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Preliminary Report on Prototype In-Situ Stress Testing
for the Yucca Mountain Project, Nevada

by

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CONVERSION FACTORS

For readers who wish to convert measurements from the metric system of units to U.S. customary units, the conversion factors are listed below.

Metric unit	Multiply by	To obtain U.S. customary unit
millimeter (mm)	0.03937	inch (in.)
centimeter (cm)	0.3937	inch (in.)
meter (m)	3.281	foot (ft)
kilometer (km)	0.6214	mile (mi)
square centimeter (cm ²)	0.1550	square inch (in. ²)
gram (g)	0.035	ounce (oz)
meganewton per square meter (MN/m ²).	145	pounds per square inch (psi)
megapascal (MPa)	145	pounds per square inch (psi)

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INTRODUCTION

Knowledge of in-situ stress is of fundamental importance in repository design considerations. Tuffs at Yucca Mountain are under pressures (stresses) that are caused by the weight of overlying rock (lithostatic load) and by regional tectonic stresses superimposed on the lithostatic load. These total stresses on the in-place rock are referred to as in-situ stresses. As part of the activities to assess the geomechanical properties of rock at Yucca Mountain, tests are planned to characterize in-situ stresses. Data on in-situ stresses are necessary input for thermal stress models and other models that are used to assess the stability of underground openings and to determine failure criteria for rock excavations. The prototype in-situ stress test described here has been performed at G-Tunnel, Nevada Test Site, where the rock conditions and stress state are similar to those that will be encountered during Exploratory Shaft Facility (ESF) testing at Yucca Mountain. Observations and results of prototype testing will refine equipment needs and test procedures to determine in-situ stresses in brittle, fractured, welded tuff.

Field work at G-Tunnel was performed during April to November 1989 and consisted of in-situ stress and deformation measurements and the description and documentation of the number, spacing, infilling, and orientation of fractures in recovered core and borehole walls. Laboratory work included instrument preparation and calibration, equipment preparation and testing, and tests on selected cores. Analyses consisted of data reduction and interpretation.

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We appreciate the assistance of John S. Teuscher, Colorado School of Mines, who prepared the core samples for testing and performed bulk density and unconfined compressive strength tests.

PURPOSE AND OBJECTIVES OF PROTOTYPE TESTING

The purpose of the activities planned under the prototype in-situ stress test is to test field procedures and instrumentation and to determine in-situ stresses under conditions very similar to those expected in the ESF.

The specific objectives of the tests are the following:

1. Determine in-situ stresses at three stations (boreholes) in G-Tunnel using the USBM borehole deformation gage, and determine the adequacy of the gage for use in the ESF.
2. Determine the deformation moduli of the rock mass using borehole dilatometer equipment and assess its adequacy for use in the ESF.
3. Evaluate the effect on drilling procedures when air circulation is used during the drilling for overcore stress-relief measurements.
4. Field test and verify the use of borehole camera systems for the inspection of boreholes and evaluate the suitability of these systems for Exploratory Shaft (ES) tests.
5. Develop or modify existing procedures and gain experience for such operations in highly fractured, welded tuff.

Detailed Test Description

Overcore stress-relief techniques relieve the stresses from a rock core and measure the change in size and shape of the core as the rock deforms as a result of the removal of stress. Rock expands different amounts in different directions, depending upon the orientation and magnitude of the stress field in the area from which the sample has been taken and its mechanical properties. The overcore stress-relief techniques are described briefly below.

Prototype Overcore Stress-Relief Test

The stress-relief testing was conducted in two phases. The first phase involves borehole dilatometer measurements in which a borehole dilatometer is used to determine the in-situ rock mass modulus of deformability by measuring the volumetric change of a dilatable membrane in contact with the borehole wall (Bourbonnais, 1985). These measurements are used in conjunction with core analysis and borehole TV camera logs to determine stress-relief measurement locations.

The second phase uses a deformation gage in overcored boreholes and consists of several steps. First, a small-diameter hole is drilled and a deformation gage is mounted in the hole. A large-diameter, thin-walled, core barrel and bit then is used to core around (overcore) the small hole. Overcoring relieves the stresses on the inner hole and the deformation gage

registers the changes in the size and shape of the wall of the inner hole as the rock deforms from the stress removal. After overcoring, the deformation gage is removed from the hole and the core is freed from the bottom of the hole and retrieved. The oriented core is then tested to determine the modulus of deformation of the rock. The modulus of deformation and strain measurements are used to calculate stress distributions normal to the axis of the borehole. To determine the orientation and magnitude of the stress tensor with the USBM gage method, it is necessary to have stress-relief data from at least three orthogonal (nonparallel) overcore holes. In order to sample similar rock-fabric conditions, overcore stress-relief holes and dilatometer core holes have been drilled parallel to each other.

Historical Perspective and Background

To obtain information on the present state of stress in fractured, welded tuff in G-Tunnel, the stress-relief technique has been selected as the primary method. This passive method of obtaining data on the in-situ stress state in shallow boreholes (<50 m or 160 ft) has witnessed steady progress in development and application, is firmly based in theory, is supported by extensive literature spanning more than 2 decades, and has met with general acceptance in the rock-mechanics community (Kim and Franklin, 1987). Its chief alternative is the active method of hydraulic fracturing.

Various techniques are available for monitoring stress changes in response to environmental influences (for example, Gloetzel cells, flatjack pressure cells, IRAD rigid-inclusion stressmeter), but they are not standard techniques for measuring the preexisting in-situ state of stress.

The three principal candidate instruments considered for use in the ESF are the U.S. Bureau of Mines (USBM) deformation gage (Hooker and Bickel, 1974), the Australian (CSIRO) hollow-inclusion cell (Worotnicki and Walton, 1976), and the doorstopper method (Leeman, 1969). The USBM and doorstopper methods both require three noncoplanar boreholes for a complete determination of the stress state; the CSIRO method requires only one borehole. The USBM and CSIRO methods of measurement require that the length of the recovered core be at least twice its diameter to allow for satisfactory stress relief and for testing of the core to determine its elastic properties. The CSIRO and doorstopper instruments must be cemented to the borehole wall, introducing potential difficulties in getting satisfactory bonding, and the bonding material could conceivably interfere with subsequent chemical testing of the rock. Both methods also require the use of numerically or experimentally derived geometrical or strain-coupling constants for data interpretation.

The USBM method was selected as the primary method for prototype testing, principally because of its inherent simplicity, relative stability, and long history of use in many rock types. The USBM gage is inserted and oriented in

the pilot hole, and because of its circuit design (full Wheatstone bridge), it is insensitive to heat-induced expansion in the pilot hole (Swolfs, 1982). The selection of the USBM or CSIRO method (displacement or strain measurement) will be based on the prevailing site and rock conditions in the ESF. The USBM gage will be the primary instrument, but the CSIRO cell may be used as an alternate instrument if stresses are sufficiently high, relative to the thermal response of the cell, to allow accurate measurements.

Biaxial Elastic Modulus Tests

The biaxial pressure-cell method of elastic-modulus testing (Fitzpatrick, 1962) is required to interpret the strain-relief data from the USBM gage or CSIRO cell. Most of the rock in the ESF is expected to be reasonably isotropic. If the rock is isotropic or nearly so, then this technique of modulus determination is adequate for data interpretation using standard methods. Its chief advantage is that the modulus measurements can be made in the field immediately after overcoring and performed at pressures in the biaxial device that approximate the stress level in the rock prior to overcoring. If biaxial testing reveals significant rock anisotropy, then the alternative of laboratory triaxial core testing on oriented samples cored from test overcores will be used to determine the required anisotropic elastic parameters (Aggson, 1977).

Petrology and Rock-Fabric Analysis

Microscopic rock-fabric analysis will supply detailed information on fracturing and mineral grains; that information will be used to evaluate variations observed in the elastic rock parameters needed for interpreting the overcoring data. Directional variation of intact rock modulus is caused primarily by microscopic fractures (McWilliams, 1966), which rock-fabric analysis will identify. The standard petrographic techniques that are used for rock-fabric analysis have no reasonable alternatives.

Prototype In-Situ Stress-Relief Project Drilling

A unique element of the overall testing program in the ESF at Yucca Mountain is the condition that the natural, unsaturated rock-mass environment be protected from artificially induced fluids that could compromise other planned hydrologic and geochemical tests. Therefore, all drilling associated with the USBM overcore in-situ stress-relief determination tests is scheduled to use air as the circulating medium for drilling. In preparation for testing at the Yucca Mountain ESF, a prototype testing project to evaluate the effects of using air circulation on drilling equipment (pilot hole and overcoring bits) and drilling parameters (penetration rates and hole alignment) was

initiated in the spring of 1989. These field-scale tests were conducted from April to November 1989 at G-Tunnel, Nevada Test Site, located 40 km northeast of Yucca Mountain. Rock mechanics test drifts in this existing underground testing facility, operated by Sandia National Laboratory, have been excavated in a sequence of welded and nonwelded tuffs with physical properties and in-situ states similar to those expected at Yucca Mountain (fig. 1).

Drill-Hole Layout

The prototype in-situ stress project studies conducted at G-Tunnel required the coring of drill holes with three different diameters. The dilatometer borehole-deformation test required the coring of a 3.0-in.* (NQ) drill hole. The conventional USBM in-situ stress-relief determination test requires the coring of two different diameter holes along the same drill-hole axis (Bickel, 1985; ISRM, 1987). First, a 1.5-in. (EX) diameter pilot hole is cored. The USBM deformation gauge is emplaced within the pilot hole. The gauge is then overcored using a 6-in. diameter narrow-kerf coring barrel and bit.

Figure 1 shows the location of the four prototype in-situ stress-relief project test sites within the G-Tunnel underground test facility. At each test site, a set of one NQ and one overcore stress-relief hole was drilled parallel to each other and approximately 3 ft apart. Table 1 gives the drill-hole site number, location, surveyed orientation (degrees, minutes, seconds), tunnel wall to total depth length, and drilling circulation medium (air or water). Surveying of drill-hole alignment and final hole orientation was done by Holmes & Narver, Inc., Mercury, Nev.

*Measurements of dimensions and rates of advance are listed in English units in this section and in the appendix to conform with standard drilling practices.

Figure 1.--Location of prototype in-situ stress test sites, G-Tunnel, NTS.

TABLE 1
Prototype In-Situ Stress Test
Drillhole Statistics

Borehole Number	G-Tunnel Location	Surveyed Orientation¹	Total Depth m	Circulation Medium
IS-1	Controlled Blasting Alcove	S 54°50'38" W 2 deg. up	15.24	air
IS-2	Demon- stration Drift	S 37°43'48" E 2 deg. up	14.66	air
IS-3	Demon- stration Drift	S 39°09'44" W 60 deg. up	15.24	air
IS-4	Laser Drift	S 83°03'49" W 3 deg. up	15.64	water

¹ Boreholes surveyed by Holmes and Narver

Drill-Site Lithology

The majority of drilling was confined to the densely welded tuff unit of the Grouse Canyon Member of the Belted Range Tuff of Miocene age. The Grouse Canyon Member in the area is approximately 12 m thick (Zimmerman and Finley, 1987). Drill holes at sites IS-1 and IS-2 (fig. 1), both drilled approximately horizontally, were started and terminated in the densely welded tuff. At IS-2, slightly lower in the welded tuff section than IS-1, there were several isolated pockets of rubble flow material consisting of rounded to subrounded lithic and volcanic clasts up to 1.5 in. in diameter. Site IS-3 holes (fig. 1), drilled at an angle of 60 degrees up with respect to IS-2, penetrated densely welded tuff, pockets of rubble, a thin (0.5 to 1-ft-thick), reddish-brown transition zone, and terminated within the overlying, less densely welded and bedded tuffs of the Paintbrush Tuff. Drill-hole site IS-4 in the Laser Drift (fig. 1) was begun and terminated within the same densely welded tuff as the other three sites, although it is slightly higher in the unit.

Air Versus Water Drilling Medium

The Site Characterization Plan (SCP) emphasizes that the Yucca Mountain test facility environment be protected, as much as possible, from artificially induced water that could compromise hydrologic and geochemical test data. Therefore, one of the goals of the prototype testing program was to evaluate the use of air, rather than water, as a drilling medium. NQ, EX, and OCH drill holes at sites IS-1, IS-2, and IS-3 were drilled using air and a dust-collector system. For comparison purposes, water was used as the drilling medium at the IS-4 site.

In general, the penetration rate for the continuous coring of NQ-diameter drill holes was slightly faster with water, although not significantly so. More time is involved with setting up and maintaining the air-drilling/dust-collection system than the water system. However, a factor such as drill-rig position within a drift (the small working space requires that drill strings be built or pulled from a drill hole one 5-ft-length at a time) can double the drilling time for a given hole. There was little to no observable difference in hole diameter consistency, wall smoothness, or core recovery between the two drilling fluids.

NQ Dilatometer Drill Holes

The NQ diameter drill holes were continuously cored using diamond impregnated coring bits. Appendix 1 contains lithologic logs compiled from NQ core at all four test sites (table 1). Fracture location information based on core, noted under the column labeled "Core Break" on the lithologic logs, indicates the highly fractured nature of the densely welded tuff. Fractures

that are predominantly oriented nearly perpendicular to the core axis and with ragged planes may be the result of the drilling process and/or core handling. Where there is a question of whether or not the fracture is naturally occurring, the break is labeled with a "db?" on the lithologic core log. Sections of core that are so broken that no single piece is a complete core cylinder are represented on the core logs with a hachured pattern and labeled "Broken Zone". Additional information on fracture frequency is provided in a following section on borehole TV camera analysis.

Review of Standard USBM Overcore Drilling Procedures

The following is a brief, generalized description of the drilling sequence and required hole specifications associated with the standard, and normally quite successful, USBM overcore stress-relief method (Bickel, 1985):

(1) Using a 6-in. diameter starter barrel with centered EX 1.5-in. diameter bit, a pilot hole is drilled into the working face until the starter barrel cuts about a 1/4-in. slot. The starter barrel is removed and the EX-size pilot hole extended approximately 5-7 ft with a double-tube EX core barrel, diamond bit, and reaming shell. The pilot-hole core is inspected to determine drill-hole wall condition and fracture plane locations. This information determines the positioning of the USBM borehole gauge. Overcore stress-relief test zones should ideally have an intact length of two-to-three times the diameter of the larger 6-in. diameter hole.

(2) If a satisfactory test zone of sufficient length is not located, the larger 6-in. diameter hole is advanced by coring with a 18-to-24-in. long, thin-walled masonry core barrel and diamond bit. At least two to three trips into and out of the hole are usually necessary to completely remove the EX-cored pilot hole. This procedure of EX pilot-hole coring, followed by larger diameter hole advancement continues until a suitable test zone is found. Often, rock in front of a potential test zone must be removed so that the entire stress-relief test zone length of 12 to 18 in. can be overcored in one continuous overcoring run.

(3) Following the overcore drilling of the test zone, the core is removed for biaxial deformation modulus testing. These steps are repeated as necessary for each successive measurement along a given drill hole.

Problems Encountered During G-Tunnel Prototype Testing

Site configuration

Figure 2 shows an example of typical drill-rig alignment, perpendicular to the tunnel axis, associated with overcore stress-relief testing in

Figure 2.—Specialized tricone hole-opener bit. Centered "stringer" uses EX pilot hole as a guide.

underground workings. This drill-hole alignment dictates that the drill strings be extended or withdrawn one 5-ft section at a time. Figure 3 shows the dust-collection system utilized during prototype testing. The system consists of a mounted drill-hole collar, removable steel frame for holding rubber membranes snugly around the drill rod, and flexible vacuum hoses for removing cuttings from the drill hole. The steel frame had to be unbolted and the rubber membranes removed each time the drill string was pulled for bit changes or to recover core. Therefore, when working under these conditions, and especially when working time is limited, it is crucial that the drilling operation be streamlined in order to eliminate any unnecessary drill string handling.

EX pilot hole

The first stress-relief test hole (IS-2) was designed to be 50-ft long and inclined 3 degrees up from the horizontal. This hole was cored using EX coring bits from various manufacturers. The bits exhibited various port configurations and contained premium grade surface-set diamonds or were diamond-impregnated with different matrices. All bits penetrated well for approximately 1 yd, with an average penetration rate of 1 in./min. or faster. However, after a meter of coring with air, the bits became "polished" and penetration rates rapidly dropped to 3 ft/hr. Another major problem resulting from bit polishing was that little or no usable EX core could be recovered, even when using a new bit and double-tube core barrel. Core inspection, therefore, could not be used to provide information on drill-hole wall conditions (cavities) and fracture locations. This information is critical in determining gauge positioning. In rock masses where intervals of intact rock suitable for testing are few, the ability to locate all possible test zones is extremely important to the success of the stress determination effort.

Larger diameter overcore holes

The highly fractured, brittle, and heterogeneous nature of the welded tuff and the presence of volcanic debris rubble-zone pockets required the removal, by coring, of up to 10 ft of untestable rock between stress-relief test zones. Core barrels used to advance the larger diameter hole encountered the same diamond-polishing problems as the EX-size bits. In order to proceed with hole advancement in a timely manner, new core barrels were used on every other rock-removal run. Stop-gap remedies, such as dressing the bit with a brick, only increased penetration rates for a short period of time. Reduced bit life required the purchasing of more core barrels than originally expected.

Figure 3.--View of centered EX pilot hole and flat working face produced by the specialized tricone hole-opener bit.

When following the suggested overcore drilling procedures, the larger diameter overcore hole is advanced by coring. The length of the core runs is dictated by the length of the core barrels, normally 2-3 ft. A BQ-size drill string, supported every 10 ft by a rotating stabilizer, drives the coring barrel. However, reduced air velocity within the large annulus behind the core barrel resulted in a build-up of cuttings along the floor of the drill hole. Additionally, because air is a poor lubricating medium, the increased cutting buildup around the stabilizers caused friction between the stabilizers and drill-hole wall, which resulted in considerable drill-string binding and chatter. Binding and chatter increased with drill-hole length. To eliminate chatter, drill-rig RPM's were reduced until eventually a satisfactory penetration rate could not be produced, even with a new barrel. The use of four 1/8-in. diamond reaming spacers, welded to the core barrel and stabilizers, failed to solve the cutting buildup problem.

Modifications of Drilling Procedures

EX pilot-hole drilling

A key element in the evolution of the drilling procedure modifications was the use of a small-diameter borehole TV camera. The ability to log and record on VCR tape the EX borehole wall allowed the investigators to carefully survey the pilot hole and accurately determine the nature and location of discontinuities. Elimination of the requirement for EX core recovery allowed the EX coring bit to be replaced with an EX plug bit, and a 50-ft-long pilot hole could be drilled in one continuous setup. Polishing of plug-bit diamonds did occur but did not slow penetration rates to the degree experienced with the coring bits.

Larger diameter overcore hole advancement

As stated earlier, removing long sections of unsuitable rock between test zones in highly fractured rock is commonly necessary. Removing rock with the core barrels was unacceptable from the standpoint of time and expense. Drilling out the annulus with standard tricone bits was not possible in the horizontal holes because the weight of the heavy bit caused the hole to deviate. This, in turn, caused the EX pilot hole to become off center, which is unacceptable for overcore stress measurements. To resolve this hole-deviation problem, a 1-ft-long EX-diameter drill rod (stinger) was centrally attached to a tricone hole-opener bit (fig. 4). This specialized bit produced the following positive results:

(1) The larger diameter hole could be advanced as far as necessary along the existing EX pilot hole in one continuous setup.

Figure 4.--Double string setup used to overcome drill string binding and chatter. Photograph shows BQ drive rod attached to core-barrel expander head nested inside of stationary NQ-diameter support stabilizer.

(2) The cuttings produced by the tricone bit were small enough to be easily removed from the hole by the dust-collection system or blown out with air. The removal of broken core and rock fragments from the larger diameter borehole, with a core barrel shovel, can be time consuming and difficult. Clogging of the EX pilot hole can be a problem, especially in vertical down holes. The specially designed hole-opener tricone bit pulverizes the rock, and the stinger protects the EX hole from clogging with debris. Therefore, overcore drilling with air circulation in vertical down holes should be feasible to the depths desired.

(3) The stinger follows the EX pilot hole during the larger hole advancement drilling. Figure 5, a view at the back of the larger diameter overcore hole, shows the perfectly centered EX hole and a flat working face resulting from the use of the stinger-mounted tricone bit.

(4) Only one modified tricone bit was needed to complete the hole advancement drilling required for a 50-ft-long test hole. The fabrication cost for one of the tricone bits was only slightly more expensive than the cost of one narrow-kerf impregnated coring bit.

Drill-string binding and chatter problems encountered during larger hole advancement and USBM overcore stress-relief test drilling were solved by nesting the rotating BQ drive rods inside of stationary NQ rods equipped with centering stabilizers. Figure 5 shows this double string arrangement. In practice, the entire set of BQ drive rods are encased within NQ-sized rods. NQ centering stabilizers are inserted into the NQ string at 10-ft intervals. For additional stability, a NQ rod stabilizer can be positioned next to the core-barrel expander head so that only a maximum of 18 in. of rotating BQ rod is exposed at the end of an overcore run. Overcore runs completed to a depth of 50 ft drilled smoothly at RPM's high enough to produce the desired penetration rates of 1-in./min.

Summary and Recommendations

The adaptations to the recommended standard drilling procedures used for USBM overcore stress determination fall into three distinct areas:

(1) Drilling of the entire EX pilot hole in one continuous run, followed by an initial hole survey to identify the precise number and location of all possible test zones. If the survey shows insufficient test zones at depths where measurements are needed, the hole can be immediately extended and/or abandoned and relocated. Where rock conditions are poor, this procedure can save a great deal of time and expense.

Figure 5.-- View of borehole TV camera monitor showing fracture measuring gage.

The key element to this procedural change is the use of a small-diameter borehole TV camera or, less preferably, a borescope. The TV camera log is more precise in measuring the location of fracture planes and other borehole discontinuities. In highly fractured rock, this is even more critical because test zones are short in length and few in number. During our prototype studies, if gauge placement had been solely dictated by core recovery, the entire test would have been abandoned.

(2) Use of an EX pilot-hole-guided tricone bit for removal of large intervals of broken and unsuitable rock from the larger diameter hole proved to be extremely beneficial. The ability to advance the larger diameter hole without repeated pulling of the drill string meant a substantial saving in working time and bit cost. Coring and drilling in both horizontal and vertical down drill holes, in highly fractured rock, are more manageable because the rock mass can be pulverized and easily removed with air suction or water. This change of method and equipment eliminates some problems that can limit the depth to which overcore stress-relief tests can be performed in horizontal and vertical down holes.

(3) Use of rotating drive rods, nested within a set of stationary centering rods, eliminated most drill-string binding and chatter problems and damage to the borehole wall. The double-barrel system is easily built or pulled in manageable increments. The ability to drill smoothly at high RPM's is another major factor enabling the successful completion of overcore stress-relief tests along horizontal drill holes.

Borehole TV-Camera Logging

Borehole TV-Camera Systems

Borehole TV-camera surveys evolved into an integral part of the prototype in-situ stress-testing project. By the completion of the fourth overcore stress hole, two different camera systems, a Westinghouse TV-1253 and a Rees R-93, were being employed. The Westinghouse camera was used to inspect the EX pilot hole because the camera body is slightly smaller than the Rees camera body. The Rees camera system was used to inspect the NQ and larger diameter overcore holes. The components of the two systems are essentially the same and consist of a camera body with interchangeable lenses; transmission cable; camera control unit for remotely adjusting light intensity, focus, and rotation; and a black and white TV monitor. A VCR recorder, adaptable to both camera systems, was added to make the system more useful and to meet project documentation requirements.

Two different types of lenses, an axial and a side scan, were employed with each system. The axial lens provided a view directly down the axis of the borehole and the side-scan lens rotated through a 360-degree scan showing

a closeup of the borehole wall. In a 7.6-cm-diameter borehole, the side-scanning lens shows a rectangular viewing field of approximately 5 x 3.8 cm.

Core Inspection Versus Camera Logging

Recovered core is valuable in that it provides specimens for detailed composition analysis, microfracture, and physical-property testing. However, it is not very helpful in the determination of fracture plane position and character of other rock mass discontinuities. Even where competent core is recovered, it is often difficult to determine if a fracture is naturally occurring or has been caused by drilling and/or core handling. Histograms (see figs. 15 and 16) comparing the foot-by-foot fracture frequency determined from recovered NQ core and observed with a borehole TV camera are presented in the discussion of prototype dilatometer testing. If any core is lost during coring and/or handling, it is impossible to tell the exact location of fracture planes within the borehole. Dip angles for fractures can be measured on recovered core segments, but their true orientation within the rock mass is unknown. Additionally, the fracture aperture width cannot be determined.

Borehole TV-camera surveys can determine the precise location and orientation of a fracture plane ellipse, a very important factor in doing overcore stress measurements in highly fractured rock. Detailed surveys can also determine true fracture orientation and aperture. Viewing of broken zones within the borehole can often determine the reason for poor core recovery. The zone may consist of multiple, closely spaced fractures, the intersection of multiple nonparallel fracture planes, or a highly fractured fault zone. If the rock mass is highly variable in composition and has inclusions, it is sometimes possible to orient recovered core by matching up unique features and/or structures observable in the core and the TV-observed borehole wall.

Overcore Pilot-Hole Surveys

The implementation of a small-diameter borehole TV camera was a key element in the adaptation of drilling procedures that made it possible to continue the overcore in-situ stress analysis project in highly fractured rock. After the initial 15-m-long EX pilot hole was drilled, the TV borehole camera, employing both axial and side-scan lenses, was used to closely inspect and review (by means of VCR tape) the borehole wall conditions. From these camera logs, it was possible to graphically plot the location of fracture planes and other discontinuities, such as vugs and broken zones. From this plot, the investigators can determine at the onset all possible overcore stress-measurement zones along the length of the hole. Because of the highly fractured nature of this rock, measurement zones are generally few in number and may be concentrated in very short lengths of the borehole.

NQ Borehole Surveys

Borehole camera surveys of the NQ-sized dilatometer deformation test holes provided information on the location, orientation, frequency, and aperture width of fractures. In addition, detailed axial and side-scan monitoring showed the location of open vugs, breakout zones, and large open fractures that could damage the dilatometer membrane and/or affect the deformation test results.

Results of In-Situ Stress Measurements

The results of the prototype in-situ stress measurements (table 2) were disappointing because of an insufficient number of measurements and the confusing variation of the calculated secondary principal magnitudes and orientations of stress. Several factors contributed to this outcome, the most important of which is the intense fracturing of the brittle tuffs which severely limits the potential zones of measurement. Fracturing of this intensity can create significant stress anomalies and concentrations (Amadei and Goodman, 1982; Lemos and Brady, 1983) that may obscure the regional stresses. In addition, alteration, vugs, and soft lithic fragments interfered with the placement and stability of the gage during overcoring, resulting in unsatisfactory measurements. At this time, it is not known whether or not the USBM instrument will be the primary stress gage for the ESF investigations at Yucca Mountain. In order to determine stresses in the highly fractured rock, it may be necessary to employ an additional type of gage, possibly the CSIRO hollow-inclusion instrument. Another reason for the unsatisfactory stress measurements is that our main emphasis was placed on first establishing optimum drilling procedures including the location of all satisfactory test intervals in this difficult rock, sometimes to the detriment of stress measurements. After the standard USBM drilling technique was changed, the number of successful measurements increased. As we noted above, six successful measurements out of eight overcore runs were obtained from the last hole (IS-1). These modified drilling procedures were described in the previous section.

Satisfactory stress measurements are those that displayed a "normal" stress-relief curve, as shown in figure 6. Unsatisfactory stress measurements had erratic, incomplete, or otherwise abnormal stress-relief curves. An example of an unsatisfactory measurement is shown in figure 7. Even the calculated stresses for the unsatisfactory measurements vary considerably, both in magnitude and direction (table 2). There is no immediately apparent cause for the range of stresses measured in IS-1. The causes of these variations may be differences in rock properties (tables 3-8), effects of excavation, and natural (i.e., tectonic) stress variations. It should be

TABLE 2
Secondary Principal Stresses
in Plane of Measurement

Overcore #	Depth m	P MPa	Q MPa	Inclination of P to vertical	Remarks
IS1-1	3.58	3.8	-2.2	39°	In disturbed zone
IS1-2	8.53	8.6	5.4	71°	
IS1-3	9.35	5.0	2.7	10°	
IS1-4	9.96	1.7	1.2	84°	
IS1-5	10.54	10.8	5.4	41°	
IS1-6	12.27	7.8	4.0	13°	
IS2-1	1.85	16.6	3.9	74°	
IS4-4a	8.64	8.3	1.1	66°	

noted that P' , the maximum principal stress, is inclined at a moderate to high angle to the vertical. This preliminary finding is in general agreement with the published results of other investigators (Hooker, V.E., Aggson, J.R., and Bickel, D.L., written commun., 1971; Haimson, 1983). These authors determined that the maximum horizontal stress is approximately 8 MPa and the vertical stress is approximately 7 MPa (Haimson, 1983) in G-Tunnel at 400-m depth. The ratio of maximum to minimum principal stresses in IS-1 averages 1.8 when the initial measurement is discounted.

In general, overcore stress-relief measurements in non-welded tuffs are more consistent than those, like ours, that are made in fractured, welded tuff. It is obvious that more measurements will be needed to obtain an acceptable understanding of stresses in the welded tuff.

We are currently investigating the amount and cause of anisotropy in the rock matrix and its possible effect on rock stresses by testing small (1 3/8 in.) oriented cubes taken from rock at the USBM gage-sensor locations. Fabric elements have been studied in oriented thin sections and compared with the recorded strain magnitudes and directions.

Figure 8 is a typical biaxial pressure test from hole IS-1. The data available show that there is little variation of modulus with direction. The average modulus for the tests in IS-1 is 2.64×10^6 psi (ranging from 1.24×10^6 to 3.44×10^6 psi). Several overcore samples were retested (table 3) and, in general, the modulus increased with successive tests, probably because of fracture closure. Inspection of the core and a review of our field notes revealed that, at several locations, fractures, vugs, and lithic fragments are present in the core near the plane of the stress-relief sensors. In some cases, an abrupt jump in the strain-relaxation curve coincided with the overcore bit intersecting these features. This would usually abort the overcoring operation. In addition, when the core separates along such structures, it is impossible to test it in the standard biaxial test.

The effect of fractures on the rock-mass modulus can be estimated with the NQ borehole dilatometer, but only volumetrically in an adjacent borehole. Therefore, if fractures are important causes of abnormal stress variations, we will have to employ a different gage in some fractured intervals to avoid the bias inherent in sampling only the larger blocks.

Another disappointment was our lack of sufficient measurements to determine the 3-dimensional stress field, one of the prototype study goals. The necessary additional hole was not drilled because all prototype activities were suspended when G-Tunnel was closed in December 1989.

Overcore Stress Relief Plot
Drill Hole IS-2, Demonstration Drift, G-Tunnel

Figure 6.--Stress-relief curves showing the typical response of the USBM deformation gage during a "normal" overcore test.

Overcore Stress Relief Plot
Drill Hole IS-2, Demonstration Drift, G-Tunnel

Figure 7.--Stress-relief curves of an unsatisfactory overcore test. The annular core containing the USBM deformation gage apparently broke in pieces during the test.

TABLE 3
Deformation Moduli of IS-1 Overcores,
Biaxial Compressive Loading to 6.9 MPa,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Overcore #	Depth m	E _{ave} GPa	E _{min} GPa	E _{max} GPa	Date
IS1-1	3.58	21.59	17.38	26.62	10/18/89
IS1-1	3.58	25.17	22.14	29.52	12/14/89
IS1-2*	9.03	23.86	20.07	28.41	12/05/89
IS1-3	9.35	11.17	5.17	21.10	10/25/89
IS1-3	9.35	11.10	5.10	18.41	12/13/89
IS1-3	9.37	13.03	5.10	23.59	01/28/90
IS1-6	12.27	14.41	8.55	23.72	11/02/89
IS1-6	12.27	14.55	9.45	23.45	12/15/89
IS1-6	12.24	14.21	9.66	22.48	01/29/90
IS1-7	14.25	16.00	14.06	18.28	12/12/89
IS1-8	14.50	18.83	15.24	21.79	12/11/89

* Annular core located between OC-2 and OC-3

TABLE 4
As-Received Bulk Densities of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
0.47	9.43	4.74	1.99	166.38	367.97	2.21
1.30*	9.85	4.77	2.06	175.65	379.69	2.16
1.93	9.94	4.50	2.21	158.33	364.20	2.30
2.99	9.96	4.52	2.20	160.08	358.15	2.24
3.09	10.22	4.52	2.26	163.71	368.00	2.25
3.51+	9.97	4.51	2.21	159.17	350.45	2.20
3.60	10.02	4.52	2.22	160.69	350.40	2.18
3.84	9.98	4.51	2.21	159.33	360.41	2.26
4.01	10.01	4.51	2.22	159.62	358.30	2.24
4.16	10.10	4.51	2.24	161.15	366.05	2.27
4.38	9.93	4.50	2.21	158.26	364.90	2.31
4.53	9.56	4.51	2.12	152.62	349.25	2.29
4.63	9.28	4.52	2.05	148.65	338.75	2.28
4.74	9.95	4.51	2.21	158.94	366.95	2.31
4.85	9.56	4.51	2.12	152.53	350.25	2.30
4.96	9.95	4.52	2.20	159.38	368.23	2.31
5.07	10.27	4.52	2.27	164.69	380.75	2.31
5.18	9.69	4.51	2.15	154.96	359.05	2.32
5.27	9.35	4.51	2.07	149.44	345.41	2.31
5.65	9.83	4.51	2.18	157.28	367.15	2.33

* No P & S velocities measured

+ No S velocities measured

TABLE 4 (con'd)
As-Received Bulk Densities of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
5.74	10.01	4.51	2.22	159.71	369.65	2.31
5.95	10.14	4.51	2.25	161.79	375.60	2.32
6.21	10.20	4.51	2.26	163.20	375.71	2.30
6.36	10.08	4.51	2.24	160.74	361.55	2.25
6.47	9.97	4.51	2.21	159.08	366.80	2.31
7.10	10.13	4.51	2.25	162.18	365.85	2.26
7.27	10.11	4.50	2.25	160.95	372.05	2.31
8.14+	9.99	4.50	2.22	158.86	349.50	2.20
8.31	10.14	4.49	2.26	160.42	364.05	2.27
8.41	9.73	4.48	2.22	153.33	345.80	2.26
8.51	9.74	4.49	2.17	154.62	351.95	2.28
8.62	10.14	4.51	2.25	162.06	364.31	2.25
8.71	10.14	4.52	2.24	162.79	365.15	2.24
8.86	10.10	4.50	2.24	160.88	366.25	2.28
8.96	10.06	4.51	2.23	160.69	366.90	2.28
9.06	10.07	4.50	2.24	160.40	367.85	2.29
9.19	9.65	4.51	2.14	154.06	351.81	2.28
10.37	10.06	4.52	2.23	161.42	358.05	2.22
10.68	10.12	4.51	2.24	161.65	369.50	2.29
11.07	10.12	4.50	2.25	161.20	365.70	2.27

+ No S velocities measured

TABLE 4 (con'd)
As-Received Bulk Densities of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
11.21	10.14	4.51	2.25	162.06	367.31	2.27
11.51	10.01	4.51	2.22	160.16	366.40	2.29
11.65	10.11	4.50	2.25	160.86	364.31	2.26
11.83+	9.73	4.51	2.10	151.76	340.91	2.25
11.95	9.73	4.50	2.16	155.07	356.05	2.30
12.17	10.23	4.50	2.27	162.95	371.95	2.28
12.28	9.94	4.50	2.21	158.33	359.45	2.27
12.37	9.91	4.51	2.20	158.12	357.12	2.26
12.43	9.68	4.50	2.15	154.27	347.10	2.25
12.75	10.04	4.50	2.23	159.92	358.30	2.24
13.46	10.27	4.51	2.28	163.95	360.50	2.20
13.70	10.19	4.51	2.26	162.77	367.05	2.26
13.88	10.69	4.51	2.37	170.47	38203	2.24
14.04	9.64	4.51	2.14	154.16	347.10	2.25
14.80	9.59	4.51	2.13	153.36	336.85	2.20

+ No S velocities measured

TABLE 5
As-Received Bulk Densities of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
0.82	10.10	4.74	2.13	178.14	410.30	2.30
1.22	8.87	4.74	1.87	156.47	363.15	2.32
1.31	10.11	4.74	2.13	178.36	402.55	2.26
1.52	9.81	4.73	2.07	172.32	401.40	2.33
2.44	9.97	4.74	2.10	175.97	406.80	2.31
2.56	9.98	4.75	2.10	176.89	397.45	2.25
2.83	10.12	4.75	2.13	179.39	410.70	2.29
3.05	9.78	4.74	2.06	172.63	400.82	2.32
3.57	9.50	4.74	2.00	172.63	386.70	2.24
3.84	9.82	4.75	2.07	173.36	398.30	2.30
3.96	9.83	4.74	2.07	173.48	398.85	2.30
4.30	9.96	7.74	2.10	173.48	406.79	2.34
4.39	9.92	4.75	2.09	174.21	407.21	2.34
4.75	9.72	4.75	2.05	172.21	397.35	2.31
4.94	10.01	4.75	2.11	177.39	408.75	2.30
5.03	10.08	4.76	2.12	179.38	410.45	2.29
5.24	9.86	4.75	2.08	174.80	398.59	2.28
5.36*	9.96	4.75	2.10	176.57	400.50	2.27
5.88	9.98	4.75	2.10	176.78	413.15	2.34
6.00	10.30	4.77	2.16	183.99	421.40	2.29

* No P & S velocities measured

TABLE 5 (con'd)
As-Received Bulk Densities of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
6.31	9.96	4.75	2.10	176.44	411.15	2.33
6.43	9.85	4.74	2.08	173.74	402.25	2.32
6.85*	9.88	4.73	2.09	173.60	387.52	2.23
6.95	9.97	4.74	2.10	175.90	404.50	2.30
7.35	9.99	4.75	2.10	177.06	404.90	2.29
7.47	9.95	4.77	2.09	177.88	402.85	2.26
7.65	9.78	4.76	2.06	174.11	397.02	2.28
7.99	9.67	4.75	2.04	171.34	390.42	2.28
8.23	9.95	4.77	2.09	177.77	403.45	2.27
8.35	10.10	4.75	2.13	178.98	395.80	2.21
8.50	10.03	4.75	2.11	177.74	403.55	2.27
9.66	10.00	4.74	2.11	176.41	401.55	2.28
9.91	9.52	4.77	2.00	170.16	385.88	2.27
10.18	10.21	4.75	2.15	180.93	419.20	2.32
10.30	10.11	4.75	2.13	179.15	412.25	2.30
10.67	10.02	4.76	2.10	178.31	411.75	2.31
10.85	9.58	4.76	2.01	170.42	393.57	2.31
11.06	9.96	4.78	2.08	178.70	402.60	2.25
11.22	9.68	4.74	2.04	170.76	393.25	2.30
11.31	9.77	4.75	2.06	173.09	398.22	2.30

* No P & S velocities measured

TABLE 5 (con'd)
As-Received Bulk Densities of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Length cm	Diameter cm	L/D	Volume cc	Weight gm	Density gm/cc
12.31	10.00	4.76	2.10	177.90	415.70	2.34
12.59	9.99	4.75	2.10	179.98	423.70	2.39
12.86	10.03	4.77	2.10	179.24	427.70	2.39
12.98	9.47	4.76	1.99	168.59	403.75	2.39
13.17	10.01	4.75	2.11	177.38	427.38	2.41
13.75	9.80	4.77	2.05	175.16	405.45	2.31
14.11	10.07	4.75	2.12	178.45	419.45	2.35
14.20	9.72	4.75	2.05	172.21	410.70	2.38
14.39*	10.10	4.75	2.13	178.98	421.30	2.35
14.54	9.47	4.77	1.99	169.28	396.60	2.34

* No P & S velocities measured

TABLE 6
Dynamic Moduli of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
0.47	2.21	3.70	2.01	0.54	0.29	23.04
1.93	2.30	3.76	2.52	0.67	0.09	31.84
2.99	2.24	3.59	1.90	0.53	0.31	21.19
3.09	2.25	3.72	2.19	0.59	0.23	26.55
3.60	2.18	4.02	2.54	0.63	0.17	32.91
3.84	2.26	3.95	2.47	0.63	0.18	32.54
4.01	2.24	3.82	2.35	0.62	0.20	29.69
4.16	2.27	4.04	2.60	0.64	0.15	35.29
4.38	2.31	3.97	2.54	0.64	0.16	34.58
4.53	2.29	3.97	2.60	0.65	0.13	34.99
4.63	2.28	4.11	2.78	0.68	0.08	38.06
4.74	2.31	4.14	2.66	0.64	0.15	37.59
4.85	2.30	4.17	2.56	0.61	0.20	36.18
4.96	2.31	4.25	2.72	0.64	0.15	26.53
5.07	2.31	4.22	2.71	0.64	0.15	39.02
5.18	2.32	4.11	2.62	0.64	0.16	36.95
5.27	2.31	4.07	2.63	0.65	0.15	36.75
5.65	2.33	4.24	2.75	0.65	0.14	40.17
5.74	2.31	3.64	2.52	0.69	0.04	30.51
5.95	2.32	4.16	2.58	0.62	0.19	36.75

TABLE 6 (con'd)
Dynamic Moduli of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
6.21	2.30	4.04	2.63	0.65	0.13	35.95
6.36	2.25	3.80	2.60	0.68	0.06	32.25
6.47	2.31	3.89	2.66	0.68	0.06	34.65
7.10	2.26	3.87	2.58	0.67	0.10	33.10
7.27	2.31	4.23	2.34	0.55	0.28	32.38
8.31	2.27	3.85	2.57	0.67	0.10	32.98
8.41	2.26	3.81	2.42	0.64	0.16	31.71
8.51	2.28	3.87	2.54	0.66	0.12	32.95
8.62	2.25	3.71	2.44	0.66	0.12	31.01
8.71	2.24	3.68	2.45	0.67	0.10	29.58
8.86	2.28	4.05	2.67	0.66	0.12	36.41
8.96	2.28	4.04	2.64	0.65	0.13	35.91
9.06	2.29	4.17	2.49	0.60	0.22	34.64
9.19	2.28	4.13	2.89	0.70	0.02	38.85
10.37	2.22	3.78	2.47	0.65	0.13	30.61
10.68	2.29	3.81	2.82	0.74	-0.11	32.42
11.07	2.27	3.75	2.47	0.66	0.12	31.02
11.21	2.27	3.81	2.53	0.66	0.10	31.97
11.51	2.29	3.68	2.68	0.73	-0.06	30.92
11.65	2.26	3.65	2.39	0.65	0.12	28.92

TABLE 6 (con'd)
Dynamic Moduli of IS-1 Core,
Controlled Blasting Alcove,
G-Tunnel, NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
11.95	2.30	3.81	3.20	0.84	-0.67	15.54
12.17	2.28	3.96	2.56	0.64	0.14	34.07
12.28	2.27	3.81	2.48	0.65	0.13	31.55
12.37	2.26	3.72	2.52	0.68	0.08	31.00
12.43	2.25	3.77	2.47	0.66	0.13	31.02
12.75	2.24	3.67	2.48	0.68	0.08	29.76
13.46	2.20	3.78	2.49	0.66	0.11	30.28
13.70	2.26	3.90	2.61	0.67	0.10	33.87
13.88	2.24	3.85	2.54	0.66	0.11	32.08
14.04	2.25	3.91	2.48	0.63	0.16	32.10
14.80	2.20	3.90	2.48	0.64	0.16	31.39

TABLE 7
Dynamic Moduli of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
0.82	2.30	3.72	2.41	0.65	0.14	30.46
1.22	2.32	4.02	2.45	0.61	0.21	33.70
1.31	2.26	3.96	2.57	0.65	0.13	33.74
1.52	2.33	3.97	2.59	0.65	0.13	35.32
2.44	2.31	3.73	2.70	0.72	-0.05	32.00
2.56	2.25	3.33	2.19	0.66	0.12	24.17
2.83	2.29	3.63	2.40	0.66	0.11	29.28
3.05	2.32	3.97	2.72	0.69	0.06	36.39
3.57	2.24	3.96	2.68	0.68	0.08	34.75
3.84	2.30	3.79	2.73	0.72	-0.04	32.91
3.96	2.30	3.76	2.54	0.68	0.07	31.75
4.30	2.34	3.83	2.66	0.69	0.04	34.44
4.39	2.34	3.74	2.76	0.74	-0.10	32.09
4.75	2.31	3.83	2.73	0.71	-0.01	34.09
4.94	2.30	3.91	2.66	0.68	0.01	32.87
5.03	2.29	3.89	2.66	0.68	0.06	34.35
5.24	2.28	3.81	2.70	0.71	0.00	33.24
5.88	2.34	4.21	2.87	0.68	0.06	40.86
6.00	2.29	4.04	2.75	0.68	0.05	36.37
6.31	2.33	3.66	2.70	0.74	-0.10	30.57

TABLE 7 (con'd)
Dynamic Moduli of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
6.43	2.32	3.63	2.53	0.70	0.03	30.59
6.95	2.30	3.65	2.50	0.68	0.06	30.48
7.35	2.29	2.90	2.11	0.73	-0.06	19.17
7.47	2.26	3.15	2.36	0.75	-0.13	21.90
7.65	2.28	4.07	2.78	0.68	0.06	37.36
7.99	2.28	3.52	2.76	0.78	-0.29	24.66
8.23	2.27	3.57	2.58	0.72	-0.05	28.71
8.35	2.21	3.76	2.31	0.61	0.19	28.07
8.50	2.27	3.46	2.36	0.68	0.06	26.80
9.66	2.28	3.17	2.38	0.75	-0.14	22.21
9.91	2.27	3.58	2.86	0.80	-0.39	22.65
10.18	2.32	3.19	2.02	0.63	0.16	21.96
10.30	2.30	3.60	2.58	0.72	-0.03	29.70
10.67	2.31	3.19	2.33	0.73	-0.07	23.33
10.85	2.31	3.96	2.63	0.66	0.14	36.43
11.06	2.25	3.46	2.34	0.68	0.08	26.61
11.22	2.30	3.22	1.99	0.62	0.19	21.68
11.31	2.30	3.46	2.57	0.74	-0.12	26.74
12.31	2.34	3.08	2.03	0.66	0.12	21.60
12.59	2.39	4.21	2.97	0.71	0.01	42.59

TABLE 7 (con'd)
Dynamic Moduli of IS-4 Core,
Laser Drift, G-Tunnel,
NTS, Nevada

Depth m	Density gm/cc	Vp km/sec	Vs km/sec	Vs/Vp	PR	E GPa
12.86	2.39	4.38	3.07	0.70	0.02	45.95
12.98	2.39	4.06	3.31	0.82	-0.49	26.71
13.17	2.41	4.08	2.75	0.67	0.09	39.73
13.75	2.31	4.14	2.79	0.67	0.08	38.84
14.11	2.35	4.06	2.93	0.72	-0.05	38.33
14.20	2.38	4.15	2.70	0.65	0.11	38.52
14.54	2.34	3.53	2.53	0.72	-0.03	29.06

TABLE 8
Unconfined Compressive Strengths
of Core Samples from IS-1 and IS-4

BOREHOLE IS-1		BOREHOLE IS-4	
Depth (m)	Strength (MPa)	Depth (m)	Strength (MPa)
0.47	65.14	0.82	-
1.30	16.75	1.22	94.39
1.93	86.00	1.31	63.85
2.99	9.31	1.52	86.71
3.09	56.81	2.44	65.51
3.51	40.46	2.56	48.55
3.60	53.09	2.83	56.29
3.84	73.72	3.05	100.72
4.01	82.84	3.57	89.85
4.16	73.60	3.84	75.74
4.38	84.73	3.96	63.09
4.53	102.36	4.30	94.64
4.63	78.80	4.39	90.03
4.74	99.24	4.75	71.80
4.85	71.14	4.94	94.48
4.96	87.36	5.03	-
5.07	89.70	5.24	64.62
5.18	103.41	5.36	60.76
5.27	86.94	5.88	96.75
5.65	120.48	6.00	107.0
5.74	104.87	6.31	80.16
5.95	95.06	6.43	85.31
6.21	101.52	6.85	38.13
6.36	44.24	6.95	66.00
6.47	82.18	7.35	66.28

TABLE 8 (con'd)

Depth (m)	Strength (MPa)	Depth (m)	Strength (MPa)
7.10	87.4	7.47	69.28
7.27	84.83	7.65	80.20
8.14	43.20	7.99	62.97
8.31	95.25	8.23	52.07
8.41	65.20	8.35	56.89
8.51	93.69	8.50	36.10
8.62	74.69	9.66	77.95
8.71	74.36	9.91	41.55
8.86	78.63	10.18	50.15
8.96	95.57	10.30	64.07
9.06	75.56	10.67	54.93
9.19	88.27	10.85	—
10.37	79.89	11.06	50.81
10.68	95.57	11.22	47.60
11.07	81.09	11.31	81.05
11.21	76.53	12.31	37.67
11.51	66.05	12.59	68.25
11.65	67.65	12.86	92.86
11.83	46.26	12.98	77.89
11.95	68.76	13.17	85.61
12.17	98.90	13.75	43.42
12.28	54.62	14.11	68.33
12.37	65.01	14.20	101.44
12.43	68.15	14.39	78.71
12.75	—	14.54	67.28

TABLE 8 (con'd)

Depth (m)	Strength (MPa)	Depth (m)	Strength (MPa)
13.46	62.52		
13.70	-		
13.88	74.86		
14.04	79.54		
14.80	69.15

Drill hole: IS-1

Overcore #: 8

Depth: 47 ft.-7 in.

Figure 8.--A typical biaxial pressure test from hole IS-1.

It is important to the aims of the project to have the capability of obtaining stress estimates in and adjacent to fracture zones and faults. For example, a more adequate stress study should include sampling stresses on either side of the Ghost Dance fault as an aid in forecasting movement potential.

ROCK FABRIC AND AUTHIGENIC MINERALOGY

This section provides a petrographic description of the mineralogy and rock-fabric features of the welded Grouse Canyon Member samples that were tested. Mineralogy and rock fabric are important factors that may influence physical properties.

The central portion of the Grouse Canyon Member examined in the Laser Drift is a densely welded, devitrified tuff containing phenocrysts of sanidine, anorthoclase, quartz, and pyroxene in decreasing abundance (Connolly and others, 1983). Lithic fragments are devitrified tuff fragments with potassium feldspar phenocrysts. Shardy material is more deformed or strained around phenocrysts and lithic fragments.

Microfractures and their mineralized surfaces were studied using a Cambridge 2250 Mark 2 scanning electron microscope (SEM) with an attached energy dispersive system (EDX). Microfracture orientations in the Grouse Canyon commonly are perpendicular and parallel to the plane of welding and are abundant around stress concentration points of phenocrysts (fig. 9A). This latter feature may explain the scatter in the microfracture orientation data; at least four trends with average directions of N. 75° W., N. 29° W., N. 3° E., and N. 70° E. have been determined from a limited number of microfractures (fig. 9B) (Diehl and others, 1990).

Authigenic mineral phases commonly seal microfractures, which indicates precipitation of these minerals from fluid moving through the microfractures. Insufficient amounts of fracture-coating minerals were available for quantitative analysis; therefore, only qualitative descriptions are provided. Backscatter imagery on the SEM shows two generations of authigenic minerals filling microfractures—an early iron, titanium, and rare earth mineral coating the walls of the microfractures, and a later potassium feldspar filling the centers (fig. 10).

Figure 9.— A, Microfractures commonly radiate from stress-concentration points of phenocrysts, which may account for the scatter in orientation data; B, average microfracture directions.

Figure 10.—Backscatter imagery shows two generations of mineral fill in a microfracture. An early iron and rare earth mineral coats the walls of the fracture and a later potassium feldspar mineral fills the center.

The microfractures are believed to be extensional in origin. A profile of shut-in pressures from hydraulic-fracture testing in two vertical holes near the GTUF site (Warpinski and others, 1981) indicates that the least horizontal stress magnitude in the welded Grouse Canyon Member is half as much as in the adjacent nonwelded units. Thus, the welded unit may be in a state of extension, probably due to lateral spreading in the more ductile nonwelded tuff.

Although the Grouse Canyon Member is currently in the unsaturated zone, the hydrous mineral phases that coat fracture surfaces and seal microfractures indicate that fluid movement has occurred along these structures. This shows that the microfractures were once well-connected flow paths for fluid movement.

PROTOTYPE DILATOMETER TESTING

Instrument Description

The Probex-I Dilatometer (fig. 11), purchased from Rocktest, Inc., is a high-pressure borehole dilatometer that measures deformation of the rock mass under applied pressure. It consists of a high-pressure dilatable membrane, a dual-action hydraulic module, and a measuring module that contains a linear-variable differential transformer (LVDT). The LVDT is fixed to the "up-hole" end of the dual-action piston and follows its displacement through the loading and unloading cycles. Incremental piston displacement is converted to volume change by a calibration factor supplied by the manufacturer. Hydraulic fluid (water) is pumped into the membrane when confined in either a borehole or a calibration cylinder and the change in pressure is measured by the pressure transducer. Pressure- and volume-change data are displayed on the PRO-R2 readout unit and recorded in a notebook. These data are used to construct pressure vs. volume plots (fig. 12) for each test.

The outer diameter of the dilatable membrane is 2.9 in.* and the expandable portion is 18 in. long. The pressure capacity of the membrane is a function of the borehole diameter: 4,350 psi in ≤ 3.125 -in.-diameter holes; 3,000 psi in ≤ 3.25 -in.-diameter holes. Dilatometer tests are not run in boreholes that are greater than 3.25 in. in diameter to prevent the rupturing of the membrane.

*Measurements of dimensions and pressures are listed only in the text in English units to conform with standard field practices.

Figure 11.—Schematic representation of the Probex-1 dilatometer system with PRO-PR2 readout unit.

Figure 12.--Schematics of a pressure-volume curve.

The dilatable membrane is fairly stiff and an internal pressure of approximately 100 psi is required to begin to expand it (membrane inertia), but to fully engage and compress a nominal 3-in.-diameter borehole, a pressure of about 500 psi is required. Therefore, pressure-volume plots begin at a pressure of 500 psi. Field calibrations are performed before and after each set of borehole measurements.

Laboratory Experiments

Calibration experiments on the Probex-I Dilatometer (fig. 13) began in February 1989. The Probex-I Dilatometer is customarily calibrated by using a thin-walled steel cylinder with internal diameter of 3.000 ± 0.025 in. To aid in the performance assessment of this borehole instrument, a thick-walled aluminum test cylinder was obtained that measures 36 in. long, 9 in. outside diameter, and contains a coaxial cavity 3 in. in diameter. The calibration procedures and computations were as prescribed in the manual provided by the instrument manufacturer.

Initial tests in the steel and aluminum cylinders yielded results that were not satisfactory because the volume intake was considered to be too great (fig. 14, lower part). The instrument was drained of its pressurizing fluid (water) and the dilatable membrane was removed to inspect the internal parts of the instrument. Some water was observed in the upper well of the membrane beyond the O-ring seal on the up-hole side. Moreover, a shallow but persistent helical groove was observed along the up-hole portion of the steel core that crossed over the O-ring seal. Suspecting this to be a possible source for leakage, the instrument was returned to the manufacturer on 3/8/89 for inspection and repair of the machining flaw.

The dilatometer was returned on 3/23/89. Inspection of the steel core revealed that the helical groove had been ground off, except at the rims of the O-ring seal. During tests on the instrument, the manufacturer noted some loss of fluid (<1 cc) but claimed that this small amount would not adversely affect the performance of the instrument. We repeated all tests as before (fig. 14, upper part) and again observed small amounts of water (~2 cc) where there should be none. The manufacturer agreed to inspect and test the instrument at no cost to us except for transportation. Before shipping the device on 4/21/89, all O-rings initially supplied by the manufacturer on the steel core were replaced by a new batch purchased in Denver. These new O-rings, although of the same dimensions as the "old" ones, appear to be somewhat stiffer.

Figure 13.--Probex-1 (NX) dilatometer (in box) showing expandable membrane (standing upright) and aluminum (thick wall), and steel (thin wall) calibration cylinders.

Figure 14.--Comparison between calibrations in steel and aluminum cylinders at various dates - 2/24, lower diagram - 4/4 and 7/21, upper diagram.

The manufacturer subjected the dilatometer to three sets of tests: membrane tests on a manufacturer-supplied steel core, steel-core test in a steel model of the membrane, and rubber membrane/steel core tests of the kind performed numerous times before by both parties. No leaks were detected in any of these tests. Moreover, routine tests found the instrument to be in calibration. The dilatometer was returned on 7/17/89. We repeated our series of tests (fig. 14, upper part) and observed no leaks. Evidently, the new O-ring batch may have provided an effective seal.

The calibration experiments in both metal cylinders, performed in the laboratory and summarized in figure 14, show an improvement in both response and reproducibility since the initial tests in February 1989. However, work will continue to improve the performance of this instrument; for example, by finding ways to increase the effective stiffness of the pressurizing fluid. In the meantime, field tests at G-Tunnel were scheduled for the week of August 14, 1989.

Field Testing at G-Tunnel

The Probex-I Dilatometer was used to make a series of measurements in the G-Tunnel complex during the period from August 14-17, 1989, at site IS-4 in the Laser Drift, and from November 27-30, 1989, at site IS-1 in the Controlled Blasting Alcove. These tests were an integral part of the prototype in-situ stress-testing program. Information on the two measurement sites is summarized in table 1.

The horizontal NQ holes were core-drilled and, consequently, were reasonably smooth-walled. The recovered cores were logged to establish a profile of fracture frequency along the length of the holes. A borehole video camera was used to map the locations and abundance of fractures along the walls of the boreholes (figs. 15 and 16). The measurement sites in the boreholes were initially set at 3-ft intervals starting at 48-ft depth. However, based on camera sightings of large vugs, cavities, and open fractures, any of which could cause damage to the dilatometer membrane, the individual measurement site was moved up- or down-hole to a more favorable location.

The Probex-I Dilatometer, along with all essential mechanical and electronic equipment, was taken into the G-Tunnel complex a day in advance to allow adjustment to site-specific atmospheric and temperature conditions. The following day, the dilatometer was prepared for service and calibrated in the steel cylinder. Distilled water was used as the pressurizing fluid. Failure or rupture of the dilatable membrane during borehole measurements would require replacement of the membrane unit and recalibration of the entire device. A discussion of the field calibrations of the dilatometer will be presented later in this section.

Figure 15.--Observed fracture frequency, drill hole IS-1, Instrumentation Alcove, G-Tunnel.

Figure 16.--Observed fracture frequency, drill hole IS-4, Laser Drift, G-Tunnel.

A typical plot of pressure versus volume change at one location in borehole IS-1 is shown in figure 17. At each measurement location, two consecutive pressure cycles were run in 500 psi increments: (1) from 500 to 3500 to 1000 psi and (2) from 1000 to 4000 to 500 psi. Depending on the amount of hysteresis observed in the two pressure cycles, the measurement would have been repeated to improve the results. The maximum pressure reached during the pressure cycles was a function of the initial borehole diameter at the measurement location. Because the holes were core-drilled, the diameters were well within tolerances along the lengths of both holes except at highly fractured rubble zones.

During each pressure cycle at a measurement location, the data displayed in the PRO-R2 readout unit was manually recorded on a data sheet shown in figure 18. The slope of each cycle in the pressure-volume curve (fig. 17) was determined by linear regression and recorded on the computation sheet shown in figure 19. The slope of the second pressure cycle (fig. 17) was usually steeper and considered representative of the deformation modulus at each measurement location. This information and the results of the calibration tests are then used to compute the deformation modulus, E_R .

The results of the dilatometer measurements in boreholes IS-1 and IS-4 are shown in figure 20. The distribution of the deformation moduli with depth is highly variable, as might be expected in a fractured rock mass. The dependence of the deformation moduli on fracture frequency is shown in figure 21. In this plot, the deformation moduli are normalized by an assumed rock-mass modulus of 3×10^6 psi and show an overall decrease with increasing number of fractures per meter. This relationship can be further emphasized as shown in figure 22 by averaging the X and Y values in each of the five intervals of fracture frequency (0-5, 5-10, etc.) of figure 21. The curve fit to the averaged data represents the constitutive equation proposed by Duncan and Goodman (1968) for an equivalent continuum model of a regularly fractured rock mass:

$$E_a/E = (1 + \lambda * E/kn)^{-1} \quad (1)$$

where E_a is the apparent deformation modulus, E is the assumed rock-mass modulus, λ is the fracture frequency, and Kn is the normal stiffness of the fractures. The relationship observed in figure 22 is of qualitative value only; the quantitative aspects of these measurements are very much in question for the reasons outlined in the next section.

Figure 17.--Pressure-volume curve for dilatometer test at 30 ft depth in borehole IS-1, Demonstration Drift Alcove, G-Tunnel, NTS, Nevada.

[illegible]

Figure 18.--Data-recording sheet for Prototype ISS test - Borehole dilatometer.

PROBEX-1 DILATOMETER TEST - PROTOTYPE ISS - G-TUNNEL - NTS - NEVADA

DATE: / / ; TIME: ; OPERATORS:

HOLE: ; DEPTH:

Linear Regression - HP-15C

Data Input: cc - ENTER; psi - Σ +; B - Σ f(L.R.); A - $\Sigma x^2 < y$; r - $\Sigma f(y, x) < y$

	Cycle #1	Cycle #2	Cycle #3	Cycle #4	Cycle #5
B					
A					
r					

Modulus Calculation

Poisson's ratio - <1>

k = 0.01815 cc/unit

v_o = 2006 cc

a = x 10⁻³ cc/psi

b = 7.014 x 10⁻⁴ cc/psi

c = a - b = x 10⁻³ cc/psi <2>

$\Delta v / \Delta P_b$ = x 10⁻³ cc/psi <3>

v_m = cc <4>

$$E_m = 2(1 + \frac{\text{<1>}}{\text{<4>}})(2006 + \frac{\text{<3>}}{\text{<2>}}) \frac{1}{(\text{<3>} - \text{<2>}) \times 10^{-3}} = \text{.....} \times 10^6 \text{ psi}$$

Figure 19.--Data-reduction sheet for the Probex-1 dilatometer test - Prototype ISS, G-Tunnel, NTS, Nevada.

Figure 20.--Variation of deformation modulus with depth in drill hole IS-1 (solid line) and drill hole IS-4 (dashed line).

Figure 21.--Modulus-reduction factor as a function of fracture frequency for the Grouse Canyon tuff, G-Tunnel, NTS, Nevada.

Figure 22.—Same as figure 19. X and Y values are averaged in each of the five cells ($\lambda = 0-5$, $5-10$, $10-15$, $15-20$, and $20-25$). Curve is best fit of equation (1) to averaged data.

Discussion, Conclusions, and Recommendations

The calibration factor identified as $\langle \epsilon \rangle$ in figure 17 enters into the calculation of the deformation modulus of the rock mass as a first-order parameter and the accuracy and precision of this parameter becomes increasingly acute in high-modulus rocks. The results of the calibration tests performed at the field sites have been less than satisfactory as can be seen in figure 23. The scatter observed in this plot is far too great to gain acceptable confidence in the field measurements of deformation modulus. Several factors that, by themselves or in concert, degrade the level of confidence in the field results have been identified: (1) gas dissolved in the pressurizing fluid or trapped in the membrane reducing the apparent stiffness of the system, (2) leakage across ineffective seals, and (3) unknown or poorly understood characteristics of the three main parts of the dilatometer. The latter refers to the anelastic behavior and resiliency of the membrane material and its response to pressure cycling, the nature of membrane inertia, and the stability of the displacement and pressure-monitoring devices and their associated electronic circuits at sites outside the laboratory.

With the closure of the G-Tunnel complex on November 30, 1989, it was no longer possible to pursue the prototype work on the dilatometer. The results obtained to date probably underestimate the rock-mass modulus by 30 percent or more. A comparison with the modulus values obtained in the biaxial loading tests of annular overcores (fig. 24) confirms this conclusion.

This section concludes with the following recommendations:

- o Calibration procedures and experiments should be optimized to increase the accuracy and precision of the calibration factor.
- o Stiffness of the membrane assembly should be increased by reducing its effective volume; that is, fill the membrane cavity with solid material such as rods or beads.
- o A low-volume pump or pressure intensifier should be incorporated into the system to increase the ease and efficiency of the calibration and borehole tests.
- o Real-time data acquisition equipment should be part of the package used in the field.
- o Performance and evaluation tests of the dilatometer system should be performed in a material of known modulus, such as a 36-in. or greater ingot of aluminum that incorporates a central 3-in. diameter hole.

Figure 23.--Calibrations of the Probex-I dilatometer done underground at G-Tunnel, NTS, Nevada.

Figure 24.--Comparison of modulus values obtained in (A) the biaxial loading tests of annular overcores and (B) dilatometer tests in an NQ borehole at site IS-1.

LABORATORY MEASUREMENTS

Laboratory testing was done to provide supplementary data useful in the evaluation of field measurements. The following properties were measured: as-received bulk density, ultrasonic compressional- and shear-wave velocity, and unconfined compressive strength. Bulk density values and velocities were used to calculate dynamic elastic moduli. Test specimens were selected from the NQ diameter (4.5 cm) core recovered from drill holes IS-1 and IS-4. Only core specimens that met, after end-milling preparation, the ISRM and ASTM required length-to-diameter ratio of 2:1 were selected. The testing sequence for an individual specimen was as follows: (1) Core samples were dry cut using a trim saw. (2) Specimen ends were milled to ISRM required parallelism. (3) Specimens were weighed and length and diameter measured. (4) Ultrasonic compressional- and shear-wave velocities were determined. (5) Specimen was loaded to failure in unconfined compression.

Test specimen requirements dictate that samples be selected from the more competent sections of the core. Therefore, the values listed in tables 4-8 cannot necessarily be considered to be representative of the highly fractured rock mass at G-Tunnel. Tests were conducted using ISRM and ASTM standards.

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APPENDIX I

NQ CORE LOGS

CORE LOG SYMBOL EXPLANATION

57	Measured fracture dip angle with respect to core axis.
db	Core fractures interpreted to result from drilling and/or core handling. Typically the break is approximately perpendicular to core axis and generally is not planar.
db?	Indicates fractures that may be natural but that have characteristics similar to drilling or core handling induced fractures.
Broken zones. Core is highly fractured welded tuff and no intact cylinders of core are present.
Rubble zone. Composed of generally nonwelded debris trapped between welded and nonwelded tuff units.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-1 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-1 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-1 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-1 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-1 HOLE SIZE: NQ (3.0 inch) ORIENTATION: HORIZ.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-2 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-2 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-2 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-2 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-2 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-3 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-3 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-3 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-3 HOLE SIZE: NQ(3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-3 HOLE SIZE: NQ ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-4 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-4 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-4 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-4 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.

PROTOTYPE INSITU STRESS TEST (1.2.4.1.5.1 G)

HOLE #: IS-4 HOLE SIZE: NQ (3.0 inch) ORIENTATION: Horiz.