G. Zahary

CANADA

DEPARTMENT OF ENERGY, MINES AND RESOURCES

MINES BRANCH

OTTAWA

STRESS MEASUREMENTS AT ELLIOT LAKE

D. F. COATES AND F. GRANT

Reprinted from The Canadian Mining and Metallurgical Bulletin, Transactions, Vol. LXIX. pp. 182-192, 1966

A COMPARISON OF TWO METHODS FOR MEASURING STRESS IN ROCK

W. L. VAN HEERDEN AND F. GRANT

Reprinted from The International Journal of Rock Mechanics and Mining Sciences, Vol. 4, pp. 376-382, 1967

Reprint Series RS 68

Price 25 cents

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ROGER DUHAMEL, F.R.S.C.
Queen's Printer and Controller of Stationery
Ottawa, Canada
1968

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Stress Measurements at Elliot Lake†

Annual General Meeting, Quebec City, April, 1966

Transactions, Vol. LXIX, 1966, pp. 182-192

SUMMARY

As the problems concerning rock mechanics in Canada occur mainly in the mineral industry, the Federal Government recently established a field research group at Elliot Lake, Ontario. One of the first projects with which this group has been occupied is that of measuring the in situ stresses in and around mines.

The most intensive work has been done in one of the local uranium mines located on the south limb of the west-plunging syncline that contains the orebody. The average dip of the formation here is 14 degrees. Mining is being conducted at depths of from 600 ft. to 1,400 ft. below surface.

Stresses of up to 15,600 psi, measured in the pillars, are higher than expected. Even higher stresses may exist, because, in some of the stress-measuring holes, the cores disced and measurements could not be taken. These stresses, however, do not cause working of the pillars.

The field stress results were also surprising in that, at the location of the measurements, the vertical stress was found to be about twice the gravitational stress, with the horizontal stresses being about three times the gravitational stress. Clearly, the rocks still retain stresses from the folding, faulting and intrusive action that has occurred in geological time.

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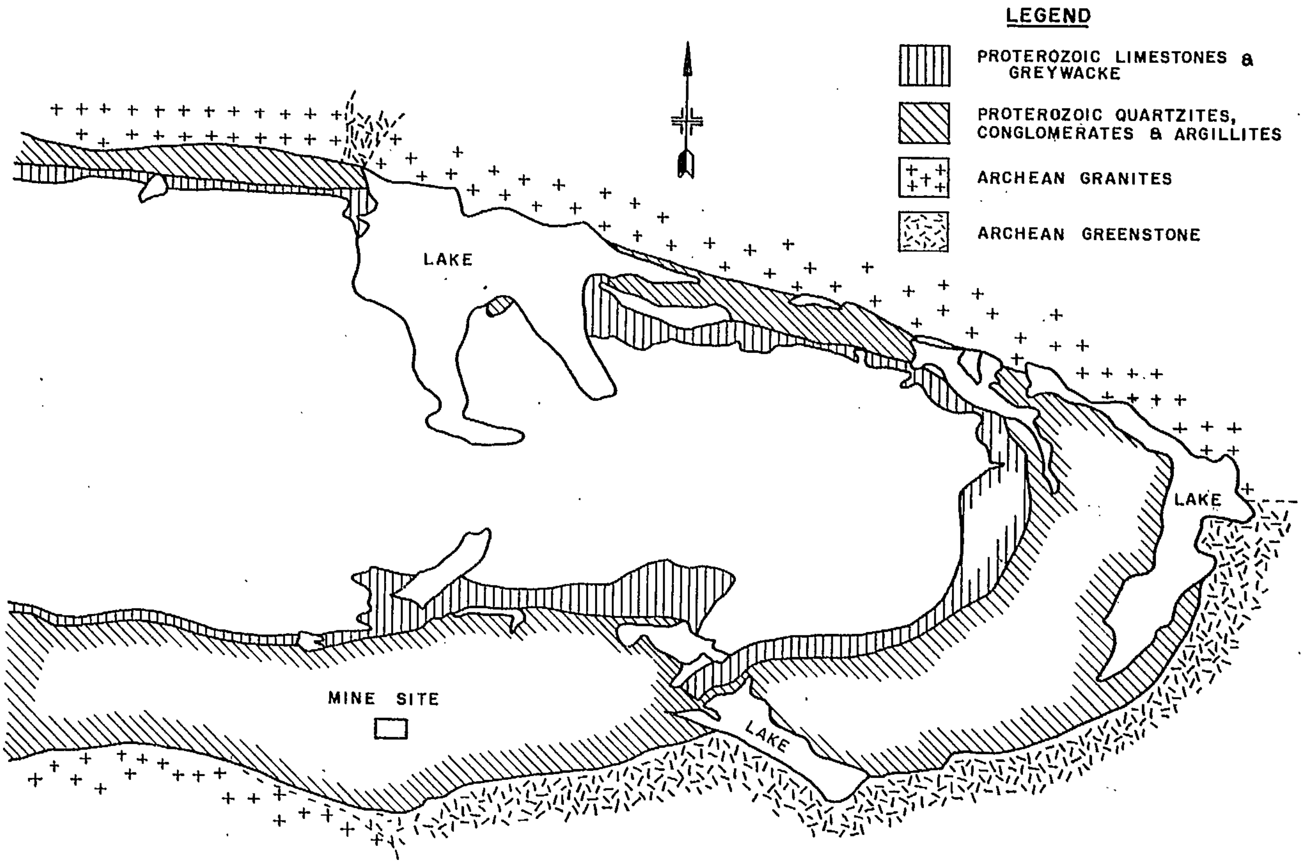

Figure 1.—Regional geology of Elliot Lake, showing the trace of the syncline plunging to the west, with the mine located on the south limb dipping to the north in the Proterozoic quartzite (after Ref. 3).

Introduction

Rock Mechanics in Canada

MINING consists essentially of breaking rock and maintaining the surrounding rock in a stable condition. Hence, the basic science of mining is rock mechanics, and mining research is substantially rock research.

The mineral industry in Canada is currently excavating about 350 million tons of rock per year while the construction industry excavates about 40 million tons per year; consequently, the group that is likely to have the most problems in rock mechanics is the mining industry.

In the light of these facts, a little over a year ago the Federal Government decided to expand applied research in this subject by establishing a new field research group at Elliot Lake. This group is now conducting studies both in the immediate area and in mines across the country.

Ground Control at Elliot Lake

Figure 1 is a geological plan of the Elliot Lake area showing a trace of the broad syncline in the Precambrian Proterozoic formation that contains the uranium ore of the Elliot Lake mines (1). Folding of the syncline caused the upper layers to ride over the lower, producing observable striae. Various families of faults have also contributed to a relatively active stress history, which seems to have left a legacy of field stresses much greater than could be attributed to gravitational effects alone.

Some common aspects of ground control problems at Elliot Lake can be seen in the following figures. *Figure 2* shows the breaking in the hanging wall to a former beach where the bonding, or cohesion, between the layers is obviously low. *Figure 3* shows the roof in a drift at a depth of 3,500 ft. below the ground surface, where a fault zone has produced local weakening and encouraged working of the ádjacent ground.

Figure 4 shows a stope at a depth of 700 ft. below the ground surface, where the stress concentration at the upper corner of the face seems to be great enough to produce local crushing. *Figure 5* also shows the crushing and shear effect of the stress concentration at the upper corner of a face in a stope at a depth of 3,000 ft. *Figure 6* shows a pillar at a depth of 2,600 ft., where slabbing has occurred due to the shearing induced by the stress relief in the surface ground resulting from crushing at the upper corner. This action is thought to be the same as the discing which occurred in some of the core and is described below.

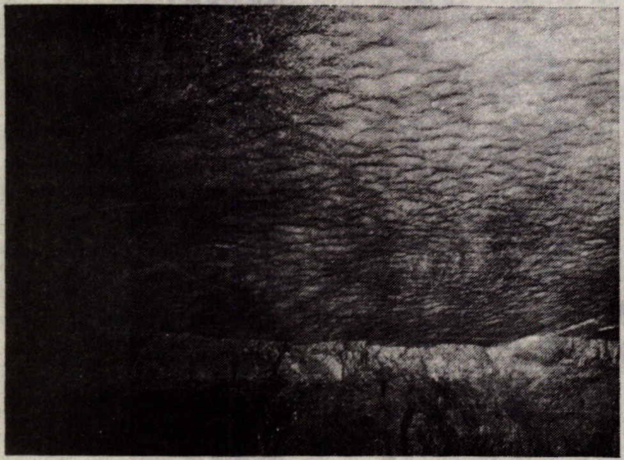

Figure 2.—Breaking in the roof of a stope to a former beach where inter-layer bonding is weak (courtesy of Professor H. R. Rice).

Figure 3.—The working and failure of rock adjacent to a fault zone in the roof of a drift in one of the mines at a depth of 3,500 ft. (courtesy of Professor H. R. Rice).

Figure 4.—Crushing occurring at the sharp, upper corner of the face, where the stress concentration is a maximum, in a stope at a depth of 700 ft. (courtesy of Professor H. R. Rice).

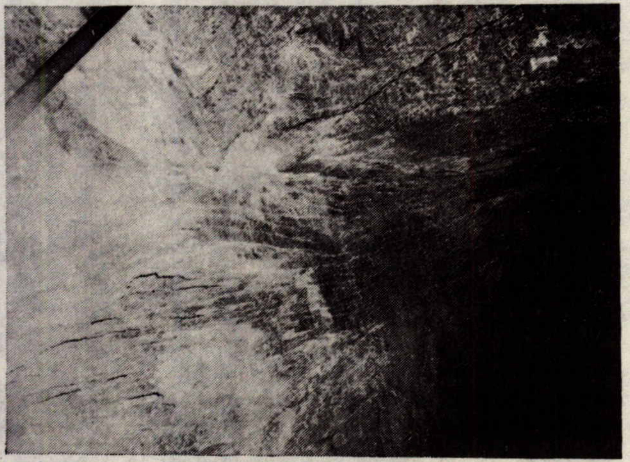

Figure 5.—Extensive crushing at the upper corner of a face, where the stress concentration is a maximum, in a stope at a depth of 3,000 ft.; note the trace of the shear planes which penetrate into the rock (courtesy of Professor H. R. Rice).

The shearing mechanism of slabbing that is postulated here is somewhat controversial. Our concept is that the mechanism is similar to that which would occur if you had two cubes of rubber cemented together on a vertical surface; if these two cubes were compressed by pushing down with a hand on each and then one hand is taken away, the cube that was relieved would tend to shear away from the one that was still under compression, and this mechanism would be reinforced if the second one received the weight that was formerly being taken by the first one.

Elliot Lake Research

In an attempt to obtain new information that will remove some of the speculation about the causes of these various ground control problems, the Mines Branch, as soon as the Elliot Lake group was established early in 1965, started an intensive program of stress measurements underground. Stresses were measured first in nine pillars in three typical stopes. Then, measurements were taken in a cross-cut out to a projected internal shaft as far away from the mining zone as possible to obtain an indication of the field stress conditions. Finally, measurements were taken in one of the major vertical dikes intersecting the mine adjacent to the test stopes.

Geology

Regional Geology

The measurements described below were obtained in the Pre-cambrian Proterozoic formation. The Proterozoic rocks at this location were deposited on a major unconformity which had been eroded flat, although shallow troughs had formed in a northwest-southeast direction in some softer volcanics within the greenstone. The formation containing the ore of the mines in this area is a river deposit forming a monolithic quartzite some 700 ft. thick. The actual orebody is generally 6 to 12 ft. thick and lies about 30 ft. above the Archean erosion surface (1, 2).

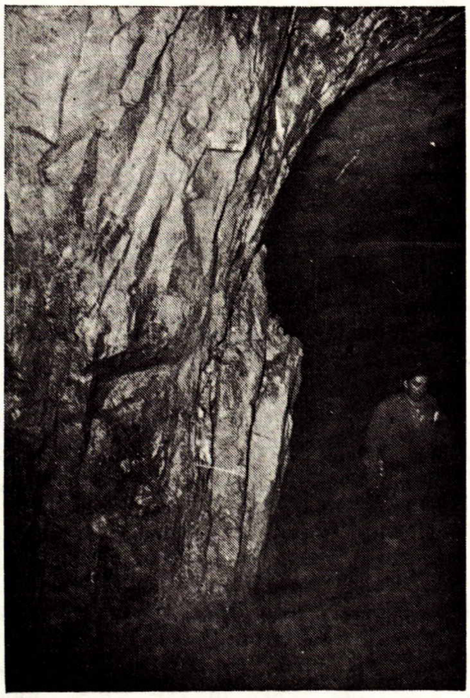

Figure 6.—Splitting or slabbing on the sides of a pillar, possibly due to the shear stress resulting from the decrease in stress in the slabs following removal of the rock above and aggravated by the increase in stress resulting from the slabbing on the interior rock; in a drift at a depth of about 2,600 ft. (courtesy of Professor H. R. Rice).

As a rule, the Proterozoic rocks here are only gently folded and slightly metamorphised. These sediments record three renewals of sedimentation after deformation or up-lift. Locally, the formation has been folded into a broad syncline, as shown in *Figure 1,* with the east-west axis plunging westerly at about 5 degrees (3). The mine in which the stress measurements were made is on the south rim of the syncline and is being worked from the 600-ft. to the 1,400-ft. level.

Faults occur regionally, with strikes in the northwest direction. In addition, there is a main east-west fault. Movements have generally been from the south up and to the west. The initial folding also resulted from regional compression in the north-south direction.

Relaxation of this compression possibly accounts for some of the northwest and east-west steep-angle faults which were subsequently intruded by diabase dikes. The most predominant of these dikes trend in an east-west direction. The dikes range from stringers to intrusions about 100 ft. wide and in many cases are consistent over several miles. Some diabase sills were formed at the same time. Renewal of compression in the northwest direction resulted in thrust faults offsetting these dikes. At a later time, some thin (less than 1 ft.) basic dikes were intruded (1, 2).

Mine Geology

The mine in which the measurements were made is located in the south limb of the syncline. The orebody here, as shown in *Figure 7,* has a dip of between 13 degrees and 18 degrees north, with an east-west strike. The dip varies locally in the stopes, where measurements of from 10 to 25 degrees were noted (2).

The test area seemed to be relatively free from faults. To the east of the area, the predominant strike of near-vertical faults is N 80° E; to the west, it is about N 80° W. These are normal faults, with a downward displacement on the north side of a few inches to a few feet (2).

The joints in the mine are usually normal to the bedding and strike N 80° W, although a few were seen striking N 75° E. In the stopes in which the measurements were made, the joints have a strike close to east-west. They dip normal to the vein in most cases, although a few vertical dips were noted. The continuity of the joints does not seem to exceed 75 ft. on strike, and few could be traced that penetrated both roof and floor. The few joints striking N 75° E may have been associated with the faulting to the east. The absence of joints in the 1009 test stope can be correlated with the lack of variation in dip as compared to the 9 W 9 and the 1109 stopes, where the dips varied by as much as 5 degrees (2).

Nature of the Orebody

The ore generally occurs in closely packed conglomerates with a pyritic matrix. The close-packed conglomerates grade into loose-packed conglomerates with streaks of pyrite or, in some cases, no pyrite. The matrices of all the conglomerates comprise quartzite with a small amount of microline (2).

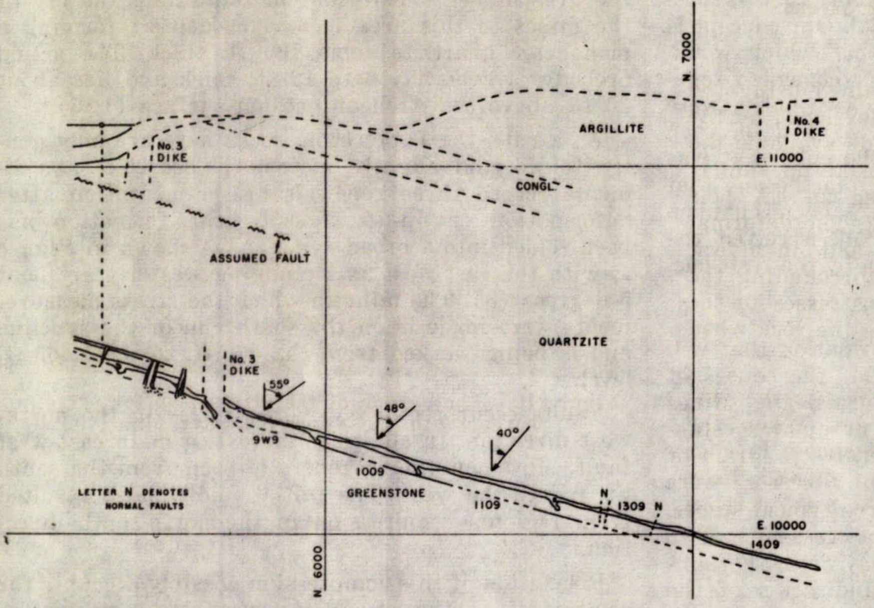

Figure 7.—Vertical section through the mine where measurements were taken showing the possible horizontal faulting, as indicated by the location of No. 3 dike, and also showing the average angle of inclination of the major principal stresses in the pillars in the three different stopes; the depth below ground surface of these stopes varied from 850 ft. to 1,010 ft.

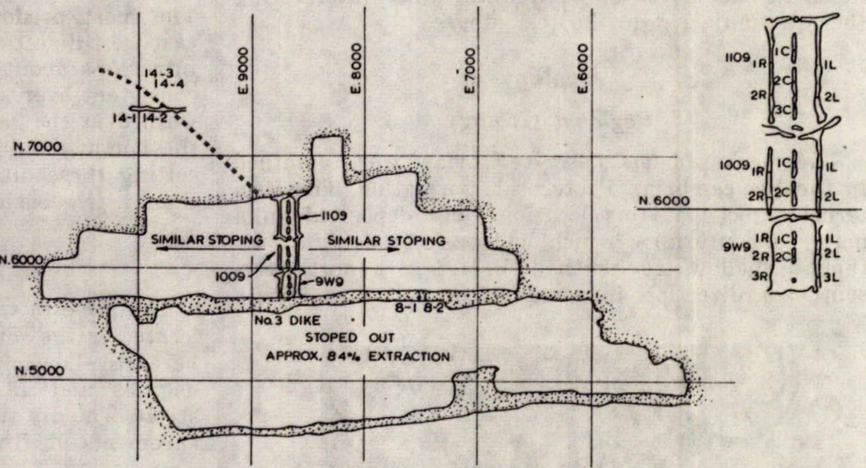

Figure 8.—The plan of part of the mine, showing the prevailing geometry and the location of the stress-measuring holes in the pillars, in the drifts away from the mining zone and in the dike above the stopes.

The pebbles of the conglomerates are generally quartz, with occasionally some black chert. They are between 0.5 and 2 inches in diameter and have a rounded to sub-rounded shape. Some angular fragments, obviously crushed pebbles, can also be seen (2).

The quartzite beds are as much as 5 ft. thick, and the grain size varies between 0.5 and 2 mm. The grains are sub-round, closely packed and cemented with sericite (2).

In the mine area, the orebody lies close to the Archean basement at the outcrop, is about 30 ft. above the basement in the test area and increases to about 50 ft. to the northwest. Above the orebody, there is about 500 ft. of massive, monolithic quartzite. This is overlain by a greenstone conglomerate up to 30 ft. thick, followed by 50 to 100 ft. of argillite. Four vertical diabase dikes 50 by 80 ft. thick, two of which are shown in *Figure 8*, cut the orebody in an east-west direction in the mine area (2).

Intrusives

The main diabase dike, No. 3, up-dip from the test area, as shown in *Figure 8*, is massive, fine to medium grained, has a width of 60 ft. and dips about 80 degrees to the south. The orebody in the adjacent areas rolls down on the south side and up on the north side. Normal faulting has been concentrated south of the dike. The dike rock has not been noticeably altered and outcrops on the surface about 300 ft. south of its upward projection from the intersection of the orebody, as shown in *Figure 7* (2). This displacement, or bedding thrust, may be evidence of additional regional compression in the north-south direction after the intrusion of the dikes.

Mining

The stope configuration is the conventional room and pillar type, as shown in *Figure 8*. Sill drifts are driven on strike, with stope lengths of between 200 and 350 ft. Rooms, or raises, are turned off up-dip, with a central pillar 10 ft. wide extending up-dip and two 65-ft. stopes on either side. These stopes are separated from the adjacent rooms by 10-ft. rib pillars. A crown pillar, nominally 15 ft. in breadth, is left between the top of the rooms and the next sill drift (3). In time, some of the crown pillars are mined out.

To commence stoping, twin raises are driven on either side of the central pillar from one level to the next. The face is then advanced, slashing the outer walls of the raise outward both up- and down-dip. The mining height varies from about 7 to 14 ft., 10 ft. being typical.

Method of Measurement of Stress

Theory

The stress measurements were obtained using the borehole deformation meter recently developed by the U.S. Bureau of Mines, who kindly made available the fabrication details so that we could make up similar units. The method consists of drilling an EX hole (1.5 inches in diameter) into the rock. The deformation meter is inserted into the hole and, in effect, acts as a feeler gauge. As the rock surrounding the hole is under compression due to the stresses in the ground, the hole is actually drilled into the rock when it is in a deformed state (4).

The stress in the rock around the hole is then relieved by overcoring with a 6-inch bit. When the stress is relieved the rock expands, as shown in *Figure 9,* which increases the diameter of the EX hole. The deformation meter then measures this increase in diameter in specific directions.

By measuring the increase in diameter of the EX hole resulting from the release of the surrounding stress by overcoring, it is possible to calculate the stresses acting in the plane perpendicular to the axis of the hole. In other words, just as a hole in rock would be deformed inwards as stress was applied to a rock mass, in the same way a hole is deformed outwards when stress is relieved from a rock mass—the deformation being dependent on the amount of stress being relieved and the modulus of deformation of the rock.

Operation

The drilling is done with a skid-mounted machine (BBS-2 gear box and winch) with an NX hydraulic head, powered by a two-cylinder, four-stroke, piston-type compressed air motor. Two photographs of the machine are shown in *Figure 10.*

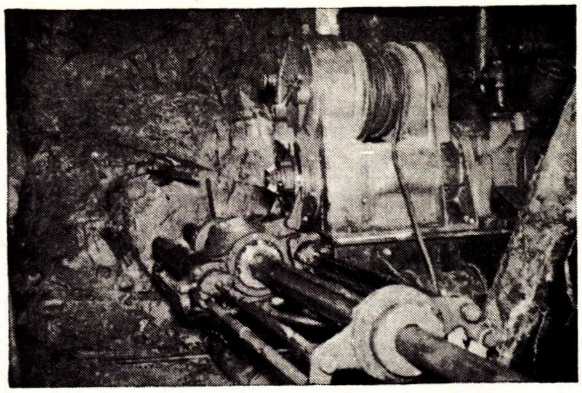

Figure 10.—Photograph of the drill set-up for overcoring with the 6-inch-diameter bit.

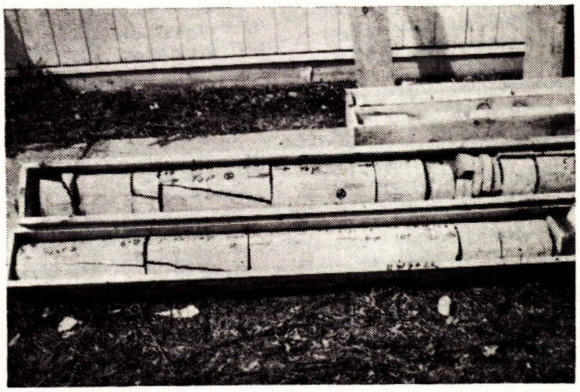

Figure 11.—Photograph of some of the core, showing the detailed marking and some minor discing (courtesy of R. C. Parsons).

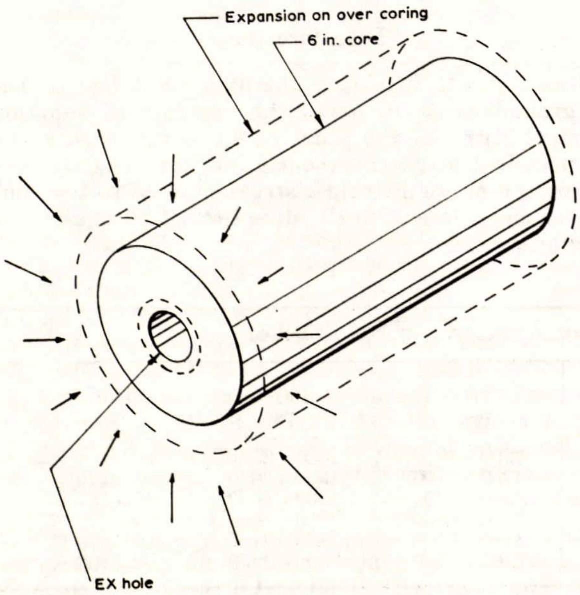

Figure 9.—Pictorial representation of the expansion occurring during the overcoring drilling as a result of stress relief.

Initially, a short length of 6-inch core is taken from the rock face to get past the surface fractured zone. Guides are then placed in the hole to center the EX core barrel. The EX hole is then drilled about 10 ft. ahead of the 6-inch core face.

The deformation meter is initially placed in the EX hole at the desired orientation 6 to 9 inches from the collar. The cable of the meter is brought out through the drill rods and attached to a strain-indicating device. The difference between the initial reading and the final reading during the overcoring operation is then a measure of the amount of deformation or expansion of the EX hole.

After overcoring one position, the 6-inch core is broken off the end of the hole, withdrawn and marked for the location of the deformation measurement, as shown in *Figure 11.* The cores are then shipped to the laboratory for determination of the modulus of deformation, uniaxial compressive strength, specific gravity and mineralogical identification.

In some cases, measurements were not obtained owing to the breaking of the core during the overcoring drilling. This occurred either from the presence of weakly cemented joint planes or from the discing and fracturing resulting from the relief of stress on the 6-inch core.

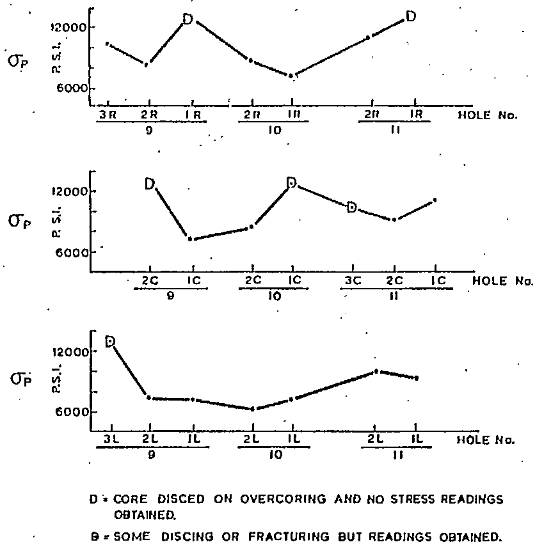

Figure 12.—Graph of the pillar stresses normal to the walls, along the lines of the left and right rib pillars and the central pillar, down-dip through the three stopes and showing where discing occurred.

Table I—Average Pillar Stresses

Hole No.	B ft.	z ft.	σ_1 psi	σ_2 psi	θ deg.	σ_v psi	σ_p psi
9 - 1R	9.4	870	disced	—	—	—	—
9 - 2R	8.3	860	11,000	4,500	55	6,640	8,200
9 - 3R	7.7	850	14,200	4,500	53	8,150	10,360
9 - 1L	14.5	870	10,500	4,900	59	6,380	7,700
9 - 2L	8.7	860	9,600	3,600	49	6,180	7,630
9 - 3L	10.7	850	15,600	11,000	54	12,590	13,700
9 - 1C	10.7	870	7,500	2,400	53	4,320	5,480
9 - 2C	8.8	860	disced	—	—	—	—
10 - 1R	10.2	910	9,500	3,600	52	5,830	7,270
10 - 2R	15.2	890	8,650	4,450	37	7,130	8,010
10 - 1L	14.8	910	8,500	3,700	50	5,680	6,840
10 - 2L	14.2	890	8,000	3,200	50	5,180	6,340
10 - 1C	8.7	910	14,000	6,100	42	10,460	12,260
10 - 2C	9.7	890	10,500	4,000	51	6,540	8,150
11 - 1R	8.3	1000	disced	—	—	—	—
11 - 2R	13.6	980	10,600	9,650	80	9,650	10,460
11 - 1L	13.6	1000	10,000	4,700	37	8,080	9,190
11 - 2L	10.7	980	10,800	4,640	25	9,700	10,580
11 - 1C	8.2	1010	13,300	4,350	50	8,040	10,210
11 - 2C	11.2	990	11,400	4,000	46	7,570	9,320
11 - 3C	10.3	960	11,900	5,600	45	8,750	10,230

B = breadth of pillar
z = depth from ground surface
σ_1 = major principal stress in pillar
σ_2 = intermediate principal stress in pillar
σ_v = vertical stress in pillar
σ_p = stress normal to walls in pillar
θ = angle from vertical to north, or down-dip, to σ_1

Experimental Results

Pillar Stresses

The results of the measurements in the pillars are shown in Table I. *Figure 8* shows the location of the holes where measurements were taken. The breadth of the pillars varied from 8 to 15 ft., with the typical breadth being about 11 ft. The depth below ground surface to the measuring holes varied from 850 to 1,010 ft. The major principal stresses that were measured varied from 7,500 psi to 15,600 psi, but in three of the holes measurements could not be obtained because of discing during overcoring. This probably indicates even higher stresses at these locations. *Figure 12* shows the variation of the pillar stresses normal to the walls along the lines of the rib and center pillar.

The inclination of the major principal stress, as shown in *Figure 7*, was neither vertical nor normal to the plane of the ore but was inclined up-dip at an angle of about 30 degrees to the orebody. This inclination shows that shear stresses exist on planes parallel to the walls; however, the magnitude of these stresses is relatively low. It is probable that the most significant pillar stresses are those normal to the walls (roof and floor). They vary from 5,500 psi up to 13,700 psi, and are possibly higher at the locations where discing occurred. *Figure 12* shows these normal stresses plotted along lines down-dip.

Field Stresses

The location of the holes away from the stoping and those in the dike just up-dip from the test stopes are also shown in *Figure 8*. Table II shows the results of the stress measurements at these other locations. At the location of holes 14-1 to 14-4, the vertical stress appears to have an average value of about 2,500 psi whereas the depth below the surface is only between 1,280 and 1,360 ft. (equivalent to about 1,600 psi). The horizontal stress parallel to strike is about 5,000 psi; the horizontal stress perpendicular to strike is about 4,500 psi.

Dike Stresses

Measurements in No. 3 dike at a point 840 ft. below ground surface showed the presence of a major principal stress of the order of 4,000 psi, with a minor principal stress of about 1,500 psi. The inclination of the major principal stress is close to horizontal and more or less in the direction of plunge of the syncline.

Table II—Stresses Away from Stoping Area and in No. 3 Dike

Hole No.	Direction of Hole	z ft.	σ_1 psi	σ_2 psi	θ deg.
14 - 1	W	1280	4500 - 5700	1000 - 3000	88 - 101 N
14 - 2	N	1280	4700 - 6400	1800 - 2900	92 - 114 E
14 - 3	S	1360	4500 - 5800	1900 - 4600	89 - 115 E
14 - 4	W	1360	3100 - 4800	2100 - 3000	65 - 72 N
8 - 1	N	840	4200 - 6000	800 - 2100	61 - 78 E
8 - 2	N	840	2200 - 4300	800 - 2200	50 - 96 E

z = depth from ground surface
σ_1 = major principal stress
σ_2 = intermediate principal stress
θ = angle from vertical to σ_1

Rock Properties

The results of laboratory tests conducted on the cores, or rock substance, are shown in Table III (5). These values may or may not be representative of the corresponding properties of the rock mass; more work is required to be able to measure these properties *in situ.*

For a complete classification of the rock substance, the pre-failure deformation characteristics and failure properties were determined. The criterion of more than 2μ in./in./hr. was used to classify viscous, as opposed to elastic, deformation. The characteristic value for the sample was obtained by plotting the test data on a strain versus log-time graph and extrapolating to 200 min. (6,7). The failure properties are characterized by the amount of plastic, or irrecoverable strain, before failure, more than 25 per cent of the total strain indicating a plastic, as opposed to a brittle, rock (7).

Fracturing in Holes

Considering the very high stresses that exist in many of the pillars and considering the stress concentration effects of making a hole in such a stress field, it might be expected that the walls of the holes would fail as a result of surface stresses of the order of 36,000 psi. An examination of the holes showed that some flaking did occur—mainly in the rib area at 90 degrees to the major principal stress.

As these observations were being gathered, the depths of the surface cracks in the pillars were recorded. Table IV shows the observations that were made concerning flaking in the 6-inch holes and the depth of surface fracturing in the pillars. Flaking took place in two of the three holes where discing occurred, where maximum stresses would be expected, but was also noted in other holes with much lower stresses. Several of the holes with high measured stresses did not produce flaking. In fact, there seems to be an equal distribution of the proportion of holes flaking to those not flaking over the entire range of measured stresses. Furthermore, if the flaking was due to the stress concentrations, it must be argued that this would be a progressive mechanism with the working into the rib areas continuing until the hole collapsed. This did not occur—only the initial shallow flake being produced in all cases.

The depth of the surface fracturing was commonly about 2 ft., with the range being from as little as 0.5 ft. up to an entire pillar containing fractures which would not likely be due simply to surface working or blast effects.

Table III—Rock Substance Test Data (5)

Samples	Uniaxial Compressive Strength (Standard Deviation) psi	Modulus of Deformation (Standard Deviation) psi	Viscous Strain Rate μ/hr.	Maximum Plastic Strain to Total Strain Ratio %	Classification
Quartzite	37,800 (5,840)	11.6×10^6 (0.3×10^6)	0	0	Very strong, elastic, brittle
Conglomerate	31,600 (6,970)	10.8×10^6 (1.7×10^6)	1.0	2	Very strong, elastic, brittle
Diabase	31,000 (10,900)	13.5×10^6 (1.7×10^6)	1.4	3	Very strong, elastic, brittle

Table IV—Observations on Flaking in 6-Inch Holes and Surface Fracturing in Pillars

Hole No.	Distance from Collar ft.	Areas of Flaking ins.	Inclination of Bisector of Flaking Areas deg. N.	Inclination of σ_1 deg. N.	Depth of Surface Fracturing from Collar of Hole ft.
9 - 1L	1.8	1 x ½	35	59	2
9 - 2L	—	—	—	—	0.5
9 - 3L	—	—	—	—	2
9 - 1C	—	—	—	—	2
9 - 2C	1.3	2 x 5	54	disced	2
9 - 1R	0.6 — 1.8	5 x 1	36	disced	2
9 - 2R	0.3 — 1.0	4 x 1½	27	55	2
9 - 3R	—	—	—	—	2
10 - 1L	0.6 — 0.8	2 x ½	34	50	1½
10 - 2L	0.3 — 0.9	2 x 2	38	50	1
10 - 1C	1.2	2 x 5	20	42	0.2
		2 x 5	44		
10 - 2C	—	—	—	—	1
10 - 1R	0.5	2 x 3	11	52	2
		2 x 3	27		
10 - 2R	—	—	—	—	2
11 - 1L	—	—	—	—	0.5
11 - 2L	—	—	—	—	1.5
11 - 1C	0.4	1 x 1	37	50	1
11 - 2C	—	—	—	—	1.5
11 - 1R	—	—	—	—	Fractured throughout
11 - 2R	—	—	—	—	1

Conclusions

1.—*The magnitude of the stresses measured in the pillars*, up to 15,600 psi, was unexpectedly high. The maximum measured stress has a component normal to the walls which is equal to twice the stress that would be calculated using the extraction ratio and the gravitational load, and more than twice the predicted load that would take into account the structural action of the deflections of the walls (8). Furthermore, whereas a down-dip component would normally be anticipated in the pillars, a strong up-dip component was actually found to exist. These stresses, however, cause no working of the pillars.

2.—*The field stress results were also surprising* in that, at the location of the measurements, the vertical stress was found to be approximately twice the gravitational stress and the horizontal stresses about three times the gravitational stress. Both can only be explained by crustal action. The presence of these high horizontal stresses explains the presence of the up-dip component of the pillar stresses.

It is possible that the presence of horizontal stresses greater than vertical stresses might be normal in the earth's crust, as indicated by the work of others and as shown in *Figure 13* (9). The measurements obtained at Elliot Lake, together with other measurements that have been obtained by the Mines Branch, are included in this figure (10, 11). It seems that it is not uncommon to have a horizontal stress at the surface of about 1,100 psi, increasing with depth at the rate of about 1,900 psi per 1,000 ft. of depth. At this rate, the average horizontal stress at the lower levels at Kirkland Lake might be as much as 16,000 psi.

3.—*Non-homogeneous field stress conditions* were shown to exist in this area at Elliot Lake. The structural history of folding, faulting and intrusive action have left residual stresses that probably vary with distance from the intrusives, the faults and the axes of the folds. It can be seen that a diagonal band of high stress exists in the 9 W 9 stope, with discing occurring in holes 1R and 2C and hole 3L having the highest measured stress. A similar band seems to exist in the 1109 stope, where holes 1R and 3C produced discing as did adjacent hole 10-1C. Furthermore, holes 14-1 to 14-4 showed considerable variation in field stress measurements from point to point, and this has since been corroborated by a completely new technique.

4.—*No correlation with depth* of the magnitude of the pillar stresses could be seen. It might, therefore, be concluded that the depth below the surface, or the gravitational loading, was not relevant to the pillar stresses due to the presence of residual, as opposed to gravitational, vertical stresses; however, this is not thought to be the case, but merely indicates that the variation with depth is obscured by the wide variations arising from other factors.

5.—*No obvious correlations with local geological features* were observed. An examination of individual logs of the cores together with the joint and stratigraphic patterns indicated only one broad correlation —1009 stope contained the fewest number of visible joints or faults and produced the lowest pillar stresses. The other stopes contained a high frequency of joints, which may have been associated with minor faulting (2).

6.—*A rough correlation with the breadth of the pillars* seems to exist. This is consistent with the structural action of the pillars on the wall deflections (8). Pillars with a breadth of between 7.5 and 9.5 ft. were subjected to approximately 2,000 psi greater stress than those pillars with a breadth of between 9.5 and 11.5 ft.; the latter, in turn, were subjected to stresses approximately 2,000 psi greater than those pillars with breadths of between 11.5 and 15 ft.

Acknowledgments

The authors would like to thank the following individuals for their assistance in this work: T. S. Cochrane, I. Bain, R. C. Parson, A. St. Louis, W. Zawadski, J. St. Onge, J. Sullivan, S. Cook and M. S. Ellis.

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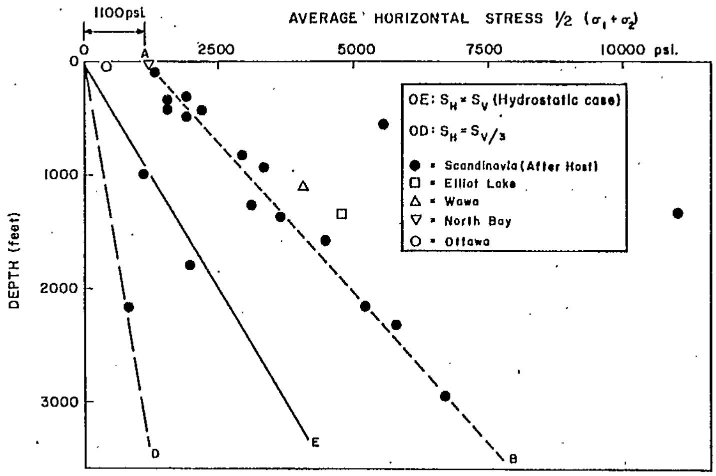

Figure 13.—Graph of horizontal stresses measured in Scandinavia and Canada and their variation with depth, showing horizontal stresses at the surface of 1,100 psi increasing with depth at the rate of 1,900 psi per thousand feet (after Ref. 9, 10, 11).

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APPENDIX

The phenomenon of discing in cores is generally accepted as a manifestation of high stress in the rock (13, 14). *Figure 14* shows some of the discs obtained in this work at Elliot Lake. Experiments in the laboratory have shown that it is possible to create discing by the compression of part of the length of a core (12, 14). Furthermore, it is suggested here that the slabbing of thin layers of rock that is seen in drifts and stopes is the result of the same mechanism; i.e., by the initiation of shearing associated with high compressive stress, a surface slab of rock is relieved of its stress and hence it slabs or discs. *Figure 15* shows some such slabs collected by Professor R. G. K. Morrison at Elliot Lake, and *Figure 16* shows some from his collection from other mines.

The stress distribution resulting from the compression of part of the length of a core has been solved using the theory of elasticity (15). *Figure 17* shows the results of this solution. *Figure 17(b)* shows the variation of the shear stress on the transverse plane at the origin, which has a maximum value of 0.27p, where p is the external pressure applied radially around the core. *Figure 17(c)* shows the variation of the axial stress on the transverse plane through

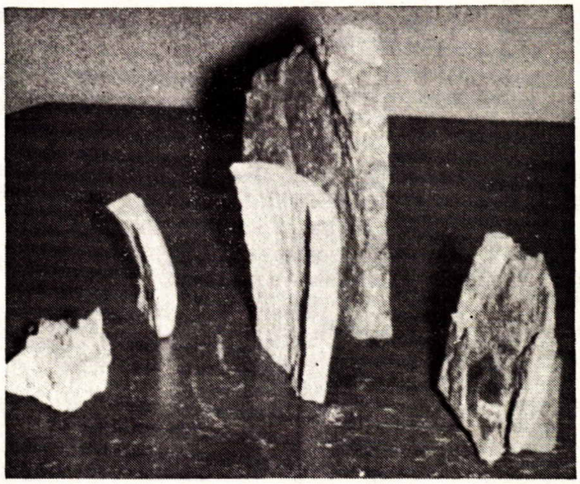

Figure 15.—Thin slabs, possibly analogous to discing, collected in mines at Elliot Lake by Professor R. G. K. Morrison.

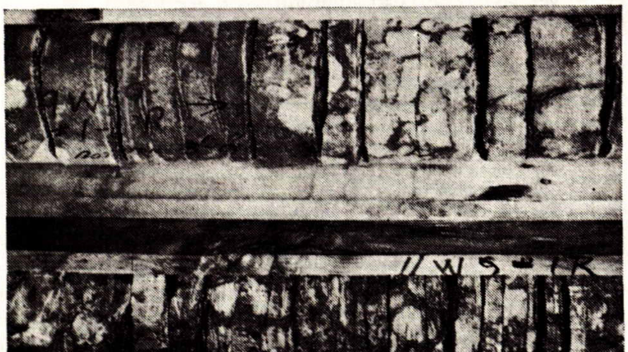

Figure 14.—Photograph of the discs obtained in hole No. 11-1R with thicknesses of from 0.5 to 3 inches (courtesy of R. C. Parsons).

Figure 16.—Thin slabs collected from other mines by Professor R. G. K. Morrison.

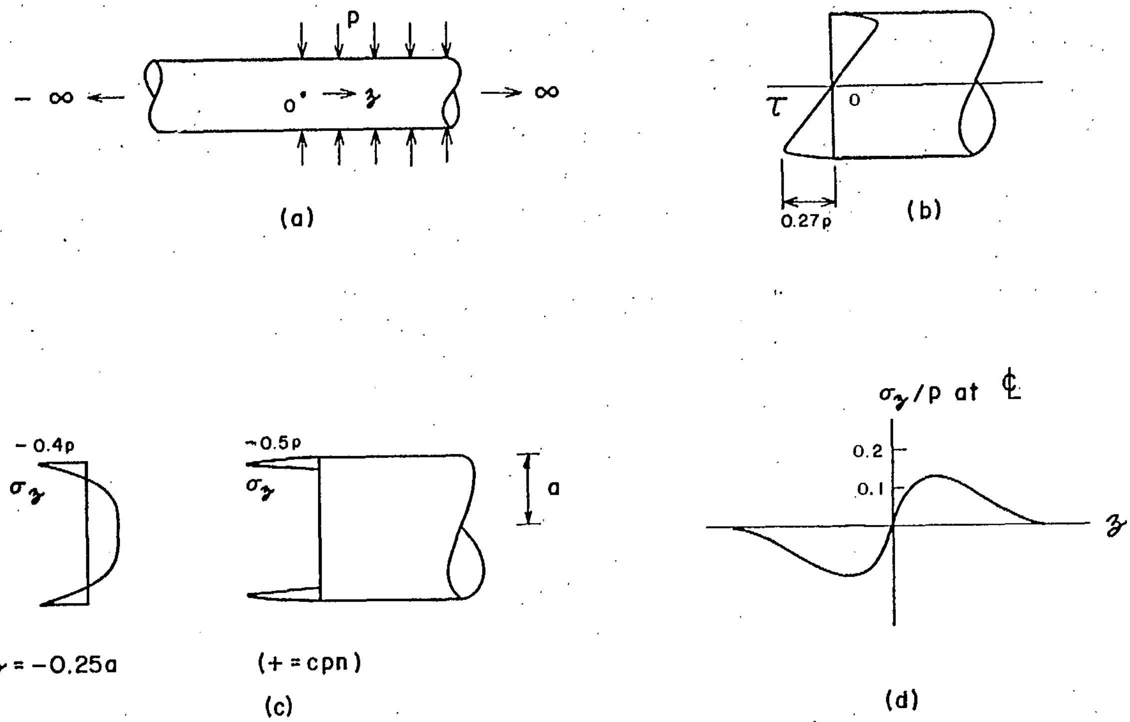

Figure 17.—Stress distribution in the core stub resulting from stress relief as determined by the Theory of Elasticity: (a) core extends to infinity and is squeezed with pressure for half its length; (b) variation of shear stress, τ, on the transverse plane at the origin; (c) variation of axial stress, σ_z, at the origin and on the transverse plane 0.25a into unloaded core stub; and (d) variation of axial stress, σ_z, at the center of the core along the core (after Ref. 15).

the origin and on a transverse plane 0.25 times the radius of the core away from the origin into the overcored stub. These graphs indicate that a tensile stress is created on the surface of the core in the axial direction with a maximum magnitude of 0.5p. *Figure 17(d)* shows the variation of the axial stress along the axis of the core, with compression occurring in the stub length and tension occurring within the material still subjected to compressive stress.

By postulating that discing results from shearing and is initiated by the maximum shear stress, the following formula should relate the strength of the material to the stress that has been relieved:

$$\tau_m = 0.27p \qquad (1)$$

Alternatively, it could be assumed that discing is caused by the initiation of a crack at the surface of the core by the tensile stresses acting parallel to the axis, σ_z. The formula then relating strength to stress would be:

$$\sigma_z = -0.5p \qquad (2)$$

In previous testing, it was found that under triaxial stress conditions the Elliot Lake quartzite had an angle of internal friction of about 55 degrees (16). Using this value, it is calculated that the cohesion, according to Mohr's Strength Theory, is equal to $0.157Q_u$, where Q_u is the uniaxial compressive strength. In this case, following a previous suggestion (17), a calculation could be made of the stress in the ground, assuming it to be hydrostatic, before relief caused the discing. Based on *Equation 1*, such a calculation could be made with the following equation:

$$p = 0.58Q_u \qquad (3)$$

Alternatively, if the discing is caused by the tensile stress and, following Griffith's theory, the tensile

Table V—Measurements Pertaining to Discing in 5.62-in.-diam. Cores

Hole No.	Uniaxial Compressive Strength in Cores[x] psi	Width of Discs. in.	Depth from Collar ft.	Major Principal Stress psi
9 - 1R...	34,200	1.2	2 to 9	11,000[y]
9 - 2R...	39,700	0.9	1.0	11,000[y]
	39,700	1.3	4.5	11,000[y]
10 - 1R...	38,000	0.7	6.0	9,500
		6.0	6.5	9,500
10 - 2R...	20,900	0.5	1.6	8,650
10 - 1C...	40,600	1.0	2.0	14,000
11 - 1R...	42,000	0.9	1.5	13,300[y]
		1.3	3.5	13,300[y]
11 - 3C...	36,700	2.0	1.5	11,900
		1.5	8.0	11,900

[x]Average of hole or of adjacent holes on strike
[y]Maximum in adjacent holes

strength is assumed to be $\frac{1}{8}Q_u$, then the following equation, based on *Equation 2*, would be used:

$$p = 0.25Q_u \qquad (4)$$

A second approach is to use a strength-of-material type of analysis, as shown in *Figure 18(a)*, where each half of the stub of the core is considered to be analogous to a cantilever beam. This analysis results in a similar distribution of shear stress across the core, as shown in *Figure 18(b)*, to that obtained with the theory of elasticity. It is not suggested that this analysis is rigorous; however, it indicates that the maximum shear stress is related to the relative length of the cantilever, L/R, or the thickness of the discs, as has been suggested by others to be significant

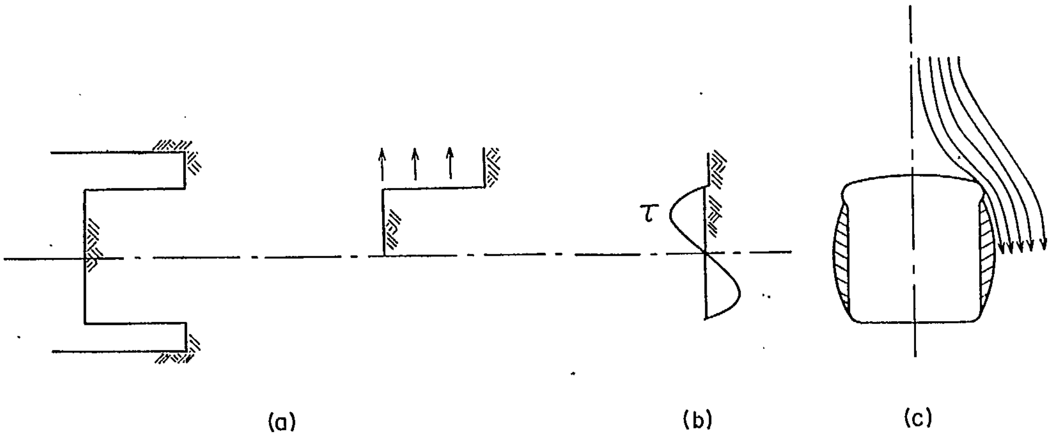

Figure 18.—(a) A cantilever analogue for stress relief producing shear stresses in the stub of the core; (b) variation of the shear stresses across the core; (c) analogous shearing due to stress relief producing slabbing in a face.

(14). *Figure 18(c)* shows the analogous case of slabbing on a face. From this analysis, the maximum shear stress would be calculated according to the following equation:

$$\tau_m = \frac{k\sigma_u(2R)L}{0.5\,R^2} \qquad (5)$$

where k is the ratio of the maximum shear stress to the average shear stress across the section, σ_u is the relieved stress (assumed to be uniaxial), L is the length of the core stub or thickness of the disc and R is the radius of the core. As the factor k for a semicircle is about 1.4, *Equation 5* can be rewritten for the assumption of discing due to shear stresses as follows:

$$\sigma_u = 0.0882\,Q_u\,R/L \qquad (6)$$

Alternatively, assuming that discing is initiated by the tensile stress associated with the pure shear stress of *Equation 5*, i.e., being equal in magnitude and on planes at 45 degrees to the shear planes, the following equation is obtained:

$$\sigma_u = 0.0702\,Q_u\,R/L \qquad (7)$$

Table V shows the following: the holes in which discing occurred; the average uniaxial compressive strength either obtained in that hole or, where discing occurred throughout, in the adjacent holes on strike; the width of the discs; their location in the pillar; and the major principal stress either in that pillar or, where measurements could not be made, the maximum measured in an adjacent hole. One paradox here is that none of the EX core disced, although cores were drilled through the zones where the 6-inch cores subsequently disced.

As a trial calculation, the figures for hole 10-1C can be examined, with the additional factor being recognized that the stress concentration at the end of the hole is 1.53 so that the field stress, S, equals p/1.53 or σ_u/1.53 (17):

Equation 3	p	= 23,500 psi
	S	= 15,400 psi
Equation 4	p	= 10,200 psi
	S	= 6,700 psi
Equation 6	σ_u	= 10,100 psi
	S	= 6,600 psi
Equation 7	σ_u	= 8,000 psi
	S	= 5,200 psi

From these figures, it would seem that some correlation between discing and stress might be established. More information, however, is required on the mechanism of failure, the applicability of laboratory testing, the effectiveness of the stress concentration acting over the infinitesimal element adjacent to the bottom of the hole, and the analysis for biaxial stress, as opposed to hydrostatic stress as assumed in *Equations 3* and *4* and uniaxial stress in *Equations 6* and *7*.

(*Reprinted from The Canadian Mining and Metallurgical Bulletin, May,* 1966)

Printed in Canada

Int. J. Rock Mech. Min. Sci. Vol. 4, pp. 367–382. Pergamon Press Ltd. 1967. Printed in Great Britain

A COMPARISON OF TWO METHODS FOR MEASURING STRESS IN ROCK

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(*Received* 20 *January* 1967)

Abstract—As part of an extensive program of measuring rock stresses, a series of comparative measurements were made in a Canadian uranium mine by means of a strain cell developed at the South African Council for Scientific and Industrial Research and of a borehole deformation meter developed at the U.S. Bureau of Mines. An analysis of the results, taking account of certain sources of error, shows that there is satisfactory agreement between the results obtained with the two methods.

1. INTRODUCTION

DURING recent years increasing consideration has been given to the problem of measuring the stress in rock around underground excavations. LEEMAN [1] has published an excellent review of the instruments and techniques available for this purpose. In Canada an extensive program has been pursued since 1963, and this paper, in effect, is a second progress report (the first having been published last year [7]).

The instruments that are most often used today can be divided into two classes. One class, the borehole deformation gauge, measures the changes in length of one or more diameters of a borehole drilled in the rock when the stress in the rock surrounding the gauge is relieved by drilling an annular groove concentric with the borehole. Because the over-coring hole is usually 6 in. or more in diameter, the method is expensive and practical drilling difficulties restrict the depth at which a measurement can be obtained.

The second class of instrument, which takes the form of a so-called strain cell containing electrical resistance strain gauges or a photoelastic disc, is glued on the flattened end of a borehole drilled in the rock. It measures the change in strain of the rock on which it is glued when the stress is relieved by extending the length of the borehole by means of a coring drill. The small size of the hole required and the simplicity of the overcoring and installation procedure make it possible to obtain measurements at a considerable depth inside the rock mass.

In this paper the results obtained using a borehole deformation meter developed at the U.S. Bureau of Mines [2] (referred to as a deformation meter in this paper) with the results given by a borehole strain gauge device [1] developed at the South African Council for Scientific and Industrial Research (referred to as a strain cell in this paper) are compared.

Both instruments were tested in the laboratory [3, 4] and were found to perform satisfactorily under uniaxial and biaxial loading conditions. In this paper the performance

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of the two instruments, under the more complex stress conditions which are encountered underground, are compared.

2. DESCRIPTION OF THE INSTRUMENTS

The instruments used for the measurements have been described elsewhere [1, 2] so that only a brief description will be included here.

(a) *U.S. Bureau of Mines Borehole Deformation Meter*

The borehole deformation meter, which was developed in the U.S. Bureau of Mines, measures the changes in length of a single diameter of a borehole. The measuring element is a beryllium copper cantilever on which four resistance strain gauges are mounted and connected to form a Wheatstone bridge. The cantilever also produces the force required to keep the piston in contact with the sidewalls of the borehole.

When the deformation meter is installed in a borehole, any change in diameter of the borehole is transmitted via the piston to the cantilever.

The change in bending strain produced in the cantilever is measured on a conventional strain indicator. Changes in diameter of as little as 50 μin. can be measured with this meter. The reference diameter of the instrument can be changed by changing either the piston length or the spacing stud directly opposite the piston. This allows the meter to be adjusted for use in an oversize hole. The gauges on the cantilever are waterproof and dustproof, and the meter has a temperature sensitivity of less than 10 μin/°F.

(b) *CSIR Strain Gauge Strain Cell* (*'Doorstopper'*)

This strain cell was developed at the South African Council for Scientific and Industrial Research. It measures changes in strain in three directions on the end of a borehole. The measuring element is a conventional rectangular strain gauge rosette, the individual gauges of which are oriented to measure changes in strain in the vertical, 45° and horizontal directions. The leads from the gauges are connected to 4 pins in an insulated connector plug. Both the plug and the gauges are encapsulated in a silicone rubber compound which provides physical protection as well as waterproofing for the strain gauges.

The strain cell is installed into the borehole by means of a special installating tool [5, 6]. In this method the rock on which the cell is cemented is stress-relieved by extending the length of the borehole.

3. METHOD OF MEASUREMENT

(a) *Theoretical*

In using either of the methods described above, the principal stresses, σ_1 and σ_2, acting in the plane normal to the borehole, are obtained by measuring the deformation or strain in at least three different directions. The calculation of the magnitudes and directions of σ_1 and σ_2 from the measurements depends, in general, on the assumption that the measurements are insensitive to any normal stress parallel to the direction of the borehole.

For purposes of comparing the two instruments underground, the boreholes in which the measurements were made were drilled parallel to each other. Thus at any depth of the boreholes the stresses to be measured could be considered to be the same. Any differences could only be the result of inaccuracies introduced by the instruments, inaccuracies in determining the elastic constants of the rock and inaccuracies due to the degree to which the above assumption was not fulfilled.

(i) *U.S.B.M. Deformation Meter*. The formulae to calculate the stresses from borehole deformation measurements obtained with the U.S.B.M. Deformation Meter were derived by MERRILL and PETERSON [3]. They are as follows:

If

u_1 = deformation measured across a diameter of the borehole (see Fig. 1)
u_2 = deformation measured in a direction 60° anticlockwise from u_1 (usually the vertical direction)
u_3 = deformation in a direction 120° anticlockwise from u_1
θ_1 = angle measured anticlockwise from σ_1 to the direction of u_1
D = diameter of the borehole
E = modulus of elasticity of the rock

then

$$\sigma_1 + \sigma_2 = \frac{E}{3D}(u_1 + u_2 + u_3)$$

$$\sigma_1 - \sigma_2 = \frac{\sqrt{2}E}{6D}\left[(u_1 - u_3)^2 + (u_2 - u_2)^2 + (u_1 - u_3)^2\right]^{\frac{1}{2}}$$

from which the magnitudes of σ_1 and σ_2 can be determined. The direction of σ_1 is given by:

$$\tan 2\theta_1 = \frac{-\sqrt{3}(u_2 - u_3)}{2u_1 - u_2 - u_3}.$$

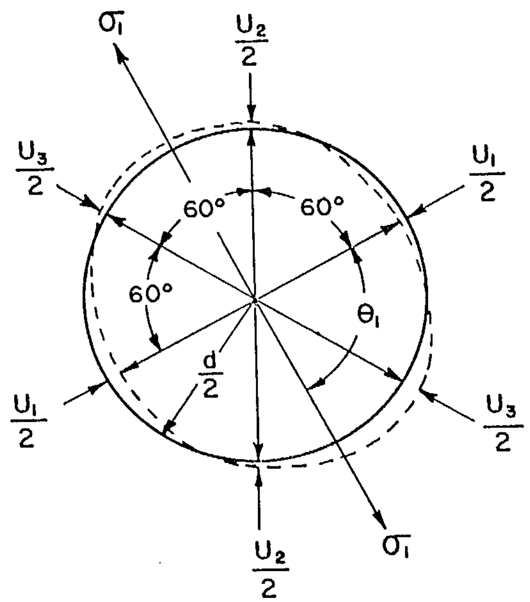

FIG. 1. Direction of stresses and deformations (U.S.B.M. deformation meter).

(ii) *CSIR Strain Cell*. The formulae used to calculate stresses from strain measurements obtained with the CSIR Strain Cell are well-known strain rosette equations derived from the theory of elasticity. They were included in a recent publication by LEEMAN [4].

Let the difference in the strain readings in the vertical, 45° and horizontal directions before and after overcoring be e_V, e_{45} and e_H respectively.

The principal strains in the rock on the end of the borehole are given by:

$$e_{1,2} = \tfrac{1}{2}\left[(e_H + e_V) \pm \sqrt{(\{2e_{45} - (e_H + e_V)\}^2 + (e_H - e_V))^2}\right].$$

The principal stresses σ_1' and σ_2' in the rock on the flat end of the borehole are:

$$\sigma_1' = \frac{E}{1 - \nu^2}(e_1 + \nu e_2)$$

$$\sigma_2' = \frac{E}{1 - \nu^2}(e_2 + \nu e_1)$$

where

E = modulus of elasticity of the rock
ν = Poisson's ratio of the rock.

The principal stresses σ_1 and σ_2 in the rock surrounding the borehole can be obtained from

$$\sigma_1 = \frac{1}{1{\cdot}53}\sigma_1',\ \sigma_2 = \frac{1}{1{\cdot}53}\cdot\sigma_2.$$

The direction of σ_1 can be determined from:

$$\tan\theta = \frac{2(e_1 - e_H)}{2e_{45} - (e_H + e_V)}$$

where θ is measured anticlockwise from the horizontal direction (see Fig. 2).

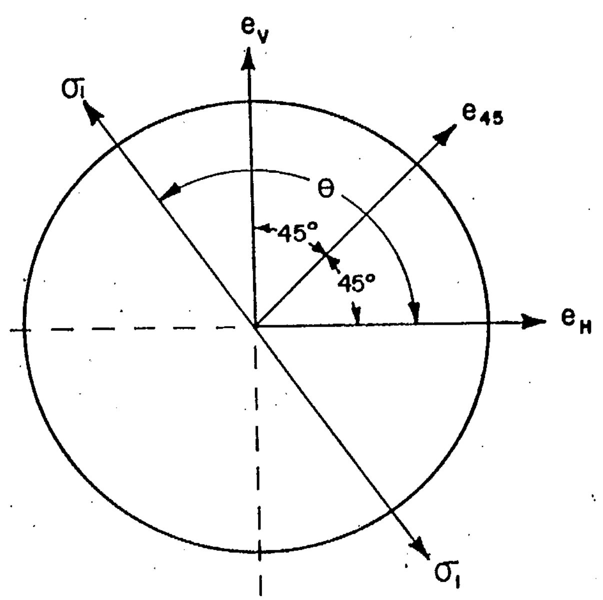

FIG. 2. Direction of stresses and strains (CSIR Strain Cell).

(b) *Experimental*

(i) *U.S.B.M. Deformation Meter*. The procedure used to install the deformation meter and to relieve the stress in the rock surrounding it involves the following sequence of steps.

A 6 in. hole is drilled sufficiently far into the rock to get beyond the fractured zone near the rock face. The core is removed, and guides are placed in the hole to center the EX core barrel. An EX hole is then drilled into the center of the end of the 6 in. borehole to a depth of 10 ft or more ahead of the end of the 6 in. hole.

The deformation meter is placed in the EX hole 6–9 in. from the end of the 6 in. hole and oriented to measure changes in the vertical diameter of the borehole. The cable from the meter is brought out through the drill rods and connected to a strain gauge bridge, an initial reading is taken, and the 6 in. drill is advanced. Readings are taken at regular intervals until the meter is overcored. A final reading is taken and the core is removed.

The difference, u_1, between the initial reading and the final reading during the overcoring operation is a measure of the borehole deformation in this direction.

The meter is then installed deeper in the *EX* borehole and oriented to measure changes in diameter in a direction 60° anticlockwise to the direction of u_1. It is again overcored. and a measurement u_2 is obtained.

(ii) *The CSIR Strain Cell.* To use this strain cell a BX size ($2\frac{3}{8}$in. diameter) borehole is drilled into the rock to the depth at which the stresses are to be determined.

The end of the borehole is ground flat and smooth with a square-faced diamond bit and a flat-faced diamond impregnated bit. The end of the borehole is dried by wiping it with a piece of cloth soaked in a suitable solvent.

The strain cell to be used is cleaned, plugged into the installing tool and smeared with a uniform layer of glue. The complete assembly is pushed up to the end of the borehole.

On reaching the end of the borehole the tool is oriented and the cell is pushed against the end of the hole by applying sufficient pressure on the installing rods.

Once the glue has set and the strain readings become constant, the tool is removed, leaving the strain cell adhering to the end of the borehole.

The drill using a BX coring bit is inserted in the borehole, and the rock to which the strain cell is attached is cored out. On removal from the borehole the strain cell, adhering to a length of core, is plugged into the installing tool and a final strain reading is taken. The strain resulting from the stress-relieving operation is the difference between the initial and final strain readings.

The end of the hole is flattened and the procedure repeated.

(iii) *Tests performed on* 6 *in. core samples*. Each of the 6 in. core pieces in which a deformation meter measurement was obtained was returned to the laboratory. Cylindrical specimens with their axes parallel to the directions in which field measurements were taken were prepared from them.

Simple uniaxial compression tests were performed on these specimens from which the modulus of elasticity of the rock was determined. Lateral strain gauges were glued to some of these specimens in order to determine the Poisson's ratio of the rock.

(iv) *Tests performed on BX core samples.* As a confirmation that the strain cells were properly glued to the rock, tests were conducted on half-inch thick discs cut from the ends of the core to which the strain cells were attached.

Compressive loads were applied to the cylindrical surface of the discs at two diametrically opposite points and parallel to, say, the vertical strain gauge in the rosette. Readings from all the strain gauges were taken at fixed increments of loads.

The response of the gauges was plotted against the applied load, and if there were no peculiarities in the curves it was assumed that the bond between the strain cell and the rock was adequate. The modulus of elasticity and the Poisson's ratio of the rock were obtained from the core by means of uniaxial compression tests in the laboratory. Unfortunately some of the core pieces were not recovered and others were not suitable for compression testing so that these tests were performed at about half the stress-relieving stations.

4. DESCRIPTION OF THE TEST SITES

(a) *Test-Site 1—Measurements in rock undisturbed by stoping*

Measurements were made with both instruments in No. 14 extension drift in the hanging-wall of the mine. The drift is 1400 ft below surface and the average dip of the formation is 14°.

Three horizontal boreholes were drilled in the sidewall of the drift as shown in Fig. 3. Borehole No. 1 was used for deformation meter measurements and boreholes No. 2 and 3 for strain cell measurements.

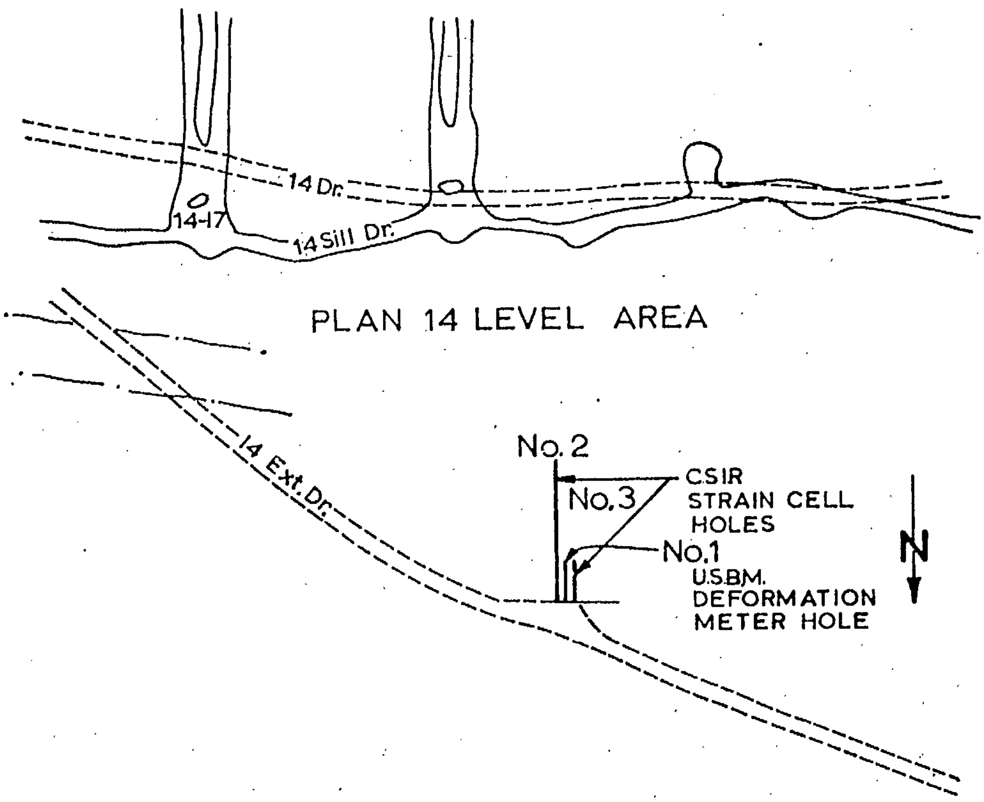

FIG. 3. Position of the boreholes in the 14 extension drift.

(b) *Test Site 2—Measurements in pillars*

Both instruments were used to determine the stresses in several narrow pillars in a mined-out area of the mine. The ore is mined on a room-and-pillar system [7]. The stopes are approximately 250 ft long on dip by 140 ft wide and are separated by pillars 10 ft wide.

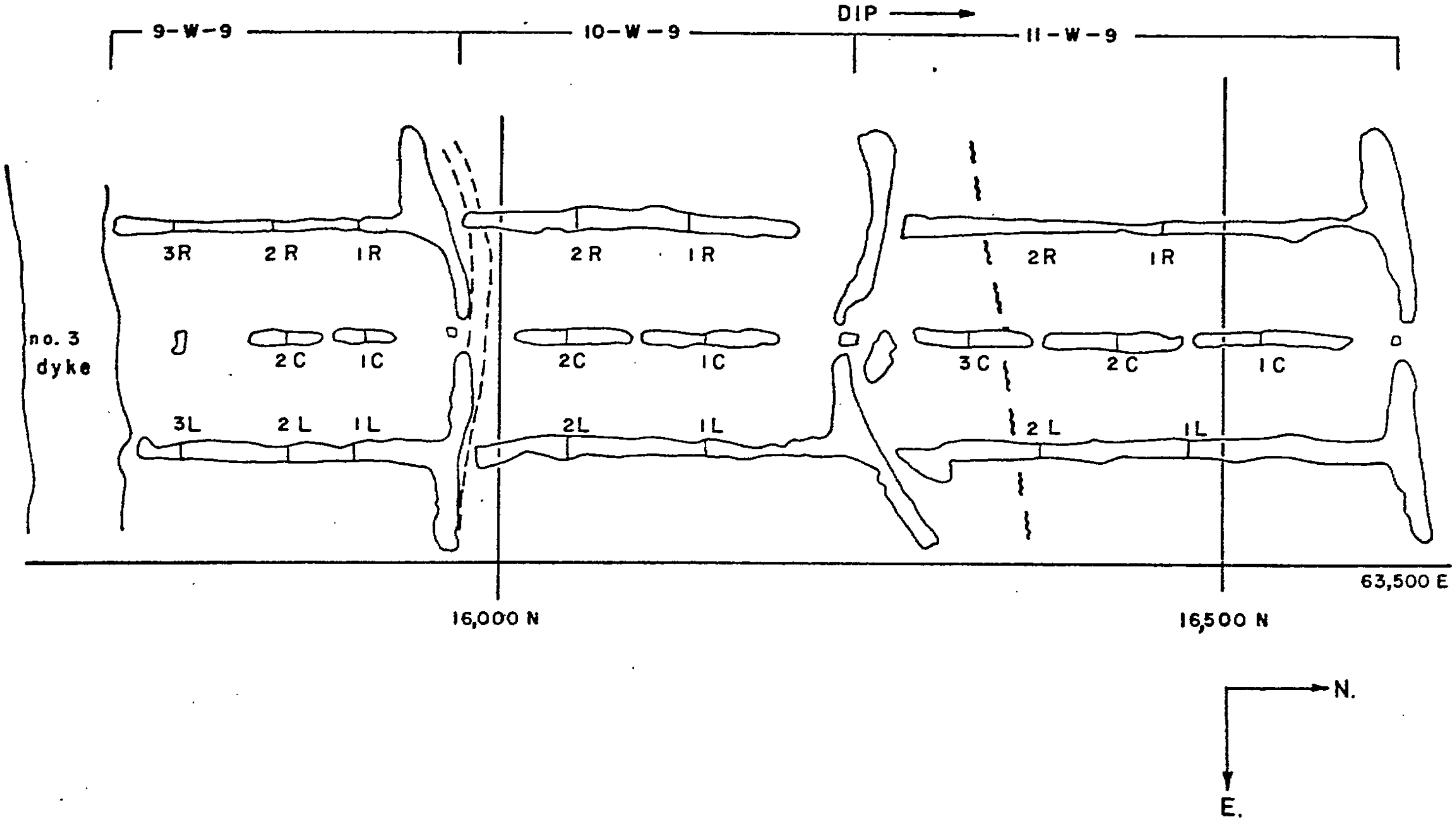

FIG. 4. Plan of stopes showing pillars in which measurements were taken.

Holes for the strain cell measurements were drilled parallel and close to holes in which the deformation meter measurements were taken. The pillars in which measurements were taken as well as the general layout of the stopes are shown in Fig. 4.

5. EXPERIMENTAL RESULTS

(a) *Field stresses, Test Site* 1

The results of the measurements made with each instrument in the 14 level extension drift are given in Table 1 and Figs. 5–7. Figures 5 and 6 show the variation, with distance into the solid, of the major and minor principal stresses (σ_1 and σ_2) respectively. Figure 7 shows the variation of the major principal stress direction with distance into the solid.

TABLE 1. RESULTS OBTAINED IN THREE BOREHOLES DRILLED IN 14 EXTENSION DRIFT

U.S.B.M. Deformation Meter results—Borehole No. 1

Depth of hole (ft)	Borehole deformation in. × 10^{-4} u_1	u_2	u_3	Principal stresses (psi) σ_1	σ_2	Angle (degrees from vertical) θ
5	15·75	3·0	13·1	5542	2580	84
10	10·10	0·5	13·1	4469	1573	95
12	4·90	0·5	12·0	3497	940	101
14	8·60	0·75	8·6	3220	1307	88
16	1·60	1·50	5·2	3457	1209	70
18	13·50	2·25	5·6	4017	1427	69
20	14·25	3·70	10·5	4806	2448	80
25	15·0	6·75	13·87	5514	3543	86
30	15·0	8·25	10·5	5060	3540	70

CSIR Strain Gauge Strain Cell—Borehole No. 2

Depth of hole (ft)	Strain on end of hole (μin/in) e_V	e_{45}	e_H	Principal stresses (psi) σ_1	σ_2	Angle (degrees from vertical) θ
4	18	361	557	4700	1000	82
6	−31	242	400	3300	340	83
8	24	330	355	3300	380	70
11	−0	220	352	3000	530	83
14	−44	106	273	2200	80	90
18	−5	278	344	3070	320	74
21	88	242	322	2800	1200	79
25	−2	163	374	3140	900	92
30	134	355	565	5100	2000	90

CSIR Strain Gauge Strain Cell—Borehole No. 3

Depth of hole (ft)	Strain on end of hole (μin/in) e_V	e_{45}	e_H	Principal stresses (psi) σ_1	σ_2	Angle (degrees from vertical) θ
4	786	466	202	6800	2900	90
6 ft 11 in.	207	229	520	5000	2200	110
10 ft 1 in.	70	185	400	3200	145	98
12 ft 5 in.	−18	88	388	3400	185	103
13 ft 2 in.	13	190	352	2900	670	90
15	11	114	388	3400	600	102
19	−74	198	370	2950	−110	84

In analysing the deformation meter results, two obviously erroneous results were rejected. Plots were made of borehole deformation u_i ($i = 1, 2, 3$) vs. distance from the collar of the borehole for each orientation of the meter in the borehole. Values for the borehole deformations u_1, u_2 and u_3, determined at intervals, along the length of the borehole, by interpolation from these plots were used to calculate the deformation meter results shown in Figs. 5–7.

The strain cell results in Figs. 5–7 are the average of measurements made in boreholes No. 2 and 3. Except for the first 10 ft the scatter in the results obtained in these two boreholes was less than 10 per cent. The scatter in the results over the first few feet can be attributed mainly to the fact that these measurements were obtained during a period when the strain cells and installing equipment were undergoing initial evaluation tests when the underground crew was not yet fully acquainted with the installation procedure.

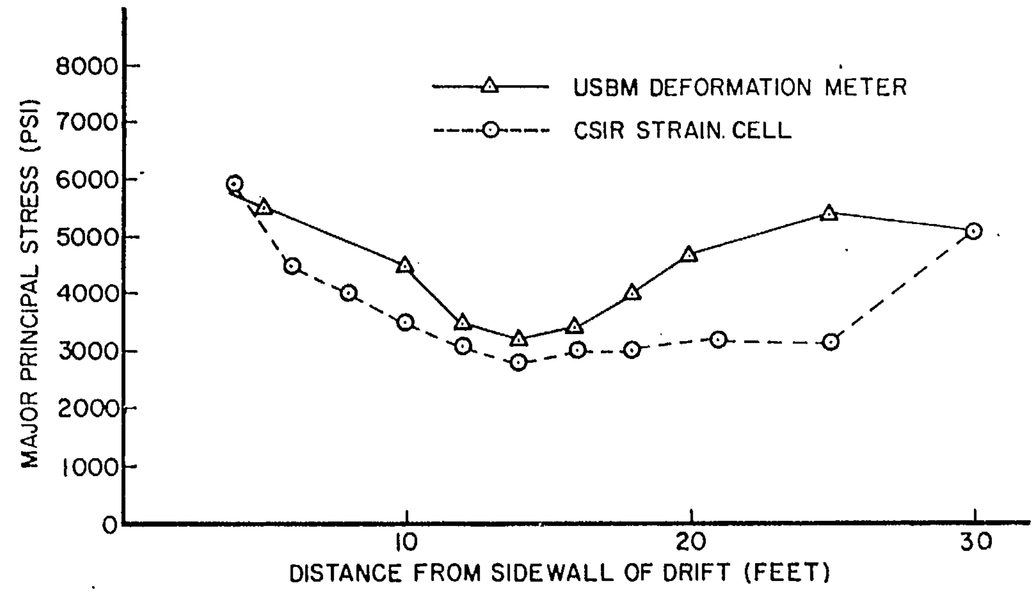

FIG. 5. Graph showing variation of major principal stress (as determined with each of the two methods) with distance into the solid.

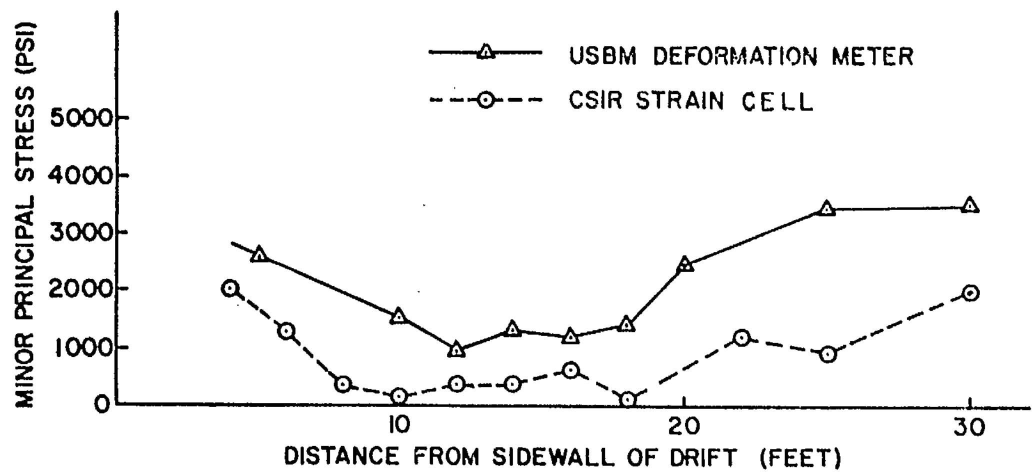

FIG. 6. Graph showing the variation of the minor principal stress (as determined with each of the two methods) with distance into the solid.

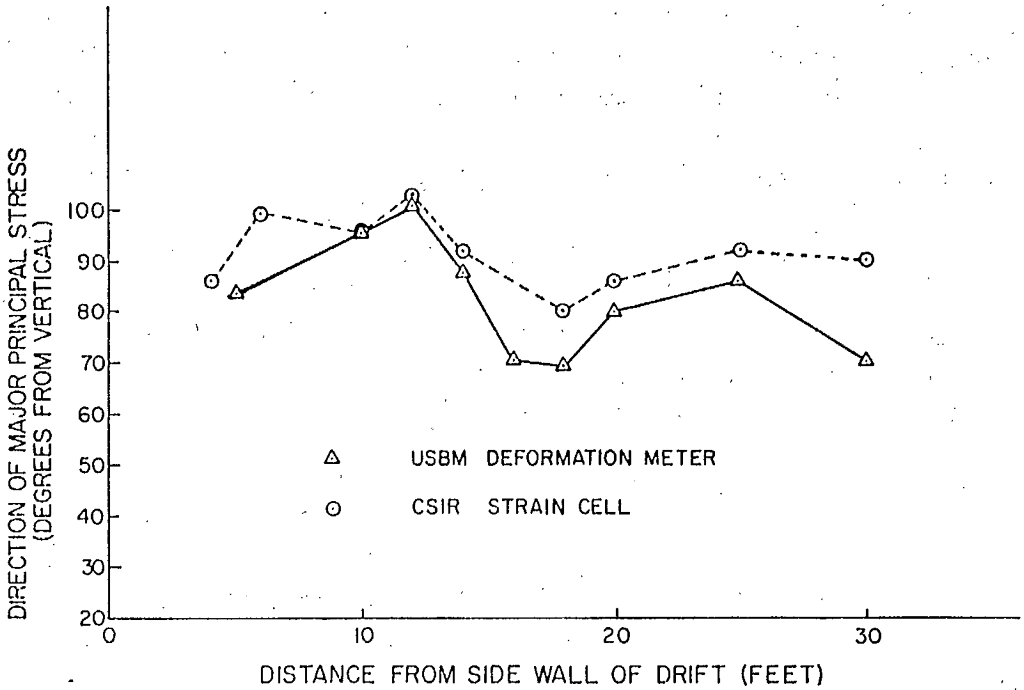

FIG. 7. Variation of the major principal stress direction with distance into the solid.

Measurements were obtained in borehole No. 2 for a distance of 80 ft into the solid. Since the deformation meter measurements were discontinued at a depth of 30 ft, the strain cell results between 30 and 80 ft could not be used for comparison purposes and are, therefore, not included in this paper.

As can be seen from Fig. 7, both methods showed that the major principal stress is acting in a direction of between 70 and 100° from the vertical direction (i.e. close to the horizontal direction). The magnitudes of both major and minor principal stresses as determined with the deformation meter are consistently higher than those indicated by the strain cell. It is believed that some of the reasons for this are as follows:

(1) In some cases the deformation meter may have been installed too close to the end face of the 6 in. overcoring hole when the initial reading was taken. For this reason the diameter of the hole at the point of measurement may have been affected by the stress concentration ahead of the 6 in. overcoring hole. Although a distance of 6 in. (one diameter) was used, a decrease in the borehole diameter was detected as soon as the overcoring hole was advanced. This might mean that the deformation meter was not completely outside the zone of influence of the overcoring hole; thus the measurements would be, therefore, slightly high because the stress concentration effect is included in the reading.

(2) In some cases the overcoring operation had to be discontinued while the diameter of the borehole was still changing. This resulted in measurements which were either too high or too low depending on the direction in which they were taken [9].

(3) The construction of the strain cells used in these tests was such that the strain had to be transmitted to the strain gauges via a thin piece of Araldite shim. This resulted in strain readings which were as much as 6 per cent too low.

These errors increased the difference between the results obtained with the two instruments.

TABLE 2. RESULTS OBTAINED FROM STRAIN CELL MEASUREMENTS IN THE PILLARS

Hole No.	Depth of hole (in.)	Strain measurements (θin/in)			Principal stresses (psi)		Angle from vertical to the direction of the maximum principal stress (deg.)
		e_V	e_{45}	e_H	Maximum	Minimum	
9W9–1C	22	534	1045	490	8400	1470	44
9W9–1C	48	542	1078	500	8760	2160	42
9W9–1C	60	350	890	423	7050	400	47
9W9–1C	72	485	940	335	7400	520	41
9W9–1C	96	322	819	392	6450	520	47
			Mean		7610	1014	44
9W9–2C	27	516	1015	508	8200	1710	45
9W9–2C	39	480	990	480	7950	1360	45
9W9–2C	51	490	1070	392	8350	200	43
9W9–2C	64	466	1052	503	8350	1030	46
			Mean		8210	1075	45
9W9–2L	23	725	44	334	8500	1750	33
9W9–2L	36	600	40	282	7050	1400	33
9W9–2L	48	572	22	167	6300	1000	30
9W9–2L	60	458	97	494	7050	2170	45
9W9–2L	72	528	66	493	7800	2070	44
9W9–2L	84	545	158	533	7700	2760	45
			Mean		7400	1858	38
9W9–3L	24	1650	308	786	18300	5300	32
9W9–3L	36	870	485	750	10000	5700	40
9W9–3L	48	1130	352	800	13500	5250	36
			Mean		13930	5400	36
9W9–1R	24	685	1350	595	10800	1530	43
9W9–1R	47	595	1195	440	9400	600	41
9W9–1R	61	505	1225	440	9500	−275	44
9W9–1R	80	585	1510	575	11600	−387	45
9W9–1R	96	510	1300	520	9750	242	46
			Mean		10210	342	44
9W9–2R	27	800	1495	585	12000	1470	41
9W9–2R	48	710	1325	540	10600	1500	41
9W9–2R	72	560	1370	705	10900	1340	48
9W9–2R	83	635	1185	485	9500	1350	41
			Mean		10750	1415	43
9W9–3R	40	425	1375	880	11100	1420	54
9W9–3R	52	580	1375	635	10900	920	46
9W9–3R	69*	1070	2140	880	17000	1890	43
			Mean		13000	1890	47

TABLE 2 (*contd.*)

Hole No.	Depth of hole (in.)	Strain measurements (μin/in)			Principal stresses (psi)		Angle from vertical to the direction of the maximum principal stress (deg.)
		e_V	e_{45}	e_H	Maximum	Minimum	
11W9-2C	27	660	1440	545	11700	856	43
11W9-2C	36	700	1275	530	10703	2116	41
11W9-2C	50	1040	1105	170	10619	1993	24
11W9-2C	60	650	1120	355	9205	1266	39
11W9-2C	69	650	1246	495	9960	1500	44
			Mean		10437	1546	42
11W9-3C	40	670	1096	492	9400	2360	40
11W9-3C	48	525	1025	485	8700	1610	44
11W9-3C	72	715	1270	520	10800	1800	40
11W9-3C	80	600	1145	490	9600	1470	45
11W9-3C	90	725	1180	410	10030	1480	38
			Mean		9710	1744	41

* This reading obtained only 7 in. from the sidewall of the pillars.

(b) *Pillar stresses, Test Site* 2

The results of the strain cell and deformation meter measurements are shown in Tables 2 and 3 respectively. Since the three readings u_1, u_2 and u_3 required to complete a deformation meter measurement were obtained over a distance of approximately 3 ft, fewer measurements could be obtained in the narrow pillars with this method. Average values of u_1, u_2 and u_3 were, therefore, determined to calculate the stresses in the pillars.

Table 2 shows that the strain cell results are fairly consistent over the center section of the pillar in which measurements were obtained. In view of this and the fact that few deformation meter results were obtained, it was considered reasonable to compare average values of the stresses and their directions in the pillars. The results are compared in Table 4.

Both methods showed that the major principal stress acts in an up-dip direction approximately 45° from the vertical direction. The magnitudes of the major principal stresses as obtained from deformation meter measurements were again slightly higher than those obtained from strain cell measurements. The difference in the magnitudes of the minor principal stresses obtained with the two methods is much greater. In some cases the deformation meter values are three times as high as the strain cell values.

It is believed that this difference is due to errors which were introduced in the u_3 deformation meter measurements taken in a direction 120° from the vertical direction. As a check the stress values obtained from the strain cell measurements were used to calculate the deformations of an EX borehole in the three directions in which deformation meter measurements were taken. The calculated values of u_1, u_2 and u_3 and those measured by the deformation meter are given in Table 5. These clearly indicate that the agreement between the calculated and measured u_3's are generally poor. On the other hand the agreement between the calculated and measured values of all u_1's and u_2's is satisfactory with only one exception, namely, in borehole 9W9-2L. Thus it can be concluded that, had the u_3 measurements been more satisfactory, the stresses obtained by the deformation meter would have conformed even more closely with those given by the strain cells.

TABLE 3. RESULTS OBTAINED FROM DEFORMATION METER MEASUREMENTS IN THE PILLARS

Hole No.	Depth of hole (in.)	Deformation measurements (10^{-4} in.) u_1	u_2	u_3
9W9–1C	37	28	—	—
9W9–1C	63	—	—	4·8
9W9–1C	80	—	12	—
Mean		28	12	4·8
9W9–2C		Core disced all the way		
9W9–2L	21	—	21	—
9W9–2L	31	32	—	—
9W9–2L	44	—	—	2·8
9W9–2L	55	—	14	—
9W9–2L	77	32	—	—
9W9–2L	89	—	—	18·2
Mean		32	17·5	10·5
9W9–3L	22	—	2·9	—
9W9–3L	34	48	—	—
9W9–3L	46	—	—	29
9W9–3L	68	—	33	—
Mean		48	31	29
9W9–1R	72	—	18·0	—
		Core disced most of the time		
9W9–2R	33	—	8·0	—
9W9–2R	45	36·63	—	—
9W9–2R	58	—	—	0
9W9–2R	80	—	19·0	—
Mean		36·63	13·5	0
9W9–3R	42	—	12	—
9W9–3R	53	48	—	—
9W9–3R	66	—	—	3·6
9W9–3R	76	—	19·2	—
Mean		48	15·6	3·6
11W9–2C	30	—	23·3	—
11W9–2C	46	32·4	—	—
11W9–2C	59	—	—	−0·7
11W9–2C	72	—	—	−1·4
11W9–2C	80	—	—	0
11W9–2C	92	44·0	—	—
11W9–2C	102	—	14·5	—
Mean		38·2	18·9	−0·7
11W9–3C	24	—	20	—
11W9–3C	41	33	—	—
11W9–3C	54	—	—	0
11W9–3C	65	—	20	—
Mean		33	20	0

TABLE 4. COMPARISON OF THE MAGNITUDES AND DIRECTIONS OF THE PRINCIPAL STRESSES AS DETERMINED WITH EACH OF THE METHODS IN THE PILLARS

Borehole No.	Type of measuring instrument	Principal stresses (psi) Maximum	Minimum	Angle from vertical to direction of maximum principal stress (deg.)
9W9–1C	Strain Cell	7610	1014	44
9W9–1C	Deformation Meter	8680	3220	53
9W9–2C	Strain Cell	8210	1075	45
9W9–2C	Deformation Meter	—	—	—
9W9–2L	Strain Cell	7400	1858	38
9W9–2L	Deformation Meter	10475	5425	50
9W9–3L	Strain Cell	13930	5400	36
9W9–3L	Deformation Meter	16800	12000	57
9W9–1R	Strain Cell	10210	342	44
9W9–1R	Deformation Meter	—	—	—
9W9–2R	Strain Cell	10750	1415	43
9W9–2R	Deformation Meter	11640	3220	44
9W9–3R	Strain Cell	13000	1410	47
9W9–3R	Deformation Meter	14000	3600	52
11W9–2C	Strain Cell	10437	1546	42
11W9–2C	Deformation Meter	11975	3025	45
11W9–3C	Strain Cell	9710	1744	41
11W9–3C	Deformation Meter	11500	2600	42

TABLE 5. COMPARISON OF THE CHANGES IN DIAMETER OF AN EX BOREHOLE AS MEASURED WITH THE DEFORMATION METER AND AS CALCULATED FROM MEASUREMENTS MADE WITH THE STRAIN CELL

Borehole No.	Principal stresses determined from strain cell measurements (psi)		Borehole deformations calculated from strain cell measurements (10^{-4} in.)			Borehole deformations measured with the deformation meter (10^{-4} in.)		
	Maximum	Minimum	u_1	u_2	u_3	u_1	u_2	u_3
9W9–1C	7610	1014	26·0	11·8	−3·9	28	12	4·8
9W9–2C	8210	1075	30·0	12·0	−6·5	—	—	—
9W9–2L	7400	1858	23·0	15·7	−2·3	32	17·5	10·5
9W9–3L	13930	5400	40·7	30·4	3·8	48	31	29·0
9W9–1R	10210	342	35·6	14·6	−8·9	—	18·0	—
9W9–2R	10750	1415	35·8	17·4	−5·9	36·6	13·5	0
9W9–3R	13000	1410	45	16·5	−6·2	48	15·6	3·6
11W9–2C	10437	1546	38·0	18·1	−5·6	38·0	18·9	−0·7
11W9–3C	9710	1744	31·0	18·0	−4·35	33	20	0

(c) *Rock properties*

The uniaxial compression tests performed on BX core samples which were obtained from overcoring tests with the CSIR Strain Cell produced values of the modulus of elasticity and Poisson's ratio of the rock with a standard deviation of less than 5 per cent. The following mean values were obtained:

Modulus of elasticity $11{\cdot}5 \times 10^6$ psi
Poisson's ratio $0{\cdot}2$.

All the results, except the values of Poisson's ratio, were obtained from compression tests on specimens prepared from the 6 in. overcoring cores [8]. Although the same mean values were obtained, the standard deviation was found to be 15 per cent for the modulus of elasticity and 25 per cent for Poisson's ratio.

When analysing strain or borehole deformation measurements, obtained from overcoring tests, the results are usually more accurate if the elastic constants (E and ν) obtained at each measuring station are used for the calculation of the stresses at that point. However, since the constants could not be determined at every overcoring station, it was considered reasonable to use the mean values given above for all calculations in this paper.

6. CONCLUSIONS

(1) Both instruments show the same general variation of σ_1 and σ_2 with distance from the sidewall of the 14 level extension drift. At distances of between 20 and 25 ft from the sidewall the difference between the results is large. However, considering the accuracy with which the elastic constants of the rock could be determined, the inherent accuracy of the instruments and other errors, the results obtained with the two methods are generally in good agreement.

(2) With the exception of the u_3 measurements all other measurements (u_1 and u_2 measurements) obtained with the deformation meter in the pillars are in good agreement with measurements obtained with the strain cell. The directions of the principal stresses obtained at both sites using the two instruments are in good agreement.

(3) The results of these experiments are a good indication of the reliability of both instruments for determining the stresses in unfractured hard rock.

(4) The deformation meter has the advantage that it can be used under wet conditions. It has the disadvantage that it is not rigidly fixed in the hole during the overcoring operation and it can, therefore, move while being overcored.

(5) The strain cell has the advantage that more measurements can be obtained with it within a given distance. Although it has the disadvantage at the present time, that it can only be used on dry rock, chemicals are now available for improving the bond of cements to wet surfaces which may solve this problem.

Acknowledgements—This work is part of an extensive program of stress measurements being conducted in Canada under the supervision of Dr. D. F. Coates. The assistance of J. St. Onge, W. Zawadski, A. V. St. Louis, H. Montone, E. Timms and P. Pike of the Mining Research Laboratories at Elliot Lake as well as the assistance of Dr. W. M. Gray in editing the paper are acknowledged.

The cooperation and assistance of the mining officials at the Nordic Mine of Rio Algom Mines Limited was much appreciated.

The permission of the Canadian Department of Mines and Technical Surveys and the South African Council for Scientific and Industrial Research to publish this paper is gratefully acknowledged.

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