HOW TO COPE WITH CUTTER ROOF PROBLEM

A. Wahab Khair

ABSTRACT

This paper presents results of the experiments carried out in Beth Energy No. 33 Mine for the purpose of alleviating problems associated with cutter roof. Cutter roof problems have delayed the advance of the entry development considerably and maintenance cost of these entries was very high due to the requirement of very extensive and heavy artificial supports. The study includes assessment of the problem, investigation of the geology and lithology of the mines, study of in-situ stresses and corrective measures to combat the problem. Two approaches were selected as the corrective measures: 1) implementing yield pillar concept in order to reduce the effects of in-situ horizontal stresses, 2) designing an effective roof reinforcement system and application procedure.

Results of this study revealed that roof strata is weak and in-situ stresses are very high at the problem area which results in development of cutter roof. Implementation of yield pillar concept was found to be ineffective because cutter roof often developed at the entry face concurrent to the development. However, proper design and implementation of roof reinforcement system stabilized the roof and eliminated roof caving during both entry development and retreat of longwall face, despite existence of the cutter roof, at a much lower cost.

INTRODUCTION

Beth Energy No. 33 Mine is located within the Allegheny mountain section of Appalachian plateau province. The local geologic structures include folds, faults, joints, and cleats in the coal (1). The mine’s geologic structures have been influenced mainly by Ebensburg Anticline and Wilmore Syncline (2). Current longwall panels lay along the western limb of the Wilmore Syncline, oriented south 30° west as shown in Figure 1.

Fig. 1 Mine layout and location of experiments and tests.

Overburden depth varies from 183–244 m and topographic relief is less than 152 m. The coal seam is part of the Lower Kittanning formation and its thickness varies from 1.22 – 1.52 m and consists of two parts. The lower part is 1.12 – 1.22 m of soft coal and the remainder is bony coal. The coal seam dips 9° to 6° to the east—south direction.

Panel entries are driven 61 m wide, consisting of three 5.5 m wide entries and two pillars 24.4 x 30.5 m centers. Some entries indicated existence of fractures

11th International Conference on Ground Control in Mining, The University of Wollongong, N.S.W., July 1992.
in roof strata near parallel to the face direction and extended through the coal seam (see Figure 2). Two types of cutter roof were observed. The first type, initiated right at the coal face and extended into the roof, resulted in breakdown and caving of the roof strata and produced irregular roof line (see Figure 3). The second type was initiated behind the face a short distance from the entry face (working area) and extended toward the face (see Figure 4). In both cases following initiation of the cutter roof the entries experienced closure and resulted in push down of the roof strata (see Figure 5). This cutter roof problem delayed the advance of the entry development considerably and maintenance cost of these entries was very high due to the requirement of very extensive and heavy beam supports. Furthermore the abutment pressure of the retreat longwall face often resulted in the collapse of the beam support and caving of the roof (see Figures 6–7).

Fig. 2 Fractures, in the roof and rib, parallel to the crosscut.

Fig. 3 Irregular roof line resulted from roof breakdown during entry development.

Fig. 4 Cutter roof initiated in delay, behind the entry face and extended toward the face.

Fig. 5 Fractured and pushed down roof strata, following development of cutter roof.

11th International Conference on Ground Control in Mining, University of Wollongong, N.S.W. July 1992.
Mines deformation gauge and recommended procedures (3.4) Depth of the holes were limited due to excessive pressure and the collapse of the wall of the holes at the sides. Assuming $n < r < c_0 < 0$ (negative values being compressive), the average combined in–situ stresses and additional stress due to effect of entry development were found to be $-52$, $-44.5$, and $-31$ MPa, respectively. The average maximum (S) and minimum (S) horizontal stresses were found to be $-49$ and $-28$ MPa, respectively. The direction of these stresses with respect to direction of main and panel entries are shown in Figure 1. Further assessment indicated that once the cutter roof develops, the ribs converge toward the cutter zones, pushing down the roof. Maximum lateral strata movements take place at the lower strata and minimum at the higher strata above the roof line as depicted by the schematics in Figure 8.

Fig. 6  Collapsed beam support under the pressure of caving roof.

Fig. 7  Cutter roof resulted in caving of the roof at the headgate.

Fig. 8  Schematics of cutter roof development and roof strata movements.

STUDY PROGRAM

The study program included: 1) assessment of the problem and 2) in–mine experiments.

Assessment of the Problem:

To assess the problem, mechanical properties of rock/coal and in–situ stress associated with the mine have been determined. Laboratory tests on the coal seam indicated that its compressive strength is 4.4 MPa, angle of internal friction is $37^\circ$ and the triaxial stress constant is 3.7. Roof strata is highly laminated with poor bonding strength and is mostly shale, and has a compressive strength of 144 MPa. Floor is shale and relatively strong with exception of some fireclay. In–situ stresses were measured utilizing the U.S. Bureau of

11th International Conference on Ground Control in Mining, The University of Wollongong, N.S.W., July 1992.
In-Mine Experiments:

Since in-situ stresses are found to be very high in the mine and roof strata are weak, two approaches were selected to remedy the problem. The first approach was to reduce in-situ horizontal stresses through proper panel geometry and yield pillar size. The second approach was to reinforce the roof strata.

First approach: An experimental panel entry system was designed and implemented as shown in Figure 9. The behavior of the roadways were monitored while changing entry system's geometry and configuration. This panel entry began after a transition area, with an abutment pillar flanked by two yield pillars in 12 m x 42 m x 12 m center configurations with four 5.5 m - 6.1 m wide entries (see Figure 9). Number 1 entry was kept advance of the others because of hauptide system, advanced entries often experienced cutter roof in the mine, therefore development of entry No. 4 (the lower entry in Figure 9) was kept advance of the others in order to transfer cutter roof from No. 1 entry to No. 4 entry. Following the change of sequence of advance of entries, cutter roof also developed in No. 4 entry. Both No. 3 and No. 4 entries, with cutter roof, were supported with additional beam support utilizing a 59.1 kg (130#) rail with yieldable legs.

![Diagram](image)

Fig. 9. Experimental panel entry system with experimental roof support systems.

The proper yield pillar sizes were determined utilizing Wilson's hypothesis and associated equations [5]. When roof and floor were strong in comparison to the coal seam, the width of the yield zone, for total yield pillars, was found to be 1.7 m, thus the smallest dimension for the totally yielded pillars was determined to be 3.4 m. For partial yield pillars (i.e. yield pillar with elastic core), a 6.1 m core was required to give a total dimension of 9.5 m. When the roof and floor rocks were soft as coal seam, the above dimensions were found to be 2.6 m, 5.2 m, 7.6 m, and 12.8 m, respectively. In other words the calculated size of the yield pillars were found to be a minimum of 3.4 and a maximum of 12.8 m depending on the local geology and type of yield pillars to be employed. The yield pillars were reduced from 6.1 m to 4.5 m using 10.6 m x 45.5 m x 10.6 m center configuration as shown in Figure 9. The reason for further reduction of the size of the yield pillar was to provide more support for the roof strata; hence stopping cutter roof. However, reduction of yield pillar size change in sequence of entry advanced did not alleviate the cutter roof problem from No. 1 entry. Therefore, the size of the abutment, rigid pillars were reduced to approximately 21.1 m in order to have better interaction between the yield pillars. The panel development proceeded in a 10.6 m x 27.3 x 10.6 m centers, yield- abutment -yield pillar system (see Figure 9). The modified entry system eliminated the cutter roof problem in No. 4 entry, however cutter roof was still evident in No. 1 entry. After completion of four rooms (2 blocks), the yield pillars seemed stable without signs of deterioration except at the corners which made the intersection very large. Hence, additional supports and superbolts were utilized at the No. 2 and No. 3 intersections. A 73 m long caving occurred in the No. 3 entry with exception at the intersection which was supported with additional bolts, superbolts. Once again panel entry configuration was changed to abutment - yield pillar arrangement of 36.4 m x 12.1 m center proceed with 38.5 m x 10 m center (see Figure 9).

Stress-strain relationships for pillars during entry development is presented in Figure 10. From Figure 10 it is obvious that both yield and rigid pillars behave elastically at a depth of 3 m, however physical appearance of the yield pillar exhibited some yield at the ribs in the crosscuts as shown in Figure 11. The lateral movements and yielding of the ribs provided some relaxation in the roof strata which resulted in 9.8 cm roof to floor convergence and 4.2 cm strata separation (see Figure 12). However the relaxation in the roof strata didn't alleviate cutter roof development. During retreat of the longwall face abutment pressure yielded the small yield pillars completely while the yield zone in rigid pillars reached a depth of approximately 1.5 m (see Figure 13). Figure 13 shows stress changes within the pillars at depths of 1.5 m and 3 m.
Fig. 10  Stress–strain relationships for pillars during entry development.

Fig. 11  Yield pillar exhibiting failure at the crosscut.

Fig. 12  Roof to floor convergence during entry development.

Fig. 13  Stress changes within the pillars during retreat of longwall face.

As the longwall face passed by the monitored pillars, a 1.5 m yield zone developed in both yield and rigid pillars as indicated by a drop in measured stresses. Due to further retreat of the longwall face a 3 m yield zone developed in the yield pillar which is almost center of the yield pillar, hence resulting in total yielding of the small pillars while load has been transferred onto the rigid pillar. Figure 14 shows physical appearance of both yield (left side) and rigid (right side) pillars as the longwall face has been passed by this section. The stress–strain history within both yield and rigid pillars at depths of 1.5 m and 3 m are presented in Figure 15. The stress–strain history shown in Figure 15 substantiate development of yield zones, 3 m deep in yield pillar while 1.5 m deep in rigid pillar. Total yielding of the small pillars resulted in roof to floor convergence of 43 cm and strata separation of 20 cm as shown in Figures 16 and 6. Figure 16 shows deformation characteristic of No. 2 entry while Figure 6 shows condition of the No. 1 entry at the headgate. From the behavior of the roadways it was evident that implementing yield pillars could not stop the development of cutter roof nor could it stabilize roof condition in preventing caving.

Second approach: The second approach was a) to increase roof strata reinforcement by utilizing proper bolting pattern and density, b) while keeping the fracture roof together transfer downward pressure of the roof strata to the ribs by utilizing truss bolts. A primary roof bolting system was designed and implemented (see Figures 17–18). The use of angle
Fig. 14 Physical appearance of both yield (left) and stiff (right) pillars during retreat of longwall face.

Fig. 15 Stress–strain history for both yield and stiff pillars during retreat of longwall face.

Fig. 16 Roof to floor convergence during retreat of longwall face.

Fig. 17 Schematic of designed primary roof bolting system pattern ($1' = 0.3048$ m).

Fig. 18 Implemented designed primary roof bolting system.

It was found to be very effective. It reduced bending stress in the roof strata as the result of the lateral convergence and pushdown of strata at the cutter side (see Figures 4 and 19). Observation of the roof strata movement through borescope, indicated that most of the strata separation occurred within 2–3 m of the roof resulting overloading of the long bolts (see Figure 20). Furthermore installation of angle bolts was time consuming and was found to be needed only at the cutter side hence the primary roof bolt system was modified (see Figure 21). To keep fractured rock intact supplemental support system, truss bolt set and yieldable legs, were implemented as shown in Figures 22 and 23. The combination of the support systems was found to be the solution to the problem, stabilizing the roof strata even in fractured states as shown in
Fig. 19  Shows performance of angle bolt at the cutter side.

Fig. 20  Shows failed roof bolt due to the dynamic action of roof strata separation.

Fig. 21  Schematics of modified primary roof bolting system (1' = 0.303 m).

SUPPLEMENTAL SUPPORT SYSTEM

6' RESIN BOLT
6' TRUSS BOLT SET

YIELDABLE LEGS

Fig. 22  Schematics of supplemental roof support system (1' = 0.303 m).

Fig. 23  Shows in mine primary and supplementary roof support system (before additional yieldable leg is installed in the center of entry).

Figure 24. Figure 24 shows roof strata behind the longwall face in the gob. The roof reinforcement system was modified further in order to be cost effective (see Figure 25A—C). Due to the variation of in-situ stresses some areas in the mine experienced severe pressure, and excessive lateral roof motion, which resulted in failure of the angle bolts in both primary and supplementary roof bolts systems. In—mine measurements indicated that pressure build-up started approximately 9 m behind the entry face and reached to its full level 15—24 m behind the entry face. The length of the angle bolt in primary roof bolting system was reduced allowing the anchored zone to move with the roof strata. Later on the primary roof bolting system was changed to all vertical posture (see Figure 25A) and supplemented with truss and yieldable legs. The final modification of roof reinforcement system was in cycle implementation of primary and supplemental support system as Figure 25B—C. To avoid over stressing of the truss bolt system three options were considered: a) use
truss bolts in passive manner, that is to say no tension to be applied in angle bolts, b) place wooden blocks or collapsible block, between truss bolt bearing plate and roof strata, to absorb excessive strata separation and lateral movements, and c) install truss bolts in delay time, 15–30 m feet behind the entry face. The most effective of the three was found to be the second option.

ANALYSIS AND CONCLUSIONS

Results indicated that in-situ horizontal stresses are very high in the mine and strata are weak, highly laminated with poor bonding condition. In an unconfined state, strata separation occurs and strata buckles down resulting in failure of the roof and development of cutter roof.

Implementation of yield pillar concept was found to be ineffective. Relaxation of the roof strata due to yielding of the pillar further deteriorated the laminated strata. On the other hand, roof reinforcement and supplemental support system was found to be the solution to the problem. Increasing density of bolting and proper pattern increased friction resistance between laminated strata hence reducing lateral movement of roof strata. Furthermore, under action of truss bolt, bending stress in the roof strata was reduced, uplift confinement in the roof strata increased and downward vertical pressure in the roof strata was transferred onto the ribs. Major success is attributed to the immediate application of supplemental support which kept roof strata under confinement and constraint during development of additional pressure. Hence the integrity and self support of laminated roof strata was preserved.

ACKNOWLEDGMENT

This project was co-sponsored by Beth Energy Mines, Inc. and West Virginia University’s Energy and Water Research Center (EWRC), and the College of Mineral and Energy Resources. The writer acknowledges the assistance of Daping Xu, Research Assistant, in preparation of this paper.
REFERENCES


