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REPORT OF INVESTIGATIONS/1992

Rock Mechanics Investigations at the Lucky Friday Mine

(In Three Parts)

3. Calibration and Validation of a Stope-Scale Finite-Element Model

By W. G. Pariseau, J. K. Whyatt, and T. J. McMahon

UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

deg	degree	pct	percent
ft	foot	psi	pound per square inch
in	inch		

ROCK MECHANICS INVESTIGATIONS AT THE LUCKY FRIDAY MINE

(In Three Parts)

3. Calibration and Validation of a Stope-Scale, Finite-Element Model

By W. G. Pariseau,[1] J. K. Whyatt,[2] and T. J. McMahon[2]

ABSTRACT

The U.S. Bureau of Mines has been conducting a series of rock mechanics investigations at the Lucky Friday Mine in the Coeur d'Alene Mining District of northern Idaho. The present study uses rock deformation measurements to validate and calibrate a geomechanical model of the experimental Lucky Friday underhand longwall (LFUL) stope on the 5,100-ft level of the Lucky Friday Mine, Mullan, ID. The calibrated three-dimensional, finite-element model and measurements of rock mass displacement show a correlation of 0.795. The calibration reduces rock mass modulus to 14 to 17 pct of laboratory values and rock mass strength (with an energy rule) to 37 to 41 pct of laboratory values. This model suggested that stress-related ground control problems would be encountered west of the stope where the vein makes a right-angle turn and then splits. Rock burst activity was concentrated in this area during mining of the LFUL stope.

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INTRODUCTION

Rock bursts have a long history in the Coeur d'Alene Mining District of northern Idaho (*5*)[3] and are threatening the future of district mines. The U.S. Bureau of Mines has long been involved in developing alternatives to the traditional overhand cut-and-fill stoping method (fig. 1*A*) that would reduce rock burst hazards and be amenable to mechanization. One of these methods, the underhand longwall cut-and-fill method (fig. 1*B*), was chosen for testing at the Lucky Friday Mine, Mullan, ID (fig. 2). The experimental stope, dubbed the Lucky Friday underhand longwall (LFUL) stope (fig. 3), was tested under a cooperative agreement among the Bureau, Hecla Mining Co. of Coeur d'Alene, ID, and the University of Idaho at Moscow, ID. Additional information on stope design and operation have been reported by Werner (*14*) and Noyes, Johnson, and Lautenschlaeger (*6*), respectively.

The principle of a single advancing face, which is central to the underhand longwall method, is not new. In fact, the South African High-Level Committee on Rock Bursts and Rock Falls recommended the use of longwalls as a means of reducing rock burst hazards associated with mining remnants (or sill pillars) as early as 1924 (*3*). Longwall mining is now standard practice in the deep gold mines of South Africa. However, other mechanisms of rock bursting, such as shear fracturing and slip, have been identified in the Coeur d'Alene District (*4*), where the greatest premining stress is horizontal compression (*15*).

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

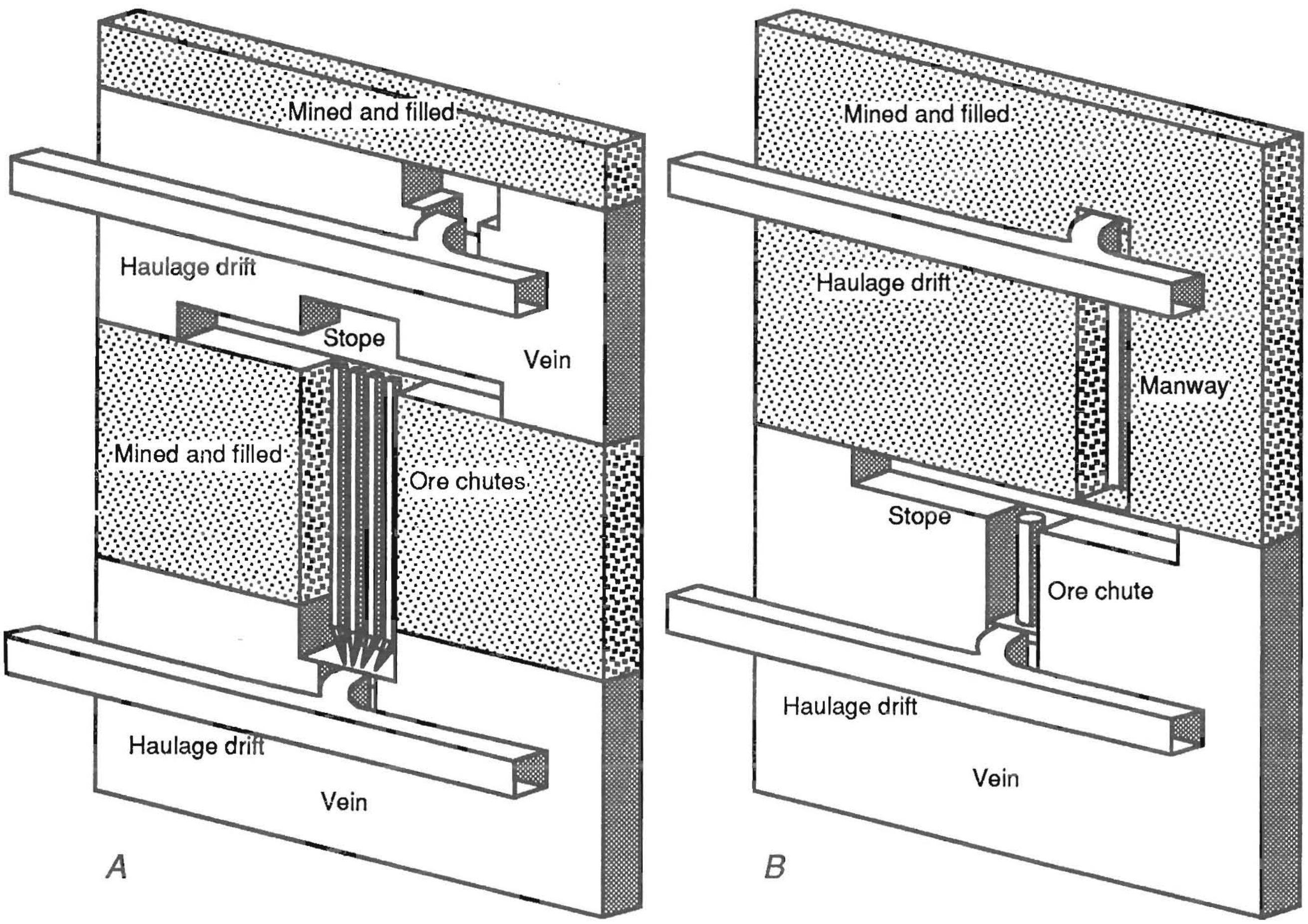

Figure 1.—Schematic of cut-and-fill mining methods. *A*, Overhand; *B*, underhand.

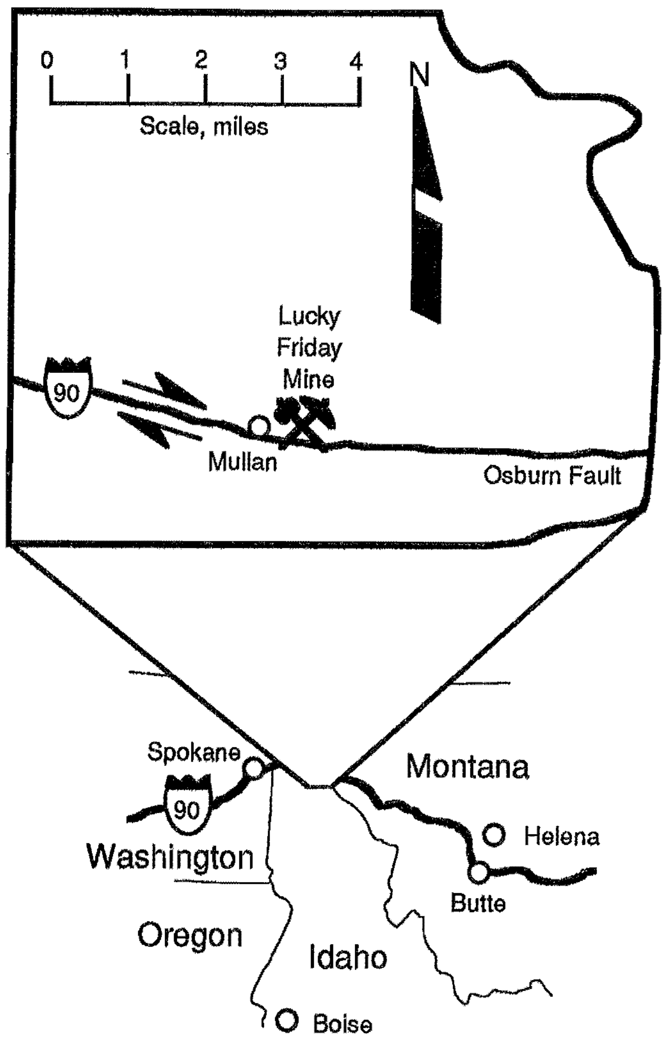

Figure 2.—Location of Lucky Friday Mine.

Recent research (*17*) on explicit modeling of shear fractures has also suggested adopting the underhand longwall method to reduce rock burst hazards.

Optimization of stope and stope sequence designs, as well as investigations of new mining methods, requires some anticipation of how the rock mass will respond to mining. This can be achieved with a calibrated finite-element model. Both two- and three-dimensional versions of the LFUL stope model were formulated with the finite-element programs UTAH2 and UTAH3, respectively. The two-dimensional model is based on a plan view of the LFUL stope and is used to calculate conditions before and after mining. The three-dimensional model allows for sequential excavation and analysis of the LFUL stope as it is mined at depth; this model can readily simulate increments of excavation as monitored by various instruments. Both models ignore the effects of backfill, which was examined in the second Report of Investigations (RI) of this series (*18*).

Model validation and calibration consisted mainly of two comparisons, or back-analyses. The first comparison was between displacements measured in the mine near the study stope (presented in the first RI of this series) (*19*) and displacements calculated from output of the finite-element model. The second comparison was between plastic zones in the model and estimates of the extent of yielding ground near the stope as determined through observation. In combination, the two comparisons allow not only for validation of the model, but also for establishing scale factors for elastic and strength properties that can be used to anticipate rock mass response to alternative stope sequences and designs.

Success of the back-analysis procedure in developing a valid model depends on selecting an appropriate material model and the availability of adequate measurements of ground movement. These measurements in turn depend on a well-planned and successfully implemented instrumentation program. Validity can be measured in part by comparing a comprehensive set of field data with predictions from the calibrated model. Unfortunately, problems such as instrument failure often result in a data base that is incomplete or too small to allow reliable statistical analyses or model calibrations. However, when the data base is adequate, excellent validations have been achieved, for example, at the Carr Fork Mine (*8*), at the Homestake Mine (*12*), and in earlier studies of conventional overhand cut-and-fill stoping at the Lucky Friday Mine (*2*, *7*, *9-10*).

ACKNOWLEDGMENTS

Close cooperation of the Hecla Mining Co. and the University of Idaho is gratefully acknowledged. Fred Brackebush, manager of mining research for Hecla (now president of Mine Systems Designs, Kellogg, ID), was instrumental in organizing research efforts. Dave Cuvelier and Mike Werner, mining engineers, Hecla Mining Co., coordinated mine access. The contributions of Bureau of Mines staff, including Doug Scott, geologist, and Mark Board, mining engineer (now with Itasca Consulting Group), were invaluable to planning and monitoring the instrumented stope.

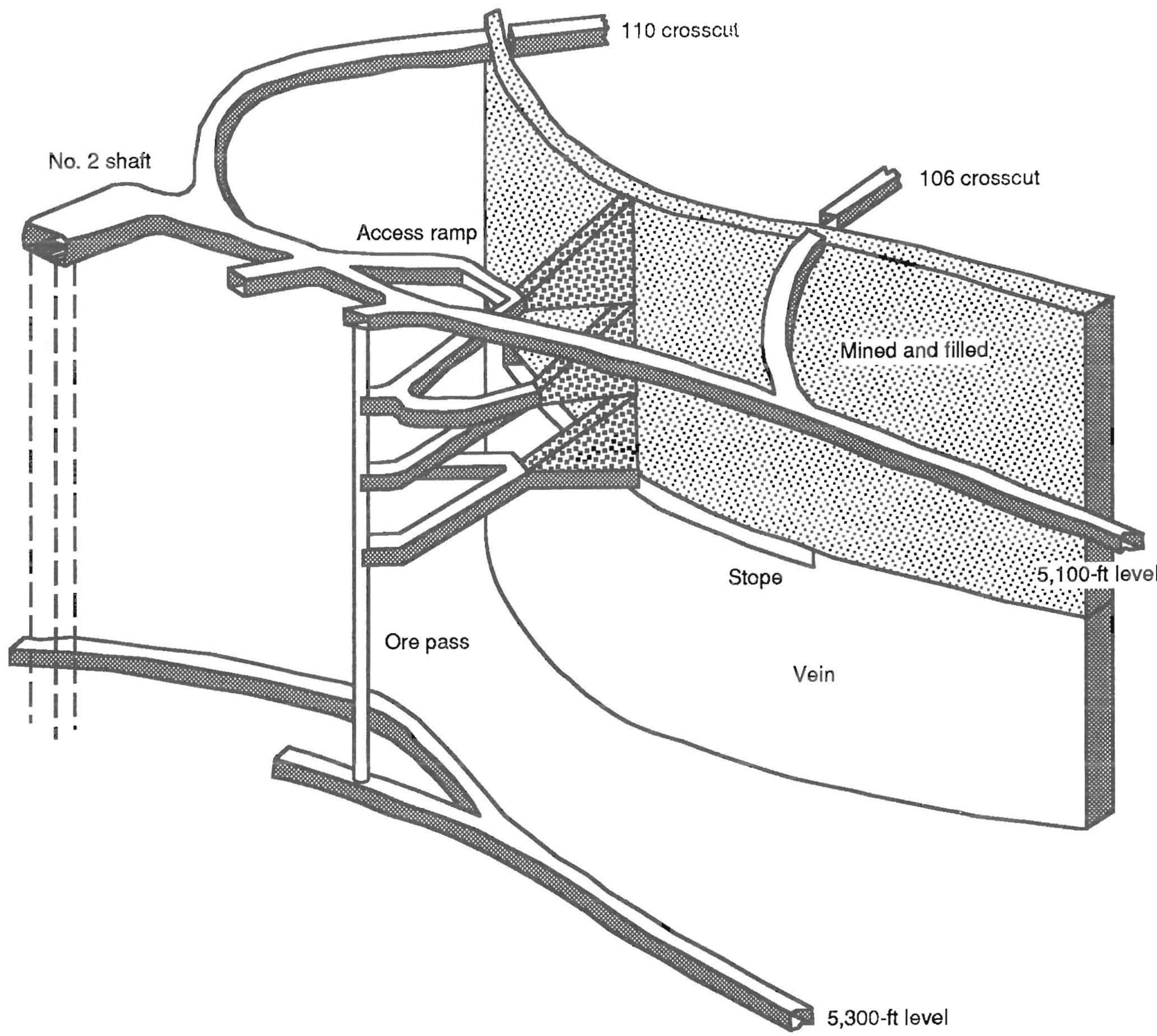

Figure 3.—Underhand longwall mining method as implemented in LFUL stope.

ROCK MASS BEHAVIOR

Construction of a numerical model of rock mass behavior around the LFUL stope required mathematical definition of the characteristics of rock mass behavior. These characteristic behaviors were evaluated by the model constitutive law on the basis of several parameters, including rock mass loading, distribution of various rock types, shape of mine openings, rock strength, and rock deformability.

CONSTITUTIVE LAW

The character of rock mass response to mining is defined by a constitutive law. In this model, an elastic-plastic constitutive law was assumed; that is, the rock was assumed to behave as an elastic, isotropic material until loading exceeded strength, at which point the material acted as a perfectly plastic material. This constitutive law has been successfully used in several studies of rock mass response to mining, including studies of conventional overhand cut-and-fill stoping at the Lucky Friday Mine (*2, 7, 9-10*) and elsewhere (*8, 12*).

However, time-dependent phenomena may also contribute to rock movement, especially at elevated stress states near the elastic limit (*16*). At the Lucky Friday Mine, although field data indicated that most rock mass movement occurred with excavation, some time-dependent deformation was observed during an 8-month idle period. This relatively small amount of time-dependent deformation suggested that while time-dependence can be safely ignored for temporary mine openings, it could become a factor in permanent mine openings. Because the stope is a temporary mine opening, time-dependence was ignored in this study.

ROCK MASS LOADING

Loading is applied primarily by in situ stresses that may be concentrated by nearby mine openings. Generally, these openings were included in the model, and the in situ stress state could be applied as a boundary condition far from mine openings. In this study, the premining stress state was determined from a review of overcore stress measurements in the Coeur d'Alene Mining District (*15*). The magnitude and orientation of the principal stresses are shown in table 1. The cartesian components of stress relative to compass and finite-element coordinates are shown in table 2.

Table 1.—Principal stresses used in LFUL stope study, pounds per square inch

(Compressive stresses are negative)

Stress	Magnitude	Dip,[1] deg	Azimuth,[2] deg
σ_1	-5,753	90	0
σ_2	-5,886	0	50, N50E
σ_3	-7,695	0	140, N40W

[1]Dip = angle down from horizontal (positive).
[2]Azimuth = angle clockwise from north (positive).

Table 2.—Premining stress components used in LFUL stope study, pounds per square inch

(Compressive stresses are negative)

Stress	Type	Direction	Magnitude
σ_{xx}	Normal	East-west	-6,632
σ_{yy}	do.	North-south	-6,947
σ_{zz}	do.	Vertical	-5,753
γ_{xy}	Shear	East-north	891
γ_{yz}	do.	North-vertical	0
γ_{zx}	do.	Vertical-east	0

ROCK TYPES

There are three major rock types in the study stope region: vitreous quartzite, which is the most common, sericitic quartzite, and argillite. The ore is found in a mineralized quartz vein. Laboratory rock properties are shown in table 3; elastic moduli and strengths were estimated by Pariseau and Moon (*11*) on the basis of studies by Chan (*1*). All rock types were considered isotropic, and no directional characteristics were assumed. Consequently, shear modulus, G, was directly related to Young's modulus, E, and Poisson's ratio, ν. Compressive strength was considered to be a nonlinear function of confining pressure with shear strength, R_o, directly related to tensile strength, T_o, and unconfined compressive strength, C_o.

Standardized laboratory tests provided starting values for a model analysis. Defects in a rock mass generally make the rock mass more deformable and weaker than the rock in intact laboratory test specimens, so rock mass elastic moduli and strengths are generally less than values determined by laboratory testing.

A study of stope geology and seismicity by Scott (*13*) was used to position the vein and sericitic quartzite in the footwall (fig. 4). A summary of LFUL stope geology is also included in the first report of this series (*19*).

Table 3.—Laboratory values for rock properties, pounds per square inch

Rock type	$E \times 10^6$	ν	$G \times 10^6$	C_o	T_o	R_o
Vitreous quartzite ..	7.300	0.29	2.830	32,500	2,600	5,307
Sericitic quartzite ..	1.550	.27	.610	11,400	1,500	2,387
Quartz vein	1.650	.28	.640	15,100	1,550	2,793

E = Young's modulus.
C_o = unconfined compressive strength.
G = shear modulus.
ν = Poisson's ratio.
R_o = unconfined shear strength.
T_o = tensile strength.

NOTE: $G = E/[2(1+\nu)]$, $R_o = [C_oT_o/3]^{1/2}$.

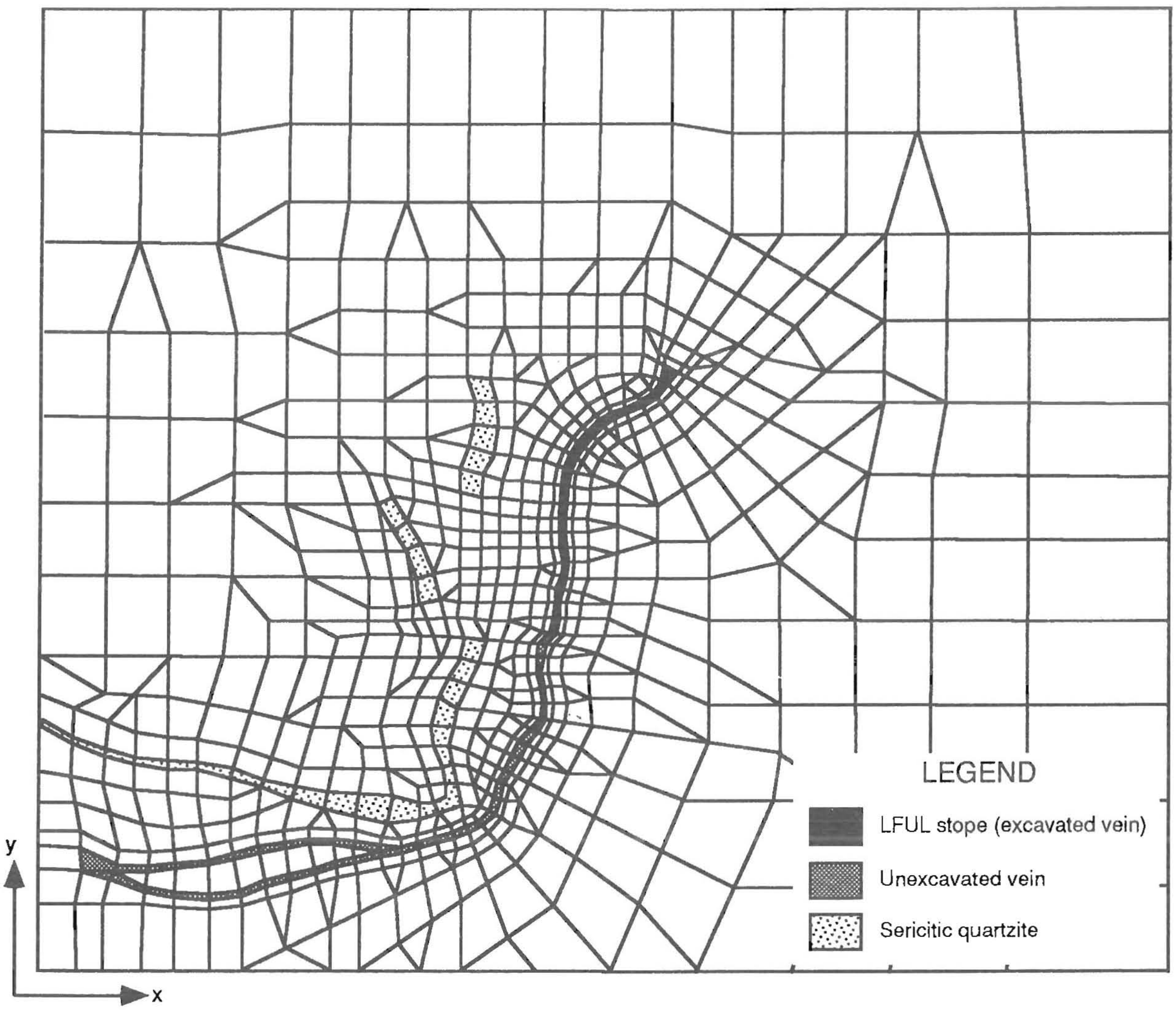

Figure 4.—Two-dimensional plan view mesh showing sericitic quartzite beds and location of LFUL stope in Lucky Friday vein.

MINE MODEL FORMULATION

The elastic-plastic constitutive behavior assumed for the rock mass is described mathematically by several equations, including those for equilibrium, strain-displacement, stress-strain, and strength relationships. While analytic (exact) solutions are possible for highly idealized combinations of loading, rock mass geologic structure, and mine geometry, such solutions were not feasible for the complex conditions found in the LFUL stope, and a numerical approach was necessary. Several numerical methods are available for developing approximate solutions to these equations, depending upon assumptions and mine geometry.

The finite-element method, a particularly convenient and powerful type of numerical method, can handle real-world complexities, such as different materials, directional rock properties, curved and faulted geologic contacts, and sequential excavation and filling of stopes. Transformation of geologic information and excavation dimensions into a numerical form required subdivision of the study region into a mesh of small homogeneous units known as finite elements. Relatively small elements were used near the excavation walls for numerical accuracy, while larger elements were used away from the walls for economy of computer storage and running time. Element boundaries were typically along geologic contacts and stope walls.

To implement the finite-element method, the programs UTAH2 and UTAH3 were chosen. UTAH 2 and UTAH 3 are capable of handling anisotropic, nonhardening, gravitating, elastic-plastic materials that may be initially stressed. Elastic and plastic anisotropies may be defined independently. Both linear (Drucker-Prager) and nonlinear yield criteria are available. Strength is thus stress dependent as is appropriate for rock and soil. Elastic strain increments are computed using a generalized Hooke's law; plastic strain increments are computed using the associated flow rule. Thus, the initial response of an element is elastic, but with increasing load, the element eventually will reach its elastic limit. Beyond the elastic limit, stress changes are limited by material strength and yielding. Yielding of the rock mass may physically occur in several ways, for example, by extension of microcracks, slip along existing joints and fractures, and extension of fractures. The result is modeled as macroscopic yielding and plastic flow of the rock mass. These capabilities were more than adequate for the constitutive behavior defined for modeling the LFUL stope.

The UTAH programs differ in their capabilities of modeling stope geometry. UTAH3 can model complex stope geometries in three dimensions. UTAH2, on the other hand, assumes a very long opening that can be simplified to a cross section in two dimensions with plane strain boundary conditions in the third. Problems amenable to this simplifying assumption can be solved considerably faster than equivalent problems using three-dimensional geometries. Thus, it is common practice to run a quick two-dimensional model to test assumptions and model performance before undertaking the construction of more time consuming and expensive three-dimensional models.

TWO-DIMENSIONAL MODEL

Figure 4 shows a two-dimensional plan view of the 5,100-ft level of the Lucky Friday Mine after the level was divided into finite elements. The LFUL stope was started on the 5,100-ft level where the vein is close to vertical. There are 794 elements and 777 nodes represented in figure 4, which is a relatively small two-dimensional mesh.

Simulation of LFUL stope excavation was accomplished by excavating 22 elements along the vein in a single, 500-ft-long cut with laboratory values of rock properties. The result was a shaft-like excavation of indefinite vertical extent where displacements were confined to the horizontal plane; vertical shear stresses remained zero while other stress components changed as a consequence of mining. Figure *5A* is a plot of principal stresses in the two-dimensional model after excavation; figure *5B* shows the corresponding displacement pattern. No element failures occurred despite induced tensile stresses near the footwall and hanging wall of the stope.

Since progressive excavation of the LFUL stope was mainly vertical, successive LFUL cuts (and fills) cannot be followed. Hence, installation of instruments and subsequent changes in instrument readings cannot be simulated in plan view. An alternative two-dimensional, tunnel-like model is possible in a vertical section perpendicular to the vein. Successive LFUL stope lifts could be followed in vertical section, but the curved course of the vein could not be represented in such a model.

THREE-DIMENSIONAL MODEL

The real world is three dimensional, and a more accurate and versatile model should be three dimensional to follow the progress of mining along the strike of the vein in the horizontal direction as well as cut by cut in the vertical direction. In this regard, advances in computer technology have brought three-dimensional capabilities to the stage of development reached by two-dimensional models at the time the first cut-and-fill simulations were conducted by Pariseau and Kealy (*9*).

An outline of the region enclosed by the three-dimensional mesh is shown in figure 6. The mesh extended 1,780 ft east-west, 1,520 ft north-south, and 1,600 ft vertically. These dimensions were sufficient to reduce the influence of the outer mesh boundaries on stope wall stresses and displacements to a negligible amount. The mesh was constructed by digitizing mine maps of the 4900-, 5100-, 5150-, and 5300-ft levels. Intermediate levels were generated by linear interpolation between the digitized levels. The steep dip of the vein and beds of sericitic quartzite produced only slight changes in element shape and size from level to level. The entire minable length of the vein was defined in the mesh, so that the final mesh consisted of 30,966 elements and 31,080 nodes in 39 three-dimensional element layers. The coordinate system was the same as that used in the two-dimensional model with the addition of the vertical Z-coordinate.

The simulated mining sequence began with the first cut of the LFUL stope on the 5100-ft level using the premining stress state given in table 3. Simulation of each cut involved 10 loading increments to follow any nonlinearities that might have arisen from element yielding. Output from the first cut provided the starting stress state for the second cut, and so on. Each LFUL cut was the width of the vein, about 10 ft deep, and approximately 500 ft long along the strike of the vein. The simulated mining sequence follows the actual sequence, described in detail in Williams, Whyatt, and Poad (*19*).

The calculated displacements allowed calculation of stope closure and anticipated extensometer readings; closure was the relative displacement between points on opposite sides of an opening, and extensometer readings indicated the relative displacement between the collar of the extensometer borehole and anchor points positioned in the borehole. Care was taken to model the correct portion of stope excavation that each instrument monitored to ensure a valid comparison with actual measurements.

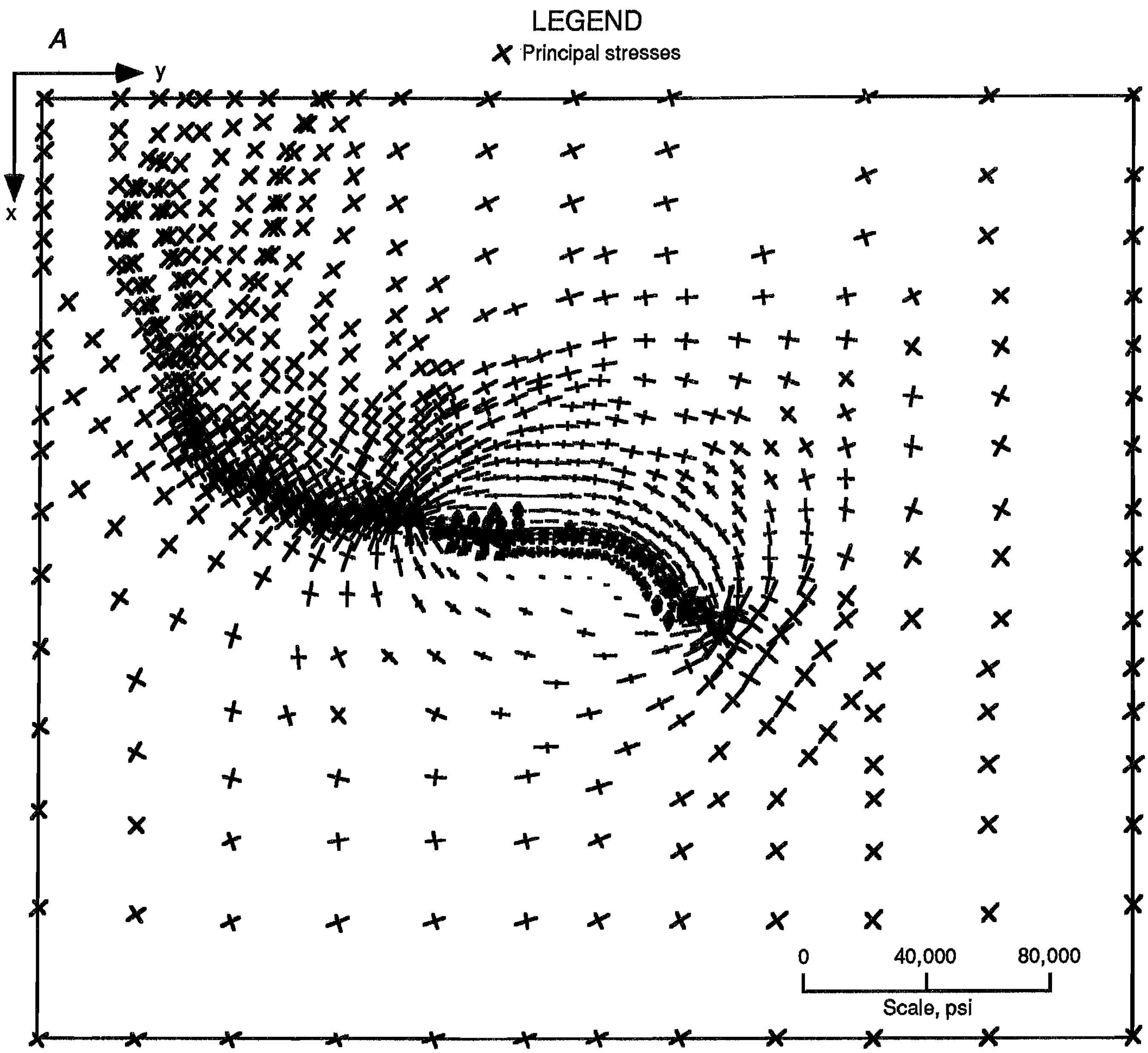

Figure 5.—(*A*) Principal stresses calculated by plan view model.

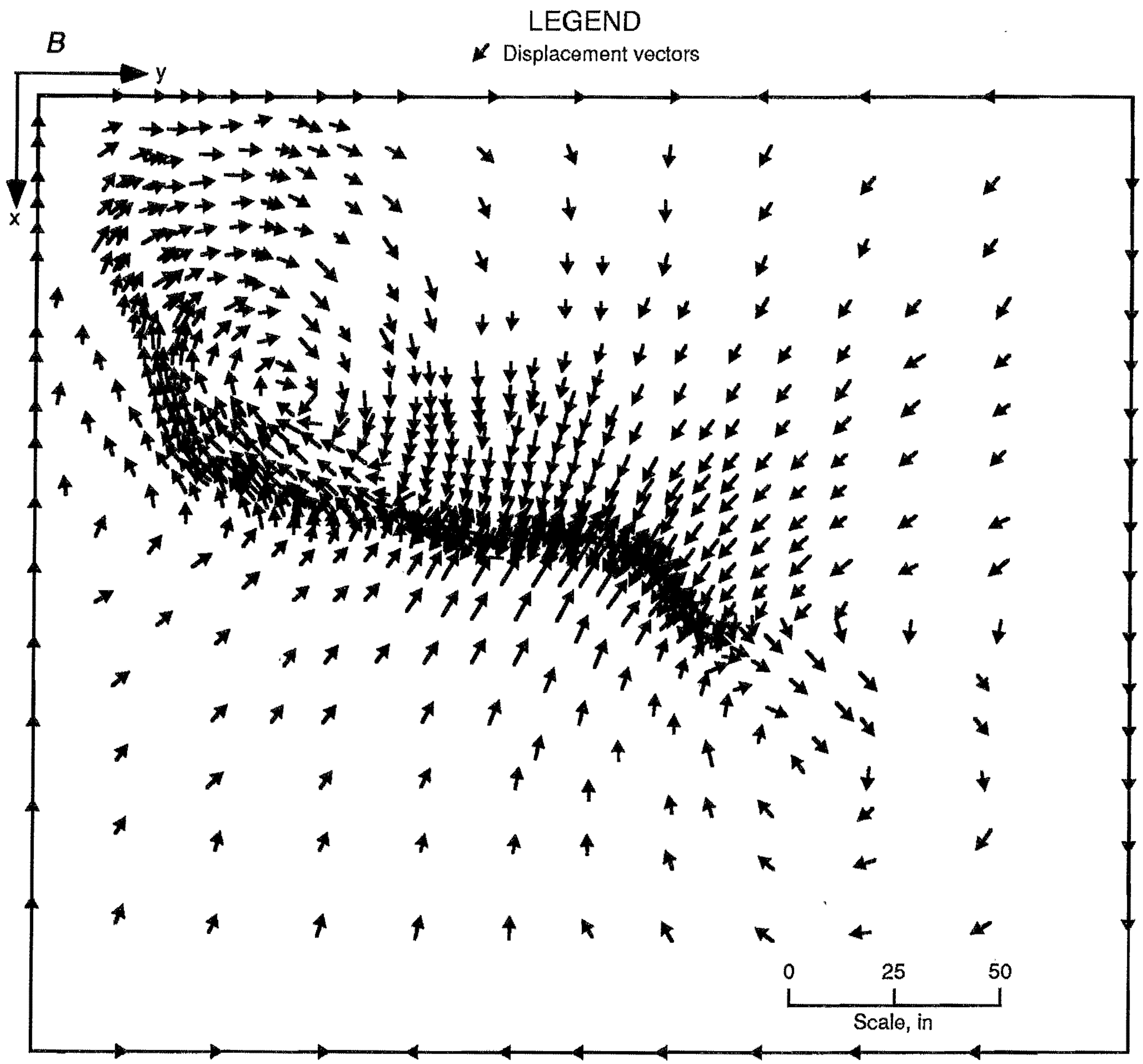

Figure 5.—(*B*) Displacement vectors calculated by plan view model—Continued.

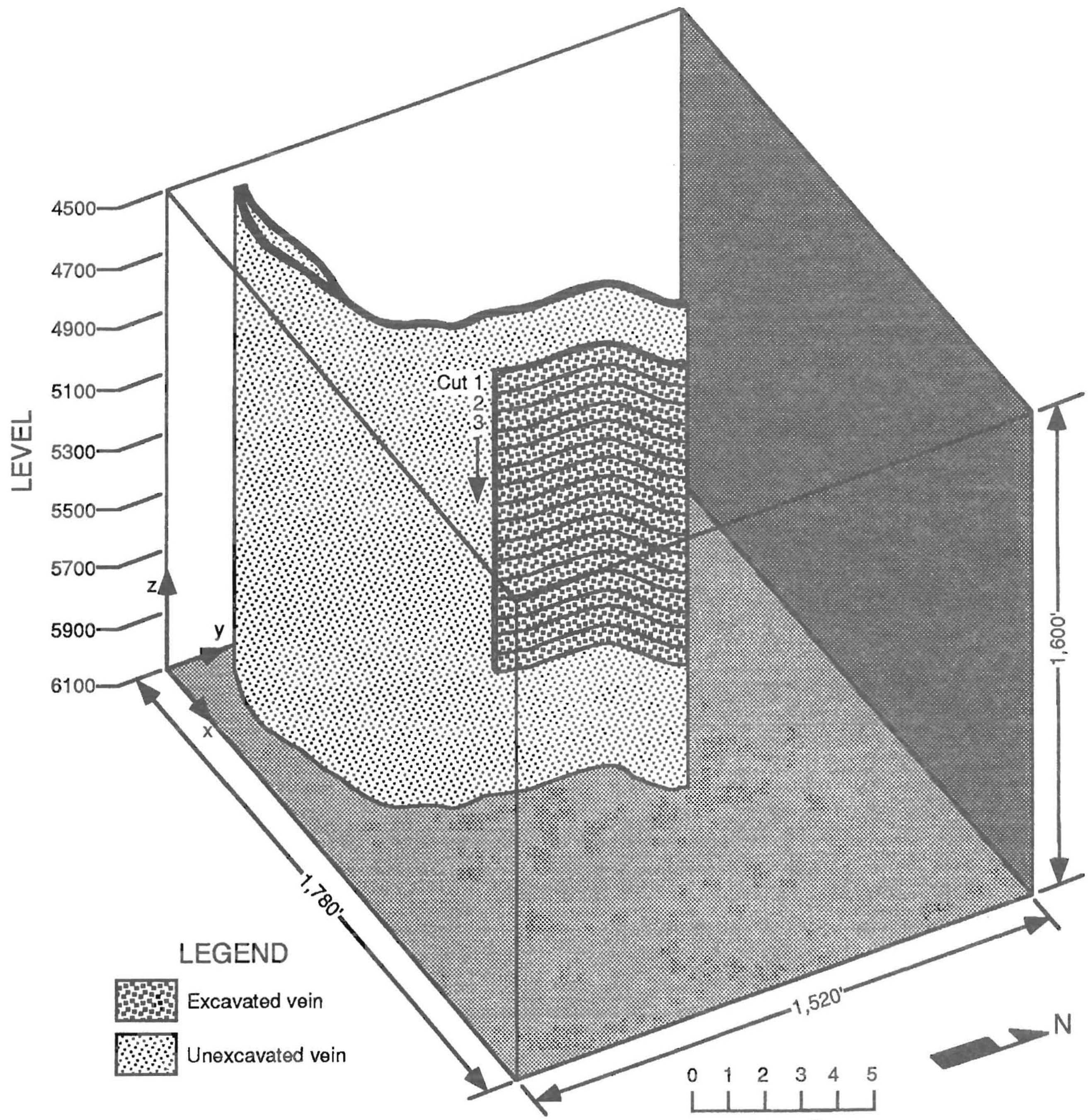

Figure 6.—Three-dimensional view of LFUL stope in modeled block.

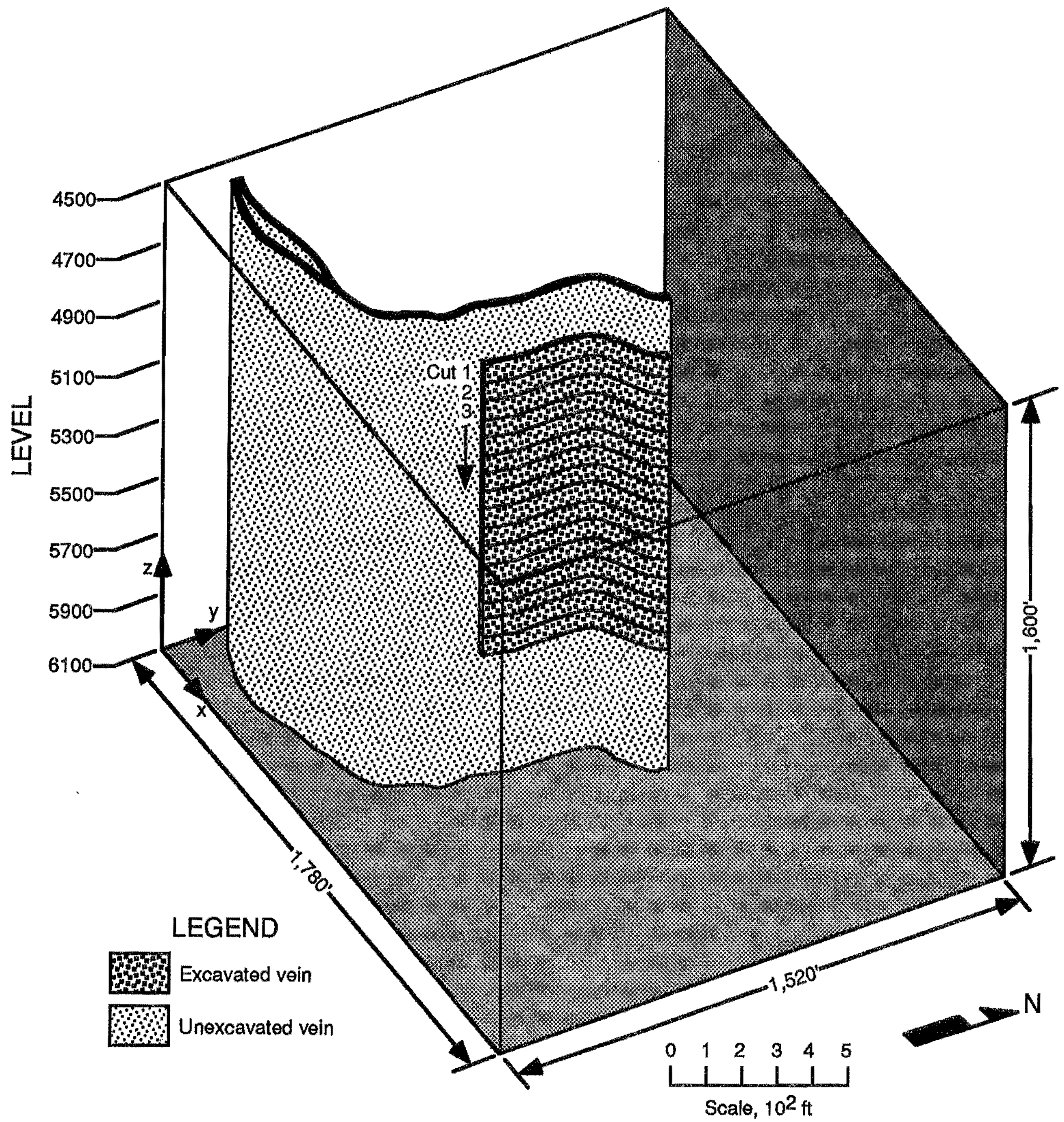

Figure 6.—Three-dimensional view of LFUL stope in modeled block.

MINE MODEL CALIBRATION AND VALIDATION

Model calibration is the process of adjusting model parameters so that the model approximates field measurements and observations. Adjustment is concentrated in rock elastic modulus and strength because laboratory measurements of these quantities are generally much greater than in situ values. Validation provides some sense of the relative success of the calibration. The following sections describe calibration and validation of the model for the elastic modulus of the rock mass using intact elements and the strength of the rock mass for elements that had failed.

ELASTIC (INTACT) ELEMENTS

Modeling excavation of a stope cut with a finite-element method requires solution of a system of equations that has the form

$$\Delta F = K \Delta U, \tag{1}$$

where ΔF = change in node forces,

K = stiffness matrix,

and ΔU = change in node displacements.

In the purely elastic range of response, Young's modulus E of, for example, vitreous quartzite, can be factored from K so that

$$\Delta F = E K' \Delta U, \tag{2}$$

where K' is dimensionless. The node forces are determined from the stresses and are independent of the right-hand side of equations 1 and 2. It follows that the product $E\Delta U$ is a constant, and therefore

$$\Delta U_l = (E_m/E_l)\Delta U_m, \tag{3}$$

where $_l$ = analysis with laboratory rock properties,

and $_m$ = rock mass properties taken from mine measurements.

Thus, the displacements calculated by computer mining of a stope cut are related to displacements measured during mining of the same increment by the ratio of true rock mass to laboratory Young's modulus. This relationship assumes that measured displacements result from elastic rock mass deformation. Equation 3 can be used to determine relative displacements between two arbitrarily chosen points 1 and 2, which may represent anchors for stope closuremeters or borehole extensometers. Thus,

$$[\Delta U_l(2) - \Delta U_l(1)] = (E_m/E_l)[\Delta U_m(2) - \Delta U_m(1)]. \tag{4}$$

Equation 4 has the simple linear form $y = ax$, where dependent variable y corresponds to relative displacement calculated from finite-element analyses using laboratory rock properties, independent variable x corresponds to relative displacements measured in the mine, and slope a corresponds to the ratio of rock mass to laboratory Young's modulus. Hence,

$$E_m = a(E_l). \tag{5}$$

Figure 7 shows a plot of calculated versus measured changes in closure and extensometer readings taken over a period of 4 years while 12 cuts in the LFUL stope were being mined. The vertical scale in figure 7 is one-tenth the horizontal scale, so that line slopes are exaggerated by a factor of 10. A regression analysis that allows for a calculated intercept results in a correlation coefficient r of 0.795 (r^2 = 0.632), so that 63 pct of the variability is explained by a linear relationship between calculated and measured changes in instrument readings.

The regression equation is

$$y = (0.138)x + 0.05 \text{ in.} \tag{6}$$

The slope of the regression line is the elastic-properties scale factor and has the value 0.138. A regression analysis that forces the regression line to pass through the origin results in a slightly smaller correlation coefficient (0.748, r^2 = 0.560) and a somewhat greater slope (a = 0.174). These results indicate that laboratory values of elastic moduli should be reduced somewhat less than an order of magnitude (factor of 10) for use in the analysis of the rock mass response to mining; that is, the rock mass elastic properties should be about 14 to 17 pct of the laboratory values.

If the rock mass is linear or only weakly nonlinear with quite localized yielding, then scaling the elastic moduli to rock mass values and recalculating the finite-element model displacements is equivalent to scaling the original regression data. A subsequent regression analysis of rescaled data on the original mine measurements should produce a new regression line with a slope of 1 (45° inclination) without changing the correlation coefficient. Figure 8, a plot of rescaled data and the resulting regression line (with intercept), shows that this is indeed the case. The vertical and horizontal scales are the same, so that regression line inclination is the actual value.

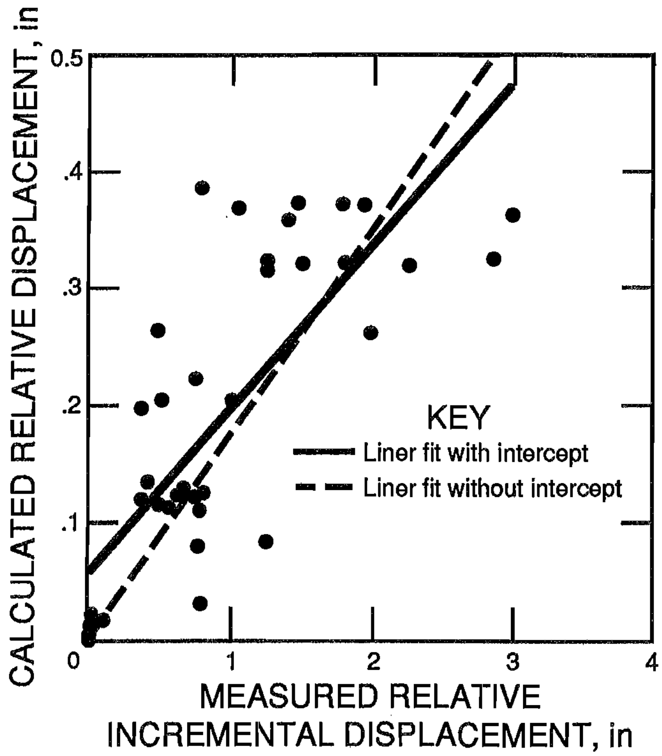

Figure 7.—Plot of measured versus calculated displacement at instrument locations. Calculated displacements based on laboratory measurements of rock properties. A two-parameter linear fit and a linear fit with zero intercept are plotted.

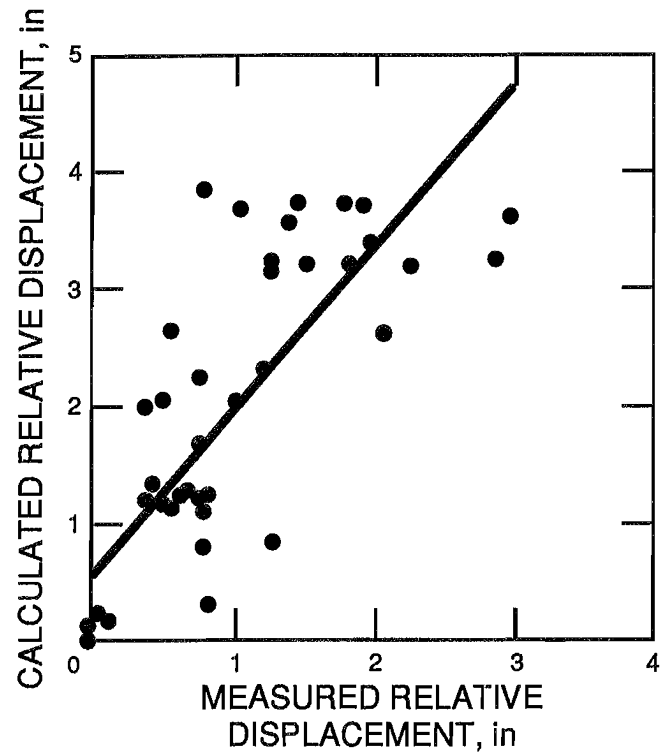

Figure 8.—Plot of measured versus calculated displacement at instrument locations. Calculated displacements based on an order-of-magnitude reduction of Young's modulus obtained from laboratory values. A two-parameter linear fit is plotted.

PLASTIC (FAILED) ELEMENTS

The strength properties scale factor cannot be obtained by simple regression analyses because of the nonlinearity associated with deformation beyond the elastic limit. A more subjective procedure can be applied that compares calculated extent of element yielding to the extent of yielding ground actually observed in the mine by inspection and interpretation of instrument readings. The usual indication of yielding is a relatively large increase in measured displacements compared with previous displacement changes and earlier stope cuts. However, some yielding in the Lucky Friday Mine occurs violently as rock bursts. If model strengths are too high, the calculated displacements will be too small to follow the actual displacements associated with stope wall yielding.

In this regard, detection of extensive yielding requires instruments to be located where yielding is occurring. Yielding may become widespread, but it will not be discernible in mine measurements when the measurements are made in nominally elastic ground. However, manually read closure point measurements were necessarily made in open, accessible areas during mining of the first few cuts in the isolated ore block where rock mass response was largely elastic. Remotely read closure point and borehole extensometer data are much more likely to indicate yielding when it takes place, provided the instruments survive mining and filling. Although few of the mine instruments survived excavation of more than a single cut, some stope wall closure meters and fill pressure cells installed in the fifth cut continued to operate through excavation of several cuts. These data may be examined for scale factors of elastic and strength properties.

Yielding at the levels of cuts 1, 5, and 10 after simulated mining to the present depth (5244-ft level) was examined for two cases. The first case used laboratory rock properties; the second case an order of magnitude reduction in modulus and strength. No yielding occurred in the first case. Considerable yielding occurred when elastic and strength properties were reduced, primarily in the footwall (fig. 9). Further examination of the extent of the yield zone at reduced rock mass moduli and strengths showed that a remnant of ore left between the original LFUL stope and the split had been stressed beyond the elastic limit and had yielded. Elevated stresses in remnants are to be expected, and the resulting yielding signals a

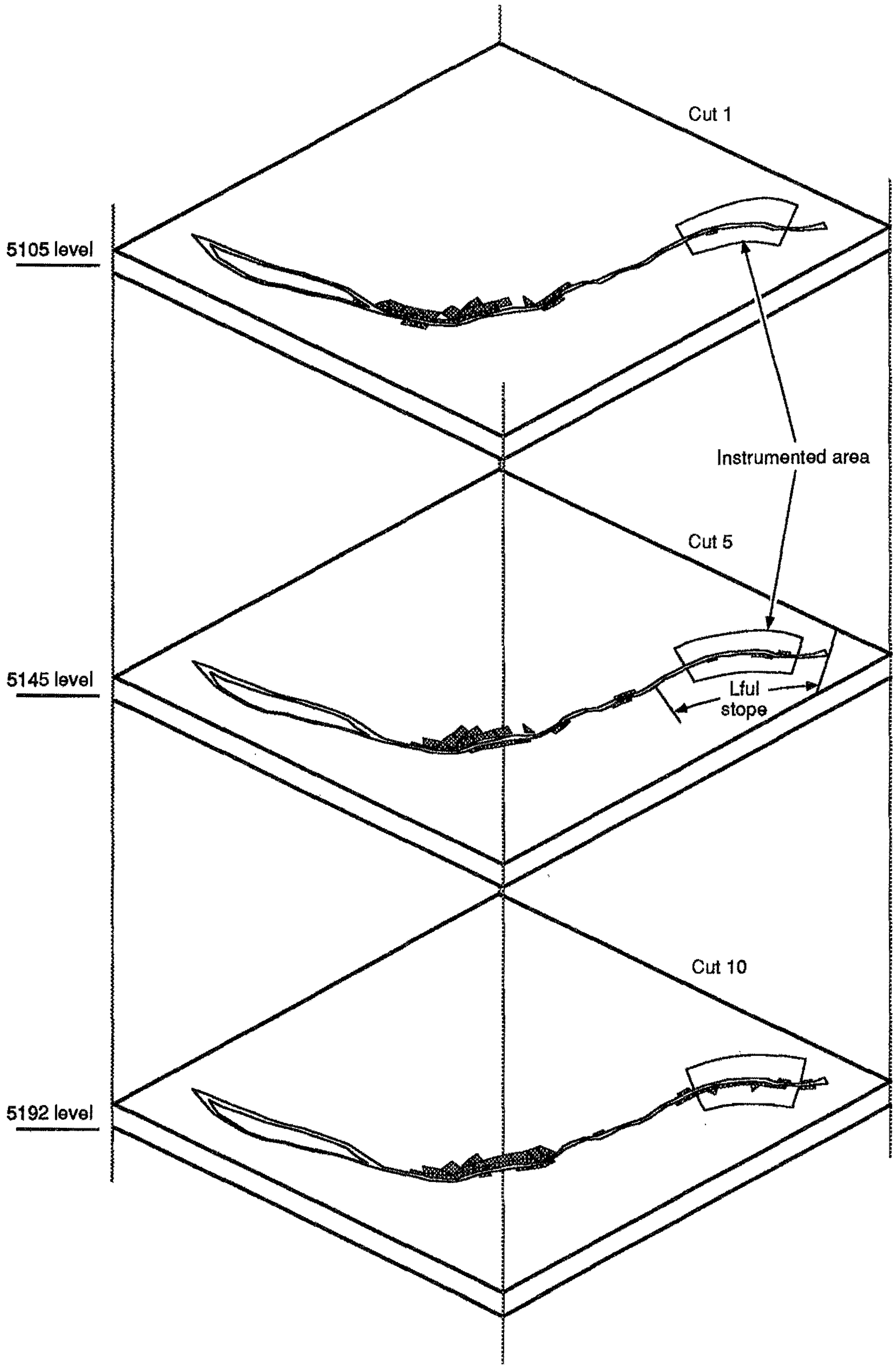

Figure 9.—Plastic zones after excavation of 14 cuts in LFUL stope (and simultaneous historical mining) for model with order-of-magnitude reduction in rock mass strength. Shaded area is equivalent to yielding ground.

potential failure of ground. In actuality, considerable rock burst activity took place in these areas during LFUL stope mining, confirming that stresses were exceeding strengths.

The reason for the apparent insensitivity of calculated instrument readings to the strength scale factor is clear from the results shown in figure 9. The measurement points and readings had been taken in stable ground where yielding was negligible. Extensive yielding did take place away from the LFUL stope when the strength scale factor was reduced.

The data do not allow for an accurate determination of the best strength scale factor that should be used in subsequent analyses. However, two empirical rules that relate the strength scale factor to the modulus scale factor provide some assistance. One rule is to maintain the strain-to-failure, ϵ_f, constant under uniaxial compression. This rule states that, at failure, $\epsilon_f = (C_o/E)$ is a constant (C_o and E are unconfined compressive strength and Young's modulus, respectively). Thus, if the laboratory value of E is scaled by a multiplier of 0.1, for example, then C_o must also be scaled by 0.1.

The second rule maintains the strain energy, Σ, constant at failure under uniaxial compression. According to this rule, $\Sigma = (C_o)^2/E$ is a constant. Thus, if the laboratory value of E is scaled by 0.16, for example, then the strength should be scaled by 0.40 ($= \sqrt{0.16}$). The second rule gives a less drastic reduction in strength for a given modulus reduction. When applied to a modulus reduction factor in the range 0.14 to 0.17, as determined by regression analysis, the strength scale factor range is 0.37 to 0.41. Scale factors of 0.16 and 0.40 for elastic moduli and strengths should be satisfactory for future LFUL model design analyses.

DISCUSSION AND CONCLUSIONS

A set of 42 field measurements taken from the experimental LFUL stope was used to calibrate and validate a numerical model for stope design in the Lucky Friday Mine. The quality of calibration depended on several factors, including estimates of boundary conditions, laboratory measurements of rock properties, relevance of constitutive law assumptions, and adequacy of the field data set. Elastic model calibration produced an elastic modulus scale factor of 0.14 to 0.17 and was validated with a correlation coefficient of 0.75. Location of field measurements in areas shielded from yielding prevented an equally good calibration for strength properties. An energy rule was applied that suggested a corresponding strength scale factor of 0.37 to 0.41. While a correlation coefficient could not be developed, a model with an order-of-magnitude reduction in strength produced considerable yielding in areas of the mine with extensive rock burst activity. A full-strength model showed no yielding.

The calibrated and validated three-dimensional, finite-element model of the LFUL stope explains two major physical observations:

1. Insensitivity of calculated instrument readings to rock mass strength, and
2. Location of concentrated rock burst activity in a remnant that evolved during mining, and in the hook and split parts of the vein (fig. 9).

As such, finite-element modeling is a useful design tool for assessing potential rock mass yielding (including concentrated rock burst activity) and evaluating alternative mining plans. Furthermore, the model can project the sensitivity of instrument locations to rock mass yielding, which should be an important consideration in planning further calibration studies. Ideally, the model should be run extensively before installation of the calibration instruments and used to optimize their location.

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APPENDIX.—MEASURED AND CALCULATED VALUES FOR STOPE INSTRUMENTATION

Table A-1.—Actual closuremeter and corresponding model measurements of stope closure

Point	Cut	Actual measurement, in	Model measurement, in			
			Base	10%S	50%E	10%E
SC1-W3 ...	2	1.968	0.339	0.339	0.679	3.393
SC1-W4 ...	2	1.800	.320	.321	.641	3.206
SC5-E1 ...	6	.786	.385	.391	.781	3.906
SC5-W1 ...	6	2.908	.360	.361	.727	3.636
	7	1.250	.323	.317	.635	3.174
	8	1.250	.315	.314	.628	3.139
SC5-W3 ...	6	1.380	.357	.359	.718	3.588
SC5-W1A ..	8	2.250	.318	.315	.627	3.135
	9	2.000	.261	.262	.524	2.621
	10	1.000	.203	.204	.408	2.040
	11	.000	.219	.219	.438	2.192
SC5-W1B ..	8	1.500	.320	.317	.633	3.165
	9	.500	.264	.264	.528	2.641
	10	.500	.205	.206	.412	2.060
	11	.750	.223	.223	.445	2.223
SC10-E1 ..	11	.960	.670	.668	1.332	6.661
SC10-E2 ..	11	.870	.676	.674	1.346	6.732
SC10-W1 ..	11	.690	.655	.651	1.301	6.508
SC10-W3 ..	11	.550	.649	.647	1.294	6.471

Table A-2.—Actual MPBX and corresponding model measurements of rock mass deformation

Depth, ft	Cut	Actual measurement, in	Model measurement, in			
			Base	10%S	50%E	10%E
		MPBX SC1-E2				
5	2	0.800	0.031	0.037	0.070	0.334
25	2	1.250	.084	.085	.170	.850
		MPBX R1				
5	2	0.028	0.002	0.002	0.004	0.019
5	3	.010	.002	.002	.005	.022
15	2	.027	.006	.006	.014	.061
15	3	.009	.007	.007	.014	.067
25	2	.032	.012	.012	.026	.118
25	3	.017	.012	.012	.026	.122
40	2	.038	.023	.023	.048	.229
40	3	.033	.022	.022	.044	.217

Table A-3.—Actual manual extensometer and corresponding model measurements of stope closure

Point	Cut	Actual measurement, in	Model measurement, in			
			Base	10%S	50%E	10%E
SC1-W1 ...	106-1	1.45	0.372	0.372	0.738	3.714
	106-2	.67	.129	.129	.257	1.286
SC1-W3 ...	106-1	.38	.371	.372	.742	3.712
	106-2	.43	.134	.134	.268	1.343
SC1-W4 ...	106-1	1.92	.370	.371	.741	3.704
	106-2	.81	.125	.125	.250	1.252
SC1-W5 ...	106-1	1.77	.371	.372	.742	3.713
	106-2	.78	.111	.111	.221	1.106
SC1-E1 ...	106-1	1.04	.368	.368	.736	3.680
	106-2	.56	.113	.113	.225	1.127
SC1-E2 ...	106-1	2.86	.323	.323	.646	3.229
	106-2	.77	.080	.080	.161	.804
SC1-E3 ...	106-1	.36	.198	.198	.395	1.974
	106-2	.11	.018	.018	.035	.176
SC1-E4 ...	106-1	.07	.184	.184	.368	1.838
	106-2	.03	.012	.012	.024	.121

Table A-4.—Actual closuremeter and corresponding model measurements of closure in 106 raise

Point	Cut	Actual measurement, in	Model measurement, in			
			Base	10%S	50%E	10%E
106-E	2	0.750	0.122	0.122	0.243	1.215
	3	.500	.116	.116	.232	1.162
106-W	2	.625	.124	.124	.248	1.242
	3	.375	.120	.120	.239	1.212

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