between the charge and the free face. For the intermediate stem a grout plug (poured in place) was used at the mid-point of the column. The plug was more dense than gravel and reflected part of the shock pressure toward the charge, assisting the bridging action (as discussed in Chapter 4). This stem appeared to be very effective in single-borehole configurations. The purpose of the intermediate stem was to protect the second charge in the borehole from deadpressing and hence, becoming desensitized. Every single borehole test conducted with multiple charges per hole was successful. That is, the second charge always detonated at the proper time and with the proper detonation velocity. In addition, the instrumentation used in many of the single-borehole tests indicated little or no shock loading of the charges would occur prior to initiation of the delayed explosive column.

Such was not the case for the multiple-borehole tests. As stated earlier, problems began to develop with Test C-5, the first multiple-borehole multiple-charge per hole test. In the upper level, three of the eight holes apparently did not perform properly. Delayed charge performance was even worse in the D-series tests; Tests D-2 and D-3 were particularly bad. In Test D-2 the third level of explosive slurry (IRECO 1205C) failed to

detonate. The test site was accessible because a rock ledge remained and undetonated slurry was still present in the borehole. Upon an examination of the explosive under a microscope, it was evident that the glass beads utilized to sensitize the slurry were crushed. Apparently the slurry had been subjected to very large compressive stress as a result of the intermediate stem failing to perform its function.

Even greater evidence of intermediate stem failure was found in Test D-3. In this test, the third level was TNT rather than slurry because deadpressing was less of a problem with the TNT. Post-test inspection of the test site revealed that, again, an unbroken rock ledge remained. In this case, it was observed that when the first charge in the hole detonated the intermediate stem was displaced downward. This stem motion resulted in a compression of the TNT charge to nearly one and one-half times its original density. Fortunately the delay caps were in the end of the explosive away from earlier detonated charges, and even with that degree of compaction, the TNT detonated. Upon this second detonation, the intermediate stem was pushed back up the borehole approximately 0.6 m. The third level charges detonated at the proper times and with the proper velocities. The fourth level of charges in Test D-3 experienced a similar history; since the explosive was a slurry instead of pelletized TNT, half of the charges failed to detonate and two of the remaining four detonated low order.

This piston-like behavior of the intermediate stems was obviously caused by interactive effects of explosives going off in boreholes surrounding the intermediate stem since they performed in a perfectly acceptable manner when tested in a single-borehole situation. Perhaps the explosives detonating in adjacent boreholes caused the diameter of the intermediate stem to change slightly which, when the borehole resumed its original dimension, permitted the stem to move freely upward and downward in the borehole from the gas action. This effect is comparable to a reduced coefficient of friction between the stem column and the borehole wall as discussed in Chapter 4. It is imperative that problems with intermediate stemming be resolved, especially in designs in which the stem experiences shock loading from adjacent holes.

D. Reversal of Throw Direction

A major disappointment in the Anvil Points Oil Shale Program was the failure of the three large D-series tests to completely fragment the 18-m interval between the upper and lower rooms. In all three tests, an unbroken shale ledge remained.

As stated previously, in Test D-3 the third level charges was displaced downward an average of one meter, but some moved as much as two meters. In addition, CORTEX and pin-probe data indicated that four of the eight charges in the fourth level failed to detonate and two of the remaining four detonated low

order. This was attributed to poor performance of the intermediate stem which failed to hold when subjected to a shock wave from charges in adjacent or common boreholes. This resulted in large stem movements which caused displacement and compaction of the third-level TNT charges and deadpressing of the slurry used in the fourth level. In view of this, it is amazing that the bottom two levels resulted in as much fragmentation as occurred. Perhaps this was due to the interval being very lean shale.

The results for Test D-2 were similar. Large motion of the intermediate stems in the third level was observed. This caused the third-level slurry explosives to be dead-pressed, preventing detonation. Therefore, it appeared that Test D-1 was the only valid test of the retort design. Test D-1 was a three-level, 24-charge configuration with the two upper levels moving rock upward and the lower level moving rock downward. Initially, the reasons postulated for the failure of Test D-1 were the existence of a very rich layer of shale (75 gal/ton) just above the remaining rock ledge and an inadequate reversal plane design. In Test D-1, the uppermost level detonated first, then the second and third levels were detonated after a 20 ms delay each. Within each level there was a 10 ms delay between the center and edge holes. Figure 5.1 shows the relationship of four boreholes and charges through one section of the retort. Figure 5.2 shows the breakage resulting from the detonation of the charges. The very rich E and F beds are located just above the remaining shale ledge. Figure 5.3 shows the superposition of the breakage from Test D-1 and the breakage resulting from Test RP-1, which was a test to examine the breakage resulting from two levels of charges that were to fragment an interval of shale by moving one level up and the second level down. There were six boreholes in Test RP-1, three contained explosives to fragment the shale upward and three contained explosives to fragment the shale downward. The geometry was chosen to be a representation of the lower two levels of a multiple-level test such as Test D-1 and was intended to simulate one-quarter of a sixteen-borehole test. The charges in Holes 1 and 2 were undelayed, the explosives in Holes 3 and 4 were delayed 15 ms, and in holes 5 and 6, the delay was 30 ms. The rich bed interval in RP-1 was located nearly at the surface of the bottom room. By comparing Tests D-1 and RP-1, a similarity is noted in the breakage pattern and the remaining shale ledge. Since the rich beds were at different locations relative to the charges for the two tests, and since there was no difficulty encountered in breaking these beds in Tests RP-1, it was concluded that these rich layers were not the cause for failure of Tests D-1 (or Test RP-1). This implies that the reversal plane design used in Test RP-1 and throughout the D-series was faulty. In Test RP-1, it was apparent that charges in the upper level of the retort were expected to break the shale between the free surface and the upper one-fourth length of the charge. The lower or reversal plane charges, on the other hand, were expected to break the same interval (on the lower surface) and the shale surrounding the bottom portion of the upper

Fig. 5.1. Vertical cross section of four holes for a sixteen - hole pattern for Test D - 1.

Fig. 5.2. Vertical cross section of the post test rubblization pattern of Test D - 1.

Fig. 5.3. Vertical cross section of the post test patterns for Test D - 1 and RP - 1.

charges. In retrospect, it appears the explosive was expected to fragment more shale than was practicable. The problem of breakage at the reversal plane is not easily solved and certainly more thought must be given to a solution. If the reversal plane can be eliminated from the retort design, a solution is not necessary.

E. Conclusions

The failure to successfully fragment a rubble bed for VMIS retorting during the Colony and Anvil Points program can be attributed to three factors, namely blasting cap performance, stemming and explosive performance, and a reversal plane requirement. If the stemming problem could be solved (especially the intermediate stem), then the blasting and explosive performance problems could be eliminated to a large extent. This is especially true if the current trend toward electronic detonators continues. Furthermore, if the requirement for a reversal plane could be eliminated, then successful VMIS formation could be a reality. This was demonstrated by Test C-5 where three levels were fragmented quite successfully. This design would essentially involve having a working room above the intended retort from which all drilling and loading would take place. An unbroken rock ledge would intentionally be left from which to work the subsequent burning operations. In conclusion, therefore, the one remaining problem deserving the most attention which would lead to VMIS retorting would have to be further researched in intermediate stem design.

CHAPTER 6 - SUMMARY

Each chapter of this report concludes with a summary and the reader can consult these sections for details. This chapter will only provide the most important conclusions from the research program.

A large amount of data and experience was gained from the fragmentation of oil shale the research programs at the Colony and Anvil Points Mines during the 1978-1983 time period. The data obtained includes basic cratering knowledge such as optimum and critical DOB, typical fragment size distributions for simple and complex blast designs, expected rock mass motion for several commercial blasting agents, delay blasting cap performance, behavior characteristics of several stem column designs, and information on the influence of timing and spacing of the explosive charges on fragmentation results in oil shale. Most importantly, several problem areas were identified with regard to construction of large, single blast VMIS retorts. These problems are formidable and considerable research will be required to solve them.

Only part of the data has been analyzed and documented in this report. The subjects presented represent the authors views about some of the most important results from the research. The mini-retort tests of the Consortium Program (Tests D-1, D-2, and D-3, and Tests C-1, C-2, and C-5) were very informative. These tests were analyzed in limited detail and are reported in this document. There is, however, much more data and a report devoted to the analysis and interpretation of the results of these tests should be written.

Oil shale is a very inhomogeneous rock in terms of strength, density, kerogen content, discontinuities, and layering. Therefore, results from previous fragmentation research on other rock types were not extremely helpful in predicting the outcome of fragmentation blasting in oil shale. As a consequence, some aspects of the programs were directed at providing fundamental cratering and fragmentation data, such as crater sizes, fragment size distribution, charge DOB influences, and effects of charge-timing and spacing. Chapter 3 provides such fragmentation data from two oil shale mines on shales ranging in kerogen content from 5 to 80 gallons/ton in kerogen content. The scaling rules based on dimensional analysis, that is, dividing the DOB by the cube root of the explosive weight or the crater volume by the explosive weight, does not appear to adequately describe the results obtained. Three main conclusions were obtained from the analysis of the fragmentation and crater data (1) the ranges of DOB used in the tests were not large enough to completely characterize the results, (2) equal lengths of stem and explosive in a borehole may not be the optimum loading for the best fragmentation, and (3) within the limits investigated, deeper burial of the explosive does not greatly affect the crater

volume, however, the average particle size increases significantly with burial depth.

Instrumentation of the tests with CORTEX and ionization pins produced valuable data for evaluating cap and the blasting agent performance. From these measurements, sympathetic or pre-detonation of an explosive resulting from the shock wave produced by neighboring explosive charges was not observed as feared. However, deadpressing, poor cap performance, and low-order detonation were observed and influenced the fragmentation results.

Rock mass motion instrumentation provided measurements of mixed value. The accelerometer and velocity gauges had inadequate range and frequency response to produce meaningful measurements near the explosive charge, but provided accurate results at ranges beyond three meters. The computer-controlled data acquisition system performed very well, although the analog bandwidth of the transient digitizers and signal conditioners were marginal for extremely fast signals. Stress measurements obtained at the Colony Mine were good, but the interpretation proved difficult because the directional characteristics and calibration factors of the gauges were not accurately known. Despite these difficulties, calculated and measured values for acceleration, velocity, and stress agreed surprisingly well. This inspired confidence both in the simulation models developed by Los Alamos and Science Applications, Inc. and in the field measurements. The most important result from the rock motion data was the low particle velocities (2 to 6 m/s) measured at a depth of about three meters in a multiple-borehole, multiple-charge array. These low values suggest that timing delays between charges need to be at least 50 ms to adequately produce a free face for the next lower level of explosives, otherwise the burden is much too large for good fragmentation. These low velocities were a contributing factor in the failure of the mini-retort fragmentation tests.

Stemming performance in a complex blast array was determined to be very important for good fragmentation of the shale, especially the stem column separating the charges in the same borehole. Chapter 4 details the analysis of the stem behavior data.

The failure to produce an adequate rubble bed for in situ retorting was probably due to three main factors, poor blasting cap performance, poor stemming performance causing poor explosive performance, and the reversal plane requirement. Insufficient delay times and wide variations in kerogen content were secondary factors contributing to degraded blasting performance.

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