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DEEP CUT-GROUND CONTROL AND WORKER SAFETY IN COAL MINES

E. R. Bauer
NIOSH
Pittsburgh, PA

C. J. Bise
The Pennsylvania State Univ.
University Park, PA

ABSTRACT

The trend in underground room-and-pillar coal mining is to employ remote-control continuous mining machines and extended cuts, 12 m (40 ft) deep or more. This system of coal extraction, adopted by over 435 mines, can create additional worker-safety hazards. To address the ground control and worker-positioning hazards, a combination of statistical analyses and numerical modeling was conducted. Initially, the reported roof-fall incidents and fatalities were reviewed to delineate the ground-control hazards. Then, the application of the Coal Mine Roof Rating (CMRR) for estimating safe extended-cut depths was evaluated. Finally, computer modeling to predict roof displacements during extended-cut mining was completed. The paper describes the results of these studies and their impact on the safety of extended-cut mining.

INTRODUCTION

The mining of extended cuts in the U.S. underground coal mining industry has been steadily increasing. From 1988 to 1995, the number of mines approved to mine extended cuts increased by 108% (Bauer, 1998). According to the Mine Safety and Health Administration (MSHA), in 1995, approximately 45% of the producing underground bituminous coal mines had approval to mine extended cuts. Extended cuts are taken during room-and pillar mining, retreat mining, and for longwall gate road development. The popularity of extended-cut mining assures that mines will continue to request approvals for the use of this coal extraction system.

There are safety concerns associated with the mining of extended cuts, including ground control and remote-control operation. The safety-and-health issues of remote-control mining, including operator positioning during extended cutting, and human factors analysis of remotely operated mining systems, have been presented by King and Frantz (1977), Love and Randolph (199 1), Steiner et al. (1994), and Bauer et al. (1995). These authors suggest that the safety of workers engaged in extended-cut mining can be enhanced through a systems approach to ergonomic design of equipment tasks, and work processes. Much of the research on the ground-control aspects of extended-cut mining has been conducted in the recent past (Bauer et al., 1993, 1995, and 1997, Grau and Bauer, 1997, and Bauer, 1998). These authors reported on the roof-fall hazard potential of employing extended cuts; preliminary conclusions that based only on whether or not a mine had an extended-cut approval indicate that extended-cut mining was no more hazardous than the mining of 6 m (20 ft) deep cuts.

In addition, all MSHA coal mine roof-fall fatality reports for 1988 through 1996 were analyzed to determine how often extended cutting was a contributing factor. Extended-cut mining was considered a contributing factor if the fatality occurred at the active mining face and the depth exceeded 6 m (20 ft). This nine-year period was examined because all fatality reports were available, and because the mines with extended-cut approval were known with a fair degree of certainty. A total of 106 roof-fall fatalities occurred in underground bituminous coal mines in the U.S. from 1988 through 1996. Extended cutting was a contributing factor in 26 (24.5%) of the fatalities. To evaluate if extended-cut mining causes more roof-fall fatalities, the percent of extended-cut fatalities was compared to the percent of mines approved to mine extended cuts. For the years 1988-1996, the average percentage of extended-cut roof-fall fatalities (24.5%) is slightly higher than the percentage of mines with extended-cut approval (23.2%). It is important to note that this compares the nine-year averages and that currently, nearly 50% of all mines are approved to use extended cuts. For example, in 1995, 47% of the mines were approved to use extended-cut mining while only 37% of the fatalities involved extended cutting. This comparison suggests that extended-cut mining is no more hazardous to mine workers from a roof-fall perspective than is the mining of a nonextended cut.

ROOF-FALL FATALITY CHARACTERISTICS

Fatality characteristics extracted from the reports included geologic influences, fall location, victim location, and action of the victim. Many of the roof-fall fatalities occurred because of illegal actions on the part of the victim. For instance, 42% of the extended-cut fatalities occurred when the MSHA-approved depth-of-cut was being exceeded. In addition, 61% of the victims in extended-cut and nearly 34% of the victims in nonextended-cut roof-fall fatalities were inby permanent supports when fatally injured (Bauer, 1998).

Visibility during extended cutting could be a problem for continuous mining machine (CM) operators because the added depth may restrict their ability to recognize changing roof and geologic conditions causing them to travel inby permanent supports for better visibility. Geology was reported as a contributing factor in 81% of the extended-cut and 76% of the nonextended-cut roof-fall fatalities. The geologic anomalies cited included slickensided slips, laminated and unconsolidated shale roof, fractures, cutter roof, cap coal, draw rock, horsebacks, hillseams, and highly-jointed rock. The CM operator's ability to detect changing geology was only slightly reduced because the reports stated that the geologic influences went undetected in 29% of the extended-cut and 21% of the nonextended-cut fatalities (Bauer, 1998).

Another concern of extended cutting is that by exposing a larger unsupported area, the falls will be more likely to run out of the unsupported area and ride over the bolted areas. The reports indicated that in over 62% of the nonextended-cut and in only 46% of the extended-cut fatalities, the fall rode over or in between the roof bolts. For extended cuts and nonextended cuts where a distance was reported, the falls rode past the bolts an average of 1.7 m (5.6 ft) and 3.8 m (12.4 ft), respectively.

Finally, the fatality-report analysis was used to evaluate if mine workers performing certain job tasks were more likely to be fatally injured by a roof fall during extended-cut mining than during the mining of nonextended cuts. During the remote-control mining of extended cuts, workers either operating or helping on the CM were over three times more likely to be involved in a roof fall fatality than they would when mining a nonextended cut (46% vs. 15%). Workers involved with support operations were only slightly more likely to be a victim of a fatal roof-fall incident during extended cutting than during nonextended cutting (31 % vs. 29%) (Bauer, 1998).

REPORTED ROOF-FALL ACCIDENTS

The reported roof-fall accidents were analyzed to estimate the roof-fall potential of extended cutting. Mine operators are required to report any fall of bolted roof if certain criteria are met as specified in 30 CFR 75.223 (U.S. Code of Federal Regulations, 1994a). These roof falls are termed "accidents" by MSHA and are reported by mine operators on MSHA Mine Accident; Injury, and Illness Report Form 7000-1 (30 CFR 50.20) (U.S. Code of Federal Regulations, 1994b). All reported roof-fall accidents for 1989-1991, 1993, and 1995 were extracted from MSHA's Health and Safety Analysis Center (HSAC) Accident Database, which is a compilation of all the Form 7000-1 reports received by MSHA from mine operators. Data from these years were utilized because they represent the years that an accurate list of mines with extended-cut approval was available. The data were categorized and compared by whether or not the mine had an extended-cut approval. Then, incidence rates, normalized with respect to employee-hours worked underground, were calculated. Table 1 lists the reported roof-fall accident incidence rates for extended and nonextended-cut mines and shows that the incidence rate of reported roof-fall accidents for extended-cut mines is less than the rate for nonextended-cut mines and all underground coal mines. This suggests that there is no significant difference in potential for worker injury from roof falls between the two mine types.

DEPTH DETERMINATION USING CMRR

Determination of the safe depth-of-cut is usually the responsibility of the CM operator or section foreman. Their decision is based on experience, how they perceive the roof is behaving, and the visual signs of roof instability if present. Because this decision process is subjective in nature, there is some inherent error which could prove disastrous if an unsafe depth is selected and a roof failure occurs. To improve the depth-decision process, the Coal Mine Roof Rating (CMRR) was evaluated as a possible depth-determination alternative.

CMRR was selected because it estimates the structural competence of coal mine roof and considers bedding most important (Mark et al., 1994). CMRR addresses the lithologic factors that weaken bedded coal-measure rocks such as discontinuities, moisture sensitivity, and rock strength rather than just the geologic characteristics of the rock. CMRR also includes a strong-bed adjustment, a moisture sensitivity factor, a surcharge adjustment when weaker rocks overlay the bolted interval, and an adjustment for structurally weak contact surfaces rather than just for lithologic changes (Molinda and Mark, 1994).

Other mine-specific information that was used in the analyses included entry width, average safe/successful cut depth, extended-cut status, and overburden. This information was collected during underground mine visits in approximately 40 mines. An example of the mine-specific data is listed in Table 2. Entry width was included because research has shown that the entry width, width-to-height ratio, and opening shape can influence the stability of the mine roof (Stefanko, 1983, and Obert et al., 1960). Combined with the overburden thickness, the stresses around an opening will influence the stability of the mine roof. The average cut depth and extended-cut status are two parameters that are determined after mining of extended cuts has commenced. Extended-cut status is an estimate of a mine's ability to extract extended cuts based on the percentage of cuts mined that are extended cuts. The scale is listed as a footnote to Table 2. The average cut depth is

Table 1. - Accident incidence rates based on underground employee-hours worked, 1989-1991, 1993 and 1995.

Mine category	Total roof-fall accidents	Total underground hours worked	Incidence rate[1]
All[2]	13,666	571,701,998	4.78
Extended cut	7,479	314,250,741	4.76
Nonextended cut	6,187	257,451,257	4.81

[1]Incidence rate = $\frac{\text{No. of reported roof-fall accidents}}{\text{Total underground-hours worked}}$ x 200,000.

[2]Includes only data for underground bituminous coal mines.

Table 2. - Example of mine specific information used for estimating safe extended-cut depths (Mark, 1998)

Coal seam	State	CMRR	Entry width, m (ft)	Avg. cut depth, m (ft)	Ext. cut status[1]	Overburden, m (ft)
Blue Creek	AL	50	6.2 (20.3)	7.6 (25)	2	549 (1,800)
Wadge	CO	55	6.1 (20.0)	12.2 (40)	1	549 (1,800)
Herrin No. 6	IL	72	5.5 (18.0)	12.2 (40)	1	305 (1,000)
Springfield No. 5	IL	50	5.9 (19.5)	12.2 (40)	1	183 (600)
Springfield No. 5	IN	29	5.7 (18.8)	4.6 (15)	3	76 (250)
Harlan	KY	63	5.5 (18.0)	12.2 (40)	1	549 (1,800)
Lower Freeport	OH	68	5.9 (19.5)	12.2 (40)	1	183 (600)
Pittsburgh	PA	40	5.2 (17.0)	9.1 (30)	2	46 (150)
Hiawatha	UT	70	6.1 (20.0)	12.2 (40)	1	457 (1,500)
Powellton	WV	34	5.6 (18.3)	5.2 (17)	3	122 (400)
Hanna	WY	43	5.6 (18.3)	12.2 (40)	1	213 (700)

[1]Status: 1 = Always extended cuts (app. 75-100% of the time), 2 = Sometimes extended cuts (app. 25-75% of the time), and 3 = Almost never extended cuts (under 25% of the time).

Table 3. - Best-fit equations for estimating safe depths-of-cut (Bauer, 1998)

Mining stage	Best-fit equation (using English units of measure)	Std. dev., +/- ft (m)	Coefficient of determination (R^2)
Design	CutDepth = 10.1 + 0.442(CMRR)	7 (2.1)	0.455
Pre-approval	CutDepth = 10.1 + 0.442(CMRR)	7 (2.1)	0.455
Post-approval	CutDepth = 35.4 + 0.164(CMRR) - 6.64(Status)	5.5-6	0.641
	or	(1.7-1.8)	
	CutDepth = 47.0 - 18.56(Status)		0.609

either the depth of a majority of the cuts mined or the extended-cut depth when extended cuts are routinely mined.

Regression analysis using the statistical software package MINITAB, Release 10Xtra was completed to evaluate the relationship between depth-of-cut and one or all of the five variables listed in Table 2. Regression analysis determines which of the five variables best predicts the cut depth, and provides the best-fit equation for estimating safe cut depths. The equation for the simple linear regression model that was used to represent this statistical relationship is

$$Y_i = \beta_0 + \beta_i X_i , \qquad (1)$$

where: Y_i = Response (dependent) variable, depth-of-cut;
β_0 = Y intercept of the regression line;
β_i = Slope of the regression line; and
X_i = Predictor (independent) variable.

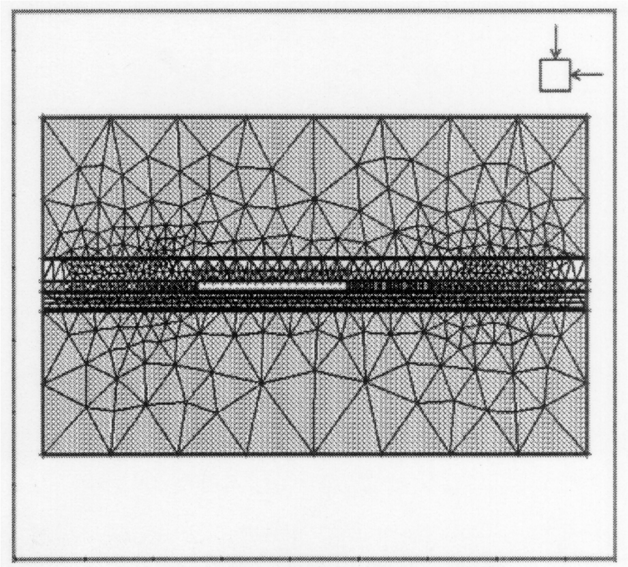

Figure 1. – Staged Model (Bauer, 1998).

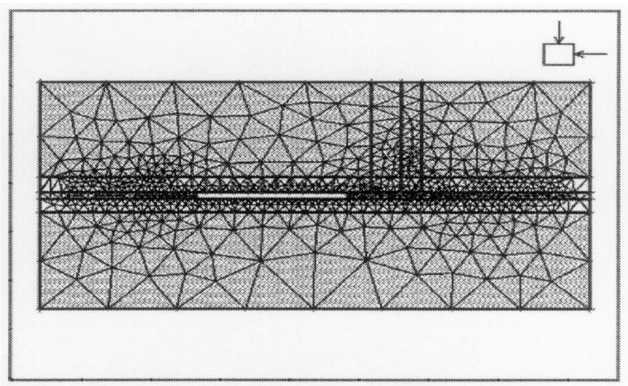

Figure 2. – Joint Element model (Bauer, 1998).

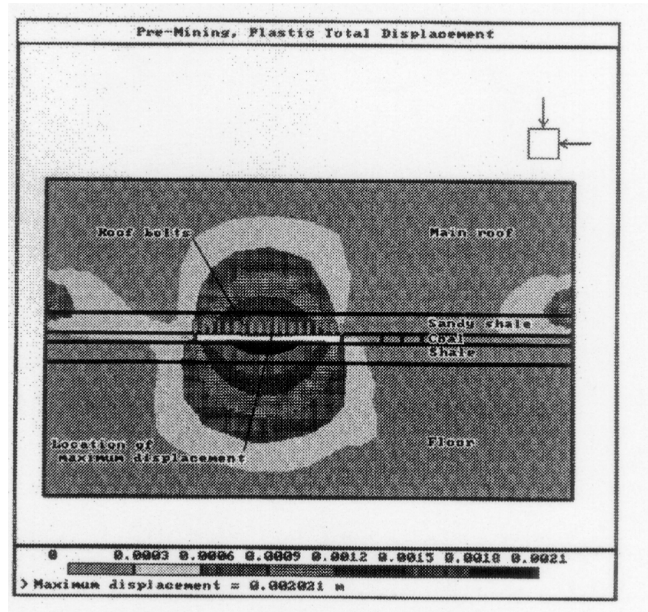

Figure 3. – Pre-mining staged model output of plastic total displacement (Bauer, 1998).

The independent variables (X_I) were CMRR, entry width, average cut depth, extended-cut status, and overburden. The regression models were completed using the independent variables that were available for three different mining situations: (1) During the design stage (prior to opening a mine); (2) Preextended-cut approval (after mining has commenced but before applying for an extended-cut approval); and (3) Post-approval (after mining extended cuts when a change in the approved depth is desired). Table 3 provides the best-fit equations as determined by the regression analyses for the three mining stages. In all three situations, CMRR was the

Table 4. – Failure criteria for Phase^2 modeling (Bauer, 1998)

Location	Rock type	Thick., m (ft)	Unit wgt., MN/m^3 (pcf)	Young's Mod., MPa (psi)	Poisson's Ratio	Mohr-Coulomb		
						c, MPa (psi)	phi, deg.	Tensile, MPa (psi)
Main roof . .	Sandstone	12 (40)	0.022 (175)	28000 (4.0 x 10^6)	0.18	2.8 (400)	25	20.0 (2,857)
Imm. .roof . .	Sandy shale	3 (10)	0.0214 (170)	25375 (3.6 x 10^6)	0.19	0.75 (107)	20	14.9 (2,133)
Coal	L. Kitt.	1.5 (5)	0.01 (80)	3250 (4.6 x 10^5)	0.37	0.20 (28.57)	35	2.8 (400)
Floor	Shale	2.75 (9)	0.0214 (170)	28420 (4.1 x 10^6)	0.14	1.5 (215)	19	37.4 (5,335)
Main floor .	Sandstone	12 (40)	0.022 (175)	28000 (4.0 x 10^6)	0.18	2.8 (400)	25	20.0 (2,847)

Table 5. – Bold Properties for Phase[2] modeling (Bauer, 1998).

Bolt characteristic	Value
Type	Full-column resin
Length, m (ft)	1.52 (5)
Diameter, mm (in)	15.875 (0.625)
Modulus, MPa (psi)	205,800 (2.9 x 10^7)
Peak capacity, MN (lbs)	0.2 (60,000)
Residual capacity, MN (lbs)	0.02 (6000)
Pre-tensioning, MN (lbs)	0.0 (0.0)
Out-of-plane spacing, m (ft)	1.22 (4)
In-plane spacing, m (ft)	1.22 (4)

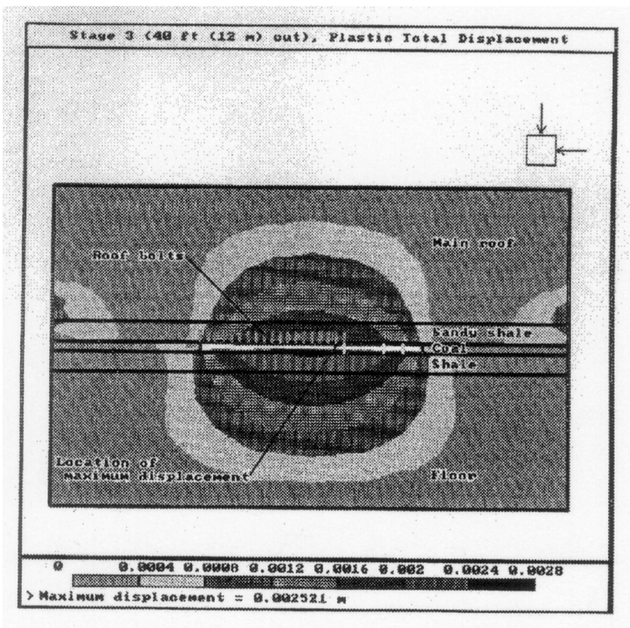

Figure 4. - Plastic total displacement for 12 m (40 ft) cut, staged model (Bauer, 1998).

R^2 values are considered low by standard statistical methods, they are reasonable for a mining based analysis where the environment is highly variable, uncontrollable, and unpredictable. Ultimately, these equations are not "design" equations and can not be used to determine an exact, safe depth-of-cut. They can be used simply as a starting point for mine operators and regulatory agencies to estimate a range of cut depths that could possibly be safely mined (Bauer, 1998).

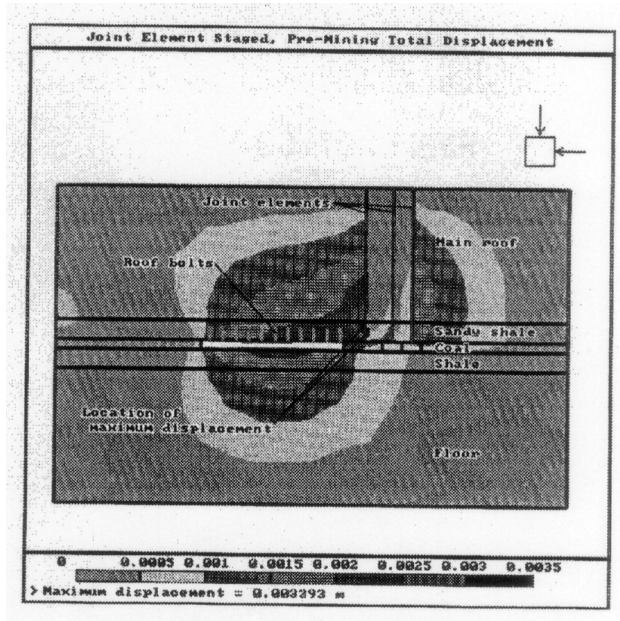

Figure 5. – Initial pre-mining displacements, joint-element staged model (Bauer, 1998).

NUMERICAL MODELING

Two-dimensional finite-element numerical modeling of roof and pillar reactions during extended-cut mining was completed using the computer program Phase[2] (Rock Engineering Group, 1997). Roof displacement was predicted as the depth-of-cut increased from 6 m (20 ft) to 12 m (40 ft) to evaluate where maximum displacement would be expected and to determine if the area where the remote operator is most likely to be positioned is safe. The effect of discontinuities on mine-roof stability was modeled using joint elements to simulate geologic discontinuities in the mine roof. Awareness of how discontinuities affect mine-roof stability can lead to an increased level of safety while extracting extended cuts.

To analyze strata displacements, as the depth-of-cut increases, a side elevation view, staged model was constructed (Figure 1). The staged model represented the extraction sequence of a typical 6-m-deep (20-ft-deep), 1.5-m-high (5-ft-high) cut of coal, that was extended to 12 m (40 ft) in successive 3-m-deep (I 0-ft-deep) cuts. Strata displacements were predicted along the centerline of the entry. The effect of geologic discontinuities was investigated by adding joint elements in the roof and overburden to the original model (Figure 2). For both models, the material properties for the coal seam, immediate roof and floor, main roof, and main floor were from an actual mine site (when available) and representative values extracted from the literature (Table 4). The bolt parameters used are listed in Table 5. Initially, linear elastic material properties were used to estimate the strength factors of the roof, coal, and floor, and to predict where elements were failing, then plastic material properties were used to evaluate strata displacements as the cut was extended from 6 to 12 m (20 to 40 ft) deep. The

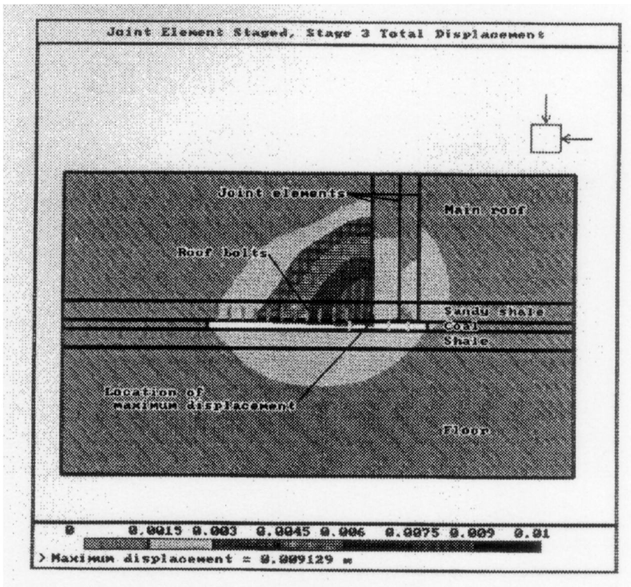

Figure 6. – Stage 3 output, joint-element staged model, 12 m (40 ft) cut (Bauer, 1998).

field stresses used were gravitational, based on an overburden of 152.4 m (500 ft), and unit weight of overburden of 0.022 MN/m^3 (175 pcf). This resulted in a vertical stress of approximately 3.85 MPa (550 psi), using the relationship that σ_v = 1.1 x overburden depth in m (ft). In addition, studies at the mine in question indicated a higher than normal horizontal stress (Campoli et al., 1996), thus the vertical and horizontal stresses were assumed to be equal. The horizontal stress (σ_H) was also 3.85 MPa (550 psi).

The plastic modeling followed the elastic modeling and was used to predict the location of maximum displacement. For the pre-mining stage, which included a bolted roof area, the plastic modeling predicted a maximum displacement in the immediate roof of 0.18 cm (0.07 in) (Figure 3). As the depth-of-cut increased, the predicted displacement also increased, finally reaching 0.25 cm (0.095 in) at a cut depth of 12 m (40 ft) (Figure 4).

Once an understanding of how "solid" mine roof strata reacts during the mining of extended cuts was obtained, the effects of geologic discontinuities on the stability of the roof during extended-cut mining could be studied. This analysis was important because the review of the roof-fall fatality reports revealed that geology was a contributing factor in 76% of the nonextended-cut roof-fall fatalities and in 81 % of the extended-cut roof-fall fatalities. Joint elements were used to simulate geologic discontinuities such as clay veins, slickensided surfaces, cracks or fractures, fault planes, cutter roof and horizontal stress effects, and jointing. These discontinuities affect the roof similarly, by severing the strata into discontinuous beams. The joint-element models used the same material properties as before. The joints were given properties that would simulate the worst case scenario. This included a joint aperture of 0.069 cm (0.027 in); normal and shear stiffness, joint tensile strength, and cohesion were all equal to 0.0 MPa (0.0 psi). Slip along the joint was allowed and the joint was permitted to deform from both the far field stresses and the induced stresses due to the excavation. A vertical joint was placed approximately at the midpoint of each stage, trending perpendicular to the direction of mining.

The joint-element staged models were completed using the effects of the excavation and jointing as inputs to each subsequent model. Figure 5 presents the initial pre-mining displacements. As in the previous displacement models, the displacement of the roof, floor, and coal ribs into the excavation, as represented by the deformed mesh, were exaggerated for illustrative purposes. The result of placing joints in the mine roof is an initial displacement (slip along the joint) of 0.34 cm (0. 13 in), located along the first joint just ahead of the face. The maximum displacement in the bolted roof was 0.25 cm (0.1 in), which was nearly 43 percent more than what was predicted by the pre-mining, "solid" roof model. As the subsequent stages were modeled, the maximum displacement continued to increase until finally, after stage 3 at a cut depth of 12 m (40 ft), the predicted maximum displacement was 0.91 cm (0.36 in) (Figure 6). The models predicted that only the first joint had a significant impact on roof displacement, while the depth-of-cut and remaining two joints had only a minimal influence.

SUMMARY AND CONCLUSIONS

This research was conducted to address the ground-control and worker-safety concerns of extended-cut mining. Safer extended-cut mining will reduce mine-worker injuries associated with this extraction system. The research efforts included the following investigations: roof-fall fatality and accident analysis; determination of improved methods of estimating safe depths-of-cut; and numerical modeling.

The review of the roof-fall fatality reports from 1988 through 1996 indicated that extended-cut mining was a contributing factor in 24.5% of the roof-fall fatalities. For the same years, the average percent of mines approved to extract extended cuts was only 23%. From an industry-wide perspective, the incidence rate of reported roof-fall accidents was slightly less for extended-cut mines (4.76) than for nonextended-cut mines (4.81).

In addition, it was demonstrated that the Coal Mine Roof Rating (CMRR) and extended-cut status can be used to estimate safe depths-of-cut on a mine-wide basis. The best-fit equation determined for cut depth during the design stage and pre-approval stage was:

$$\text{Cut Depth} = 10.1 + 0.442\ (\text{CMRR}).$$

For the post-approval stage, either of the following best-fit equations applies:

$$\text{Cut Depth} = 35.4 + 0.164\ (\text{CMRR}) - 6.64\ (\text{Status}),$$

or

$$\text{Cut Depth} = 47.0 - 8.56\ (\text{Status}).$$

No method, other than the experience of the CM operator or section foreman, was found to accurately predict the safe cut depth on a cut-by-cut basis.

Finally, numerical modeling of roof and pillar reactions during extended-cut mining predicted that a maximum roof displacement of 0.25 cm (0. 1 in) would occur in the center of the entry at the last row of bolts. In addition, when a joint (geologic discontinuity) was present, nearly 4 times the displacement would occur. The modeling indicated that the area encompassing the 2nd row of bolts outby should remain stable, and be a safe place for the remote operator to be positioned while mining an extended cut.

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