

Rock Mechanics Study of Shaft Stability and Pillar Mining, Homestake Mine, Lead, SD

(In Three Parts)

2. Mine Measurements and Confirmation of Premining Results

UNITED STATES DEPARTMENT OF THE INTERIOR



UNITED STATES BUREAU OF MINES

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Report of Investigations 9576

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and Pillar Mining, Homestake Mine,
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By **W. G. Pariseau, J. C. Johnson, M. M. McDonald,
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Bruce Babbitt, Secretary**

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Rhea Lydia Graham, Director**

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Metric Units

cm	centimeter	mm	millimeter
kg	kilogram	MPa	megapascal
m	meter	pct	percent

U.S. Customary Units

ft	foot	psi	pound per square inch
in	inch	tr oz	troy ounce

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ABSTRACT

A U.S. Bureau of Mines case study of pillar recovery in high-grade ore near the Ross shaft at the Homestake Mine, Lead, SD, has demonstrated the usefulness of the finite-element method for evaluating shaft pillar mining plans and shaft stability. This report focuses on mine measurements and calibration of the two-dimensional computer model and is the second in a series of three Reports of Investigations describing the Ross shaft study. In this study, borehole extensometers and other instruments were installed to provide data for model verification and to monitor the shaft. Results of the recalibrated two-dimensional model (UTAH2) confirmed the premining stability evaluation.

However, after mining began, concern was expressed because cracks and other signs of ground motion appeared at considerable distances from the area of active pillar mining. In part 3 of the study, an intensive three-dimensional modeling effort using UTAH3 was initiated. The results again showed that the observed effects were within expectations and that the shaft would remain safe. Three-dimensional analyses of alternative pillar mining scenarios indicated that more of the shaft pillar ore reserve could be recovered than previously thought.

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INTRODUCTION

Because of the importance of shaft pillar design to the mining industry, researchers from the U.S. Bureau of Mines (USBM) initiated a study to investigate the extraction of valuable reserves within the Ross shaft pillar at the Homestake Mine, Lead, SD. The study was a cooperative effort and involved the USBM, the Homestake Mining Co., and the University of Utah, Salt Lake City, UT. Table 1 shows the chronology of the main project phases.

Table 1.—Project chronology

Phase	Topic	Beginning date
1	Premining stability analysis.	April 1987.
2	Installation of instruments and validation of two-dimensional model.	October 1987.
3	Three-dimensional stability analysis.	August 1990.
4	Installation of additional instruments and update of three-dimensional model.	June 1994.

The Homestake Mine is located in the northern Black Hills of South Dakota. Figure 1 shows the general layout of the mine, which is the oldest and deepest in North America. Development extends to the 8000 level (2,440 m or 8,000 ft below surface), with the Yates and Ross shafts providing access. About 8,400 kg (270,000 tr oz) of gold and 1,500 kg (50,000 tr oz) of silver are recovered from 1.5 million metric tons (1.7 million short tons) of ore milled each year. Most of the ore reserve in the Ross shaft pillar lies between the 3200 and 3800 levels on the west side of the shaft. Stopping methods are mainly mechanized cut-and-fill and vertical crater retreat.

Pillar mining began below the 3650 level in late 1988. Shortly afterward, movement was observed on the 3200 level, where the shaft had been damaged in the early 1950's. In fact, it was the

experience in the early 1950's that led to definition of the existing shaft pillar. Additional pillars within the shaft pillar were then defined in response to the perceived threat of renewed ground movement.

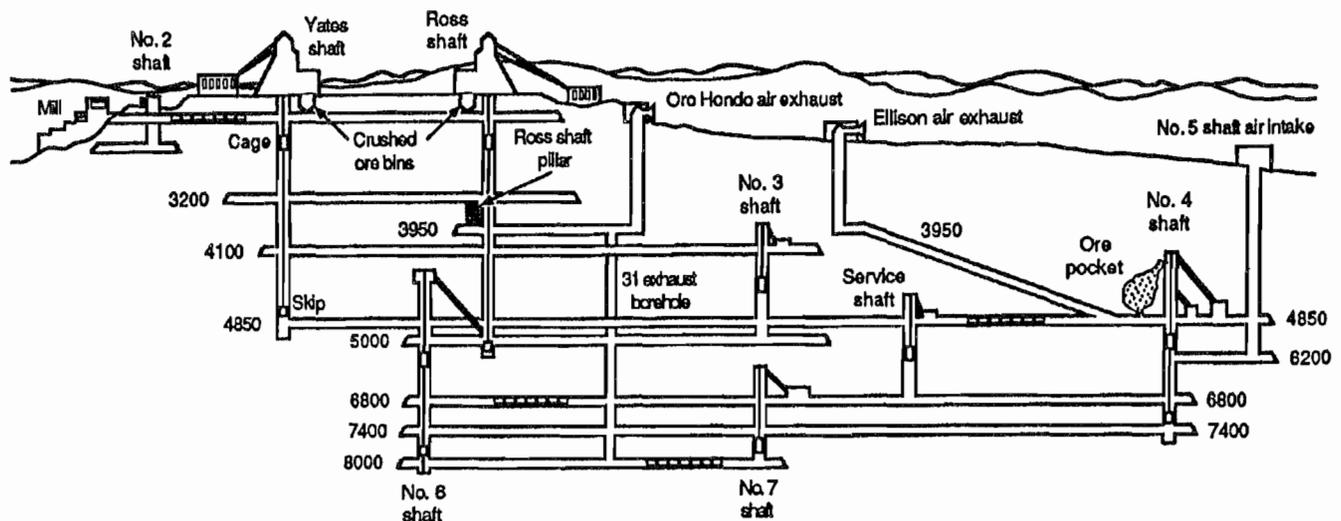
The first Report of Investigations (RI) of this series (Premining Geomechanical Modeling Using UTAH2) described the general objectives of the study, site geology, practical shaft stability criteria, and the approach taken to the problem (Pariseau and others, 1995). Also described in detail were two-dimensional, finite-element simulations of (1) historical mining leading to the present shaft pillar and (2) future mining of the ore reserve in the shaft pillar. The results indicated that the Ross shaft remained in elastic ground and that no large, catastrophic ground failures were likely. Hence, the proposed pillar mining plan was safe.

In this second RI (part 2), instrument calibration and updating of the original two-dimensional, finite-element model are described. The premining analysis focused on a plan view of the 3500 level that allowed for sequential excavation of old stopes followed by mining in the shaft pillar. This RI focuses on a vertical section that allows for sequential, lift-by-lift simulation of cut-and-fill extraction of the ore reserves in the shaft pillar and addresses several numerical modeling concerns that arose during the premining study.

The basic input data were (1) stope geometry and geological descriptions obtained from mine maps and sections, (2) in situ stresses obtained using a number of measurement techniques at various locations in the mine (Johnson and others, 1993), (3) rock and fill properties obtained from laboratory tests, and (4) the pillar mining sequence established by planning engineers.

Several types of instruments were installed at the study site. Measurements taken from borehole extensometers positioned near the plane of the section being analyzed were compared with computer estimates to validate and calibrate the numerical

Figure 1



Schematic of Homestake Mine development.

model. The results justified the use of an anisotropic, elastic-plastic material law in the model. Calibration provided the scale factors for adjusting elastic moduli and strengths to rock mass conditions at the study site. The initial scale factors for elastic and strength properties were 0.36 and 0.80, respectively, which were obtained from an earlier study of vertical crater retreat mining (Pariseau, 1985).

An important objective of the current analysis was to verify the adequacy of the rock properties scale factors and thus to test the reliability of the premining analysis very early in the shaft pillar mining project. Once calibrated, the model could then be used to analyze alternative mining sequences in the shaft pillar.

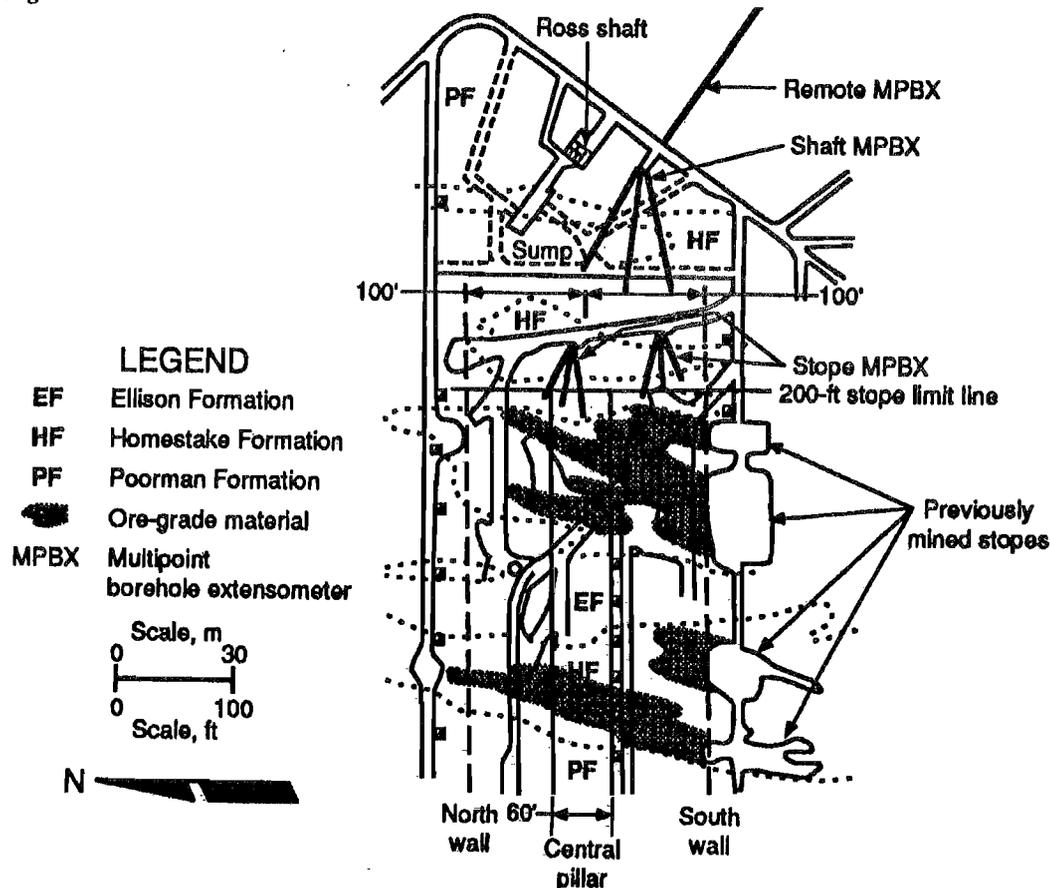
The approach to the particular problem of model calibration was (1) to install instruments prior to stoping for the purpose of measuring rock mass response to the first lifts taken in the shaft pillar and (2) to install instruments near the shaft to monitor shaft stability. Stope instruments would provide initial data for model validation, calibration, and updating. Shaft instruments would provide objective measurements of ground movement around the shaft in response to pillar mining and would also warn of any potential threat to shaft stability independently of numerical model results. This work is in support of the USBM mission to improve the safety and productivity of mining.

INSTRUMENTS

Two sets of instruments were installed, one in the stope and one in the shaft. Stope instruments consisted entirely of multipoint borehole extensometers (MPBX's). Shaft instruments were primarily MPBX's, but also included several shaft set load cells, borehole temperature sensors, strain potentiometers, and rock bolt load cells. Figure 2 shows the MPBX's in boreholes collared on the 3650 level. The remote MPBX's are directed away from the shaft into undisturbed ground.

The MPBX's in the main stope were arrayed in twin fans of three holes each that extended from the hanging wall drift toward the stopes on the north and south sides of the original shaft pillar. These sides are defined by the north and south pillar walls, illustrated in figure 2. The hanging wall drift provided access to the motor barn and connected with ramps to the shaft pillar stopes.

Figure 2



Multipoint borehole extensometers on 3650 level. Dotted lines indicate contacts between formations.

NUMERICAL QUALITY

The issue of numerical quality was addressed in some detail before conclusions were drawn from a final series of two-dimensional stope and shaft analyses using UTAH2 (Pariseau and others, 1991),³ a computer program that handles cutting and filling of initially stressed anisotropic rock masses. A generalized Hooke's law was used to relate stresses and strains in the purely elastic domain. A nonlinear, anisotropic yield condition appropriate for geologic media was employed to limit the range of purely elastic deformation (Pariseau, 1972). Associated flow rules were selected when yielding occurred and deformation progressed beyond the elastic limit into the elastic-plastic range.

Seventeen computer analyses were completed using an updated version of the original vertical section through the shaft pillar. The update was based on geologic and production planning information developed from drilling completed after the premining analysis. Figure 3 shows a portion of the mesh in the vicinity of the D-limb stope and hanging wall drift.

The 17 runs were divided into two categories. The first category consisted of "one-shot mineouts" in which the entire ore reserve was excavated in a single cut by the computer. One-shot runs save computer time and operator effort and are useful for laying out the problem. The second category consisted of sequential excavation analyses. Sequential excavation analyses follow the mining sequence and are needed to obtain estimates of instrument readings, which in turn are necessary for model calibration.

Several kinds of numerical questions relating to both categories of runs were addressed. These questions concerned the effects of mesh size, the effects of including the hanging wall drift, the amount of computational effort required, and the effects of the presence of an old stope near the shaft pillar. Results are summarized in table 2.

Mesh Size

More economical runs favor small meshes, while better quality runs favor larger meshes with boundaries well away from the excavation. Comparing run 1 with run 2 showed the desirability of enlarging the mesh by adding a border (figure 3). The calculated displacement then increased by approximately 50 pct, indicating that the original mesh borders were close enough to the excavation to affect the results significantly. This result was expected; indeed, the reason for doing the comparison run was to verify this effect.

Hanging Wall Drift

A more subtle and difficult numerical question concerned modeling extensometer response. The difficulty arose when attempting to represent the small hanging wall drift containing the extensometer hole collars and the much larger stopes of the shaft pillar in the same mesh. Mesh refinement suitable for a drift-size excavation would lead to an enormous number of elements when extended over the entire mesh, while mesh refinement suitable for the stopes would not allow accurate

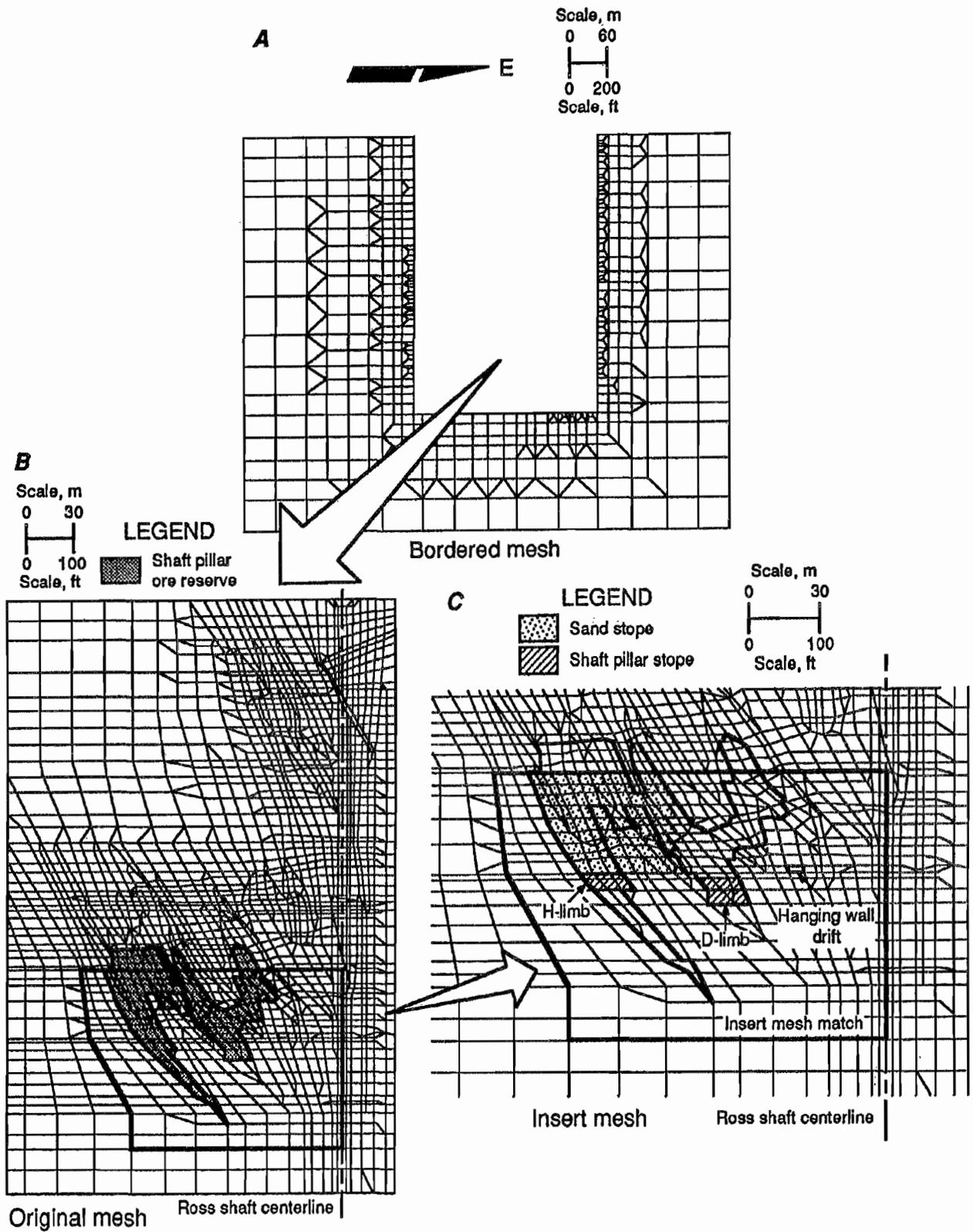
³ See also Pariseau, W. G. Interpretation of Rock Mechanics Data (Volume 2) (A Guide to Using UTAH2). Contract HO220077, Univ. UT. USBM OFR 47(2)-80, 1978, 41 pp.

Table 2.—Finite-element runs and displacement estimates

Run	File name	Description	Displacement ¹	
			mm	in
ONE-SHOT				
1	Original	Original report run, old mine geometry, no border.	-4.0	-0.158
2	Owbor	Old mine geometry, with border (border effect).	-5.9	-0.232
3	Borone	New mining geometry with border (stope geometry effect).	13.7	0.539
4	Cut1	1st lift without hanging wall drift.	-3.0	-0.117
5	Hwcut	1st lift with hanging wall drift.	1.4	0.056
6	Bcut	2 lifts, 1st and bottom lift, with hanging wall drift.	14.8	0.584
7	Bcut>hafmat	Same as run 6, but at 1/2 original strength.	15.9	0.624
8	Sand0	Excavation of old sand-filled stope 9 m (30 ft) south of shaft center.	-4.1	-0.161
9	Sand1	Sand stope plus 2 lifts without hanging wall drift.	12.1	0.475
10	Sand2	Sand stope plus 2 lifts with hanging wall drift.	17.1	0.675
11	Qsand1	Same as run 9, but with 2 times load steps and iterations (computational effort).	12.1	0.475
SEQUENTIAL				
12	Seq>bcut	Sand stope, 2 lifts, without hanging wall drift (sand stope first, lifts next).	16.1	0.632
13	Seq>cut12	Same as run 12, but at 1/2 strength.	17.7	0.698
14	Seq>hwbcut	Sand stope, 2 lifts, with hanging wall drift.	21.1	0.832
15	Seq>hwcut12	Same as run 14 but at 1/2 strength.	22.8	0.898
16	Seq>hlmb>bcut	Same as run 14 without hanging wall drift.	16.0	0.628
17	Seq>hlmb>qtrbcut	Same as run 16 at 1/4 strength.	20.4	0.803

¹Relative displacement between anchor point and collar parallel to an extensometer hole in the plane of the analysis. A positive displacement implies elongation or stretching of the hole.

Figure 3



Finite-element mesh of a vertical section through Ross shaft pillar. A, Border mesh surrounding analyzed region; B, mesh of analyzed region (Ross shaft centerline); C, detail of B showing insert mesh match.

modeling of the much smaller hanging wall drift. A compromise was to model the hanging wall drift with elements available in the stope-scale mesh.

Although the modeling was coarse, the usefulness of the compromise was seen when run 4 was compared with run 5 (table 2). The sign of the displacement was reversed, and there was a noticeable change in magnitude when the hanging wall drift was modeled even crudely. Comparisons of run 9 and run 10 showed an increase of about 40 pct when excavation of the hanging wall was included in the stope-scale mesh (figure 3). Sequential runs 14 and 16 (table 2) showed the same effect—an increase of more than 30 pct when excavation of the hanging wall drift was modeled.

These results indicate that extensometer boreholes should be collared in openings that can be represented in the numerical model of choice. If the excavation containing the collar is not represented in the model, even if only crudely, then comparisons of model results with instrument readings will be suspect. Mine instruments should be located with regard not only to convenience of access, but also to model requirements and constraints. Instrument planning and modeling should be done in concert, so that more comparisons can be made between mine measurements and model calculations.

Computational Effort

Comparing run 11 with run 9 showed that the computational effort and convergence obtained were satisfactory. No change in calculated displacement occurred in the third significant digit when the computational effort was quadrupled (double load steps and double iterations).

MODEL CALIBRATION

A two-pass, insert mesh technique was used for model calibration. This technique allowed mesh refinement in the area of greatest interest, that is, in the region between the stope hanging wall and the hanging wall drift. Figure 4 shows the refined insert mesh in relation to the updated and enlarged parent mesh. Mesh refinement in the region between the stope hanging wall and hanging wall drift was particularly desirable for estimating the extent of yielding around stope extensometer collars and downhole anchor points.

The most important results from the insert mesh were (1) obtaining the estimated extensometer readings necessary to determine the scale factor for elastic properties and (2) determining the extent of yielding in the vicinity of the stope extensometers, which was used to determine the scale factor for the strength properties.

ELASTIC PROPERTIES SCALE FACTOR

The relative displacements between the anchor and collar points in holes E14, E15, and E16 (figure 5) amounted to about 3.8 cm (1.5 in) at day 440, whereas in holes E17, E18, and E19, relative displacement came to about 2.5 cm (1 in) (figure 6).

Old Stope

A limitation of two-dimensional analysis is that stopes excavated along strike, and thus out of the plane of analysis (perpendicular to strike), cannot be rigorously taken into account. Although old stopes that were completed before installation of extensometers cannot directly affect subsequent readings, there is an indirect effect in that the rock mass becomes more deformable and compliant. An example is shown in figure 3, where a large, sand-filled stope near the south wall of the shaft pillar was projected onto the analysis plane. Comparing run 10 with run 6 (table 2) showed that this indirect effect was noticeable and increased the calculated displacement by approximately 16 pct.

Strength Reduction

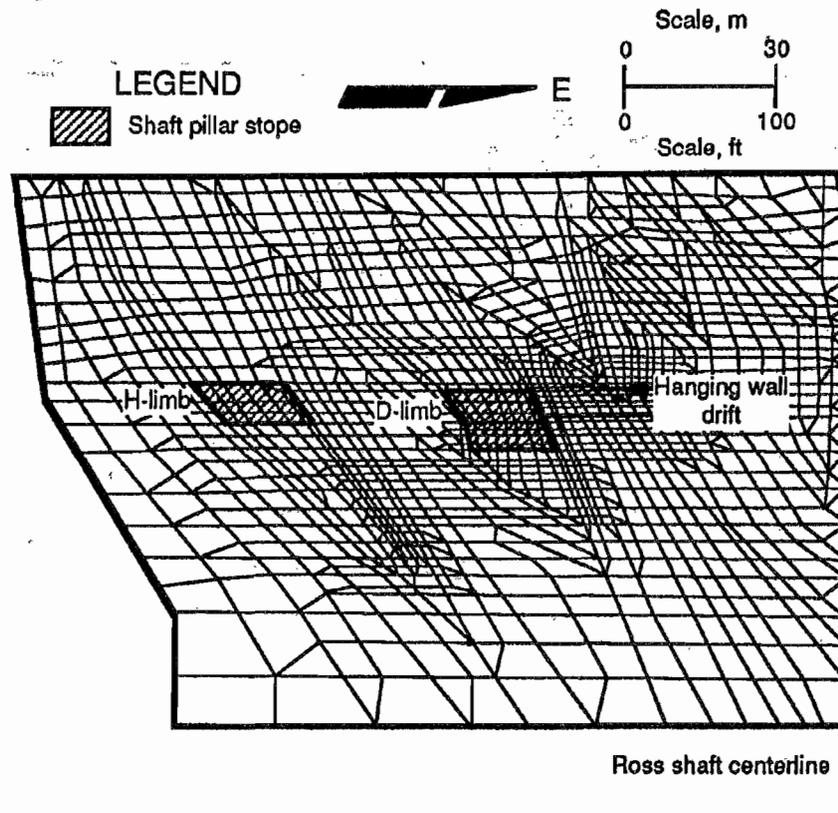
The potential effect of a 50-pct reduction in strength on calculated extensometer readings can be judged by comparing run 14 with run 15, and run 12 with run 13. Extensometer estimates were increased by 8 to 10 pct, depending on how the hanging wall drift was modeled at the point where the actual instrumentation holes were collared. Comparing run 16 with run 17 shows that the effect of strength reduction was nonlinear. An additional 50-pct reduction (quarter strength) resulted in a more than 27 pct increase in estimated readings relative to the full-strength estimate. The nonlinearity was a consequence of yielding in the stope hanging wall and in the skin of the hanging wall drift.

Figures 5 and 6 also show that the two anchor points in each hole appeared to move about the same amount; thus, relative displacements between these anchor points were small. Approximately 16 weeks after completion of installation (November 1987) and approximately 10 weeks after mining began, most of the extensometers showed somewhat more than 2.5 cm of displacement. Some anchors were lost. Small rock falls, blasting, and wire breakage were possible causes. These actual displacements compared well with the largest calculated displacement of 2.3 cm (0.9 in).

Extensometers E14, E15, E16, and E19 showed relative displacements of about 0.064 cm (0.025 in) between downhole anchor points 1 and 2. Finite-element results indicated about 0.023 cm (0.009 in) of relative displacement. E17 and E18 showed about 0.318 cm (0.125 in) of relative displacement between anchor points 1 and 2, while the corresponding result from the finite-element analysis was about 0.706 cm (0.278 in). E12 and E20 showed similar behavior, that is, a large relative displacement between the collar and the bottom anchor point, but a small relative displacement between anchor points 1 and 2 in the hole.

The large displacements of anchor points down the hole relative to the collar points and the small displacements

Figure 4



Refined insert mesh for two-pass calibration analyses.

between anchor points in the hole indicated an elastic response downhole and an inelastic response near the hole collars. The inference was that stress concentrations in the periphery of the hanging wall drift were sufficient to cause localized yielding and displacement in excess of that expected on the basis of a purely elastic response. Displacements relative to collar points could not, therefore, be used to determine the scale factor for elastic moduli. However, relative displacements between downhole anchor points can be used if they are within the elastic range. Since the results from the finite-element analysis straddled the few applicable measured results, there was no indication that a change in the assumed scale factor (0.36) for the elastic moduli was necessary or justified.

STRENGTH PROPERTIES SCALE FACTOR

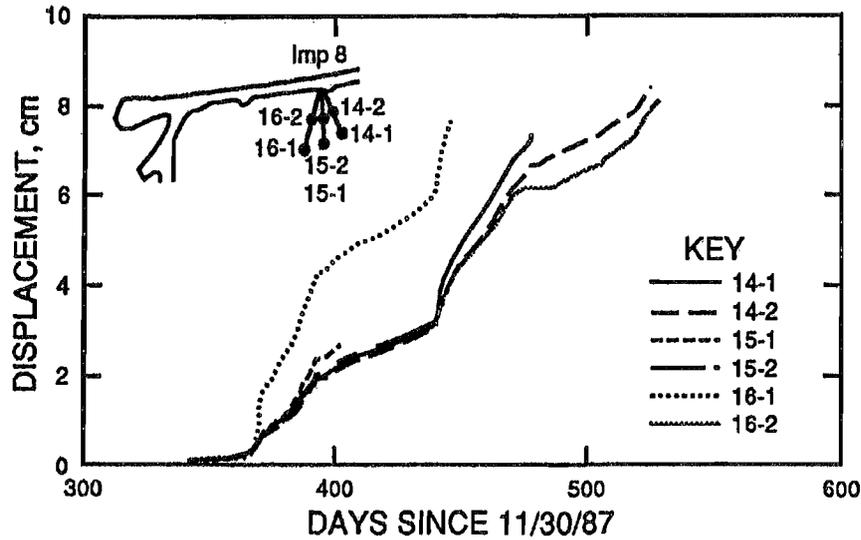
The scale factor for strength properties was determined by systematically reducing the strengths of the finite-element model until the extent of inelasticity or yielding in the model matched the inelasticity observed. In fact, two series of finite-element analyses were done, one with and one without the sand stope. Both were done using the two-pass, insert mesh technique. The first run simulated an excavation step on a relatively coarse mesh; the second simulated the same excavation step using the refined insert mesh. The excavation sequence itself represented the major stope cuts taken in the shaft pillar. Strength scale factors of 0.8, 0.6, 0.4, and 0.3 were used in both series.

Observations at the study site indicated that localized yielding occurred in the hanging wall drift as a consequence of stoping in the shaft pillar. Figure 7 shows the physical appearance of the hanging wall drift. A minimum requirement for a finite-element model is that no yielding occur with the development cut that excavates the hanging wall drift. Yielding near the hanging wall drift should occur only with stope cuts and should be localized. Some yielding in the stope model was justified because of several rock falls in the stopes and the loss of several deep extensometer anchor points. In this regard, the orientation of foliation had a noticeable effect on the yielding pattern. A 60° dip was used in all analyses. This angle was representative of the general orientation of the major stratigraphic units at the study site (Poorman, Homestake, and Ellison Formations), which are anisotropic (orthotropic).

There was no yielding in any of the finite-element analyses as a result of excavating the development cut in the hanging wall drift, nor did yielding occur near the hanging wall drift as a result of excavating stope cuts on the H- and D-limbs at strength scale factors of 0.8 and 0.6, respectively, in the series without the old sand-filled stope. Some yielding occurred at 0.4; more extensive yielding occurred in the shaft pillar stope hanging wall at a strength scale factor of 0.3.

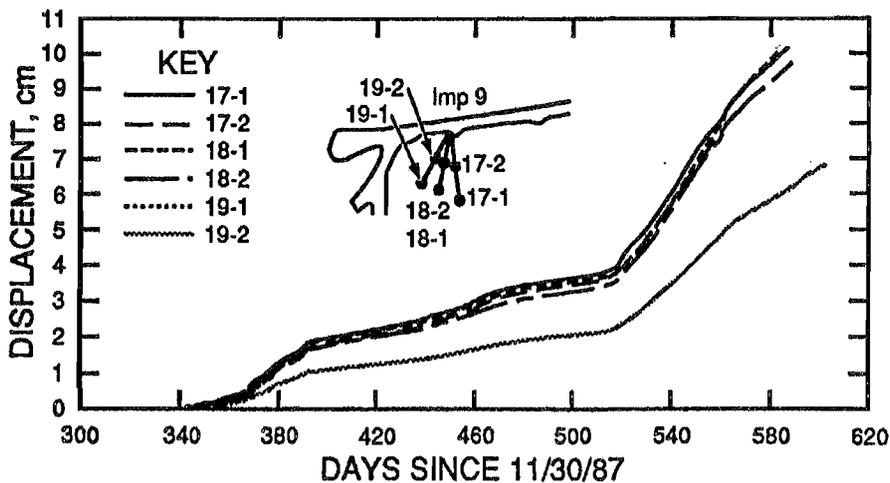
The inclusion of the sand stope in the analyses resulted in yielding in the stope hanging wall at all strength scale factors. Figure 8 shows the extent of yielding between the hanging wall drift and the shaft pillars stope with the sand stope present.

Figure 5



Recorded slope displacements, extensometers E14, E15, and E16. Imp = individual measurement pod.)

Figure 6



Recorded slope displacements, extensometers E17, E18, and E19. (Imp = individual measurement pod.)

For direct comparison, the calculated extent of yielding without the sand slope is also shown in figure 8 using a strength reduction factor of 0.6.

The justifiable amount of strength reduction depends on the importance given to the influence of the sand slope. A relatively small value of 0.3 might be justified if the sand slope had no influence on present stoping. A large value would be justified under the greater influence of the sand slope. Some intuitive guidance in the matter can be found in considering the strain-to-failure and the strain energy density of a uniaxially loaded test specimen. Strain-to-failure is simply the ratio of unconfined compressive strength to Young's modulus. If the strain-to-failure as a dimensionless quantity is considered to be independent of

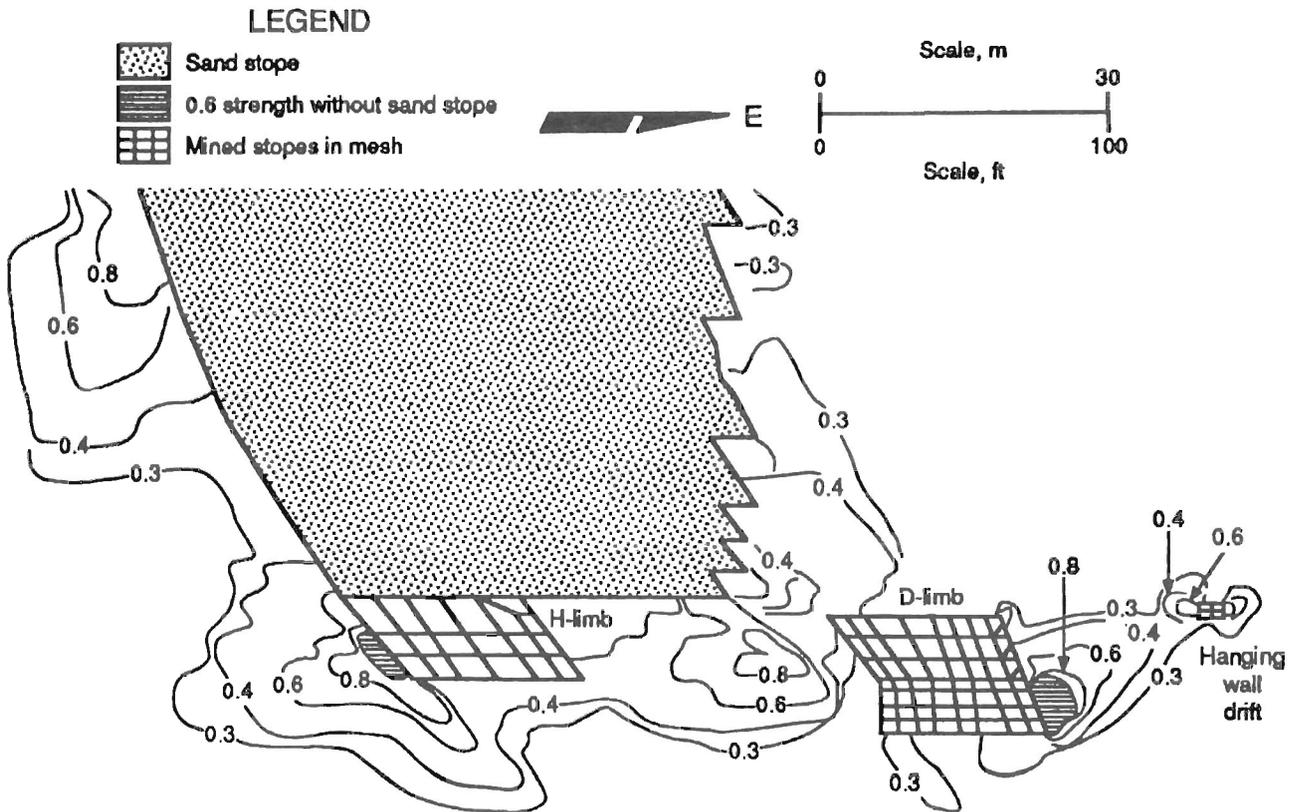
scale, then the strength and modulus scale factors should be equal. This suggests a strength scale factor of 0.36. However, if the strain energy density at failure is considered scale invariant, then the strength scale factor is equal to the square root of the modulus scale factor. This suggests a strength scale factor of 0.6, since the modulus scale factor is 0.36. These two criteria were in close agreement with the range of strength scale factors suggested by the finite-element results. Considering the large size and nearness of the old sand-filled slope to the shaft pillar stopes and the more appealing energy scaling rule, the strength scale was reduced to 0.6 from the original value of 0.8 for future analyses.

Figure 7



Sloughing on stope side of hanging wall drift. Shaft side remains intact. Note pronounced foliation.

Figure 8



Extent of yield zone as a function of strength scale factor.

SHAFT ANALYSIS RESULTS

The important results from the shaft analysis were the displacement estimates and the finding that there was *not* a potential for large-scale yielding and catastrophic rock mass motion near the Ross shaft. A quantitative index to evaluating safety and stability is the factor of safety, which is the ratio of strength to stress. Yielding is associated with a safety factor of 1. Safety factors greater than 1 indicate elastic ground.

SHAFT WALL DISPLACEMENTS

The maximum horizontal and vertical displacements along the Ross shaft are presented in table 3. The most important runs were the first three. Run 1 was from the original analysis, while run 2 was the same but used an enlarged mesh for greater

Table 3.—Maximum shaft wall displacement

Run	Vertical ¹		Horizontal ²	
	mm	in	mm	in
ONE-SHOT				
1 ...	10.7	0.42	-12.7	-0.50
2 ...	6.9	0.27	-19.3	-0.76
3 ...	16.8	0.66	-38.9	-1.53
4 ...	0.5	0.02	-2.5	-0.1
5 ...	0.8	0.03	-0.5	-0.02
6 ...	0.5	0.02	-0.8	-0.03
7 ...	1.3	0.05	-1.8	-0.07
8 ...	4.6	0.18	-9.4	-0.37
9 ...	6.1	0.24	-11.9	-0.47
10 ..	6.4	0.25	-11.9	-0.47
11 ..	6.4	0.25	-9.9	-0.39
SEQUENTIAL				
12 ..	2.0	0.08	-1.8	-0.07
13 ..	2.01	0.08	-2.0	-0.08
14 ..	2.0	0.08	-2.0	-0.08
15 ..	2.3	0.09	-2.0	-0.08
16 ..	2.0	0.08	-1.8	-0.07
17 ..	2.3	0.09	-2.3	-0.09

¹Positive values indicate settling.

²Negative values indicate movement toward stope.

accuracy. Runs 1, 2, and 3 involved excavation of the entire ore reserve.

Shaft wall displacement was thus an estimate of what the total displacement would be several years in the future after the shaft pillar had been mined. The updated analyses using the new mining geometry indicated a maximum horizontal displacement of about 3.9 cm (1.5 in) and a vertical displacement of 1.7 cm (0.7 in).

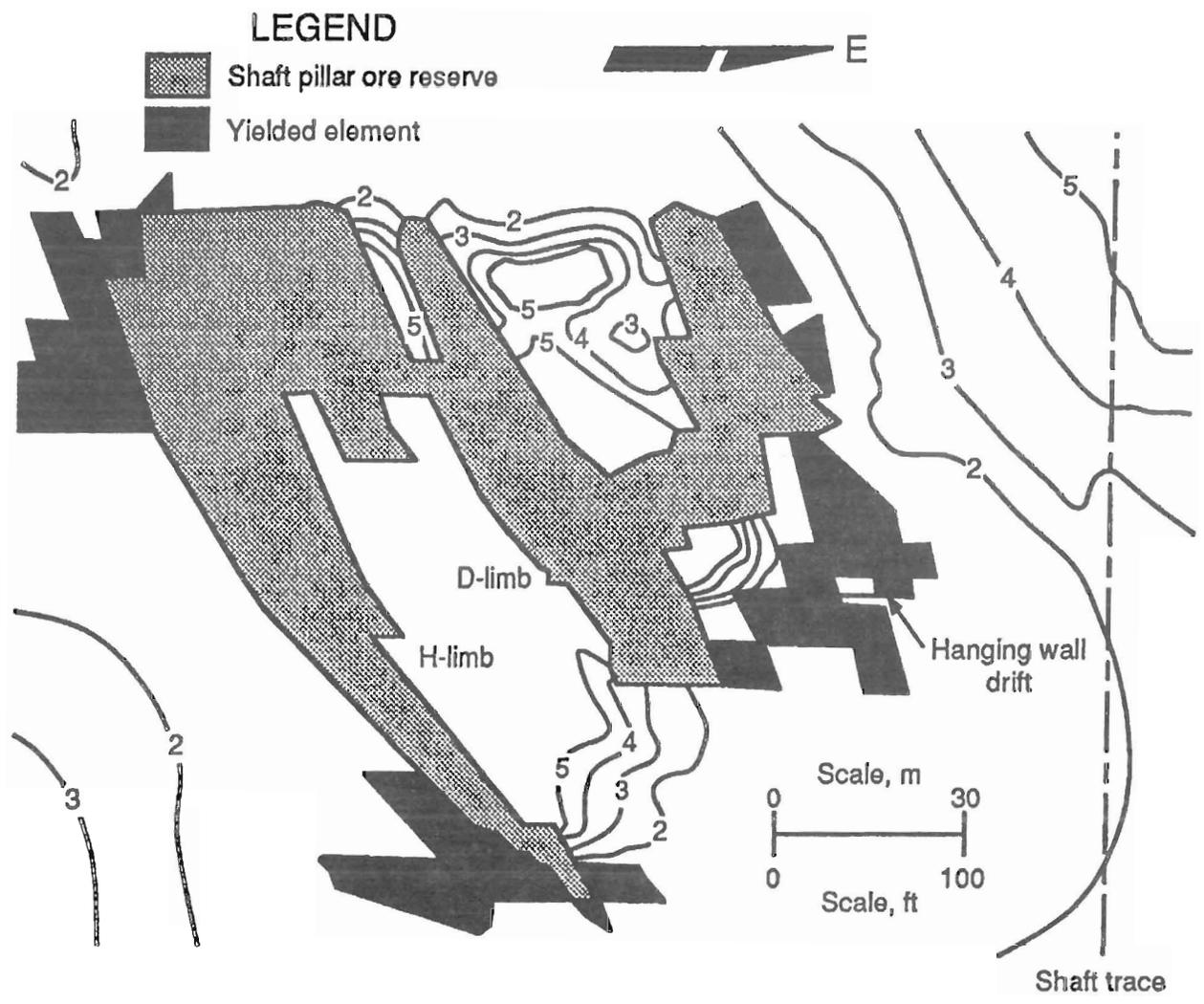
SHAFT WALL YIELDING AND SAFETY FACTOR

The extent of yielding after the entire ore reserve in the shaft pillar had been mined is shown in figure 9. The pattern of yielding was similar to that observed earlier. Yielding first occurred on the footwall side of the stoped area in the upper left of the excavation and also in the hanging wall at the toe of the stoped area, where another sharp corner was present in the mesh. Yielding was confined to the stope walls and did not extend to the shaft centerline.

Contours of the local factors of safety are also shown in figure 9, rounded to the nearest whole integer. The safety factor along the shaft centerline varied from 2 to 5. However, because the shaft cannot be explicitly represented in the plane of analysis, stress concentration effects at the shaft wall were absent, meaning that the actual safety factors would be less. Hence there was the possibility that small-scale yielding at the shaft wall would not be revealed by the analysis. Small-scale yielding in the skin of the shaft wall would be handled by ordinary ground control measures and would not be expected to pose a threat to shaft operations.

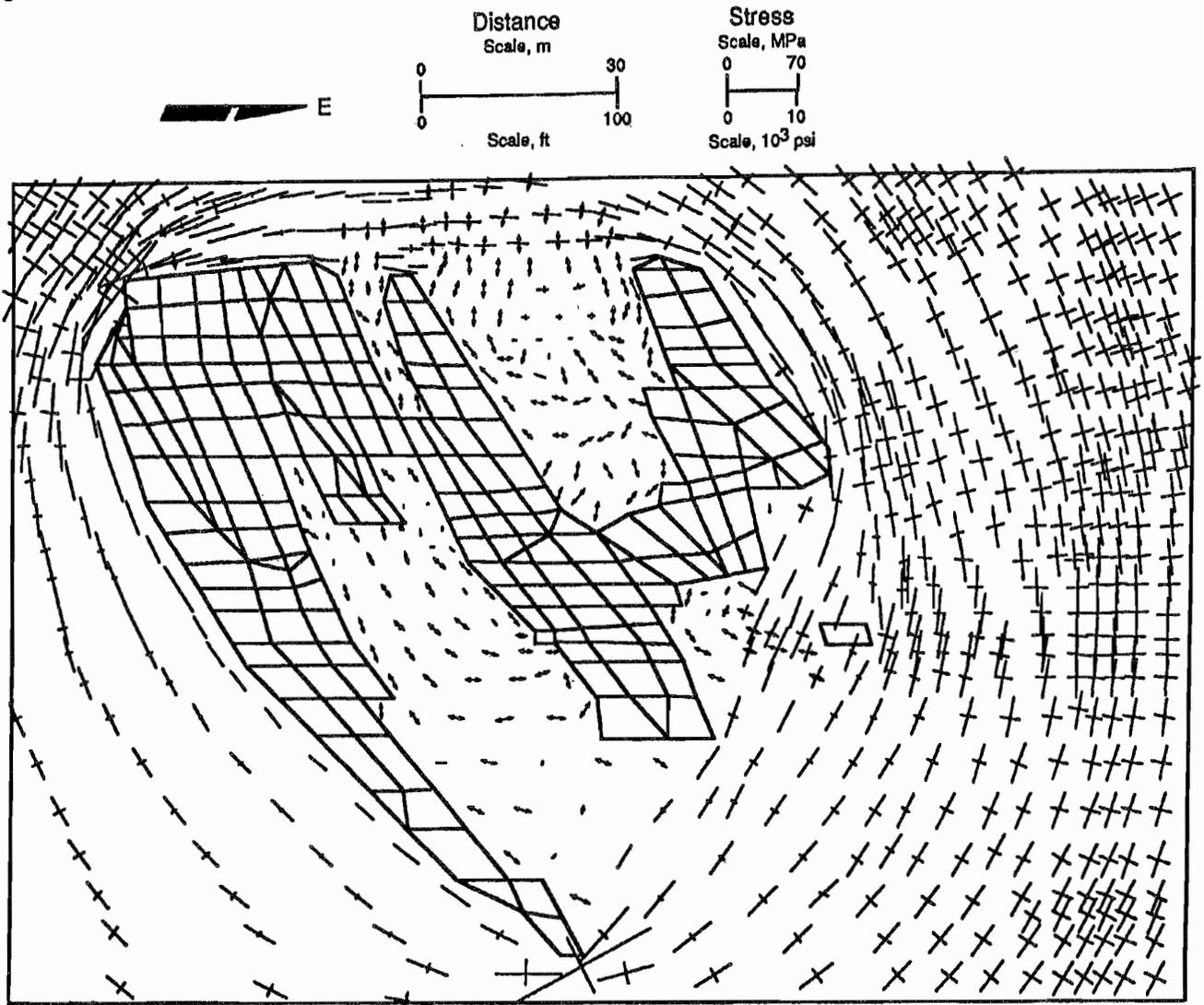
The distribution of principal stresses after the ore reserve was mined is shown in a window of the mesh in figure 10. An arrowhead indicates tension. Comparing data in figures 9 and 10 shows that the zones where the safety factor was relatively low were associated with compressive stress states. High safety factors were present in zones of induced tension.

Figure 9



Window of mesh showing extent of yielding after ore reserve in shaft pillar has been mined. Safety factor contours also shown.

Figure 10



Distribution of principal stresses in window plot of mesh.

SUMMARY AND CONCLUSIONS

Advances in rock mechanics and mine design suggest that it is possible to devise methods for recovering valuable ore left in shaft pillars in hard-rock mines. A case study involving a shaft pillar in high-grade ore near the Ross shaft at the Homestake Mine demonstrated the usefulness of this new technology. A two-dimensional, premining stability evaluation (part 1 of this three-part study) indicated that the shaft remained in elastic ground as the ore reserve was mined. Large-scale ground motion and the potential for catastrophic failure were not indicated. Thus, the effects of pillar mining did not appear to pose a threat to shaft stability.

Subsequent definition drilling and detailed mine planning allowed the construction of more accurate models of geology and stope geometry in the shaft pillar. Installation of borehole extensometers in the hanging wall of the stope in advance of mining provided data for a check on elastic moduli and strength scale factors assumed from a previous study at the mine. Scale factors adjust values of rock properties obtained in a laboratory to values obtained from a rock mass and are essential for calibrating models and for reliable design analysis.

A number of concerns about modeling were successfully addressed in a series of two-dimensional analyses of a vertical section through the center of the shaft pillar. Questions of mesh size, computational effort, and effects of out-of-plane stopes were considered. Modeling of small development openings,

such as a new hanging wall drift, and mesh refinement were also examined in order to maintain numerical quality and reliability of the analysis.

Comparisons of extensometer readings taken during the initial stages of mining with calculated readings using the updated mesh showed that some displacements were overestimated while some were underestimated. Thus, although the data were quite limited, there was no indication that the elastic moduli scale factor of 0.36 should be changed.

A systematic reduction of strength using scale factors ranging from the original value of 0.8 through 0.3 was used in the updated model in conjunction with a two-pass computational technique to determine a scale factor for rock mass strength at the study site. The two-pass technique allows one to use a refined mesh in the vicinity of the first stopes in the shaft pillar and to follow the progress of yielding in greater detail. Comparisons of the extent of yielding in the finite-element model with that observed at the study site indicated that a strength scale factor of 0.6 was appropriate.

Subsequent updated and calibrated two-dimensional, finite-element models showed that the Ross shaft remained in elastic ground and substantiated the conclusion from the original premining study that shaft pillar mining did not pose a serious threat to shaft safety. No large, catastrophic movement of ground near the shaft was indicated.

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