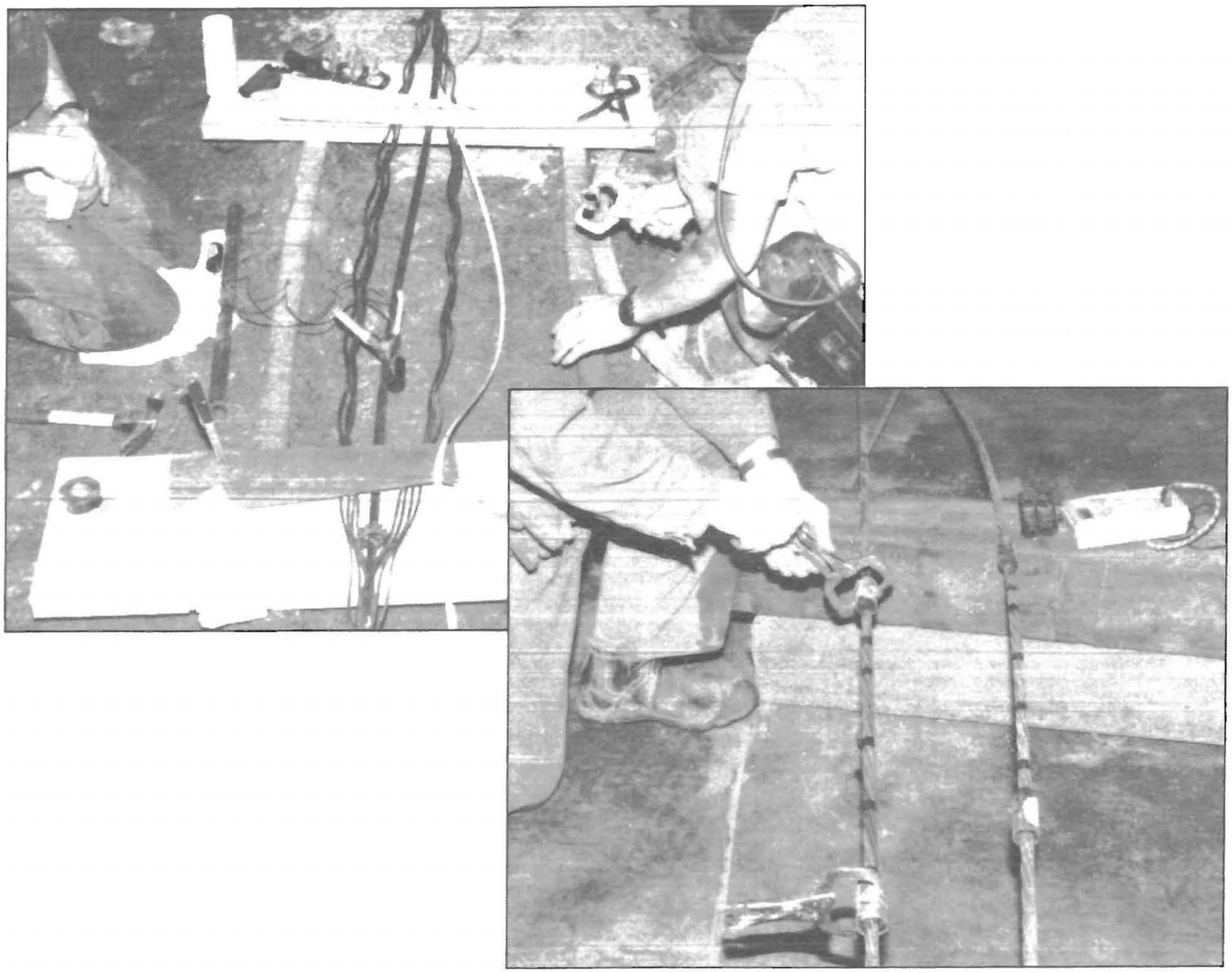
RI 9474

REPORT OF INVESTIGATIONS/1993

Field Evaluation of Cable Bolt Supports, Homestake Mine, Lead, SD

By J. M. Goris, T. M. Brady, and L. A. Martin



United States Department of the Interior

Bureau of Mines

U.S. Department of the Interior
Mission Statement

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally-owned public lands and natural resources. This includes fostering sound use of our land and water resources; protecting our fish, wildlife, and biological diversity; preserving the environmental and cultural values of our national parks and historical places; and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to ensure that their development is in the best interests of all our people by encouraging stewardship and citizen participation in their care. The Department also has a major responsibility for American Indian reservation communities and for people who live in island territories under U.S. administration.

Cover: Installation of strain gages on cable bolt supports. These strain gages were used to monitor the behavior of rock masses supported with cable bolts during mining.

Report of Investigations 9474

Field Evaluation of Cable Bolt Supports, Homestake Mine, Lead, SD

By J. M. Goris, T. M. Brady, and L. A. Martin

UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary

BUREAU OF MINES

Library of Congress Cataloging in Publication Data:

Goris, J. M. (John M.), 1938-

Field evaluation of cable bolt supports, Homestake Mine, Lead, SD / by J.M. Goris, T.M. Brady, and L.A. Martin.

p. cm. — (Report of investigations; 9474)

Includes bibliographical references (p. 28).

1. Mine roof bolting—Testing. 2. Cables—Testing. 3. Rock mechanics—Mathematical models. 4. Homestake Mine (S.D.). I. Brady, T. M. II. Martin, L. A. (Lewis A.). III. Title. IV. Series: Report of investigations (United States. Bureau of Mines); 9474.

TN23.U43 [TN289.3] 622 s—dc20 [622'.28] 93-17727 CIP

CONTENTS

ILLUSTRATIONS

ILLUSTRATIONS—Continued

TABLES

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

deg	degree	lb/ft^3	pound per cubic foot
ft	foot	Ω	ohm
h	hour	pct	percent
in	inch	psi	pound per square inch
in^2	square inch	st	short ton
in/in	inch per inch	ϵ	strain, in/in
lb	pound	ϵ/lbf	strain per pound (force)
lbf	pound (force)	yd^3	cubic yard
lb/ft^2	pound per square foot		

FIELD EVALUATION OF CABLE BOLT SUPPORTS, HOMESTAKE MINE, LEAD, SD

By J. M. Goris,[1] T. M. Brady,[2] and L. A. Martin[3]

ABSTRACT

The U.S. Bureau of Mines, in a cooperative project with the University of Utah, Salt Lake City, UT, and Homestake Mining Co., Lead, SD, conducted in situ monitoring and numerical modeling of rock masses supported with cable bolts in two mechanized, cut-and-fill stopes and a shaft pillar at the Homestake Mine.

Extensometers were used to measure rock displacement, while cable bolt strain gauges were installed to measure loads on both conventional and birdcage cable bolt supports. These instruments provided an assessment of rock mass behavior during mining as well as essential data for verifying results from computer analyses. The numerical modeling program FLAC was used to analyze cable bolt patterns and supports.

Results from the evaluation study indicated that cable bolt strain gauges and extensometers were effective instruments for monitoring the behavior of rock masses supported with cable bolts. Also, numerical modeling of the 45-48N stope using the FLAC code provided displacement values comparable with measurements from field extensometers.

[1]Mining engineer.
[2]General engineer.
[3]Mechanical engineer.
Spokane Research Center, U.S. Bureau of Mines, Spokane, WA.

INTRODUCTION

The U.S. Bureau of Mines, in cooperation with the Homestake Mining Co., Lead, SD, and the University of Utah, Salt Lake City, UT, conducted a field demonstration to evaluate the behavior of cable bolt supports in three separate areas of the Homestake Mine. The first area was supported with both conventional and birdcage cable bolts (site 1, figure 1). This site was located between the 6500 and 6650 levels of the 45-48N stope, 21 ledge. The second area was the hanging wall of the D-limb stope between the 3500 and 3650 levels near the Ross shaft (site 2, figure 1) and was supported with conventional cable bolts. The third area was the Ross shaft pillar at the 3350 level (site 3, figure 1). The primary objectives of the project were to (1) monitor the behavior of the rock supports in these three areas with extensometers and cable bolt strain gauges and (2) model the rock masses using the Fast Lagrangian Analysis Continuum (FLAC) numerical code for site 1 and the UTAH II and UTAH III codes for sites 2 and 3. Field data collected from the extensometers and cable bolt strain gauges were used to verify the models. In addition, conventional and birdcage cable bolt supports were compared to evaluate their relative effectiveness. This work was conducted as part of the Bureau's program to improve mineral resource recovery.

ACKNOWLEDGMENTS

Special thanks go to Mike King and Bill Hand, electronic technicians; Doug Scott, geologist; and Gene Stone and Mike Jones, engineering technicians, all of the Bureau's Spokane Research Center, who were instrumental in the development of the data collection system and installation of the instruments. Special acknowledgment is also given to Steve Orr, Greg Struble, Jerry Pfarr, and Brent Lamore, mining engineers at the Homestake Mine, for their assistance in locating test sites, and to William G. Pariseau, professor of mining engineering, and Fei Duan, graduate student, of the University of Utah, for their assistance in installing instruments and providing technical interpretation of the data.

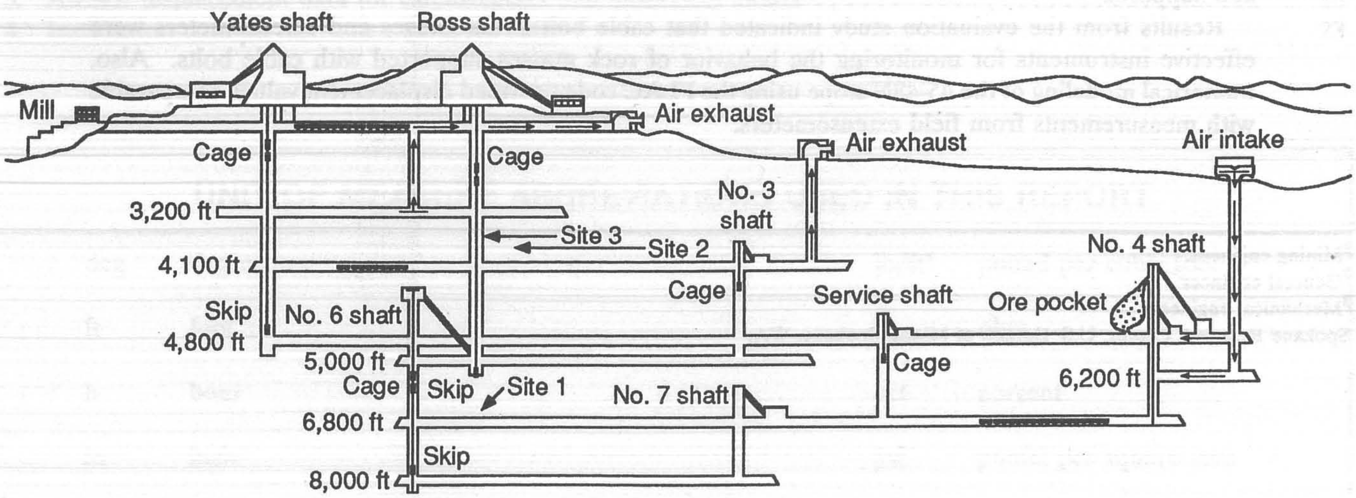


Figure 1.—Generalized cross section of Homestake Mine.

GEOLOGY OF HOMESTAKE MINE

The Homestake Mine is located in the Black Hills of South Dakota in the town of Lead. The three major formations in the Lead area, the Poorman, the Ellison, and the Homestake, are all Precambrian in age. The Poorman Formation is the oldest and contains ankeritic phyllite and ankerite mica schist interbedded with chert, graphite, and disseminated pyrite.

The most famous formation in the Black Hills is the Homestake Formation, which is a generally competent cummingtonite and sideroplesite schist containing various amounts of chlorite, pyrrhotite, arsenopyrite, ankerite, and graphite. Gold is found only in the Homestake Formation, generally disseminated in arsenopyrite crystals. Minor amounts of fine gold are disseminated in chlorite and pyrrhotite [Rye and Rye, 1974 (*1*); Noble, 1950 (*2*)].[4]

The Ellison Formation is metamorphosed quartzite overlying the Homestake Formation. The Ellison is composed of phyllite, dark quartzite, micaceous quartzite, minor amounts of chert, and calcareous mica schist. This formation can range up to 3,000 ft thick where it has been folded [Dewitt, Redden, Wilson, Buscher, and Dersch, 1986 (*3*)]. It is younger than the Homestake Formation and tends to have weak planes resulting from grain orientation parallel to foliation and bedding.

45-48N STOPE

The 45-48N stope is located in the 21 ledge between the 6500 and 6650 levels and contains about 233,000 st of minable ore. Bureau personnel mapped the extent of the ore body from a drift on the 6500 level and from the 45-48N stope when it was at the 6590 level. In addition, rock core samples were obtained from boreholes drilled from the drift on the 6500 level down to the stope below.

The ore body lies within the Homestake Formation. It is very irregular in shape and is approximately 200 ft long by 45 ft wide. Between the 6500 and 6650 levels, it dips from 35° to 60° east with a near-vertical plunge to the south. The contact between the Homestake and the overlying Ellison Formation creates planes of weakness along localized graphite layers, and these can cause ground control problems during mining. Figure 2 shows a typical cross section of the 45-48N stope area.

[4] Italic numbers in parentheses refer to items in the list of references at the end of this report.

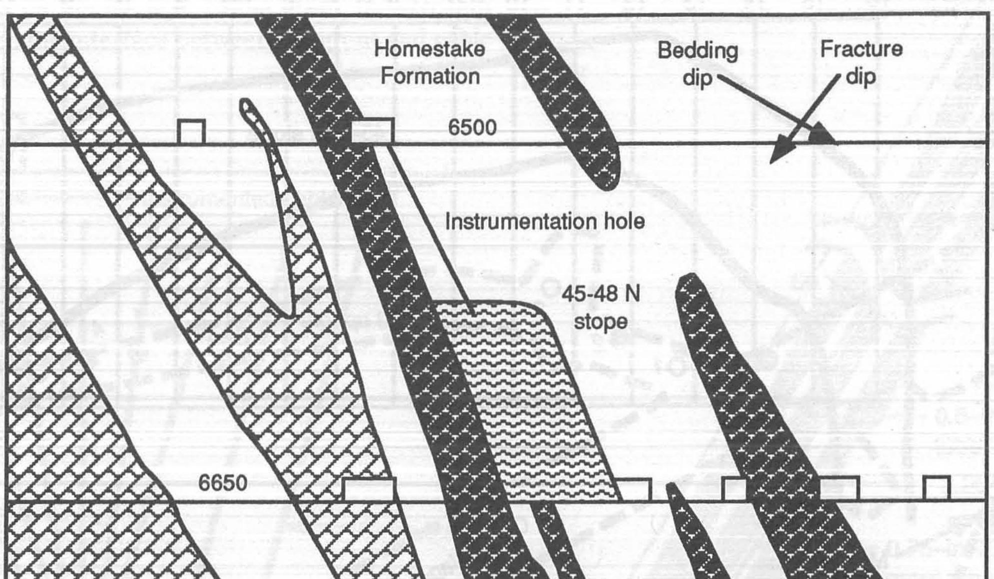


Figure 2.—Cross section of geology surrounding drift on 6500 level and stope on 6580 level.

A lack of support can lead to weakening and raveling of the walls and back. When the dip is below 65°, the rock tends to slide along the graphite layers and cause slabbing in the back. Many of the faults and shear zones have been filled with quartz, which is more rigid than the surrounding rock. However, problems arise when induced stresses from previous mining cause initial raveling failures followed by long-term instability.

In addition to geologic conditions, the geometry of the stope and the sequence of mining have a great effect on the stability of the walls and back. Stress concentrations from earlier mining can be sufficient to cause slabbing of the back. A previously mined vertical crater retreat stope just west of the 45-48N stope was responsible for a great deal of induced stress in the walls and back of the 45-48N stope and complicated ground control efforts [Schmuck, 1979 (*4*); Pharr, 1991 (*5*); Goris, Duan, and Pfarr, 1991 (*6*)].

MINING AND GROUND CONTROL

The 45-48N stope was mined using a mechanized cut-and-fill method in which pneumatic rubber-tired, two-boom jumbos and 2- and 3-1/2-yd^3 load-haul-dump units (LHD's) were the primary pieces of equipment. Ore areas were accessed from a main ramp and breast mined to allow a continuous flow of rock each day. Each stope had several areas or headings, allowing for much flexibility and a high degree of productivity.

Ground control for this stope consisted of double 60-ft-long conventional and birdcage cable bolt supports as well as 5-ft-long Split Set rock bolts.[5] Stations for the cable support patterns began at the south end of the stope at station 0 and continued north to station 16 (fig. 3).

[5]Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

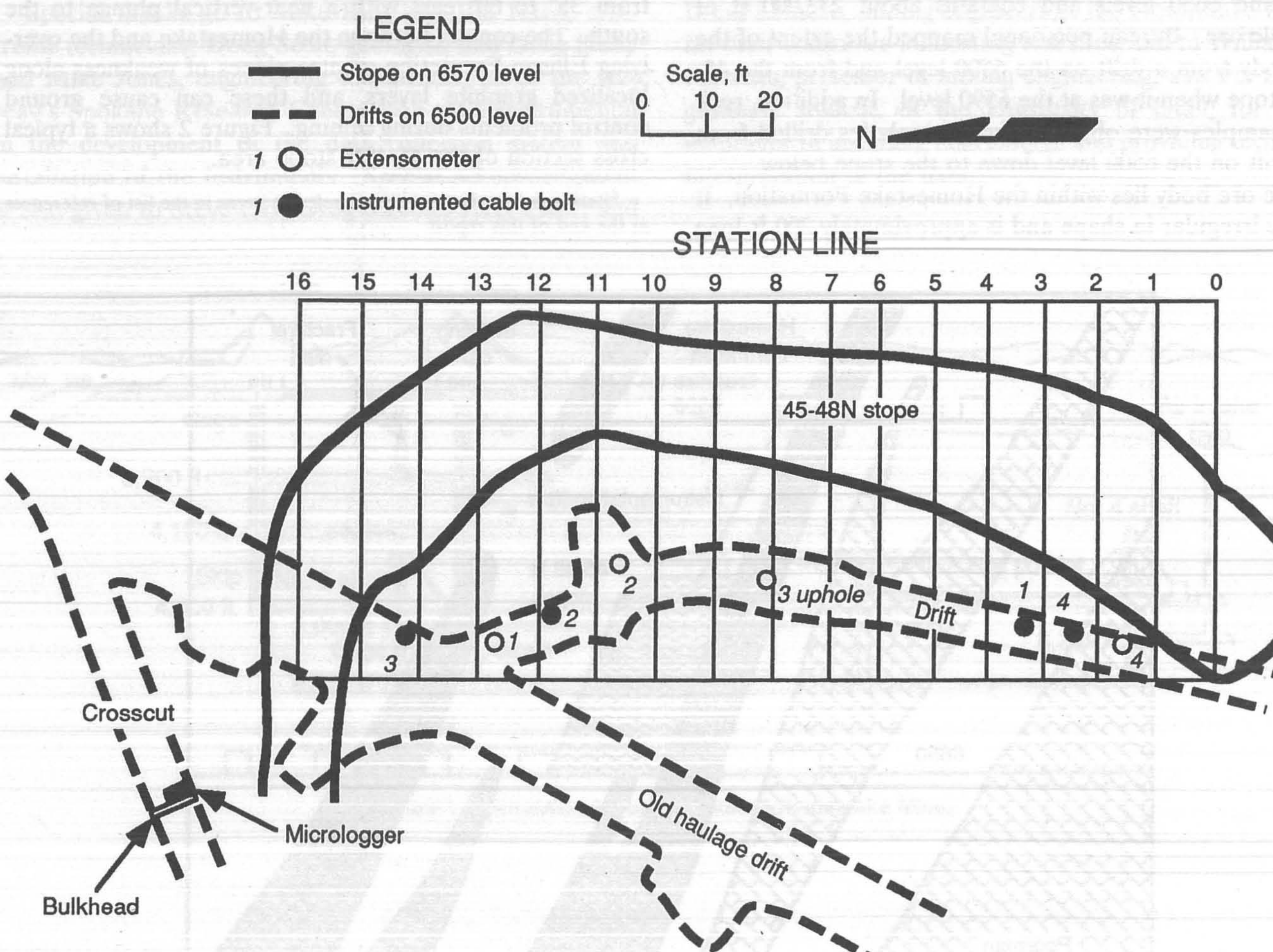


Figure 3.—Plan view of cable bolt supports from stope on 6580 level.

Conventional cable bolts were installed on 10-ft centers between stations 0 and 8. Birdcage cables were placed on 11-ft centers between stations 9 and 16. A typical cross section of the stope showing the fan-shaped configuration of the cable supports is shown in figure 4.

CABLE BOLT SUPPORTS

Cable bolt supports were first introduced to the mining industry around 1970 as a means of reinforcing ground prior to mining. These supports consist of one or more steel cables grouted into a drill hole in the rock (see figure 5). These supports vary in length, but 60-ft lengths or greater are common. There are two kinds of cables used for these supports, conventional and birdcage.

Conventional Cables

Conventional cables are made from high-strength steel that has an ultimate strength of approximately 58,000 lbf and a modulus of elasticity of approximately 29.5×10^6 psi. They are usually 0.6 to 0.625 in. in diameter and consist of seven wires. Figure 6 shows a conventional cable on the left.

For conventional cable bolt supports to be effective, it is necessary that the load from the rock be transferred to the cable through the grout. Therefore, the capacity of the system depends on the strength of the grout, the strength of the cable, and the interface between the grout and cable and between the rock and grout. Failure may occur in one or more of the following modes:

1. Failure at the grout-cable interface,
2. Failure at the rock-grout interface,
3. Failure of the cable, or
4. Failure of the rock around the cable bolt.

Of these, failure at the grout-cable interface appears to be the most common [Goris, 1990a (*7*)].

One primary advantage of cable bolts is that a long bolt can be placed in the rock even in confined areas because the cable is flexible. For example, in a drift with a 10-ft-high roof, the length of a rock bolt is restricted to 10 ft or less; however, because cables will bend, much longer supports can be placed in this same opening. As mentioned before, 60-ft-long cable bolts are common.

Cable bolts work well in fractured ground because the entire length of the cable is bonded to the rock with grout. Also, cable bolt support patterns can be designed to respond to various types of ground movement. In highly stressed rock, a single cable in each hole allows a large degree of rock deformation to occur, thereby redistributing the load to the pillars. Double cables, on the

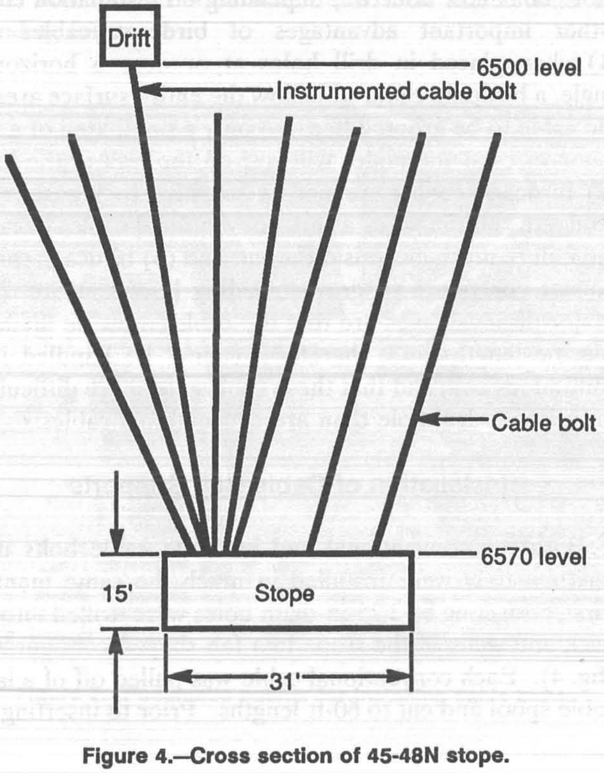


Figure 4.—Cross section of 45-48N stope.

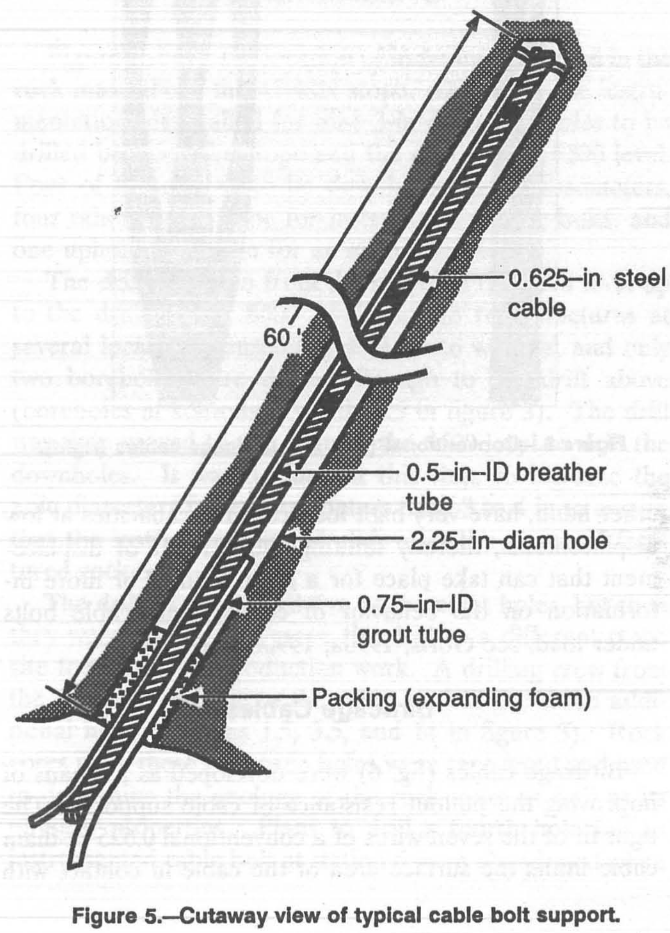


Figure 5.—Cutaway view of typical cable bolt support.

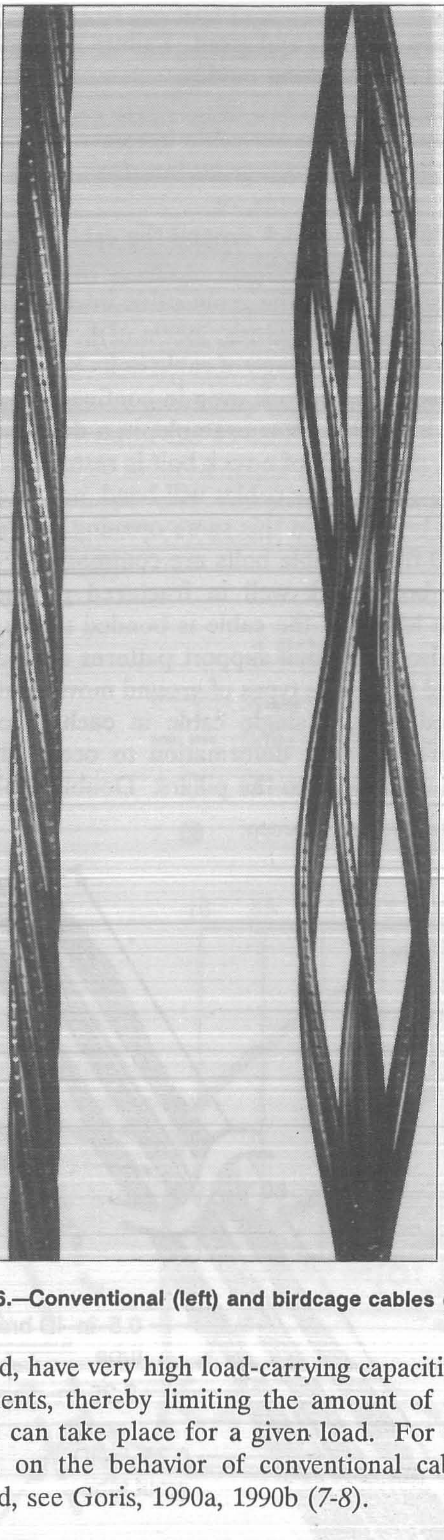


Figure 6.—Conventional (left) and birdcage cables (right).

other hand, have very high load-carrying capacities at low displacements, thereby limiting the amount of displacement that can take place for a given load. For more information on the behavior of conventional cable bolts under load, see Goris, 1990a, 1990b (*7-8*).

Birdcage Cables

Birdcage cables (fig. 6) were developed as a means of improving the pullout resistance of cable supports. The tight fit of the seven wires of a conventional 0.625-in-diam cable limits the surface area of the cable in contact with the grout to approximately 2.62 in^2 per linear inch of cable. It is at this grout-cable interface that the resistance to pullout is developed. A manufacturer in Australia designed a technique for separating the seven wires of a conventional cable and then recombining them to form an open cable with a series of nodes and antinodes spaced at about 7-in intervals along the cable (fig. 6). This process is called birdcaging and can be started or ended at any point along the length of the cable. When grout is placed around the birdcage cable, the wires of the cable tend to form reinforced nodes that behave as anchors along the length of cable.

Researchers at the Bureau conducted laboratory pull tests on both single and double birdcage cables to determine their load-displacement behavior when compared to single conventional cables [Goris, 1990a, 1990b (*7-8*)]. Basically, these tests showed that both single and double birdcage cables have higher load-carrying capacities than conventional cables. Tests conducted on single birdcage cables showed that the maximum load-carrying capacity is between 36 and 79 pct greater than the load-carrying capacity of conventional cables, depending on the location of the node with respect to a failure plane. Pull tests conducted on double birdcage cables showed an increase of 88.5 to 94.5 pct over conventional double cables.

The cost for a birdcage cable is approximately 35 pct greater than the cost of a conventional cable; however, the cost of the cable alone represents only about 10 to 15 pct of the total cost of a cable bolt support. Therefore, the use of birdcage cables would increase total costs of cable bolt supports about 3.5 to 5.2 pct. This makes birdcage cable bolts cost effective, depending on installation costs. Other important advantages of birdcage cables are (1) when placed in drill holes at or near a horizontal angle, a birdcage cable will allow the entire surface area of the cable to be grouted because only a small area of a few outer wires at the nodes will rest on the wall of the hole; (2) birdcage cables are not as sensitive to grease, rust, mud, etc., on the wires as are conventional cables because the failure mechanism is different; and (3) birdcage cables do not contribute to grout bleeding [Goris, 1990b (*8*)]. Major disadvantages are that the cables must be made to specific lengths and, therefore, cannot be handled in a continuous coil, and that these cables are more difficult to push into a drill hole than are conventional cables.

Installation of Cable Bolt Supports

Both the conventional and birdcage cable bolts used for this study were installed in much the same manner. First, 60-ft-long by 2.25-in-diam holes were drilled into the back and walls of the stope in a fan-shaped configuration (fig. 4). Each conventional cable was pulled off of a large cable spool and cut to 60-ft lengths. Prior to inserting the first of the two cables, a 4-in-long piece of one of the outer six wires on the cable was bent back at about a 130° angle to form a hook. This helped to keep the cable in the hole until grout could be placed. The cable was then inserted into the hole using a pneumatic pusher (fig. 7). A 0.5-in-diam plastic breather tube was attached to the second cable and again one of the outer six wires on the cable was bent back to form a hook. The second cable and the breather tube were then inserted into the hole. A 15-ft by 0.75-in-diam plastic grout tube was then inserted about 3 ft into the hole; the remaining portion of the tube was later connected to the grout pump.

Once the cables and tubes were in place, water was sent through the breather tube to flush the hole. The bottom 12 in of the hole was then plugged by placing expanding foam into it and allowing the foam to harden. This process sealed the hole and prevented grout from running out.

The birdcage cables came from the manufacturer as double cables precut to 60-ft lengths and coiled for ease of handling. In the mine, the coils were released by one crew member standing in the center of the coil and cutting the wire ties that bound the coiled cable. These cables were also installed with a pneumatic pusher, but because of the nodes and antinodes, the cables were very difficult to push into the hole. The 2.25-in-diam holes were too small for double birdcage. A 2.75-in-diam hole would have helped to reduce cable installation time. The rollers on the pusher had to be replaced several times. As with conventional cables, a breather tube was attached to the cable (birdcage) prior to being installed. After the cable, breather tube, and a short length of grout tube were in place, the hole was flushed with water and then sealed with expanding foam.

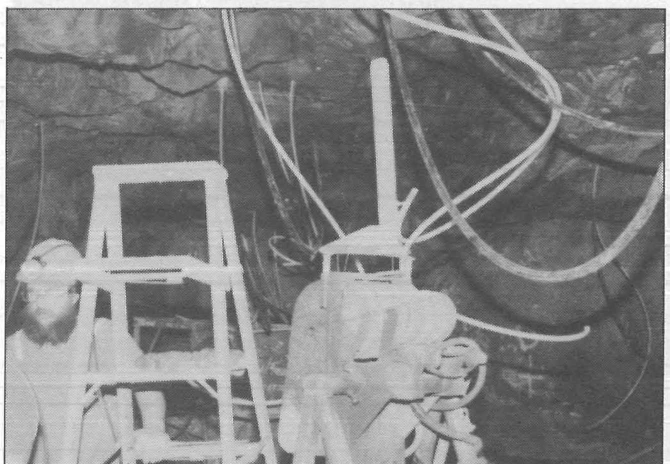


Figure 7.—Pneumatic pusher for installing cable bolts.

The grout for the cable bolts consisted of Type I,II portland cement and water mixed at a ratio of 0.45 parts water to 1.0 part cement by weight. The grout was mixed in a shear-type mixer and pumped with a pneumatic Moyno pump. Grout was pumped into the holes until it began running out of the breather tube. Then the bottom 12 in of both the grout and breather tubes were folded over and tied off to prevent the grout from draining out of the hole. A two-person cable bolt crew can install and grout about 500 ft of cable per shift.

Four cable bolts equipped with strain gauges were installed from the drift on the 6500 level down into the rock mass above the 45-48N stope. This allowed the electrical leads from the gauges to extend out of the top of the hole and to be connected to the data collection system. The holes for these cables were 3 in. in diameter. The bottom of the drill holes were first plugged using wooden plugs and expanding foam. A 0.75-in-diam grout tube was then attached to the bottom end of each instrumented cable, and both the cable and tube were lowered into the hole by hand. Since these were downholes, a breather tube was not required. The grout was then mixed in a shear mixer and pumped into the hole. As the grout filled the hole, the grout tube was slowly removed, allowing the hole to be filled without voids.

INSTRUMENTS

Figure 3 shows the location of instruments placed in the rock mass above the 45-48N stope. Originally the instrumentation plan called for nine 3-in-diam boreholes to be drilled between the stope and the drift on the 6500 level. Four of these were to be downholes for extensometers, four others were to be for instrumented cable bolts, and one uphole was to be for an extensometer.

The drilling began from the stope on the 6570 level up to the drift on the 6500 level. Large rock fractures at several locations caused the drill bit to wander, and only two boreholes were drilled through to the drift above (boreholes at stations 2.5 and 10.5 in figure 3). The drill was then moved to the drift on the 6500 level to drill the downholes. It was decided at this time to increase the hole diameters for extensometers from 3 to 4 in to ensure that the instruments would slide past the zones of fractured rock.

The drillers completed five of the eight holes, but then they were required to move the drill to a different stope site for scheduled production work. A drilling crew from the Bureau was sent to the mine to drill the three additional holes (stations 1.5, 3.5, and 14 in figure 3). Rock cores from these last three holes were recovered and used to determine the geology of the rock mass as well as its physical properties. Plans to drill a fourth hole for an instrumented cable bolt at station 6 were abandoned when the walls of the drift on the 6500 level between stations 4 and 9 began to move and rock began to spall, making the area unsafe to work in. A safety engineer from the mine inspected the drift and decided that it could not be used for access. Consequently, the downhole was repositioned from station 6 to station 2. Table 1 provides information on the instrumentation holes and shows the type of support used in the vicinity of each instrument.

Extensometers

Extensometers were installed in the rock mass to monitor movement. Each extensometer had either five or six groutable rebar anchors, depending on the depth of the hole. Each anchor was connected to 0.25-in-diam continuous fiberglass rods inserted into 0.5-in-diam nylon tubing to protect the rods from the grout. The rods were then connected to linear potentiometers in the drift on the 6500 level at the collar of the hole (fig. 8). Each potentiometer had a range of 4 in and was housed in a protective head assembly. Once the extensometer was inserted into the borehole, the hole was filled with a cement grout. Figure 9 shows the head of extensometer 3 at station 8 (uphole). Movements of the anchors were detected by the potentiometers and were read electronically using a data collection system.

Cable Bolt Strain Gauges

Cable support strain gauges function on the principle that changes in electrical resistance are caused by stretching of electrical wires, and thus by measuring the change in resistance, investigators can determine the amount of load being placed on selected cables.

The gauges used in this field test were 25 in long with a thin, nickel-chromium wire (0.01 in. in diameter) wound into spiral grooves between the outer strands of the cable. The thin wire was protected by a plastic tube and was terminated at both ends by a molded rubber anchor bonded to the cable by quick-setting epoxy (fig. 10). Each anchor was then protected by a plastic sleeve to prevent grout from restricting movement when the cable was loaded [Choquet and Miller, 1987 (*9*)].

Although readouts from the gauges can be either in volts or strain, it was decided to generate the measurements in strain. Figure 11 shows a cable bolt strain gauge being tested in a Bureau laboratory. Figure 12 shows the strain versus load behavior of two cable bolt strain gauges. Both gauges were attached to a conventional 0.625-in-diam cable. In the first test, the cable was not embedded in grout. In the second test, both the gauge and the cable were embedded in a cement grout column and allowed to cure for 7 days before testing. The tests were conducted by placing the instrumented cable in a test machine (fig. 11), securing the ends of the cable with barrel-and-wedge anchors, and then loading the cable. Continuous readings of strain and load were recorded during the test. Both curves in figure 12 are linear, although the gauge embedded in grout showed a little less strain for a given load than the free-standing gauge. A value of 1.13×10^{-7} ϵ/lbf was obtained from the gauge embedded in grout, and this value was used as input to the

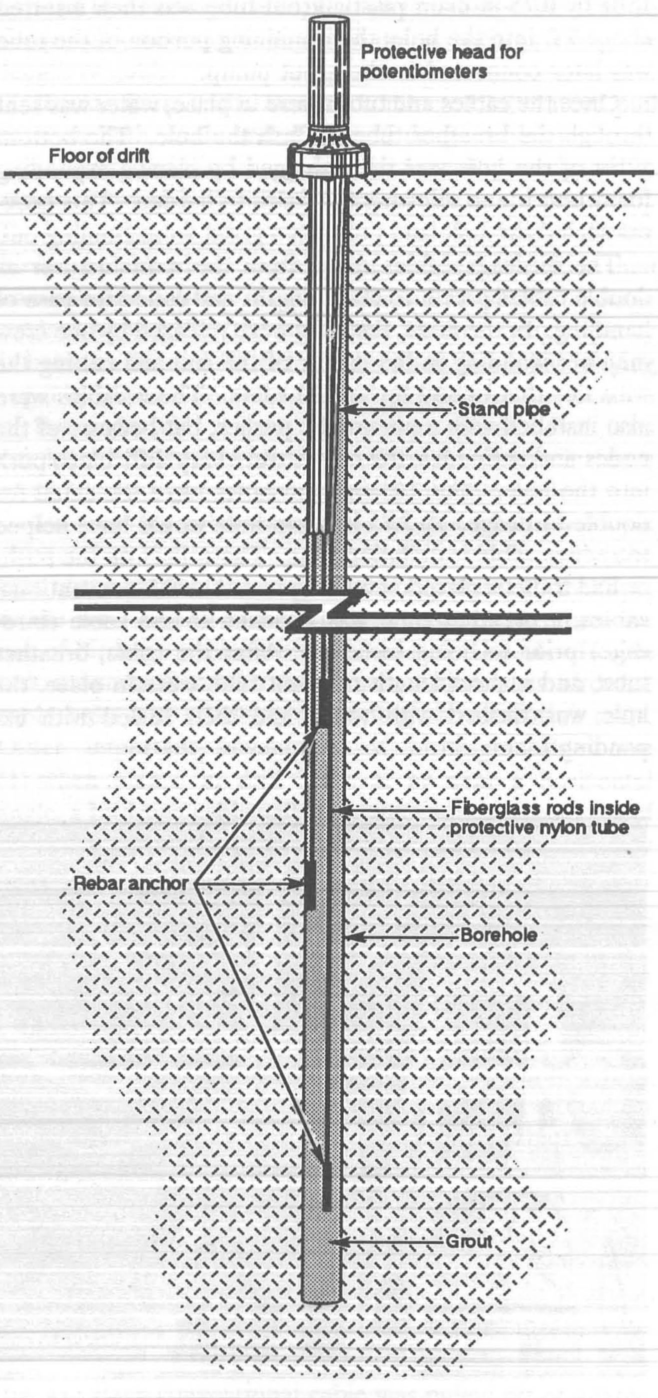


Figure 8.—Profile of extensometer with groutable anchors.

data collection system. Output from the data collection system was in pounds of force.

Long-term creep tests (82 days) were run on the gauges by the gauge manufacturer and the results showed that creep is as much as -0.000425 ϵ (in compression). The load associated with this strain can be approximated by the equation

$$\epsilon = (P/AE)$$

or

$$P = \epsilon AE,$$

where P = load, lbf,

ϵ = strain, in/in,

A = cross-sectional area of the cable, in^2,

and E = Young's modulus of the cable, psi.

Therefore, given a strain of -0.000425 ϵ, a cable area of 0.22 in^2, and a Young's modulus of 29.5×10^6 psi, for example, the load should be $P = -0.000425(0.22)(29.5 \times 10^6) = -2{,}758$ lbf (in compression). This would represent the largest amount of creep based on the manufacturer's data.

Table 1.—Holes and instruments placed in 45-48N stope

Instrument and hole	Station	Hole drilled by	Core recovery	Type of support in area
Extensometer:				
1	12.5	Homestake	No	Birdcage cables.
2	10.5	..do.	No	Do.
3 (uphole)	8.0	..do.	No	Not applicable.
4	1.5	Bureau of Mines	Yes	Conventional cables.
Cable:				
1	3.0	..do.	Yes	Do.
2	12.0	Homestake	No	Birdcage cables.
3	14.0	Bureau of Mines	Yes	Do.
4	2.5	Homestake	No	Conventional cables.

Figure 9.—Head of extensometer at station 8 (uphole).

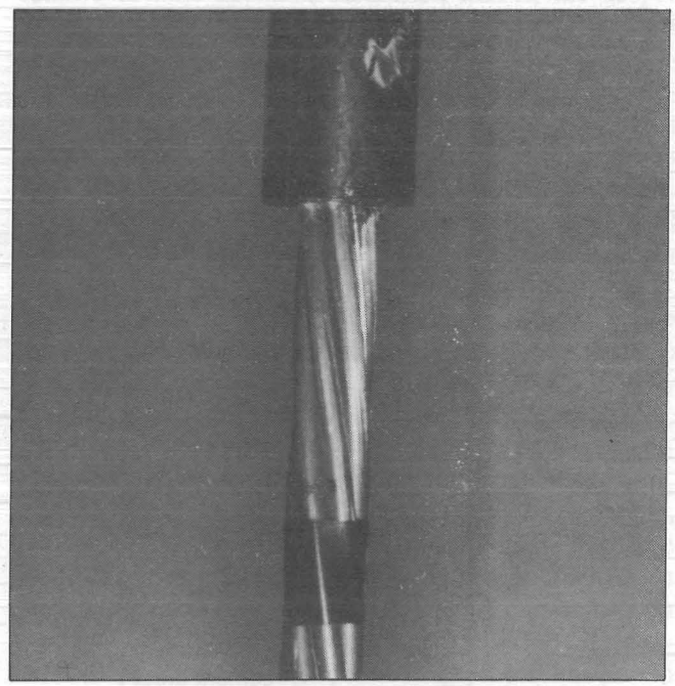


Figure 10.—Closeup of cable bolt strain gauge.

The strain gauges selected were developed for use on conventional 0.625- or 0.6-in-diam cables and could not be placed on birdcage cables. However, project personnel were successful in combining conventional and birdcage cables into a single tendon so that the strain gauges could be used. Various numbers of outer wires from birdcage cables were placed around a conventional cable, after which a strain gauge was attached to the conventional cable (fig. 13). A series of these conventional cable-birdcage cable combinations were made and tested to determine how many outer wires of birdcage cable were required to duplicate pull-test results obtained with a double birdcage cable [see Goris, 1990b (*8*) for details on conducting pull tests]. The combination cables tested included conventional cables with 6, 8, and 10 outer birdcage wires. Load-displacement curves for these tests are shown in figure 14. Each curve represents the average of five pull tests. The figure shows that a conventional cable with eight outer birdcage wires closely approximates the pullout behavior and maximum load-carrying capacity of double birdcage cables. Therefore, two 60-ft-long cables with this configuration were ordered from a supplier in Australia. These cables were then instrumented with five cable bolt strain gauges each and installed in the 45-48N

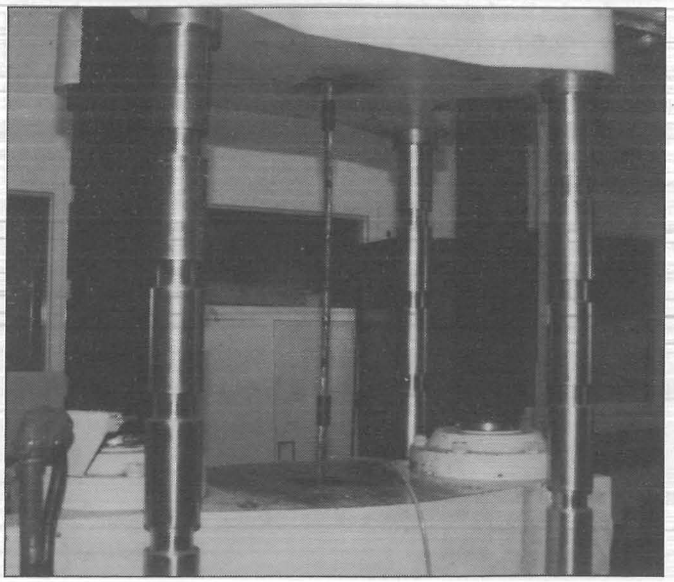


Figure 11.—Cable bolt strain gauge being tested in laboratory.

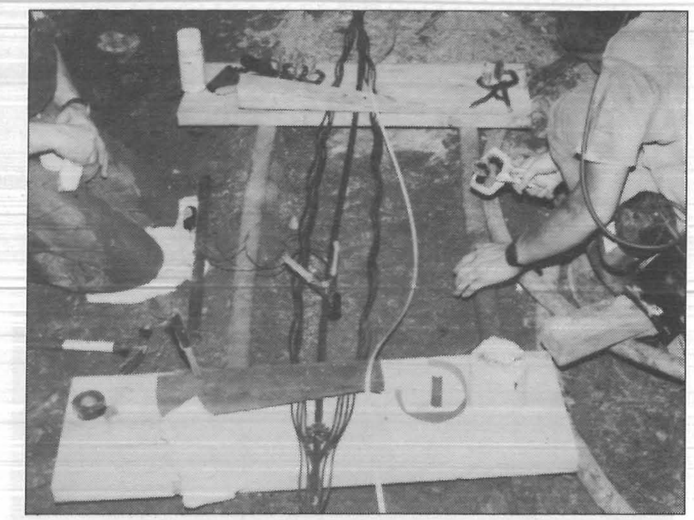


Figure 13.—Strain gauge being placed on combined conventional and birdcage cable.

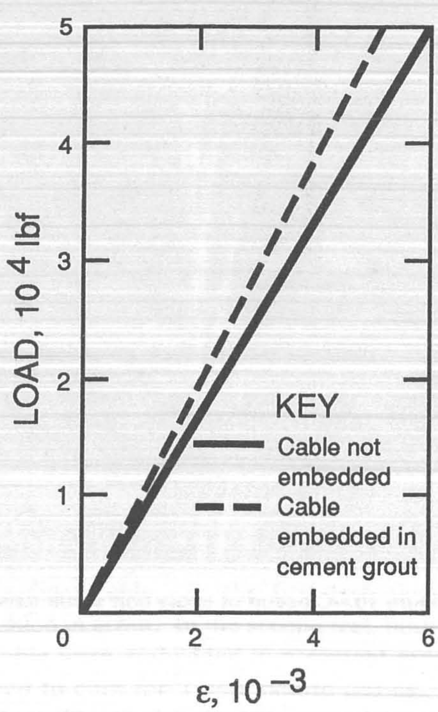


Figure 12.—Load-strain curves for cable bolt strain gauges.

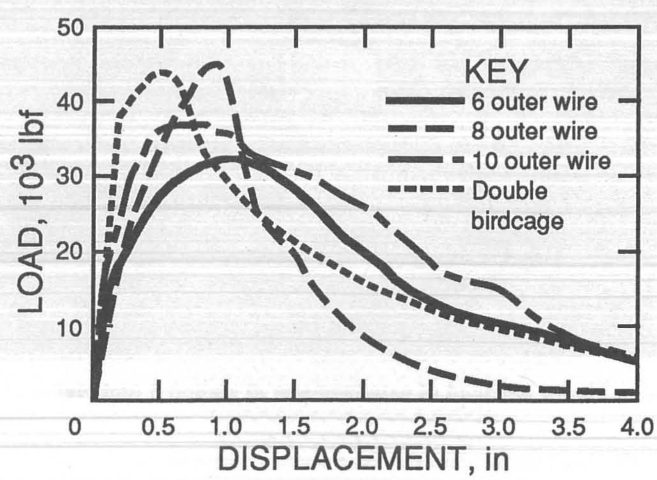


Figure 14.—Load-displacement curves for tests on combined conventional and birdcage cables.

stope at stations 12 and 14 (fig. 3). Figure 15 shows an instrumented birdcage cable prior to being placed in a hole.

Sites for Extensometer Anchors and Cable Bolt Strain Gauges

The locations of the strain gauges and extensometer anchors are shown in figure 16 along with the stope lift numbers. Mining of the stope began on the 6500 level and the instrumented cable bolts and extensometers were installed after three lifts had been mined.

Properly locating each extensometer anchor and cable bolt strain gauge was crucial to obtaining useful data. Instrumentation plans called for placing a series of four extensometer anchors and four cable bolt strain gauges near the center of each of the first four lifts (4 to 7) mined in the rock mass and then positioning another series just below the collar of the boreholes (lift 8). However, a number of unforeseen problems made this difficult. While the three boreholes (stations 2.5, 10.5, and 12) drilled from the stope up to the drift on the 6500 level remained open until instruments could be placed in them, only one of the four holes drilled from the drift on the 6500 level down into the rock mass broke through to the stope (station 12.5). However, this hole became plugged about 48 ft below the collar and could not be reopened. The remaining three holes (1.5, 3, and 14) were drilled to depths of approximately 60 ft, then flushed with air and water. As a result, in the first lift (lift 4), there were only two strain gauges and one extensometer anchor.

Data Collection System

Electrical signals from both the extensometers and the cable bolt strain gauges were transmitted to a microcomputer datalogger (micrologger) via electrical cables. The micrologger was programmed to scan each gauge every 8 h, convert electrical signals to digital values, and store the results in a 0.72-megabyte storage module. The data were then retrieved and reduced. Figure 17 shows the data collection system on-site at the mine. The location of the micrologger with respect to the instruments is shown in figure 3.

RESULTS

During the period the instruments were monitored (September 30, 1988 to August 31, 1990), three lifts were mined in the 45-48N stope, and major blasting occurred in adjacent cut-and-fill stopes as well as in vertical crater retreat stopes. It is unclear how much these other mining

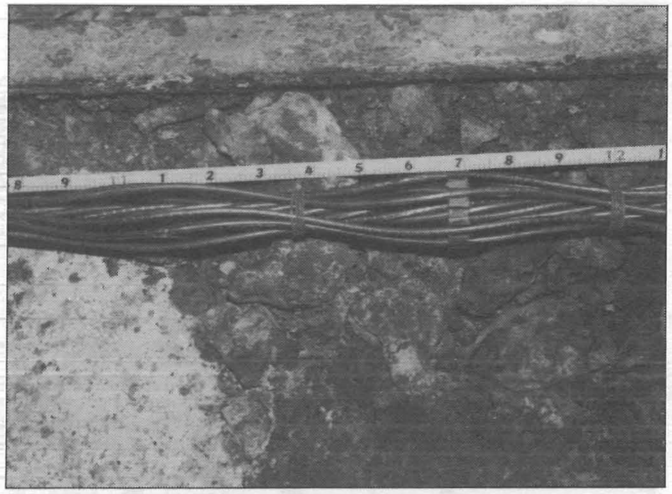


Figure 15.—Combined conventional and birdcage cables.

activities around the stope influenced readings from the instruments because records of the blasts were not kept by mine personnel.

Movement recorded by extensometers 2 and 1 at stations 10.5 and 12.5, where the birdcage cables were installed, is shown in figures 18*A* and 18*B*, respectively. Leveling of the curves indicates that the anchors were lost and the potentiometers became stationary. In studying these curves, the sequence in which the anchors were lost relative to time appears reasonable; that is, the deeper anchors (6, 5, 4, etc.) were lost first as mining progressed from the 6570 toward the 6500 level. For example, in figure 18*A*, anchor 6 was located in lift 4, which was approximately 13 ft high, between the 6570 and 6557 level. This anchor was lost after 120 days of mining. Anchor 5 was lost after 250 days, and anchor 4 was lost after 310 days. Anchor 3 continued to be read up to 600 days. Lift 7 had not been mined at the time the data collection system was removed from the mine.

It is important to note that in figure 18*A*, all of the displacement values are positive, which indicates that the rock at each anchor location was moving down and away from the drift on the 6500 level and toward the stope. There is some question about displacement values for anchors 1 and 2, at least for the first two lifts, lifts 4 and 5. These anchors were only 5.5 and 11 ft below the collar, but the displacement values are approximately the same for both anchors. It appears that the floor in the drift on the 6500 level was heaving as a result of mining in the area around the drift, and this caused the head of the extensometer to move. Photographs of the drift over the period that the instruments were being monitored indicate that a great deal of rock movement was taking place. Figure 19 shows haulage tracks being displaced by floor movement.

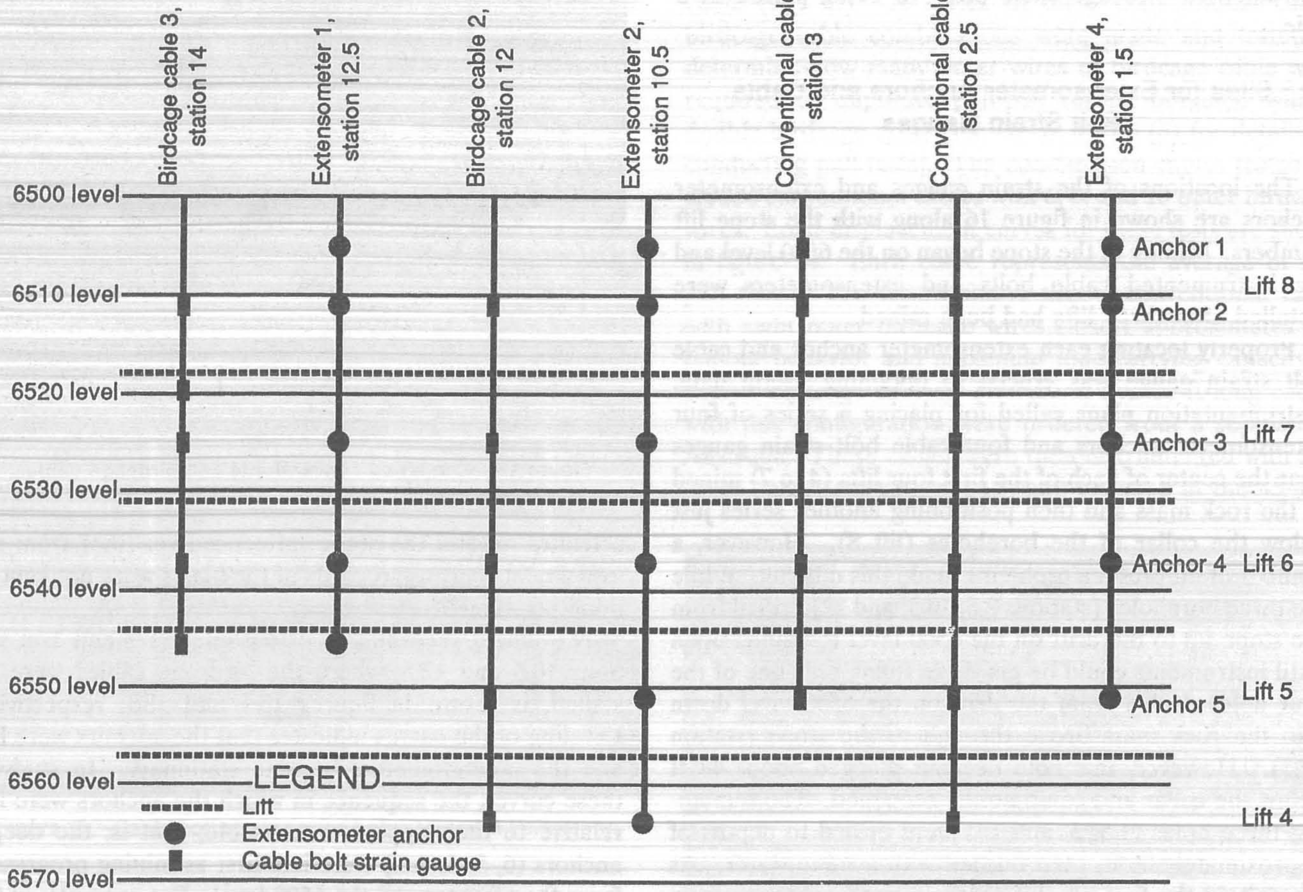


Figure 16.—Locations of extensometer anchors and cable bolt strain gauges.

At station 12.5, there were no extensometer anchors in lift 4 above the 6570 level (fig. 16). Displacements of anchors 1 through 5 for this extensometer during the first 200 days were most likely caused by floor movement in the drift on the 6500 level (fig. 18*B*).

Anchor 5 appears to have been lost after only 100 days, but it does not appear reasonable that the loss was caused by mining in the stope because the anchor was located in lift 5 about 25 ft above the stope back. There is no information to show conclusively why the anchor behaved as it did, but perhaps the fiberglass rod was sheared off somewhere in the hole as a result of movement along a joint. It is also of interest that there apparently was a horizontal separation in the rock mass between 11 and 25 ft below the collar of the hole, as indicated by the clusters of displacement values for anchors 1 and 2 and for anchors 3, 4, and 5 (fig. 18*B*). It was estimated that this separation was about 20 ft down from the collar because it was at this location that drilling water was lost during drilling of the hole.

Movement of the rock around the drift was also indicated by extensometer 3 at station 8 (fig. 3), which was placed in an uphole in the back of the stope. Anchors 1 through 5 were placed 5, 10, 20, 35, and 50 ft, respectively, from the collar of the borehole. Displacement readings from all five anchors approximated one another (fig. 18*C*), indicating that most of the movement was caused by rock movement around the drift. Consequently, when displacement values for any of the anchors were determined, the corresponding displacement value for anchor 1 at that point in time was subtracted to indicate actual movement of the anchor.

Data in figure 18*D* came from extensometer 4 located at the back of the stope area at station 1.5. This extensometer was installed 150 days after the first three extensometers and after lift 4 had been mined.

The ore vein narrowed toward the back of the stope and the ore content dropped considerably at this point. Also, the roof at the south end of the stope was difficult to support, and the area required a great deal of barring down of loose rock after each blast. As a result, the height of the stope in this region was usually 3 ft higher than in the north end. Consequently, miners in the stope decided not to mine past station 3 after lift 4. Therefore,

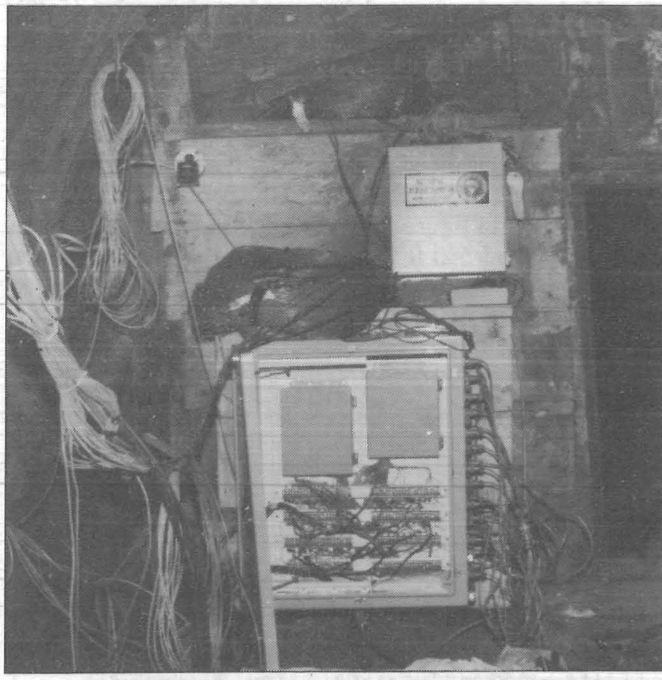


Figure 17.—Micrologger on site at mine.

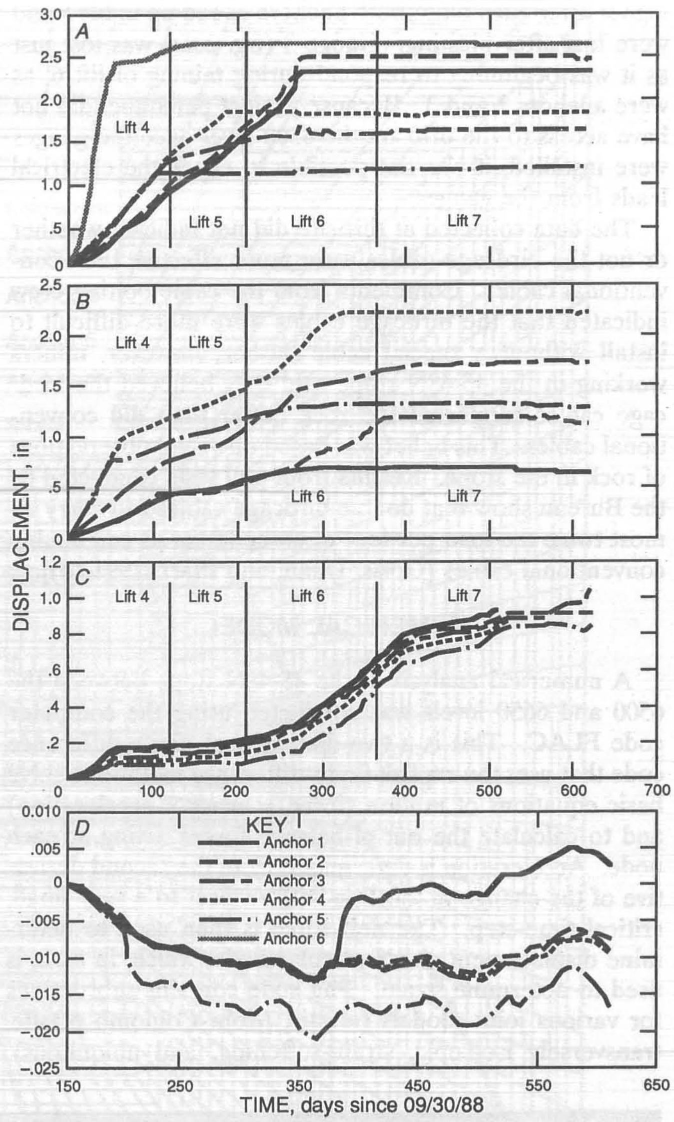


Figure 18.—Time-displacement curves for extensometers. *A*, Extensometer 2; *B*, extensometer 1; *C*, extensometer 3; *D*, extensometer 4.

the amount of displacement shown in these curves is insignificant and is most likely associated with background vibrations in the rock caused by nearby mining.

Data received from instrumented cable bolts 1 and 2 at stations 3 and 12 (figs. 20*A-B*) were very scattered because the electrical connections repeatedly shorted out and made contact again, so little information was obtained about the behavior of the rock. This was unfortunate because reliable data would have helped determine the effectiveness of the cable supports. Strain gauges on these cables were the first to be installed by the crew under field conditions, and it is felt that the installation technique may not have been adequate to protect the electrical connections from water. The condition of the electrical wires leading from the instruments to the data collection system was repeatedly checked for damage and for proper connections, but the condition of the wires down the holes could not be checked because the holes were grouted.

Although the 45-48N stope is deep (6,500 ft) and no ground water is visible, there are still large volumes of water migrating through these deep rock masses. The water comes from mining equipment, such as water coolers to cool ventilation air and drilling equipment, and activities such as backfilling of stopes. The water enters fractures in the rock and penetrates shrinkage cracks in the grout columns; it can even move along the centers of the steel cables, where it eventually comes in contact with the instruments.

The strain gauges on instrumented cable 3 at station 14 (fig. 20*C*) responded very well for several months, but then two of the gauges began to show very low negative load values, indicating that the cables were placed in compression. Part of the negative values were, however, likely the result of creep in the gauges, as discussed earlier. When gauges are installed on cables, they are placed under some strain; therefore, in addition to gauge creep, negative readings may be caused when rock fractures close and cause the cable to compress.

Strain gauges on cable 4 at station 2.5 (fig. 20*D*) responded better than gauges on cables at stations 3 and 12 (figs. 20*A-B*). Cable 4 was installed after lift 4 had been mined. The bottom anchors (5 and 4) were to be placed near the center of lifts 5 and 6, respectively; however, because the rock in the roof at the south end of the stope was difficult to support and the stope was generally 3 ft higher after each blast, both anchors ended up in lift 5 and were lost after blasting. Gauge 3 (fig. 20*D*) was lost just as it was beginning to respond during mining of lift 6, as were anchors 2 and 3. Because project personnel did not have access to the drift on the 6500 level once the gauges were installed, it was not possible to check the electrical leads from the gauges.

The data collected at this site did not indicate whether or not the birdcage cables were more effective than conventional cables. Comments from the cable bolting crew indicated that the birdcage cables were more difficult to install without a special cable pusher; however, miners working in the 45-48N stope said they believed the birdcage cables supported the rock better than did conventional cables. This belief was based on visual observations of rock in the stope. Results from pull tests conducted by the Bureau show that double birdcage cables can carry almost twice the load per foot of embedment as can double conventional cables [Goris, Duan, and Pfarr, 1991 (*6*)].

NUMERICAL MODEL

A numerical analysis of the 45-48N stope between the 6500 and 6650 levels was conducted using the computer code FLAC. This is a two-dimensional, finite-difference code that uses the explicit finite-difference method to solve basic equations of motion (force = mass × acceleration) and to calculate the out-of-balance forces acting at each node. Acceleration is determined from the second derivative of the change in location with respect to a very small, critical time-step. The new force is then used to determine displacement at a particular node, which, in turn, is used to determine strain. The code contains subroutines for various joint models (elastic, Mohr-Coulomb plastic, transversely isotropic, strain-softening, and ubiquitous). The Mohr-Coulomb model was selected for the 45-48N stope because it is a reasonable representation of the behavior of the rock and the fill materials.

To model the 45-48N stope region, a computer mesh (fig. 21) was developed based on the geology and stope geometry at station 10.5 (fig. 3). This mesh contained 5,600 elements and approximated the profile of the 45-48N stope as seen in figure 2. The mesh represented an area 500 ft wide and 500 ft high around the stope. The inner grid was 20 by 40 ft, with each grid square representing an area 5 by 5 ft. The outer grid was also 20 by 40 ft, but here each grid square varied from 5 to 12.5 ft. The model assumed simple mine geometry and did not include any previously mined stopes or drifts in the vicinity of the 45-48N stope other than those shown in figure 21.

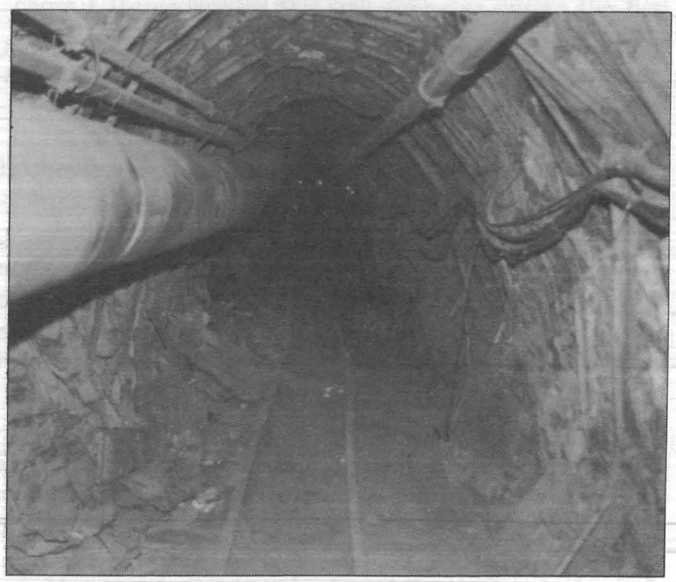


Figure 19.—Drift on 6500 level.

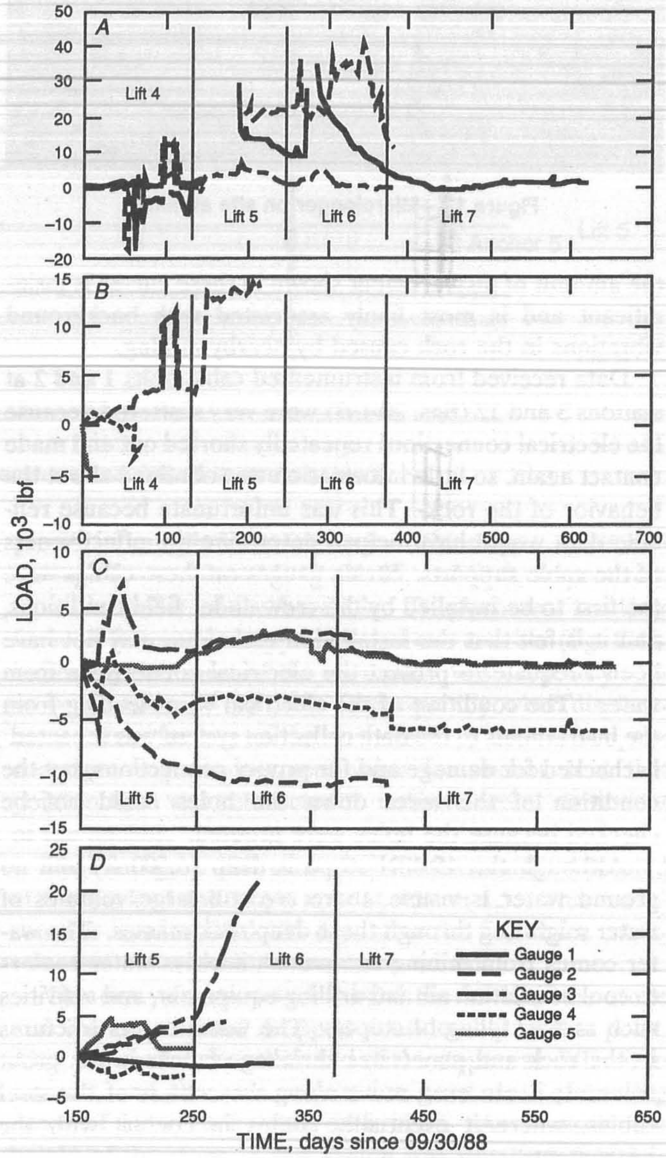


Figure 20.—Time-load curves for instrumented cable bolts. *A*, Cable bolt 1; *B*, cable bolt 2; *C*, cable bolt 3; *D*, cable bolt 4.

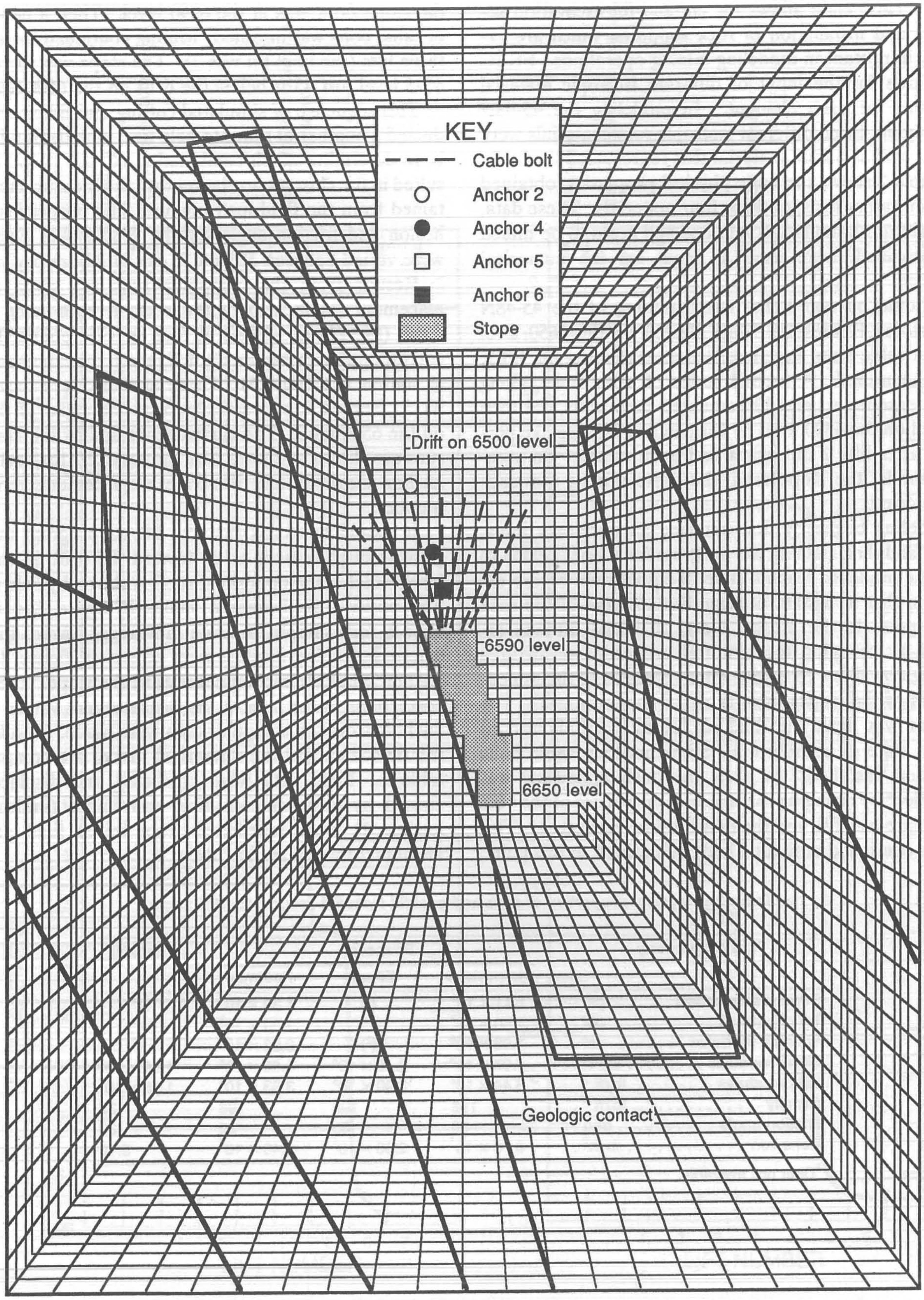


Figure 21.—Computer mesh for numerical analysis of 45-48N stope.

The FLAC code allows for cut-and-fill excavation sequences and installation of rock supports, which are extremely important in modeling mining operations. FLAC also allows for the input of different isotropic material properties for each element. In modeling the 45-48N stope region, three different isotropic rock materials were used, as well as a sand material that represented the backfill. Table 2 shows the basic physical properties, obtained from laboratory tests, of these four materials. These data, in addition to boundary conditions and areas to be mined and backfilled, were used as input for the numerical model.

It should be noted that initial mining of the 45-48N stope at the Homestake Mine started on the 6650 level and that cable bolts were installed after the first lift, lift 1, had been mined. A second series of cable bolts, along with extensometers and instrumented cable bolts, was then installed on the 6590 level by Bureau and Homestake personnel after lifts 2 through 4 had been mined. To accurately model the stope area, the same mining sequence was followed.

Modeling began by first entering all necessary information and then running the program for a brief period to allow the model to reach a state of equilibrium before excavation, that is, a state of zero displacement of the rock at any point in the mesh. Next, an initial lift was excavated by changing the material properties of an area 15 ft high by about 25 ft wide (this represented the mined-out area of the stope) from those of rock to a void. The program was again run and all forces were allowed to reach a state of equilibrium. Sixty-foot-long cable bolts were then "installed," the area previously mined out was "backfilled" by changing the material properties of the void to simulate backfill, and the first lift was "excavated." This process continued until lift 4 had been mined; at this point, the stope floor was at the 6590 level. Then a new series of cable bolts was installed. Mining continued for another three lifts (the final lift was 7). The stope was then at the 6545 level just 45 ft below the drift on the 6500 level.

This sequence of simulated cutting and filling was conducted a number of times to help fine-tune the model, that is, to determine which combination of rock properties resulted in the closest correlation with the measurements obtained from the field instruments. The properties of cohesion and friction angle for the Homestake Formation were varied between 50 and 75 pct of laboratory values.

Results from the computer runs are given as displacement (in inches) of the rock at each node in the mesh (fig. 21) as well as at four specific points (fig. 21) corresponding to anchors 2, 4, 5, and 6 for extensometer 2 at station 10.5 (fig. 16). These anchors were located 11, 38, 51, and 64 ft, respectively, from the collar of the hole on the 6500 level. The curves generated by the computer runs are shown in figures 22*A* through 22*D* for each of the anchor locations and illustrate displacement versus lift number.

It should be noted that the FLAC program indicates positive displacement as up toward the drift on the 6500 level, which is the opposite direction shown by field results. The positive displacement values in figure 22*A* for anchor 2, for example, indicate movement toward the drift, whereas positive displacement values in figure 18*A* for anchor 2 indicate movement toward the stope. Therefore, corrections in the sign of the displacement values for the anchors from the computer data were made. Results from the field data and the numerical model are compared in table 3. Field data were corrected and displacements caused by extensometer head movement (indicated by anchor 1) were subtracted from initial field data for anchors 2 though 6 (fig. 18*A*).

Table 2.—Physical properties of rock and backfill material around 45-48N stope

Physical property	Formation			Backfill
	Homestake	Ellison	Poorman	
Compressive stress ..... psi	1.41×10^4	(1)	1.39×10^4	(1)
Tensile stress .... psi	1.50×10^3	(1)	1.39×10^3	(1)
Young's modulus .... psi	1.08×10^6	(1)	9.90×10^6	(1)
Shear modulus .... lb/ft²	1.97×10^8	2.40×10^8	1.62×10^8	1.89×10^6
Bulk modulus .... lb/ft²	3.44×10^8	2.90×10^8	3.33×10^8	1.45×10^6
Density .... lb/ft³	198	198	198	135
Friction angle .... deg	30	30	30	30
Cohesion .... lb/ft²	3.70×10^5	2.80×10^5	2.40×10^5	3.10

[1]Data not available.

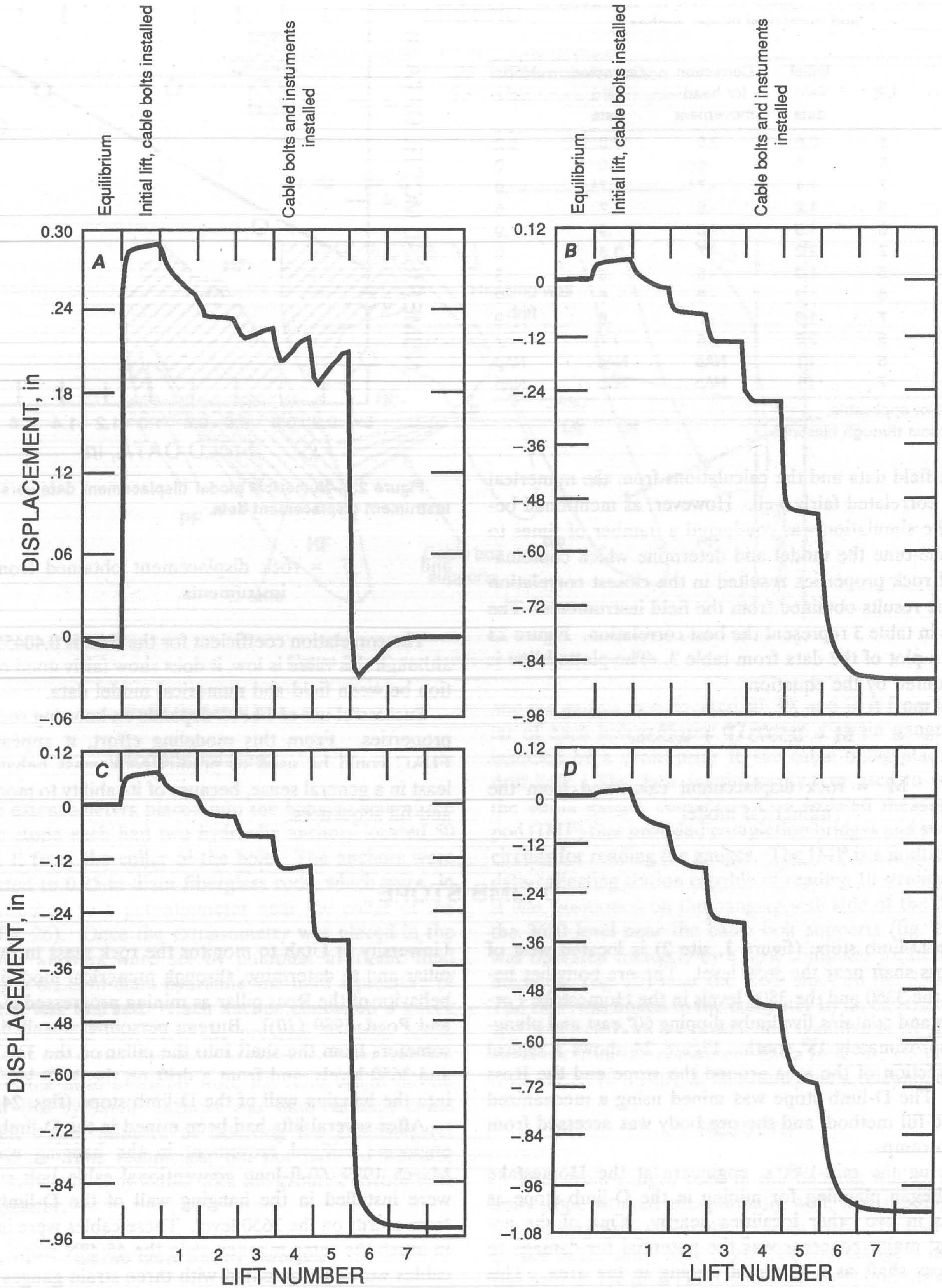


Figure 22.—Displacement curves from numerical analysis of 45-48N stope. ***A*****, Anchor 2;** ***B*****, anchor 4;** ***C*****, anchor 5;** ***D*****, anchor 6.**

Table 3.—Anchor displacement data for extensometer and numerical model, inches

Anchor	Lift	Initial field data	Correction for head movement	Corrected field data	Model data
2	5	0.5	-0.5	0.0	-0.2
	6	.6	-.6	.0	.0
	7	1.4	-.7	.7	.0
4	5	1.2	-.5	.7	.5
	6	1.5	-.6	.9	.9
	7	2.2	-.7	1.5	.9
5	5	1.0	-.5	.5	.3
	6	1.2	-.6	.6	.9
	7	1.3	-.7	.6	.9
6	5	2.2	-.5	1.7	.7
	6	([1])	NAp	NAp	NAp
	7	([1])	NAp	NAp	NAp

NAp Not applicable.
[1]Anchor lost through blasting.

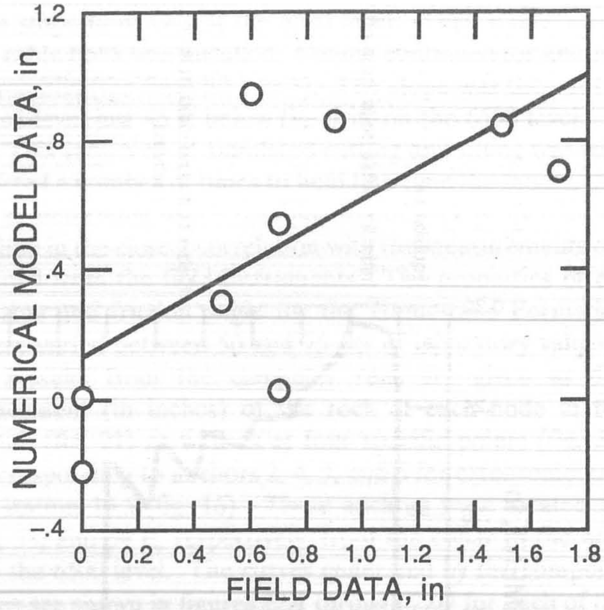


Figure 23.—Numerical model displacement data versus field instrument displacement data.

The field data and the calculations from the numerical model correlated fairly well. However, as mentioned before, the simulation was conducted a number of times to help fine-tune the model and determine which combination of rock properties resulted in the closest correlation with the results obtained from the field instruments. The results in table 3 represent the best correlation. Figure 23 shows a plot of the data from table 3. The plotted line is represented by the equation

$$M = 0.5073 \cdot F + 0.1358,$$

where M = rock displacement calculated from the numerical model

and F = rock displacement obtained from field instruments.

The correlation coefficient for the data is 0.404554, and although this value is low, it does show fairly good correlation between field and numerical model data.

Successful use of FLAC depends on knowing rock mass properties. From this modeling effort, it appears that FLAC could be used to predict rock mass behavior, at least in a general sense, because of its ability to model cut-and-fill sequences.

D-LIMB STOPE

The D-limb stope (figure 1, site 2) is located west of the Ross shaft near the 3650 level. The ore body lies between the 3350 and the 3800 levels in the Homestake Formation and contains five limbs dipping 60° east and plunging approximately 15° south. Figure 24 shows a typical cross section of the area around the stope and the Ross shaft. The D-limb stope was mined using a mechanized cut-and-fill method, and the ore body was accessed from a main ramp.

During the mid-1980's, engineers at the Homestake Mine began planning for mining in the D-limb stope as well as in two other localities nearby. One of the engineers' major concerns was the potential for damage to the Ross shaft as a result of mining in the area. This concern prompted assistance from the Bureau and the University of Utah to monitor the rock mass in the Ross pillar and to determine, through numerical modeling, the behavior of the Ross pillar as mining progressed [Johnson and Poad, 1989 (*10*)]. Bureau personnel installed extensometers from the shaft into the pillar on the 3350, 3500, and 3650 levels, and from a drift on the 3650 level down into the hanging wall of the D-limb stope (figs. 24-25).

After several lifts had been mined in the D-limb stope, engineers noticed movement in the hanging wall. In March 1989, 60-ft-long conventional cable bolt supports were installed in the hanging wall of the D-limb stope from a drift on the 3650 level. These cables were installed in much the same manner as in the 45-48N stope. Three cables were instrumented with three strain gauges each.

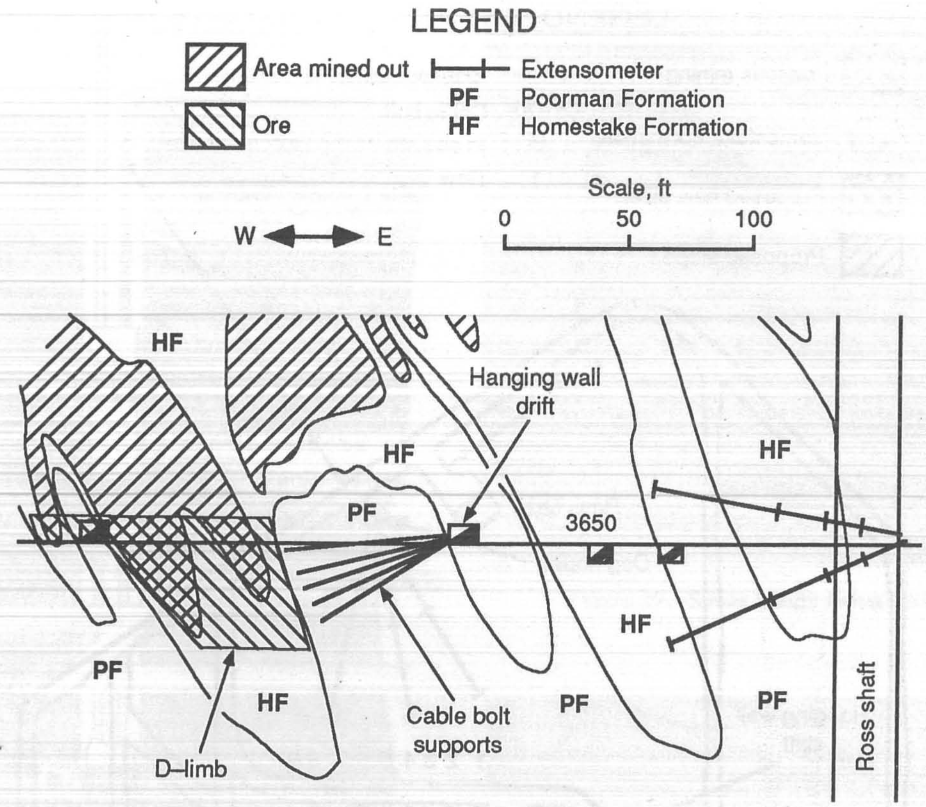


Figure 24.—Cross section of D-limb stope area.

INSTRUMENTS

Extensometers

The extensometers placed into the hanging wall of the D-limb stope each had two hydraulic anchors located 50 and 15 ft from the collar of the hole. The anchors were connected to 0.25-in-diam fiberglass rods, which were, in turn, attached to a potentiometer near the collar of the hole (fig. 26). Once the extensometer was placed in the hole, the anchors were set by pumping hydraulic fluid through a 0.125-in-diam hydraulic line until a pressure of 2,000 psi was reached. Each anchor contained a check valve that would not allow the pressure to drop below 1,500 psi. Once the downhole anchors were set, the anchors on the head assembly were set. One major advantage of the hydraulic anchors was that readings could be taken within minutes of inserting the extensometer in the hole. The technicians did not have to wait for grouts to cure as they would with the grouted anchor-type extensometer.

Cable Bolt Strain Gauges

The cable bolt strain gauges were the same type as those used in the 45-48N stope. The gauges were placed on downhole cables located in rows 1, 3, and 6 (fig. 25), and the gauges were located 50, 30, and 10 ft from the collar of each hole. Figure 27 shows a strain gauge being installed on a cable prior to the cable being placed in a drill hole. The data acquisition system used to monitor the strain gauges consisted of an isolated measurement pod (IMP) that provided completion bridges and switching circuits for reading the gauges. The IMP is a multichannel data-collecting station capable of reading 10 strain gauges. It was positioned on the hanging wall side of the drift on the 3650 level near the cable bolt supports (fig. 28) and was operated remotely by a host computer located in the doghouse (fig. 25) near the Ross shaft on the 3500 level. The IMP was linked to the computer by an electrical cable and received both its power and commands to scan the channels from the computer. Data from the gauges were transmitted to a Bureau laboratory via a telephone link.

RESULTS

The strain gauges used in the hanging wall of the D-limb stope worked exceptionally well, and readings correlated very well with readings from extensometers placed earlier in the same area (figs. 29*A*-*C*). Gauge 3 in row 1 was nearest the drift (10 ft down the hole) and showed the highest loading (fig. 29*A*), indicating that the rock near the drift was moving and loading the cables. This was consistent with extensometer data showing nearly equivalent

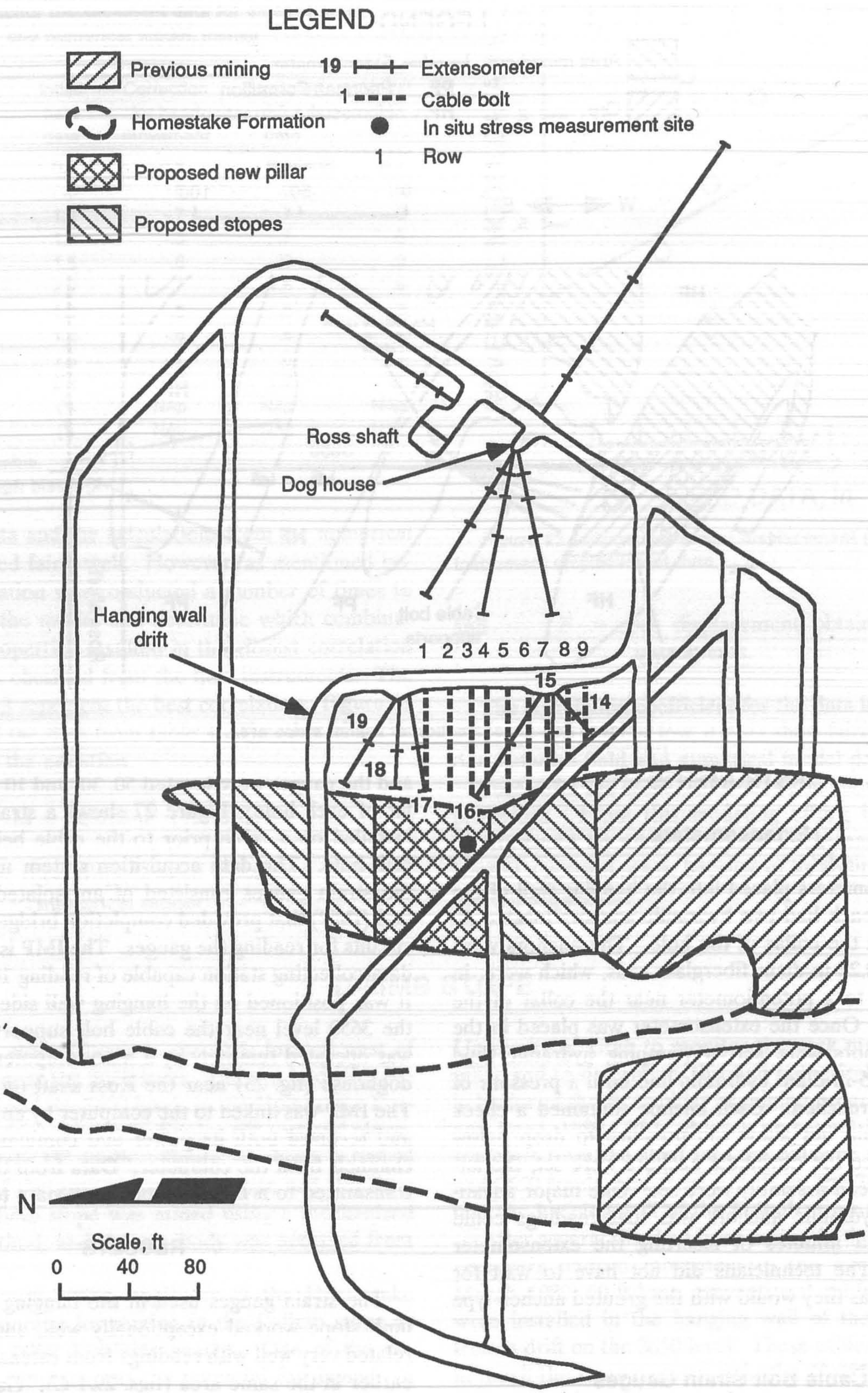


Figure 25.—Plan view of 3560 level near Ross shaft.

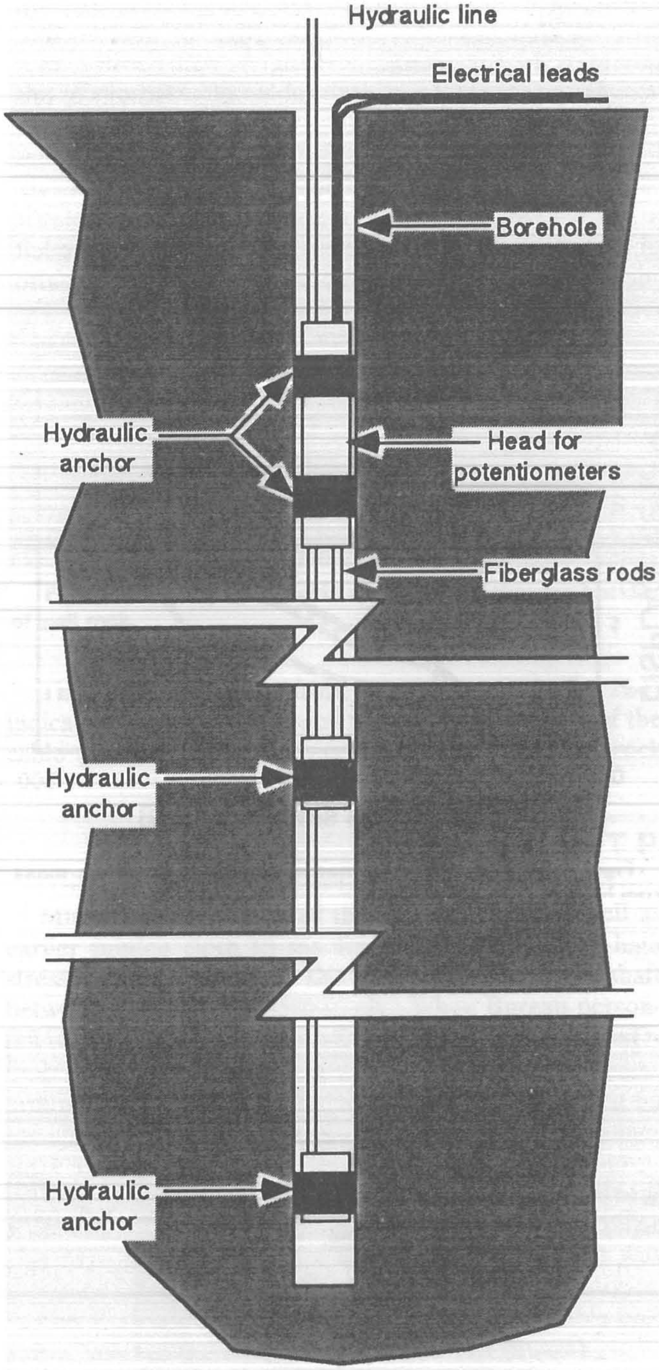


Figure 26.—Profile of extensometer with hydraulic anchors.

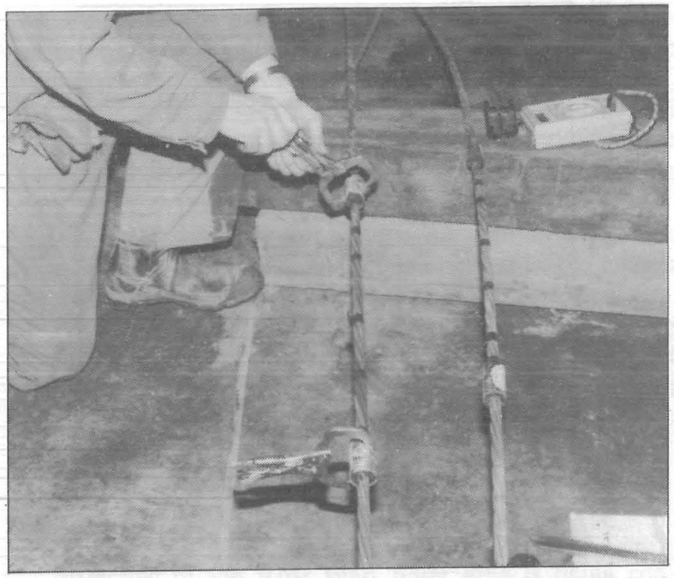


Figure 27.—Strain gauge being installed on cable.

Figure 28.—Isolated measurement pod unit on wall of hanging wall drift.

displacements for anchors in the same holes (fig. 30). Anchors 17-1, 17-2, 18-1, and 18-2 (fig. 30) were used to determine movement of the hanging wall because they were close to the instrumented cable bolts. Results of this rock movement are seen in figure 31 in which an old air door frame in the hanging drift on the 3650 level has buckled. The collars of the holes for the cable bolts are on the left side of the photograph and have white plastic tubes protruding out from the rock. The rock on the left side of the drift where the cable bolts were placed is also buckling (fig. 32).

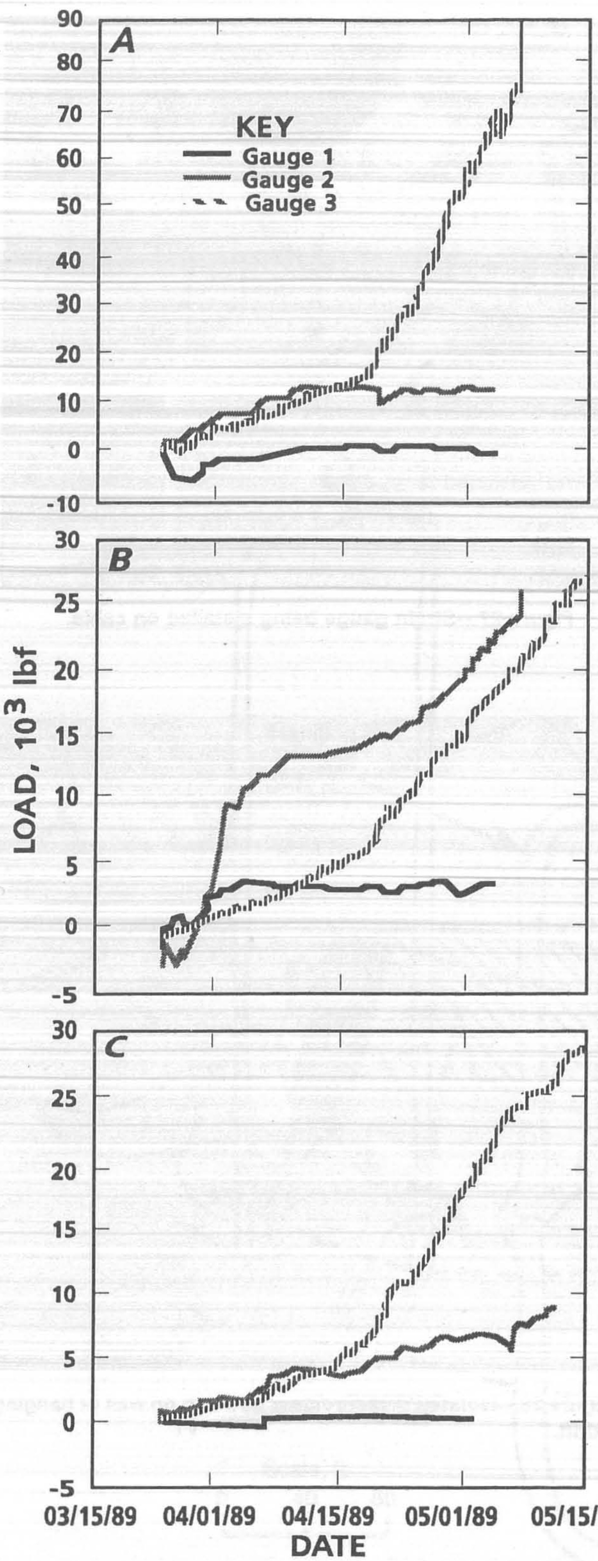


Figure 29.—Time-load curves for instrumented cable bolts. *A*, Row 1; *B*, row 3; *C*, row 6. Gauge 1 located 50 ft down hole; gauge 2 located 30 ft down hole; gauge 3 located 10 ft down hole.

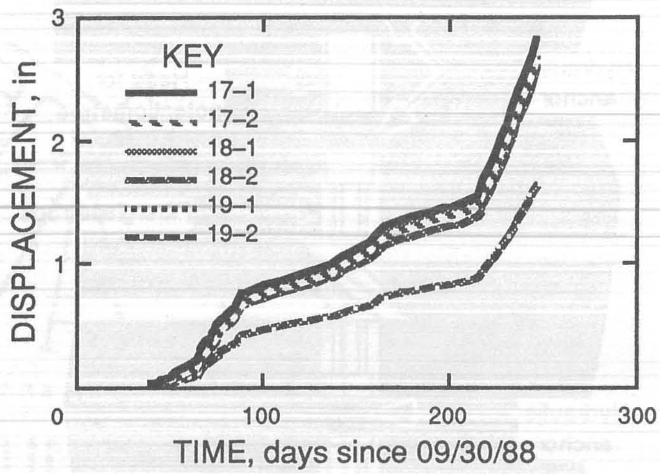


Figure 30.—Data from extensometers placed in rock mass from hanging wall drift.

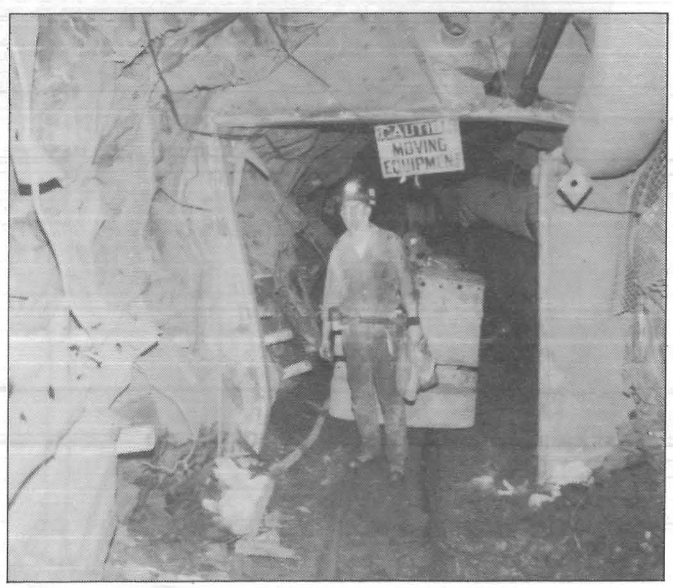


Figure 31.—Hanging wall drift on 3650 level showing buckling of old air door frame.

Figure 32.—Hanging wall drift on 3650 level showing buckling of wall rock.

The last few readings from gauge 3 in row 1 (fig. 29*A*) indicated that loads exceeded the ultimate strength of the cable (58,000 lbf). This was most likely caused by rock shearing near the gauge, resulting in excessive elongation of the cable and gauge wire. Therefore, the gauge wire was stretched abnormally, indicating a very high load. Readings from gauges 1 and 2 in row 1, located 10 and 30 ft from the stope hanging wall, respectively, showed less loading, thereby indicating that the rock was not moving very much in this area.

The load histories for gauges on two other cables (figs. 29*B-C*) showed similar behaviors; that is, the gauges nearest the drift showed the highest loads.

All of these cable bolt strain gauges stopped functioning toward the end of May 1989; however, they provided excellent information on the behavior of the cables and have proven to be very useful tools in analyzing rock mass behavior.

NUMERICAL MODEL

Modeling of the Ross shaft pillar area is being conducted by personnel at SRC in cooperation with William G. Pariseau, professor, of the University of Utah. The finite-element codes being used are UTAH II and UTAH III. Preliminary results of this work appear in Johnson and Poad [1989 (*10*)], Johnson and Orr [1990 (*11*)], and Brady and Johnson [1989 (*12*)].

ROSS SHAFT PILLAR, 3350 LEVEL

Mining in the vicinity of the D-limb stope, as well as earlier mining close to the Ross shaft, has caused high stresses and movement in the rock mass around the shaft between the 3200 and 3500 levels. When Bureau personnel were placing instruments in the hanging wall of the D-limb stope, they also placed extensometers into the pillar around the Ross shaft to monitor the effects of mining on the shaft pillar near the 3350 level. In mid-1990, Homestake Mine personnel became concerned about rock movement in the shaft on the 3350 level, and consequently installed approximately 70 cable bolts to reinforce the pillar (fig. 33). The procedure for installing the cable bolts was the same as that used in the 45-48N and D-limb stopes. At this time, Bureau personnel also installed 2 cable bolt strain gauges on 6 of these cable bolts (12 gauges).

INSTRUMENTS

Extensometers

The extensometers placed in the shaft pillar (figs. 33-34) were the same type as those placed in the hanging wall of the D-limb stope. Each extensometer had five anchors placed 25, 50, 75, and 100 ft from the hole collars. Extensometer 1 (fig. 34*A*) was placed in an uphole at an angle of 35° while extensometer 2 was placed in a downhole at an angle of -38°.

Cable Bolt Strain Gauges

The cable bolt strain gauges were the same type as used in the 45-48N and the D-limb stopes. The fan-shaped configuration for cables in rows 1 through 14 is shown in figure 34; installation information is given in table 4.

Table 4.—Installation data on instrumented cable bolts (gauges 1 and 2 were installed on 50-ft cables, while the remainder were installed on 60-ft cables)

Gauge	Cable	Row	Hole	Distance of gauge from collar of hole, ft
1	1	9	1	10
2	1	9	1	40
3	2	9	5	10
4	2	9	5	50
5	3	8	1	10
6	3	8	1	50
7	4	8	5	10
8	4	8	5	50
9	5	6	4	10
10 ...	5	6	4	50
11 ...	6	3	4	10
12 ...	6	3	4	50

RESULTS

The loading history for instrumented cable bolts 1 through 6 is shown in figure 35 as collected between October 1990 and early January 1991. A malfunction of the data collection system caused a gap in the data between November 18 and December 2. Most of the gauges appeared to be functioning well except gauge 10 (fig. 35*E*), which exceeded the capacity of the cable (58,000 lb) between early November 1990 and late January 1991. The cause of these extreme values is unknown.

A pattern emerged with regard to loading behavior on the cables. Gauges 1, 3, 5, and 7 on cables in rows 8 and 9 (fig. 35) and gauges 10 and 12 on cables in rows 6 and 3 (fig. 33) all showed loading, although the loads were small. Gauges 2, 4, 6, and 8 did not show any loading. As mentioned earlier, the data from gauge 10 were questionable.

These data indicated that the rock near the haulage drift and adjacent to the shaft was moving slightly and loading the cables (fig. 34*B*). In addition, data from gauge 12 indicated that the rock to the southwest of the shaft was also moving and loading the cables (fig. 35*F*). The two extensometers in this region have not indicated any rock movement (fig. 36); curves are more or less level and consistent.

The cable bolt strain gauges and extensometers in this region continue to function as of this writing and will continue to be monitored for the near future.

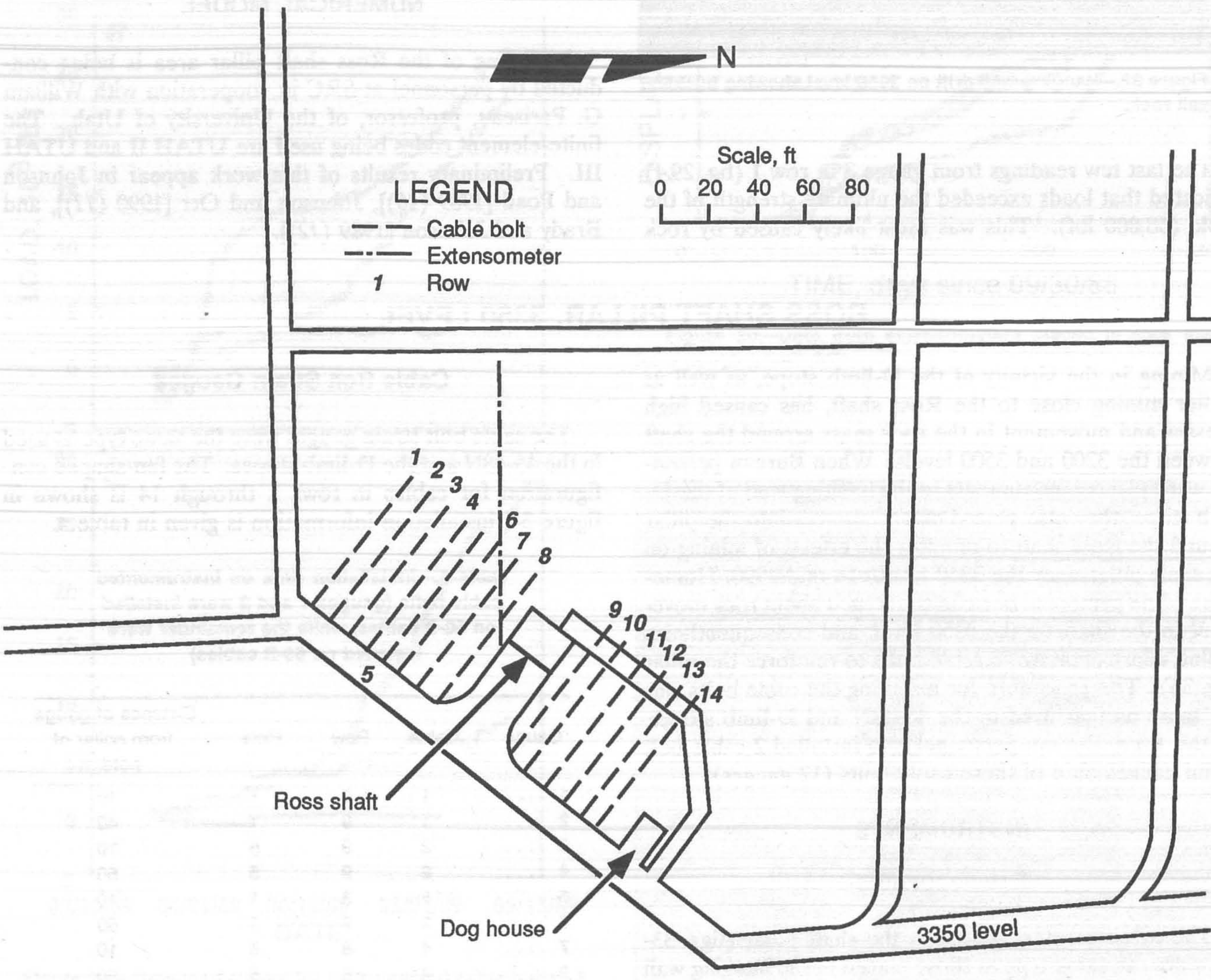


Figure 33.—Plan view of 3350 level showing location of cable bolts installed in shaft pillar.

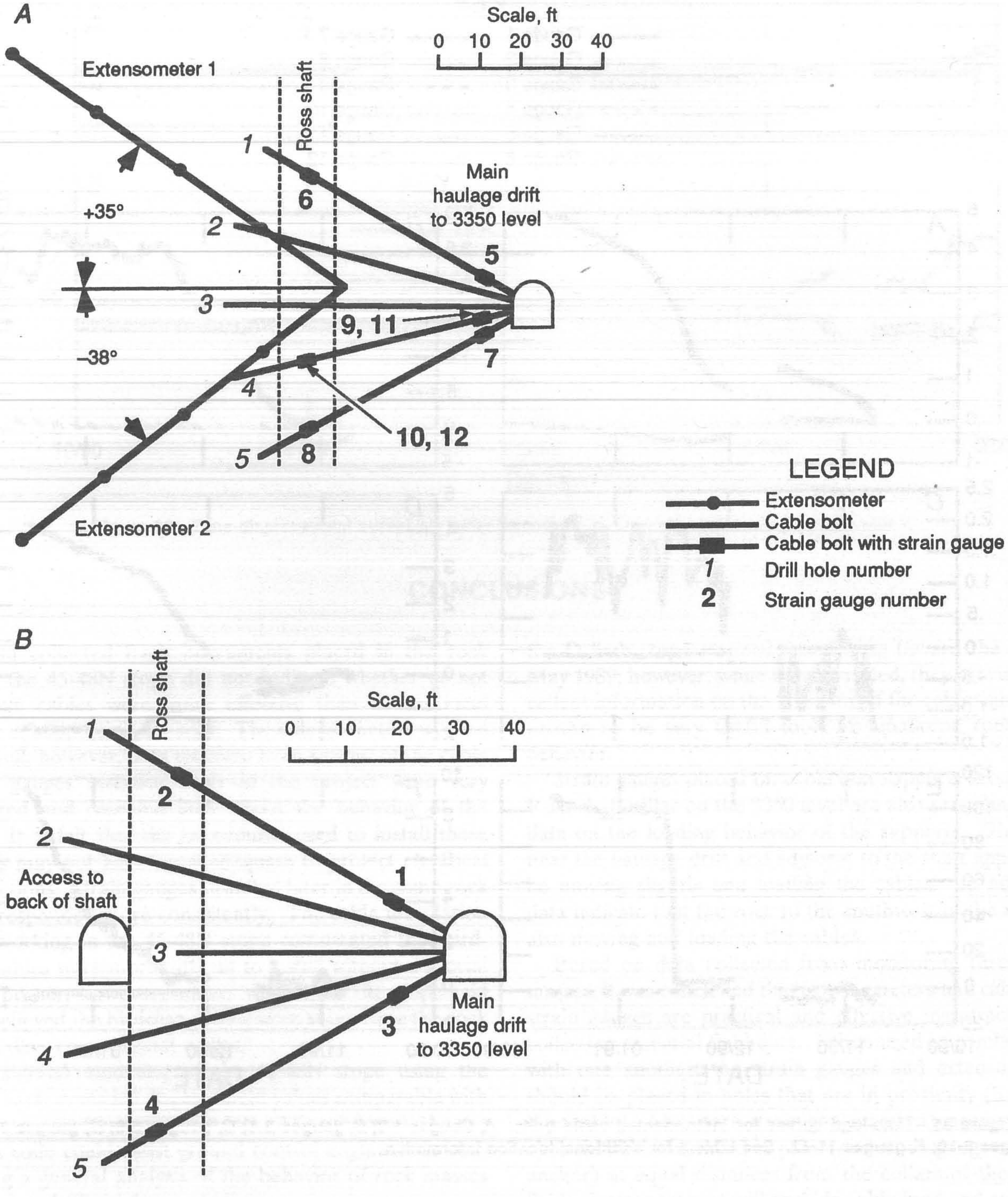


Figure 34.—Cross section of 3350 level showing typical fan-shaped configuration of cable bolts. *A*, Rows 1-8; *B*, rows 9-14.

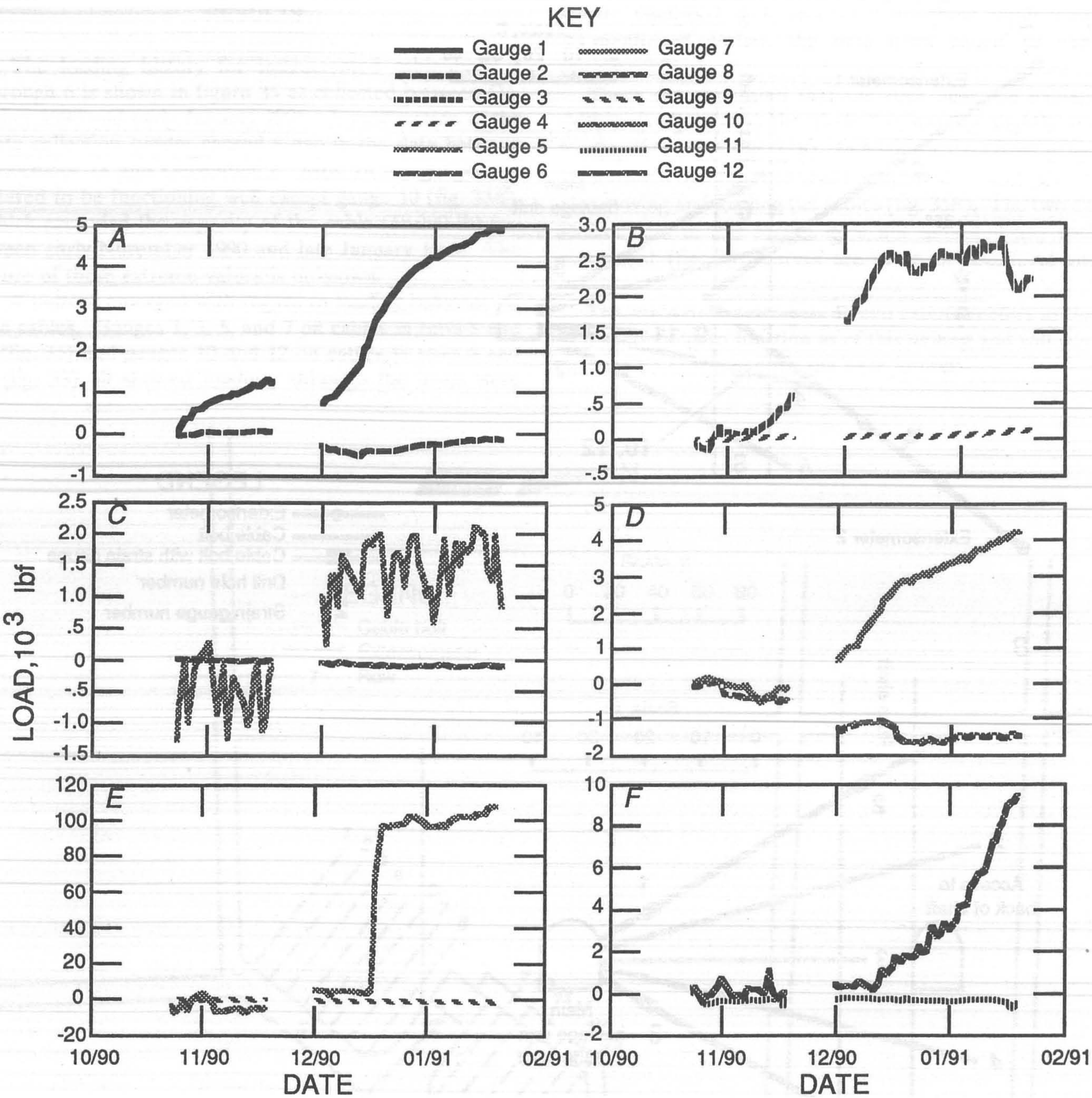


Figure 35.—Time-load curves for instrumented cable bolts. ***A*, Gauges 1-2; *B*, gauges 3-4; *C*, gauges 5-6; *D*, gauges 7-8; *E*, gauges 9-10; *F*, gauges 11-12. See table 5 for additional location information.**

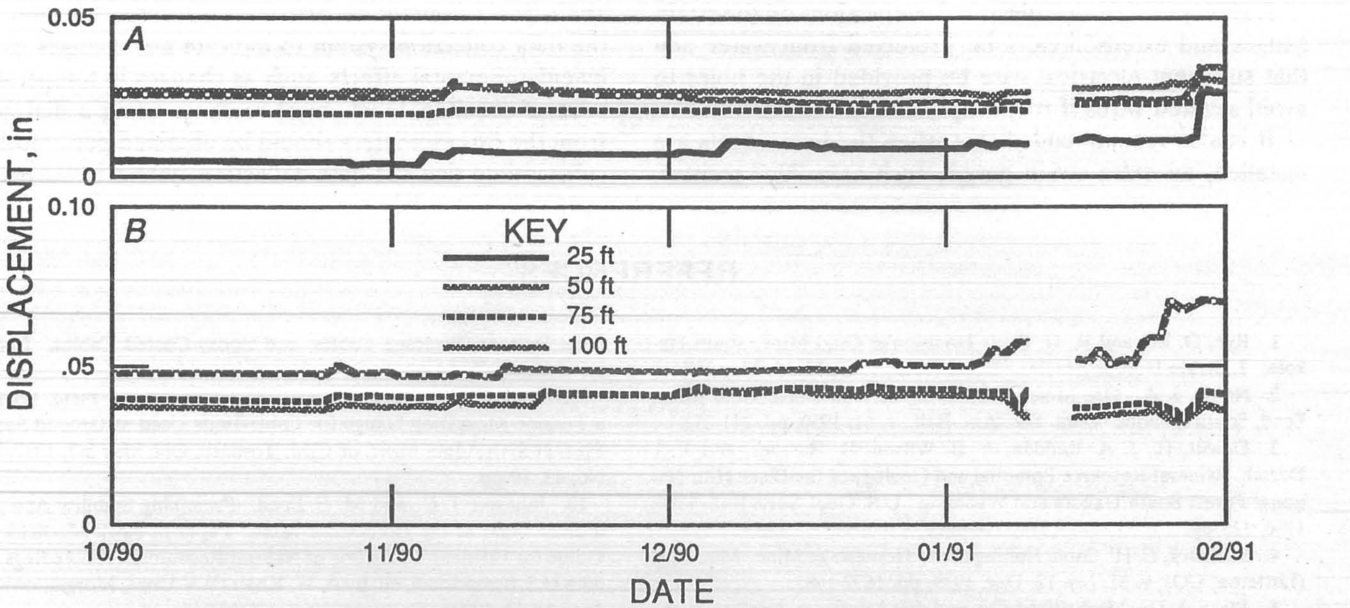


Figure 36.—Time-displacement curves for extensometers. *A*, Extensometer 1; *B*, extensometer 2.

CONCLUSIONS

Data collected from instruments placed in the rock above the 45-48N stope did not indicate whether or not birdcage cables were more effective than conventional cables at supporting the roof. The extensometers worked very well; however, data received from several of the cable strain gauges installed early in the project were very scattered and revealed little about the behavior of the rock. It is felt that the procedures used to install these gauges may not have been adequate to protect electrical connections. Strain gauges installed later in the same rock mass responded more consistently. The cable installation crew working in the 45-48N stope commented that birdcage cables were more difficult to install without a special cable pusher; however, miners working in the stope said they believed the birdcage cables were supporting the rock better than conventional cables.

Numerical modeling of the 45-48N stope using the FLAC code provided displacement values comparable with field extensometer measurements. At a minimum, the use of this code could assist ground control engineers in conducting a general analysis of the behavior of rock masses in a cut-and-fill mining operation.

Strain gauges placed on cables in the hanging wall of the D-limb stope worked exceptionally well, and data correlated well with readings from extensometers placed earlier in the same area. Data from both the strain gauges and the extensometers indicated that the rock in the hanging wall near the stope was stable, but that the rock surrounding the hanging wall drift was moving and loading the cables. All of the cable bolt strain gauges placed near the D-limb stope stopped functioning toward the end of May 1989; however, while they operated, they provided excellent information on the behavior of the cables and have proven to be very useful tools for analyzing rock mass behavior.

Strain gauges placed on cable bolt supports around the Ross shaft pillar on the 3350 level are also providing good data on the loading behavior of the supports. The rock near the haulage drift and adjacent to the shaft appears to be moving slightly and loading the cables. In addition, data indicate that the rock to the southwest of the shaft is also moving and loading the cables.

Based on data collected from monitoring three rock masses, it was concluded that extensometers and cable bolt strain gauges are practical and effective instruments for collecting essential field data. When used in combination with one another, the strain gauges and extensometers should be placed in holes that are in proximity (3 to 6 ft if possible). Also, each strain gauge and extensometer anchor should be installed in pairs (one strain gauge and one anchor) at equal distances from the collars of the holes. Such an arrangement will provide cable load and rock displacement information on equivalent planes (fig. 16). It is also important that instrumented cable bolts be installed at the same spacings as regular cable bolts. The number of cable bolt supports per unit area should not be increased just because some cables have strain gauges on them. This way, the instrumented cable bolt will experience the same loading conditions as the standard cable bolts.

It is very critical that electrical connections on the strain gauges and extensometers be protected from water and that sufficient electrical wire be provided in the holes to avoid severed wires if rock displacement occurs.

It is also recommended that when the instruments are installed, an extra strain gauge, such as a 70-Ω resister, and a potentiometer be placed in a secured location near the data collection system to indicate any changes caused by environmental effects, such as changes in temperature. Also, if possible, mechanical readings using a dial gauge from the extensometers should be obtained periodically to substantiate that the data collection system is working.

REFERENCES

1. Rye, D. M., and R. O. Rye. Homestake Gold Mine, South Dakota: I. Stable Isotope Studies. Econ. Geol., v. 69, No. 3, May 1974.

2. Noble, J. A. Ore Mineralization in the Homestake Gold Mine, Lead, South Dakota. Geol. Soc. Am. Bull., v. 61, 1950, pp. 221-252.

3. Dewitt, E., J. A. Redden, A. B. Wilson, D. Buscher, and J. S. Dersch. Mineral Resource Potential and Geology of the Black Hills National Forest South Dakota and Wyoming. U.S. Geol. Surv. Bull. 1580, 1986, 135 pp.

4. Schmuck, C. H. Cable Bolting at the Homestake Mine. Min. Eng. (Littleton, CO), v. 31, No. 12, Dec. 1979, pp. 1677-1681.

5. Pfarr, J. D. Mechanized Cut and Fill Mining as Applied at the Homestake Mine. Min. Eng. (Littleton, CO), v. 43, No. 12, Dec. 1991, pp. 1437-1439.

6. Goris, J. M., F. Duan, and J. Pfarr. Evaluation of Cable Supports at the Homestake Mine. CIM Bull., v. 84, No. 947, Mar. 1991, pp. 146-150.

7. Goris, J. M. Laboratory Evaluation of Cable Bolt Supports (In Two Parts). 1. Evaluation of Supports Using Conventional Cables. BuMines RI 9308, 1990a, 23 pp.

8. _____. Laboratory Evaluation of Cable Bolt Supports (In Two Parts). 2. Evaluation of Supports Using Conventional Cables With Steel Buttons, Birdcage Cables, and Epoxy-Coated Cables. BuMines RI 9342, 1990b, 14 pp.

9. Choquet, P., and F. Miller. Development and Field Testing of a Tension Measuring Gauge for Cable-Bolts Used as Ground Support. Pres. at 89th Annu. Meet. of CIM, Toronto, ON, May 3-7, 1987, Paper No. 43, 19 pp.

10. Johnson, J. C., and M. E. Poad. Premining Stability Analysis of a Shaft Pillar at the Homestake Mine. Paper in Rock Mechanics as a Guide for Efficient Utilization of Natural Resources: Proceedings of the 30th U.S. Symposium, ed. by A. W. Khair (WV Univ., Morgantown, WV, June 19-22, 1989). Balkema, 1989, 175-182 pp.

11. Johnson, J. C., and S. A. Orr. Rock Mechanics Applied to Shaft Pillar Mining. Pres. at 1990 SME Annu. Meet., Salt Lake City, UT, Feb. 26-Mar. 1, 1990, 10 pp.; also in Int. J. Min. and Geol. Eng., v. 8, 1990, pp. 385-392.

12. Brady, T. M., and J. C. Johnson. Comparison of a Finite-Difference Code to a Finite-Element Code in Modeling an Excavation in an Underground Shaft Pillar. Paper in 3rd International Symposium on Numerical Models in Geomechanics—NUMOG III (Niagara Falls, ON, May 8-11, 1989). Elsevier, 1989, pp. 608-619.

INT.BU.OF MINES,PGH.,PA 29797

*U.S.G.P.O.: 1993--709-008/80029