FINAL TECHNICAL REPORT
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Project Title: **DEVELOPMENT AND DEMONSTRATION OF ALTERNATE ROOM-AND-PILLAR MINING GEOMETRY IN ILLINOIS**

ICCI Project Number: DEV05-11
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ABSTRACT

A typical room-and-pillar mining method uses pillars of equal size for ease of layout and simplified standard mining sequences. In such a geometry, pillars around the panel center have lower safety factors while pillars next to the barrier pillars have higher safety factors, because pillar size is based on the highest expected load while pillar loading increases from edge pillars (lowest) to central pillars (highest). This leads to a higher risk of roof falls in the central entries where the conveyor belt and other critical infrastructure are located. The extraction ratio achievable with this geometry is also sub-optimal.

The alternate mining geometry concept is a novel method of using unequal pillar sizes with larger pillars in the center and smaller pillars at the edges of a mining section. This allows equalizing pillar and floor safety factors across the entire width of a mining section while simultaneously achieving a higher extraction ratio. Furthermore, cut sequencing in a panel with alternate geometry can also be optimized to realize a higher production rate at a lower production cost.

In this project, the 'Alternate Mining Geometry' concept was further developed for demonstration in the sub-main area of a mine in Southern Illinois. The project team conducted studies to: 1) monitor geotechnical and operations performance of a currently practiced regular geometry, 2) develop an alternate mining geometry in cooperation with the mining company, and 3) monitor geotechnical and operational performance of the alternate geometry demonstration area during mining.

A preliminary analysis of results in this first phase of the overall project, indicate that pillars in the regular geometry exhibit higher convergence (as would be expected) around the centre of the section as compared to the edge. The alternate geometry was designed to increase the extraction ratio of the panel from 53.37% to 56.33% without any sacrifice in modeled safety factors. The developed alternate geometry was approved by the company for demonstration.

EXECUTIVE SUMMARY

Background

The overall project goal is to develop and demonstrate advanced mining technologies that will improve face productivity and reduce production costs while improving safety and working conditions in Illinois coal mines. As part of an ongoing research program to realize the above stated goals, the project team has already developed and demonstrated incremental improvements through higher productivity in current mining geometries, better dust control and the potential of reducing out-of-seam dilution. This has been accomplished with funding support provided by the Illinois Clean Coal Institute and coal mine operators throughout Illinois. Tasks in this project aim at demonstrating the innovative 'Alternate Mining Geometry' concept at an Illinois mine for increasing productivity, reducing production costs, reducing out-of-seam dilution, improving ground stability and increasing extraction ratio. The demonstration was conducted in a sub-main development at a Southern Illinois mine to achieve:

1. Higher extraction ratio with the alternate geometry.
2. Higher pillar and floor safety factors around the center of the sub-main section that will preserve the integrity of belt and power entries over a longer period.
3. Higher long-term stability of the overall sub-main area that will preserve the integrity of primary airways.
4. Smaller sub-main foot print allowing more reserves for panel mining.
5. Higher production and productivity potential through cut sequence optimization.

To achieve the above objectives, the study was divided into two phases. In the first phase of the study, three major tasks were planned and executed.

Task 1: Collect Baseline Data on Current Mining Geometry

This task primarily dealt with collection of data on the currently practiced geometry that was later used in the development of an alternate mining geometry for sub-main entries of a mine in Southern Illinois. Convergence data and pillar stress data were collected and used to develop a structural model for the current geometry. Industrial engineering and ventilation studies data were also collected and will be used in Phase II of the project.

As expected, preliminary results indicate higher loads on central pillars as compared to edge pillars. Plots of convergence data over the demonstration period show that the roof strata in the regular geometry bends to form a concave shape, which indicates higher loading on central pillars and entries. Central entries are subject to higher stress conditions and thus more liable to experience roof falls.

Task 2: Develop Alternate Geometry for Sub-Main Demonstration

The goal of this task was to use ground control and production models of Task 1 to develop an alternate mining geometry with more uniform safety factors across the panel

and higher extraction ratio to demonstrate in a sub-main section of a mine.

Finite Element analyses of the regular and various alternate geometries were performed and an alternate geometry identified for demonstration. The demonstrated geometry would increase the extraction ratio of the panel from 53.37% to 56.33% while reducing the panel footprint from 715 feet to 625 feet in widths. Pillar safety factors calculated using the conservative tributary area theory ranged from 3.16 for the smallest size pillar to 4.48 for the largest central pillars. The Vesic-Speck floor safety factors varied from 1.42 for the smallest size pillar to 1.87 for the largest size pillar.

The project team collaboratively worked with company management and mine operations staff to develop the alternate geometry and its implementation plans for field demonstration.

Task 3: Monitor Alternate Geometry Demonstration Area During Mining

The goal of this task was to monitor the performance of the alternate geometry demonstration area before and during mining. Two rows of convergence stations and rib stress measurement stations were installed in the demonstration area and the area is being regularly monitored. Analysis of the data will be performed during Phase II of the project.

OBJECTIVES

The goal of this project was to develop an alternate mining geometry for a sub-main development at a mine in Southern Illinois. The objectives of this demonstration were:

1. Higher extraction ratio with the alternate geometry.
2. Higher pillar and floor safety factors around the center of the sub-main section that will preserve the integrity of belt and power entries.
3. Higher long-term stability of the overall sub-main area that will preserve the integrity of primary airways.
4. Smaller sub-main foot print allowing more reserves for panel mining.
5. Higher production and productivity potential through cut sequence optimization.

To achieve the above broad objectives, the study was divided into two phases. In the first phase of the project, three major tasks were proposed. The first task involved collecting baseline data for the regular geometry as practiced currently at the mine. The second task involved designing the actual alternate geometry to be implemented in the demonstration area. The final task involved establishing a monitoring system in the alternate geometry demonstration area for long-term evaluation of the concept. Specific objectives of each task are given below.

Task 1: Collect Baseline Data on Current Mining Geometry

The first task primarily dealt with collection of data to be used in development of an alternate mining geometry for sub-main entries. Specific objectives of this task included:

1. Establishing a convergence monitoring system in the regular geometry area.
2. Establishing a rib stress measurement system in the regular geometry area.
3. Collecting geotechnical and lithological data.
4. Performing industrial engineering studies in the regular geometry area.
5. Conducting a ventilation survey in the regular geometry area.

Task 2: Develop Alternate Geometry for Sub-Main Demonstration

This task dealt with using data collected in Task 1 and ground control and production models developed at SIUC to design an alternate mining geometry for demonstration in sub-main entries. Specific objectives of this task included:

1. Designing several potential alternate geometries with higher extraction ratios.
2. Modeling these geometries using Panel3D and Phase2 Finite Element Analysis (FEA) software to evaluate structural stability of individual pillars and the panel as a whole.
3. Identifying suitable alternate geometry design options for company management to evaluate.
4. Preparing a report on the alternate geometry demonstration concepts as well as

pillar and floor safety factors in the proposed alternate geometry to assist company officials in obtaining necessary experimental permits.

Task 3: Monitor Alternate Geometry Demonstration Area During Mining

The goal of this task was to monitor the performance of the alternate geometry demonstration area during mining. Specific objectives of this task include:

1. Establishing a convergence monitoring system in the demonstration area.
2. Establishing a rib stress measurement system in the demonstration area.

INTRODUCTION AND BACKGROUND

Traditionally, room-and-pillar mining is associated with pillars of uniform size both in mains and panels (Figure 1). Though this layout lends to simplified mining cut sequences, the extraction ratio of the panel is sub-optimal. Room-and-pillar mining layouts, with about 50% extraction ratio typically result in a small amount of subsidence in the form of a trough due to settlement of pillars on the weak floor strata. Safety factors against roof failure and pillar failure based on coal strength vary across and along a panel for a given mining geometry. The term pillar safety factor (PSF) refers to failure of pillar based on coal strength while floor safety factor (FSF) refers to failure of floor or foundation failure. Factor of safety for roof failure in bending or shear failure is considered for roof stability. Safety factors vary within the panel because stress distribution across a panel is non-uniform. Designs for Illinois coal mines require minimum PSF and FSF to be 1.5 and 1.3, respectively.

To overcome the disadvantage of lower extraction ratio and reduced safety factors in the central belt entries, alternate mining geometries were proposed to the mining industry. Alternate mining geometries (Figure 2) have unequal pillar sizes, such that central entries have larger pillars as compared to end ones. Other possible alternate geometries may include a large pillar near the panel edge such that the load on smaller edge pillars can be transferred to the larger pillar. Additional advantages of this system include: (1) increased stability of the entire mining development, (2) increased extraction ratio, (3) enhanced productivity through cut sequence optimization, and, (4) reduced roof fall risk in the central belt entries. Roof failures due to pillar and floor instability are more probable around the center where critical infrastructure is located. For such a panel, stresses on pillars are highest around the center and lowest around the edges, thus putting belts and other infrastructure at risk.

Selection of alternate geometries is based on an optimization procedure that utilizes computed values of loading and strength of pillars spatially in a mining layout, and the load transfer from mined-out areas to unmined areas. Modeled safety factors for conventional (Figure 1) and alternate (Figure 2) geometries at a previous demonstration mine in Illinois are shown in Figures 3(a) and 3(b). It is apparent that moving from edge pillars to the center, safety factors decrease for the regular geometry while they increase

for the alternate geometry. Greater safety factors mean that vertical displacement is minimized resulting in less spalling and roof control problems. This is particularly beneficial in the center entry where the conveyor belt operates as it reduces the risk of roof falls.

Previous Demonstration of Alternate Geometry at a Deep Mine in Illinois

Chugh and Pytel (1992) researched ways to develop alternate mining geometries with variable size pillars along and across a panel based on pillar and floor safety factors and pillar settlement considerations. That research utilized the following three technical concepts impacting the mining geometry design: a) variability of safety factors along and across the panel due to differential pillar loading and settlements, b) elastic-plastic behavior of coal pillars resulting in the arching effect, and, c) elasto-visco-plastic behavior of the weak floor strata with load transfer from smaller pillars to larger pillars and panel barrier pillars. These concepts were analyzed and optimized using two-dimensional and three-dimensional SIU Ground Mechanics Models (Chugh and Pytel, 1992a; Chugh et. al., 2004). Objectives of the optimization procedure were to maximize coal recovery and productivity while maintaining appropriate pillar safety factors and immediate roof stability (Chugh and Pytel, 1992b). Based on results from these studies, an alternative mining geometry was demonstrated at a deep (~600 feet) coal mine in Illinois. The existing mining geometry at this mine utilized 80-ft. x 80-ft. pillars (c-c) with 20-ft. wide entries throughout the panel (Figure 1). SIU researchers developed and demonstrated the alternate mining geometry in an operating unit as shown in Figure 2 (Chugh et. al., 2001, Chugh et. al., 2003) in cooperation with the coal company. This geometry increased pillar sizes in the panel center to 90-ft x 75-ft (c-c) and decreased pillar sizes near the panel barriers to 50-ft x 75-ft (c-c). The implemented alternative geometry (Figure 2) increased extraction ratio by 2.3% from 45.7% to 48.0% while simultaneously improving both pillar and floor safety factors leading to significantly increased overall ground stability in the center portion of the panel where the belt conveyer system is located. A map of the demonstration area of this alternate mine geometry demonstration is presented in Figure 4.
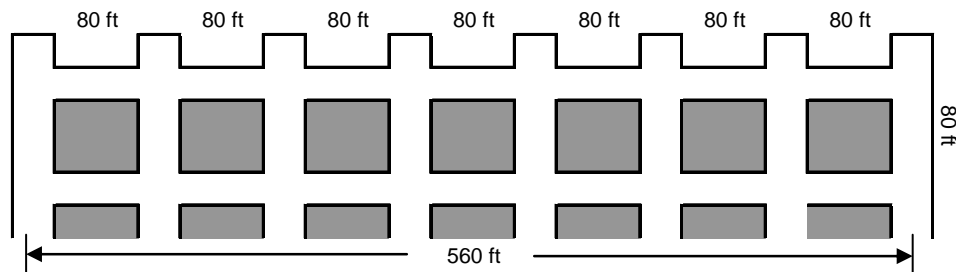

Figure 1.   Traditional Room-and-Pillar Mining Geometry From a Previous Demonstration
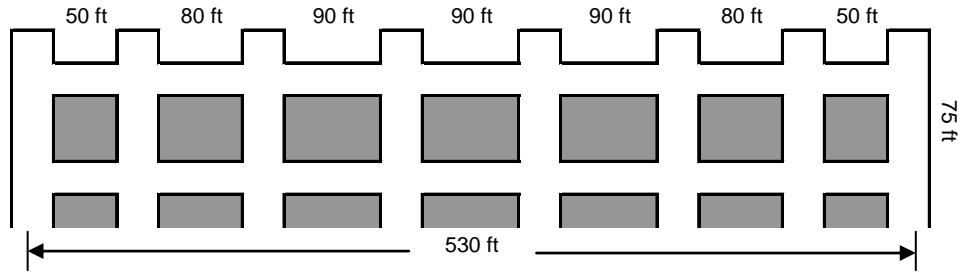
Figure 2.   Alternate Room-and-Pillar Mining Geometry From a Previous Demonstration
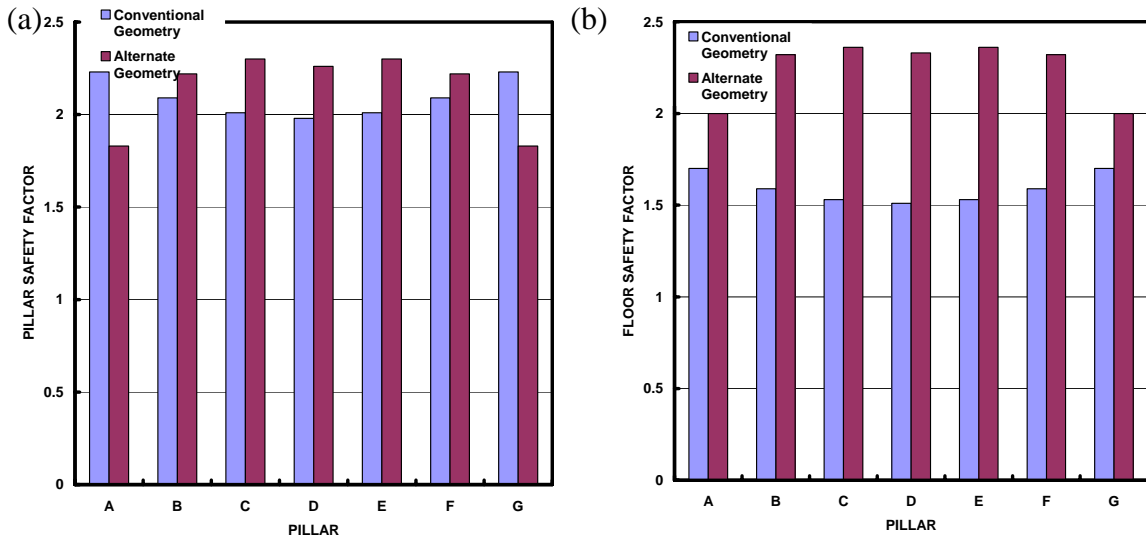

Figure 3.   Comparison of (a) PSF and (b) FSF for Regular and Alternate Geometries (Chugh-Pytel geometry)
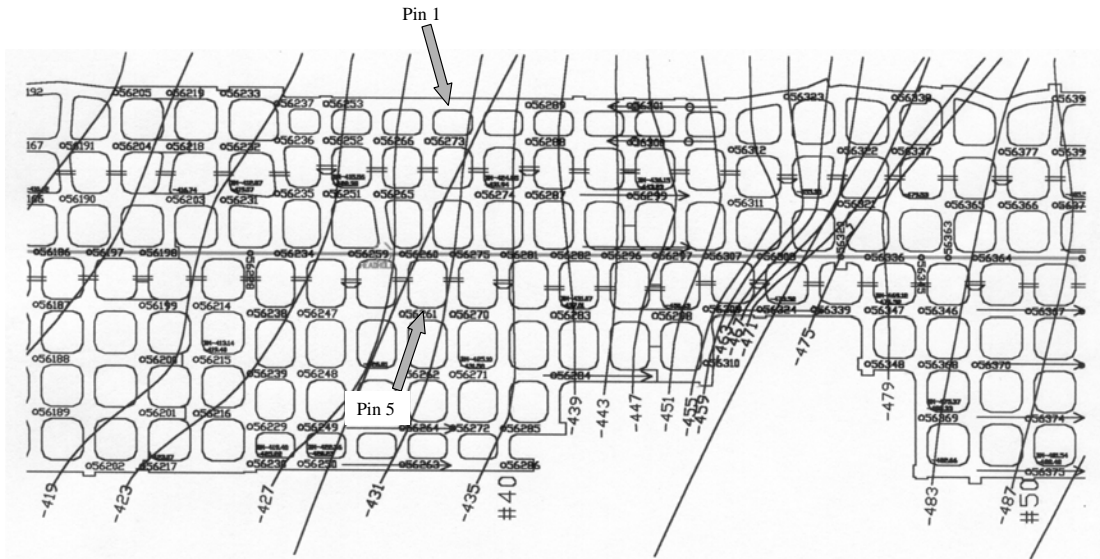

Figure 4.   Alternate Geometry Demonstration Area at Previous Demonstration Site

Ground conditions were observed over a period of one year through convergence measurements and visual observations of pillar rib spalling and floor heave, both for the traditional mine geometry as well as for the alternate mine geometry. These observations and studies indicated that ground conditions improved for both pillar and floor strata and rib-rash was comparable in both areas. Roof to floor convergence also diminished significantly in the alternate mining system. The lower convergence implied a reduced possibility of future squeezes in mined-out areas. In addition to increased extraction ratio for the panel, there were increases in productivity as lesser time was required to mine the outside edges of the panel because one cut blow–throughs were possible between the outer two entries. This allowed more time to be spent in the center of the panel where cycle times are lower and production rates higher. Overall, the development and demonstration of this alternate geometry was considered a success.

Geometry Practiced at the Proposed Demonstration Site

The demonstration site for this project mines Herrin #6 coal in Southern Illinois at a depth of 230 feet. The coal seam thickness averages 6.5 feet and it is overlain by gray shale 1-3 ft. thick followed by a competent limestone bed about 6 feet thick. The immediate floor stratum is claystone 1-3 ft. thick with 8.0% average moisture content. Standard entry width is 20 feet. Twelve sub-main entries are mined on 65 feet centers. Cross-cuts are also mined on 65 feet centers creating 45 feet square pillars. The extraction ratio within the mining area in the current geometry is 53.37%.

Comparison of the Two Demonstration Sites

There are some basic differences between the previous demonstration site and the demonstration site used in this project. Salient differences in geology and mine layout have been tabulated in Table 1.

Table 1.   Differences Between Previous and Current Demonstration Sites

Demonstration	Mining Depth (feet)	Weak Floor Strata Thickness (feet)	Seam Thickness (feet)	Panel Width (feet)	Panel Entries (#)
Previous	600	1.95	6	550	8
Current	231	2.33	6.5	645	12

Economic Benefits of Alternate Geometry

In addition to delivering higher extraction ratio, the alternate mining geometry has the potential for higher production per unit shift and lower ground control costs. Though no accurate economic analyses to assess gains due to better ground control have been conducted yet, it is expected that major economic benefits would accrue due to the lower incidence of roof falls and associated clean-up costs and productivity delays due to falls in belt entries and travel ways. Accurate quantification of these benefits are planned to

be performed in Phase II of this project. It is expected that benefits will result from the following:

1. Increased extraction ratio - Extraction ratios in typical mains and sub-main mined in Illinois vary from 46-48%. Use of alternate geometry designs will help mines to increase their extraction ratios by about 3%. Cost advantages accrue because of higher returns on the same feet of development advance. Also, the production cost of the incremental coal is expected to be low.
2. Decreased footprint of main and sub-main entries - Smaller edge pillars will narrow the total width of main or sub-main sections. Not only will this increase the overall safety factor of the section but it also implies that a smaller portion of the coal reserve is committed to developing mains and sub-main. This frees up coal reserves for panel mining where extraction ratios are higher and production costs are lower.
3. Improved production per unit shift through cut sequence optimization - Smaller edge pillars could potentially change a number of two cut cross-cut hole-throughs to single-cut hole-throughs while simultaneously decreasing haul distances due to decreased footprint.
4. Reduced convergence in central mining entries - Reduced convergence reduces the risk of roof falls. Of particular importance may be the reduction in the frequency of roof falls in the central belt entry, where roof falls can completely halt production.
5. Reduced rib spalling - Higher pillar safety factors around central entries should lead to reduced rib spalling. This effectively increases the usable portion of the entry width and reduces time and cost associated with cleaning up the sloughing material.
6. Increased stability of long-term mine development - Alternate mining geometry is expected to provide higher long-term stability which should be beneficial for mains and sub-main which must stand for longer periods of time.

EXPERIMENTAL PROCEDURES

This project was proposed to collect baseline performance data on regular geometry with uniform size pillars currently used in sub-main at the mine and compare it to an alternate geometry with non-uniform size pillars designed and demonstrated as part of this project. Associated geologic, geotechnical, and mine environment data were also collected and compared. In this first phase of the project, three major experimental and modeling procedures were performed. These included: 1) Establishing convergence stations, 2) Establishing rib stress measurement stations, and 3) Finite Element Analysis (FEA) modeling of the regular and alternate geometry. These are described in detail below.

Convergence Stations

Convergence stations monitor total movement between roof and floor. A schematic diagram of a convergence station used in this study is given in Figure 5. For installation of these stations, a plumb bob was hung from a roof bolt to accurately mark a point

vertically below it. A 1.5 feet deep hole was drilled into the floor using a Schroeder drill. A convergence pin (length of 1 foot and diameter of 5/8-inch) with polished head was then grouted into the hole in the floor. A plastic spacer was inserted over the bolt and the spacer was filled with sponge type filling material. The spacer was then covered with a metal ring with a string attached to it. This allowed quick access to the bolt during convergence monitoring.
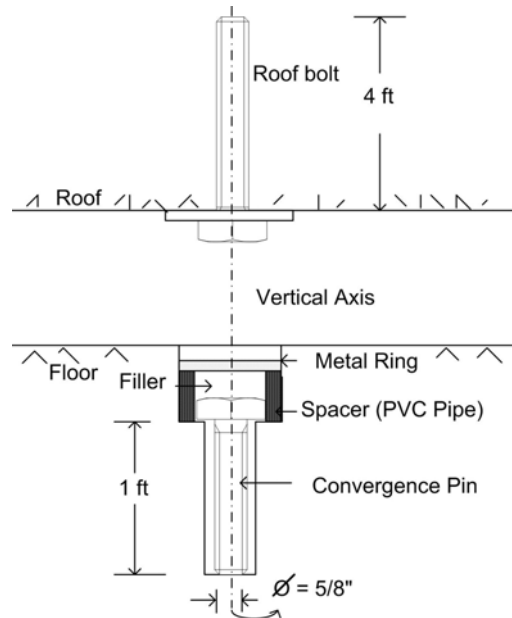

Figure 5.   Schematic Diagram of a Convergence Station

Five (5) rows of convergence stations were established with three (3) in the regular geometry and two (2) in the alternate geometry. These stations were located both in intersections as well as in entries adjacent to pillars of all sizes. There were 14-18 convergence stations per row and each row installed was in the cross-cut behind the last open cross-cut at the time of installation.

Convergence readings were taken at suitable intervals using an Invar Tube extensometer (Soil Test Inc.). The extensometer consisted of two concentric tubes, one fixed and the other movable. The displacement of the movable tube could be read on a dial gage (0 +/- 0.001 inch) to determine total linear distance between the roof and the floor pins below it. Comparison with previous readings was used to determine convergence/ divergence values at each point over the time interval.

Rib Stress Measurement Stations

Rib stress measurement stations were used to monitor incremental stresses in pillar ribs of the regular and alternate geometries. To establish a rib stress measurement station (Figure 6), three holes were drilled as a rectangular rosette and bolts were grouted into each hole. Bolts were 2 feet in length and 5/8-inch in diameter. These bolts were polished at the top and the sides to reduce measurement errors. Five (5) rows of rib stress measurement stations (three in the regular geometry and two in the alternate geometry)

with 4-6 stations per row. Each row was installed in the pillars just oubye the last open cross-cut at the time of installation. Horizontal, vertical and diagonal measurements were taken at suitable intervals using a vernier caliper (0 +/- 0.001 inch). These data were utilized to assess incremental vertical and horizontal stresses in pillar ribs using strain rosette equations (Goodman, 1980).
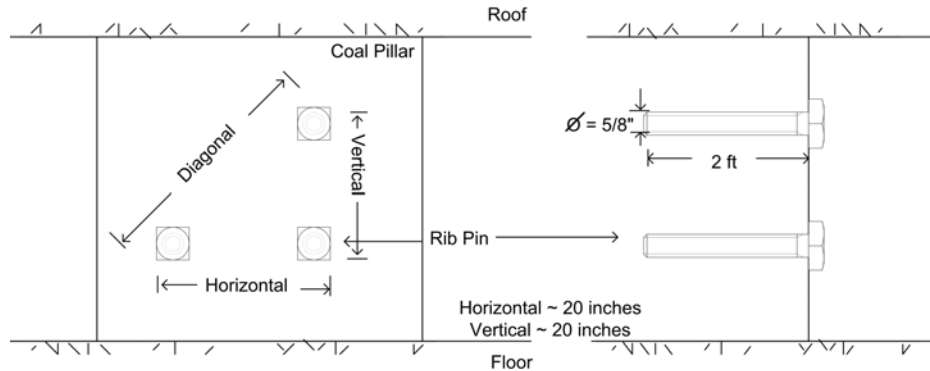

Figure 6.   Schematic Diagram of a Rib Rtress Measurement Station

Finite Element Analysis (FEA)

Two dimensional finite element models were created using Phase2 (RockScience Inc.) FEA software to model regular as well as alternate geometries. Half barrier pillar width of 50 feet was assumed on both sides of the panel. Regular and alternate geometry sections were modeled based on the lithology of boreholes closest to the regular and alternate geometry demonstration areas. Table 2 gives the lithology of the borehole closest to the area with regular geometry while Table 3 gives the lithology of the borehole closest to the alternate geometry. Table 4 gives values of geotechnical parameters for different strata used in FEA models. The models were analyzed with both vertical and horizontal stress. Vertical stress of 325 psi was applied about 80 feet above the coal seam on the model and was simulated as uniform loading on the model. Uniform horizontal stress of 1,000 psi was applied to the model. This was simulated by setting a displacement in the negative direction due to applied 1,000 psi horizontal stress. This allowed different lithologies to assume different horizontal stresses based on their stiffness. The bottom portion of the model was restrained in the vertical direction, but was allowed to move horizontally, thus allowing displacement of the rock mass. The model was meshed with graded four-node quadrilateral elements with a gradation factor of 0.1 away from the excavation boundary. Mesh size was non-uniform with approximately 1,500 elements per excavation (entry perimeter). The failure analysis was run using Mohr-Coulomb failure criterion. Figure 9 shows a screenshot from the ground control model for an alternate geometry design.

Table 2.   Borehole Lithology around the Regular Geometry Area

Gr. Grn. Silty Shale	6.6 ft	
Limey Shale	8.5 ft	
Gr. Limestone	3 ft	
Black Shale	3.2 ft	
Dark Fossil Shale	1.1 ft	Roof
Black Limey Shale	3.8 ft	
Dark Fossil Limestone	2.8 ft	
Black Shale	2.1 ft	
Black Shale w/Sulfur	1.1 ft	
Coal	6.9 ft	Coal
Claystone	0.6 ft	
Cong. Lime & Shale	0.5 ft	Floor
Gray Limestone	2 ft	

Table 3.   Borehole Lithology around the Alternate Geometry Area

Limestone	4.1 ft	
Grn Gr. Striped Shale	3.8 ft	
Limestone	3.9 ft	
Black Shale	2.6 ft	
Core Loss	0.9 ft	
Dark Fossil Shale	1.9 ft	Roof
Dark Fossil Limestone	1.3 ft	
Dark Fossil Shale	3 ft	
Dark Fossil Limestone	2 ft	
Black Shale w/ Sulfur	1.8 ft	
Black Fossil Shale	0.4 ft	
Black Shale	2.7 ft	
Coal	6.7 ft	Coal
Claystone	1 ft	Floor

Table 4.   Geotechnical Material Properties Used n FEA Models

	E (kpsi)	μ	Tensile Strength (psi)	Fric. Angle (deg)	Cohesion (psi)
Coal	150	0.25	250	26	850
Claystone	200	0.35	70	18	250
Shale	300	0.20	200	25	70
Sandstone	1,800	0.25	300	20	1,000
Weak Limestone	300	0.25	300	20	1,000
Limestone	700	0.20	300	20	1,000
Competent Clay	208	0.25	70	18	175

Mohr-Coulomb Failure Criterion

Mohr-Coulomb criterion is one of the most common failure criterion used to calculate rock (brittle) failure or yielding. It describes the limiting relationship between normal and shear strengths on a plane at failure (assuming tensile stress is positive). This criterion suggests that chances of failure are high when the stress at a point is close to the Mohr's circle envelope. (RocScience, 2006)

The direct shear formulation of the criterion (denoted by the strength envelope in Figure 7) is given by the following equation:

$$\tau = C + \sigma_n \tan \phi \tag{1}$$

where, C is the cohesive strength,
ϕ is the angle of internal friction,
σ_n is the normal stress, and
τ is the shear strength

The Phase2 FEA program also calculates equivalent Mohr-Coulomb parameters for non-linear failure envelopes over a specified stress range.

Modeling a 3D Mining Geometry using a FEA 2D Model

Phase2 is a two-dimensional finite element analysis software which assumes that the constructed geometry extends infinitely in both positive and negative 'z' directions (plain strain analysis). Thus, modeling the regular and alternate geometry with the overburden vertical stress (σ_v) will result in an erroneous calculation since the model will assume no cross-cuts in the panel. Thus, the concept of "equivalent vertical stress" on the panel was used to model the effect of cross-cuts in the mining geometry.

Figure 8 shows the cross section around a pillar that is modeled in Phase2. The FEA model geometry (Figure 9) can be derived from the 3D geometry of a panel by analysing the panel across a vertical plane (AA'). Thus, to model the effect of additional extraction in the cross-cuts, the vertical stress on the model is increased by the ratio of the area carrying the load prior to mining to the area of the pillar after mining. In the z-direction, the load was carried by the area EFGH prior to mining and by the area ABCD after mining.

Thus, effective stress on the pillar $= \sigma_v \times \dfrac{\text{area(EFGH)}}{\text{area(ABCD)}}$

$$= \sigma_v \times \dfrac{(D + E) \times W}{D \times W}$$

where, D = length of the pillar,
W = width of the pillar, and
E = entry width
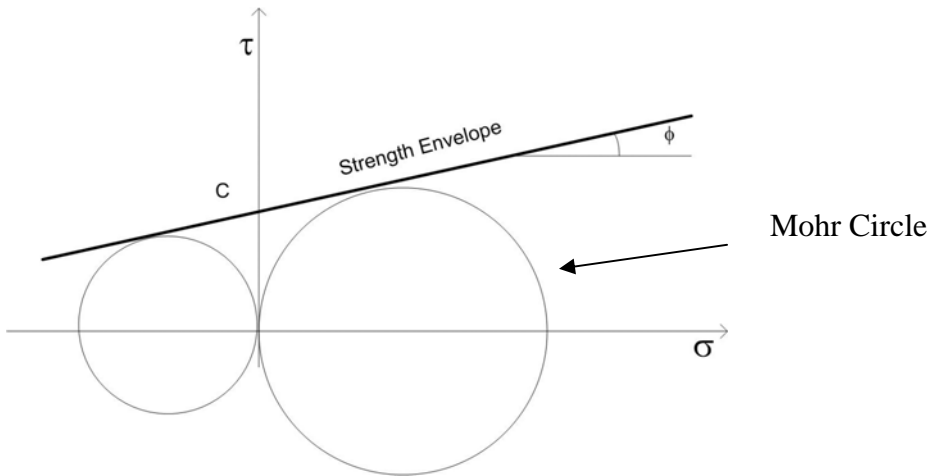
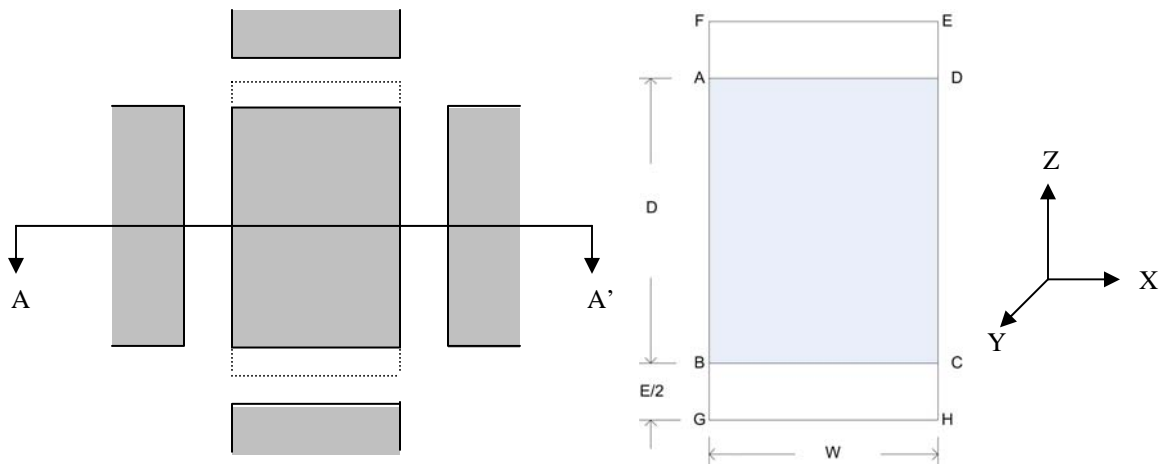Figure 7.   Mohr-Coulomb Failure Envelope in Graphical Form

Figure 8.   Calculation of Equivalent Vertical Stress

325 psi

1000 psi

1000 psi

745 feet

Y

X

Z

Figure 9.   Screenshot of FEA Model of Alternate Geometry Using Phase2 FEA Software

Optimizing Cut Sequences

Cut sequencing refers to the order in which continous miner cuts are made as a mining section advances. Each of the numbered blocks in Figure 10 identifies a single cut and its length typically varies from 20-40 ft. based on mining conditions. All of the cuts shown in the figure make up a cut cycle, which is usually coordinated with belt and power moves required to keep mine infrastructure in close proximity to mining operations at the face. Optimizing the sequence of cuts provides for efficient utilization of mining equipment while meeting requirements for a stable mine environment such as providing sufficient quantities of ventilation air and reducing exposure to dust. In designing the mine geometry with pillar and floor safety factors, consideration must also be given to entry spacing with the objective of minimizing change-out distances and the amount of time spent mining crosscuts.

22	41	42	23	43	44	24	45	46	25	47	48	26	49	50	27	51	52	28
15			16			17			18			19			20			21
8	29	30	9	31	32	10	33	34	11	35	36	12	37	38	13	39	40	14
1			2			3			4			5			6			7

Figure 10. Example of a Cut Cycle Showing Individual Cuts To Be Mined

Dynamic programming is an optimization technique that has been used in other mining scenarios to solve the problem of scheduling mining sequences. It is a recursive or step-by-step approach with decisions made after analysis at each step that provide information used in succeeding analyses and decisions. The following must be defined in order to use dynamic programming in optimizing cut sequences:

- stages,
- feasible states at each stage,
- optimal value function, and
- recurrence relation.

In room-and-pillar cut sequences, stages are cuts. The objective at each stage is to select

a cut for mining that best satisfies the optimal value function. Only those cuts that are feasible can be selected from. To be feasible, the cut must be accessible by way of previous cuts that have been mined and bolted. The optimal value function can be a maximization or minimization problem, such as maximization of production in a given timeframe (unit shift productivity) or minimization of haulage distance or delays (equipment utilization). The recurrence relation takes into consideration the repetitive nature of cut sequences, such as haulage distances based on mining geometry, roof bolting constraints based on time and space, and movement of the miner, also based on time and space. Other irregular factors, such as the presence of abnormal geologic conditions that create constraints, can be formulated and included as well. In dynamic programming, an optimal decision is made at each stage, and this process continues through all stages until the optimal value function is optimized in the last stage giving an optimal cut sequence.

Development of a minimization optimal value function to evaluate alternate geometry cut sequences was begun as part of this project with the objective of minimizing cut cycle time. All of the cuts in a predetermined mining cycle such as that shown in Figure 1 must be mined. At the end of each cut, the decision must be made as to which cut to mine next. At each decision point, the optimal value function is used to select the feasible cut with the lowest cycle time. For the purposes of this evaluation, cycle time is a function of car haulage distance between the face and feeder, miner tram distance from the previous cut, the ventilation configuration and a constraint based on time available for roof bolting previously mined cuts.

The optimal value function is given as:

Minimize CCT

where, $CCT = \{BF * VF * [(HD * COF) + (TD * CHF)]\}$ (2)

and,   CCT   = cut cycle time
         BF   = bolting factor or constraint
         VF   = ventilation factor or constraint
         HD   = car haulage distance from face to feeder
         COF   = change-out factor or constraint based on change-out distance
         TD   = miner tram distance from previous cut
         CF   = cable handling factor or constraint.

A brief explanation of each component follows.

Bolting Factor
To be considered feasible, a cut must be accessible through previously mined cuts. However, complete accessibility requires previously mined cuts to be bolted. Thus, the bolting factor is used to render the most recently mined cuts unlikely candidates for the next cut. Depending on the bolting capacity of the mine, the bolting factor can also be used to maintain a one or two cut buffer between mining and bolting functions.

Ventilation Factor
Cross-cuts can be the most difficult cuts in the cut sequence cycle due to long and awkward change-outs for the haulage equipment and the inefficiencies of "turning" a cross-cut. However, keeping cross-cuts caught up with main entry advance is critical in maintaining adequate ventilation and providing haulage equipment access to change-out points that are as close to the face as possible. The ventilation factor is used to make feasible cuts that would begin or complete a cross-cut preferable to other cuts. It is also used to insure that cross-cuts are "turned" in the proper direction based on scrubber configuration and direction of ventilation air flow at the face.

Haulage Distance
This component is simply the measurable distance from face to feeder that each haulage unit must travel. The most direct route should be used.

Change-out Factor
Change-out distance is included within haulage distance for loaded cars but must still be accounted for otherwise because the miner has to wait for an empty car to arrive after the loaded car leaves. Because change-out distance is a major factor in minimizing cut cycle time, the change-out factor is utilized to give preference to feasible cuts with the shortest change-out distance.

Tram Distance
This component is simply the measurable distance that the miner has to tram from the just completed cut to the next feasible cut.

Cable Handling Factor
As the miner moves from cut to cut, it is either pulling slack cable or picking it up. Little time and effort is involved in pulling cable whereas considerable time and effort is required to handle the miner cable when slack has to be picked up. The cable handling factor promotes sequencing of cuts such that the number of times that slack cable has to be picked up is minimized and when it is done, the miner cable is positioned so that the miner is able to make a number of cuts without rehandling the cable.

At this point, shift reports from the mine hosting the alternate geometry demonstration are being examined to compare actual cut sequences mined with the cut sequence suggested by the dynamic programming model. This comparison will enable a more accurate definition of each of the constraint factors. The final model factors will be reported in Phase II of this project.

RESULTS AND DISCUSSION

Task 1: Collect Baseline Data on Current Mining Geometry

Three rows of convergence stations and rib stress monitoring stations were installed in

the regular geometry. Monitoring results for regular geometry in the 3rd row are summarized in Figure 11. Central entries and pillars (e.g. entry 6 - 7), exhibit higher convergence values as compared to edge pillars (eg. entry 11-13). Roof strata in the regular geometry bends to form a "concave' shape, which indicates higher loading on central pillars and entries. However, since floor strata on the left-hand side was weaker than on the right-hand side, the absolute value of convergence on this side is larger.
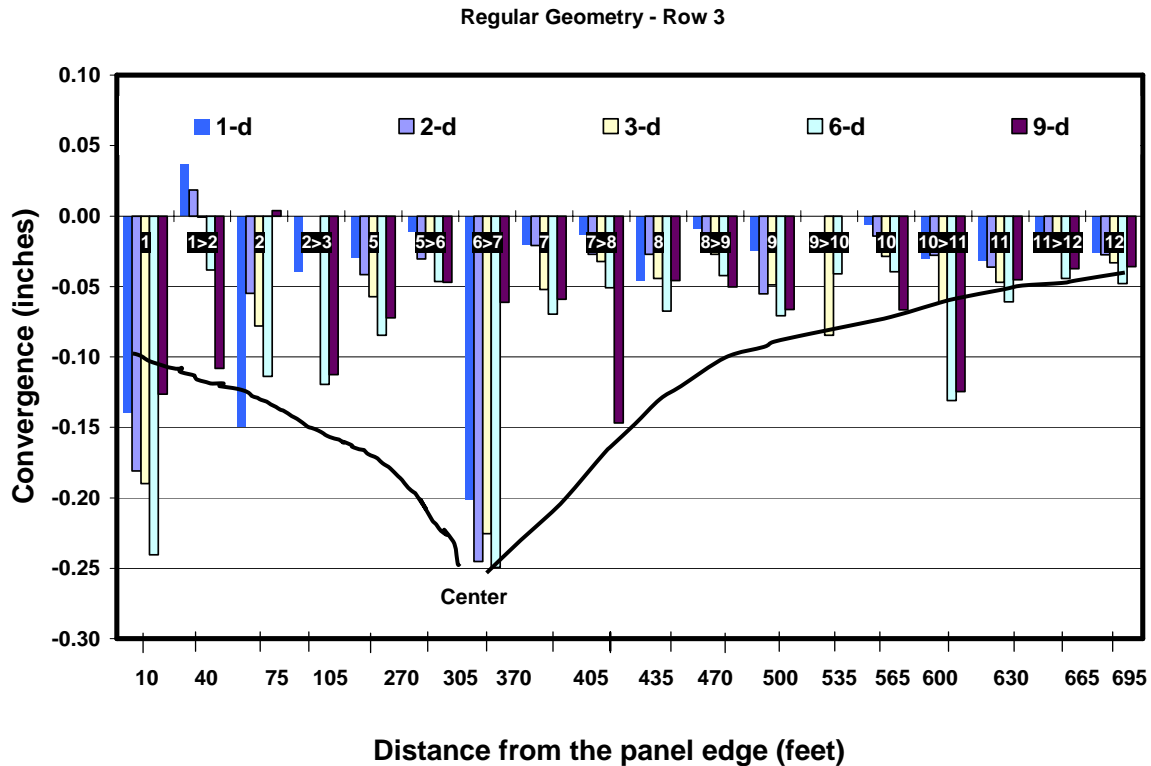
Figure 11.   Convergence Observed over Various Days in Row 3 of Regular Geometry

Regular monitoring of rib stress stations was conducted and the changes in the vertical, horizontal and diagonal distances between the rosettes recorded. The ratio of change in distance of vertical, horizontal and diagonal to the original distances were the strains in the 0, 45 and 90 degree directions, respectively. These were used to calculate the principal strains using strain transformation equations. Results will be included in the Phase II report.

Apart from convergence readings and rib stress measurements, the project team also collected geotechnical data from the mine and its company headquarters. These were used for safety factor calculations in Task 2. Table 5 gives the mining and geotechnical parameter values obtained from tests conducted by the mine.

Table 5.   Mining Parameter Values Obtained From Company Tests and Data

	Parameter	Value
1.	Average bearing capacity (based on 9-in. X 9-in. plate loading tests) of immediate floor strata	725 psi
2.	Immediate floor thickness (maximum)	2.33 feet
3.	Moisture content of floor strata	8.0 %
4.	Seam height	6.5 feet
5.	Overburden depth (maximum)	231 feet
6.	Entry width	20 feet
7.	Critical size coal strength (σ_{cc})	900 psi

Task 2: Develop Alternate Geometry for Sub-Main Demonstration

Design of alternate geometries was based on an optimization procedure that utilized computed values of loading and strength of pillars in a mining layout, and load transfer across pillars and to unmined areas. It also included production modeling and cost and delay analysis of mining systems to achieve a better overall system for producing coal at a lower cost. Out of many geometries modeled, two alternate geometry options were short-listed for this site (Figure 12b, 12c). Both geometries achieved the desired benefits of greater extraction ratio, increased ground stability and improved productivity. One option followed the pattern established in the first demonstration site with the largest pillars around the panel center and pillar size continuously decreasing towards the panel edges as shown in Figure 12(b). The other option had the largest pillars around the panel center but also included a slightly larger pillar near each edge of the panel with smaller pillars on either side as shown in Figure 12(c). These larger pillars were positioned such that the load from the smaller pillars can be effectively arched onto larger pillars and barrier pillars.

Both geometries above were modeled using Phase2 FEA software. Safety factors and expected convergence in the entries were also calculated from the model. Simple production models were run to predict the production potential of proposed alternate geometries. These analyses were performed in collaboration with mine operations staff and corporate office technical professionals. The project team in concert with mine management decided to demonstrate alternate geometry #1 (Figure 12-b) at the mine. This was because the previous demonstration had successfully demonstrated this configuration. This geometry increased the extraction ratio of the panel from 53.37% to 56.33% while reducing the panel footprint from 715 feet to 625 feet. After finalizing the mining geometry to be demonstrated, the project team calculated pillar and floor safety factors using tributary area theory to help mine management in obtaining an experimental permit for field demonstration.
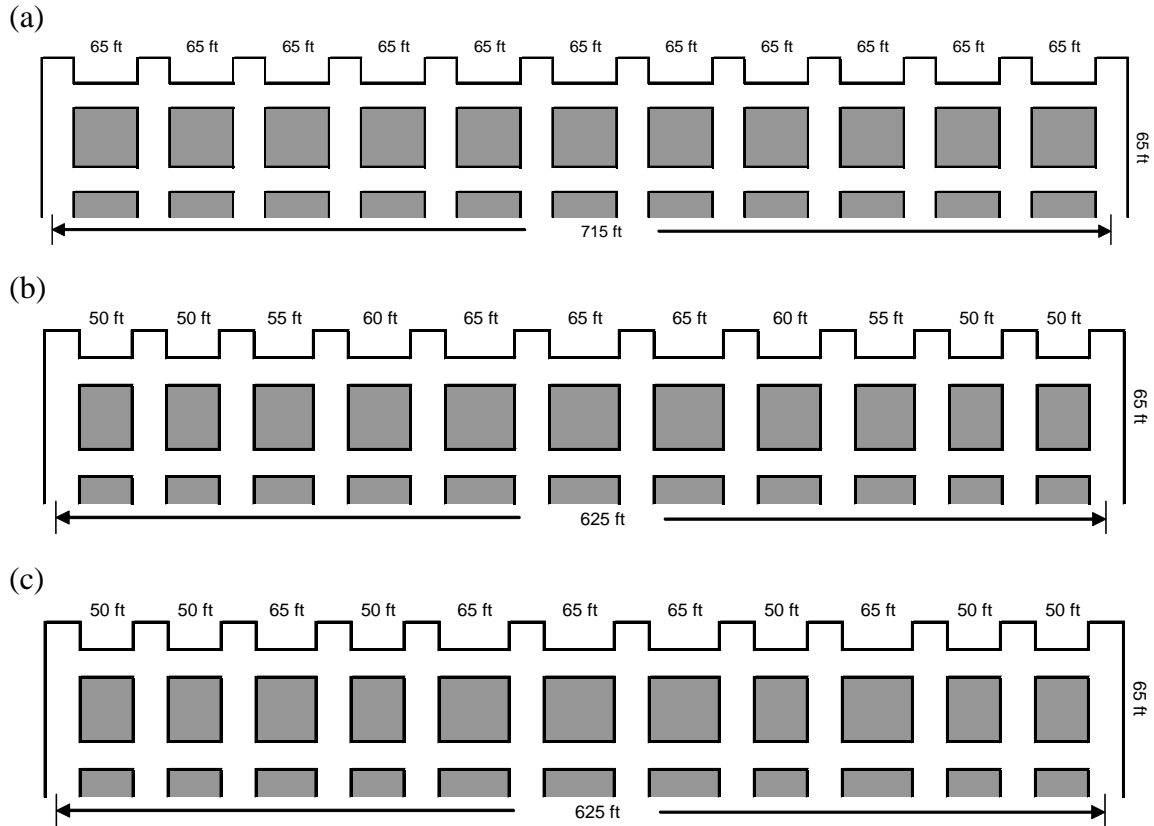
18

(a)

| 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft | 65 ft |

65 ft

715 ft

(b)

| 50 ft | 50 ft | 55 ft | 60 ft | 65 ft | 65 ft | 65 ft | 60 ft | 55 ft | 50 ft | 50 ft |

65 ft

625 ft

(c)

| 50 ft | 50 ft | 65 ft | 50 ft | 65 ft | 65 ft | 65 ft | 50 ft | 65 ft | 50 ft | 50 ft |

65 ft

625 ft

Figure 12.   (a) Regular Geometry, Alternate Geometry Options (b) #1 and (c) #2

Calculation of Pillar and Floor Safety Factor using Tributary Area Theory

Pillar and floor safety factors were calculated for each of the various pillar sizes in the alternate geometry shown in Figure 13. For instance, pillar type I indicates size of 50-feet x 65-feet. Vesic-Speck approach (Chugh and Hao, 1992)) was used to calculate the floor safety factors. Vesic-Speck floor safety factors are conservative estimates since they assume the angle of internal friction (ϕ) to be zero, while the value of ϕ typically varies between 15-18 degrees for Illinois mines. Furthermore, tributary area loading represents maximum loading on pillars. Calculations for PSF and FSF for Pillar type I are shown as an example below.

Pillar Type	I		II	III		IV		III	II		I
	50	50	55	60	65	65	65	60	55	50	50
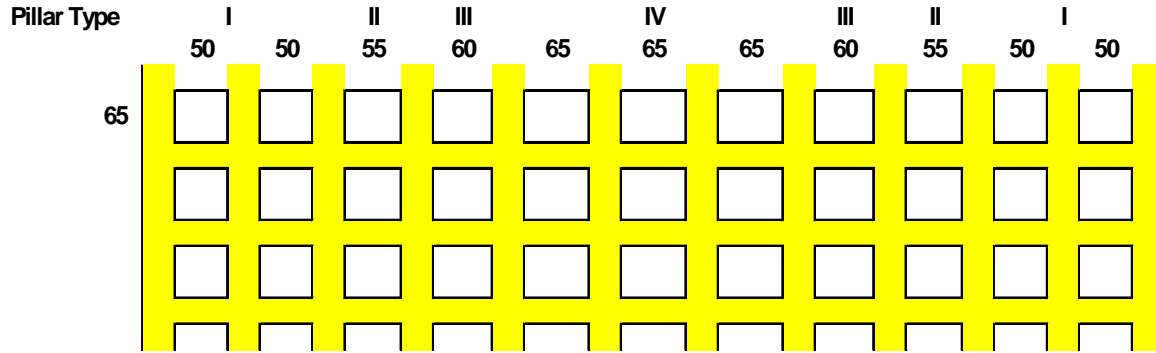
Figure 13.   Pillar Types of the Demonstrated Alternate Geometry

Calculation of Pillar Safety Factor (Pillar Type I)

Using the Holland formula (Holland, 1964; Holland, 1973), in-situ pillar strength is given by:

In-situ pillar strength $(\sigma_p) = (\sigma_{cc})$ X $\sqrt{\dfrac{Wp}{h}}$

where, Wp = pillar width = 30 feet
      h = seam height = 6.5 feet
and, σ_{cc} = Critical size coal strength = 900 psi

Thus,   In-situ pillar strength (σ_p) = 900 X $\sqrt{\dfrac{30}{6.5}}$ = 1,933 psi.

Now, Pillar Safety Factor = $\dfrac{\sigma_p}{1.1 x D / 1-e}$

where, D = depth of cover = 230 feet
      e = extraction ratio = 58.5 % (for Type I pillar)

Thus, Pillar Safety Factor = $\dfrac{1933}{1.1 x 230 / 1-0.585}$ = 3.16

Calculation of Floor Safety Factor (Pillar Type I)

The authors used results of plate loading tests conducted by mine personnel.

Average Bearing capacity (based on 9-inch x 9-inch plate loading tests) = 725 psi

Cohesion $(S_1) = \dfrac{\text{Bearing Capacity}}{N_c^*}$ (Chugh and Hao, 1992)

         = 725 / 6.17 = 117 psi,
where, N_c^* = 6.17 (assuming $\phi = 0$)

Now, Ultimate Bearing Capacity (Pillar) is given by:

$$q_O = S_1 N_m \ ,$$

where, N_m = modified bearing capacity factor

Vesic (1970) proposed the following equation for the determination of N_m:

$$N_m = \frac{KN_C^*(N_C^* + \beta - 1)[(K+1)N_C^{*2} + (1+K\beta)N_C^* + \beta - 1]}{[K(K+1)N_C^* + K + \beta - 1][(N_C^* + \beta)N_C^* + \beta - 1] - (KN_C^* + \beta - 1)(N_C^* + 1)}$$

where, K= ratio of unconfined shear strength of lower hard layer (S_2) to upper weak layer (S_1)

and $\beta = \dfrac{BL}{[2(B+L)H]}$ can be found from the width (B), length (L) and thickness (H) of the foundation. B and L correspond to pillar width (W_p) and pillar length (W_l), respectively.

Thus, UBC = 865 psi
σ_p = 1.1 X D / (1-e) = 610 psi (for Type I pillars)

Floor Safety Factor = 865/ σ_p = 865/610 = 1.42

Table 6 lists calculated floor and pillar safety factors for different pillars in the alternate geometry. It can be observed that pillar safety factors are greater than the required 1.5 and floor safety factors are greater than the required 1.3. Thus, the extraction ratio could be further increased in the alternate geometry without negatively impacting structural stability of the panel. Since limited geologic data was available for the area, it was decided to limit the initial demonstration to a 3% increase in extraction ratio.

Table 6.   Safety factor of different pillars in the panel

Pillar Type	Pillar Width (solid) feet	Pillar Length (solid) feet	Extraction Ratio (%)	Pillar Floor Bearing capacity (psi)	Vesic-Speck Floor Safety Factor	Pillar Safety Factor (Holland)
I	30	45	58.5	865	1.42	3.16
II	35	45	55.9	911	1.59	3.64
III	40	45	53.8	950	1.73	4.07
IV	45	45	52.1	985	1.87	4.48

Figure 14 shows the safety factor at various points along the floor and roof in regular and alternate geometries obtained from FEA modeling. The regular geometry have higher safety factors at the edge pillars and lower in the centers. Since PSF and FSF values are larger than 3, pillar sizes could be reduced to increase extraction ratio without affecting structural stability of panel or individual pillars.

The developed alternate geometry has relatively uniform PSF values (~3.5) across the

panel. These values are much higher than required to ensure structural stability of coal pillars. FSF values for the alternate geometry are higher near the center than around the edges as designed. Values (2.75 to 3.0) are still much higher than required. Thus, there is potential to increase extraction ratio further without compromising structural stability of the panel. It was decided to limit extraction ratio gains to about 3% to gain operational experience with this alternate geometry before increasing extraction ratio further.

(a)

(b)
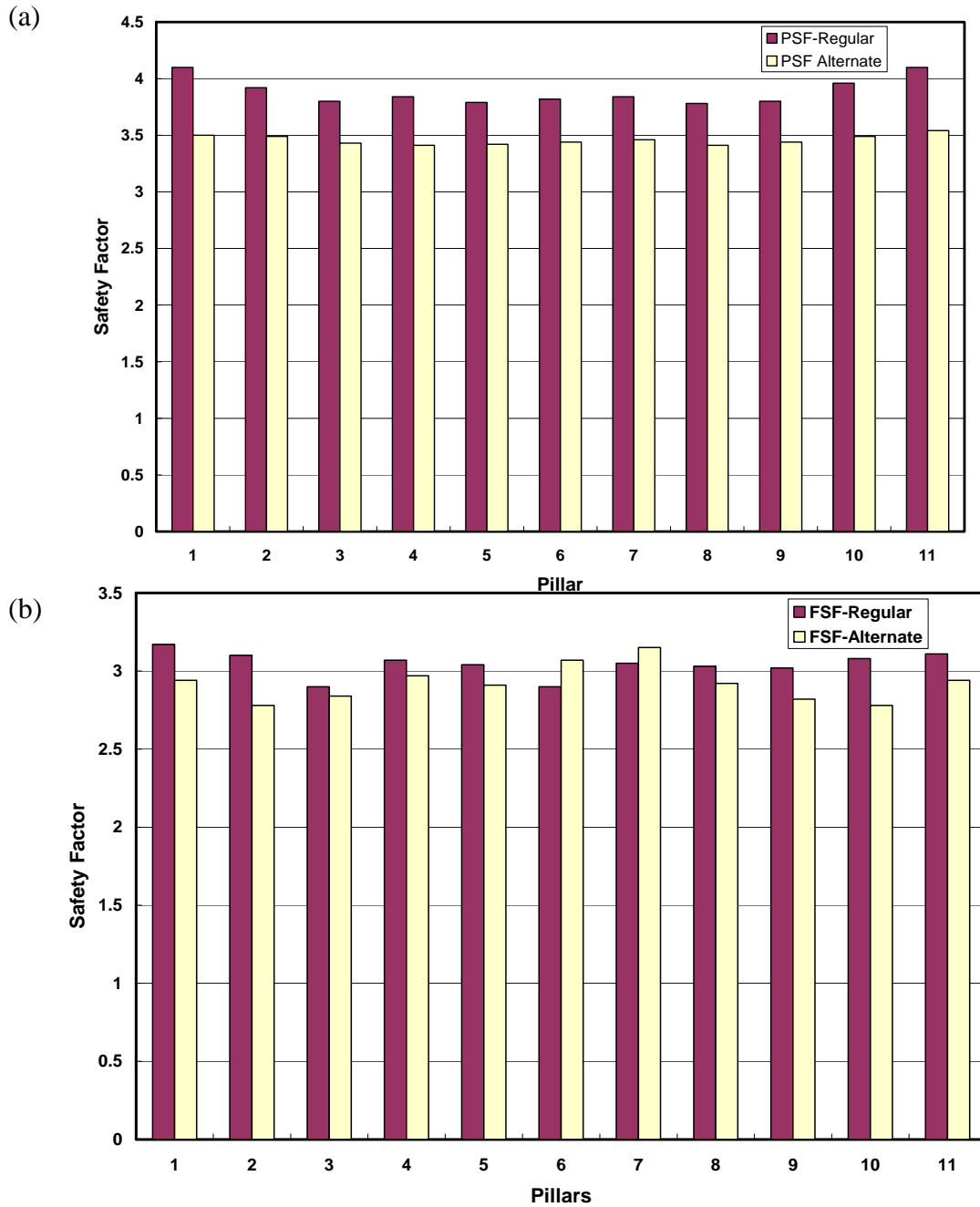
Figure 14.   Comparison of Safety Factors of (a) Pillar and (b) Floor Across a Panel for Regular and Alternate Geometries

Figure 15 shows point values of safety factors as computed by the finite element analysis model for different geometries. Numbers in bold represent safety factors in regular geometry while numbers above them represent safety factors in alternate geometry #1. Figures 15(a) and (b) show that the regular geometry is over designed around outside entries, but it is also more than adequate around the center of the panel. This would suggest that maintaining the regular geometry pillar size in the center of the panel while decreasing pillar sizes around the edges should achieve the desired objective of extracting more coal while maintaining ground control stability. It also indicates that alternate geometry #1 would be stable. Similar analysis for alternate geometry #2 indicated that alternate geometry #1 was more stable than alternate geometry #2. It can also be observed that safety factors obtained from FEA modeling are higher than that of the safety factors obtained using the tributary area method (Table 5). This is because the FEA model assumes angle of internal friction to be between 18 to 26 degrees (see Table 3) for various strata. In addition, the model incorporates lateral stress of 1,000 psi. Figure 16 shows FEA model screenshots of the safety factor of an alternate geometry around the central and edge pillars.
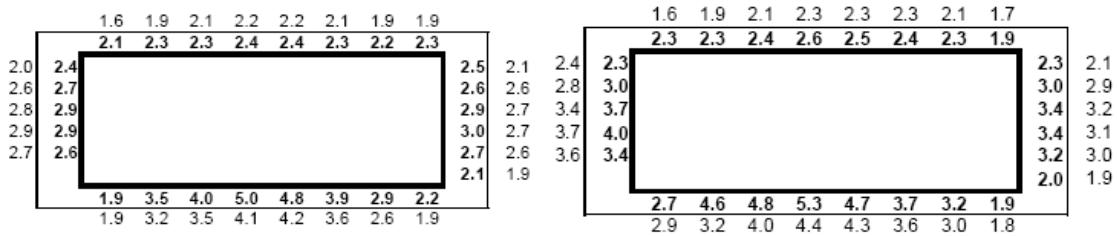
Figure 15.   Safety Factors from Phase2 Model Output for: (a) Central Entry, (b) Edge Entry for Alternate Geometry#1
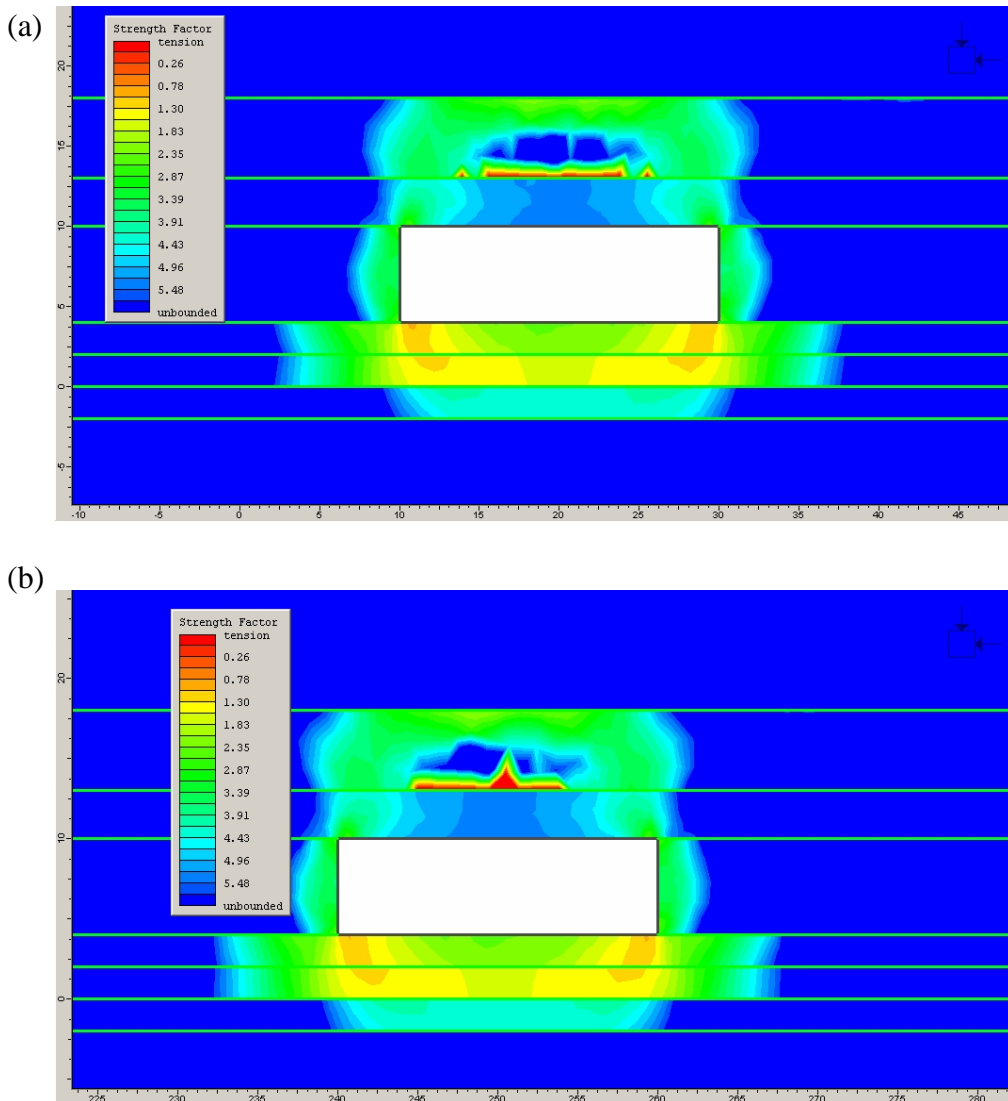
Figure 16.   Phase2 Screenshots of Safety Factors Around the (a) Edge Entry, (b) Central Entry in Alternate Geometry#1

Productivity Analyses

Underground coal mining involves repetitive elemental tasks often referred to as cycles. Productivity improvements involve minimizing cycle times and eliminating non-productive delays in those cycles. In regular geometry, following a standardized cut sequence may not produce maximum productivity. Furthermore, regulations typically restrict maximum cut depth to 40 ft. Since the pillar size in regular geometry is 55 ft, two cuts are required to hole through every crosscut given the uniform pillar size throughout the section and at least one of those cuts is a short one, which reduces productivity. Also, negative productivity occurs at the panel edges where wider than needed pillars mean longer haulage distances and longer cycle times. The continuous miner power cable has limited reach and reaching the end of the panels may sometimes require shifting of the

cable slack. Apart from being a non-productive delay, it may also restrict movement of other equipment and decrease equipment utilization.

A production and cost model of the current geometry mining sequence was developed using the SIU-Suboleski (SSP) Model (Chugh, et al., 2005) to establish a base case scenario. This model was calibrated with industrial engineering data collected by the mine's process improvement team. The sub-mains used a 12-entry split air super section producing around 3400 raw tons/ shift working 213 days/ year and 3 shifts/day. The panel used two Joy 12 CM continuous miners, four battery ram cars (9-10 ton capacity) and three double boom roof bolters. The alternate geometries were also modeled using SSP to quantify potential productivity gains related to shorter CM cable distances and haulage cycles associated with narrower panels. Cut sequences were developed to maximize the number of crosscuts that can be completed in a single cut due to smaller pillars at the panel edges. Apart from assessing the production and production cost of these individual geometries, the delays associated with various cuts in both traditional and alternate geometries were compared. This will be used in future work to delineate the cuts which have the most delays to improve the cut sequences of the alternate geometries. This analysis assumes similar production and equipment characteristics for both the geometries.

Table 7 shows the production characteristics of the regular geometry and the two alternate geometries. As hypothesized, alternate geometries have higher production potential at lower costs. This is because smaller edge pillars could potentially change a number of two-cut cross-cut hole-throughs to single-cut hole-throughs while simultaneously decreasing haul distances. Alternate geometry #1 has more number of single cut hole-throughs as compared to alternate geometry #2. It can also be observed that the production cost of coal in alternate geometry #1 is 3.15% cheaper as compared to the regular geometry. This is because the incremental cost of coal at the edge pillars is lower for the alternate geometries as compared to the regular geometry.

Table 7.   Comparison of Productivity Characteristics

	Regular Geometry	Alternate Geometry #1	Alternate Geometry #2
Extraction Ratio (%)	53.37	56.33	56.33
Production (tons/unit shift)	3,434	3,572	3,460
Section Production Cost* ($/ton)	6.34	6.14	6.30

* Refers to cost until the coal from the face area is dumped on section belt. It does not include any outbye costs.

In addition to production and cost analysis of the different geometries, the project team delineated the delays in each cut in each of the modeled geometries. Wait times in a mining system can indicate both loss of productive time as well as overcapacity of the system. In a CM-batch haulage system, overcapacity is indicated by wait on the cars by the miner while loss of production time is indicated by wait of miner on cars (Chugh

et.al., 2005). Frequency distribution of the wait times were plotted for different geometries. The wait times were obtained from SSP model intermediate computations. Positive wait times indicate a wait on car (miner waits on cars) while a negative wait time indicates a wait on miner (car waits on the miner). Figure 17 shows the histogram of wait times of the different geometries as obtained from the SSP model intermediate output. Analysis of the histogram shows that in almost all of the cases the wait times are negative which indicate the batch haulage units are waiting on the miner. Comparing the three histograms, we observe that alternate geometry #1 has more bias towards the zero wait time. This would indicate that alternate geometry #1 is a more matched system as compared to the other geometries.
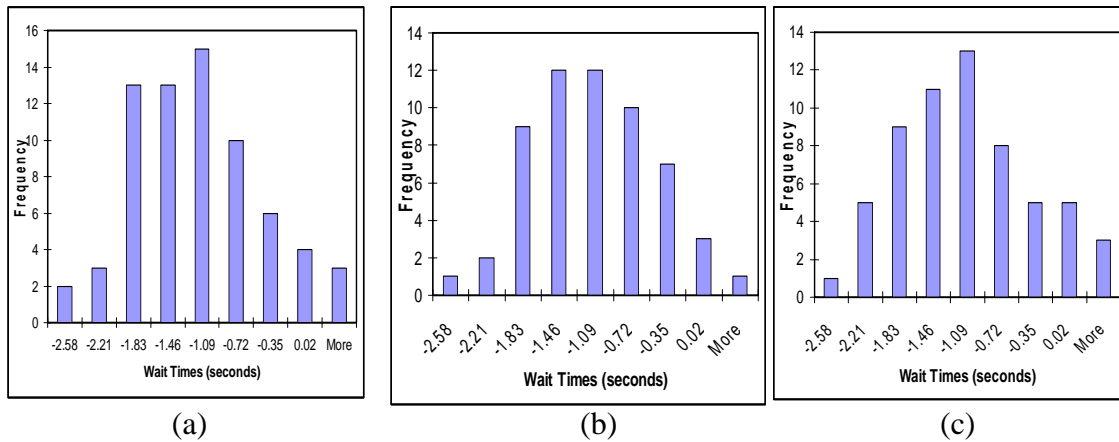
Figure 17.   Histogram of Wait Times for Different Geometries (a) Regular, (b) Alternate#1, and (c) Alternate#2

Task 3: Monitor Alternate Geometry Demonstration Area during Mining

The project team installed two rows of convergence monitoring stations and rib measurement stations in the alternate geometry demonstration area. Figure 18 shows the demonstration area for alternate geometry. Regular monitoring of convergence as well as rib stress in the demonstration area will be done as part of Phase II of the project.
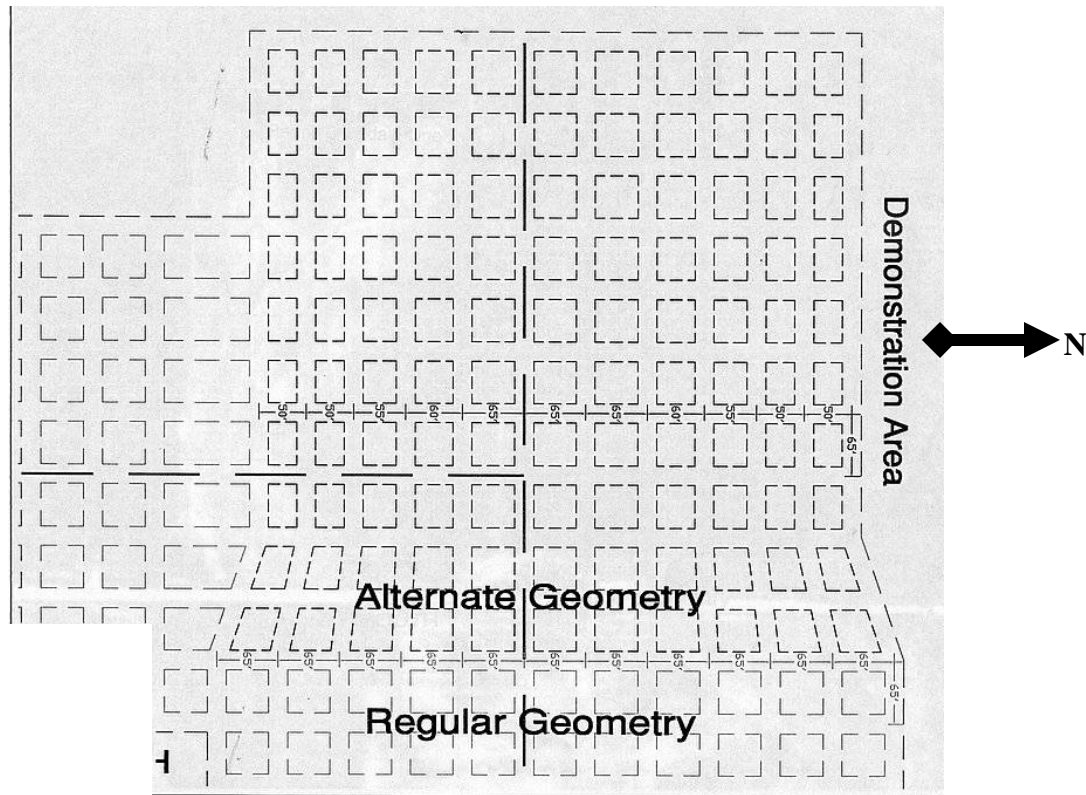
Figure 18.   Demonstration Area

CONCLUSIONS AND RECOMMENDATIONS

Detailed analysis of data collected during this project will be done as part of Phase II. However, conclusions derived from studies to date are presented below:

1. Higher convergence is observed in central entries of the regular geometry.
2. FEA modeling of the developed alternate geometry shows that it has relatively uniform PSF values (~3.5) across the panel. This value is much higher than required to ensure structural stability of coal pillars. The FSF values for the alternate geometry are higher around the center than around the edges as designed. The values are still much higher (2.75 to 3.0) than required.
3. Based on calculated PSF and FSF values for the proposed alternate geometry, additional increases in extraction ratio are possible.
4. Modeling indicates that the production potential of the proposed alternate geometry is higher as compared to the regular geometry due to the ability to accomplish single cut blow-throughs between outside entries reducing the number of miner moves required.

No recommendations are included since data analysis for alternate demonstration has not been completed.

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