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EFFECTS OF ROCK PROPERTIES ON EXPLOSIVE SOURCE MODELING: PRELIMINARY RESULTS

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ABSTRACT

The purpose of this study was to investigate possible reasons for unexpectedly low ground motion measurements obtained during some events at the Nevada Test Site. Of particular interest were the unexpected low results obtained for the Mission Cyber event which was the first test conducted in the P-tunnel complex.

If the measurements made do not agree with the results predicted by computations, then either the computations or the measurements are incorrect. This report examines the possibility that the error could lie with the computations. A second report entitled "An Examination of the Experimental Results Obtained from Mission Cyber and Disco Elm: Are They Different?" will discuss the possibility that the fault could be with the measurements.

We use a three-part approach in the report to examine the possibility of an error existing in the computational predictions of ground motion. We first used WONDY (a Lagrangian, finite difference, hydrodynamics code) to predict the velocities measured in model tests conducted at SRI in both tuff and Indiana Limestone. The computer code used did not incorporate a damage model, which would serve to weaken the material once subjected to high level shock, nor did it incorporate strain rate effects which would elevate the properties above those measured in static laboratory testing. In spite of these deficiencies in the code, the results predicted for the tuff agreed quite well with the results obtained at SRI. The results obtained from the simulation of the response of the Indiana Limestone were not in as good agreement as those obtained from the tuff.

The second aspect of the report involved an examination of the media in which the tests in question were conducted by comparing the properties and quality of the rock at the Disco Elm and the Mission Cyber sites. We also compared the quality of the rock located in the P-tunnel complex with tuff located at other sites within the NTS. The reason for these comparisons was to determine if there were aspects associated with the rock at either of the event locations that were not taken into account in the ground motion predictions. The results of these examinations were very informative. At all locations that we examined there were sections of the tuff that were of near zero strength, sandy in appearance, and very friable. In vertical core holes approximately 40% of the section was of this nature. In horizontal core holes there was less variation in the material properties but some very weak sections did exist - especially in the holes from the N-tunnel complex. The examination also revealed the presence of joints within the core but these were not large in thickness nor extent and were judged to be of little significance. The material properties used within the codes to predict ground motion are based on static tests conducted on core samples from the stronger parts of the exploratory holes. That is to say, the very weak material is of insufficient strength to make test samples from. The properties used in the code therefore do not reflect the true properties of the entire section but only those of the strongest parts of the section. This could prove to be of importance in computer simulations of ground motion.

The third approach taken in this report was to utilize the WONDY code to predict the effects on ground motion that the presence of open joints and layers of very weak material might have on computed results. The results of an open joint were found to be extremely severe. For a situation of a 1-kt event it was found that a gap of 5.3 cm located 68 m from the charge would result in a loss of 60% of the peak particle velocity. The core examinations however indicated that it was unlikely for such open joints to exist. The results from our investigations of the presence of

very weak layers in an otherwise strong test bed showed that such possibilities should be included within computer predictions. Our calculations show, for example, that a layer of 1.33 m of weak rock at a location of 68 m from ground zero of a 1-kt event could result in a loss of 25% of the peak particle velocity. Our examination of the core at the test site showed the presence of layers of weak rock of up to 20 m in extent at the N-tunnel complex - but not in the vicinity of the Mission Cyber or Disco Elm events in the P-tunnel complex.

The conclusion of the report is that even though the presence of weak layers could result in grave errors in the computer-predicted response to a nuclear explosion that this was not the case in the Mission Cyber/Disco Elm case. The material around both the Mission Cyber and the Disco Elm events was extremely competent rock - out to ranges well beyond where all of the instrumentation was placed. The material properties, as determined from static testing of the core from those gage locations, should reflect the situation that existed and the computer predictions obtained should be accurate.

The presence of this very weak material probably should not be ignored as it has been in past calculations. Better means are needed to test this very weak material to determine the properties that should be used in the codes. These tests should include some at higher strain rates to determine if these materials are extremely strain rate sensitive. It is also important to conduct small scale dynamic tests in models which contain weak layers to see if the code predictions are accurate for this type of material.

1. Objectives

This study was motivated by the measurement of unexpectedly low free field ground motion (acceleration) and stress levels from certain nuclear tests conducted at the Nevada Test Site (NTS). Many events conducted at the NTS are instrumented to aid in predicting the response of the media to shock as well as to verify yield. In most cases a pre-test prediction of the expected ground response is carried out well in advance of the test for the purposes of certifying containment. When the ground motions measured during the event are not in relative agreement with the values predicted it is cause for great concern.

Of particular interest in this study were the unexpected results obtained from the Mission Cyber event. Mission Cyber was the first of three tests conducted in P tunnel by the Defense Nuclear Agency (DNA). Mission Cyber, Disco Elm, and Distant Zenith were all conducted in the P-tunnel complex within a few hundred meters of one another. Accelerometers and stress gages fielded on the Mission Cyber event by Sandia National Laboratories (SNL) indicated that peak free field stresses and velocities were lower than predicted by previous "nominal" events by factors of two to three while peak accelerations were lower by a factor of nearly ten. The ground motion measurements from the Disco Elm and Distant Zenith events, on the other hand, were as expected. An understanding of the cause of anomalous results in nuclear testing is essential because of the need to accurately calculate expected ground motions for containment and verification purposes.

Individuals we talked to from both DNA and SNL indicated that the device yield for Mission Cyber was nominal and that the ground motion gage records were reliable. The reason or reasons for the energy not coupling well with the tuff at the working point are not known. There were rumors as to evidence being available - other than the lower than expected ground motion - which would substantiate that poor coupling occurred. These rumors were: (1) the measured seismic signals were lower than expected for the device yield as determined by radiochemistry and other evidence; (2) portions of the line-of-sight (LOS) tunnel exhibited less crushing than would have been expected for nominal yield; and (3) seismic refraction surveys conducted in the LOS and bypass drifts near the working point for Mission Cyber prior to the test required greater amounts of explosive to produce seismic signals than expected. On the other hand, there were also rumors that other indicators were such that the coupling of the energy to the tuff was good. These were: (1) measurements of the pressure and fluid flow in the LOS pipe were as expected close to the working point; and (2) the permanent displacements measured after going back into the tunnel near the site were as would be expected for the device yield.

Other underground tests have displayed anomalous ground motion results so Mission Cyber is not unique. Possible causes for obtaining anomalous behavior in the past (whether the test was conducted in alluvium or in tuff) have included questions on having misjudged the saturation at the working point, having made an error on the percentage of air-filled voids in the rock section, or possibly having a different mineralogy than perceived.

If the measurements made in the field do not agree with the computer predictions then one of two possibilities exist. Either the problem is with the computation or with the measurements. If the calculations are the source of the problem there are many possibilities as to what could cause the error. For example, the codes being used could be inadequate in their physics, the wrong material properties might

have been used as input to the code, the rock could be different from the rock that was assumed to be present in the test bed, or the geometric model being used by the code to approximate the rock mass could be in error. Reasons for the measurements being lower than expected could include the presence of a rock type other than what had been expected, improper use of the instruments, or simply poor performance of the instruments.

We have chosen to look closely at both of these possibilities. This report will present results of a study of the problems that could lead to the computation results being incorrect. A second report entitled "An Examination of The Experimental Results Obtained from Mission Cyber and Disco Elm: Are They Different?" will cover the possibility of the measurements being in error.

2. Introduction

Ground motion resulting from shock loading is well described by H.C. Rodean [1] starting from the time of the explosion and ending with distant seismic signals. Rodean examines inelastic response of the rock to nuclear explosions; the thermodynamics, stress wave dynamics, equations of state for the materials; the elastic wave propagation; and the attenuation of seismic signals.

The pressures that a rock experiences from a nuclear explosion range from 100 GPa very near the site to stresses well below the elastic region in the far field, and consequently, the material properties dictating the behavior of the rock must be known over 12 orders of magnitude in pressure (SI units). The properties that must be known to calculate rock response include Hugoniot data from 200 GPa to the elastic stress, static and (possibly) dynamic failure surfaces, the elastic pressure of the rock, the pressure at which the voids are eliminated (crush pressure), the water saturation level, the amount of air-filled voids, the density, and the bulk sound speed. Obtaining a complete set of properties is a formidable task since rocks are generally inhomogeneous, anisotropic, and vary even within a given stratigraphic section, as well as being composed of many stratigraphic sections.

What is really needed to permit accurate ground motion predictions by the codes is not known. The following items are a list of technical issues we felt to be important. These issues were identified after reading the pertinent literature and having discussions with experts in the field. These are: (1) What are the most important rock properties needed to understand propagation and attenuation of waves induced from dynamic high energy source? (2) Are the results of dynamic experiments in the 10^7 to 2.0×10^9 Pa range needed or are statically determined properties adequate. (3) If dynamic data are needed, must the tests be conducted under confining stresses? (4) What are the effects of joints and weak layers on wave attenuation? (5) How do we use the results from small scale static and dynamic experiments to predict full scale response? (Are there specimen size effects?) and (6) How important are saturation, gas-filled porosity, and in-situ stresses in code predictions?

These questions formed the background for the direction that our study took. In order to answer some of these questions we formulated an approach that consisted of three parts. The first was to use a typical hydrodynamics code to calculate the response of small-scale particle velocity experiments conducted by Nagy and Florence [2] on P-tunnel tuff. Our logic was that if the current codes are valid we should be able to predict accurately the results of such small-scale, well-controlled experiments. We also wanted to see how well we could predict similar results obtained in a less porous rock. Miller and Florence [3] conducted similar tests in Indiana Limestone. The use of the code required certain properties as input for both of the rock types being investigated. From the literature we compiled Hugoniot data, failure surface data, elastic and crush pressure values, and other physical parameters to generate an input file for WONDY. We used this hydrodynamics code, written by SNL, to simulate the experimental data.

A qualitative analysis of core from P and N tunnels and from Pahute Mesa was the second part of our investigation. The reason for evaluating the core was to get an idea of the quality and strength of the tuff throughout the stratigraphic section above the working point in P tunnel. What we found was very enlightening. Namely that there are thick sections in which the P- and N-tunnel tuffs are very weak, not competent, and of poor quality in terms of the rock's ability to propagate

stress waves. In all, two vertical cores were examined from P tunnel near the Mission Cyber and Disco Elm events, one vertical core from N tunnel near the Misty Echo event, and one vertical core from Pahute Mesa near the Towanda event. In addition, core from a horizontal hole near the Mission Cyber working point was examined as was core from a horizontal hole near Misty Echo.

The third part of our investigation involved using WONDY to calculate (using the properties for Indiana Limestone) the effects that an open joint or bedding plane might have on stress wave propagation. The situation where a layer of weak material is located between two stronger materials was also investigated. The intent in both of these cases was to determine the attenuation in the outgoing wave as it propagates across these discontinuous layers. Thicknesses of the layers used in the computations were several millimeters (corresponding to several meters when scaled to a one kiloton test).

The results of these three approaches are discussed in the following sections. The final section of the report gives our recommendations for continuing work.

3. Response of Porous NTS Tuff to Explosive Loading

We wanted to examine a representative hydrodynamics code and determine if the rock properties used as input to the code to predict ground motion were realistic or unrealistic. Over the years a number of different codes have been used to predict rock motion from an explosive source. We wanted to know if these input parameters (the rock properties) made sense from the standpoint of the results obtained in static (or dynamic) laboratory testing or if the input parameters were used simply because they predicted the correct (or nearly correct) ground motions.

The code that was used to predict the rock response was WONDY V, a one-dimensional, finite-difference wave propagation code developed and maintained by SNL. The code integrates the one-dimensional Lagrangian differential equations of motion by use of finite difference analogs. The geometry employed can be rectilinear, cylindrical or spherical. The method of artificial viscosity is employed to solve problems involving the propagation of pressure discontinuities, or shock waves. Equations of state and constitutive relations for a number of classes of materials are provided as options within the code. Solids are assumed to obey the Mie-Gruneisen equation of state and can be treated as either hydrodynamic or elastic-plastic. For the elastic-plastic case, plastic flow occurs on the basis of the von Mises yield criterion where the yield strength can be infinite, constant or variable. Equations of state for ideal gasses, detonation high explosives, and initially porous or distended solids are also provided. In the calculations conducted for this report, an equation of state was utilized which uses the p-alpha model for air-void crush up, and the shear strength is defined as a function of mean stress. The code is quite popular and additional details can be obtained from user manuals which are available from SNL.

WONDY was run in the spherical geometry mode - even though the tests were conducted in a cylinder. The charge was spherical and, until stress waves reach the boundaries and begin to reflect back into the model, the spherical case and the experiment are identical. Once reflection begins to occur, the results of the two begin to depart radically.

The code calls for information on the rock which includes density of the uncompressed solid, bulk sound speed, Gruneisen parameters, shear modulus, initial distension ratio (an indication of the air voids present), elastic-limit pressure, crush up pressure, bulk sound speed of the "foam" material, the shear failure surface, and constants defining the pressure-volume relationship for the material. Of particular importance to the accuracy of the code predictions are the pressure-volume behavior of the material, the shear failure surface, and the elastic and crush up pressures (P_e and P_c).

The experimental results from small scale explosions in tuff (and RMG 2C4 grout) by Nagy and Florence [2] provided the particle velocity versus time data used for comparisons with WONDY calculations. Figures 3.1 and 3.2 show the experimental setup and Figure 3.3 shows the records from particle velocity gages located 1.0, 2.0, and 5.0 cm from the center of a 1.0-cm diameter PETN explosive charge. The signals labeled 408 (*Saturated*) and 409 (*Saturated*) were averaged to produce a single record for use in comparison with the computational results. The density of the PETN was 1000 kg/m³ and the charge had a mass of 3.75x10⁻⁴ kg. The tuff specimen was a core 28 cm diameter by 28 cm height and was split in half. Several grooves were cut concentric to the specimen axis for installation of the wire loops used for the particle velocity gages. A spherical hole was milled in the center of

Figure 3.1. Configuration of particle velocity experiments in geologic materials.

(a) Installation of Particle Velocity Loops in Geologic Cores

(b) Experimental Arrangement

Figure 3.2(a) and (b). Loop installation and experimental setup.

Figure 3.3. Particle velocities in P-tunnel tuff.

each specimen half for the spherical explosive. After placing the loops in the grooves and the explosive in the hole, the two halves were assembled and the tuff sample placed in a confinement chamber. The chamber was then aligned in an external magnetic field whose flux lines were parallel to the cylindrical axis of the tuff specimen. The core was from P tunnel (the specific stratigraphic section is not known) and several tests were conducted on dry and saturated tuff specimens. WONDY calculations were performed using saturated tuff property data found in the literature in an attempt to match the particle velocity data from the experiments.

Tuff properties data were from many sources [4-17]. The Hugoniot data were in the form of shock velocity - particle-velocity ($U_s - U_p$) and pressure-volume ($P - V$) curves. Table 3.1 is a list of all the tuff Hugoniot data, but separated into saturated and dry tuff data. These same data are shown in Figures 3.4-3.7. There is considerable scatter of the tuff Hugoniot data which is probably due to the variations in saturation, porosity, density, mineral content, and strata from which the tuff originated. Some of the tuff samples were not characterized in terms of the above properties and in many cases only the densities were determined.

The tuff U_s-U_p data were fit by the linear least squares method over the range of the experiments to the equation

$$U_s = C_0 + S U_p \quad (3.1)$$

where C_0 is the bulk sound speed in m/s and S is the slope of the line.

The saturated tuff data appeared to have three distinct linear regions and hence were fit with three lines given below.

$$\begin{aligned} \text{Region 1 } U_s &= 2234 + 1.632U_p \text{ where } U_p < 700 \text{ m/s} \\ \text{Region 2 } U_s &= 2912 + 1.052U_p \text{ where } 700 < U_p < 2100 \text{ m/s} \\ \text{Region 3 } U_s &= 1315 + 1.658U_p \text{ where } U_p > 2100 \text{ m/s} \end{aligned} \quad (3.2)$$

The $P - V$ data were fit to a cubic equation by the least squares method to the form

$$P = K_0 \eta (1 + K_1 \eta + K_2 \eta^2) \quad (3.3)$$

where $\eta = U_p/U_s$, $K_0 = \rho_0 C_0^2$, and ρ_0 is the initial density of the solid tuff. Also, C_0 is the bulk sound speed for the solid material. The actual equation used is given by

$$\begin{aligned} P &= 2.077 \times 10^{10} \eta (1 + 3.447 \eta + 4.367 \eta^2) \\ \text{with } K_0 &= 2450(2912)^2 \end{aligned} \quad (3.4)$$

The density value for calculating K_0 is an average grain density for tuff and the bulk sound speed is from the fit of the $U_s - U_p$ data for Region 2 given above. Figure 3.6 illustrates the fit to the experimental data. Also shown is the result of a fit to a quadratic form which was not used in the WONDY input file.

The pressure associated with the break between Regions "1" to "2" represents the crush pressure, $P_c = 5.79 \times 10^9$ Pa. The elastic pressure, P_e , is not as well defined, but examination of the low stress data indicates a likely value of 3.0×10^8 Pa.

Saturated & Wet NTS Tuff

NTS & Lab Data

Figure 3.4. $U_s - U_p$ Hugoniot data for saturated and wet NTS tuff.

Saturated & Wet NTS Tuff

NTS & Lab Data

Figure 3.5. $P - V$ Hugoniot data for saturated and wet NTS tuff.

Saturated & Wet NTS Tuff

NTS & Lab Data

Figure 3.6. $P - \eta$ Hugoniot data for saturated tuff, and cubic and quadratic fits to the data.

Dry NTS Tuff

NTS & Lab Data

Figure 3.7. $U_s - U_p$ Hugoniot data for dry NTS tuff.

Dry tuff data indicate two regions represented by two straight line segments. The least-squares fits are given by

$$\begin{aligned} \text{Region 1 } U_s &= 1557 + 1.288U_p \text{ where } U_p < 3100 \text{ m/s} \\ \text{Region 2 } U_s &= 540 + 1.620U_p \text{ where } U_p > 3100 \text{ m/s} \end{aligned} \quad (3.5)$$

Figure 3.7 shows the dry tuff equation-of-state data in the $U_s - U_p$ plane.

Another set of properties data for saturated tuff needed for WONDY calculations is a representation of the failure surface, that is, shear strength versus mean pressure. Compressive static yield strength data were compiled from the literature [17] and the data fit to the equation

$$Y = Y_0 - Y_1 e^{(-Y_2 P)} \quad (3.6)$$

where Y_0 , Y_1 , and Y_2 are coefficients and P is the mean stress. The coefficients used are

$$Y_0 = 0.9 \times 10^8 \text{ Pa}, \quad Y_1 = 0.8 \times 10^8 \text{ Pa}, \text{ and } Y_2 = 0.25 \times 10^{-8} \text{ Pa} \quad (3.7)$$

Figure 3.8 is a plot of the yield surface equation. This yield surface represents tuff material that is competent, well consolidated, and dense. In other words, the tuff test specimens were for stronger tuff. The coefficients for the weaker tuff are

$$Y_0 = 0.3 \times 10^8 \text{ Pa}, \quad Y_1 = 0.2 \times 10^8 \text{ Pa}, \text{ and } Y_2 = 0.28 \times 10^{-8} \text{ Pa} \quad (3.8)$$

This yield surface is also shown in Figure 3.8.

The saturated tuff properties information for the WONDY input data that best matches the SRI experiments are listed in Table 3.2. The table includes the average properties data for the actual tuff material as well as other equation-of-state data such as ratio of the specific heats, Gruneisen gamma, initial distention ratio, and the data discussed above. Figures 3.9-3.11 show comparisons between the experimental data and the numerical simulations for the three gages at ranges of 1, 2, and 5 cm. We were able to match the rise time and amplitude for the two close-in gages while the match is not as good for the release part of the signal. At the farthest position the agreement is poor when matching the signal start, amplitude, and release on the back side. However, the rise time was matched reasonably well. These calculations were performed without incorporating a damage model or introducing strain rate effects for the tuff.

Parameter studies indicate that the calculations are sensitive to the crush and elastic pressures and, to a lesser extent, to the failure model. Varying the distention ratio a few percent had little effect on the calculated particle velocities. There is some sensitivity to the zone size for the tuff specimen. For the best-fit calculation the tuff contained 250 zones or 0.56 mm per zone. A smaller zone size tends to round the sharp corners on the rise.

The pressure range for these particular calculations on tuff based on the SRI experiments was 2.0 GPa to elastic conditions or about 0.1 GPa. This pressure is in the elastic-plastic regime of the tuff and hence these calculations are not applicable for predicting the response above 10.0 GPa pressure.

Failure Envelope Upper Limit

Area 12 Tuff

Figure 3.8. Failure Envelope for Tuff.

PV1 (Nagy & Florence)

Sat Tuff P-Tunnel JWL/State7

Figure 3.9. Experiment and WONDY comparisons for the 10-mm gage.

PV3 (Nagy & Florence)

Sat Tuff P-Tunnel JWL/State7

Figure 3.10. Experiment and WONDY comparisons for the 20-mm gage.

PV6 (Nagy & Florence)

Sat Tuff P-Tunnel JWL/State7

Figure 3.11. Experiment and WONDY comparisons for the 50-mm gage.

TABLE 3.1. Data on all Tuff

Density (g/cc)	Sat (%)	Por (%)	Pressure (kb)	Particle Velocity (m/s)	Shock Velocity (m/s)	Whose Data	Tuff Origin
1.46	15	40	0.16	6.3	1740	Larson	Mt St Helens
1.46	15	40	0.22	8.3	1815	Larson	Mt St Helens
1.73	sat	34	0.7	11	3678	Larson	Tunnel
2	96	35	0.8	16	2251	Bass	NTS
2	96	35	1.1	21	2258	Bass	NTS
2	96	35	1.2	24	2262	Bass	NTS
2	96	35	1.4	28	2267	Bass	NTS
2	96	35	1.8	35	2276	Bass	NTS
2	96	35	2.2	44	2288	Bass	NTS
2	96	35	2.4	48	2294	Bass	NTS
1.73	sat	34	2.77	61	2625	Larson	Tunnel
1.46	15	40	1.83	65	1928	Larson	Mt St Helens
1.86	sat	34	2.5	65	2068	Larson	Mt St Helens
1.46	15	40	2.2	87	1732	Larson	Mt St Helens
2	96	35	5	96	2357	Bass	NTS
1.84	sat		4.4	115	1940	Larson	Mt St Helens
1.46	15	40	3.65	132	1894	Larson	Mt St Helens
2	dry		7.55	146	2586	Lysne	Ue19b1
2	dry		8.11	147	2759	Lysne	Ue19b1
1.74	sat		7.4	156	2726	Lysne	U12g tunnel
1.84	sat		6.2	162	2160	Larson	Mt St Helens
1.76	sat		6.3	162	2210	Lysne	U12g tunnel
1.95	dry		5.5	164	1720	Lysne	Ue19b1
1.5	dry		4.83	168	1917	Lysne	U12g tunnel
2	96	35	8	180	2469	Bass	NTS
1.73	sat	34	9.2	186	2859	Larson	Tunnel
1.84	sat		8.1	194	2320	Larson	Mt St Helens
1.86	sat	34	7	195	1930	Larson	Mt St Helens
1.84	sat		10	220	2420	Larson	Mt St Helens
1.34	dry		3.53	228	1155	Lysne	U12g tunnel
1.46	15	40	5.9	253	1597	Larson	Mt St Helens
1.84	sat		12.1	266	2520	Larson	Mt St Helens
1.84	sat		12.9	271	2640	Larson	Mt St Helens
1.977	90	33	19	285	3380	Furnish	N tunnel
1.819	78	33	18.1	291	3430	Furnish	MC3 P tunnel
1.84	sat		13.9	295	2570	Larson	Mt St Helens
1.74	sat		13.8	310	2558	Lysne	U12g tunnel
2	dry		12.7	314	2022	Lysne	Ue19b1
1.76	sat		13.2	316	2373	Lysne	U12g tunnel
1.76	sat		12.8	318	2287	Lysne	U12g tunnel
1.46	15	40	7.65	319	1643	Larson	Mt St Helens
1.95	dry		11.4	326	1793	Lysne	Ue19b1
1.5	dry		9.21	332	1849	Lysne	U12g tunnel
1.84	sat		16.7	335	2730	Larson	Mt St Helens
1.83	97	33	15.5	335	2530	Furnish	MC2 P tunnel
1.46	15	40	8.87	366	1660	Larson	Mt St Helens
1.46	15	40	8.77	370	1623	Larson	Mt St Helens
1.34	dry		7.2	372	1444	Lysne	U12g tunnel
1.812	78	33	22.4	376	3290	Furnish	MC2 P tunnel
1.46	15	40	9.75	381	1753	Larson	Mt St Helens
1.84	sat		20.2	390	2840	Larson	Mt St Helens
1.46	15	40	9.95	398	1712	Larson	Mt St Helens
1.46	15	40	9.3	405	1573	Larson	Mt St Helens
2	dry		23.2	430	2698	Lysne	Ue19b1

Table 3.1. (continued)

Density (g/cc)	Sat (%)	Por (%)	Pressure (kb)	Particle Velocity (m/s)	Shock Velocity (m/s)	Whose Data	Tuff Origin
1.95	dry		22.5	436	2646	Lysne	Ue19b1
1.84	sat		23.4	437	2920	Larson	Mt St Helens
1.74	sat		22.2	456	2798	Lysne	U12g tunnel
1.74	sat		20.8	462	2587	Lysne	U12g tunnel
1.76	sat		22.1	470	2672	Lysne	U12g tunnel
2	96	35	28	472	2857	Bass	NTS
1.5	dry		14.5	480	2014	Lysne	U12g tunnel
1.84	sat		28.2	488	3090	Larson	Mt St Helens
1.34	dry		11	530	1549	Lysne	U12g tunnel
1.988	90	33	42.8	554	3860	Furnish	N tunnel
1.844	78	33	42.2	637	3590	Furnish	MC3 P tunnel
1.83	97	33	42.5	669	3470	Furnish	MC2 P tunnel
1.909	dry	24	36	680	2820	Kusubov	U12a
1.54	61	36	14	719	1974	Bass	NTS
1.46	15	40	20.5	720	1950	Larson	Mt St Helens
1.602	dry	33	27	730	2290	Kusubov	U12a & b
2.01	dry	19	47	780	3010	Bass	Pahute Mesa
1.971	90	33	64.2	792	4110	Furnish	N tunnel
1.844	78	33	57.1	837	3700	Furnish	MC3 P tunnel
1.7	dry		38.9	869	2627	Lombard	U12a tunnel
1.771	dry	33	41	880	2630	Kusubov	U12b
1.72	sat	27	53	900	3420	Bass	Area 16
1.46	15	40	29.6	910	2228	Larson	Mt St Helens
1.83	97	33	62.4	919	3710	Furnish	MC2 P tunnel
1.46	dry	27	31	950	2240	Bass	Area 3
1.46	dry	27	39	1000	2680	Bass	Area 16
1.97	sat		77	1003	3878	Shipman	Rainier Mesa
1.54	61	36	40	1049	2568	Bass	NTS
1.988	90	33	97.7	1059	4640	Furnish	N tunnel
2.19	wet	21	103	1070	4400	Bass	Pahute Mesa
1.844	78	33	86.8	1153	4090	Furnish	MC3 P tunnel
2.01	dry	19	86	1160	3670	Bass	Pahute Mesa
2.01	dry	19	84	1170	3590	Bass	Pahute Mesa
1.83	77	33	86.6	1182	4000	Furnish	MC2 P tunnel
1.97	dry	21	80	1190	3430	Bass	Pahute Mesa
1.97	dry	21	80	1190	3400	Bass	Pahute Mesa
1.901	wet	33	96	1230	4090	Kusubov	U12b
1.901	wet		96	1230	4090	Lombard	NTS
1.869	wet		94	1236	4050	Lombard	U12b tunnel
1.89	wet	33	94	1240	4050	Kusubov	U12b
2.046	90	33	118.7	1257	4620	Furnish	N tunnel
1.898	wet	33	100	1270	4130	Kusubov	U12b
1.898	wet		95	1270	4130	Lombard	U12b tunnel
1.64	dry	29	67	1280	3170	Bass	Rainier-pink
1.4	dry	40	58	1320	3120	Bass	Rainier-white
1.931	dry	24	95	1320	3720	Kusubov	U12a
1.69	dry		77	1329	3433	Lombard	U12b tunnel
1.69	dry	33	77	1330	3430	Kusubov	U12b
1.4	dry	40	57	1330	3050	Bass	Rainier-white
1.661	dry		74	1340	3330	Lombard	U12b tunnel
1.661	dry	33	74	1340	3330	Kusubov	U12b
1.23	dry	39	46	1390	2710	Bass	Rainier-buff
1.97	sat		131	1401	4749	Shipman	Rainier Mesa
1.618	dry	33	76	1420	3290	Kusubov	U12a & b

Table 3.1. (continued)

Density (g/cc)	Sat (%)	Por (%)	Pressure (kb)	Particle Velocity (m/s)	Shock Velocity (m/s)	Whose Data	Tuff Origin
1.83	97	33	119.6	1440	4540	Furnish	MC2 P tunnel
1.74	sat	27	108	1450	4260	Bass	Area 16
1.844	78	33	131.9	1529	4680	Furnish	MC3 P tunnel
1.46	dry	27	82	1570	3560	Bass	Area 16
2.01	dry	19	134	1580	4230	Bass	Pahute Mesa
1.46	dry	27	85	1580	3700	Bass	Area 3
1.855	wet		136	1590	4610	Lombard	U12b tunnel
1.855	wet	33	136	1590	4610	Kusubov	U12b
1.97	dry	21	130	1600	4130	Bass	Pahute Mesa
1.897	wet		133	1600	4400	Lombard	U12b tunnel
1.897	wet	33	133	1600	4400	Kusubov	U12b
1.861	wet	33	132	1610	4410	Kusubov	U12b
1.861	wet		130	1610	4411	Lombard	U12b tunnel
1.882	dry		132	1626	4299	Lombard	U12a tunnel
1.882	dry	33	132	1630	4300	Kusubov	U12b
1.64	dry	29	113	1710	4030	Bass	Rainier-pink
1.713	dry	33	109	1720	3710	Kusubov	U12b
1.713	dry		109	1720	3710	Lombard	U12b tunnel
1.604	dry		105	1730	3780	Lombard	U12b tunnel
1.604	dry	33	105	1730	3780	Kusubov	U12b
1.74	92	38	140	1750	4695	Bass	NTS
1.97	sat		170	1802	4781	Shipman	Rainier Mesa
1.4	dry	40	91	1820	3570	Bass	Rainier-white
1.4	dry	40	91	1820	3560	Bass	Rainier-white
1.23	dry	39	81	1870	3530	Bass	Rainier-buff
1.97	sat		189	1879	5105	Shipman	Rainier Mesa
1.97	sat		191	1934	5012	Shipman	Rainier Mesa
2.19	wet	21	221	1950	5170	Bass	Pahute Mesa
1.898	dry	24	170	1990	4530	Kusubov	U12a
1.869	wet		171	2020	5010	Lombard	U12b tunnel
1.869	wet	33	189	2020	5010	Kusubov	U12b
1.54	61	36	125	2079	3988	Bass	NTS
1.692	dry	33	151	2080	4290	Kusubov	U12a & b
2.01	dry	19	197	2090	4690	Bass	Pahute Mesa
1.97	dry	21	195	2090	4730	Bass	Pahute Mesa
1.97	dry	21	194	2100	4690	Bass	Pahute Mesa
1.583	dry	33	143	2120	4270	Kusubov	U12a & b
1.9	dry	24	199	2210	4730	Kusubov	U12a
1.796	wet		197	2240	4790	Lombard	U12b tunnel
1.903	wet		224	2250	5230	Lombard	U12b tunnel
1.914	wet	33	224	2250	5230	Kusubov	U12b
1.46	dry	27	143	2280	4280	Bass	Area 3
1.74	92	38	212	2301	5103	Bass	NTS
1.889	wet	33	223	2310	5110	Kusubov	U12b
1.65	dry		181	2310	4760	Lombard	U12b tunnel
1.63	dry	33	173	2320	4570	Kusubov	U12a & b
1.89	dry	24	208	2340	4710	Kusubov	U12a
1.4	dry	40	137	2360	4160	Bass	Rainier-white
1.4	dry	40	136	2360	4110	Bass	Rainier-white
1.76	dry		200	2380	4763	Shipman	Rainier Mesa
1.744	dry	33	202	2430	4780	Kusubov	U12b
1.46	dry	27	152	2460	4240	Bass	Area 16
1.46	dry	27	153	2500	4200	Bass	Area 3
1.46	dry	27	147	2500	4030	Bass	Area 16

Table 3.1. (continued)

Density (g/cc)	Sat (%)	Por (%)	Pressure (kb)	Particle Velocity (m/s)	Shock Velocity (m/s)	Whose Data	Tuff Origin
2.01	dry	19	295	2600	5650	Bass	Pahute Mesa
2.01	dry	19	311	2790	5540	Bass	Pahute Mesa
1.75	sat	27	270	2810	5490	Bass	Area 3
1.97	dry	21	304	2820	5470	Bass	Pahute Mesa
1.46	dry	27	202	2900	4780	Bass	Area 3
1.97	sat		390	3002	6597	Shipman	Rainier Mesa
1.4	dry	40	211	3050	4950	Bass	Rainier-white
1.97	dry	21	393	3310	6030	Bass	Pahute Mesa
1.97	sat		453	3328	6904	Shipman	Rainier Mesa
1.64	dry	29	326	3350	5940	Bass	Rainier-pink
1.23	dry	39	239	3370	5760	Bass	Rainier-buff
1.748	dry	33	354	3430	5910	Kusubov	U12a & b
1.76	dry		416	3588	6584	Shipman	Rainier Mesa
1.76	dry		425	3673	6570	Shipman	Rainier Mesa
1.77	dry	33	458	3820	6760	Kusubov	U12a & b
1.97	sat		680	4190	8233	Shipman	Rainier Mesa
1.76	dry		594	4407	7663	Shipman	Rainier Mesa
1.97	sat		874	4748	9341	Shipman	Rainier Mesa
1.76	dry		796	5111	8851	Shipman	Rainier Mesa
1.76	dry		909	5444	9490	Shipman	Rainier Mesa
1.76	dry		1009	5819	9849	Shipman	Rainier Mesa
1.97	sat		1278	5883	11028	Shipman	Rainier Mesa
1.97	sat		1331	6019	11228	Shipman	Rainier Mesa
1.76	dry		1261	6490	11035	Shipman	Rainier Mesa

TABLE 3.2. WONDY Input Data for Best Fit

Parameter	Value
Number Zones	250
Tuff Thickness (m)	0.14
Solid Density (kg/m ³)	2450
Solid Bulk Speed (m/s)	2910
Solid modulus (GPa)	20.75
K1 Coefficient	3.447
K2 Coefficient	4.367
Solid Shear Modulus (GPa)	12.45
Poisson's Ratio	0.25
Cp/Cv - 1	0.25
Gruneisen Gamma	1.5
Distention Ratio	1.32
Elastic Pressure (MPa)	6
Crushup Pressure (MPa)	30
Energy Zero Strength (MJ/kg)	2.511
Actual Bulk Speed (m/s)	2234
Sublimation Energy (MJ/kg)	11.72
Melt Energy (MJ/kg)	2.511
Y0 Yield Strength (MPa)	0.9
Y1 Yield Strength (MPa)	0.8
Y2 Yield Strength (MPa)	2.5E-08

4. Simulation of the Response of a Competent Rock

It was felt that the best test of the codes would be to attempt to predict the response of a well-behaved, competent, low-porosity rock to an explosive load. The results of this check would be even more precise if the size of the test utilized for the exercise was relatively small. The small size would eliminate any concerns about the properties as determined from static or dynamic testing using small size laboratory specimens being drastically different from the material properties of the larger rock mass. The fact that the rock was competent and relatively non porous would eliminate any uncertainties concerning unduly high air-void content or large variations in properties.

We found a very nice set of experiments that had been conducted at SRI by Miller and Florence [3] in 1991. Of particular interest were the results of a number of tests conducted in Indiana Limestone. Rock specimens of the well-characterized material in the form of cylinders that were 27 cm in diameter and 27 cm high were used in the test series. The wave source was a 1 cm diameter sphere containing 3/8 gram of PETN powder packed to density of 1 gm/cm³. Particle velocity histories were measured in the tests at nine different ranges using concentric copper loops placed in machined grooves along the midplane of the rock specimen. A constant axial magnetic field was applied to the specimen, and the particle velocities were measured by monitoring the induced voltage generated by each loop as it moved through the magnetic field. Figure 4.1 shows the experimental set up used by SRI, and Figure 4.2 gives typical results from tests that were conducted in dry/frozen Indiana Limestone. (These results were nearly identical to those obtained from tests conducted earlier on dry Indiana Limestone at room temperature at SRI). The properties for the Indiana Limestone that were used in the SRI tests are given in Table 4.1. Many of these properties were taken from Heard, et al. [18]. Figure 4.3 presents the failure envelope for Indiana Limestone as determine from reference [18]. The upper curve is for dry limestone, the middle curve for a 50% saturation state, and the lower curve for the fully saturated material. The code we used requires a failure surface of the form $Y=Y_0-Y_1e^{-EP}$, where Y is the level at which shear failure occurs, Y_1 , Y_0 , and E are fitting parameters, and P is the mean stress. From the data shown, the upper curve would be nicely represented by $Y=7.5-7.25e^{-0.125P}$.

Figure 4.4 presents the displacements as a function of distance from the charge as obtained from an integration of the velocity pulses shown in Figure 4.2. Both the value of the outward displacement (initial pulse) and the total displacements are shown in Figure 4.4. Figure 4.5 presents a plot of the log of the outward displacement versus the log distance from the charge center and shows two distinct regions of behavior. From the edge of the charge (at 0.5 cm) to about 2 cm, the displacement can be represented by a power function of distance to the negative 2.375 power. For distances greater than 2 cm, the displacements can be expressed as a power function of distance to the negative 1.26 power.

The pressure-volume relationship (Hugoniot) was obtained from experimental work by Larson [10], Larson and Rodean [11], Kusubov [14], Larson and Anderson [16], Grady, Murri, and Mahrer [19], Grady, Hollenback, and Schuler [20], Furnish [21], and van Thiel [22]. Data obtained from these sources are shown in Figure 4.6. The curve fitted to the data was $P=1.42 \times 10^7 \eta (1-4.11\eta + 14.79\eta^2)$, where η is the ratio of the particle velocity to the shock wave velocity. The elastic limit pressure was taken to be 0.6 Kbar and the crush up pressure was taken to be 24 Kbar. These values

Figure 4.1. Experimental set-up for tests conducted on Indiana Limestone at SRI.

(a) 10-20 mm

(b) 25-40 mm

(c) 50-80 mm

Figure 4.2. Velocity results obtained from tests on Indiana Limestone at SRI.

Failure Envelope Indiana Limestone

Figure 4.3. Failure Envelope for Indiana Limestone.

Experimental Displacement Versus Distance From Charge

Figure 4.4. Experimental displacements in Indiana Limestone vs distance from charge center.

Figure 4.5. Log-Log Plot of displacement vs distance - Indiana Limestone.

Figure 4.6. P - V data for Indiana Limestone.

were determined from an examination of the pressure-volume data for the limestone and a knowledge of where material transitions occur.

As indicated above, the failure surface was determined from experimental work by Heard, et al. [18], and in the code the function used was $Y=30-29e^{-0.050P}$. This represents a failure surface of greater strength than determined from static testing and represents an attempt to recognize that there is an increase in strength during dynamic loading. Figure 4.7 shows this failure surface in relation to the actual data obtained in static testing. For the range of mean stresses likely to be seen during the dynamic loading the elevation of the strength is approximately a factor of two.

The results from the numerical simulation are shown in Figure 4.8. Figure 4.8a shows a comparison of the experimental results and the numerical simulation at a location of 1 cm from the charge center. Figure 4.8b gives the same comparison at 2.5 cm from the charge center, and Figure 4.8c gives a comparison at 5 cm from the charge center. At 1 cm the amplitude of calculated velocity matches fairly well the experimental value, but the calculated pulse width for the positive velocity is too large. At 2.5 cm the calculated amplitude is too high by about 30% and the positive pulse width is too small. At 5 cm from the charge center the calculated velocity magnitude is about 60% too large and the width of the calculated positive pulse is too small.

Figure 4.9 gives a summary of the comparisons between the calculation and the experiment. In Figure 4.9a the calculated velocity magnitudes are compared with the experimental values at various distances from the charge, and in Figure 4.9b the width of the calculated positive velocity peaks are compared with the experimental values. Both the amplitudes and the pulse widths are not in good agreement with the experimental values. The agreement with the amplitude values are better than the agreement with positive pulse widths. In addition to an elevation of the failure surface, this attempt at agreement also assumes a more densely packed PETN charge (1.4 gm/cm^3 compared to an actual value of 1 gm/cm^3) and a larger amount of air voids than are felt to be present (an initial distention ratio of 1.3 as compared to a value of 1.2).

To give an indication of how changes in the failure surface affect the simulated results, the computations were repeated with other failure surfaces. Figures 4.10 and 4.11 show two other failure surfaces used. The failure surface shown in Figure 4.10 is given by $Y=30-29e^{-0.075P}$ and represents a change in the exponent from the surface used earlier. Notice that the maximum value of strength is the same but the maximum is reached faster. By comparing Figure 4.3 with Figure 4.10 it can be seen that at a mean pressure of 10 Kbar the difference in failure strength between the two cases is 16 Kbar for the 0.075 exponent versus 12.5 Kbar for the 0.050 exponent. The failure surface shown in Figure 4.11 is given by $Y=15-14.5e^{-0.125P}$ and represents a material that is slightly weaker than the material originally presented. The failure strength at a mean pressure of 10 Kbar for the original material (exponent of 0.050) is 12.5 Kbar compared to a failure strength of only 11 Kbar for the weaker material (smaller asymptote, unconfined strength and exponent of 0.125). A comparison of results obtained with the three different failure surfaces are shown in Figure 4.12 at points located 1, 2.5, and 5 cm from the charge center. The material with a maximum strength of 15 Kbar is a little too weak to give good agreement at the close-in gage as shown in Figure 4.12a. Although the peak velocity is best predicted by this failure surface, the positive pulse width is much too great and the material behaves in almost a hydrodynamic fashion. The positive pulse width at the 1 cm location is best predicted by the

Failure Envelope Indiana Limestone

Figure 4.7. Failure Envelope used in WONDY calculations for dry Indiana Limestone.

Gage 1 @ 1.0 cm Velocity vs. Time

Figure 4.8a. Velocity results from WONDY for Indiana Limestone compared to experimental results at 1 cm from charge center.

Gage 4 @ 2.5 cm

Velocity vs. Time

Figure 4.8b. Velocity results from WONDY for Indiana Limestone compared to experimental results at 2.5 cm from charge center.

Gage 7 @ 5.0 cm

Velocity vs. Time

Figure 4.8c. Velocity results from WONDY for Indiana Limestone compared to experimental results at 5 cm from charge center.

Figure 4.9a. Summary of positive peak velocities from WONDY and SRI experiments.

Figure 4.9b. Summary of positive pulse width from WONDY and SRI experiments.

Failure Envelope Indiana Limestone

Figure 4.10. Alternate failure surface for Indiana Limestone - slightly stronger.

Failure Envelope Indiana Limestone

Figure 4.11. Alternate failure surface for Indiana Limestone - slightly weaker.

Gage 1 @ 1.0 cm

Velocity vs. Time

Figure 4.12a. Comparison of results for different failure surfaces at 1 cm from charge center.

Gage 4 @ 2.5 cm

Velocity vs. Time

Figure 4.12b. Comparison of results for different failure surfaces at 2.5 cm from charge center.

strongest material, i.e., the one with the 0.075 exponent. Figure 4.12b gives the comparisons at the 2.5 cm location. At this location the weakest material is still behaving a bit too hydrodynamically. All three materials result in too large of an amplitude and the best fit to pulse width is felt to be from the intermediate strength material (exponent 0.050). Figure 4.12c shows the comparison at a location 5 cm from the charge center. Once again the best fit is felt to come from the intermediate strength material. At this location, however, all three representations over predict the peak value and under predict the pulse width. The weakest material does not give a very good estimate of the negative pulse. It is interesting to note that all three materials give almost the same prediction for the positive velocity maximum amplitude (6.4 m/s). None do a good job in predicting the experimental results.

This lack of agreement is felt to be mainly due to the fact that the version of the WONDY code being used does not incorporate a damage model as some of the other codes do. That is to say in the numerical simulation shown, the shear strength of the limestone is always the same and is not decreased due to previous high amplitude loading. The codes that have been most successful in predicting the response of small models to explosive loading do incorporate some type of damage criteria [23]. In this material we also used a gamma law to describe the behavior of the explosive. In the earlier match with the tuff data, a JWL model was used for the explosive detonation and probably does a better job of predicting the correct behavior.

Modifications are being made to the WONDY code that we are using and, once the damage model has been successfully incorporated, the new code will be used to predict more accurately the response of the SRI models. For the time being, however, the agreement is acceptable for the purposes we have in mind which will be described in Section 6.

Gage 7 @ 5.0 cm

Velocity vs. Time

Figure 4.12c. Comparison of results for different failure surfaces at 5 cm from charge center.

Table 4.1. Properties of Indiana Limestone
(from [3])

Density	As Received	2.277 gms/cc	
	Dry	2.269 gms/cc	
	Saturated	2.425 gms/cc	
	Grain	2.699 gms/cc	
Porosity	Total	15.90 %	
	Interconnected	15.54 %	
	Occluded	0.36 %	
Crush Strength	As Received	45.8 MPa	
	Dried	53.3 MPa	
P-wave Speed	Dry	4.21 mm/ms	
	Saturated	4.35 mm/ms	
S-wave Speed	Dry	2.14 mm/ms	
	Saturated	4.35 mm/ms	
Static Constants	Bulk Modulus	15 GPa	
	Young's Modulus	22.5 GPa	
	Shear Modulus	9 GPa	
	Poisson's Ratio	0.25	
Dynamic Constants	Dry	Bulk Modulus	26.5 GPa
		Young's Modulus	27.8 GPa
		Shear Modulus	10.4 GPa
		Poisson's Ratio	0.33
	Saturated	Bulk Modulus	31.8 GPa
	Young's Modulus	28.6 GPa	
	Shear Modulus	10.6 GPa	
	Poisson's Ratio	0.35	

5. NTS Core Analysis

We looked at core from Holes UE12P.01, UE12P#4, U12P.02 IH-1, UE12N#14, U12N.23 UG-1, and UE19P. The first two numbers represent P tunnel designations for cores from vertical holes. U12P.02 IH-1 is the designation for a horizontal core that nearly intersected the Mission Cyber event working point in P tunnel which is located in Aqueduct Mesa. Figure 5.1 shows the location of the vertical cores relative to the Mission Cyber event in drift U12P.02 and the Disco Elm event in drift U12P.03. The Distant Zenith event in drift U12P.04 had a working point about 260 m beyond either of the other two events. Figure 5.2 shows the location of the horizontal core holes in P tunnel. This core hole was approximately the same level as the gage holes for monitoring the ground motion on the Mission Cyber event. Figure 5.3 is a map of the vicinity of the Misty Echo event and shows the location of the vertical (UE12N#14) and horizontal core holes (U12N.23 UG-1). Hole UE19P is a vertical core hole near the Towanda event on Pahute Mesa in Area 19. The ground motion data from Towanda have been simulated very well by App and Brunish [24], so we wanted to develop an impression of the quality of the tuff in which that test was conducted.

The core was evaluated by the authors in qualitative terms according to the apparent strength of the tuff, the density, the porosity, and the appearance. A strength range was ascribed to the tuff on a scale of 0 - 10 with zero assigned to the sandy and crumbly material and ten assigned to the dense, competent, and strong material. The denseness was estimated by hefting sections of the core, while the porosity was evaluated by how quickly the tuff sections absorbed water. The appearance included the sandy and crumbly condition, the color, and the presence of pumice inclusions and large voids.

Figures 5.4-5.9 show plots of the apparent strength versus depth for all the core holes analyzed. Tables 5.1 and 5.2 show the correlation of our impressions with test data of the tuff from core holes UE12P#4 and UE12N#14. Abundant test data were not available for the other core holes.

An analysis of core hole UE12P#4 plotted in Figure 5.4 indicates a weak cap rock layer, then approximately 300 ft of strong tuff rock, followed by 300 ft of weak tuff. At the working point (880 ft) and for about 120 ft above and 80 ft below the working point the rock strength is high (about 7.5 on our scale). There are layers in the section at 60, 760, and 960 ft, each about three to five feet thick that consist of sand and hence have zero strength. The average strength for the entire section is 4.9, and within 60 m of the working point the average relative strength is 5.1. Figure 5.10 shows pictures of the core in UE12P#4 taken at three different depths. Figure 5.10a shows a picture taken of the core in the interval from roughly 680 ft through 690 ft. In this area the tuff is very weak - in the range from 0 to 1 on our scale. Figure 5.10b shows the core in the range of 820 ft to 830 ft. This rock has a strength of between 7 and 8 on our scale and, in our opinion, is quite competent. Figure 5.10c shows a picture taken in the range from 880 ft through 890 ft. In this interval the tuff is more broken than that shown in Figure 5.10b but we judged its strength to be about the same - between 7 and 8 on our relative scale.

The relative strength of the companion core hole U12P.02 IH-1 (horizontal) near the Mission Cyber working point is shown in Figure 5.5. As shown in the figure, we found that all of the tuff in the horizontal hole had high rock strength (5-8 on our scale). The core over most of the length was like that shown in the picture presented as Figure 5.10b. A few boxes from a second horizontal core hole,

Figure 5.1. Surface geology and P-tunnel layout.

UE12P#4 MISSION CYBER

Figure 5.4. Relative strength vs depth for the vertical core hole UE12P#4 near Mission Cyber.

U12P.02 IH-1 MISSION CYBER

Figure 5.5. Relative strength vs length for the horizontal core hole U12P.02 IH-1 near Mission Cyber.

UE12P#01 OLD HOLE

Figure 5.6. Relative strength vs depth for the vertical core hole UE12P#01.

UE12N#14 MISTY ECHO

Figure 5.7. Relative strength vs depth for the vertical core hole UE12N#14 near Misty Echo.

U12N.23 UG-1 MISTY ECHO

Figure 5.8. Relative strength vs length for the horizontal core hole U12N#23 UG-1 near Misty Echo.

UE19P TOWANDA

Figure 5.9. Relative strength vs depth for the vertical core hole UE19P near Towanda.

Figure 5-10a Photograph of core from hole UE12P#4 depth 680 - 690 feet

Figure 5.10c Photograph of core from hole UE12P#4, depth approximately 850 feet

U12P UG-1, were examined at a range (approximately 2100 ft) corresponding to the working point. This core was also strong rock (5-8 on our scale). The tuff within 60 m in all directions of the working point (except possibly below) was quite uniform in terms of strength, porosity, and density. In summary, based on our examination, the tuff in the vicinity of Mission Cyber working point was found to be very good rock.

The quality of the tuff from core hole UE12P.01 was found to be considerably different from that observed in the other vertical hole. Figure 5.6 shows relative strength versus depth for this hole and indicates strong material from 60 ft below the surface to a depth of 500 ft. From 500 to 880 ft the strength is quite low. Over the entire length the average relative strength is 5.0 but at the working point depths for Mission Cyber and Disco Elm the average strength was found to be quite low - 2.1 on our scale.

Our interpretation from observations of core from the four holes that we viewed is that there is a surprisingly large change in the character of the tuff through any vertical stratigraphic sequence. In addition, there was also a surprising difference in the tuff between the two vertical holes. However, the quality and consistency of the Paintbrush tuff at the Mission Cyber working point (and probably at the Disco Elm working point also) were very good. We would judge the tuff in that location to average five to six on our relative scale. There is nothing in the quality of the tuff at these working points to suggest that the explosive energy would not couple well to the rock - at least in a horizontal direction.

The core from the two N tunnel holes, UE12N#14 and U12N.23 UG-1, had a noticeably poorer quality than the P-tunnel core. Figure 5.3 shows the locations for both of these holes which are near the Misty Echo event. Table 5.2 is a listing of the relative strength and other properties data available for hole UE12N#14, and Figure 5.7 is a visual representation of the relative tuff strength as we saw it. From 240 ft to 390 ft the strength is on average very high, but the next 700 ft varies from zero strength to medium strength. (We did not view the first 240 ft of core.) As indicated in the figure, this section has multiple weak and medium strong layers. The last 210 ft of tuff is again generally strong rock, and at the Misty Echo working point (about 1300 ft) the relative strength is nearly five. The average relative strength over the section on our scale was 3.5 and within 60 m of the working point for Misty Echo the average value is 6.4. Figure 5.11 shows a number of photographs of the core from this hole. Figure 5.11a shows a photograph of the 10 ft of core at 825 ft and below. As indicated in Figure 5.7, for this range we have assigned a strength of 2, but as the picture shows, part of the core at this location is nothing more than sand-like. Figure 5.11b shows a photograph of the core between 838 ft and 848 ft. In this case we gave the tuff a strength of 4. Figure 5.11c is a close up of a piece of the core shown in Figure 5.11b and shows the wavy surface of the core, which was quite flaky to the touch. As shown in Figure 5.11d, some of the core contained quite large inclusions. This picture was taken at around 990 ft. Figure 5.11e shows a closer view of the core in this vicinity. Figure 5.11f shows core from about the 1250 ft depth in this hole. Notice from Figure 5.7 that we have given this core a rating of 10. Even though it might not look much different from the core shown in the vicinity of Mission Cyber (Figure 5.10b), which we gave a strength rating of 7.5, it was much more dense and appeared stronger to us.

In the horizontal core hole, U12N.23 UG-1, the relative tuff strength varies widely from zero to medium strength of five as indicated Figure 5.8. This is different from

Figure 5.11a. Photograph of core from hole UE 12N#14 depth approximately 825 feet

Figure 5.11c. Photograph of core from hole UE12N#14, close-up of core in Figure 5.11b.

Figure 5.11e. Photograph of core from hole UE12N#14, close-up of core in Figure 5.11d.

1253.7 1254.3 1255.1 1256.1 1257.1 1258.1 1259.1 1260.1 1261.1 1262.1 1263.1 1264.1 1265.1 1266.1 1267.1 1268.1 1269.1 1270.1 1271.1 1272.1 1273.1 1274.1 1275.1 1276.1 1277.1 1278.1 1279.1 1280.1 1281.1 1282.1 1283.1 1284.1 1285.1 1286.1 1287.1 1288.1 1289.1 1290.1 1291.1 1292.1 1293.1 1294.1 1295.1 1296.1 1297.1 1298.1 1299.1 1300.1

what we found in the horizontal hole near the Mission Cyber working point. There was very little change in the horizontal section. In this hole there are three depths (600, 1200, and 2000 ft) where there are zones with very low strengths (0 to 1.5 on our scale). The weak zone at 600 ft is 40 ft in length and has zero unconfined strength, the one at 1200 ft is also 40 ft in length but has a strength of one. The weak zone at 2000 ft is 60 ft in length and has a strength of 1.5. It is our opinion that the presence of these weak zones in this horizontal strata could have considerable negative effects on the coupling of the energy to the rest of the tuff. Table 5.3 is a listing of relative strength with length for this core hole. Near the Misty Echo working point, which corresponds to the 1700 ft mark, the strength is about five. This correlates with our interpretation of the vertical core which was done independently on a different day. The average relative strength over the length of the core for this hole was found to be 3.8. As indicated above we also viewed the core from a vertical hole near the Towanda event (UE19P). Our impression is that this tuff is generally very good rock but has a few sections of weak rock as indicated in Figure 5.9. The upper sections appeared to be stronger than the lower sections, but overall the core has an average relative strength of 5.7. This is the same relative strength that we found to exist within 60 m of the Towanda working point (2182 ft). Using average properties for the tuff in ground motion calculations to simulate the Towanda event in our opinion could give very good results in this case.

Table 5.1. Analysis for core from hole UE12P#4

Core hole collar 6404' MSL

Tma - Ammonia Tanks

Tmr - Rainier Mesa Member

Tp - Paintbrush Tuff

Qual Strength 0 (no strength) - 10 (very strong)

Depth (ft)	Strata	Qual Strength	Unconf strength (kbar)	Unconf modulus (kbar)	Density Rec'd (g/cc)	Density Dry (g/cc)	Porous (%)	Sat (%)	Air (%)
0	Tma								
20	Tma	1.5		7	1.61	1.24	48.6	76	11.7
40	Tma	0.5	0.03						
60	Tma	0.5							
80	Tmr	5		10	1.57	1.13	54.3	81.6	9.9
100	Tmr	5.5	0.12	28	1.85	1.5	39.3	92.5	3.2
120	Tmr	5.5							
140	Tmr	7.5	0.1		1.97	1.71	33.6	79.5	6.9
160	Tmr	7.5	0.74		2.39	2.32	8.1	90.1	0.8
180	Tmr	7.5							
200	Tmr	7.5	0.49		2.14	1.92	23.3	97.1	0.7
220	Tmr	7.5							
240	Tmr	7.5							
260	Tmr	7.5							
280	Tmr	7.5	0.96	200	2.42	2.31	11.7	92	0.9
300	Tmr	7.5							
320	Tmr	7.5							
340	Tmr	7.5	1.2		2.37	2.26	12.3	91.8	1
360	Tmr	7.5							
380	Tmr	7.5	1.2		2.41	2.32	8.9	98	0.2
400	Tmr	7.5	0.19		2.31	2.2	12.3	91.9	1
420	Tmr	7.5							
440	Tmr	7.5	0.5		2.13	1.94	20.2	92.9	1.4
460	Tmr	3	0.07		1.78	1.4	43	89.3	4.6
480	Tmr	3	0.07		1.73	1.4	42.7	78	9.4
500	Tmr	3	0.07	27	1.72	1.38	43.3	77.2	9.9
520	Tmr	3	0.09		1.72	1.42	41.5	72.7	11.3
540	Tmr	3		13	1.78	1.46	39	80	7.8
560	Tmr	2							
580	Tp	1							
600	Tp	1			1.91	1.57	37.8	91.5	3.2
620	Tp	1							
640	Tp	1			1.85	1.49	38.1	93.1	2.6
660	Tp	1							
680	Tp	1			1.84	1.47	41.4	91.2	3.6

Table 5.1. (continued)

700	TP	1				1.82	1.44	38.7	98.2	0.7
720	TP	1	0.04	20						
740	TP	0	0.04	27		1.8	1.48	35.4	83.2	6.3
760	TP	3.5	0.1	33		1.91	1.63	31.3	89.6	3.2
780	TP	3.5	0.16	64		1.85	1.52	36.4	88.2	4.3
800	TP	3.5				1.94	1.61	42.2	77	9.7
820	TP	7.5				1.99	1.64	38.4	99.4	0.2
840	TP	7.5	0.1	80		1.92	1.56	37.4	97.8	0.8
860	TP	7.5	0.07	22		1.84	1.5	39.1	85.1	5.8
880	TP	7.5								
900	TP	7.5	0.08	89		1.86	1.49	37.5	97	1.1
920	TP	7.5		55		1.89	1.52	37.6	98.6	0.4
940	TP	7.5	0.13	75		1.93	1.61	34	98.1	0.7
960	TP	?	0.17	81		1.81	1.39	43.4	97.8	1

Table 5.2. Analysis for core from hole UE12N#14

Core hole collar 6404' MSL										
Tma - Ammonia Tanks										
Tmr - Rainier Mesa Member										
Tp - Paintbrush Tuff										
Qual Strength 0 (no strength) - 10 (very strong)										
Depth (ft)	Strata	Qual Strength	Unconf strength (kbar)	Unconf modulus (kbar)	Density Rec'd (g/cc)	Density Dry (g/cc)	Porous (%)	Sat (%)	Air (%)	
0	Tmr	na								
10	Tmr	na								
20	Tmr	na								
30	Tmr	na								
40	Tmr	na								
50	Tmr	na								
60	Tmr	na								
70	Tmr	na								
80	Tmr	na								
90	Tmr	na								
100	Tmr	na								
110	Tmr	na								
120	Tmr	na								
130	Tmr	na								
140	Tmr	na								
150	Tmr	na				2.387	10.7	90.8		1
160	Tmr	na								
170	Tmr	na								
180	Tmr	na								
190	Tmr	na								
200	Tmr	na								
210	Tmr	na								
220	Tmr	na								
230	Tmr	na								
240	Tmr	na								
250	Tmr	8								
260	Tmr	8			2.322	2.231	9.6	94.4	0.5	
270	Tmr	8								
280	Tmr	8								
290	Tmr	8								
300	Tmr	8								
310	Tmr	8								
320	Tmr	3								
330	Tmr	3								
340	Tmr	3								

Table 5.2. (continued)

350	Tmr	3	0.06	13.9	1.725	1.376	42.8	81.5	7.9
360	Tmr	3							
370	Tp	3							
380	Tp	3							
390	Tp	0							
400	Tp	0							
410	Tp	0							
420	Tp	0							
430	Tp	0							
440	Tp	0							
450	Tp	0							
460	Tp	3			1.625	1.16	51	91.3	4.5
470	Tp	3							
480	Tp	5							
490	Tp	1.5							
500	Tp	5							
510	Tp	0							
520	Tp	5							
530	Tp	5							
540	Tp	4							
550	Tp	2							
560	Tp	1.5							
570	Tp	1.5							
580	Tp	4							
590	Tp	3.5							
600	Tp	4.5							
610	Tp	3							
620	Tp	2.5							
630	Tp	2.5							
640	Tp	1.5							
650	Tp	0							
660	Tp	1.5							
670	Tp	3							
680	Tp	3							
690	Tp	1.7							
700	Tp	0							
710	Tp	0							
720	Tp	1.5							
730	Tp	2							
740	Tp	5							
750	Tp	3.5							
760	Tp	1.5							
770	Tp	4.5							
780	Tp	5							
790	Tp	3							

Table 5.2. (continued)

4.5	Tp	0.02	2	1.828	1.42	41.2	99	0.4
4	Tp							
2	Tp							
2	Tp							
2	Tp							
4	Tp							
4	Tp							
4	Tp							
4	Tp							
4	Tp							
4	Tp							
1	Tp	0.02	5.6	1.685	1.204	50.6	95	2.5
1	Tp							
1	Tp							
1	Tp							
4	Tp							
4	Tp							
4	Tp							
4	Tp							
3	Tp							
3	Tbg							
3	Tbg							
1	Tbg							
1	Tt5							
1	Tt5		1.3					
1	Tt5	0.01						
1	Tt5							
1	Tt5		1.3					
1	Tt5	0.07						
1	Tt5		5.7					
1	Tt5	0.05						
1	Tt5							
1	Tt4 4Ku							
4	Tt4 4Ku							
4	Tt4 4Ku							
4	Tt4 4Ku							
4	Tt4 4Ku							
4	Tt4 4Ku							
8	Tt4 4Kl							
8	Tt4 4Kl							
8	Tt4 4Kl	0.18	35.3	1.964	1.627	34.5	97.6	0.8
8	Tt4 4Kl							
8	Tt4 4Kl							
8	Tt4 4Kl							
8	Tt4 4Kl	0.25	63	1.955	1.616	34.4	98.7	0.5
8	Tt4 4Kl							
8	Tt4 4Kl	0.14	29.2	1.933	1.575	36.4	98.3	0.6
8	Tt4 4Kl							
4.5	Tt4 4J							

Table 5.2. (continued)

1250	Tt4 4J	4.5	0.11	23.6	19.25	1.551	38	98.5	0.6
1260	Tt4 4J	4.5							
1270	Tt4 4J	4.5							
1280	Tt4 4J	4.5	0.11	20.8	2.05	1.736	31.8	98.8	0.4
1290	Tt4 4H	4.5							
1300	Tt4 4H	4.5	0.11	30.7	1.746	1.29	46	99.1	0.4

Table 5.3. Analysis for core from hole U12N.23 UG-1

Tt4 - Tunnel beds, Unit 4
Qual Strength 0(no strength) - 10(very strong)

Depth (ft)	strata	Qual Strength
0	Tt4	5
20	Tt4	5
40	Tt4	5
60	Tt4	5
80	Tt4	5
100	Tt4	5
120	Tt4	5
140	Tt4	3
160	Tt4	3
180	Tt4	3
200	Tt4	3
220	Tt4	3
240	Tt4	3
260	Tt4	3
280	Tt4	3
300	Tt4	4.5
320	Tt4	4.5
340	Tt4	4.5
360	Tt4	4.5
380	Tt4	2.5
400	Tt4	3
420	Tt4	3
440	Tt4	3
460	Tt4	3
480	Tt4	3
500	Tt4	3
520	Tt4	3
540	Tt4	3
560	Tt4	2.5
580	Tt4	2.5
600	Tt4	0
620	Tt4	2.5
640	Tt4	2.5
660	Tt4	2.5
680	Tt4	2.5
700	Tt4	4.5
720	Tt4	4.5
740	Tt4	4.5
760	Tt4	4.5
780	Tt4	4.5
800	Tt4	4.5
820	Tt4	4.5
840	Tt4	3
860	Tt4	4.5
880	Tt4	4.5
900	Tt4	3.5
920	Tt4	3.5
940	Tt4	3.5
960	Tt4	3.5
980	Tt4	3.5
1000	Tt4	3.5
1020	Tt4	3.5
1040	Tt4	3.5

Table 5.3. (continued)

1060	Tt4	3.5
1080	Tt4	3
1100	Tt4	3
1120	Tt4	2.5
1140	Tt4	4.5
1160	Tt4	4.5
1180	Tt4	4.5
1200	Tt4	4.5
1220	Tt4	1
1240	Tt4	4
1260	Tt4	4
1280	Tt4	4
1300	Tt4	4
1320	Tt4	4
1340	Tt4	4
1360	Tt4	4
1380	Tt4	4
1400	Tt4	4
1420	Tt4	4.5
1440	Tt4	4.5
1460	Tt4	4.5
1480	Tt4	4.5
1500	Tt4	3.5
1520	Tt4	5
1540	Tt4	5
1560	Tt4	5
1580	Tt4	5
1600	Tt4	5
1620	Tt4	5
1640	Tt4	5
1660	Tt4	4.5
1680	Tt4	4.5
1700	Tt4	4.5
1720	Tt4	4.5
1740	Tt4	
1760	Tt4	
1780	Tt4	
1800	Tt4	
1820	Tt4	
1840	Tt4	
1860	Tt4	
1880	Tt4	
1900	Tt4	
1920	Tt4	
1940	Tt4	
1960	Tt4	
1980	Tt4	
2000	Tt4	
2020	Tt4	
2040	Tt4	
2060	Tt4	
2080	Tt4	
2100	Tt4	

6. Effects of Weak Layers and Open Joints

As part of our investigation to determine why some unexpected rock motion results have been obtained in past nuclear tests, the authors spent considerable time examining the core obtained from exploratory holes at the NTS as described in the previous section. As a result of those examinations, it is our opinion that considerable variations exist in the rock properties which could be of great importance to code calculations - even within a given sub-unit of the tuff. It has been acknowledged in the past that there are large differences in the core at the NTS. What is felt to be new and important, however, is that a very large percentage of the tuff (up to 40% in some locations) is extremely weak and friable. This tuff is so weak that it has been impossible to get mechanical properties since it is too weak even to fabricate test samples. Much of this weak material appears as sand in the core trays and would have an unconfined shear strength near zero.

There was not much evidence of open joints or bedding planes in the core we examined. However, in many of the test documents there are indications of open joints and bedding planes. It is felt that these are perhaps somewhat local to the tunnels and may have been caused by the construction of the tunnels. There were numerous locations where vugs or voids were present and these varied in size from 0.2 to 2.5 cm in diameter.

Based upon our examination of the core, it is our belief that the data the codes depend upon lack input from as much as 40% of the interval, and the strength data obtained in the testing are not representative of the section since they are only averages of the values obtained from the stronger rock present. The variation in strength values obtained in the laboratory vary by a factor of ten even ignoring the very weak (near zero strength material). Most of the codes treat the tuff as a homogeneous, isotropic material or as being made up from a finite number of homogeneous, isotropic materials, and use average values obtained from the static testing as a starting point from which to obtain input values for the rock.

We wanted to see how accurate this practice was for determining rock response to a blast so we decided to impose open joints (and later a layer of weak material) in a numerical model otherwise made of a strong material. The results of this effort will be described in the sections that follow.

Open Joints/Open Bedding Planes

Figure 6.1 shows the geometry of the computational model used. Note from Figure 4.5 that the decay rate of displacement is quite high very near the charge (less than 2 cm) but becomes much lower at distances further from the charge. Very near the charge the decay is due both to spherical dispersion and losses due to unrecovered plastic flow (or deformation). Farther from the charge the decay is mainly due to geometric dispersion as the stress levels are more near the elastic values. As seen from Figure 6.1 we chose to place the open joint at a location that was 5 cm from the center of the charge. This open joint is therefore in an elastic region. In this region the velocities and the displacements caused by the outgoing stress waves are much smaller than they are nearer to the charge. We used WONDY and the properties of Indiana Limestone to determine the effect that open gaps (joints or bedding planes) would have on the signals transmitted through them. We chose to increase the gap size in steps of 0.01 mm until all of the outgoing signal was lost due to the gap being too wide. As a bit of background, the scaling laws indicate the following: the location of 5 cm for the 3/8-gram charge

Figure 6.1. Geometry used in WONDY study of open joints or open bedding planes.

Figure 6.2a. Velocities at 1 cm from charge.

scales to a distance of 68 m for a 1-kt test and a gap of 0.01 mm corresponds to 1.33 cm for a 1-kt test. For a 1-kt test therefore we are incrementing open joints by about 1.33 cm at a distance of 68 m from the center of the source.

Figure 6.2 shows the predicted velocities from WONDY. Figure 6.2a shows the response at a location 1 cm from the charge, and as would be expected, there is virtually no difference in the velocity response for the six cases shown - no gap, gaps of 0.01, 0.02, 0.03, 0.04 and 2 mm - until after 20 microseconds when reflections begin to occur. Notice from the results that the reflections are greatest for the 2 mm gap but not much greater than for the 0.04 mm gap. As the gap size decreases, so do the reflected signals until the no gap situation is reached. Please remember that the magnitudes of the reflections are greatly amplified over what would be seen in practice since the spherical geometry tends to reverse the spherical divergence and give spherical convergence. It is very unlikely that the gaps would be spherical in shape, and this spherical convergence would not occur in practical applications. Figure 6.2b gives similar results at a location 2.5 cm from the charge. The only difference between the results shown in Figure 6.2b from those in Figure 6.2a is that at the 2.5 cm location reflections are present from both the front surface of the gap (at around 17 microseconds) and from the rear surface (at around 26 microseconds). Figure 6.2c presents the results of the particle velocity at 5 cm (or at the gap location). The effect of the gap at this location is to amplify the velocity over what would have been seen if the gap had not been present (much like one sees a doubling of the velocity at a free surface). Notice from the figure (6.2c) that the velocity is double that for the no gap situation for all but the smallest gap (0.01 mm). For the smallest gap the reflection increases the velocity from a little more than 6 m/s to more than 11 m/s. Figure 6.2d shows the values of velocity predicted by the code at 6.5 cm (or 1.5 cm after the gap). The velocity values do not decrease greatly for the 0.01, 0.02, and 0.03 mm case from the no gap situation. All three of these gaps give velocity peaks of 4 m/s or more compared to a peak velocity for the no gap case of 5 m/s. The largest two gaps (0.04 mm and 2 mm) show marked decreases in the velocity values at this location. In fact, the 2 mm case was such that no velocity was transmitted through. For the 0.04 mm gap the velocity dropped from 5 m/s for the no gap situation to only 2 m/s. Notice in all cases that the area under the velocity time curve decreases as the gap size increases. This means that the biggest effect of the gap for gap sizes between zero and 0.03 mm is to decrease the particle displacement but not the particle velocity. At 0.04 mm both the particle velocity and particle displacement are decreased greatly from the no gap situation.

Figure 6.3 shows how the two surfaces that make up the gap approach one another as time increases. For the smallest gap (0.01 mm), the surfaces of the gap touch one another at about 13 microseconds and the transmission of displacements (or velocities) into the material beyond the gap begins to occur. At about 18 microseconds the surfaces begin to separate and at 26 microseconds the two surfaces are separated by about 0.022 mm. In the case of the largest gap shown (0.05 mm), the two surfaces never touch (the maximum displacement of the surface nearest the explosive is only about 0.048 mm). Figure 6.4 shows the displacements as a function of time just past the gap. For no gap the maximum displacement occurs at 17 microseconds and is approximately 0.024 mm. For the largest gap considered which permitted stress waves to pass (0.04 mm) the maximum displacement was 0.004 mm just past the gap. Figure 6.5 shows the displacements transmitted through the gap as a function of gap width. The curve is a straight line with a slope of -0.5 mm/mm. The curve extrapolates to zero displacement at a gap size of 0.048 mm. This says that for each increment of

Velocities at 2.5 cm

Air Gap at 5 cm

Figure 6.2b. Velocities at 2.5 cm from charge.

Velocities at 5 cm

Air Gap at 5 cm

Figure 6.2c. Velocities at 5 cm from charge.

Velocities at 6.5 cm Air Gap at 5 cm

Figure 6.2d. Velocities at 6.5 cm from charge.

Change in Interface Separation

Figure 6.3. Change in surface separation as a function of gap size.

Figure 6.4. Displacements calculated just past gap as a function of gap size.

Figure 6.5. Displacement transmitted across open interfaces.

0.01 mm in gap size exactly one half or 0.005 mm of displacement is lost in crossing the gap. The closer to the charge, the larger the displacement and the larger the gap needed to absorb all of the radiating particle motion. For the situation investigated for a 1-kt device at a distance of 68 m from the charge, an open joint of 6.4 cm would result in reflection of all the stress wave. If the gap were 5.3 cm in extent, the velocity would drop to 40% of the value that would exist if the gap were not present. Anything below the 5.3 cm gap would not affect the velocity to a large extent but the displacement lost across the gap would in each case be half of the gap width.

Weak Layers

In our examination of the core from the P- and N-tunnel sites and in reading the descriptions of the geology of the event locations, some evidence of open joints was found; however, none were of an extent that would approach values deemed to be of sufficient size to significantly affect the velocities (obtained from an integration of accelerations measured). As indicated in the section describing the examination of the core, we did find many places where the strength of the core was very low - too low to measure. We therefore conducted an analysis similar to the one described above for gaps, but filled the gaps with weak rock (sand-like material). Figure 6.6 shows the geometry used in the computer simulation of the weak, rock-filled gap. Recall from Figure 4.5 that the very plastic response of the particular rock being simulated occurs out to about 2 cm from the charge. We chose to place these weak layers at three different distances from the charge center - 1 cm, 3.5 cm, and 5 cm. The geometry shown in Figure 6.6 is for the weak layer located 5 cm from the charge. This case will be discussed first. The other two cases (weak layer at 1 cm and at 3.5 cm) will be discussed in later sections.

Weak Layer at 5 cm

The current case therefore is for a layer of weak rock filling a gap of varying width with the beginning of the weak layer located at 5 cm where the rock response is well within the elastic regime.

Notice from Figure 6.6 that the properties of the strong rock are the same as used in the previous exercises - namely a failure surface represented by $Y = 30 - 29e^{-0.050P}$, where P is the mean stress in kilobars. The same information was used to represent the Hugoniot data with an elastic pressure of 0.6 Kbar and a crush pressure of 24 Kbar. The properties to use for the weak material were totally unknown. We chose to use a failure surface given by $Y = 3 - 2.9e^{-0.125P}$ - a factor of about five times weaker at a mean stress of 10 Kbar. We used the same Hugoniot data but used an elastic pressure of 0.06 Kbar (and in some cases zero Kbar) and a crush pressure of 2.4 Kbar. (These values were a factor of 10 weaker.) The correct properties for this type of material are simply not known at this time and the values were simply our best guess at what they might be.

The thickness of the weak layer was varied from zero to 6 mm. Figure 6.7 presents the velocity information obtained from the WONDY Code. Figure 6.7a gives the simulated velocities at 1 cm from the charge center. The results are independent of the width of the weak layer until the reflections from the interface appear at around 22 microseconds. These reflections are much smaller than those that occurred for the air gap discussed in the section above but reflections do occur. The same density was used for both the strong and the weak rock, and the fact that reflections are occurring at the strong/weak and weak/strong interfaces indicates

Figure 6.6. Geometry used in study of effects of weak layers located at 5 cm from charge center.

Figure 6.7a. Velocities at 1 cm with weak layer located at 5 cm.

that the shock velocity in the weak material is different than the shock velocity in the strong material. As indicated in Figure 6.7a, the larger the weak zone, the larger the reflection. Figure 6.7b shows velocities at 2.5 cm from the charge (or halfway between the charge and the weak layer), and Figure 6.7c gives the velocities just at the weak rock interface. Notice from Figure 6.7c that the presence of the weak layer increases the velocity at the 5 cm location from about 6.5 m/s to nearly 10 m/s. All layer thicknesses except the very smallest (0.25 mm) give nearly the same velocity due to the reflection. The pulse width of the velocity however decreases with decreases in layer thickness. Figure 6.7d gives the velocity at a location of 6.5 cm from the charge center. This location is within the stronger rock past the weak layer. As can be seen from the figure, the presence of a single 6 mm layer of weak rock drops the velocity from 5 m/s to 1.7 m/s. The 6 mm thickness corresponds to a thickness of 8 m for a 1-kt test. At least three weak zones of greater extent were found in our observation of the horizontal core in the vicinity of N tunnel. Many weak zones of greater extent were found in the vertical core in both the P and N tunnel vicinity. It is interesting to note from the figure (6.7d) that even the smallest layer of 0.25 mm results in a decrease in velocity from 5 m/s to 3.8 m/s (or a 25% decrease). This would correspond to a weak zone of only 0.33 m for a 1-kt shot.

Figure 6.8 shows how the two surfaces of the strong rock on either side of the weak layer approach one another as a function of time for various layer thicknesses. As expected the behavior is better than if a gap were present, but the weak rock permits much more deformation to occur as the weak layer size increases. For example, with no weak layer the relative deformation between the two strong layers is about 0.01 mm. With a 6 mm weak layer the relative deformation between the two strong layers is 0.03 mm or three times as large. Figure 6.9 presents a comparison between the deformations which occur when no weak layer is present and the deformations when a weak layer is present. In Figure 6.9a the curve marked with solid squares represents the deformation that occurs at 5.025 cm from the charge center when no weak layer is present. The curve marked with crosses gives the deformation for the same point when a 0.25 mm weak layer is present. As can be seen from the figure, the deformation when the weak layer is present is only 75% of the deformation when no weak layer is present. Note from Figure 6.9d that a weak layer of only 1 mm (scaled width of 2.66 m for a 1-kt shot) results in a halving of the displacement that would occur if no layer were present.

Figure 6.10 is a summary of the results obtained in terms of displacements. It presents the loss of displacement measured in the strong rock just past the weak layer as a function of weak layer thickness. As can be seen from the figure, unlike the results obtained for an open joint, the results are very nonlinear with most of the loss occurring with small widths of weak layers.

Weak Layer at 3.5 cm

At 3.5 cm the limestone is being subjected to primarily elastic action - but certainly higher stresses than at the 5 cm location discussed above. Figure 6.11 shows the geometry used in the computational simulation for the weak layer located at 3.5 cm. The strong and weak rock were represented by the same failure surfaces as used at the 5 cm location - namely $Y=30-29e^{-0.050P}$ for the strong rock and $Y=3-2.9e^{-0.125P}$ for the weak rock. The values of P_e and P_c for the strong rock were 0.6 Kbar and 24 Kbar respectively - the same as the 5 cm location. The elastic pressure used for the weak rock was reduced from 0.06 Kbar (as used at the 5 cm location) to zero but the crush pressure was kept at the 2.4 Kbar (as was used at the 5 cm location).

Gage 4 @ 2.5 cm

Velocity vs. Time

Figure 6.7b. Velocities at 2.5 cm with weak layer located at 5 cm.

Gage 7 @ 5.0 cm

Velocity vs. Time

Figure 6.7c. Velocities at 5 cm with weak layer located at 5 cm.

Figure 6.7d. Velocities at 6.5 cm with weak layer located at 5 cm.

Figure 6.8. Separation of two strong surfaces with weak layer between.

Figure 6.9a. Comparison with and without 0.25 mm weak layer at 5 cm.

Figure 6.9b. Comparison with and without 0.5 mm weak layer at 5 cm.

Figure 6.9c. Comparison with and without 1 mm weak layer at 5 cm.

Figure 6.9d. Comparison with and without 2 mm weak layer at 5 cm.

Displacements Weak Rock Fill

Figure 6.9e. Comparison with and without 6 mm weak layer at 5 cm.

Loss of Displacement Weak Rock Fill

Figure 6.10. Loss of displacement across weak layer as a function of layer thickness.

Figure 6.11. Geometry used for weak layer located 3.5 cm from charge center.

Figure 6.12a. Velocities at 1 cm for weak layer located at 3.5 cm.

The effect of this reduction in P_e will be discussed after the results obtained at 3.5 cm are presented.

Figure 6.12 presents the results obtained for velocities at three positions - 1 cm, 2.5 cm, and 5 cm from the charge center - with weak layers of thicknesses which ranged from 0.5 mm to 6 mm. Figure 6.12a presents results at 1 cm and, as before, the only changes seen occur as reflections from the strong/weak layer interface occur. In Figure 6.12b velocities predicted at 2.5 cm from the charge are presented. This is 1 cm before the weak layer. As can be seen, the reflection occurs very shortly after the peak velocities are reached as the distance from the strong/weak interface are quite small. Note that as the thickness of the weak layer increases, the amplitude of the reflected pulse increases. Figure 6.12c shows the velocities obtained at 5 cm (or 1.5 cm past the beginning of the weak layer). The smallest layer (0.5 mm) results in a decrease of velocity from 6.5 m/s to 4 m/s, about the same percentage loss as for the same thickness weak layer at 5 cm.

Figure 6.13 shows the separation of the two strong layers of rock (the space between is filled with the weak rock). As the weak layer thickness increases, the dynamic separation between the two strong layers decreases. When no weak layer is present, the deformation between a point located at 3.5 cm and a point located at 4.2 cm is about 0.02 mm. When a 1 mm weak layer is present, the separation between a point located at 3.5 cm and 3.7 cm is about 0.042 mm. In other words, the material deforms more than twice as much when the original distance is less than a third as great (0.2 cm compared to 0.7 cm).

Figure 6.14 gives a comparison of the displacements occurring at points just past the weak layer for the various different layer thicknesses and the displacements which would occur if no weak layer were present. Figure 6.14a shows a comparison of the displacements at 3.65 cm with no weak layer and with a 0.5 mm thick weak layer. As can be seen from the figure, the displacement drops from 0.037 mm to only 0.019 mm with just this very small layer. As evident from the other graphs presented in Figure 6.14, as the layer thickness increases so does the loss in displacement. Figure 6.15 presents a summary of the data presented in Figure 6.14 - the loss in displacement as a result of the presence of the weak layer as a function of the layer thickness. As was true for the 5 cm position, the loss in displacement is very non-linear. The loss in displacement, for example, with a 0.5 mm layer is 0.018 mm. This loss appears to quickly approach a maximum value - 0.025 mm. This compares with a loss of 0.008 mm for the same thickness at 5 cm but the maximum loss seen at that location was only 0.012 mm. The 0.5 mm weak layer at 3.5 cm results in 72% of the maximum loss. At 5 cm the same thickness results in only 67% of the maximum loss. In both cases this very small weak layer results in a very significant loss in both displacement and velocity. When scaled, the 3.5 cm location corresponds to a distance of about 48 m from a 1-kt charge. The 0.5 mm thickness corresponds to a thickness of about two thirds of a meter.

Figure 6.16 shows how the change from an elastic pressure of 0.06 Kbar for the weak material to a pressure of zero Kbar affects the velocities predicted. The velocities presented in the figure are for the case of a 6 mm-thick, weak layer located at the 3.5 cm location. The only change between the two cases was that the elastic pressure (P_e) was reduced from 0.06 Kbar to zero. As can be best seen from Figure 6.16c which shows the velocities at the 5 cm location (1.5 cm past the beginning of the weak layer), the resulting change in the velocity due to this decrease in elastic crush pressure is not great. The lower elastic pressure results

Velocity at 2.5 cm

Weak Layer at 3.5 cm

Figure 6.12b. Velocities at 2.5 cm for weak layer located at 3.5 cm.

Velocity at 5 cm

Weak Layer at 3.5 cm

Figure 6.12c. Velocities at 5 cm for weak layer located at 3.5 cm.

Surface Separation Weak Rock Fill

Figure 6.13. Surface separation predictions with weak layer located at 3.5 cm.

Displacements Weak Rock Fill

Figure 6.14a. Displacement comparison at 3.65 cm with and without 0.5 mm weak layer at 3.5 cm.

Displacements Weak Rock Fill

Figure 6.14b. Displacement comparison at 3.7 cm with and without 1 mm weak layer at 3.5 cm.

Displacements Weak Rock Fill

Figure 6.14c. Displacement comparison at 3.9 cm with and without 3 mm weak layer at 3.5 cm.

Figure 6.14d. Displacement comparison at 4.2 cm with and without 6 mm weak layer at 3.5 cm.

Figure 6.15. Loss of displacement across weak layer as a function of weak layer thickness - weak layer at 3.5 cm.

Figure 6.16a. Velocity comparison at 1 cm when weak layer at 3.5 cm elastic pressure is reduced from 0.06 Kbar to 0 Kbar.

Figure 6.16b. Velocity comparison at 2.5 cm when weak layer at 3.5 cm elastic pressure is reduced from 0.06 Kbar to 0 Kbar.

Figure 6.16c. Velocity comparison at 5 cm when weak layer at 3.5 cm elastic pressure is reduced from 0.06 Kbar to 0 Kbar.

Figure 6.17. Geometry of model with weak layer at 1 cm.

in a peak velocity of about 1.2 m/s compared to a velocity of 1.7 m/s with the pressure of 0.06 Kbar. The effect of this change is the same as would occur if the thickness was halved. That is, the 0.06 Kbar pressure with a 6 mm layer gives about the same results as would a 3 mm layer with a zero crush pressure. Recall that the major change occurs at very small layer thicknesses so that this effect is not very large. Additional information on the effect of changes in both P_0 and P_c will be given later.

Weak Layer at 1 cm

At a location that is 1 cm from the charge center the limestone is being subjected to stress levels that are quite high - certainly higher than elastic values. As seen from Figure 4.5, at this location the decay rate is greater than at 3.5, and at 5 cm where the weak layers were located previously. We were curious to see the effects that a weak layer located in this very high stress regime might have on the outward propagating stress waves. Figure 6.17 shows the geometry used in the computation. Both the strong and weak rock had the same properties as used at the other two locations - i.e., $Y=30-29e^{-0.050P}$ for the strong and $Y=3-2.9e^{-0.125P}$ for the weak. P_0 was 0.6 Kbar for the strong material and zero for the weak, and P_c was 24 Kbar for the strong material and 2.4 Kbar for the weak.

Figure 6.18 shows the results in the form of velocities as obtained from the WONDY code. The velocities at a location of 0.75 cm from the charge center are shown in Figure 6.18a. Notice that since the weak layer is located only 2.5 mm from this location that reflections from the interface are seen very quickly after the maximum for the outward traveling wave is obtained. As a result, as the layer thickness increases, the positive pulse width increases greatly as does the amplitude of the peak velocity. Figure 6.18b shows the velocities just at the location of the start of the weak layer. With no weak layer the peak velocity was about 119 m/s. With the weak layers (even the smallest one of 1 mm) the velocity increases at this location to around 170 m/s. As was the case at the 0.75 cm location, as the width of the weak layer increases so does the width of the positive velocity peak. Notice, however, that the peak velocity is not very dependent upon layer thickness at this location. Figure 6.18c shows the results obtained for velocities at 2.5 cm from the charge center. This is a location that is in the strong rock past the weak layer. Note that here there is only a slight drop in the maximum velocity for the smallest layer (1 mm), and that for this size layer there is not much of a change in the positive velocity pulse. For the larger sizes of weak layers there is a drastic drop in velocity. For the 3 mm layer the velocity drops from 16 m/s for no weak layer to 9 m/s. For the largest layer investigated (6 mm in thickness) the velocity drops from 16 m/s to 6 m/s. Note also that this drop in velocity is accompanied by a very large increase in pulse width. This was not the case for the two locations that were in the elastic regime. Here the results show that the larger the drop in peak velocity, the greater the increase in positive pulse width. This would imply that even though the particle velocity decreases, the particle displacement might not since the particle displacement is given by the area under the velocity-time curve.

Figure 6.19 gives displacement results. Figure 6.19a gives a comparison of the displacement of a point located 1.2 cm from the charge center with and without a weak layer of 1 mm. For a 1 mm weak layer, the position of 1.2 cm is located in the strong rock past the weak layer. Notice that even without the weak layer that most of the displacement at 1.2 cm is of a plastic nature and that there is very little elastic displacement. This is quite different than the displacement seen further away from the charge center. Figure 6.20 shows the displacement at 5 cm. Here the

Velocity at .75 cm With Weak Layer at 1 cm

Figure 6.18a. Predicted velocities at 0.75 cm with weak layer at 1 cm.

Velocities at 1 cm Just Before Weak Layer

Figure 6.18b. Predicted velocities at 1 cm with weak layer at 1 cm.

Velocities at 2.5 cm (After Weak Layer)

Figure 6.18c. Predicted velocities at 2.5 cm with weak layer at 1 cm.

Displacement @ 1.2 cm 1 mm Weak Layer @ 1 cm

Figure 6.19a. Predicted displacements at 1.2 cm with and without 1 mm weak layer at 1 cm.

Displacement @ 1.4 cm

3 mm Weak Layer @ 1 cm

Figure 6.19b. Predicted displacements at 1.4 cm with and without 3 mm weak layer at 1 cm.

Displacement @ 1.7 cm

6 mm Weak Layer @ 1 cm

Figure 6.19c. Predicted displacements at 1.7 cm with and without 6 mm weak layer at 1 cm.

maximum displacement is 0.0025 mm and of that displacement only 40% (0.001 mm) is permanent. At 1.2 cm, on the other hand, the maximum displacement is sixteen times larger (0.04 mm) and better than 90% of it is plastic. The presence of the 1 mm layer causes a loss of displacement of about 25%. In Figure 6.19b the effect on displacement of a 3 mm weak layer is shown. At this location (1.4 cm) the maximum displacement has dropped (with no weak layer present) to 0.025 mm. If a 3 mm weak layer is present, the peak displacement drops from 0.025 mm to 0.018 mm, and the decrease in permanent displacement is comparable. Results are given for the largest weak layer investigated (6 mm) in Figure 6.19c. At this location (1.7 mm) from the charge center, the particle displacement with no weak layer present is 0.016 mm with a permanent component of 0.014 mm. When a 6 mm weak layer is added, the peak displacement drops to 0.012 mm but the problem wasn't run for a long enough time to determine the plastic component of this displacement. Figure 6.21 shows the loss in displacement as a function of weak layer thickness. The loss is shown as a percentage of the displacement that would have existed had the weak layer not been present. At the other two locations (3.5 cm and 5 cm) the loss was actually given in millimeters since the change in displacement was not great. At this location the displacement changes very quickly as the distance from the charge is increased, and therefore it is necessary to show the loss as a percentage figure rather than an absolute one.

The results obtained indicate that the response of the rock to the presence of a weak layer is quite different when the weak layer is located in the high stress regime. First, the size of the weak layer has to be larger to have the same effect on the velocity that it would have at a location in the lower stress regime. More importantly, however, the result indicates that even though the velocity may be decreased significantly, the particle displacement will be affected to a lesser degree since the lowering of the peak velocity is accompanied by an increase of the pulse width.

We wanted to investigate this case in more detail and for that reason decided to repeat the computer simulations using a slightly weaker material - both for the strong and the weak rock. We used the selection of rock shown in Figure 6.22. The strong rock has a failure surface given by $Y=15-14.5e^{-0.125P}$ and the weak rock by $Y=1.5-1.45e^{-0.125P}$. Figure 6.23 shows a comparison of the failure surfaces for the previously-used combination to the newly-selected combination. Basically the strong rock is only slightly weaker but the weaker rock is considerable weaker. Figure 6.23a shows the original choice and Figure 6.23b the weaker material.

Figure 6.24 presents the velocity results for this weaker rock with the weak layers located at the 1 cm location. Figure 6.25 gives a direct comparison of the results obtained for a 6 mm wide layer of weak rock for the two rock combinations - [15/14.5/0.125, 1.5/1.45/0.125] and [30/29/0.050, 3/2.9/0.125]. (This comparison is abbreviated 15/1.5 and 30/3 on the figures.) As shown in Figures 6.25a and 6.25b the difference is not very great at locations before the weak zone even though both locations shown are very close to the point where the weak layer starts. Notice from the figure that the velocities are higher for the weaker rock than for the stronger rock. This is due to the fact that at the very high stress levels seen at this close-in location, the weaker rock behaves in a more hydrodynamic manner and hence produces a higher velocity signal with a broader positive pulse. Figure 6.25c shows the velocity comparisons just past the weak layer and Figure 6.25d at a location 2.5 cm from the charge center. At these locations the velocity values are higher for the weaker rock and the pulse widths are greater showing once again

Figure 6.20. Displacement at 5 cm with no weak layer present.

Figure 6.21. Loss in displacement due to presence of weak layer at 1 cm.

Figure 6.22. Model using weaker rock for weak layer located at 1 cm.

Failure Envelope Indiana Limestone

Failure Envelope Indiana Limestone

Figure 6.23a. Comparison of failure surfaces for two selections of rock properties for weak layer located at 1 cm - stronger rock combination.

Failure Envelope Indiana Limestone

Failure Envelope Indiana Limestone

Figure 6.23b. Comparison of failure surfaces for two selections of rock properties for weak layer located at 1 cm - weaker rock combination.

Velocity at .75 cm With Weak Layer at 1 cm

Figure 6.24a. Predicted velocities at 0.75 cm for weaker rock combination (15/14.5/0.125) when weak layer is located at 1 cm.

Velocities at 1 cm Just Before Weak Layer

Figure 6.24b. Predicted velocities at 1 cm for weaker rock combination (15/14.5/0.125) when weak layer is located at 1 cm.

Velocities at 2.5 cm (After Weak Layer)

Figure 6.24c. Predicted velocities at 2.5 cm for weaker rock combination (15/14.5/0.125) when weak layer is located at 1 cm.

Velocity at .75 cm With Weak Layer at 1 cm

Figure 6.25a. Velocity comparison at 0.75 cm of the two rock strength combinations with 6 mm weak layer at 1 cm.

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Velocity at 1.0 cm With Weak Layer at 1 cm

Figure 6.25b. Velocity comparison at 1 cm of the two rock strength combinations with 6 mm weak layer at 1 cm.

Velocity at 1.7 cm With Weak Layer at 1 cm

Figure 6.25c. Velocity comparison at 1.7 cm of the two rock strength combinations with 6 mm weak layer at 1 cm.

Figure 6.25d. Velocity comparison at 2.5 cm of the two rock strength combinations with 6 mm weak layer at 1 cm.

Figure 6.26a. Displacement comparison at 1.2 cm with and without 1 mm weak layer at 1 cm - weaker of two rocks (15/14.5/0.125).

the tendency of the weaker rock to behave in a more hydrodynamic fashion. Figure 6.26 shows the deformation results for this weaker rock. Figure 6.26a gives a comparison of the deformation at a point located 1.2 cm from the explosive charge center with and without a 1 mm weak layer located at 1 cm. Notice that the displacement is actually greater with the weak layer than without it. This is quite different from any behavior seen previously. Figure 6.26b shows the displacement at a point located 1.4 cm from the charge center - with and without a 3 mm weak layer. In this case the displacements with and without the weak layer are almost equal, but the weak layer delays the deformation. Figure 6.26c presents information when a 6 mm weak layer is present. Once again the deformation with the weak layer is larger than the deformation that would have occurred without the layer, and the deformation is significantly delayed in time.

Figure 6.27 compares the displacements at 1.2, 1.4 and 1.7 cm from the charge for the two different failure surfaces being used when no weak layer is present. In Figure 6.27a at 1.2 cm the displacement is greater and occurs sooner for the stronger of the two materials. At 1.4 cm in Figure 6.27b there is very little difference in the two displacements, but the displacement occurs quicker in the stronger material. Figure 6.27c shows the results compared at 1.7 cm. Once again, the two displacements are nearly equal but the displacement occurs first in the stronger material. Figure 6.28 shows a comparison of the displacements that occur at the same three locations when there is a weak layer present. Figure 6.28a shows the displacements that occur at 1.2 cm when a 1 mm weak layer is present at 1 cm. The two cases shown are for the two different materials. Notice that the weaker of the two materials ($Y=15-14.5e^{-0.125P}$) results in the larger of the two displacement values. As can be seen from Figures 6.28b and 6.28c, the same is true at 1.4 cm when a 3 mm layer is present and at 1.7 cm when a 6 mm weak layer is present. Since the weaker material results in an increase of deformation as a result of its hydrodynamic behavior, the loss in displacement due to the presence of the weak layers when this weaker material is present is nearly zero - even though there is a significant decrease in velocity.

Figure 6.29 presents information on the deformation behavior for the two different choices of rock properties. Figure 6.29a shows the separation of the two strong rock surfaces adjacent to the weak layer for the weaker of the two different choices (15/1.5). Since the separation values are negative, this figure shows how the two strong rock surfaces approach one another for the different selection of weak layer thicknesses. For the 1 mm layer the original separation of the two surfaces is 2 mm. The data presented in Figure 6.29a show that this separation reduces to 1.6 mm as a result of the passage of the stress wave. On the other hand, for the 6 mm layer the original separation of the two surfaces shown was 7 mm. As a result of the passage of the shock this separation reduces to 5.2 mm. Figure 6.29b shows the same type of results for the stronger selection of rock (30/3). For the stronger rock the response (deformation) to the wave is at a slightly later time, and the deformation values are slightly smaller. For example, for the 6 mm weak layer thickness, the final separation of the strong surfaces is only 1.6 mm as opposed to 1.8 mm for the weaker combination of rock properties. All in all, the difference between the 30/3 rock and the 15/1.5 rock is not great with regard to the separation of the two strong rock layer surfaces.

Displacement @ 1.4 cm

3 mm Weak Layer @ 1 cm

Figure 6.26b. Displacement comparison at 1.4 cm with and without 3 mm weak layer at 1 cm - weaker of two rocks (15/14.5/0.125).

Displacement @ 1.7 cm

6 mm Weak Layer @ 1 cm

Figure 6.26c. Displacement comparison at 1.7 cm with and without 6 mm weak layer at 1 cm - weaker of two rocks (15/14.5/0.125).

Displacement @ 1.2 cm

No Weak Layer

Figure 6.27a. Displacement comparison at 1.2 cm for two rock strengths (30/29/0.050) and (15/14.5/0.125), no weak layer.

Displacement @ 1.4 cm

No Weak Layer

Figure 6.27b. Displacement comparison at 1.4 cm for two rock strengths (30/29/0.050) and (15/14.5/0.125), no weak layer.

Displacement @ 1.7 cm No Weak Layer

Figure 6.27c. Displacement comparison at 1.7 cm for two rock strengths (30/29/0.050) and (15/14.5/0.125), no weak layer.

Displacement @ 1.2 cm 1 mm Weak Layer

Figure 6.28a. Displacement comparison at 1.2 cm for two rock strengths (30/3) and (15/1.5), 1 mm weak layer at 1 cm.

Displacement @ 1.4 cm 3 mm Weak Layer

Figure 6.28b. Displacement comparison at 1.4 cm for two rock strengths (30/3) and (15/1.5), 3 mm weak layer at 1 cm.

Displacement @ 1.7 cm 6 mm Weak Layer

Figure 6.28c. Displacement comparison at 1.7 cm for two rock strengths (30/3) and (15/1.5), 6 mm weak layer at 1 cm.

Separation of Surfaces (Weak Layer at 1 cm)

Figure 6.29a. Separation of strong rock surfaces for weaker material (15/14.5/0.125), weak layers at 1 cm.

Separation of Surfaces (Weak Layer at 1 cm)

Figure 29b. Separation of strong rock surfaces for stronger material (30/29/0.050), weak layers at 1 cm.

Effects of Variations in Material Properties of the Weak Layer.

A series of calculations was conducted to determine the effects of variations in some of the properties of the weak layer only. Figure 6.30 shows the geometry used. The "strong" rock was represented by the weaker of the two materials discussed previously. That is, it had a failure surface defined by $Y=15-14.5e^{-0.125P}$, an elastic pressure of 0.6 Kbar, and a crush pressure of 24 Kbar. A weak layer of 1 mm thickness was placed 5 cm from the charge center and some of the properties of the weak layer were varied. Results are going to be given for a point located 6.5 cm from the charge center. This location is well past the end of the 1 mm weak layer. Figure 6.31 shows the response of the rock when no weak layer is present - a maximum velocity of about 4.9 m/s with a pulse width of about 6.5 microseconds.

Figure 6.32 shows the results of four different calculations. In all four cases, the weak rock layer had a failure surface defined by $Y=3-2.9e^{-0.125P}$. Four different combinations of P_c and P_e were investigated. Case 1 had $P_e=.3$ Kbar and $P_c=12$ Kbar, and Case 2 had twice those values as the pressures ($P_e=.6$ and $P_c=24$) - that is no change from the values used in the strong rock. As is seen from Figure 6.32 there is no effect on the velocity due to the reduction in the failure surface only. The third and fourth cases both had a crush pressure of 2.4 Kbar. The elastic pressure for Case 3 was 0.1 Kbar and for Case 4 zero Kbar. The most severe reduction in the velocity at 6.5 cm is for the case with the lowest elastic pressure. The difference between Cases 3 and 4 show the effect of reduction of elastic pressure. When the elastic pressure was reduced from 0.1 Kbar to zero Kbar, the velocity was reduced from 2.9 m/s to 2.1 m/s. The difference between Cases 2 and 3 is an effect of the reduction of both elastic and crush pressures.

Figure 6.33 shows a comparison for two cases where the weak material failure surface was given by $Y=1.5-1.45e^{-0.125P}$. In this figure Case 1 shows results where the elastic pressure is zero and the crush pressure is quite high - 12 Kbar. Comparing this result with Figure 6.31 shows that the extreme reduction of elastic pressure while still retaining a moderately high crush pressure has very little effect upon the velocity. That is, these changes only reduced the velocity at 6.5 cm from 4.9 m/s to 3.8 m/s. The other case shown in Figure 6.33 shows the effect of a reduction in the crush pressure while holding the elastic pressure constant at zero. In this case the reduction in velocity is quite significant - 1.9 m/s compared to the 3.8 m/s or a halving of the velocity. This indicates that the value of the crush pressure is quite significant.

Figure 6.34 shows a comparison between two cases where the weak material failure surface was changed while the elastic and crush pressures were kept constant. In both of the cases shown, the elastic pressure was zero and the crush pressure was 2.4 Kbar. The two failure surfaces were one with $Y=3-2.9e^{-0.125P}$ and the other with $Y=1.5-1.45e^{-0.125P}$. Since both have the same exponent the one is essentially twice as strong as the other. As can be seen from the figure there isn't a great deal of difference between the two cases. The velocity for the stronger of the two materials was about 2.2 m/s compared to 1.9 m/s for the weaker of the two.

The results of these variations in the properties of the weak materials indicate that the most important properties as far as rock response is concerned in this context is the crush pressure, with elastic pressure being next most important, and failure surface the least important.

Figure 6.30. Geometry used to investigate effects of changes in parameters describing weak layer.

Gage 8 - No Weak Layer 6.5 cm From Charge Center

Figure 6.31. Velocity at 6.5 cm when no weak layer is present.

Figure 6.32. Comparison of velocities at 6.5 cm when changes are made in elastic pressure and crush pressure - failure surface constant.

Figure 6.33. Comparison of velocities at 6.5 cm when change is made in crush pressure - failure surface and elastic pressure both constant.

Gage 8 Velocity vs. Time

Figure 6.34. Comparison of velocities at 6.5 cm when a change is made in failure surface - elastic and crush pressures both held constant.

7. Recommendations

Based upon the calculations we have made with the WONDY Code, the literature we have read on prediction of ground motion from the hydrodynamics codes, and our examination of the core from P and N tunnels we make the following comments and recommendations.

Viewing the core from the vicinity of the tunnels was an eye-opening experience. It is recommended that all LANL personnel conducting calculations to ensure containment or for verification purposes be required to view core from a sufficient number of holes to get a good physical feel for the quality of rock involved in their calculations.

The presence of a surprising amount of extremely weak and friable tuff is of importance in ground motion calculations. This material is too weak to have conventional mechanical testing conducted on it. As a result its presence is completely ignored by the codes. The essence of our calculations and our concern with the fact that the presence of this material is ignored in ground motions calculations is shown in Figures 7.1 and 7.2. Figure 7.1 shows the effect that zones of such weak materials could have on displacements measured at a location of 68 m from a 1-kt device. As is seen in the figure, a single layer of approximately 4 m of this weak material would result in displacements that are one half of what would result if no weak layer were present. This, of course assumes our WONDY calculations were correct and that the normal scaling laws apply.

Figure 7.2 presents predicted displacements when a single layer of weak material 1.33 m thick is present at a location 68 m from the working point of a 1-kt device. As can be seen from the figure, with the weak layer the radial displacement is a little less than 2 cm as compared to a displacement of more than 3 cm if no weak layer were present.

Our examination of the core showed that in the vertical direction there was a very good chance of such layers being present. In the horizontal direction for the N-tunnel core examined there were also several locations where the tuff reached this very low level of strength. It is our feeling that the effects of these weak layers are much greater than would be predicted by just averaging properties - if properties were available for averaging. For these reasons we make the following recommendations.

1. Calculations similar to the ones described in this report need to be repeated with WONDY (or a similar code) once an appropriate damage model has been introduced into the code. Additional calculations need to be conducted with a two-dimensional code as well. The one-dimensional codes are extremely cost-effective to use in parameter studies but at a certain stage their limitations become too severe. Certainly the presence of strong layers above and below such weak layers as we are describing might make a large difference in the signals ultimately seen at gage locations. That is to say, the waves may travel through the stronger rock layer with very little decay and then be transmitted into the weaker layers with velocities and displacements greater than those that arrive through the weak layer alone. These effects could be found by using a two-dimensional code.
2. Data need to be acquired on the elastic pressure and the crush pressures for this very weak tuff at different saturations and gas-filled void percentages. We feel that these data could be obtained originally in a static test but ultimately a method

Figure 7.1. Prediction on loss of displacement due to weak layers in a 1 kiloton explosion.

Figure 7.2. Comparison of displacement with and without a 1.33 m weak layer at 68 m from a 1 kiloton explosion.

needs to be developed to obtain both static and dynamic values - at least at strain rates of 103 in/in. These values should be obtained with and without confining pressures since confinement may play a large role in the results.

3. Some indication of how the failure surface for this very weak material as well as the failure surface for more competent tuff varies with strain rate needs to be determined. It will be especially challenging to devise a test that can be used to find the failure surface for this very weak material - either statically or dynamically. Once again these tests need to be conducted with and without confinement. The codes with good physical roots will need to use a higher strength at higher strain rates and a realistic damage model as the unloading progresses. Data on the behavior of tuff at higher strain rates (competent or weak) simply do not exist at this time.

4. Experimental verification of how this granular material affects outward propagating stress waves needs to be obtained. The results shown here are computational estimates. The properties used for the weak material were our best guesses. What the Hugoniot data are for this material is totally unknown. The values used for the failure surface and for the elastic and crush pressures were only guesses. We recommend that explosive tests similar to those done for the competent rock at SRI be conducted. Layers of the weak material, once it has been characterized, can be placed between larger layers of stronger material and the composite model can then be loaded by an explosive source. We would strongly recommend that stress as well as velocity be measured in the tests. Similar tests have been conducted by the authors [25] in which the volume between the two strong "rock" interfaces was left unfilled. That is, an air gap of known thickness was placed between the two layers. In our tests we used a rock grout for the strong layers which probably represents the competent tuff quite well. These tests could be repeated with the gap between the two pieces of strong grout filled with a sand-like material with known saturation and confinement.

We should indicate that we don't feel that the anomalous behavior of Mission Cyber relative to Disco Elm and Distant Zenith is in any way connected with the presence of this very weak tuff. In fact, the tuff in the vicinity of the working point for Mission Cyber and in a horizontal layer where the instrumentation was located was good competent tuff for the most part. We feel that the experimental measurements obtained from Mission Cyber and Disco Elm should be examined carefully and we plan to do this in a companion report to this one. We feel that other tests that yielded anomalous results should be examined in the same fashion as we have examined the Mission Cyber event. It is our feeling that some of these unexpected results will be shown to be a result of the presence of this very weak rock.

We also believe that the Hugoniot data that currently exist for tuff are not extremely believable. Additional accurate Hugoniot data need to be acquired for a few saturation and gas-filled porosity percentages. Data do exist but the scatter is large and the tuff behavior needs to be better clarified. It is especially important to acquire equation-of-state data for tuff in the pressure range 0.01 to 5.0 GPa.

In closing, we are impressed with what can be done with the hydrodynamics codes in the way of predicting ground motion. We do however feel that at the current time most of the codes do not predict unique results. That is to say there are multiple ways in which the codes can predict the same result. We feel that there is the tendency to let the prediction dictate the input parameters. Care should be taken

to ensure that good material properties are used as input and if agreement isn't obtained then physical reasons for this disagreement found.

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