10/10/17/88 Q8 (R)

DB# 0581-9

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Printed July 1988

In Situ Experiments of Geothermal Well Stimulation Using Gas Fracturing Technology

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Prepared by
Sandia National Laboratories
Albuquerque, New Mexico 87185 and Livermore, California 94550
for the United States Department of Energy
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Printed in the United States of America
Available from
National Technical Information Service
U.S. Department of Commerce
5285 Port Royal Road
Springfield, VA 22161

NTIS price codes
Printed copy: A05
Microfiche copy: A01

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SAND87-2241
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SAND--87-2241

DE89 001050

IN SITU EXPERIMENTS OF GEOTHERMAL WELL
STIMULATION USING GAS FRACTURING TECHNOLOGY

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ABSTRACT

The results of an experimental study of gas fracturing technology for geothermal well stimulation demonstrated that multiple fractures could be created to link water-filled boreholes with existing fractures. The resulting fracture network and fracture interconnections were characterized by mineback as well as flow tests. Commercial oil field fracturing tools were used successfully in these experiments. Simple scaling laws for gas fracturing and a brief discussion of the application of this technique to actual geothermal well stimulation are presented.

ACKNOWLEDGMENTS

The efforts of a number of individuals in organizations at Sandia National Laboratories and from REECO and EG&G contributed significantly to the experiment fielding effort: John Talbutt and Dave Amsler, project engineering and coordination; Don Werner, Allen Church, Elmer Smith and Harold Roberts, AF&F; Erv Henshaw, transmissivity; Ken Melloy and Carl Crowe, data acquisition; Mike O'Brien, geological support; Cris Salinas and Larry Arnold, photography; and Carl Davis, borehole TV logging. Henry Mohaupt led the Servo-Dynamics effort in the planning and execution of the test series. The field engineers from Servo-Dynamics were Mike Berg, Roy Green, and Gary Loman.

This work was supported by the U.S. Department of Energy at Sandia National Laboratories under Contract DE-AC04-76DP00789.

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INTRODUCTION

Drilling costs are often a substantial fraction of the overall cost of a geothermal well. It may, therefore, be more economical to stimulate an underproductive well instead of drilling a new one. There have been previous attempts in stimulating geothermal wells using hydraulic fracturing.¹ These attempts were largely unsuccessful mainly because the fractures created by hydraulic stimulation, as dictated by the in situ stress, ran parallel to existing fractures rather than intersecting them. The present study was undertaken, with funding from the U.S. Department of Energy (DOE), to demonstrate the application of gas fracturing technology to geothermal well stimulation.

Gas fracturing is a technique to rapidly pressurize a borehole with a controlled, propellant burn in order to initiate multiple, radial fractures. Generally speaking, for a specific formation the fracture patterns depend on the rate of pressurization. At slow rates, hydraulic fractures are produced; at moderate rates, multiple fractures can be created; at fast rates, the borehole is plasticly deformed (crushed), resulting in lower permeability. Details of this technique are found in Appendix A and References 2 and 3. A propellant tool was previously developed at Sandia National Laboratories (SNL) for stimulating gas wells. In situ tests showed that multiple, radial fractures could be created in open holes for a wide range of pressure loading rates.^{4,5} The geothermal application represents an extension of this technology from gas-filled to water-filled boreholes. In water-filled holes the burning characteristics of propellants and the pressure coupling between the propellant and the borehole wall are likely to be

different from gas filled holes. In situ tests are desirable to characterize the effect of these differences.

With an optimum pressurization rate, a controlled propellant burn can produce multiple fractures radiating from a wellbore with fracture length up to 10 m. Radial fractures in a non-hydraulic-fracture direction are most desirable for geothermal well stimulation since they have a higher probability of intersecting existing production fractures. While no detailed mappings are available to determine fracture spacings in production fields, field insights suggest that typical production fracture spacings should be in the range of one to tens of meters. At The Geysers, production wells are known to be drilled as close as 3 m from nonproductive wells. Multiple, radial fractures will certainly be useful for fields where fracture spacing falls within the 1- to 10-m range. Computer modeling by Glowka showed that radial fractures are superior to hydraulic fractures in correcting near wellbore damages due to drilling or silicate precipitation.⁶

The main objective of the present test series was to demonstrate the capability of the gas fracturing technique to produce multiple fractures in water-filled boreholes. Because this technology is gaining acceptance as a well stimulation technique for oil and gas wells, we were interested in evaluating the performance of commercial oil field fracturing tools for geothermal well stimulation. Therefore, Servo-Dynamics was contracted as a joint investigator with SNL for the present study. Servo-Dynamics is currently the most experienced service company in the field, carrying out several hundred fracturing jobs per year in gas and oil wells worldwide.

The experiments were conducted in the G-Tunnel complex, Rainier mesa, at DOE's Nevada Test Site (NTS).⁵ G-Tunnel is a tunnel system driven more than 2 km into volcanic tuffs at a

depth of about 425 m (1400 ft). It is ideal for fracturing experiments because it provides an in situ medium with known stress and boundary conditions and allows detailed examination of the fracture patterns through mineback.

In a previous series of experiments in horizontal boreholes, we demonstrated the feasibility of producing multiple fractures in water-filled boreholes.^{4,7} However, because the tools were relatively short, there were large end effects. Furthermore, because the test bed consisted of strata of substantially different properties, multiple fractures were observed only in a portion of the test zone. Posttest analysis suggested that in some cases the fracture patterns observed may be the result of gravity separation of water and reaction gas during the tests.⁸ Therefore, the present series of experiments was carried out in vertical boreholes using longer tools.

EXPERIMENTAL DESIGN AND PROCEDURE

The current series of tests was conducted at the CFE-2 drift in peralkaline ash-fall tuff of Tunnel Bed 5 in the Indian Trail Formation, which is approximately 30 m (100 ft) thick at the test location. Average properties of the peralkaline tuffs obtained from core samples taken from the test bed are shown in Table 1; core logs and detailed test results are given in Appendix B. The tuffs are soft, low-modulus, low-permeability, and high-porosity rocks and, as such, are well suited to mineback experiments since they allow for easy excavation with a continuous mining machine. In situ stresses have been measured in the neighboring CFE-1 drift, yielding an overburden stress of about 7 MPa (1000 psi) and horizontal stresses of about 3.4 MPa (500 psi) and 5.5 MPa (800 psi). The maximum horizontal stress (hydraulic-fracture) direction is S30°W.

Table 1

Typical Peralkaline Ash-Fall Tuff Properties

Bulk density	$1.8 \pm 0.1 \text{ g/cm}^3$
Porosity	$40 \pm 5\%$
Water saturation	$90 \pm 5\%$
Young's modulus	2-3 GPa; $0.3-0.4 \times 10^6 \text{ psi}$
Poisson's ratio	0.15
Compressive strength (unconfined)	14 MPa; $2 \times 10^3 \text{ psi}$
Tensile strength (unconfined)	0.7 MPa; 100 psi
Permeability	0.1-2 md
Compressional wave velocity	1500-2500 m/s
Shear wave velocity	900-1500 m/s

A 20-ft extension of the CFE-2 drift in the hydraulic-fracture direction was made to accommodate the test series. Five vertical holes, numbered 1 through 5, were cored into the test drift. The layout of the hole pattern is shown in Figure 1. Because we were interested in observing fracture intersections and carrying out hole-to-hole pumping tests to determine flow characteristics of the fractures, the borehole patterns were designed to facilitate the formation of fracture intersections.

Two holes (2 and 5) were placed along the hydraulic-fracture direction, 4.5 m (15 ft) apart. Two more holes were located between holes 2 and 5, on either side of the line connecting the first two, 2.4 m (8 ft) apart. A practice hole, hole 1, was located 3.0 m (10 ft) from hole 2, down the main drift direction (N42°W). The practice hole (1) was 17.4 m (57 ft) total depth (TD); the four test holes were each 12.3 m (42 ft) TD. Vertical displacement lessens the possible influence of the practice shot in hole 1 on the rest of the

Figure 1. Test Layout

tests. All except hole 2 were open holes 9.60 cm (3.78 in.) in diameter. Hole 2 was cased with N80 casing 10.1 cm (4.0 in.) I.D., 12.7 cm (5.0 in.) O.D. and perforated, with 1.5-cm (0.6-in.) diameter perforations, 8 perf/ft with 90° phasing. Two sets of perforations were parallel to the hydraulic-fracture direction, the other two were perpendicular to it. With this perforation pattern, hole 2 was designed to generate fractures in the two principal stress directions, parallel to the line connecting holes 2 and 5 and perpendicular to it, respectively. With this arrangement, we enhanced the possibility of fractures from holes 3 and 4 intersecting the fractures from hole 2.

The experimental assembly consisted of the propellant tool, the pressure transducer assembly, a rubber wiper above it, and a cable tube made of sections of AQ drill rod. Shown in Figure 2 is an assembled tool being lowered in hole 1. During the test the tool package was supported from above by the cable tube. The pressure transducer assembly houses a 138-MPa (20-kpsi) pressure transducer and passthroughs for the ignition wires. One end of the pressure transducer assembly screws onto the cable tube directly; the other end is attached to the propellant tool through a transition section (Figure 3a). The transition section has a triangular cross section (Figure 3b). The portion near the base has a 3.8-cm (1.5-in.) through hole for the pressure transducer to "see" the top of the tool directly. A blind hole near the vertex houses a maximum-pressure crush gauge. In earlier experiments, vent holes were drilled into the transition section to ensure that no air was trapped in the transition section or in the connecting tube between the pressure transducer and the top of the tool. The scheme was not entirely successful; in later experiments grease was pumped into the cavity to ensure direct pressure coupling.

All the propellant tools used were 3.7 m (12 ft) long. The Servo-Dynamic StressFrac™ tools, were used in holes 1 through 4. An SNL tool was used in hole 5. The StressFrac™ tools are

Figure 2. Propellant Tool Being Lowered into Test Hole 1

Figure 3a. Pressure Transducer Assembly and Transition Section

Figure 3b. Close-Up View of Transition Section

one-piece solid propellants with a thin aluminum skin. Three linear ignitors (instead of one as used in oil field tools) parallel to the axis of the tool were used to ensure ignition and to control initial burning rate. Otherwise, the tools were identical to commercial tools. Two sizes of StressFrac™ tools were used, one 6.4 cm (2.5 in.) in diameter with 16.6 kg (36.6 lb) of propellant and one 7.6 cm (3.0 in.) in diameter with 26.2 kg (55.7 lb) of propellant. The SNL tool used a 3-in. PVC tubing, 8.26 cm (3.25 in) I.D.; 8.89 cm (3.5 in.) O.D., filled with 17.9 kg (39.5 lb) of a 50/50 mixture of M30A and M30B artillery propellant pellets; a rapid ignition propagation (RIP) ignitor runs down the axis of the tool. Details of the M30 propellant pellets and the design of the SNL tool can be found in References 3 and 6.

The tools were installed 0.3 m (1 ft) above the bottom of the borehole. The borehole was partially filled with water such that upon installation, the water level would barely cover the top of the wiper. A sand tamp was formed above the wiper to within 15 cm of the collar to contain gas pressure during the experiment. The test zone defined by the bottom of the sand tamp and the hole bottom was typically 4.4 m (14.5 ft). A 15-cm 50/50 mix of sand and sulfate-based cement at the collar ensured containment of the stemming.

Core samples obtained during drilling were examined for existing fractures before the experiments. Some healed fractures were found, but no open fractures were observed in the test intervals. Core logs are shown in Appendix B. Before each test, a TV log and a constant-pressure permeability test were performed. For each shot, pressure history and peak pressure were measured. Observations were made of borehole-to-borehole interconnection through fracture intersections using video camera recordings. Before each shot, all holes were filled with water. In case an interconnection was made during the shot

between the test hole and any particular hole, water would eject from that particular hole (Figures 4a, 4b). Following each test, a TV log was used to examine the fractures. Typically, fractures were observed to extend 2 m above the test zone. The fracture width increased toward the middle of the test zone. A constant flow test was performed to obtain the effective permeability of the borehole. Dyed water of different colors was used in the flow tests to mark the fractures. In cases where borehole ejection occurred, permeability of ejection holes was also measured to determine the effects of a shot on neighboring holes.

At the conclusion of all the shots, the test bed was mined back to examine the fractures in detail. To prevent the near wellbore fracture patterns from being damaged during mining, the patterns were potted in place by filling the borehole with grout. The grout was sufficiently fluid to also fill the near wellbore fractures under the static head of the grout column. Mineback was accomplished by excavating a 20-m (66-ft) drift 9.9 m (32 ft) below (downhole distance) the test drift and originating from the U12g.12 drift in the N13°E direction. The centerline of the drift passed approximately 0.8 m (2.5 ft) to the right of hole 5 and 0.6 m (2.0 ft) to the left of hole 2 (Figure 1). The mineback drift was nominally 3.7 by 3.7 m (12 by 12 ft). Each successive face corresponds to a "vertical" cross section of the test bed, exposing the last 3.1 m (10 ft) of the borehole and 0.6 m (2 ft) beyond the bottom of the borehole. In operation, the continuous mining machine makes successive horizontal cuts up the face. Typically, in each sweep, a horizontal trough 0.5 m high by 3.7 m wide (width of the face) by 0.5 m deep is cut. At each cut, a strip of "horizontal" cross section is also revealed. ("Vertical" and "horizontal" are in quotation marks because they are actually curved surfaces.)

Figure 4a. Water Ejection due to Shot in Hole 5

Figure 4b. Water Ejection due to Shot in Hole 3

EXPERIMENTAL RESULTS

The present test series is designated as GTSF and each test hole number follows the GTSF designation. The results of the tests are discussed in the sequence that the shots were made because fracture patterns are significantly influenced by fractures already existing at the time of the shot. A three-dimensional pictorial representation of the fracture network in the last 3 m (10 ft) of the test zone, which is 4.4 m (14.5 ft) thick, is shown in Figure 5. Some of the minor fractures have not been included in the figure for the sake of clarity. TV logs show that in all the tests, fractures actually began to appear first at about 2 m above the test zone, in the hydraulic-fracture direction only. In cases where multiple fractures existed, the first occurrences were usually very near the top of the test zone; fracture width at the wellbore continued to increase downhole to the middle of the test zone (Figures 6a, 6b, 6c).

An overall summary of test results is given in Table 2.

Experiment GTSF-1

GTSF-1 was a practice test to help define the type of tool and pressurization rate needed for the other holes. A StressFrac™ slow-burning 6.4-cm-diameter tool was used. Pressure history for the test is shown in Figure 7. Maximum pressure, as recorded by the pressure transducer, was 34.3 MPa (4970 psi); the value given by the crush gauge was 35.2 MPa (5100 psi). This close agreement was typical of all shots (Table 3); therefore, for subsequent shots, only values obtained by pressure transducers were used. The average pressurization rate, defined as the ratio of the maximum pressure to the pressure rise time (the time required to reach the peak pressure), was 9 kPa/μs (1.3 psi/μs).

Figure 5. 3-D Reconstruction of the Fracture Network

Figure 6a. Multiple Fracture Pattern near the top
of the Test Zone

Figure 6b. Multiple Fracture Pattern Approximately
One-Fourth of Way Down the Test Zone

Figure 6c. Multiple Fracture Pattern near the Middle
of the Test Zone

Table 2

Overall Summary of Test Results

<u>Hole</u>	<u>Tool</u>	<u>Open/Cased</u>	<u>Peak Pressure (MPa)</u>	<u>Ave. Press. Rate (kPa/ms)</u>	<u>Results</u>
1	StressFrac™ 6.4 cm, 16.6 kg Slow-burning	Open	34	9	Mainly hydraulic fracture Intermittent multiple-fracture initiation
2	same as above	Cased Perforated	104	110	Fractures in two principal stress directions
4	StressFrac™ 7.6 cm, 26.2 kg Slow-burning	Open	70	350	Six major fractures Water ejection at holes 2 and 5 Fracture intersections with fractures from 2
5	SNL 8.9 cm, 17.9 kg 50/50, M30A/M30B	Open	70	33	Five major fractures Water ejection at holes 2 and 4 Fracture intersections with fractures for 2 and 4
3	StressFrac 6.4 cm, 16.6 kg Fast-burning	Open	79	172	Four major fractures Water ejection at hole 2 Fracture intersections with fractures from 2

Figure 7. Pressure-Time History for Hole 1

Table 3

Comparison of Peak Pressure Obtained by Pressure Transducers and Crush Gauges

<u>Hole</u>	<u>Pressure Transducer (psi)</u>	<u>Crush Gauge (psi)</u>
1	4,970	5,100
2	15,000	14,300
3	11,500	9,500
4	10,150	9,500
5	10,100	9,500

Hole 1 is outside the mineback zone; no information on the fracture pattern is available. The TV log shows the main fractures to be in the hydraulic-fracture direction with intermittent sections showing fracture initiation in the non-hydraulic-fracture direction. Fracture width at the wellbore was as wide as 4 cm.

Experiment GTSF-2

The second shot of the series was in hole 2, which was cased and perforated and used a tool identical to the one in hole 1. A peak pressure of 104 MPa (15,000 psi), was reached at an average pressurization rate of 110 kPa/ μ s (16 psi/ μ s). The pressure history is shown in Figure 8.

There were apparently voids in a portion of the sand tamp due to bridging of the wet sand; as a result, a section of the cable tube was under pressure during the test and was pinched (Figure 9). Also shown in the figure is a typical fragment of the aluminum casing of the StressFrac™ tool approximately 0.6 m long. Some recovered fragments had teardrop-shaped

Figure 8. Pressure Time History for Hole 2

Figure 9. Pinched Cable Tube and Tool Skin Fragment
from Test in Hole 2

slivers at the edge, indicating that partial melting had occurred. Because the aluminum skin of the tool was severely deformed before failure, the fragments were work hardened. Removing a skin fragment lodged in a fracture can be quite difficult. In many oil field applications, an acid treatment follows propellant fracturing, and fragment removal is then not a problem.

As designed, the shot in hole 2 produced fractures in the two principal stress directions (Figure 5). The hydraulic fractures are the most prominent. The southern branch is fully exposed in the mineback (Figure 10). Parts of it (not through the full depth) extend at least 7 m away from the borehole. Only 1 m of the northern branch had been mined back, but it should be of comparable length to the southern branch. Near the bottom of the hole a small (5-cm) split in the casing was observed. Fragments of the aluminum skin of the StressFrac™ tool protruded from the split for about 5 cm. Perpendicular to the hydraulic fracture to the west, there was a short fracture of about 0.5 m. The casing was found to be split on the east side along the perforations (Figure 11) about 11 m from the collar (the hole is 12.8 m TD) for about 0.5 m. The split was as wide as 5 cm. Correspondingly, near the casing split, the fracture extending from hole 2 was also quite wide, with a fracture aperture of about 0.6 cm as far as 0.5 m from the hole. This eastward fracture appears to be turning south, away from the hole, possibly toward the hydraulic-fracture direction. This change of direction becomes more and more abrupt toward the bottom end of the test zone (Figure 5). The edge of this curved fracture can be traced for at least 3 m along the east rib (sidewall) of the mineback drift.

On a more local scale, there appeared to be two sets of fractures out of each perforation, one vertical and one horizontal. The horizontal fractures (Figure 12) usually do not extend

Figure 10. Southern Branch of the Hydraulic Fracture (seen on the back, ceiling, of the tunnel) from Test in Hole 2

Figure 11. Casing Split in Hole 2

Figure 12. Horizontal Fractures out of Perforations

more than 0.3 m beyond the borehole; the vertical fractures are distinct for each perforation (Figures 13a, 13b). Due to a slight misalignment of the perforation shots, every other perforation is displaced from the neighboring one. Up to about 30 cm away, the edge of the vertical fractures takes the shape of a sine wave, with trough and valley corresponding to the perforation locations. Farther out, the fractures appear to be stranded. These individual fractures eventually merge together away from the hole.

Experiment GTSF-4

Hole 4 was shot with a 7.6-cm-diameter StressFrac™ tool. Peak pressure of 70 MPa (10,150 psi) was reached at an average pressurization rate of 350 kPa/μs (51 psi/μs). Pressure time history recorded for the test is shown in Figure 14. Pressure oscillation is apparently due to gas trapped in the transition section and the connecting tube (Figure 3a) between the pressure transducer assembly and the top of the tool. The maximum pressure reading appears to be credible since it is in agreement with the 9500-psi reading given by the maximum pressure crush gauge located outside the transition section. During the test, water ejection was observed from holes 2 and 5 (Figure 4a).

As many as six major fractures grow out of the borehole (Figures 5, 15). Posttest TV log shows that the fractures are about 5 to 7 cm wide near the borehole. Near the middle of the test zone, the borehole is greatly enlarged; the circular geometry is replaced by a star pattern (similar to Figure 6c). Borehole enlargement is not due to plastic deformation corresponding to borehole crushing (Appendix A, Reference 3); rather, it appears to have been caused by rock fragments detaching from the borehole as a result of fine fracturing near the borehole. This star pattern at the borehole is also typical of other holes with multiple fractures. Of the six fractures, two are in the

(a)

(b)

Figure 13. (a) "Vertical" Fractures out of Perforations Merging into a "Sine Wave." (b) Schematic Interpretation of Figure 13a

Figure 14. Pressure Time History for Hole 4

Figure 15. Six Fractures Growing out of Hole 4

hydraulic-fracture direction, two are toward the west, two toward the east. The two east-running fractures intersect the southern branch of the hydraulic fracture from hole 2 (Figures 16, 17). In this case, the advance of the fractures had been arrested at the intersections. The interconnections made through these intersections had caused a water ejection at hole 2. There does not appear to be any direct link between holes 4 and 5 in the mineback zone. A TV log of hole 5, after the shot in hole 4, indicated a small, localized fracture at about 9 m downhole, corresponding to a healed fracture observed in the core samples of hole 5. Apparently, this fracture had reopened and caused the ejection out of hole 5.

Experiment GTSF-5

Following hole 4, hole 5 was shot with the SNL tool. Because of the difficulty caused by trapped air in the transition section between the tool and the transducer assembly in GTSF-4, the transition section and the connecting tube were filled with grease to ensure coupling for the subsequent tests. The pressure time history is shown in Figure 18. The peak pressure was 69.6 MPa (10,100 psi); the average pressurization rate was 33 kPa/ μ s (4.8 psi/ μ s). Water ejection occurred at holes 2 and 4.

There are five major fractures extending out of hole 5 (Figure 5). The nonhydraulic fractures do not extend as far as those out of hole 4; borehole enlargement (due to fragmentation) was observed, but it was not as extensive as in hole 4. The fracture going east from the hole intersects the hydraulic fracture going south from hole 2 (Figures 19, 20, 21a, and 21b). Note that there are PVC fragments from the propellant canister trapped in the fractures (Figure 20). During mineback, similar fragments were found in the fractures as far away as 5 m. Since the PVC tube wall of the canister has an original thickness of 3 mm, the fracture aperture is at least that wide during formation of the fractures. The actual fracture width observed

Figure 16. Intersection of Two Nonhydraulic Fractures from Hole 4 with Hydraulic Fracture from Hole 2

Figure 17. Another View of Nonhydraulic Fracture from Hole 4
Intersecting Hydraulic Fracture from Hole 2

Figure 18. Pressure-Time History for Hole 5

Figure 19. Nonhydraulic Fracture from Hole 5 Intersecting Hydraulic Fracture from Hole 2

Figure 20. Nonhydraulic Fracture from Hole 5 Intersecting Hydraulic Fracture from Hole 2. (Note PVC fragment trapped in fractures.)

(a)

(b)

Figure 21. (a) Nonhydraulic Fractures from Holes 5 and 4
Intersecting Hydraulic Fracture from Hole 2;
(b) Schematic of Fracture Pattern in Figure 21a

during mineback typically varies from centimeters near the borehole to submillimeters in the fractures. It is also interesting to note that several of the near-borehole fractures did not result in any fracture extension into the formation (Figure 20). This is typical of all the experiments. As a result, posttest TV logs almost always give a more optimistic assessment of a test. The fracture going west from the hole disappears into the left rib; the extension to the rib measures a little less than 2 m. The northern branch of the hydraulic fracture from hole 5 runs very close to the southern branch of the hydraulic fracture from hole 2; at the lower part of the test zone, they probably merge in places. The southern branch of the hydraulic fracture out of hole 5 runs into the left rib at about 5 m from the hole. Where it entered the rib, slabs of rock simply slumped off after the mineback, revealing the fracture surface for a meter length over the height of the drift (Figure 22). The multicolored surface of the fracture is a vivid example of the hole-to-hole interconnections through a fracture network. The fifth fracture was first encountered 6 m (south) from the borehole near the middle of the drift; it is vertical in the upper part of the test zone but curves over to the lower extent of the southern branch of the hydraulic fracture (Figures 5, 23a, 23b).

Experiment GTSF-3

The last shot of the series was in hole 3 using a fast-burning, 6.4-cm StressFrac™ tool, resulting in a peak pressure of 79.3 MPa (11,500 psi) and an average pressurization rate of 172 kPa/μs (25 psi/μs). The pressure history is shown in Figure 24. Water ejection was observed at hole 2.

The shot resulted in four main fractures (Figures 5, 25a, 25b). The southern branch of the hydraulic fracture extends 3.5 m from the borehole. Two fractures break to the east from the borehole. One extends about 0.6 m. The other is in a direction perpendicular to the hydraulic fractures for approximately

Figure 22. Exposed Surface of the Southern Branch of the Hydraulic Fracture from Hole 5

(a)

(b)

Figure 23. (a) Edge View of Fractures from Hole 5 (Hole 5 located approximately 1 m into the face); (b) Schematic Interpretation of Fractures in Figure 23a

Figure 24. Pressure-Time History for Hole 3

Figure 25a. Fracture Patterns out of Holes 3 and 4

Figure 25b. Fracture Patterns out of Hole 3, a Closer View

1.5 m and then turns to a near-parallel direction with the drift centerline (Figure 26). The fracture was not fully excavated; it is possible that the fracture will eventually turn to the hydraulic-fracture direction. The edge of this particular fracture is visible along the right rib of the drift for about 2.5 m. The northern branch of the hydraulic fracture intersects the nonhydraulic fracture going east from hole 2. In this case, the advancing fracture did not stop at the fracture intersection; it cut across the fracture out of hole 2 (Figures 27a, 27b). This hole 3-to-hole 2 connection was the most productive fracture interconnection of the entire test series (details in the next section). In addition to the four main fractures, a fracture near the top of the test zone was observed to extend in a direction perpendicular to the hydraulic-fracture direction. At 0.6 m downhole, it comes within a few centimeters of the hydraulic fracture out of hole 2 (Figure 28); this distance increases to about 30 cm at 1 m downhole. It appears that near the top of the test zone, the two fractures might intersect.

Flow Tests

Flow tests were carried out before and after each shot to characterize the permeability of the formation surrounding a borehole. Each test involved packing off the last 4.6 m (15 ft) of a hole and injecting water into the borehole. The pressure history was then interpreted in terms of the "line source" solution for one borehole in an infinite medium.^{9,10} Therefore, the value obtained is termed "effective permeability" since the effect of fractures is not explicitly taken into account. The preshot permeability of each borehole was measured using a constant pressure injection test with a driving pressure between 0.3 and 1.0 MPa (50 and 150 psi) and a resulting flow rate of 0.2 to 1.5 liters/min (0.05 to 0.4 gal/min). The post-shot permeability was measured using a constant flow rate test with a flow rate of 19 to 76 liters/min (5 to 20 gal/min) with a corresponding pressure in the range of 1 to 1.7 MPa (150 to 250 psi). In both cases, the shut-in data following injection were

Figure 26. East-Running Fracture out of Hole 3

Figure 27a. Intersection of East-Running Fracture out of Hole 2 and North-Running Fracture (Hydraulic) out of Hole 3

Figure 27b. A Closer View of the Fracture Intersection
in Figure 27a

Figure 28. West-Running Fracture out of Hole 3 Approaching Hydraulic Fracture out of Hole 2

found to give the most consistent results. Typical preshot and postshot shut-in data are shown in Figures 29 and 30, respectively.

The results of the "effective permeability" measurements are summarized in Table 4. Generally, the primary effect of a shot is to increase the permeability of the test hole. While typical preshot permeabilities for open holes are in the range of 0.4 to 0.7 md, postshot permeabilities range between 80 and 170 md. The shots also have secondary effects, i.e., effects on neighboring holes. For example, after the shot in hole 4, the effective permeability of hole 5 increased from 0.5 to 2 md because of the interconnection between holes 4 and 5. Secondary effects are not always positive; effective permeability of hole 2 and 4 both showed a decrease as a result of shots in neighboring holes (Table 4).

Table 4

Effective Borehole Permeability (md)

	HOLE				
	1	2	3	4	5
PRE-TEST	0.4	2.0	0.4	0.7	0.5
1	78				
2		172			
4				161	2.0
5				69	79
3		142	149		

The fracture network grew with each shot. Production out of neighboring holes was observed during permeability flow tests for later shots. Two hole-to-hole flow tests were carried out between holes 2 and 3 to quantify the flow characteristics of the fracture interconnections formed in the experiments. In

Figure 29. Preshot Flow Test for Hole 4. (Δt is time borehole has been shut in; t is length of time of pressurization. See Ref. 9 for details.)

Figure 30. Postshot Flow Test for Hole 4

these experiments, production out of hole 2 was monitored while water was injected into hole 3 at a constant rate with holes 4 and 5 packed off. After an initial transient, the pressure rise became relatively slow and correspondingly, the production from hole 2 reached a constant value. In one test, the injection rate was 38 liters/min (10 gal/min), the average driving pressure was 1.3 MPa, and production out of hole 2 was 4.2 liters/min (1.1 gal/min) (Figures 31, 32). The corresponding values for the other test were 19 liters/min (5 gal/min), 0.65 MPa (95 psi), and 2.6 liters/min (10 gal/min) (Figures 32, 33). The permeability (K) aperture (w) product can be estimated using the following equation:

$$\Delta P = \frac{q\mu L}{(Kw)h} \quad (1)$$

where ΔP is the driving pressure difference, q is the flow rate through the aperture, μ is the viscosity of water, L is the path length and H is the fracture height. By using this equation, it is implicitly assumed that there is no leak-off to the matrix rock. The fracture path was the northern branch of the hydraulic fracture out of hole 3 connecting to the eastern branch of the non-hydraulic fracture out of hole 2 (Figure 5). The path length was 3.5 m, and the fracture height was assumed to correspond to the height of the test zone, 4.6 m. The fracture flow rate, q , cannot be directly measured; its lower bound is the production rate corresponding to the case of negligible leak-off from other fractures out of hole 2 under the static head in hole 2. Using the production values in Equation (1), the permeability-aperture product was calculated to be 0.04 to 0.05 darcy-meters. Observations made of water leakage out of hole 2 indicated that taking leak-out into account will increase the Kw product by no more than 50%.

Figure 31. Pressure-Time History with Injection Rate of 10 gal/min during Postshot Flow Test for Hole 3

Figure 32. Production out of Hole 2 during Flow Test at Hole 3

Figure 33. Pressure-Time History with Injection Rate of 5 gal/min during Postshot Flow Test for Hole 3

EXTRAPOLATION TO GEOTHERMAL RESERVOIRS

These tests have been very successful in ash-fall tuffs at shallow depths where in situ stresses are low. To determine the suitability of this process for geothermal applications (hard rocks at greater depths), we need to develop scaling criteria. Some simple results (Appendix A) show the effect of rock properties and in situ stresses on the initiation process and the length.

The important parameter for determining the mode of fracturing is the pressure loading rate at the point of fracturing. The minimum loading rate for multiple fracturing can be estimated from Equation 6 of Appendix A as:

$$\frac{dp}{dt} > \frac{16(\sigma_{\max} - \sigma_{\min})C}{\pi d}$$

where C is the compressional wave velocity of the rock, d is the borehole diameter, and $\sigma_{\max}$ and $\sigma_{\min}$ are the maximum and minimum horizontal stresses, respectively. Using values of 7000 ft/s for C , 0.33 ft for d , and a stress difference of 300 psi, we estimate a pressure loading rate of about 30 psi/ μ s will be required for multiple fracturing. This value is in good agreement with the test data. It may be slightly high, as should be expected, since it assumes a perfect hole with no flaws and no relaxation of the stress concentration. It also calculates the loading rate necessary to initiate fractures at the most difficult location (the highest stress concentration). For a harder rock at depth, we could expect to have values of $C = 12,000$ ft/s, $d = 8$ in., and a stress difference of 1000 psi. This would require a loading rate of about 90 psi/ μ s. This pressure loading rate can be easily obtained.

To avoid crushing of the borehole, the relation developed as Equation 22 of Appendix A is

$$\frac{dP}{dt} < \frac{4C[\sigma_o + 3\sigma_{\min} - \sigma_{\max} - 2P_i]/d}{3 \left[\sqrt{\frac{1+v}{1-v}} \sqrt{1 - \frac{(3\sigma_{\min} - \sigma_{\max} - 2P_i)}{(\sigma_o + 3\sigma_{\min} - \sigma_{\max} - 2P_i)}} - 1 \right]}$$

For these tests, $\sigma_{\min} \approx 500$ psi, $\sigma_{\max} \approx 800$ psi, $p_i = 0$, $v = 0.3$ and $\sigma_o \approx 7000$ psi. This gives a maximum loading rate of 880 psi/ μ s. The multiple-fracturing window spans well over an order of magnitude of pressure-loading-rate range.

For the geothermal case, σ_o might be 15,000 psi, $\sigma_{\min} \approx 5000$ psi, $\sigma_{\max} \approx 6000$ psi, $p_i \approx 3000$ psi, and $v = 0.2$. Other values would be the same as above. This would yield a maximum loading rate of 12,000 psi/ μ s. Thus, we would not expect crushing to occur in geothermal hardrock wells.

Estimates of fracture length are much more difficult since this is a complex fluid-mechanics/rock-mechanics interaction problem. However, Equation 34 of Appendix A was developed from simple energy considerations to give

$$L \propto \sqrt{\frac{E K_p}{\eta(1-v)^2 P(P + \sigma_{\min})}}$$

where E is Young's modulus, K_p is the energy of the propellant and η is the number of fractures. If we consider these

present tests as calibration points, we can determine β , a proportionality constant, from

$$L = \beta \sqrt{\frac{E}{\eta(1 - \nu^2) P(P + \sigma_{\min})}}$$

Then, assuming all the fracturing and burning processes are similar in the two cases, with $\eta = 4$, $E = 5 \times 10^5$ psi, average fracture lengths of 10 ft, and a pressure of 10,000 psi, we can determine β and use this to estimate geothermal results. Remember also that in a geothermal hole with twice the diameter, we can emplace four times the propellant. If $P \approx 10,000$ psi, $\sigma_{\min} = 5,000$ psi, and $E \approx 6 \times 10^6$ psi, we estimate fracture lengths of 50 ft. In a higher modulus rock, we can expect longer fractures unless the minimum stress becomes very large.

These simple scaling calculations are only order-of-magnitude estimates of the true response, but they give some understanding of how the parameters affect the fracturing results. For geothermal applications, this shows that we can reasonably expect to induce multiple fractures and that significant lengths can be obtained.

CONCLUSIONS

The present series of tests in welded tuff demonstrated that it is possible to create multiple fractures in water-filled boreholes using existing commercial tools. Multiple fractures were obtained with average pressurization rates of 33 to 350 kPa/ μ s. The fact that the operating range spans one order of magnitude in pressurization rate means that the method is practical for field applications. Peak pressures for these tests varied from 34.3 MPa (4970 psi) to 79.3 MPa (11,500 psi) for the openhole tests and reached 104 MPa (15,000 psi) for the cased/perforated test. The peak pressure is not important for openhole tests, but high pressures in casing may cause damage to the pipe.

Comparison of pre- and posttest permeability measurements showed that the effective permeability of the near-wellbore region was enhanced by a factor of several hundred by propellant fracturing. Fracture conductivity between adjacent wells, a description of the interconnectedness of the created fracture systems, was found to be about 0.04 to 0.05 darcy-meters. Pre-fracture conductivity was so low that it was not measureable.

The results of the present experiment have been extrapolated to estimate their application in hard rocks at depth, conditions that might be encountered in geothermal fields. Calculations show that under those conditions the range of pressurization rates to produce multiple fractures increases by about two orders of magnitude. Because of higher values of the Young's modulus, fracture lengths of the order of 15 m (50 ft) could be obtained. With such expected performance, propellant fracturing would certainly be useful for connecting a borehole to nearby producing fractures or as a method for near-wellbore cleanup.

Because the fracture aperture was found to be as wide as several millimeters during the shot, it will be useful in the future to test shots loaded with proppants. If the fractures can be propped near the borehole, it might then be possible to use multiple shots to achieve fracture extension.

Finally, for actual geothermal applications, it will be necessary to develop high-temperature tools. In small-scale laboratory studies some of the propellants currently in use in the StressFrac™ tools appeared to be stable to up to 340°C (650°F); however, high-temperature ignition systems need to be developed. While such a system is not currently available, the required performance is not beyond the current state of the art.

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APPENDIX A

Scaling Criteria

We have demonstrated the capability to create multiple fractures from both air- and water-filled open holes in ash-fall tuffs at shallow depths. We would now like to use this technology at much greater depths in harder and stronger rocks for geothermal applications. To scale these results to a geothermal well, we need to understand how rock properties, in situ stresses, and borehole size influence the multiple fracturing process. Finite-element models¹⁻³ have been developed that provide an excellent representation of the process, and for specific design applications, they are invaluable. However, they are not always available to everyone, and we can also lose some of the insight into the process by simply running codes.

This appendix describes some design calculations^{4,5} that were developed several years ago to help us understand the process, particularly with respect to the initiation of multiple fractures. These results are analytical and require some assumptions about the loading process. Thus, they may not be able to be applied directly without some calibrating data for a particular propellant or rock type. However, they clearly show the important effects of in situ stresses, material properties, and the borehole size and are very useful for scaling results or designs.

FRACTURE INITIATION

The most severe constraint on propellant fracturing technology is the proper loading rate for initiating multiple fractures. We know from many results at the NTS that slow loading rates cause hydraulic fractures, high loading rates result in crushing and compaction, and some intermediate loading rate will yield multiple fractures propagating radially away from the wellbore. The major problem is to determine the boundaries of these three regimes and what parameters influence the location of the boundaries in some pressure-rise-time-vs.-property/borehole space. The multiple-hydraulic and the multiple-explosive boundaries will be developed separately.

Multiple-Hydraulic Boundary

The easier problem is the onset of multiple fracturing. Several modes of possible multiple-fracture initiation have been postulated. The two most feasible are (1) nearly simultaneous initiation before a release wave around the borehole wall can "clamp" the wall and prevent any further crack initiations and (2) the "hydraulic effect," where initially a single (two-wing) crack is formed but the burning so overdrives the crack that pressure continues to rise and another set of cracks is initiated (probably at flaw nucleation sites) and so on. For two reasons, only the first mode is considered: it is the more conservative of the two and the second model is intractable in anything but a detailed finite-element model that incorporates the gas dynamics. Note that this criterion does not address the problem of how fractures propagate or why we can continue to extend more than two cracks.

If a hole is drilled in a medium with in situ stresses, stress concentrations in the tangential direction are set up around the borehole. This is illustrated in Figure A-1 for the usual case of the borehole parallel to one of the stresses and

Figure A-1. Illustration of Tangential Stresses around a Borehole Drilled in a Medium Having In Situ Stresses

thus perpendicular to the other two, which we label $\sigma'_{\max}$ and $\sigma'_{\min}$. $\sigma'_{\max}$ and $\sigma'_{\min}$ are effective stresses acting on the borehole, with an effective stress defined as

$$\sigma' = \sigma - \alpha P_i \quad (1)$$

where σ is the total stress, P_i is the pore pressure, and α is the poroelastic constant, which we assume to be unity.

The basis of breakdown analysis of hydraulic fracturing is a result of these stress concentrations. It is assumed in hydraulic fracturing that when the pressure overcomes the smallest stress at the borehole wall ($3\sigma'_{\min} - \sigma'_{\max}$) plus some tensile strength, T , of the rock, then the wall fails and a crack initiates.

For multiple fracturing, we have a similar situation with a time criterion added. Again, we expect the wall to fail when the pressure reaches

$$P' = 3\sigma'_{\min} - \sigma'_{\max} + T \quad (2)$$

where P' is the pressure above the pore pressure, or an effective pressure. We also require that the pressure then rise from this value to that of the larger stress concentration (plus the tensile strength) in a shorter time than the release wave can travel around the borehole to clamp it. At this point, the pressure needed is

$$P' = 3\sigma'_{\max} - \sigma'_{\min} + T \quad (3)$$

Note that the four-fracture pattern is both a conservative criterion as well as the commonly observed behavior.

As shown in Figure A-2, the time required to do this is just the time it takes for the wave to travel one quarter around

Figure A-2. Illustration of Release Wave Distance/Time Travel around a Wellbore

the wellbore from the cracks initiated at the lowest stress location to the highest stress location. The velocity of propagation is assumed to be the compressional wave speed, C . Then, we get a Δt of

$$\Delta t = \frac{\pi d}{4C} \quad (4)$$

Thus, the minimum pressure loading rate required to initiate fractures in two directions (four wings) is simply the difference in the minimum and maximum stress concentrations divided by Δt . This yields

$$\frac{dP'}{dt} > \frac{(3\sigma'_{\max} - \sigma'_{\min} + T) - (3\sigma'_{\min} - \sigma'_{\max} + T)}{\frac{\pi d}{4C}} \quad (5)$$

Subtracting terms and replacing P' , $\sigma'_{\max}$, and $\sigma'_{\min}$ by their total stresses, we get

$$\frac{dP}{dt} > \frac{16 (\sigma_{\max} - \sigma_{\min}) C}{\pi d} \quad (6)$$

This shows that the rise time depends upon the in situ stress difference as well as the wave speed in the rock and the borehole diameter. Note that as the stress difference goes to zero, the pressure loading rate goes to zero, which says that any loading rate will produce multiple fractures. Under ideal conditions, this may be correct, but due to flaws, inhomogeneities, and other irregularities, there probably is a minimum loading rate in practice. The more troublesome case is when the in situ stress difference is large. We then require much higher loading rates and possibly narrow the multiple fracturing window. This criterion will be compared with the tuff data at the end.

Even if the fractures do initiate by some "hydraulic" effect, as it has been called, this provides what is probably a

more conservative estimate of the loading rates necessary for multiple fracturing. Flaws will also reduce this criterion somewhat.

Explosive-Multiple Boundary

The more difficult criterion is that of the onset of crushing, which results in a maximum loading rate for the multiple-fracturing regime. Initially one would surmise that crushing is controlled by the peak pressure rather than by the loading rate. However, the NTS results have shown otherwise. A good comparison can be found in the Multifrac Experiment,⁶ where one of the tests, GF4, with a loading rate of 62 psi/ μ s and a peak pressure of 36,000 psi, resulted in multiple fracturing, while another, GF7, with a loading rate of 150 psi/ μ s and a peak pressure of 11,000 psi, resulted in crushing. Although the rock types for each shot were somewhat different, such behavior clearly shows that loading rate is the first-order effect. The problem is to determine what causes the loading rate to affect the behavior so significantly.

Comparison of such results at the Multifrac experiment (and many more recent tests) with models developed by T. Bligh⁷ and E. P. Chen⁸ shows that crushing begins near the point where the loading rate is so fast that the tangential stress around the borehole wall is compressive at early times (recall that in the quasi-static loading case, such as hydraulic fracturing, the tangential stress due to internal pressure, P , is always tensile and equal in magnitude to P). The tangential stress becomes more compressive as the loading rate increases because in very short times (microseconds), the pressure wave has traveled only a short distance into the rock because of the compressional wave speed limitation. Since the rock outside of this small zone has as yet no information that the borehole is expanding, it cannot expand to accommodate some of the strain. Thus, the borehole is expanding due to the increased pressure, but the nearby rock is

rigid because of wave speed limitations. This results in increased compression in other directions, i.e., tangentially.

An example of this behavior is shown in Figure A-3. The tangential stress calculated using Chen's⁸ analysis for two different loading rates is plotted with the pressure-loading behavior. The parameter b is a pseudo rise time. We can see that in slowly loaded boreholes, the tangential stress is nearly the same magnitude as the internal pressure (but different in sign). As the loading rate is increased, we eventually reach the point where the tangential stress is initially compressive and then lags behind the pressure.

Figure A-4 shows the two situations--one for a static pressurization with an expanded borehole and one for the dynamic case with the unstressed rock retarding expansion. Note that the borehole cannot expand as far as necessary in the dynamic case.

No In Situ Stresses -- First, we can develop a simplified analysis without in situ stress to show more clearly some of the important effects and then later we can include in situ stresses. The first consideration is the failure mechanism, but failure at intermediate loading rates is not well understood. As the first simplification, we will extrapolate from the static regime and assume a shear failure criterion in compression and a maximum tensile stress criterion in tension. For simplicity, we use a maximum shear stress criterion, but an octahedral criterion (or some other) will give similar results.

The maximum shear stress criterion is given by

$$\sigma'_1 - \sigma'_2 > \sigma_0 \quad (7)$$

where σ'_1 is the greatest effective compressive stress, σ'_2

Figure A-3. Comparison of Tangential Stress for Two Different Loading Rates along with the Pressure-Loading Behavior

Figure A-4. Illustration of Static and Dynamic Pressurization with an Expanded Borehole and with the Unstressed Rock Retarding Expansion, Respectively

is the least, and σ_0 is some strength of the material. We are concerned with the point where the greatest shear stress, or greatest stress difference, can be developed without the stress becoming tension and tensile cracks (hydraulic fractures) developing. Obviously, with the quasi-static loading, the tangential stress around the wellbore wall quickly becomes tensional and hydraulic fractures develop. No large stress differences can develop. However, with rapid loading, the tangential stress can be compressive for early times, which (1) does not allow cracks to develop but (2) allows large stress differences (high radial stress, small tangential stress) to develop. The worst possible case develops when the tangential stress has been compressive initially but is finally starting to become tensional at later times. Just before the tangential stress becomes tension, we have the largest possible stress difference, with the tangential stress, σ'_t , equal to zero and the radial stress, σ'_r , equal to the effective pressure. Substituting $\sigma'_1 = \sigma'_r$ and $\sigma'_2 = \sigma'_t = 0$ into Equation 7, we get the very simple failure criterion that

$$\sigma'_r = P' > \sigma_0 \quad (8)$$

The big problem is to determine when $\sigma'_t = 0$ and what the value of σ'_r is when this occurs.

Before continuing with the analysis, it is useful to show why no crushing will occur once tensile cracks have been initiated. As shown in Figure A-5, once cracks are initiated, we will have a state of stress that has changed significantly. The radial stress, σ'_r , at the wall will be P' ; the tangential stress, σ'_t , will be approximately P' ; and from plane strain conditions, the axial stress, σ'_z , will be $\sim 2\nu P'$. Now our failure condition is

$$(P' - 2\nu P') > \sigma_0 \quad (9)$$

Figure A-5. Illustration of Stress Change due to Initiation of Tensile Cracks

Thus, it will take considerably higher pressures to crush the rock once cracks have formed. In fact, addition of the in situ stresses will make it harder still and we know that σ_0 increases with loading. It is probably safe to say that we will not get crushing once cracks have formed unless pressures get prohibitively high (on the order of several kilobars). This is why the peak pressure is so unimportant for crushing.

The second point is that the worst case is when σ'_t is slightly tensile. Obviously, when $\sigma'_t = T$, cracks will form, but when σ'_t is just less than T , crushing is most likely since $\sigma'_r - \sigma'_t$ is maximum. By using $\sigma'_t = 0$, we tacitly assume the strength of the rock is small or abundant flaws are present in the wellbore. No loss of generality occurs.

Returning to the original criteria,

$$\sigma'_t = 0$$

$$\sigma'_r = P' > \sigma_0. \quad (10)$$

we need to find some simple way to approximate the stresses that develop around a dynamically loaded hole. One way to do this is to approximate the region around the wellbore with a thick-walled cylinder, where the borehole is the inner radius and the distance the wave has traveled is the outer radius. At some time, Δt , after ignition of the shot, the internal pressure is P' , as shown in Figure A-6, and the wave has reached the outer radius, b . The outer radius is a function of time given by

$$b(t) = a + C \Delta t \quad (11)$$

The boundary condition at $r = b$ is a zero radial displacement (u_r), that is, the material has not yet been affected by the

Figure A-6. Model Used to Approximate Stresses that Develop around a Dynamically Loaded Borehole

wave. The solutions for thick-walled cylinders are well known, and for this case, we get the stresses at the borehole wall as

$$\sigma'_t = P' \frac{[(1+\nu) - (1-\nu)b^2/a^2]}{[(1+\nu) + (1-\nu)b^2/a^2]} \quad (12)$$

and $\sigma'_r = -P'$, with $b = b(t)$. Compression is positive, and ν is Poisson's ratio.

We need to determine how accurate an approximation Equation 12 is to the correct solution. Figure A-7 shows the results of the tangential stress vs. time compared with (1) a step change and (2) a slower loading rate of the form

$$P = P_0 t e^{-t/b} \quad (13)$$

using the wave code, WONDY.⁹ This linear/exponential form is found to be a good approximation of the initial load on the rock in many tests. We can see from Figure A-7 that the thick cylinder calculation is an excellent approximation of the time when the tangential stress switches from compression to tension. We can include a factor of 1.5 in the time to bring these results more into agreement.

Using Equation 12, $\sigma'_t = 0$ when

$$(1 + \nu) = (1 - \nu)b^2/a^2 \quad (14)$$

or

$$b < a \sqrt{\frac{1 + \nu}{1 - \nu}} \quad (15)$$

Substituting Equation 11 and solving for Δt , we get

$$\Delta t = \frac{d}{2C} \left[\sqrt{\frac{1 + \nu}{1 - \nu}} - 1 \right] \quad (16)$$

Figure A-7. Comparison of Tangential Stress Versus Time for a Step Change and a Slower Loading Rate (Cylinder Approximation).

Now, our simple failure criterion states that P' must not reach σ_0 in less time than Δt , or

$$\frac{dp}{dt} = \frac{dp'}{dt} < \frac{4C\sigma_0}{3d \left[\sqrt{\frac{1+v}{1-v}} - 1 \right]} \quad (17)$$

where the 1.5 factor has been included. Let us generalize by assuming σ_0 is a yield stress determined from a static test, and we see that the rise time depends directly on the yield stress as well as the wave speed and diameter as before. It should be stressed again that the thick-wall cylinder approach is only an approximation of the true loading condition near the borehole, and the failure criterion may not be completely adequate. This does, however, point out the important parameters and shows why the loading rate is the important parameter for crushing.

In Situ Stress Effects ²— The failure analysis can be repeated with the in situ stresses, and we find that at the borehole wall, the failure criterion is given by

$$P' - (3\sigma'_{\min} - \sigma'_{\max}) > \sigma_0 \quad (18)$$

or

$$P' > \sigma_0 + 3\sigma'_{\min} - \sigma'_{\max} \quad (19)$$

To avoid hydraulic fracturing for this case, we want the time at which σ'_t just becomes equal to the minimum stress concentration, so that

$$3\sigma'_{\min} - \sigma'_{\max} = P' \frac{[(1+v) - (1-v)b^2/a^2]}{[(1+v) + (1-v)b^2/a^2]} \quad (20)$$

Solving for Δt again, we get

$$\Delta t = \left[\frac{d}{2c} \sqrt{\frac{1+v}{1-v}} \sqrt{\frac{1 - \frac{(3\sigma'_{\min} - \sigma'_{\max})}{P'}}{1 + \frac{(3\sigma'_{\min} - \sigma'_{\max})}{P'}}} \right] \quad (21)$$

If we divide the maximum pressure we can reach, given by Equation 19, by the time in Equation 21 and we convert to total stresses, we get

$$\frac{dP}{dt} < \frac{4C[\sigma_o + 3\sigma_{\min} - \sigma_{\max} - 2P_i]/d}{3 \left[\sqrt{\frac{1+v}{1-v}} \sqrt{\frac{1 - \frac{(3\sigma_{\min} - \sigma_{\max} - 2P_i)}{(\sigma_o + 3\sigma_{\min} - \sigma_{\max} - 2P_i)}}{1 + \frac{(3\sigma_{\min} - \sigma_{\max} - 2P_i)}{(\sigma_o + 3\sigma_{\min} - \sigma_{\max} - 2P_i)}}} - 1 \right]} \quad (22)$$

where we have again included the 1.5 factor. Conditions may exist where the denominator is negative. This simply implies that crushing will not occur for any loading rate under these conditions. Notice that the square-root term with the stresses is less than 1 and the stresses increase the effective yield stress so the result of the in situ stresses is to allow a higher loading rate. Thus, Equation 17 may be acceptable as a conservative guideline for most cases. Since for hard rocks σ_o becomes large and v becomes small, the acceptable maximum loading rate increases considerably. Soft rocks such as tuff are the worst case. Increasing pore pressure, as in overpressured reservoirs, facilitates crushing while underpressured zones will be hard to crush.

Comparison with NTS Results

These calculations are compared with eight tests performed in ash-fall tuffs at the NTS^{6,10}, where good stress and property data were obtained. Figure A-8 shows the loading rates for the eight tests plotted against C/d , which, from the

Figure A-8. Comparison of Loading Rates Versus $C/d(\text{s}^{-1})$ from Tests Conducted in Ash-Fall Tuff at the NTS with Calculated Criterion

developed criterion, appear to be the important parameter controlling the three regimes. Using Equation 6 and the estimated parameters shown in Table A-1,

Table A-1
Estimated Tuff Parameters

C	7000 ft/s
σ_0	7000 psi
$\sigma_{\max}$	1100 psi
$\sigma_{\min}$	800 psi
ν	0.3

we get for the hydraulic-multiple boundary,

$$\frac{dP}{dt} > 1500 \left(\frac{C}{d} \right) . \quad (23)$$

which is plotted in Figure A-8. Using Equation 22, we get, for the multiple-explosive boundary,

$$\frac{dP}{dt} > 6.8 \times 10^4 \left(\frac{C}{d} \right) . \quad (24)$$

The major discrepancy between these calculations and the NTS results is GF-7, where we observed crushing at a lower loading rate. However, it should be noted that GF-7 was conducted in a tuff much different from that of the rest of the tests. GF-7 was drilled into a much softer peralkaline tuff layer compared to the relatively competent tuff of the other shots. As an example, the permeability of GF-7 was three orders of magnitude higher than GF-4, -5, or -6. Certainly its moduli, strength, and wave speed were much lower, and the results would be shifted to the left.

It should also be pointed out that the measured pressure-loading rates are a lower bound on the loading rate the borehole wall experiences. The pressure transducer in these

tests is at the end of the fracture zone and the pressure wave undoubtedly attenuates before it reaches the transducer. For example, the pressure wave was observed in the rock 1.5 ft from the borehole (by a stress gauge) before the wave arrived at the pressure transducer. Obviously we should be very careful in extrapolating this data. The higher loading rates should have the largest errors.

FRACTURE LENGTH

We need to have some analysis to indicate the effect of rock properties and in situ stresses on the fracture length. A simple analysis is constructed here that shows the role of these parameters by using an energy approach since it is inherently simple. Essentially, the work in fracturing and dilating the cracks is equated to the energy released by the propellant.

Sneddon¹¹ gives the work necessary to dilate a 2-D fracture as

$$W_d = 2H \int_0^L \sqrt{1 - \frac{x^2}{L^2}} \left[\int_0^{W/2} P(x, \epsilon) d\epsilon \right] dx \quad (25)$$

where L is the crack wing length, W is the width at the wellbore, H is the crack height, ϵ and x are dummy variables in the width and length directions, respectively, and $p(x, \epsilon)$ is the pressure distribution. Since we do not know the pressure distribution inside the fracture, we can replace it with some equivalent constant pressure, $\bar{P}$, where

$$\bar{P} = f_1 (P - \sigma_{\min}) \quad (26)$$

where P is the pressure at the wellbore during fracturing. Using the relation

$$\epsilon = \frac{2(1 - \nu^2) L f_1 (P - \sigma_{\min})}{E} \quad (27)$$

we get

$$W_d = \frac{\eta \pi H L^2 (1 - \nu^2) f_1 (P^2 - \sigma_{\min}^2)}{E} \quad (28)$$

with η being the number of cracks.

The work in creating fracture surface is given by

$$W_f = \frac{\eta H L K_{IC}^2 (1 - \nu^2)}{2E} \quad (29)$$

where K_{IC} is the fracture toughness of the rock. The total work is now

$$W = W_d + W_f = \frac{\eta H}{E} (1 - \nu^2) \left[\pi f_1 L^2 (P^2 - \sigma_{\min}^2) + \frac{K_{IC}^2 L}{2} \right] \quad (30)$$

We can see immediately that the second term is negligible compared to the first term, and we will no longer consider it.

If we let K_p be the energy available from the burning propellant per unit length, then the total energy available is $H K_p$. Since much energy is wasted as heat, pressure losses, and useless strain energy (e.g., enlarging the borehole), the energy available for fracturing is

$$K = f_2 H K_p \quad (31)$$

where $0 \leq f_2 \leq 1$. Equating Equations 30 and 31 and solving for L , we get

$$L = \sqrt{\frac{f_2}{f_1}} \sqrt{\frac{EK_p}{\eta\pi(1 - \nu^2)(P^2 - \sigma_{\min}^2)}} \quad (32)$$

Finally, we can reasonably assume that f_2 is proportional to the difference between pressure and minimum stress. In high-stress rocks, any pressure below minimum stress level cannot do any useful work. Assuming simply that

$$f_2 = g(P - \sigma_{\min})/P, \quad (33)$$

where g depends on all the other loss terms as well, we find

$$L = \sqrt{\frac{g}{f_1}} \sqrt{\frac{EK_p}{\eta\pi(1 - \nu^2)P(P + \sigma_{\min})}} \quad (34)$$

This shows that high-modulus rocks will give longer fractures, a reasonable expectation since higher modulus rocks result in narrower cracks. The length is inversely proportional to the minimum stress level since much of the pressure must be used to overcome in situ stresses rather than crack the rock and dilate fractures.

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APPENDIX B

Core Logs

Figures B-1 through B-5 contain the core logs for each of the five test holes. Logs are shown from 20 ft to total depth (TD) and include notations for waxed samples (for core analysis), fractures, and significant lithologic changes. Generally, tunnel bed 5 rocks are fairly uniform over the thickness of the bed.

Table B-1 lists important core properties obtained from samples taken from the test holes as well as additional data obtained from nearby holes that penetrate tunnel bed 5. Permeabilities are dry Klinkenberg corrected values.

Figure B-1. GTSF No. 1 Core Log

Figure B-2. GTSF No. 2 Core Log

Figure B-3. GTSF No. 3 Core Log

Figure B-4. GTSF No. 4 Core Log

Figure B-5. GTSF No. 5 Core Log

Table B-1
Rock Properties

Depth (ft)	Dry Density (gm/cm)	Porosity (%)	Water Saturation (%)	Young's Modulus (10 ⁶ psi)	Poisson's Ratio	Compressive Strength (psi)	Compres- sive Wave Velocity (ft/s)	Shear Wave Velocity (ft/s)	Dry Permeability (mD)
21	1.39	42	99	-	-	-	-	-	3.5
25	1.46	38	91	0.37	0.14	3160	-	-	3.8
40	1.34	44	85	-	-	-	-	-	.07
40	1.58	34	93	-	-	-	-	-	3.7
32	1.27	48	94	-	-	-	-	-	0.22
35	1.52	37	86	0.13	0.15	1950	-	-	0.22

PROPERTIES FROM OTHER NEARBY HOLES

1383	1.39	39	-	0.45	0.17	5660	-	-	1.4
1381	1.73	49	96	0.13	0.10	1210	5460	2820	0.2
1394	1.86	36	92	0.58	0.27	3200	9090	5070	0.01
77	1.77	44	-	0.13	0.34	970	8440	4930	0.1
178	1.78	45	-	0.27	0.35	2060	7940	4270	<0.01
247	1.76	45	-	0.31	0.25	1560	8870	4970	0.18
1354	1.67	49	-	0.81	0.33	1110	6190	3450	0.18
1363	1.68	49	-	0.30	0.21	1230	5160	3130	0.01

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