IN-SITU MEASUREMENT OF ROCK STRESS WITH A BOREHOLE DEFORMATION GAGE

By

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ABSTRACT

The total stress in rock can be measured by any of several in-situ techniques. A method involving the measurement of borehole strain resulting from stress relief by concentric overcoring was used to investigate the total stress at a depth of approximately 2000 feet in the Pea Ridge Mine of Meramec Mining Company near Sullivan, Missouri.

The specific techniques and equipment used for twenty in-situ tests in five boreholes at two locations, and for four laboratory tests to determine the Young's modulus of elasticity of the recovered magnetite and porphyry cores, are described. Mathematical relationships, derived from those of a hole in an infinite plate, are presented together with the field and laboratory data, and the computed secondary principal stresses and stress directions.

Evaluation of the results showed the existence of a horizontal stress component having a magnitude approximately five times that which would be estimated from gravity loads alone.

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INTRODUCTION

The magnitude and direction of each component of the principal stresses in rock should be known for effective design of stable underground openings. Although rock stresses at a point are often estimated by considering the gravity load only, these stresses are also functions of the physical properties and condition of the rock, and the tectonic history of the area. Thus, one of the first problems to confront the rock-mechanics engineer is the determination of the actual stress condition at the point of interest. Absolute rock stresses cannot, however, be measured directly: they must be computed from strain measurements or stress changes following stress relief.

Earliest attempts at in-situ analysis involved strain measurements. In 1933, Lieurance reported that absolute measurements had been achieved by stress relieving a surface-mounted strain gage with a ring of overlapping drill holes. In 1938, Obert reported stress-relief measurements were made by cutting a pillar loose from the roof. The development of methods for making absolute measurements with direct-reading pressure gages soon followed. Mayer, Habib, and Marchand (1951) used pressure gages (flatjacks) in slots cut in the drift wall. Measurements by any of these methods, however, were limited to points on or near the rock surface.

A significant advance on the mechanics of stress measurement was made in 1958 by Hast. He reported the development of a gage which could be placed in boreholes twenty meters or more from the drift wall, and which indicated pressure relaxation following stress-relief by concentric overcoring. Subsequently, Obert, Merrill, and Morgan (1962) with the U. S. Bureau of Mines, reported the development of a gage which measured diametral deformation of a borehole, and which was adaptable to the concentric overcoring method of stress relief. Although this gage had certain inherent disadvantages, its overall reliability was generally recognized.

With the objective of developing an instrument which could be conveniently utilized by industry, N. E. Grosvenor, at the Colorado School of Mines, designed a three-directional gage for measuring borehole deformation. This gage, which was designed during 1962, incorporated the basic mechanical principle of the Bureau of Mines' gage and had features which were intended to further simplify its operation and improve the overall statistical reliability of the data.

The principal objective of this investigation was to evaluate the stress conditions in an operating mine. Equally important, but of necessity relegated to a subsidiary position, was an evaluation of the new equipment. The investigation was conducted at the Pea Ridge Mine of the Meramec Mining Company, near Sullivan, Missouri, which started production during April, 1964. The great depth of the mine, the occurrence of occasional rock bursts, and the Company's wish to investigate alternate mining methods provided appropriate circumstances for conducting a stress investigation. Preparations were begun late in May, 1964, and the field investigation was conducted through September 15 1964.

A complete evaluation of both the field data and the equipment performance would require that results be compared with data derived by other accepted or standard methods.

The basic fundamentals of force and stress provide the relationships which are involved in stress relief methods for absolute stress measurement. Included in this thesis are those considerations which are essential to a general understanding of the analysis. Reference is made to

Merrill and Peterson (1961) for the particular justification for deducing stresses from borehole-deformation data.

For the reader who has need to do similar work, the equipment and procedures employed are discussed in detail and appropriate design drawings are included in the appendix. All data obtained from the tests are presented in the appendix and summarized in the appropriate sections of the report.

THE CONCEPT OF STRESS

There exists within rock a distribution of stresses resulting from past and present loading conditions. Body forces, active loads -- both static and dynamic, and the unrelieved stresses from forces active at some prior time all contribute to the current stress distribution.

The mine engineer confronted with an analysis or design problem often has as an intermediate objective the determination of the magnitude and direction of all stresses in the vicinity of an underground opening. His problem may only require the determination of stresses in a given direction, or it may include a complete investigation of all stresses in the area of interest. He may be interested only in incremental stresses resulting from some structural change, or he may desire a full description of the absolute stress condition.

Regardless of his specific objective, the engineer must recognize the rather nebulous concept of stress.

Strains may be seen and measured; stresses, however, are neither visible nor directly measurable. The existing stress condition must be deduced by combining theories, empirical relationships, and the measured effects of stress.

There are numerous methods available for stress analysis. The method described herein is concerned with the measurement of an elastic response to stress change. Only basic concepts and those relationships directly pertinent to this method will be described in detail. For a more comprehensive review of the theories from which those relationships are derived, the reader will be referred to sources dealing with each such theory. The following paragraphs offer a brief review of the basic mechanics of forces upon which the specific relationships necessary for strain-relief stress analysis are based.

BASIC MECHANICS

Stresses have both magnitude and direction. They are, therefore, vector quantities which can be combined according to the laws of vectors. All parts of a stress system, such as gravity and tectonic stresses, act together, and by superposition result in an effect equal to the sum of the individual effects of the parts. Conversely, the net effect may be described in terms of other, more conveniently chosen components. The effect of all known stresses acting at any given point can be described in terms of three principal orthogonal stresses. It is fortunate that most investigations are not concerned with the origin of stresses as the net effect which is measured cannot reveal the component parts.

A frame of reference must be established if the components of stress are to be suitably described. Rectangular Cartesian coordinates are most easily applied to the solution of stress problems in rock mechanics. The three principal stresses noted above are three orthogonal vectors in the Cartesian system.

Any stress at a point in a solid may be described in terms of equivalent components which act normal to and on the surface of a unit cube at that point in the solid. Specifically, these components include a normal stress and two shear stresses on each face of the cube, or eighteen components. These are shown in Figure 1. To preserve equilibrium, opposing normal stresses and counter-operating shear stresses must be equal. Shear stresses on opposite faces of the cube must be equal in magnitude but opposite in sense. Because of these equalities, it is possible to

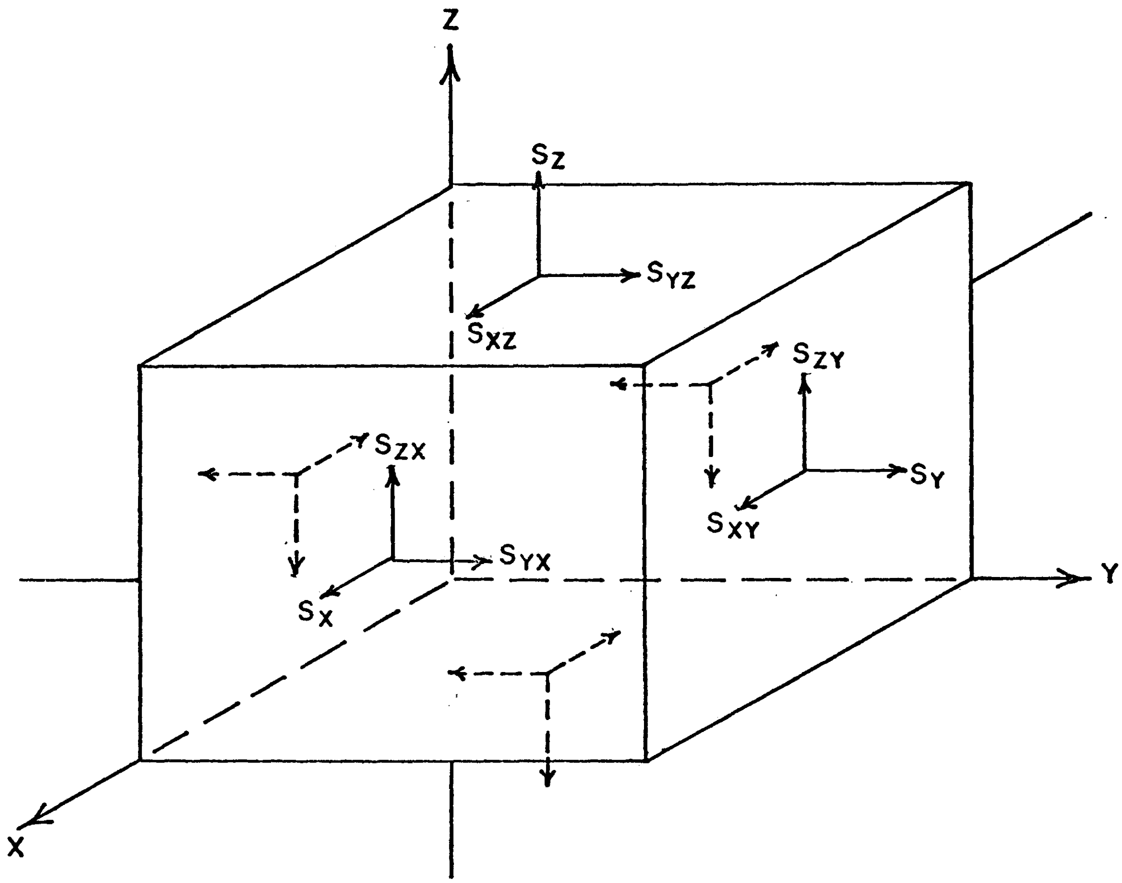


Figure 1. Components of stress at a point.

describe any static stress in terms of six components -- the others being equal and understood.

By rotating the position of the cube, or the reference axes, the three shear stresses may be eliminated and the stress system fully described by three normal components and their orientation. Such a description might represent a solution to an in-situ rock-stress investigation, and is often presented in the form of a stress or strain ellipsoid.

STRESS IN ROCK

When a stress condition is to be investigated in an elastic material, the elastic properties of that material may be utilized to deduce the stress condition according to the relationship:

$$E = \frac{\sigma}{e} \tag{1}$$

where: E = Young's modules of elasticity
σ = stress e = strain

This relationship between stress and strain is linear only for ideal materials. Most rocks are not ideally elastic but exhibit an elasto-plastic response: that is, the stress-strain relationship is neither linear nor independent of time. Generally this is not a serious handicap to rock stress analysis because both the elastic curve and the time-dependent relationship can be determined experimentally.

For most rocks the response may be considered elastic for small strains and short time intervals. Some rocks, such as salts, are important exceptions.

Other general problems associated with the determination of stress in rock often arise because of one or more of the following conditions:

1. anisotropy
2. inhomogeneity
3. discontinuity
4. environment (moisture, temperature, and pressure)

Each of the above may or may not materially affect the response of a given rock to a particular test. The degree to which each condition may have affected this investigation is discussed in section IX, Analysis of Results.

A rock mass, like any real material when subjected to one or more stresses, will deform in three directions when not fully restricted. Conversely, a change in any one boundary condition will result in a stress change on each other boundary which is not permitted to deform. Poisson's ratio describes the relationship which exists between the orthogonal strains in a given mass.

$$\text{Poisson's ratio } (\nu) = e_2/e_1 = e_3/e_1 \tag{2}$$

where: e_1 = strain in direction of applied uniaxial stress

e_2, e_3 = strains in orthogonal, unrestrained directions.

It can be shown that the relationship between stress and strain in a triaxial condition is as follows:

$$e_1 = \frac{1}{E}\left[\sigma_1 - \nu\,(\sigma_2 + \sigma_3)\right] \tag{3}$$

$$e_2 = \frac{1}{E}\left[\sigma_2 - \nu\,(\sigma_1 + \sigma_3)\right] \tag{4}$$

$$e_3 = \frac{1}{E}\left[\sigma_3 - \nu\,(\sigma_2 + \sigma_1)\right] \tag{5}$$

Unfortunately, all instruments presently available for the mechanical in-situ measurement of rock stress (by strain analysis) are either uniaxially or biaxially oriented. Furthermore, absolute in-situ determinations require that some boundary condition be altered during the test. Because the change of one boundary affects the orthogonal boundaries, and because present instrumentation is limited to biaxial measurements, information about the response in the third direction is lost during the in-situ test. It therefore becomes necessary to make an assumption about conditions in this dimension.

Three conditions might occur: the solid might deform without any change in stress in the third or unmeasured direction, the solid might be restricted so that no deformation can occur therefore requiring an adjustment of the stress magnitude, or both deformation and stress-change might occur. The mathematical analysis of an elastic response of a regular plane section can be accomplished or at least closely approximated. A three-dimensional elastic analysis can be completed for only the most regular forms.

Because of the above limitation, the mathematical expressions relating absolute stress to the borehole deformations measured during this investigation are based on plane strain analysis.[1/] These expressions, derived by Merrill and Peterson (1961) for both plane stress and plane strain, are presented in a subsequent section together with other stress relationships specifically related to the borehole stress-relief method of stress analysis.

1/ Merrill and Peterson (p.3) show that when Poisson's ratio is between 0.25 and 0.3, the difference between the deformation for the cases of plane stress and plane strain is about 6 to 10 percent. Plane strain analysis is used in this investigation so that the unknown quantity, Poisson's ratio, need not be introduced.

TECHNIQUES FOR IN-SITU MEASUREMENT OF ROCK STRESSES

There are two basic in-situ methods for evaluating the total absolute stress in rock. One involves the measurement of rock strains resulting from a change in boundary conditions, and the second is based on the functional relationships between rock properties and the stress state. The acoustic velocity of rock, for example, has been shown to correlate closely with the magnitude of the applied stress. Cannaday and Leo (1966) have developed a controlled impulse technique for such investigations. Isaacson (1962) has shown that the electrical resistivity of rock is related to the stress condition of that rock. These methods will not be further discussed due to their limited usefulness.

Instruments for measuring rock strains generally fall into one of several categories depending on the means by which the strain is sensed -- mechanical, electrical, optical or acoustical. Although the instruments vary widely in appearance, sensitivity, reliability, and cost, all perform the same basic function -- that of measuring the rock deformation which occurs as a result of some change in the existing stress or boundary condition.

Two fundamentally different strain methods may be employed. Strains resulting from the complete release of stress at some boundary may be noted and interpreted, or the stress required to completely eliminate strain at a newly created free face may be determined.

STRAIN INTERPRETATION METHODS

Hooke's law shows that stress and strain are related by Young's modules "E", where

$$E = \frac{\text{stress}}{\text{strain}}$$

If E is known, the rebound that follows the removal of stress is a direct indication of the magnitude of the removed stress, with the limitation that total strain has been within the elastic limit of the material.

Early in-situ measurements of absolute stress in underground openings were achieved by fixing strain sensing devices to a rock mass and then cutting that mass completely free from the surrounding rock -- that is "stress-relieving" the mass. The absolute stress in pillars has been determined using this approach by cutting one end of the pillar free for stress relief. Olson (1957) and Utter (1962) concentrically overcored electrical-resistance strain gages mounted on the rock surface to determine the absolute stress near the surface of an underground opening. Obert, Merrill, and Morgan (1962) described a reusable borehole gage which could be overcored. This last-noted gage is basically similar to the one used in this investigation except that it was capable of measuring deformation in only one direction during each setting and overcoring.

STRESS INTERPRETATION METHODS

Typical of the stress-interpretation methods is the "flatjack" method described by Merrill, and others, (1964). Briefly, the method requires that a deformation gage be placed on or within the rock mass to be investigated. The mass is then stress relieved, usually with a slot, and the deformation recorded. Flatjacks or other pressure devices are placed at the point where the rock was stress-relieved and the rock is restressed so that the deformation is completely reversed. The magnitude of the applied stress is an indication of the magnitude of the removed (original) stress. One significant advantage of this method over strain interpretation methods is that the elastic constant, E, of the rock need not be known. Recent applications of the stress interpretation method utilize instruments which can be placed in boreholes and overcored. Leeman (1964), describes a borehole gage designed and used by Hast as early as 1958. The Hast gage is of the solid inclusion or "high modulus" type whereby a stress is maintained on the walls of the borehole as it is being stress relieved. A strain sensor notes the change of stress to which the gage is subjected during the overcoring process.

POINT OF APPLICATION

Some methods and instruments for stress analysis are designed to be used on or near the surface of the rock; others are particularly suited for use in boreholes. Because stresses located close to an underground opening are effected by that opening, and because blast-caused wall fractures would influence the local stress pattern, it is preferable to make stress relief measurements at depth.

From relationships given by Timoshenko and Goodier (1951), it can be shown that a circular opening in an infinite plate subjected to a biaxial stress causes a tangential stress concentration of only six percent at a distance four opening radii from the center of the opening (Figure 2). Field experience on this research indicates that a depth

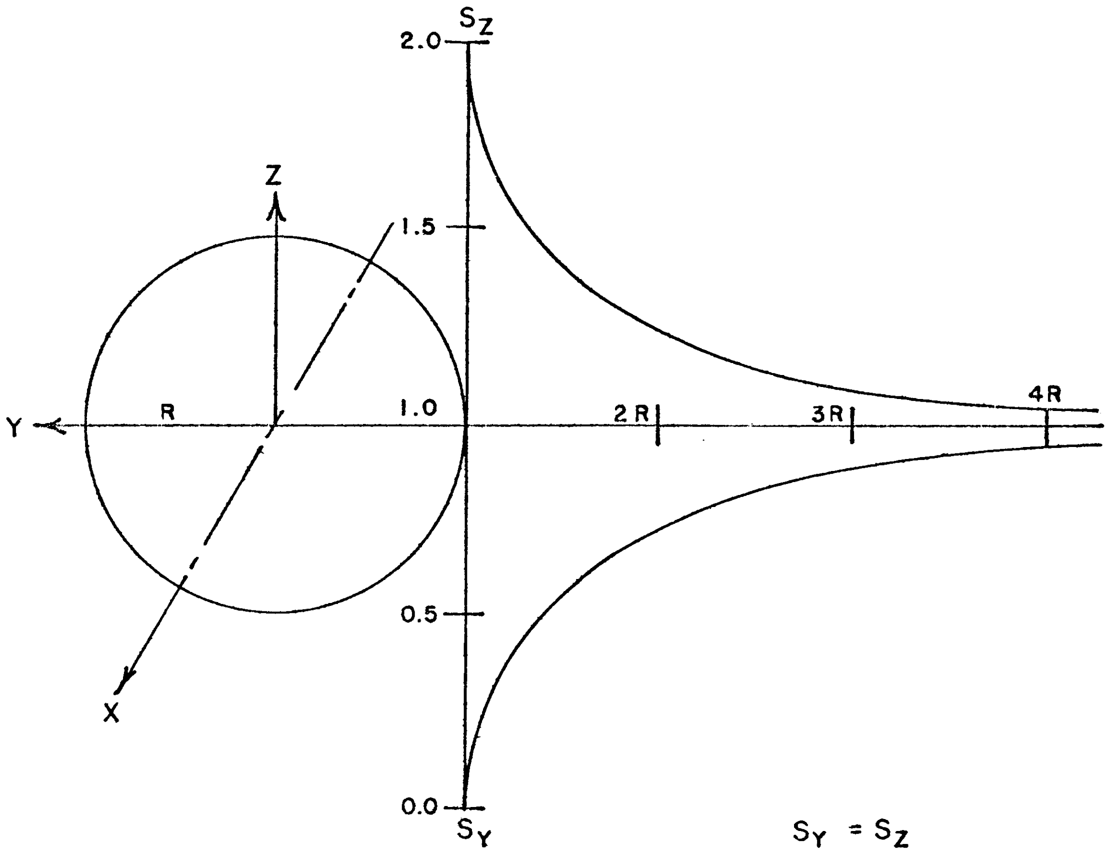


Figure 2. Stress near a circular opening.

equal to one diameter may be sufficient for good stress relief results.

Because of the improved results from stress measurements made at depth, many devices are designed for use in small-diameter boreholes at depths from a few feet to more than twenty feet. Recent work by Wisecarver of the U. S. Bureau of Mines (personal communication) has included strain measurements at horizontal distances over one hundred feet in the rock. Generally, the depth limitation is imposed only by the increasing difficulty of manipulating the equipment, and the decreasing value of additional data points along a single axis or drill hole.

The determination of the maximum, minimum, and intermediate principal stresses, and their direction of action at a point, requires the measurement of strains in at least three orthogonal planes. Two non-perpendicular planes may be sufficient if it is considered feasible to use Poisson's ratio in the stress-strain calculations. Present borehole instruments are limited to measurements either along the axis of the borehole or in a plane normal to the borehole. Extensometers of various forms are used to measure the longitudinal strain of a borehole but are limited to the measurement of changes of strain and stress and cannot measure the absolute condition. In view of these conditions, the most practical technique for obtaining the necessary field data requires that measurements be made in at least three orthogonal boreholes located as close to a point as is practical.

THE BOREHOLE STRESS-RELIEF METHOD

The computation of total stress from data obtained by the borehole stress-relief method involves two theoretical relationships between stress and strain: one is to convert the measured biaxial deformations to secondary principal biaxial stresses, and the second is to establish the stress-strain relationship for the particular rock.

BOREHOLE STRAIN MEASUREMENTS

The interpretation of borehole strain measurements is based on the deformation of a hole in an infinite plate subjected to biaxial plane stress. The equation from Merrill and Peterson (1961) is:

$$U = \frac{d}{E}\,[(S + T) + 2(S - T)\cos 2\theta] \qquad (6)$$

where:

U = deformation of hole (change in length of a diameter),

d = diameter of hole,

E = modulus of elasticity of plate,

S, T = principal and minor stresses applied at 90° separation,

θ = counterclockwise angle from S to r (see Figure 3).

When the above equation is solved for values of $\theta = 0°, 60°$, and 120°, it is shown that the secondary principal stresses, S and T, are related to the diametral deformations as follows:

$$S + T = \frac{E}{3d}(U_1 + U_2 + U_3) \tag{7}$$

$$S - T = \frac{\sqrt{2}E}{6d}[(U_1 - U_2)^2 + (U_2 - U_3) + (U_1 - U_3)^2]^{1/2} \tag{8}$$

where U_1, U_2, and U_3 are deformations at $\theta = 0°, 60°$, and 120° respectively. The complete derivation of this relationship and work substantiating its validity for a hole in rock subjected to both a uniaxial and biaxial stress field was also done by Merrill and Peterson (1961). Their investigation indicated that agreement between the known applied load and the load calculated from the hole deformation measurements was generally better than ten percent. In several instances the agreement was within one or two percent and they concluded that the relationship was valid for stress determinations using the borehole deformation method. They noted, however, that anisotropy of the rock had a considerable effect on the results, and that best results were obtained with the more nearly isotropic materials.

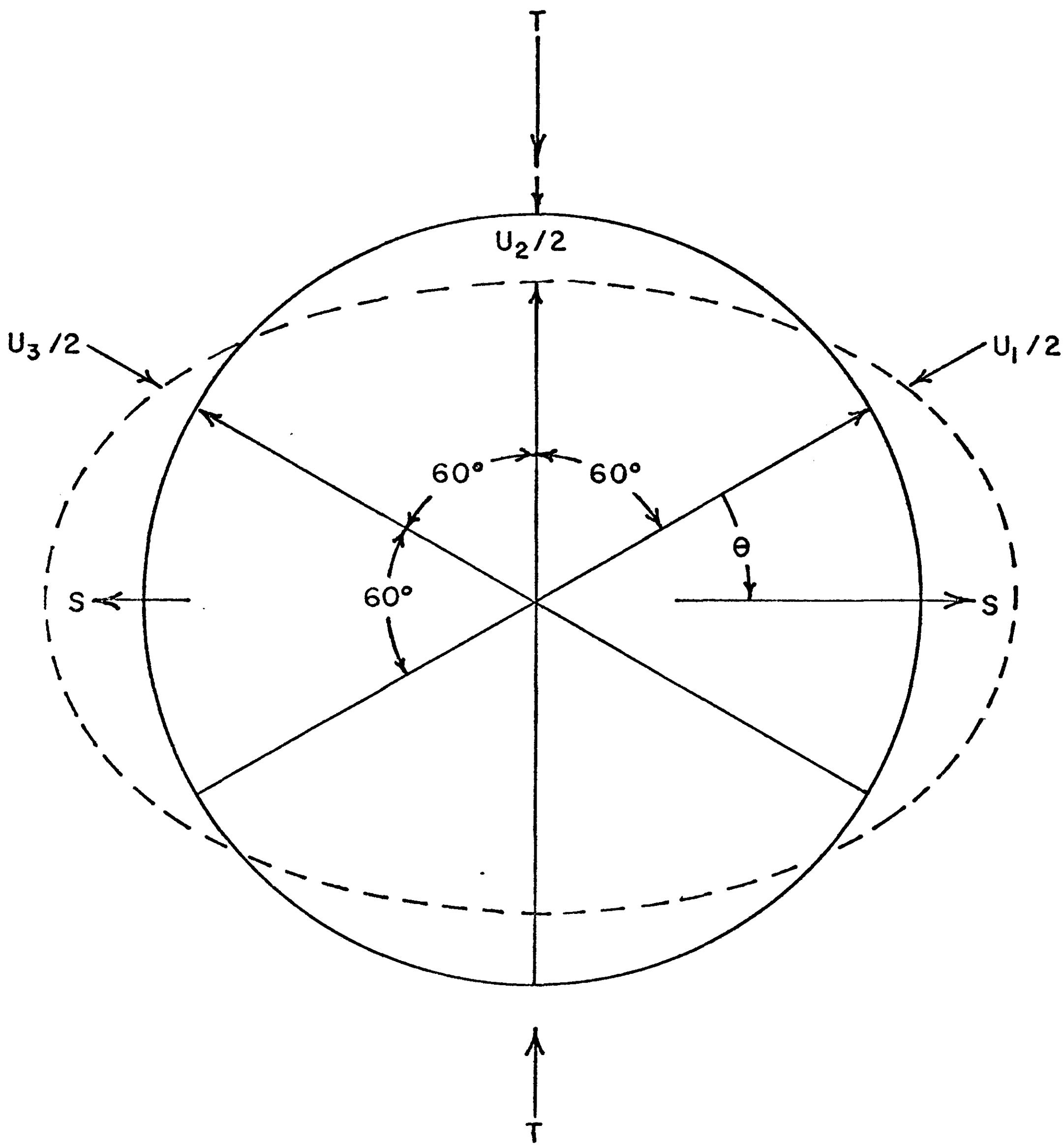


Figure 3. Sixty-degree deformation rosette.

YOUNG'S MODULUS DETERMINATION

Although there are several methods (Cannaday, 1964, and Obert, 1964) for arriving at a value of Young's modulus for use in the deformation equations (7) and (8), the method used for this research was the biaxial method developed by the Bureau of Mines. The advantages of this method are twofold: the conditions of stress in this test most nearly approach the conditions which existed at the time the strain measurement was made, and when possible, the rock tested is the same piece upon which the stress relief test was made. This second condition is particularly important. If unbroken core without voids or inclusions could have been obtained, this Young's modulus test would have been performed on every piece of stress relief core.

The relationships for radial and tangential stresses in a thick-wall cylinder subjected to an external hydraulic pressure form the basis for the test. These equations, as presented by Timoshenko and Goodier (1951), are:

$$\sigma_r = \frac{(P_o - P_i)a^2b^2}{(b^2 - a^2)\, r^2} + \frac{P_i a^2 - P_o b^2}{b^2 - a^2} \qquad (9)$$

$$\sigma_\theta = \frac{(P_o - P_i)a^2b^2}{(b^2 - a^2)\, r^2} + \frac{P_i a^2 - P_o b^2}{b^2 - a^2} \qquad (10)$$

where:

σ_r = radial stress,

σ_θ = tangential stress,

P_o = pressure on outside of cylinder,

P_i = pressure on inner surface of cylinder,

a = inner radius,

b = outer radius,

r = radial distance.

By combining these relationships with Hooke's law for tangential strain,

$$e_\theta = \frac{1}{E}(\sigma_\theta - \nu\sigma_r) \tag{11}$$

and substituting $P_i = o$, and $r = a$, we arrive at the relationship

$$e_\theta = -\frac{2\ P_o b^2}{E\ (b^2 - a^2)} \tag{12}$$

or:

$$E = -\frac{4\ ab^2 P_o}{(b^2 - a^2)\ U} \tag{13}$$

where:

E = Young's modulus of elasticity of the material

U = change in diameter of the borehole.

Fitzpatrick (1962), derives this relationship fully, and presents the results of several laboratory tests which he used to test the application of the relationship to the biaxial method for determining the elastic constants of rock.

DESCRIPTION OF MINE

All field work for the stress analysis reported here was done at the Pea Ridge iron mine of the Meremac Mining Company: a joint-venture operation of Bethlehem Steel Company and St. Joseph Lead Company. The mine is located approximately twenty miles south of Sullivan, Missouri, and north of the Old Lead Belt. Development of the mine was started in October, 1957, and the first iron ore pellets were produced in February, 1964. Planned production was 12,000 tons of ore per day and two million tons of pellets per year.

GEOLOGY

The orebody is a near-vertical, tabular mass, several thousand feet long, up to 600 feet in width, and with an undefined lower limit. The deposit strikes approximately N 60°E. The ore is primarily magnetite with minor lenticular inclusions of specular hematite. Pyrite, calcite, and apatite occur as accessory minerals throughout the mass. Horizontal limestone, sandstone, and dolomite beds with an aggregate thickness of approximately 1300 feet overlie the Precambrian basement rock which contains the ore mass. The magnetite cuts through a series of rhyolite porphyry flows with sericite occurring at the ore contact on the footwall side. Figure 4 is an idealized section through the orebody.

MINE DEVELOPMENT

The mine is developed with two vertical shafts approximately 2500 feet deep. Production levels are at 150 foot intervals starting at the 1675 foot level. Development includes a main haulage line in the footwall which closely parallels the long axis of the orebody, and alternating ventilation and stope-haulage cross cuts normal to the long axis. The stopes are 40 feet by 120 feet in horizontal section with the long dimension approximately parallel to the axis of the orebody.

Ore is broken in the stopes by horizontal ring drilling from three raises: one in the center of one side and one at each opposite corner. The broken ore is drawn through finger raises to a slusher drift and scraped directly into 20-ton, bottom-dump cars. The car dump incorporates the new Swedish design whereby a fifth wheel on the car bottom

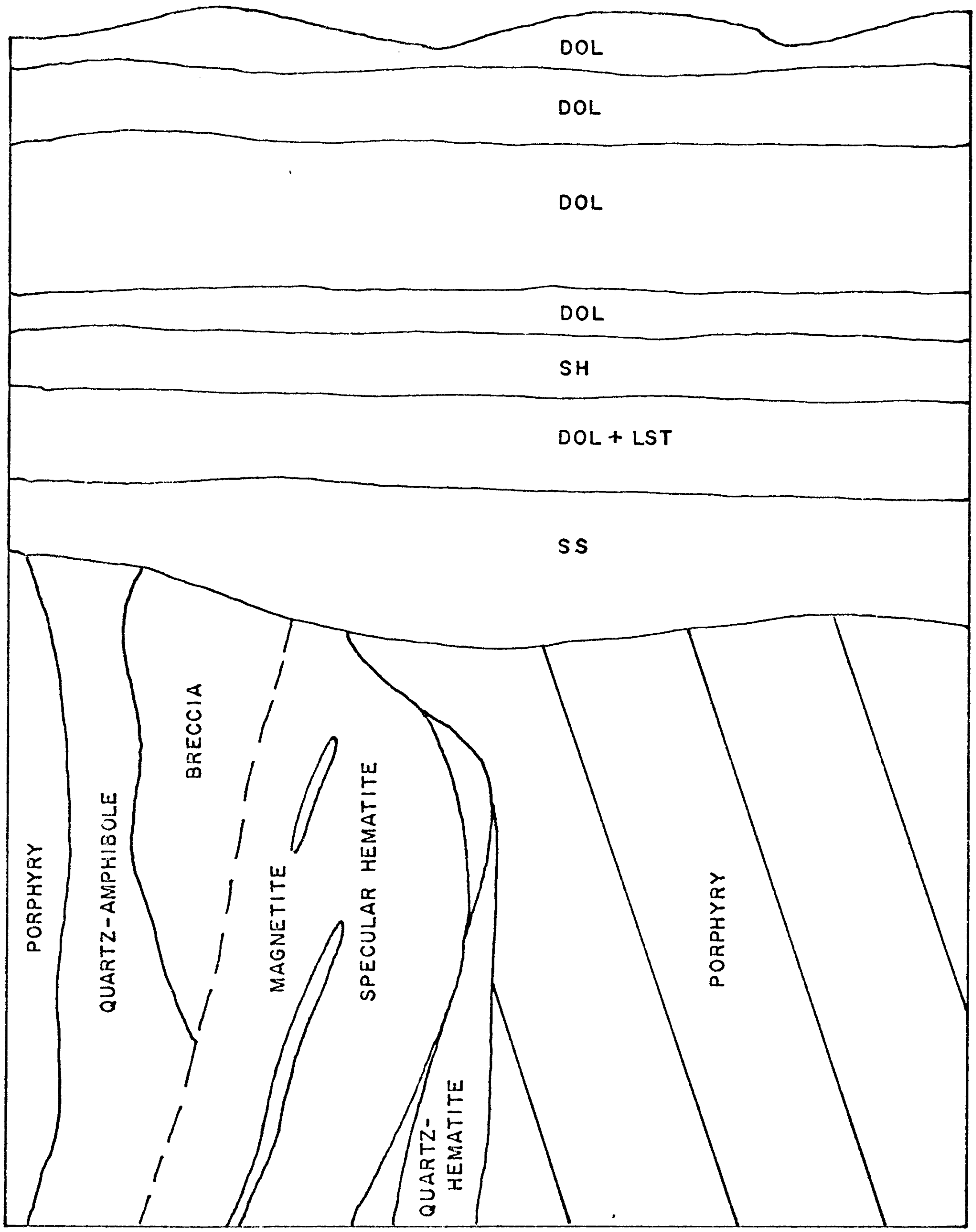


Figure 4. Idealized section through Pea Ridge orebody.

rides a lowered track which allows the bottom of the car to fall away as the car passes over the dump pocket. Ore is hoisted to the surface in two counter-balanced, 19-ton bottom-dump skips powered by a friction-drive hoist. All crushing and grinding is done on the surface. With the exception of a small surge stockpile, the ore moves directly into the mill where the hematite fraction is removed and separately concentrated and thence to the pelletizing plant.

The mine run ore which averages fifty-five percent Fe, is transferred to the mill where magnetite, hematite, and pyrite concentrates are selectively floated. The magnetite and hematite fractions are then blended as feed for the pelletizing plant.

DESCRIPTION OF TEST EQUIPMENT

The principal components of the measuring equipment used for the in-situ tests are described in the following paragraphs.

THE THREE-DIRECTIONAL BOREHOLE-DEFORMATION GAGE

The borehole deformation gage, Figure 5, was of recent design and had not been field tested prior to this research. Design details of the gage are given in Appendix II.

The gage consisted of six mechanically-independent cantilevers connected at 60 degree increments to a single body. A beryllium-copper alloy (Berylco 25) was chosen for cantilever material because it had desirable characteristics including a low and constant modulus of elasticity. The body was constructed of stainless steel. Mounted on each cantilever were two Baldwin-Lima-Hamilton type FAB 12-12 resistance strain gages; one on the top and the second on the bottom. The four resistance gages on each diametrically-

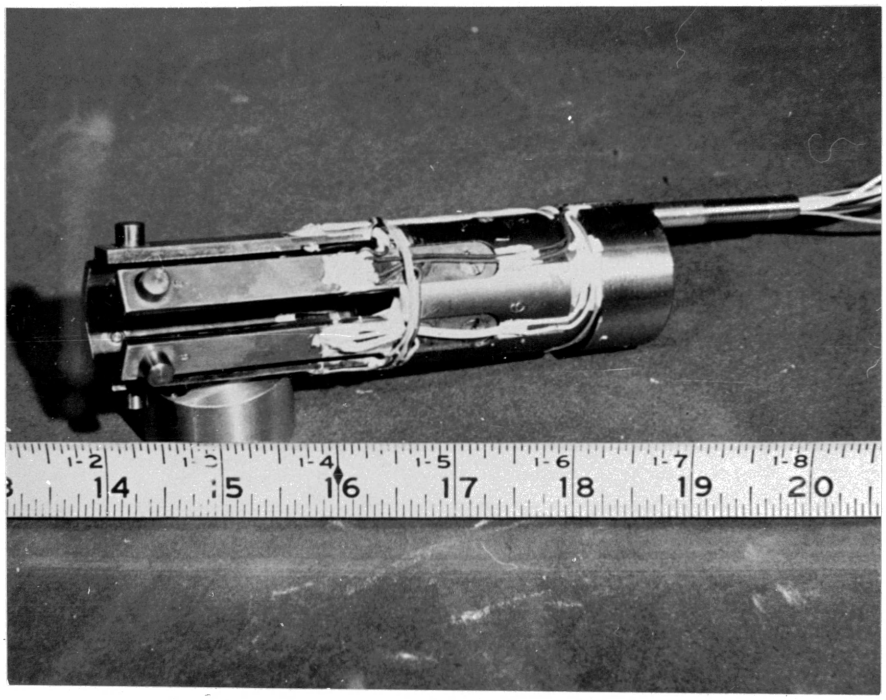


Figure 5. The borehole deformation gage.

opposed pair of cantilevers were connected as a four-arm, temperature-compensated Wheatstone bridge. The circuit used on the original gages is shown in Appendix II-c. The dotted lines in the figure represent a later modification which eliminated two lead wires from the gage. The entire gage, excluding the contact points of the cantilevers, was encapsulated with Gates Rubber Company type Am-2214 polyurethane cover to exclude moisture.

The borehole gage was inserted and positioned in the EX diamond-drill hole with a string of specially-constructed rods. These rods were made of solid steel bar 0.75 inches in diameter with machined male and female ends. The male end of each rod was 2.5 inches long and 0.551 inches in diameter. The rods were held securely together by taper pins driven through the joint. The distance between pins was exactly 34 inches.

The flexibility and weight of the insertion rods resulted in considerable bending of the long string of rods that was necessary as the work progressed into the hole. The bending, in turn, made it difficult to insert the gage into the smaller EX hole at the back of the 6-inch hole. If a new set of rods were to be made, hollow, lighter, and stiffer rods should prove to be just as strong and should facilitate the work.

The rotational position of the gage in the hole was established with a "T" handle on the end of the inserting rods. The "T" was 12 inches long and was secured to the rods with a taper pin so that its orientation relative to the gage would be constant. A positioning tool, such as that described by Wisecarver (1964) would be of considerable help in setting the gage.

The response of each pair of cantilevers on each gage was measured with the equipment described in the following section. The results of the calibration tests indicated a gage sensitivity (factor) of approximately 20 micro-inches of gage deflection per micro-inch/inch of indicator strain. The measured response of each gage is presented in Table I. The response for gage No. 1 over the full, design-deformation range is shown in Figures 6, 7, and 8. The response for gage No. 2 was similar. The calibration was, for all practical purposes, linear and without hysteresis.

TABLE I

GAGE FACTORS

Micro strains/u-in./in. Indicator Strain

Gage No.	Cantilever No. 1-1	2-2	3-3
1	22.4	22.1	22.0
2	22.4	20.8	28.2
2 (recalibrated)	26.2	22.9	29.6

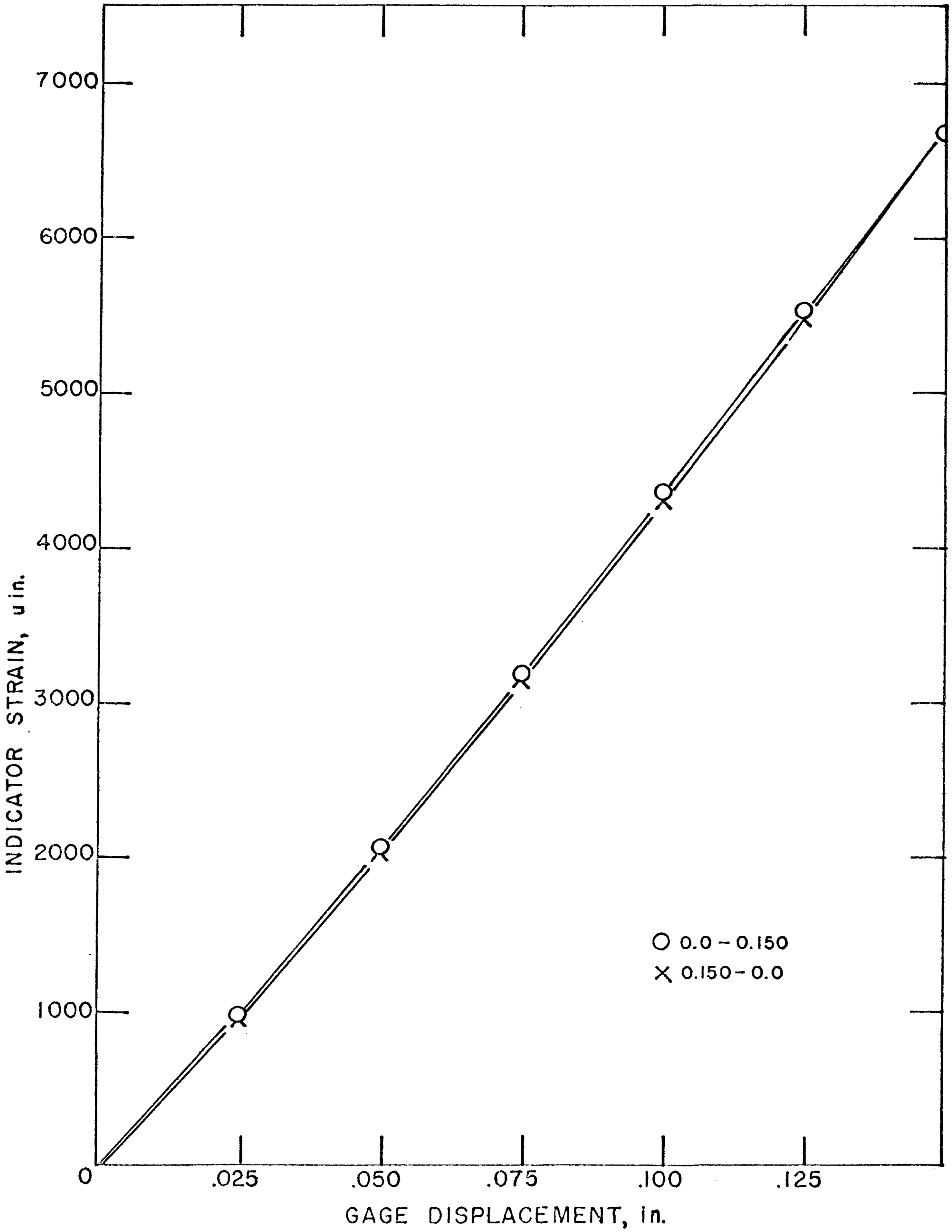


Figure 6. Calibration curve, gage 1, cantilever pair 1-4.

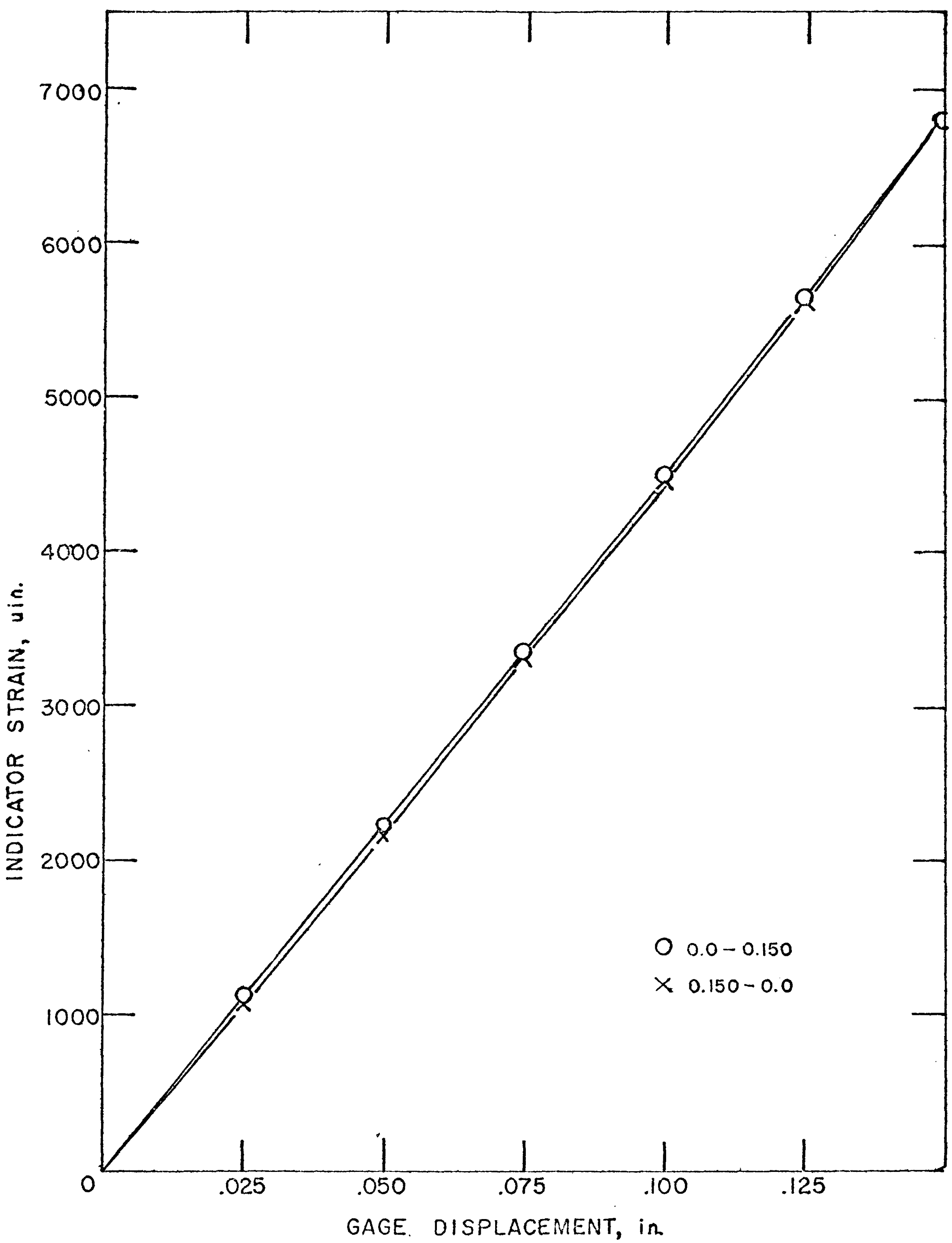


Figure 7. Calibration curve, gage 1, cantilever pair 2-5.

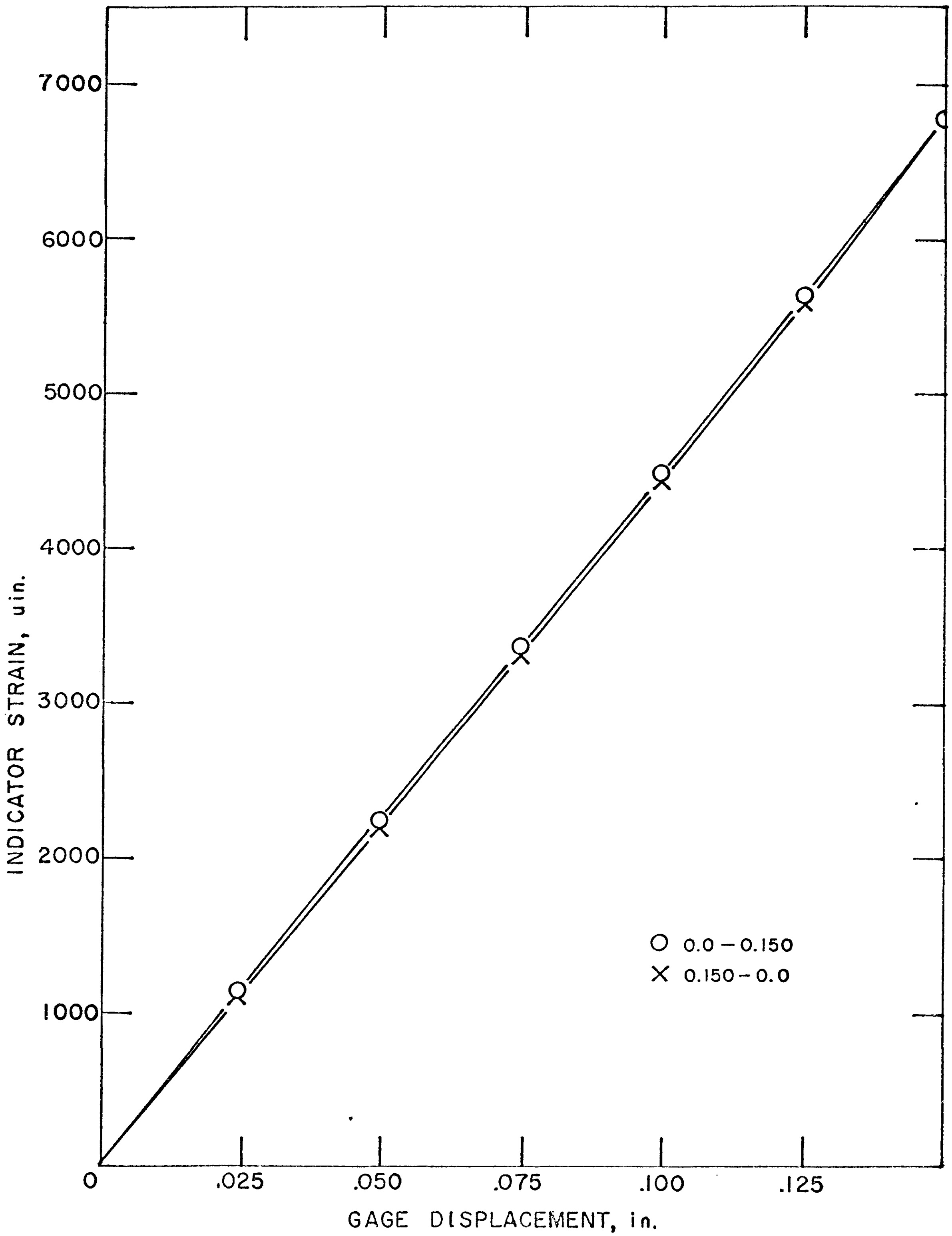


Figure 8. Calibration curve, gage 1, cantilever pair 3-6.

GAGE CALIBRATING DEVICE

The gage response as given above was determined by using a micrometer-type calibrating device. This device was constructed so that the gage body could move freely as the micrometer was adjusted, thus allowing both cantilevers to deflect an equal amount and thereby approximate the condition in the stress-relief hole. The micrometer stem was calibrated to 0.001 inch. An adjustable stop allowed the micrometer to be set at zero at the beginning of each calibrating sequence; a feature which considerably facilitated the procedure.

The gage factor may also be determined by "testing" a core of known Young's modulus in the modulus device described in a following section. The same equation (13) employed for Young's modulus determination is used and is solved for deformation instead of the modulus of elasticity. An aluminum "core" is most appropriate because it has a relatively low modulus and therefore a greater gage deflection per unit of pressure.

A deformation gage of a modified design was tested using both the micrometer calibrating jig and a 6-inch aluminum "core" in the biaxial chamber. The results of these calibration tests are presented in Figures 9 and 10 to show the correlation between the two calibration methods.

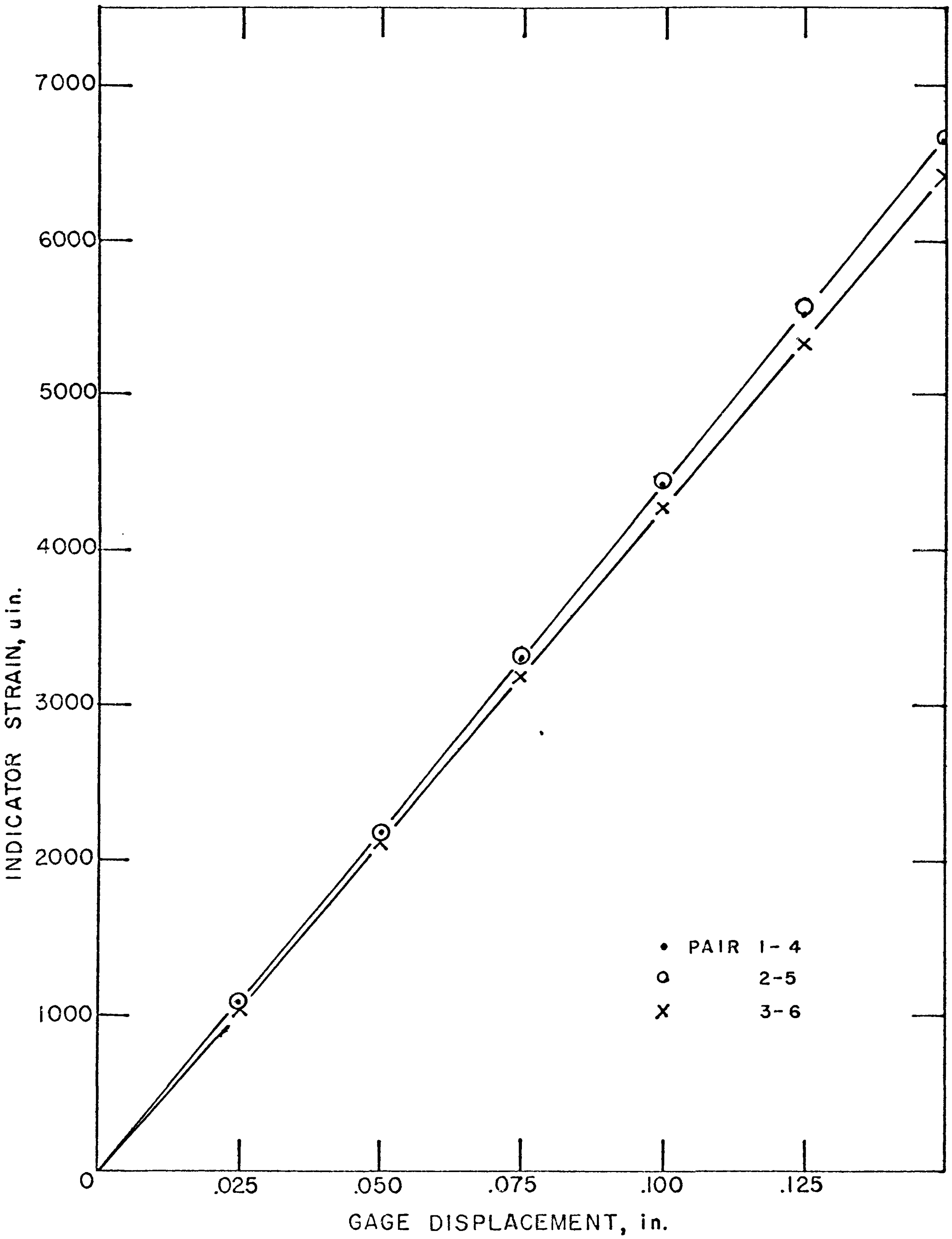


Figure 9. Calibration curve, gage B-2, using micrometer.

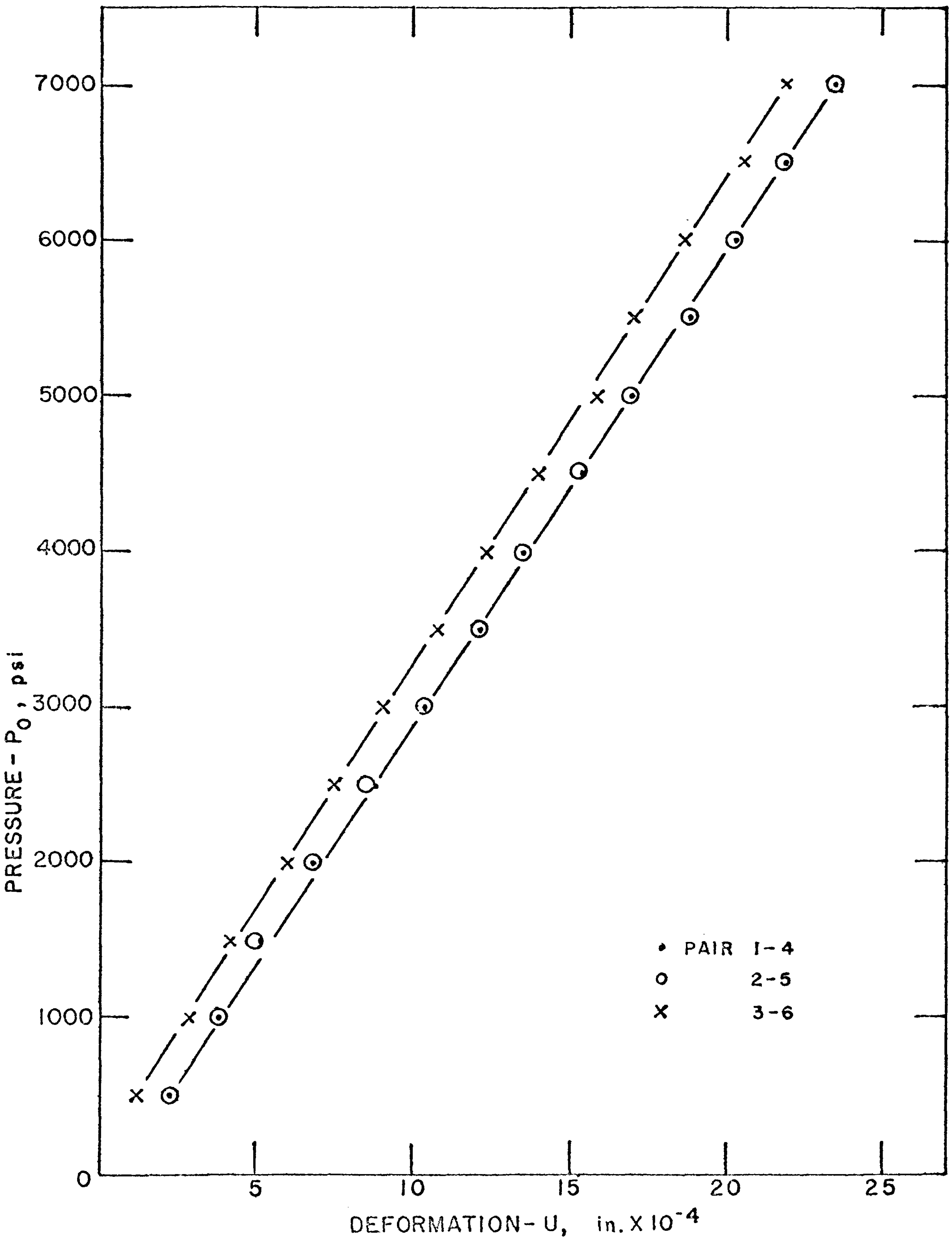


Figure 10. Calibration curve, gage B-2, with aluminum core.

STRAIN INDICATOR

Gage resistance changes were sensed with a Baldwin-Lima-Hamilton SR-4 Strain Indicator, type N; a transistorized unit which can be operated on either batteries or on 110 V AC power. The instrument used was not waterproof and required extra care to assure that moisture did not enter during storage, transportation or use. Condensation could have occurred but it is believed that this was not extensive because the instrument was used and stored in areas of near-equal temperature.

SWITCHING AND BALANCING UNIT

A Baldwin-Lima-Hamilton Switching and Balancing Unit, Model No. 225, was used between the measuring gage and the BLH Strain-gage Indicator. Although this instrument was not essential, it permitted balancing all three Wheatstone Birdge circuits. This balance in turn permitted rapid switching from one signal to another so that a continuous check could be made on all three signals as stress-relief drilling progressed. Where drilling advance was rapid, there would not have been time to make this check if a balancing unit had not been employed. Again, this instrument was not waterproof, and care was exercised to prevent moisture from entering.

YOUNG'S MODULUS DEVICE

Figure 11 shows an hydraulic device which was used to establish the modulus of elasticity of the six inch diameter cores from the stress-relief tests. The instrument was constructed in the Colorado School of Mines shops from a design described by Fitzpatrick (1962) of the Bureau of Mines.

ACCESSORY EQUIPMENT

The accessory equipment used during the field tests included a skid-mounted diamond drill with hydraulic feed, water pump, hand tools, electrical repair and test equipment, and drill bits. All items were commercially available and need not be described here. It should be sufficient to point out that the bit for the pilot hole was an EX diamond bit with a grit which had proven effective in the particular rock being cored. A reamer shell was found to be necessary for a proper fit of the gage in the hole. The stress-relief bit was a Christenson nominal 6-inch thin-wall bit with a 24-inch core barrel and was of the type which is used on an expanding mandrel. The only two parts of the drilling equipment which were designed specifically for the job were the water swivel and a pilot bar for collaring the 6-inch bit.

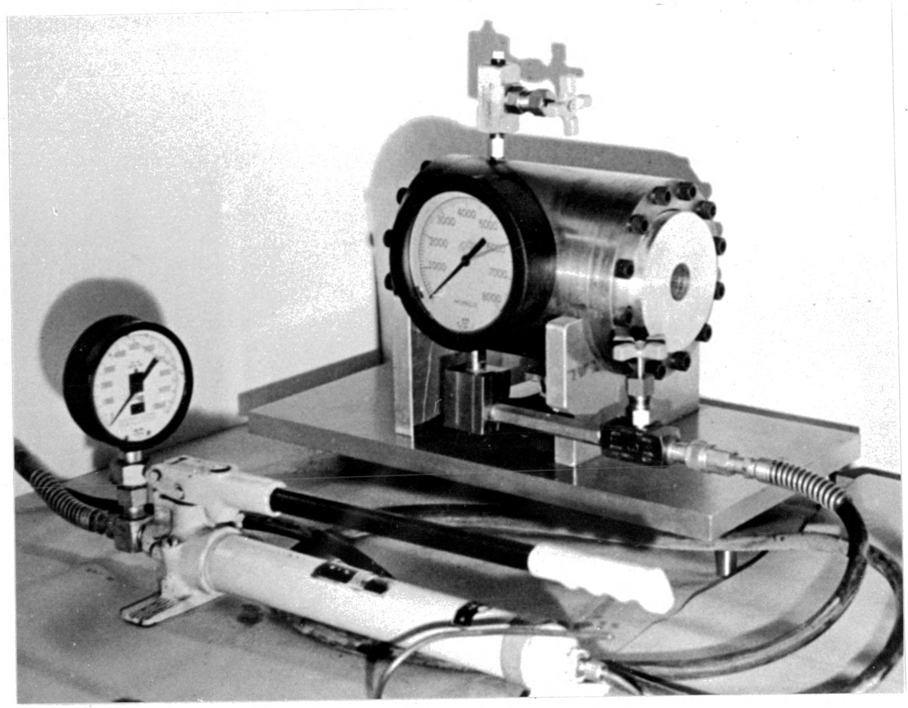


Figure 11. Young's modulus device.

The water swivel was prepared from a standard EX-rod swivel by adding a packing gland at the back end of the signal cable. The pilot bar was used only for collaring the 6-inch bit. It was constructed of 1.25 inch O.D. pipe which was welded closed at the front end and threaded at the back end to mate with the threads of the bit mandrel. The bar protruded beyond the front of the bit 6 inches.

MEASUREMENT PROCEDURES

The techniques which were used for strain measurements, Young's modulus determinations, and gage calibrations are discussed below. These techniques were developed as the work progressed, and represent the best approach that could be established within the limitations of this specific research.

STRAIN MEASUREMENT

The stress-relief procedure included the following principal steps.

Preliminary Work

The diamond drill was set level in the drift and secured using chains with turnbuckles fastened to rockbolts and to each corner of the drill. It was found that drill pressures were such that some drill movement occurred regardless of the tie-down, and minor readjustments were necessary to maintain correct alignment of the drill rod in the 6-inch hole. When misalignment occurred, the axis of the overcore bit tended to drift away from the axis of the pilot hole.

The pilot hole was drilled to a depth greater than the desired depth of strain measurements. This was done for several reasons: increasing the depth of the pilot hole after some overcoring had been completed would necessarily involve a condition in which the drill rod would be unsupported to the depth of the 6-inch hole. The additional depth of the pilot hole provided a receiver for small pieces of rock broken during the work. The EX core recovered from the extra depth also provided information about the rock type and condition immediately beyond the limits of the tests. The position of the pilot hole was chosen so that it and the orthogonal holes would be in favorable positions with regard to visible fractures. All EX core was saved and appropriately marked as to depth. This core provided information about the type and condition of the rock to be encountered.

Manipulation of Gage

The gage was placed in the pilot hole to a predetermined depth with the gage inserting tool. The gage depth was chosen on the basis of the depth of the 6-inch bit kerf, the condition of the rock, and the need for closely-spaced data points. The minimum spacing of data points was limited by the length of the gage to a minimum of 6 inches. It was essential that the gage be completely inserted into the pilot hole so that adequate support was achieved and so that pieces of rock inside the bit barrel would not hit the gage.

The orientation of the gage in the hole was established by noting the position of the "T" handle on the inserting rods. A carpenter's level and a 30-60-90 triangle allowed selection of any gage position. During tests in the first two holes the gage was set with a constant orientation. It was realized, however, that with a constant orientation, a minor but consistent malfunction of one pair of cantilevers might go unnoticed. Therefore, during all subsequent tests the gage was alternately rotated either 60 or 90 degrees. Thus, a malfunction might be detected as a result of the alternate rotation of the computed stress direction. Furthermore, such rotation would permit the computation of stress at intermediate data points by interpolation in the event a pair of cantilevers should become inoperative. This computation can be made, however, only when the gage is rotated 60 or 120 degrees because the cantilever spacing is 60 degrees.

Stress-Relief Drilling

Before drilling was begun, but after the drilling water was running, the signal from each pair of cantilevers was balanced to 11,000 u-inch of indicator strain using the BLH switching and balancing unit. If a signal change occurred after drilling was started but before 0.5 inch had been drilled, the signal was rebalanced to 11,000 u-inch. The magnitude of such changes, which seldom occurred, was usually less than 5 u-inch of indicator strain. It was presumed that they resulted from a minor adjustment of the gage in the hole brought about by the vibration of the drill. The drilling was continued to a point 0.5 inch to 1 inch beyond the point where the indicated strain became constant. The final strain reading was made with the drilling water still on.

During certain tests, particularly those where poor rock was encountered, strain was recorded at 0.5 inch intervals as the stress-relief drilling proceeded. The strain curves thus developed permitted a better evaluation of the data and facilitated an estimate of the final strain in those instances where some malfunction occurred before the test was completed. This procedure is recommended for each test when conditions permit. Figure 12 illustrates a strain curve where, unfortunately, only two cantilever pairs were operative.

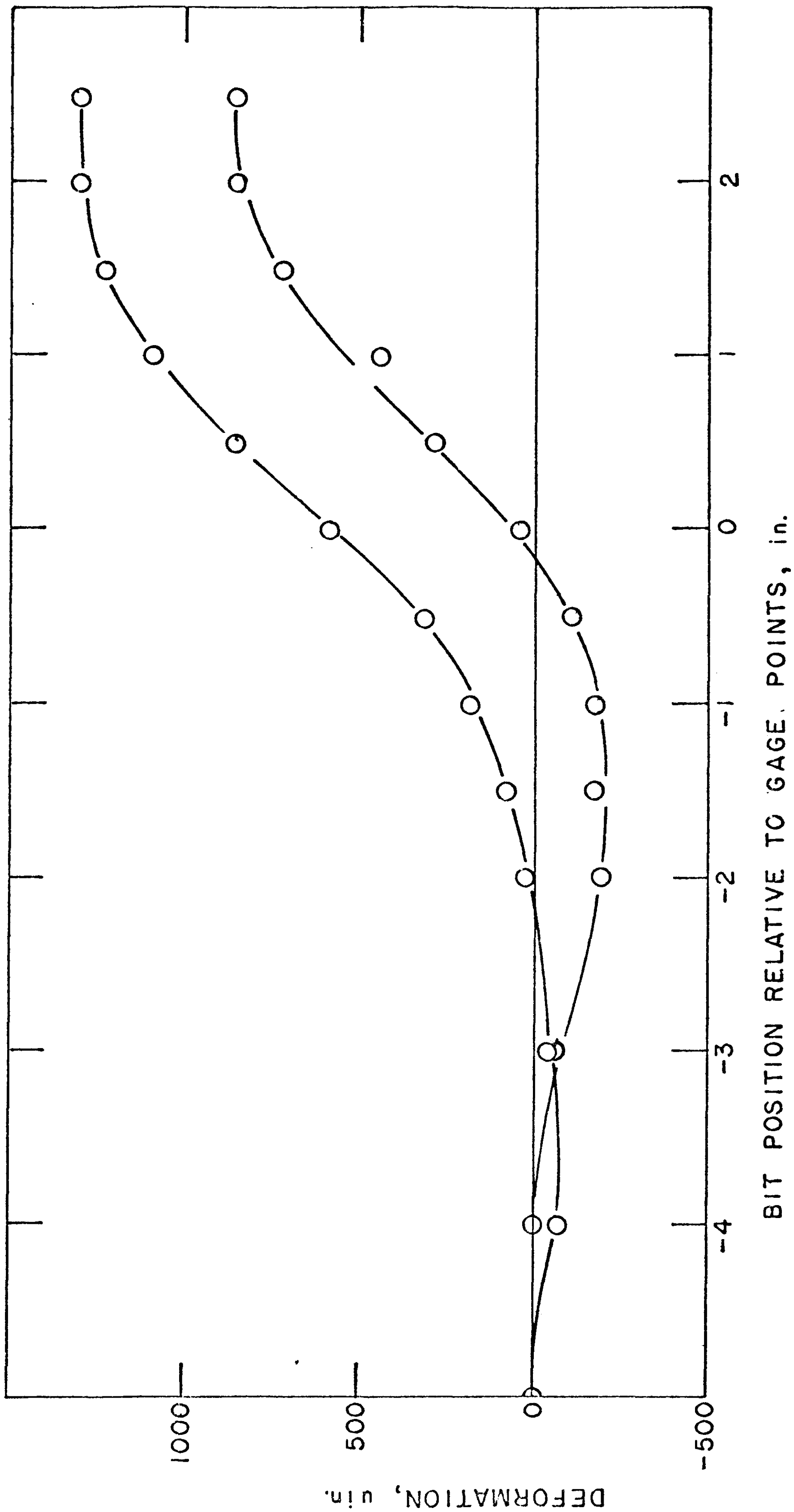


Figure 12. Typical gage response during overcoring.

All holes, except the first, were collared with a short 12-inch core barrel having a center-mounted pilot bar which protruded approximately 6 inches beyond the end of the bit. Alignment of the 6-inch hole with the pilot hole was maintained by using hardwood wedges in the collar of the hole to force the drill rod in the necessary direction, and by realigning the drill as necessary. Poor alignment occurred only in the first and second holes drilled. In the other holes the eccentricity of the centers was less than 0.25 inch.

Drilling speed varied greatly. The drill was rotated at the highest speed which could be maintained without excessive vibration of the unsupported drill rod. As the hole deepened it was necessary to drill at a lower speed to prevent excessive vibration. Average penetration of the bit in the magnetite was approximately 1 min per inch. Penetration in the syenite porphyry varied from 2 min per inch to more than 8 min per inch as the bit became worn. A high bit pressure was maintained once the bit had penetrated its full length. The actual pressure maintained was not measured.

Many cores broke or separated on fracture planes during the overcoring process. If the core did not break close to the bottom of the bit kerf, a wedge-type core breaker was used to release the core after the test was complete. Initially the 6-inch cores were removed with a barrel-type core extractor. This was both time consuming and difficult. It necessitated removing the gage cable and the bit from the drill rod and then attaching the extractor. The extractor could not be used by hand as it fit tightly in the hole and could not be forced over the core. It was subsequently found that all unbroken cores including those in the vertical (down) holes could be removed by driving a tapered bit-fishing tool wrapped with cloth into the pilot hole of the core. Care was used and no cores were broken using this method. The top, front and depth was marked on each core as it was removed. Because it was necessary to protect the gage cable from damage, all badly-fractured rock was drilled out without the gage in place.

YOUNG'S MODULUS DETERMINATION

Tests to determine the Young's modulus of elasticity were made on six magnetite cores and one porphyry core. The tests were made with the biaxial device previously described. Each core was selected for its uniformity and freedom from voids. It is recognized that by making such a selection the results may be somewhat biased. This condition was unavoidable, however, because imperfect cores failed before adequate pressure could be applied.

Each core was wrapped with a double layer of heavy polyethylene sheet which was overlapped 4 inches and secured with wide Scotch tape. The first two cores tested were subjected to pressure in increments of 500 lbs. Subsequent cores were pressurized in increments of 200 lbs. and cycled to zero. The upper pressure limit on a particular core was increased within successive cycles until that core failed. The three-directional gage was used in the pilot hole of the core to measure diametral strain. The loading rate during the test was not constant as it was necessary to stop loading while the strain readings were taken at each load increment.

GAGE CALIBRATION

Both gages used during the tests were calibrated prior to the testing program and recalibrated following the completion of tests in hole No. 3. The gages were calibrated on the jig previously described. To calibrate a gage, the indicated strain was noted for known cantilever deflections at 0.025-inch increments from 0.0 to 0.150 inch. All tests were cycled from 0 deflection to full deflection and back to zero. All tests were repeated until readings for comparable deflections on each leg of a cycle varied by less than 100 u-inch of indicated strain. The gage factors were computed by using the secant to the deflection curves through 0.0 inch and 0.150 inch deflection.

The results of the calibration tests are reported under the section "Description of Equipment."

RESULTS OF TESTS

The data obtained from the in-situ measurements and Young's modulus tests are presented in Appendix I. Summaries and comments about these data are presented in the following paragraphs.

IN-SITU TESTS

Deformation measurements were made along 5 borehole axis. Four of these were located in the magnetite (ore) rock and one was in footwall porphyry. The specific locations are shown in Figure 13. The locations were chosen on the basis of accessibility, and after proper consideration of related workings in the mine. It was assumed that the large production stopes could have a substantial effect on the total stress existing in their immediate area. Because the natural, undisturbed absolute stress was the quantity sought, it was deemed necessary that the tests be located as distant as practical from these stopes. The actual

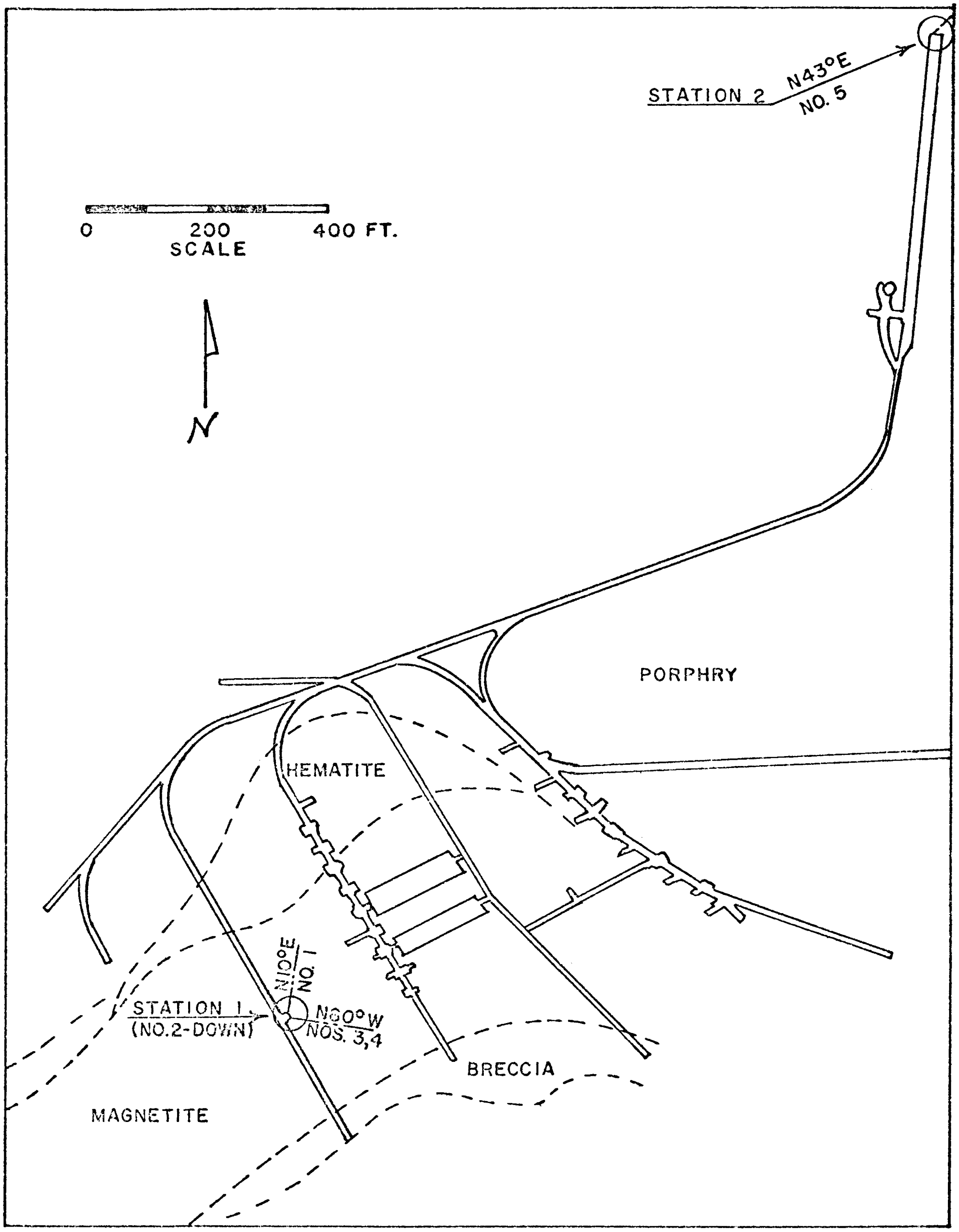


Figure 13. Location of stress-relief tests.

distance from station 1 to the nearest developed stope was approximately 200 feet and from station 2 the distance was in excess of 500 feet.

The reliability of the data, as judged by its continuity and the nature and homogeneity of the rock, varied from good to worthless for the purpose of interpretation. A discussion of the individual data is offered below.

Hole 1, Station 1 (Horizontal)

The data from this borehole was the most consistent obtained throughout the tests. Reasonable variations were noted in both magnitude and direction of the secondary principal stresses as computed from measured borehole strains.

The core from this hole contained minor inclusions and vugs but no continuous fractures in the first 102 inches. Beyond 102 inches a fracture was encountered which continued to the ultimate depth of the hole. Figure 14 shows the computed secondary stress magnitude and direction. Figure 15 has been included to indicate the effect of the fractures encountered on the strain data obtained. It can be noted that at 70 inches the indicated strain is lower than the strains at adjacent points; apparently as a result of the vug which is visible in the photograph. The flat-lying

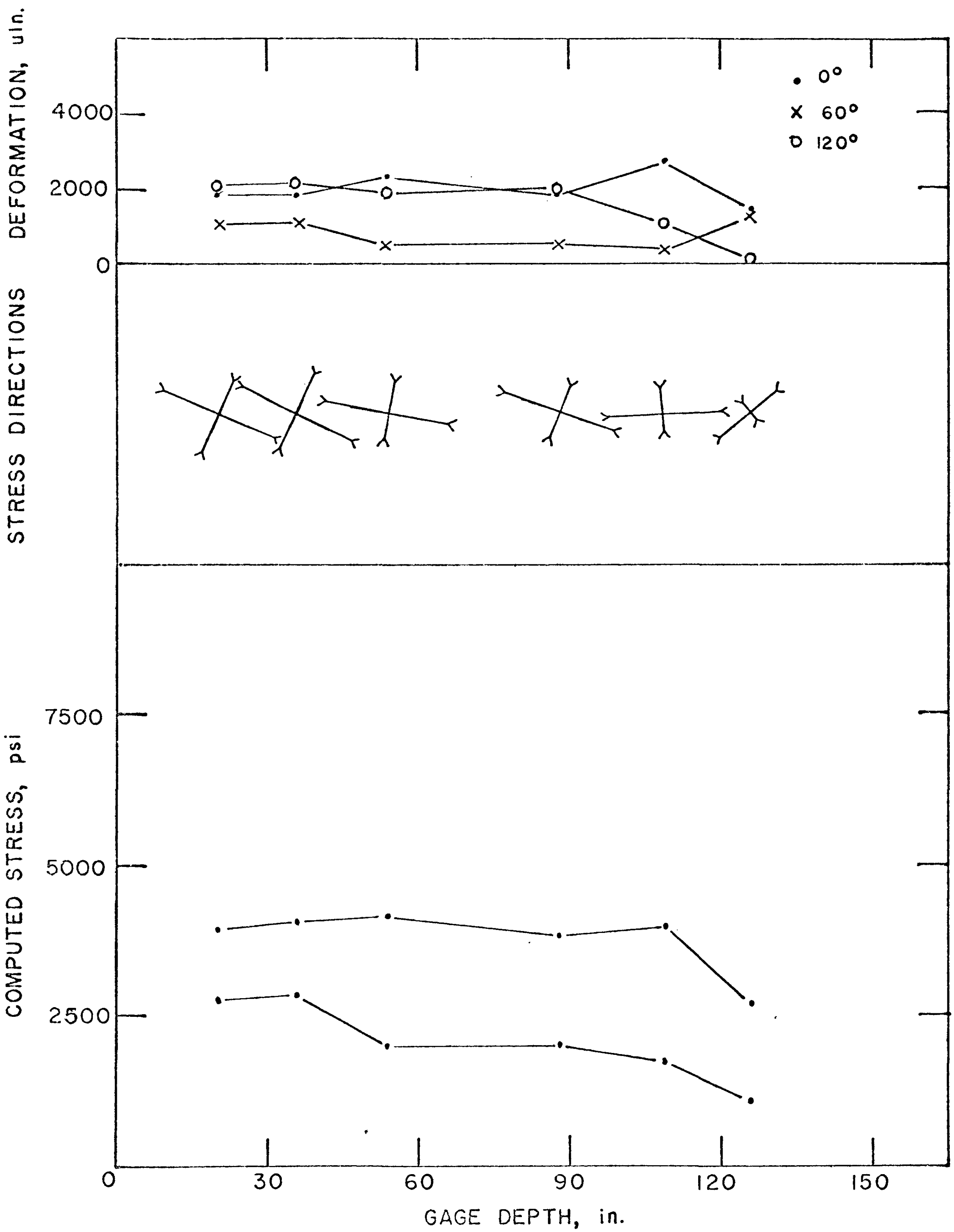


Figure 14. Stress magnitude and direction, Hole No. 1. (Horizontal)

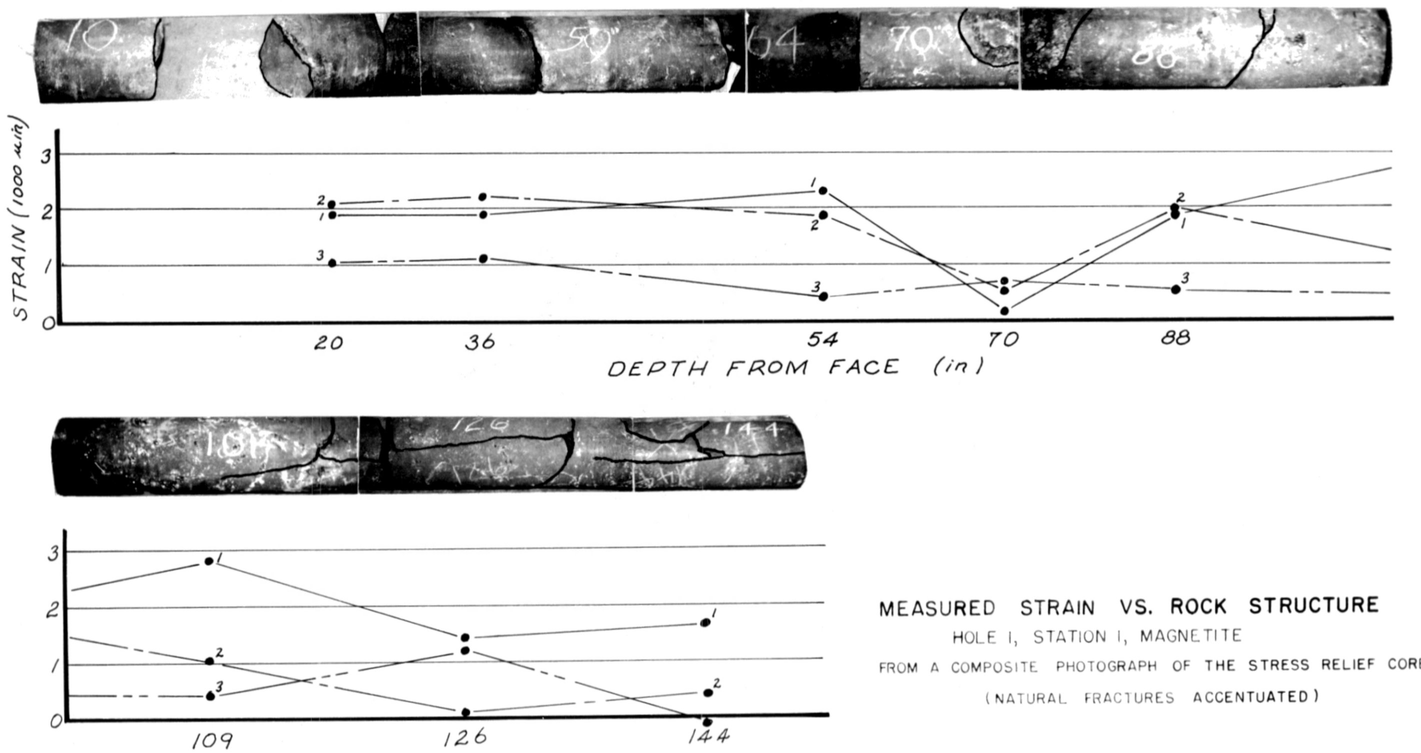


Figure 15. Measured Strain vs. Rock Structure.

fracture system which first appears at about 118 inches has a pronounced effect on the strain data recorded from that point to the end of the hole.

A high stress condition was indicated at the collar of this hole by the discing which occurred throughout the first 6 inches. The discs, shown in Figure 16, averaged approximately 0.75 inch in thickness. Their surfaces were essentially plane, and were perpendicular to the axis of the hole. Hast, 1958, attributes this discing to shearing of the core as a result of a high stress. Other than this discing there was no indication from data derived from this hole of a high stress envelope surrounding the opening.

As there were no recognized geological conditions (i.e., fractures, inclusions, and voids), and no apparent mechanical malfunctions which might cloud the validity of data from this hole, it may be assumed that the data from 0 to 102 inches are reliable, and reflect the stress condition in this zone.

Hole 2, Station 1 (Vertical downward)

The first 2 feet of rock drilled were extensively fractured; presumedly by blasting. Below these near-surface fractures there was a zone of approximately 18 inches of unbroken rock in which one set of data was obtained. A highly-developed system of fractures, dipping approximately

Figure 16. Discs from Hole No. 1.

30 degrees west, extended from 45 inches to 80 inches and prevented any measurements. All rock between these limits broke into pieces having a greatest dimension of less than 3 inches. The rock below 80 inches was as free of voids and as homogeneous as any rock encountered in the program. While only four data points were obtained in this region, there is sufficient consistency to indicate that the data is valid. Figure 17 shows the computed secondary stress magnitudes and directions.

It is of interest to note the relatively-high stress condition existing in the narrow, but competent, zone between 30 and 45 inches. Such a stress condition would be expected under the conditions indicated.

Holes 3 and 4, Station 1 (Horizontal)

These holes were parallel and 2 feet apart. Both intercepted a continuous series of fractures dipping 20 to 40 degrees to the west. These fractures rendered the data suspect and probably, by creating a column effect in the competent rock, account for the unrealistically-high stress indicated. The EX core indicated that these fractures became more prevalent with depth. In hole No. 4, the fractures were so open and extensive that all drilling water was lost. It will be noticed from Figures 18 and 19 that

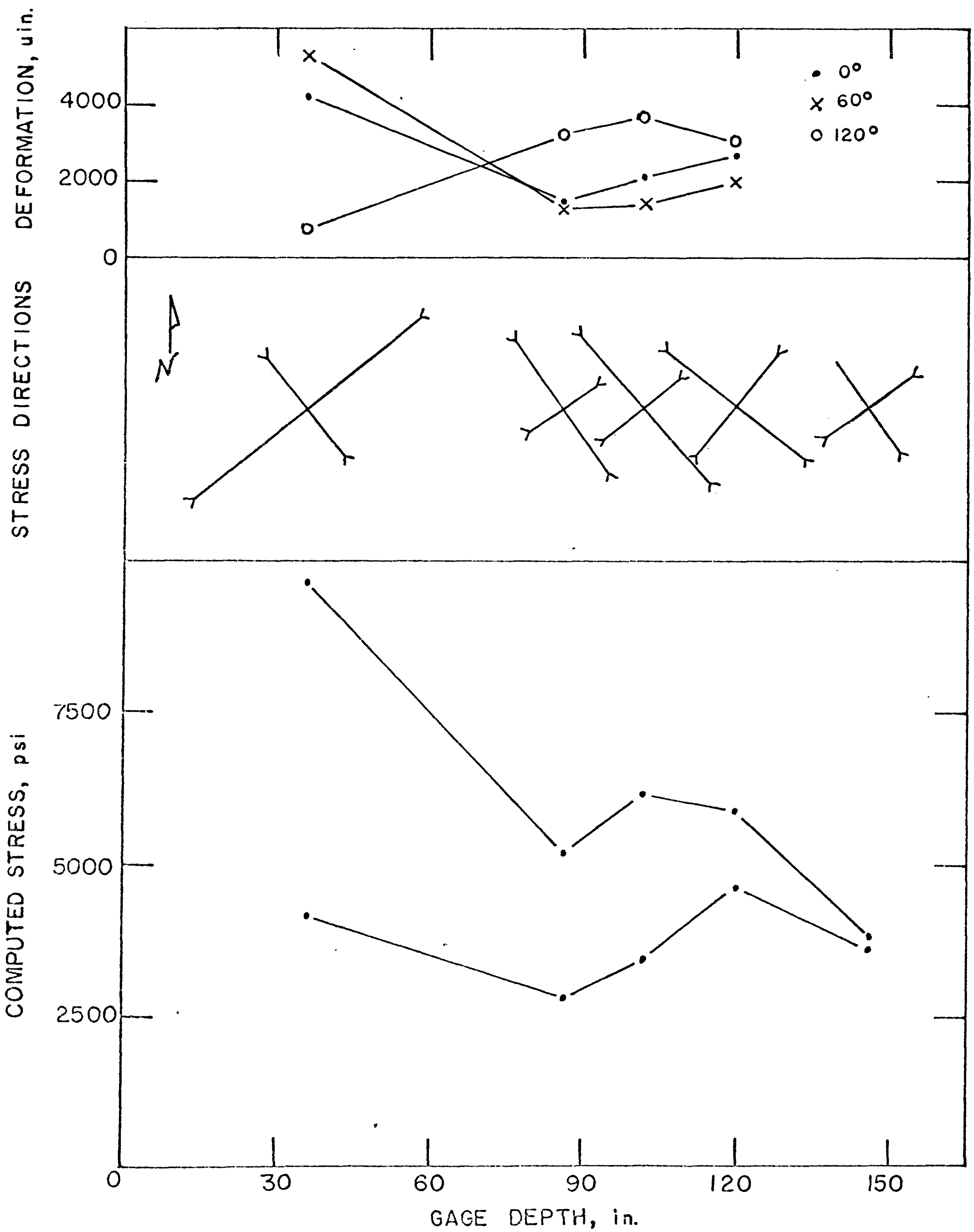


Figure 17. Stress magnitude and direction, Hole No. 2. (Vertical downward)

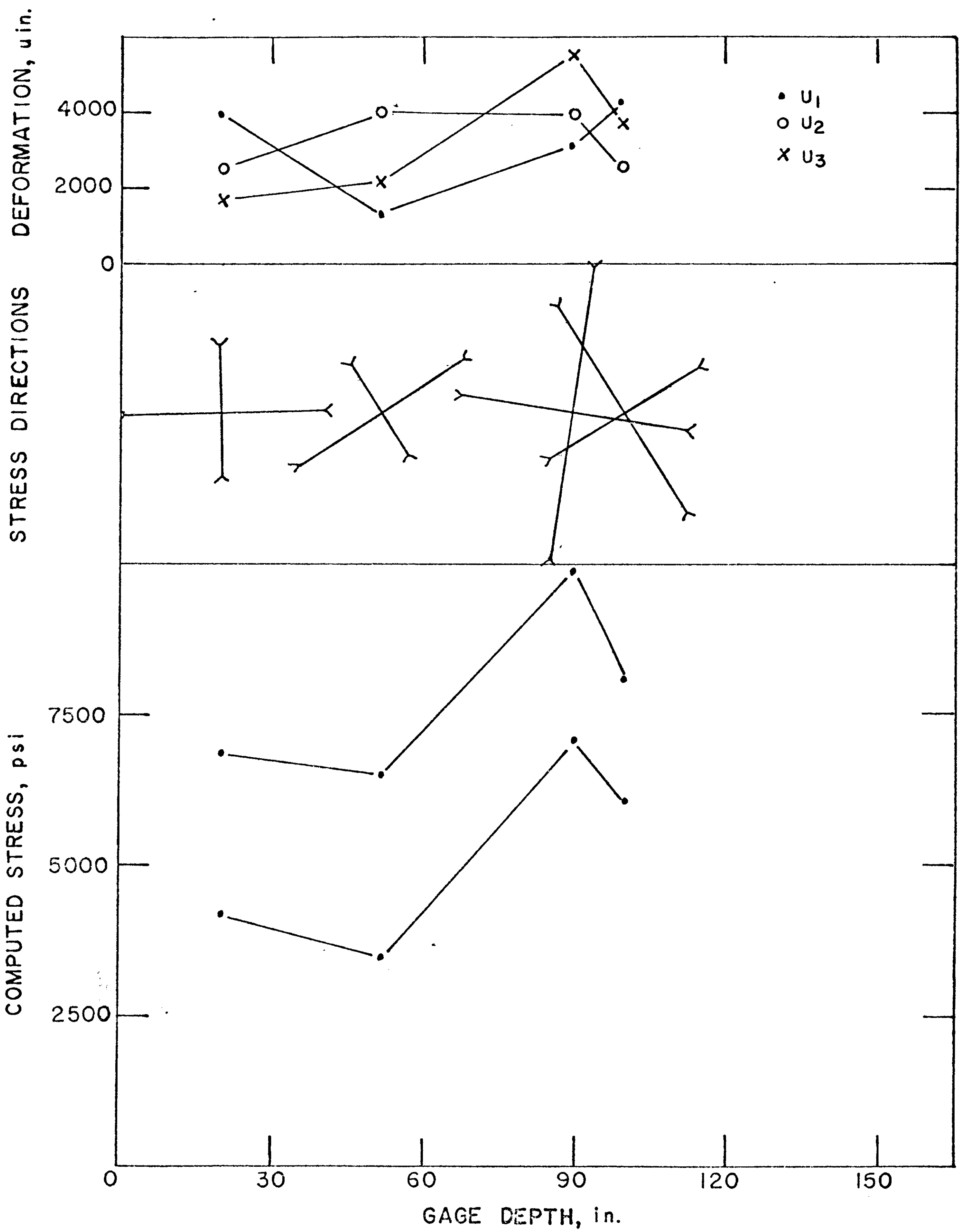


Figure 18. Stress magnitude and direction, Hole No. 3. (Horizontal)

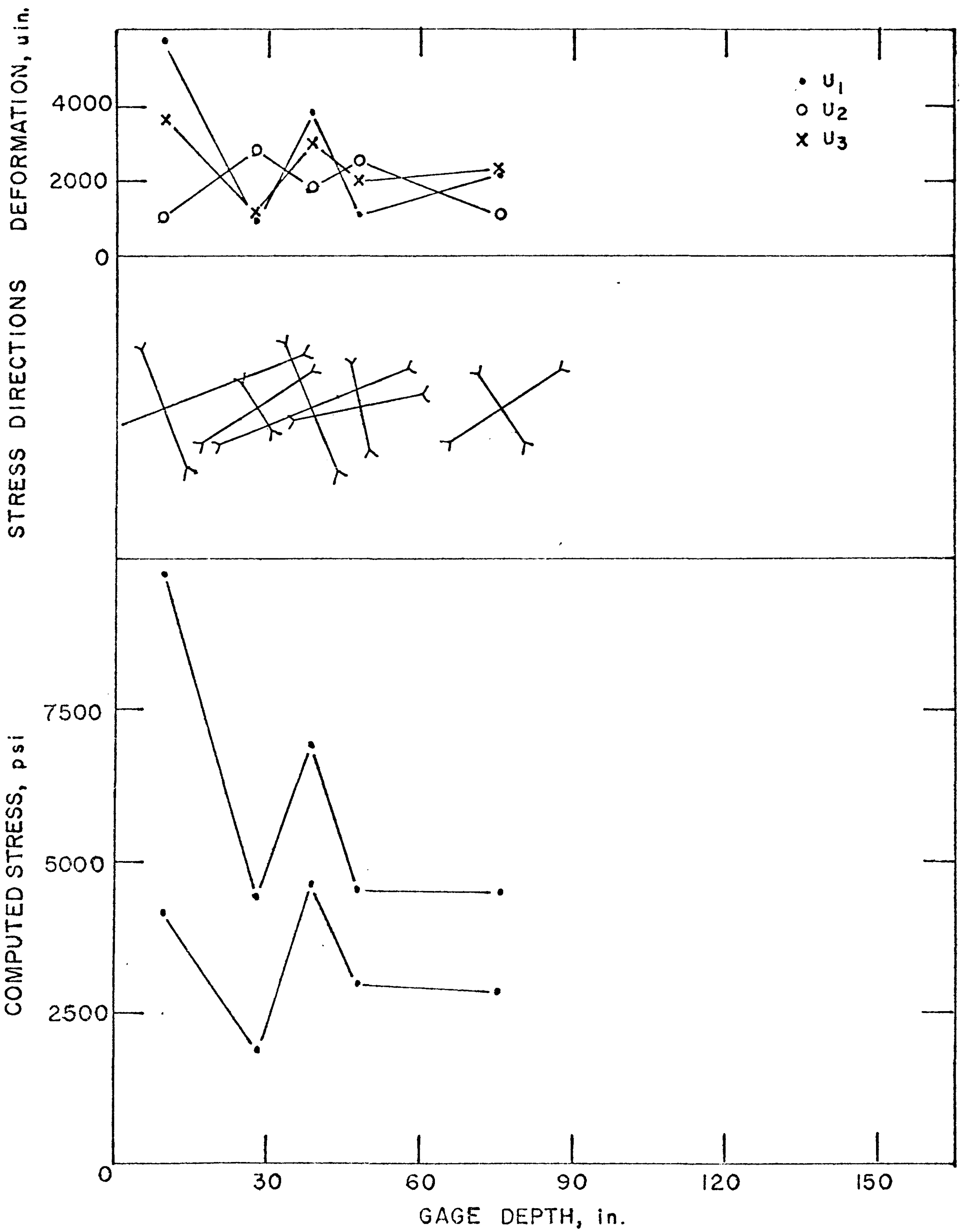


Figure 19. Stress magnitude and direction, Hole No. 4. (Horizontal)

the computed vertical stresses in these two holes are significantly greater than can be accounted for by the gravity load alone.

Hole 5, Station 2 (Horizontal

A large number of data points were obtained in this hole. Once again, however, a continuous series of fractures prevented the accumulation of a sufficient number of valid data sets. The EX core indicated competent rock throughout the first 50 inches. From 50 inches to the final depth of the hole, the rock contained numerous partings which were spaced less than 0.5 inch apart across the full diameter of the core. The principal fractures encountered had an east-west strike and dipped from 0 to 40 degrees south. It is probable that the horizontal partings account for the large recorded strain. While limited in number -- instrument failure invalidated several sets of data - those data obtained within the first 50 inches should be indicative of stress conditions at this particular location. Figure 20 shows the indicated magnitude and direction for each computed secondary principal stress.

General

The plots of measured strain shown in Figures 14 and 17 through 20 inclusive are valid except that at the last

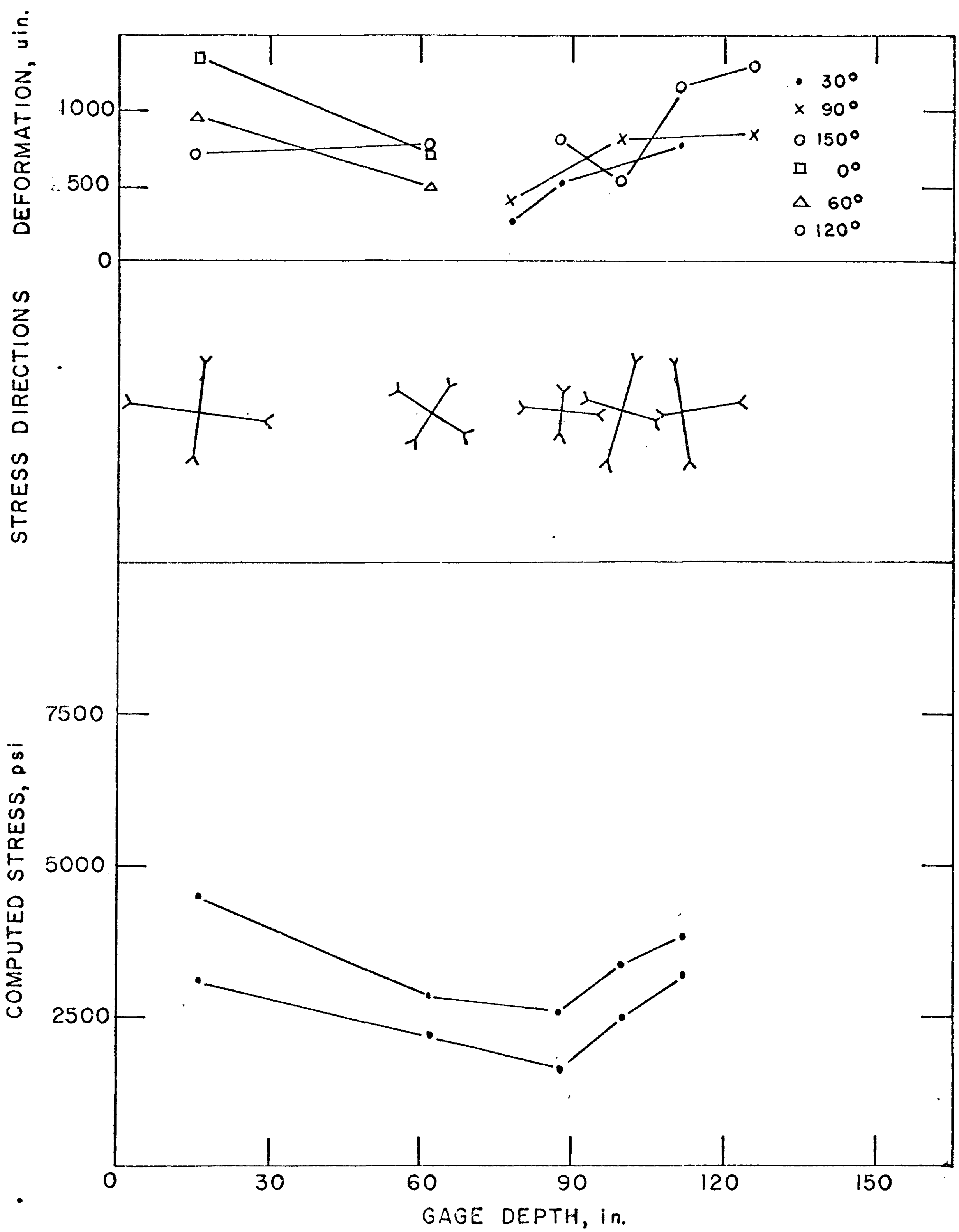


Figure 20. Stress magnitude and direction, Hole No. 5.
(Horizontal)

data point in hole No. 2 and at all points in hole Nos. 3 and 4 the gage was rotated 90 degrees alternately and therefore these strain points are misleading. At station No. 2, hole No. 5, the gage was rotated 60 degrees at alternate points and the strain plots were corrected to eliminate this rotation except at data points at 8 inches and 31 inches where the gage was rotated 90 degrees and the correction could not be applied.

Stress, strain, and principal direction, as computed for each data point, are summarized in Table II.

TABLE II

SUMMARY OF COMPUTED STRESS MAGNITUDE AND DIRECTIONS

	Depth (in.)	S (psi)	T (psi)	θ
Station No. 1				
Test Hole No. 1	20	3948	2730	-24
	36	4043	2830	-26
	54*	4143	1995	-10
	88*	3798	1982	-19
	109	3940	1735	4
	126	2687	1059	40
Test Hole No. 2	36	9665	4167	N51E
	86*	5213	2796	N35W
	102*	6187	3421	N41W
	120*	5880	4649	N51W
	146*	3835	3627	N35W
Test Hole No. 3	21	6869	4182	1
	52	6491	3398	33
	90	9903	7070	82
	100	8118	6063	122
Test Hole No. 4	10	9750	4155	21
	28*	4352	1903	33
	39	6891	4634	22
	48*	4547	2952	11
	76*	4479	2818	33
Station No. 2				
Test Hole No. 5	8	4542	3115	-7
	31*	2768	2212	-32
	39	5057	1273	
	44*	2629	1712	-5
	50	3425	2479	74
	56	3799	3196	8

*Values averaged in Table III.

YOUNG'S MODULUS DETERMINATIONS

Although seven stress-relief cores were tested to determine Young's modulus, only four tests are considered sufficiently valid to warrant consideration. The data obtained from the last complete test (i.e., the cycle on which failure occurred) on each core tested are presented in Appendix I. The stress-strain curves for the four tests that appeared to be valid are presented in Figures 21 through 24 inclusive. Based on these curves, Young's modulus was computed using the following relationship:

$$E = \frac{4\ a\ b^2 P_o}{(b^2 - a^2)U} \tag{14}$$

where:

E = modulus of elasticity of rock (psi).

a = inner radius of core (in.).

b = outer radius of core (in.).

P_o = pressure on outer surface of core (psi).

U = change in diameter of hole in core (in.).

The results obtained were:

Magnetite, dry -	8.8×10^6 psi
Magnetite, natural -	5.9×10^6 psi
Magnetite, wet -	5.7×10^6 psi
Syenite porphyry, wet -	11.2×10^6 psi

Cores were wet tested by soaking them in water for a period of five days. To maintain the natural or in-situ

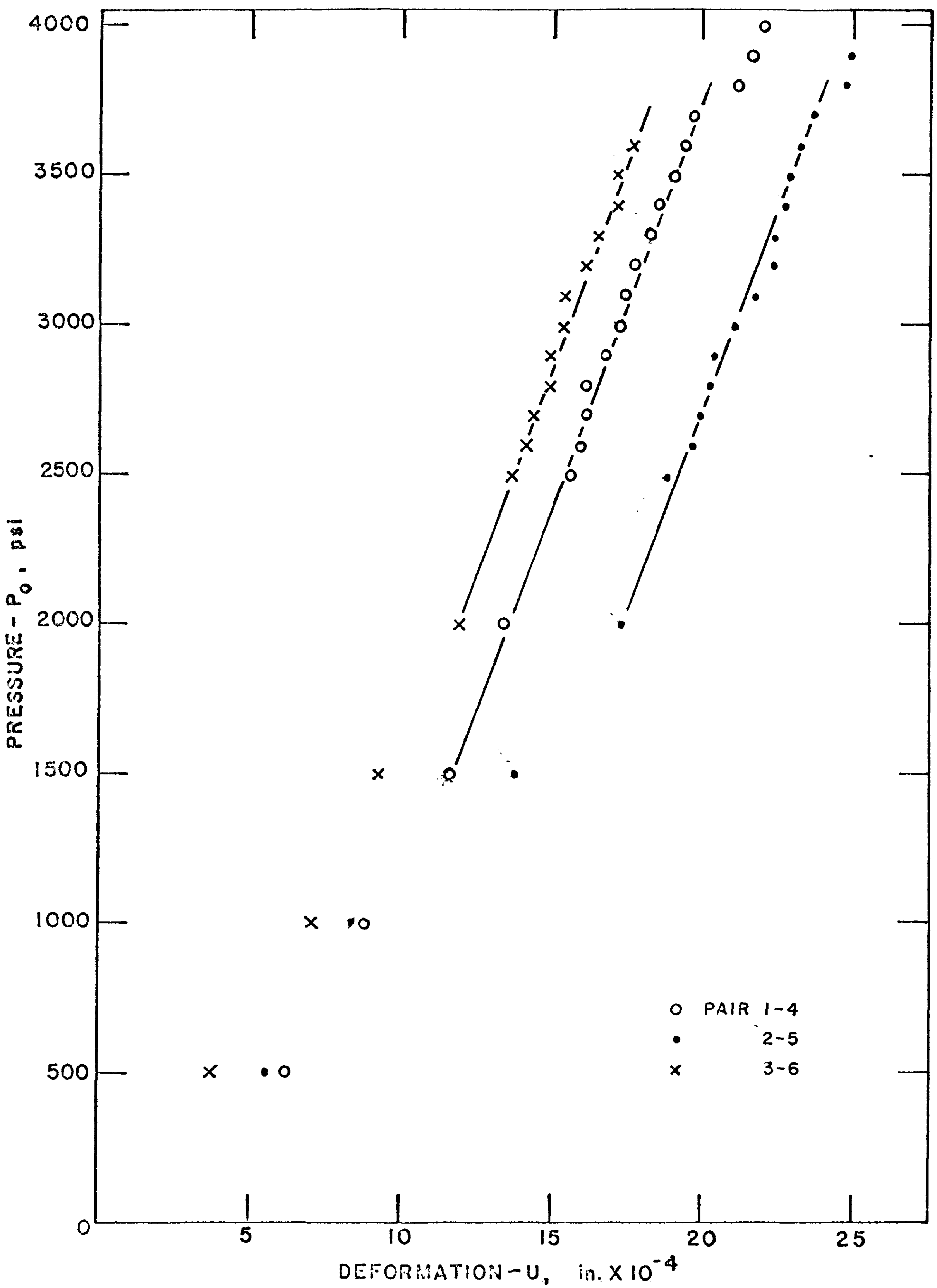


Figure 21. Young's modulus test on air-dried magnetite.

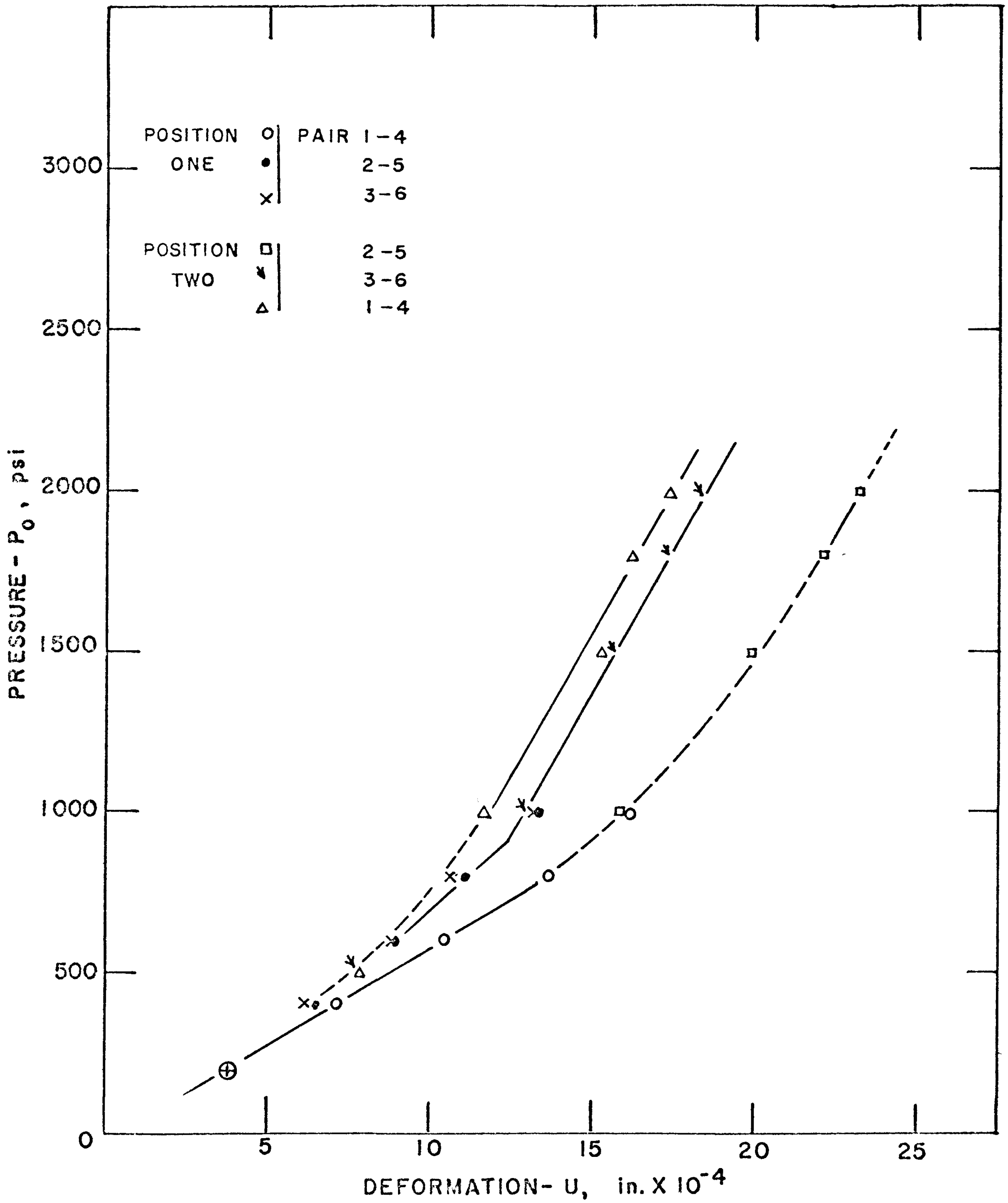


Figure 22. Young's modulus test on natural-moisture magnetite.

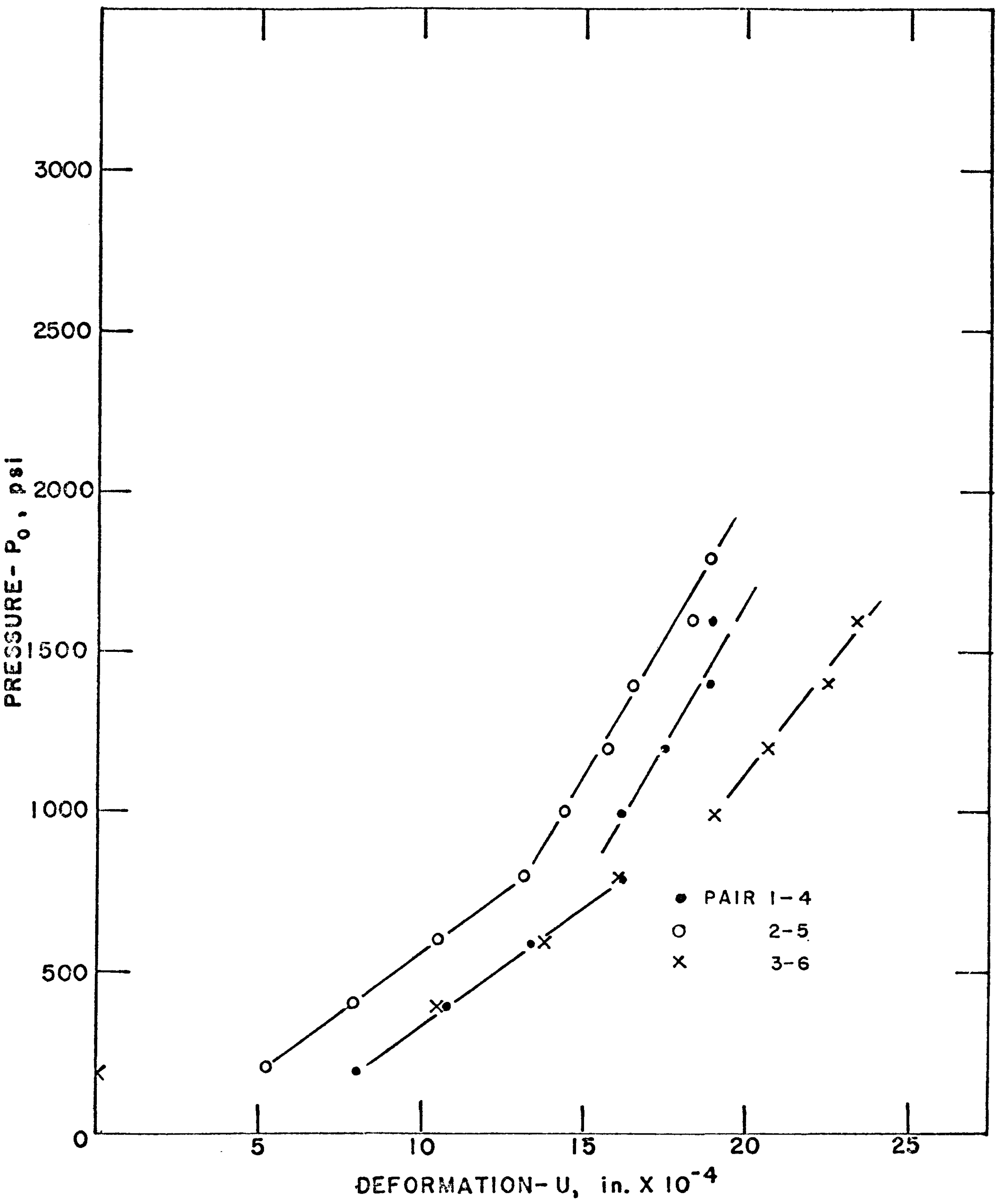


Figure 23. Young's modulus test on wet magnetite.

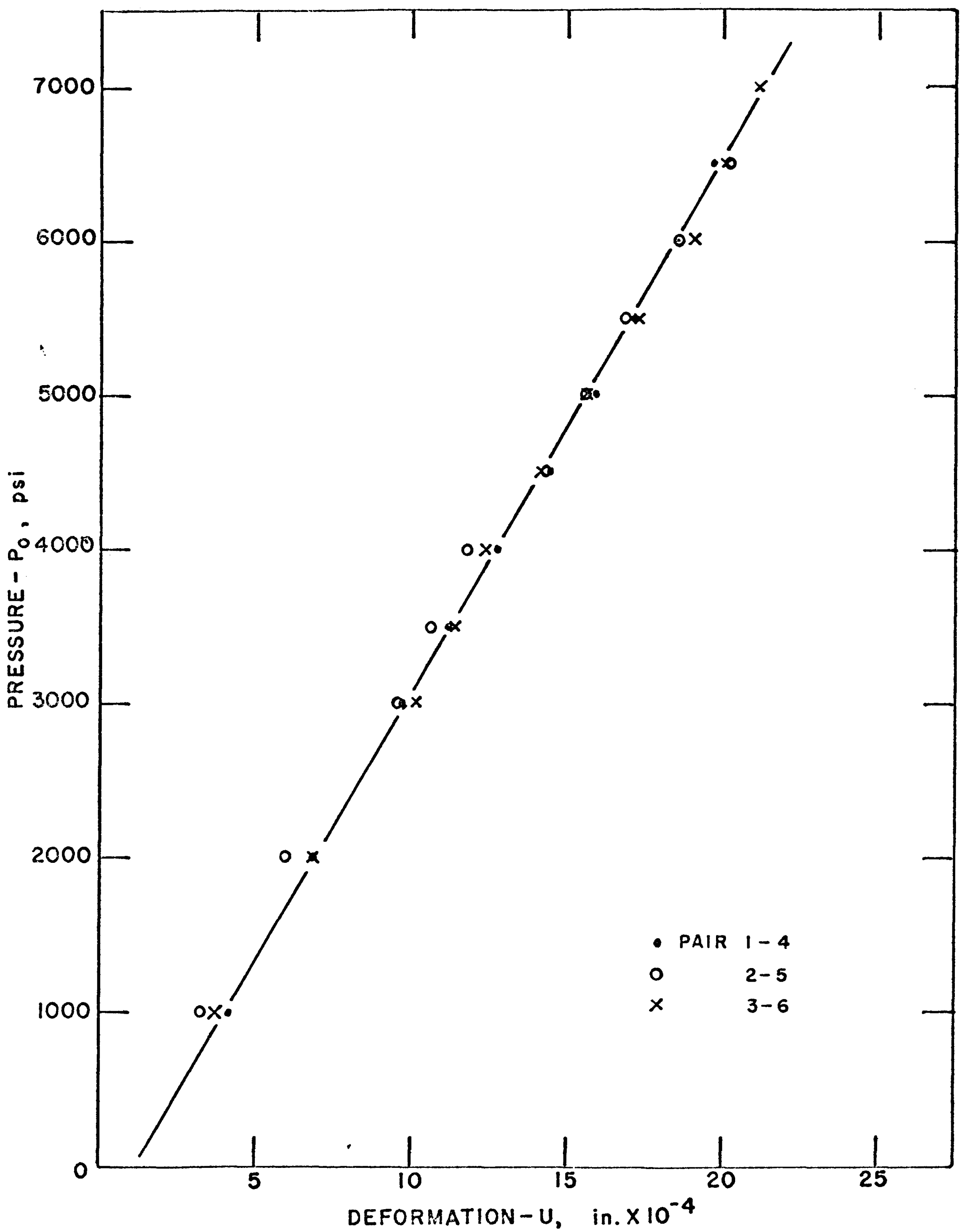


Figure 24. Young's modulus test on natural porphyry.

moisture condition, cores were sealed in plastic bags as soon as they were recovered and tested as soon as practical. It should be noted at this point that the oil pressure gage on the biaxial modulus device was not calibrated. The gage values were compared, however, with the gage values on the hydraulic pump as the core was being pressurized. The agreement was within the limit of the calibration. It must also be noted that failure of the wet magnetite core occurred at approximately the point at which the elastic curve approached a straight line. For this reason, there were not sufficient data points to establish with certainty the slope of the line and therefore the modulus of the core. The three discarded tests referred to above were not used for this same reason.

ANALYSIS OF RESULTS

The preceding sections have covered the mechanical details necessary for data collection, the data as collected, and the initial reduction of these data. The information becomes significant, however, only after such probing questions as "how good is it?" or "what does it mean?" have been answered. The following sections represent an attempt to answer these and other pertinent questions.

EVALUATION OF DATA

There are numerous conditions which may cloud the validity of the information developed through this study. Some of these, such as the mechanical properties of rock, have been noted briefly. Equipment problems, errors in experimental technique, and misinterpretation of data could also invalidate the end result.

Mechanical Properties of Rock

The general equation (6) for interpretation of bore-hole strain is valid only when applied to a material that is isotropic, homogeneous, and perfectly elastic. A condition of anisotropy would be in evidence if there were differences in the computed elastic constants in each of several directions. This, in turn, would lead to erroneous computed stress magnitudes and directions. The probability of a significant error of this type in the reported work is small. This is indicated by the results of the Young's modulus tests on the several cores. During these tests the strain was measured in three directions simultaneously. The greatest difference in strain in the three directions was 6×10^{-4} inch or about 15 percent at a stress level of 5000 psi. More important, the slope of the three stress-strain curves was identical at stress levels above 2000 psi. This means that within the elastic range of the rock the stress-strain response was identical in all directions measured by the gage. It does not follow that the rock was isotropic -- only that uniformity of the elastic modulus was evident at stress levels above 2000 psi which would be well within the range of measured stresses.

It was apparent that the magnetite was not homogeneous. Seams and isolated masses of hematite were found within the predominant magnetite. It is probable that the elastic constants of these two materials were not identical and may have introduced an error in the computed stress. The close agreement between successive strain measurements along a single axis, however, indicated that such an error must have been small if it did in fact exist. There were no instances when hematite inclusions constituted more than an estimated 20 percent of a core. Figure 25 shows a magnetite core with a more-than-normal number of these inclusions.

There was no visible evidence to indicate that the coarse-grained syenite porphyry was not homogeneous -- recognizing that within the microscopic or grain-size range the material is completely non-homogeneous.

Frequent open or filled vugs of calcite and other accessory minerals in the magnetite represented local non-homogeneities which effected the data. When such a void was visible and near the strain-gage points, the indicated strain values were erratic. Such a condition is described on page 55. When a void could be related to an erratic strain value the condition was noted and the data discounted. It is probable that there were instances of such openings either hidden in the core or just outside the 6-inch bore-hole. In these cases there would be no explanation for the inconsistent data which probably resulted.

Figure 25. Magnetite core showing inclusions of Hematite outlined in white.

By definition, a perfectly-elastic material exhibits a linear stress-strain relationship. It is apparent that none of the cores tested were perfectly elastic over the full range of interest. The porphyry core approached the perfect state very closely. The dry magnetite core was elastic within the stress range of 2000 psi to failure (approximately 3700 psi). The natural moisture and wet magnetite cores failed at approximately 2000 psi. However, it appeared that they might have been entering a linear elastic range. All magnetite cores were highly inelastic in the 0 - 2000 psi range. The effect of this condition is discussed in a following section.

In addition to the more obvious problems noted above, there is evidence to indicate that the rock core may be structurally damaged during the stress relief procedure. Obert (1962) has found microscopic cracks in the relieved core material that were not present in the same rock in place. Moreover, he found that the stress-strain curves of damaged core material from high-stress areas were concave upward whereas core from low-stress areas was undamaged and exhibited curves that were concave downward. The work reported here has shown the Pea Ridge Mine to be a high stress area. The stress-strain curves shown in the preceding section are concave upward indicating that the condition suggested by Obert may have effected the computed stress values.

Equipment Problems

There were, during the course of the test, complete equipment failures. These were usually the result of electrical shorts caused by water leaks and in these cases no data were collected. The instrument problems to be concerned with are those that were unnoticed. Temperature variations, strain-gage creep, and stray magnetic fields each could have had a small effect on the results.

The borehole gage was allowed to stand in the borehole several minutes before each test. During this period it should have approached temperature equilibrium with the rock. The introduction of the drilling water changed the temperature and some resistance drift was noted. The water was allowed to run until this drift stopped and only then was the test begun. It is nevertheless possible that some additional drift occurred due to temperature change.

Even though the strain gages were temperature compensated, the outside caliper of the gage would have been dependent on the length of the cantilever probes and the diameter of the borehole, both of which would have been temperature sensitive.

Gage drift on the order of 50 - 200 micro strains was noticed when the gage was left in place for a period of one hour or longer. This drift might be attributed to strain gage creep, to creep of the polyform covering, or to changes of rock temperature as noted above. If it were linear with respect to time, the drift would not have been significant during the 5 - 10 minute period required for a test.

Four of the five test sites were located entirely within the magnetite orebody. It is possible that there was electrical interference in either the signal cable or the wheatstone bridge due to the magnetic characteristics of the magnetite. There was no evidence to indicate that this was or was not a problem.

INTERPRETATION OF DATA

Of all the errors that may bear on the data, those involved in the determination of the elastic modulus of the rock are of greatest concern. A given percentage of error in the established value of the constant is reflected in an equal error of the computed stress value.

The deformation value (U) used in the modulus equation was established from the slope of the straight portion of the deformation curves (Figures 23 - 26). As an alternative, the deformation could have been taken directly from the data at a given pressure point. For example, the slope of the deformation curve for the natural moisture magnetite core in the range 1000 to 2000 psi was 5.5×10^{-4} inch/1000 psi. Using this value, the computed modulus of elasticity was determined to be 5.9×10^{6} psi.

If, instead of using the above noted procedure, the middle deformation curve had been used and the total deformation at 4000 psi had been used for the calculation, the indicated modulus would be 4.4×10^{6} psi or 25 percent less. This would in turn reduce all computed stresses by 25 percent, or from an approximate level of 2000 psi in the vertical direction to approximately 1500 psi.

The deformation curve for the poryphyry core was approximately linear over the range tested and crossed the deformation axis approximately at 0. The type of error discussed above would not have been a consideration in this case.

DATA SUMMARY

Certain of the computed stress values (Table II) are inconsistent with other values obtained at the same location. The consistent data have been averaged to arrive at the summary values presented in Table III.

TABLE III

AVERAGE STRESS MAGNITUDES AND DIRECTIONS

	S (psi)	T (psi)	θ°
Station 1			
Test Hole No. 1	3970	1988	18°
Test Hole No. 2	5279	3598	N40W
Test Hole No. 4	4459	2558	25°
Station 2			
Test Hole No. 1	2698	1962	18°

The relative positions of the average computed stresses are shown in Figure 26. The magnitude of the vertical stress components at each of the test points have been computed using the relationship:

$$S_v = \frac{(S + T)}{2} + \frac{(S + T)}{2} \text{Cos } 2\ (90 - \theta) \qquad (15)$$

The indicated components are:

Test Hole No.	1-1	2176 psi
	1-4	2898 psi
	2-1	2032 psi

The overlying material at station 1 consists of approximately 1300 feet of Ordovician and Cambrian sediments and approximately 600 feet of magnetite. If densities of 2.7 and 5.1 respectively are assumed, the gravity load would be approximately 2900 psi. The computed vertical components are 10 - 30 percent lower than this figure. At station 2, if an overall density of 2.5 and a total depth of 1900 feet

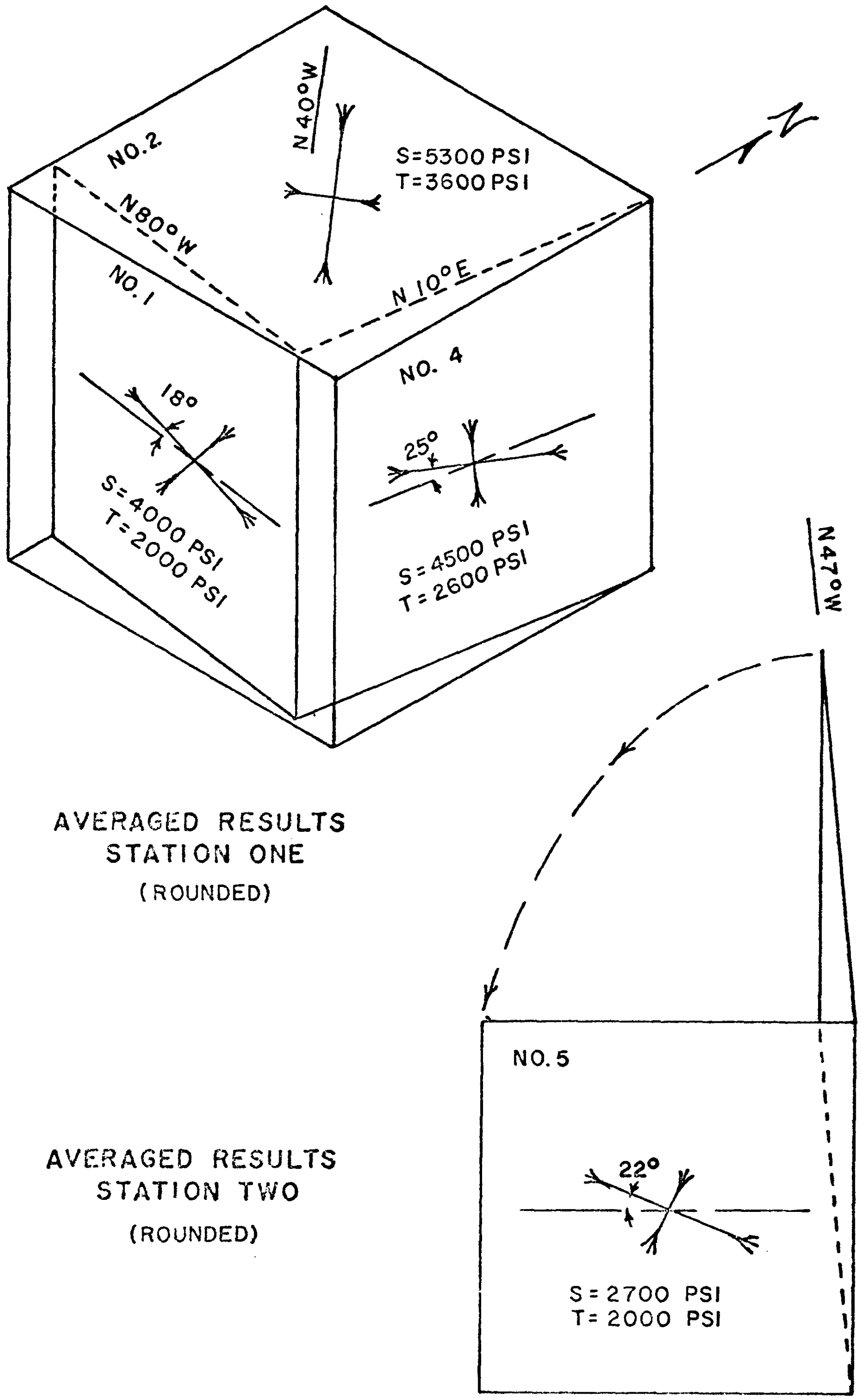


Figure 26. Relative positions of averaged computed results.

is assumed, the gravity load would be 2100 psi as compared to a computed vertical component of 2032 psi.

In the horizontal plane, the stress components computed from the vertical hole data in the directions of the planes of tests 1 and 4 are 4584 and 4292 psi respectively. The horizontal components computed from the horizontal hole data are 3781 and 4119 respectively. The differences of 19 and 4 percent appear reasonable for this comparison.

RELATIONSHIP TO MINE

The results of test hole 1-2 indicate that the principal stress direction is oriented approximately 80 degrees from the average strike of the orebody. Both tests 1-1 and 2-1 show secondary principal stress directions inclined to the NW. By chance, the plane of data from hole 2-1 was oriented only 7 degrees from the principal direction as computed from test 1-2. Therefore, if it is assumed that the same general stress field exists at station 2, the indicated principal stress direction is inclined 22 degrees above the horizontal to the west. The computed secondary principal stress at station 2 is very much less than that indicated at station 1. If, however, the strong horizontal stress is attributed to the intrusive magnetite, this condition would exist since station 2 was located a considerable distance from the orebody and beyond its longitudinal extent.

The computed angle of inclination of the secondary principal stress at test hole No. 4 is inconsistent with the above interpretation. However, since this was the most unreliable data, this contrary condition should be discounted.

CONCLUSIONS

The procedures and results of an engineering investigation must be viewed with an eye toward their ultimate purpose. The investigation reported herein must be viewed from two directions: from that of the mine operator looking for information useful for mine design and planning; and from that of the investigator concerned with technique, reliability, and practical application. The two viewpoints should be complementary.

Results of this investigation indicated that a near-horizontal principal stress existed in the mine area and that the magnitude of this stress was in excess of 4000 psi, or nearly five times as great as would be estimated considering gravity loading only. It was further indicated that the direction of this principal stress was approximately normal to the strike of the orebody. It would appear that these findings, even though somewhat qualitative, should be of practical value to the mine design engineer, because the design of large underground openings is dependent on a knowledge of the existing stress field if stability and functional suitability are to be maintained.

The results further indicated that the basic technique and equipment were adequate for in-situ testing although numerous modifications to both, as previously described, might greatly facilitate the work. It became apparent that neither the system used nor any other presently-available system could yield acceptable data where rock conditions were poor -- that is, highly fractured and fissured.

Although the accuracy of the data can not be established, there were a sufficient number of recognized potential errors to discredit most discrete values. It is believed that equipment errors could be eliminated with persistence, whereas errors attributable to inhomogeneities of the rock could be eliminated only by statistical treatment of a large number of tests.

APPENDIX I

GAGE DATA FROM IN-SITU TESTS

Gage Depth (in.)	Orientation of 1-4	Indicator Readout (u-in./in.)			Computed Deformation (u-in.)		
		1-4	2-5	3-6	1-4	2-5	3-6
Station 1 Borehole No. 1							
20"	θ	82	-	48	1841	2060	1064
36"	θ	82	103	40	1841	2142	1126
54"	θ	104	88	16	2335	1830	450
88"	θ	82	92	19	1841	1914	535
109"	θ	124	50	14	2784	1040	394
126"	θ	62	5	45	1392	104	1267
144"	θ	73	19	-4	1639	395	-113
Borehole No. 2							
36"	θ	188	37	188	4221	770	5292
86"	θ	64	154	44	1437	3203	1239
102"	θ	94	177	48	2110	3682	1351
120"	θ	122	147	72	2739	3058	2027
146"	ϕ	84	84	68	1886	1747	1914
Borehole No. 3							
21"	θ	153	110	57	4009	2519	1687
52"	ϕ	50	174	72	1310	3985	2131
90"	θ	120	172	187	3144	3939	5535
100"	ϕ	165	110	125	4323	2519	3700
Borehole No. 4							
10"	θ	254	46	165	5690	1017	3630
28"	ϕ	40	125	45	896	2762	990
39"	θ	168	83	135	3763	1834	2970
48"	ϕ	50	112	90	1120	2475	1980
76"	θ	100	45	100	2240	984	2200

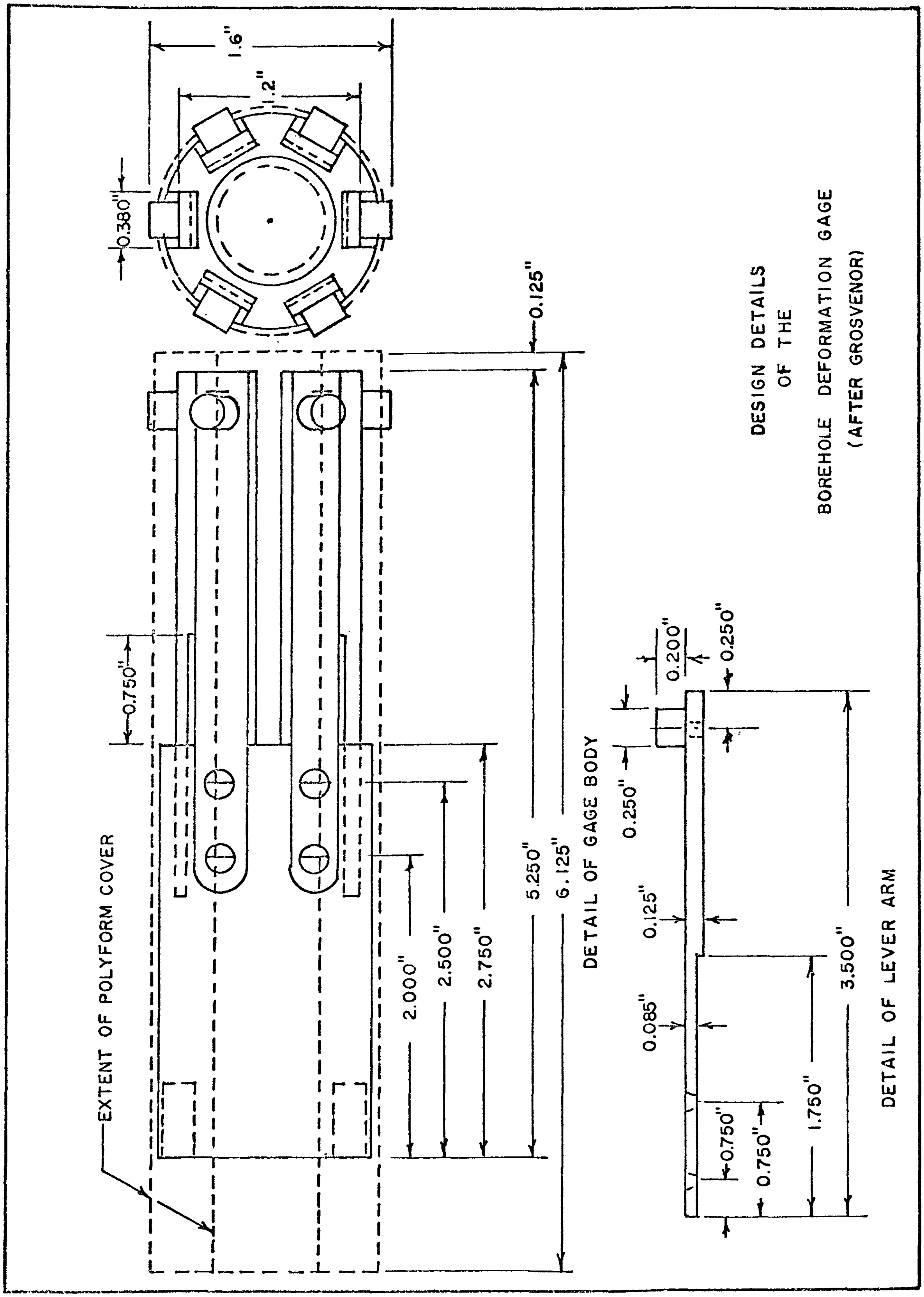
Gage Depth (in.)	Orientation of 1-4	Indicator Readout (u-in./in.)			Computed Deformation (u-in.)		
		1-4	2-5	3-6	1-4	2-5	3-6
Station 2 Borehole No. 5							
8"	θ	60	43	32	1344	950	704
31"	θ	31	23	34	694	508	748
39"	ϕ	18		12	403	(600)	264
44"	θ	23	17	50	515	(390)	795
50"	θ	24	49	37	538	(960)	814
56"	θ	35	40	48	784	(900)	1056
63"	θ	58	47	38	1299	(1120)	836
68"	θ	77		89			
82"	ϕ	75		99			
90"	θ	87		58			
99"	θ	82					
105"	ϕ	97		53			

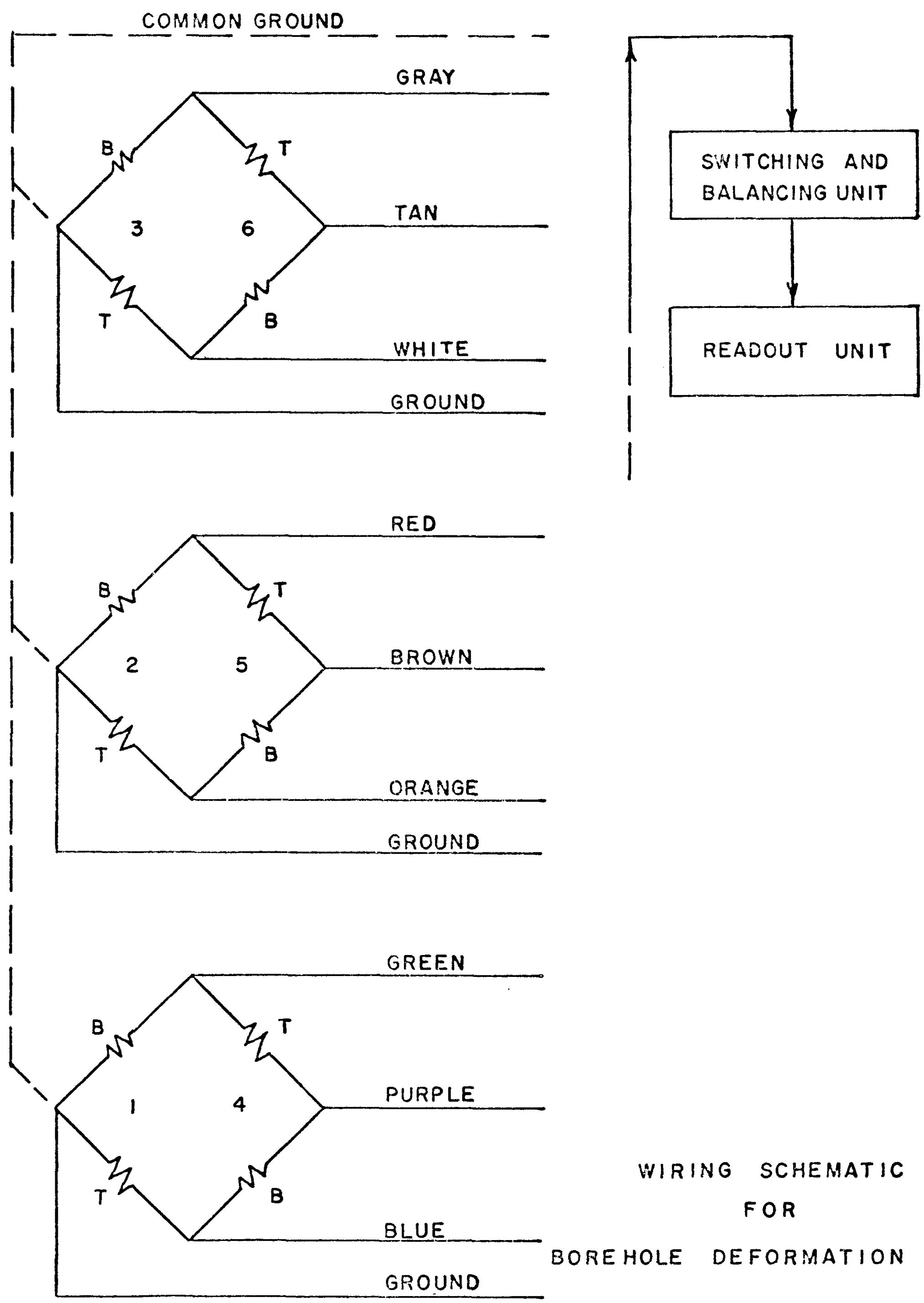
YOUNG'S MODULUS DETERMINATION

LABORATORY DATA

Pressure (psi)	Net Indicator Readout (u-in./in.)			Computed Deformation (u-in.)		
	1-4	2-5	3-6	1-4	2-5	3-6
Dry Magnetite						
500	28	25	17	627	552	374
1000	40	50	32	896	840	704
1500	52	62	42	1165	1370	924
2000	60	78	54	1344	1724	1188
2500	70	85	62	1568	1878	1364
3000	77	95	70	1725	2100	1540
3500	84	103	78	1882	2276	1716
4000	98	113	-	2195	2497	-
Natural Magnetite						
500	35	45	35	784	994	770
1000	52	72	58	1165	1591	1276
1500	68	90	71	1523	1989	1562
1800	72	100	79	1613	2210	1738
2000	78	105	83	1747	2320	1826
Wet Magnetite						
200	20	0	30	525	0	806
400	30	50	40	787	1145	1077
600	40	60	50	1050	1372	1345
800	50	70	60	1310	1600	1615
1000	55	83	60	1440	1900	1615
1200	60	90	65	1570	2060	1748
1400	63	98	70	1650	2240	1882
1600	70	102	70	1830	2330	1882
1800	72	-	-	1890	-	-
Porphyry						
1000	14	18	11	366	412	326
2000	26	30	20	682	688	592
3000	39	42	32	1022	963	947
3500	43	49	36	1128	1122	1064
4000	47	56	40	1230	1282	1183
4500	54	63	48	1413	1443	1420
5000	60	70	53	1572	1605	1568
5500	66	75	57	1730	1719	1686
6000	73	81	63	1912	1857	1863
6500	77	86	69	2015	1970	2040
7000	81	-	-	2120	-	-

APPENDIX II


DESIGN DETAILS
OF THE
BOREHOLE DEFORMATION GAGE
(AFTER GROSVENOR)
DETAIL OF GAGE BODY
DETAIL OF LEVER ARM
EXTENT OF POLYFORM COVER
1.6"
1.2"
0.380"
0.125"
0.750"
2.000"
2.500"
2.750"
5.250"
6.125"
0.200"
0.250"
0.250"
0.125"
0.085"
0.750"
0.750"
1.750"
3.500"



WIRING SCHEMATIC
FOR
BOREHOLE DEFORMATION GAGE

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