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Design Programs for Highwall Mining Operations

Ming Fan

**Thesis submitted
to the Benjamin M. Statler College of Engineering and Mineral Resources
at West Virginia University**

in partial fulfillment of the requirements for the degree of

**Master of Science in
Mining Engineering**

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2015

**Keywords: Highwall Mining, Safety, Health, Suitability, Highwall Failure,
Highwall Stability, Mine Roof Stability Assessment, Pillar Design Methods, Design
Optimization**

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ABSTRACT

Design Programs for Highwall Mining Operations

Ming Fan

Highwall mining is a hybrid of surface mining and underground mining methods, and is often the only feasible method to recover the coal reserves in the central Appalachian coalfields due to the steep terrain and the closely spaced multiple thin coal seams. Compared to the mountain-top-removal, contour, auger, and underground mining methods, application of the highwall mining method can reduce the environmental impacts, increase the recovery ratio of coal reserves, and enhance mine safety as well as productivity. Therefore, it is probable that highwall mining will be a dominant method for extracting the high-value coal resources in the Appalachian coalfields. By far, the greatest ground control safety concerns in highwall mining operations are rock falls from the highwall and mining equipment entrapment underground. These hazards are most likely caused by the instability of the highwall mine system due to insufficient mine design and difficulties encountered during mining operations.

Most of this thesis emphasizes the mine design concepts and methods to maintain the stability of mine structures. With the purpose of evaluating the stability of the entry roof, the beam theory is applied. After comparing the deflection, stress, and strain profiles of 0.05 ft sandstone and mudstone roof layers, it can be concluded that the existence of relatively thin and weak layer in the immediate roof could cause potential stability problems for the entry roof. Therefore, for highwall mining operations to be conducted in coal seams with thinly bedded roof strata, a correct decision to cut some of the thin weak roof rock layers with the main coal seam can be greatly beneficial to mining operations. The pressure arch concept is applied for the systematic design of the highwall mining operation. Within this concept, an optimization design process is developed to improve the recovery ratio of coal resources to a reasonably high level. For multi-seam highwall mining operations, the largest web and barrier pillar sizes should be selected and vertically aligned into seams. Two numerical programs, Examine2D and FLAC, are used to analyze the stability of highwall structures and to find the stable interburden thickness where no interaction between the two coal seams is expected. Numerical results show that, under given geology and mining conditions in the thesis, the stable interburden thickness is 20 ft and there are no stability issues. In the end, three spreadsheet programs are developed for the assessment of highwall mine structures, for the design of the web and barrier pillars, and for the optimization design process, based on the proposed design concepts and methodologies.

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Chapter 1 Introduction

1.1 Background

When the stripping ratio of surface mining operations loses profitability, the highwall mining method can be applied to recover the coal resources beneath the final highwall (Seib, 1993; Vandergrift, et al., 2004; Schafer, 2002). Recent advances in highwall mining systems have enhanced mining safety and increased financial gains, which makes the highwall mining an important coal production method in the United States (Schmidt, 2015). Currently, the ADDCAR system and the Cat HW300 are two principal highwall mining systems. These two systems are extremely popular in the central Appalachian region (Fiscor, 2002). Due to the advanced design and specifications of these highwall mining systems, the highwall mining method now has a healthy share of space in the central Appalachian mining market. Compared to contour mining or other surface mining methods, highwall mining provides many advantages, such as high flexibility, ability to avoid geological structures, and capacity to extract still valuable resources that are not mineable for both surface and underground mining methods. Moreover, highwall mining operations are performed by remote control, allowing workers to work in a safe environment free from hazards such as roof falls, gas, dust, flooding, and vehicle movements. Therefore, highwall mining is the desired and often the only appropriate method to recover the remaining coal resources in the Appalachian coalfields. Furthermore, due to the limitations of other mining methods, and the tighter environmental control requirements, the highwall mining method should be considered by more coal operators.

Highwall mining is a hybrid of surface mining and underground mining methods. With the miner's cutter module pushing into the coal seam through a series of push-beams, long rectangular entries are punched into the coal seam to recover part of the remaining coal resources. Web and barrier pillars are left to separate the entries and prevent the overburden strata from collapsing. As the cutter head moves forward into the coal seam, fully enclosed push-beams are inserted behind the cutter head. Then the coal is moved back to the center of the bench for stockpiling and transport through the stacking conveyor system (Schmidt, 2015). The mining cycle continues for 20 ft and the process is repeated until the operator reaches the desired depth. Once the cutter head has been fully retracted, the drum is inspected and serviced. The machine is then moved to the next entry. Furthermore, with the help of the monitoring system such as the

gamma detection system, predetermined amounts of coal can be cut accurately, and coal resources can be recovered easily in soft roof or soft floor environments.

1.2 Statement of Problem

The Mine Safety and Health Administration's (MSHA) accident and injury statistics show that highwall mining has maintained an admirable safety record. Its fatality and injury rates are nearly identical to the other surface mining methods, and are greatly lower than underground mining methods. By far the greatest ground control safety concerns in highwall mining operations are rock falls from the highwall and equipment entrapment underground. These hazards are most likely caused by the instability of the highwall mine structure system due to insufficient mine design or difficulties encountered during mining operations.

Generally, the factors affecting highwall stability during highwall mining operations can be summarized into two categories, namely geological structure and highwall structure stability (Zipf, 2005). Among the geological constraints, the hillseams are the major geological structures that could impact or limit highwall mining. However, many precautions can be taken to minimize the risk of failure associated with hillseams, such as skipping an entry where a hillseam enters the highwall. Another significant concern related to the safety of highwall mining is the instability of the highwall structure. Even though the design methods for highwall mining operations have been proposed by National Institute of Safety and Health (NIOSH) researchers, there is still considerable room to improve this mining method for the purpose of ensuring the stability of the highwall structures.

It is also critical to rationally design the highwall mining system in multi-seam conditions (Newman, 2009). In order to increase the recovery ratio of coal resources, many highwall mining operators choose to recover multiple seams in very close proximity. However, the seams being mined below or above a previously mined seam tend to be subjected to disturbances from the adjacent coal seams. Therefore, multi-seam highwall mining systems should be designed with the purpose of preventing web pillars from collapsing as well as preventing highwall failure.

1.3 Statement of Work

The objective of this research is to improve the highwall mining method to maintain the stability of mine structures for the purposes of ensuring the safety for both personnel and mining

equipment. In order to systematically design a highwall mine and increase the recovery ratio of coal reserves, three highwall mine design programs are developed.

First, this thesis studies the suitability of highwall mining in Appalachian coalfields and main challenges with highwall mining. With regard to the particular geology and topography in the Appalachian coalfields, some popular mining methods, namely mountain-top-removal, contour, auger, underground, and highwall mining methods, are compared. Then, the main challenges with highwall mining are investigated from the following aspects: types of highwall, MSHA incident statistics for highwall mining, types of highwall failures, and the factors affecting the highwall stability.

Second, in order to develop programs to systematically design a highwall mine, the stability of the highwall top surface, the stability of the highwall, and the stability of the highwall entry roof are studied. In order to assess the stability of the entry roof, the beam theory is applied. In the meantime, the gravity effect, the squeezing effect, and the interaction effect between layers are taken into consideration when evaluating the stability of the entry roof. Then, the pillar stability for highwall mining in single coal seam and the influence of multi-seam mining operations are discussed. Numerical models are constructed with Examine2D and FLAC to illustrate examples of ways to find the minimum thickness of interburden where no interaction between the two coal seams is expected. Additionally, the FLAC model is also performed to check out the stability of the entry roof and pillars, and to detect other potential failure mechanisms. Next, the pressure arch concept is applied in the panel optimization design process.

In the end, three spreadsheet programs have been developed to design highwall mine pillars with the proposed design methodologies and concepts. The first design program can be used to assess the roof stability if the thinly bedded weak layer exists in the immediate roof above the coal seam. The second design program is developed to assess the pillar stability and panel design when a single coal seam is mined based on traditional loading. In order to maximize the recovery ratio, the third design program is developed for the panel optimization design process.

Chapter 2 Literature Review

2.1 Introduction of Highwall Mining Method

2.1.1 Highwall Mining Method

As mining progresses, higher stripping ratios gradually become restrictive factors for surface mining operations. It is inevitable, at some point, for mining operations to reach an economic threshold where the cost of overburden removal surpasses the value of the coal. When stripping ratios lose profitability, recovery of coal resources under the highwall can be accomplished by other means. Generally, where the ordinary open-cut mining is difficult or impossible in a small-scale mine due to an uneconomical stripping ratio, the highwall mining systems could be applied to extract the remaining coal reserves (Seib, 1993; Vandergrift, et al., 2004; Schafer, 2002). Appalachian coal operators are discovering that highwall mining can be a safe and productive method for retrieving coal from active and abandoned highwalls (Fiscor, 2002). It is gradually becoming an important mining method in the US and accounts for 4% of the total U.S. coal production for the moment. Seventy-six highwall mining machines are currently operating in the U.S., and there are also units cutting coal currently in Russia, Colombia, and India. In the U.S., Appalachia is the preferred highwall home. There are approximately 62 active highwall mining machines at work in West Virginia, Virginia, Pennsylvania, Kentucky, Ohio and Maryland. At present, there are also operations in Alabama and Utah (Schmidt, 2015).

In the most general sense, highwall mining is the process of extracting coal reserves from the base of an exposed highwall left by the previous surface mining. Typically, this is done by using the remotely controlled mining methods to horizontally drive a series of nearly parallel entries for a significant distance (Gardner, et al., 2002; Adhikary, et al., 2002). Long rectangular entries are punched into the coal seams through the highwall operations to recover part of the remaining coal reserves. Small pillars are left to separate the entries and to prevent the overburden strata from collapsing. A typical modern highwall mining system is established on the floor of an open-pit or contour bench, lying in front of the exposed coal seam at the base of the highwall. The coal is transported from the back of the highwall miner to the surface that is incrementally extended in length as the miner advances into the coal seam. The maximum distance of the entry is determined by equipment design and geological conditions. Then the equipment is pulled back from the entry, moved along the highwall, and prepared to extract

another entry parallel to the one just mined. A schematic image of a highwall mining operation is shown in Figure 2-1.

Since a highwall mining operation only needs a narrow bench to obtain entrance to the coal seam, highwall mining systems not only offer less environmental disturbances to the surrounding land, but also are able to advance economic benefits substantially. The highwall mining system is extremely mobile and can be moved from pit to pit in a few hours, or from mine to mine in a day or two. Therefore, the efficient highwall mining systems would significantly extend the reserve life of open-cut coal mines. Moreover, since the entries are unmanned, many of the safety issues associated with underground mining are eliminated. In general, the highwall mining method is very productive, flexible and cost-effective when used in proper situations.

Figure 2 - 1 A Schematic Image of a Highwall Mining Operation (Shen, et al., 2001)

2.1.2 Advantages of Highwall Mining

Highwall mining can economically access smaller blocks of coal with high flexibility and is more readily able to avoid geological structures or other impediments to production. The mobility of the highwall mining equipment make it easy to move around a pit, from pit to pit or

from mine to mine. Therefore, its applicability improves a lot contrary to other mining methods. Due to the reason that highwall mining is operated by remote control, highwall mining machines are able to operate in thin seams without having to cut stone to make room for people to move around. Furthermore, it can also selectively extract a high-quality segment of a seam. In this sense, the safety conditions will be greatly advanced. In addition, operations are inherently safer because they are carried out by remote control, and all personnel remain outside the entries. Personnel therefore remain on the surface to avoid hazards such as roof falls, gas, dust, irrespirable atmospheres, flooding, vehicle movements in confined spaces, etc. Without the use of ventilation, highwall mining can continue in gassy seam conditions that would stop or impede underground mining.

2.2 Highwall Mining Process

The highwall mining method is a hybrid of surface and underground coal mining methods that evolved from auger mining. A typical mining cycle includes sumping (forward pushing) and shearing (raising and lowering of the cutter-head to extract the entire height of the coal seam). As the cutting cycle continues, the cutter-head is progressively pushed into the coal seam to a depth of 20 ft. Fully enclosed push-beams are inserted behind the cutter-head as it moves forward into the coal seam. The push-beam transfer mechanism stages each section with a push-beam above the power-head before insertion and moves the cut coal from the enclosed push-beams to the power-head. Then, the coal drops to the conveyor system, where coal is moved to the center of the bench for stockpiling and transport. The mining cycle continues for 20 ft and the process is repeated until the operator reaches the desired depth. After full coal extraction, the push-beams are removed from the back of the cutter-head. As it retracts, push-beams are stacked in an open pit area, minimizing the on-site size requirements. Once the cutter-head has been fully retracted, the drum is inspected and serviced. The machine is then moved to the next entry (Fiscor, 2002). As the string of push-beams is hinged at 20-ft intervals, the machine can be navigated vertically through rolls and undulations within the coal seam. Currently, the highwall mining systems are capable of handling coal seam thicknesses from 2.6 to 16 ft with a dip up to 8°.

Generally, a highwall mining operation requires crew of 6 employees: two foremen, a machine operator, a ground man, a loader operator, and a technician as needed. The operator is located in the control cabin about 20 ft above the ground where the operator has a complete view of all mining activities around the machine. The operator primarily controls sumping and cutting

operations of the continuous miner. In addition, the operator monitors six to nine screens which provide information on methane levels, hydraulic pressure, roof and floor conditions, coal thickness, entry heading and fellow employee activities. The ground man cleans debris from the top of the push-beams or conveyor cars, and ensures that each push-beam is properly locked and connected as mining progresses. The loader operator loads push-beams onto the push-beam transfer mechanism or conveyor cars behind the continuous miner during mining. At no time do any employees work underground (Fiscor, 2002).

2.3 Highwall Mining Systems

Currently, the highwall mining systems market are dominated by two manufacturers. International Coal Group, Inc. (ICG), now becoming a wholly owned subsidiary of Arch Coal, developed the ADDCAR system shown in Figure 2-2. Caterpillar developed the Cat HW300 shown in Figure 2-3, which is a continuation of the Superior Highwall Miner after purchasing Bucyrus in 2011.

2.3.1 ADDCAR Highwall Mining System

The ADDCAR Highwall Mining system is set up in front of a highwall which is a truly continuous system. This is due to the fact that coal production does not need to be interrupted to add additional conveyance cars into the highwall as the miner advances to the entry. The conveyor cars are connected together by using vertical locking pins. In this way, a continuous haulage system among the cutting machine, the launch vehicle, and a stacker conveyor is created (Gardner, et al., 2002).

The central component of the system is the launch vehicle, which is the central part of the highwall mining system, acting as a platform for the essential electrical, hydraulic, and other control functions. First of all, the launch vehicle is aligned perpendicular to the coal seam at the preferred location, which remains stationary during the mining cycle. Then a crawler-mounted continuous miner enters the highwall and begins to cut the coal. The 40-foot-long conveyor cars are added, one at a time, which are placed behind the continuous miner as the system advances under the highwall. These cars collect the coal from the continuous miner and transport it to the outside. The conveyor cars discharge the coal car-to-car without the use of an auger, which will not increase fines. One loading stacker conveyor receives the coal that is discharged from the launch vehicle and loads it into awaiting trucks. When an entry has been completed, the entire

system will be retracted, moved over, and aligned with the next entry. (ADDCAR Highwall Mining Systems, <http://www.addcarsystems.com/>).

Figure 2 - 2 ADDCAR Highwall Miner (<http://www.addcarsystems.com/broad.html>)

Figure 2 - 3 Cat HW300 Miner (<https://mining.cat.com/highwall-miner>)

2.3.2 Cat HW300 Highwall Mining System

The Cat HW300 highwall mining system is a highly efficient mining equipment which offers a safe method for recovering coal from the final highwall. This system is designed with easy maintenance and a comprehensive diagnostic system, and is able to be disassembled in modules, which all contribute to the system's outstanding productivity. The Cat HW300 offers two electric cutter head modules, which are interchangeable and able to be quickly attached to the power-head. The whole cutting cycle is fully automated, however, when the coal seam varies, it still allows the operator to make adjustments through an ampere reading.

The self-propelled Cat HW300 Highwall miner is capable of operating on benches as narrow as 59 ft. This highwall mining system is able to tram from entry to entry and transport coal resources in limited space easily. The machines can also be easily moved from pit to pit or from mine to mine. The enclosed push-beams and an anchoring system are mounted on Cat HW300 to manage roof fall problems during mining production. As roof rock is collected on the push-beams, the weight of the mining string increases. If too much load is added on the push beams, the string will be retracted and the rock will be removed by the coal loader and then reenter the entry to finish the mining cycle. The force applied to the string of push-beams must overcome changes in elevation due to undulations and rolls as well as any roof material that may

fall on the top of the push-beams. Caterpillar highwall miners are equipped with 49 push-beams, allowing mining system to penetrate nearly 1000 ft into the coal seam. (Buchsbaum, 2011; HW300 Brochure, 2013).

2.4 Highwall Mining Guidance System

Since most of the entries fail to reach the designed or predetermined depth, it is very important to maintain the coal pillars and keep parallel openings. However, this can be achieved only if the highwall miners' position and heading could be determined and controlled remotely. Therefore, it is imperative to highlight the guidance problems associated with highwall mining, which tend to decrease the recovery ratio as well as cause hazards to highwall stability. A variety of sensors, keeping the miner in the seam automatically, have been used in highwall mining operations. These sensors include a roof and floor passive gamma detector system, multiple inclinometers, a ring laser gyroscope, and a programmable logic controller (Ralston, 2001). These remote detection systems offer the following two purposes: (1) locating roof and floor rocks to avoid cutting into such rock, and (2) determining the pillar width left between the previous and current entries to maintain the pillar design width.

2.4.1 Guidance System for ADDCAR Highwall Mining System

The ADDCAR system is equipped with advanced guidance features which are able to keep the miner operating in the seam, control pillar width as well as get maximum recovery without risking roof stability. A combination of hydraulic, electrical and electronic technologies is used for lateral guidance, mining a predetermined height, pushing and pulling the system into or out of the hill and for cutting and loading the coal.

For lateral guidance system, Honeywell's Ore Recovery and Tunneling Aid (HORTA) is utilized in ADDCAR system. There are three pairs of ring laser gyroscopes and accelerometers, which are used to monitor the heading, pitch and roll of the continuous miner, shown in Figure 2-4. The operator has access to the complete picture of the active entry, past entries as well as pillar thickness, allowing the operator to navigate the miner and make adjustments. This ability is very important to improve the safety situations of highwall mining operations, since the correct size of web pillars should be kept to carry overburden load.

For vertical guidance system, the thickness of the roof and floor coal is continuously measured by gamma coal thickness sensors. With the help of these sensors, predetermined

thickness of roof or floor coal can be achieved. In order to ensure optimum quality and assist in roof control, the gamma based information is used during operation. The exact cutting height with respect to the established floor is provided with graphical data by the use of boom-mounted inclinometers. A cable reel mounted odometer is used to measure the depth of cut. This provides the highwall miners to achieve high penetrations and reduces out-of-seam dilution (Vandergrift, et al., 2004; Highwall roundtable- Highwall systems, 2006).

2.4.2 Guidance System for Cat HW300 Highwall Mining System

The operation of the Cat HW300 highwall mining system is controlled through the use of PLC (programmable logic control) technology at the heart of its machine control system. By the use of touch screen technology, the operator in the cab can not only have immediate-response control over the machine's functions, but also adjust these functions in case of the unexpected mining conditions within the entry. A comprehensive diagnostics system is constructed with the function of troubleshooting assistance and streamlines maintenance procedures. Therefore, it is not difficult to see that the mining operation can be optimized, resulting in cleaner coal products.

Figure 2 - 4 Lateral Guidance System of ADDCAR
(<http://www.addcarsystems.com/techleader.html>)

Gamma Detection systems are used to provide the machine operator with an additional knowledge to achieve the optimum cutting range. This technology is capable of steering the cutter module through the coal seam, recovering predetermined amounts of coal in the coal seam. This system also allows the extraction of coal reserves in soft roof or soft floor conditions. In the past,

it was too easy to overcut, resulting in unwanted situations which tend to cause problems for roof control and even worse cause hazards to mining machines. In order to achieve more accurate direction, Caterpillar provides a navigation and steering system. (HW300 Brochure, 2013).

2.5 Multiple Seam Highwall Mining

Many highwall mining operations recover multiple seams in very close proximity with the purpose of increasing the recovery ratio. In the eastern U.S., this situation appears frequently when a thick seam splitting into thinner seams. In the western U.S., certain very thick seams can exceed the working height of the highwall miner, and a multiple seam mining approach may be utilized (Zipf, 2005). When it comes to multi-seam highwall mining, seams being mined below or above a previously mined seam tend to be subjected to disturbances from the adjacent coal seams, especially in close proximity. The mining induced strata movements can cause serious roof and pillar stability problems, such as high stress concentration and heavy fracturing in the adjacent seam. Therefore, it is imperative to prevent web pillars from collapsing and the high possibility of highwall failure from the standpoint of ground control safety in multiple seam highwall mining situations.

The main extraction sequence for highwall mining can be classified into two categories based on the spatial relationship between workings on two adjacent seams: (1) under-mining where the upper seam is mined out prior to the mining of the lower seam; (2) over-mining where the mining of the upper seam is not commenced until the extraction in the lower seam completely finished. The multiple seam highwall mining sequence should be carefully selected to avoid operation concerns. For under-mining, it may lead to future web pillars collapse and subsidence in the upper seam, leaving unstable highwall above the active pit. For over-mining, it is easier to realize the continuity of surface mining operations and maximize the recovery ratio. This is due to the fact that over-mining makes it easier to backfill the abandoned entries from a combination of back stacking and push spoil into lower entries as the upper seam is developing. However, if the thickness of interburden is very small, it may lead to mining equipment entrapment. Moreover, because of the relatively thin web pillars, accurate surveying is required to ensure that the adjacent highwall entries do not intersect each other. For over-mining, it is required that the survey points be located before the operation of the highwall miner in the upper seam since the openings are partially backfilled. However, it is much easier to stack at the beginning of an entry if under-mining sequence is selected. This is because the upper seam is visible and positioning the highwall miner directly below is no problem.

For multiple seam mining operations, it is imperative to carefully evaluate the stability problem of highwall structures when making designs for highwall mining (Newman, 2009). Because the stability problem in multiple seams is mainly dependent on the transfer of stress through the upper and lower web pillars, properly stacking web and barrier pillars is one of the very important solutions to prevent highwall failure from occurring. If a highwall miner cut is not aligned properly with the overlying or underlying cut, the overburden stress is transferred onto the interburden. Under this situation, failure is likely to occur as web pillars punch into the interburden, which may cause great danger to highwall mining operations. Due to web pillars being very narrow and extremely long, it is very difficult to columnize web pillars between the upper and lower seams. However, it is much easier to maintain barrier pillars columnized because of their greater width. Moreover, barrier pillars are capable of supporting the load transferred from web pillars, thereby stiffening the highwall system, and thus preventing the catastrophic collapse. Therefore, it is important to design barrier pillars with a proper size because they must have adequate strength to withstand additional stresses imposed on them if all web pillars failed within a panel. In addition, in order to maintain the stacking deep within the entries, an on-board guidance system should be used. Besides properly stacked web and barrier pillars, limiting the number of entries will also lessen the possibility of highwall failure in close proximity multiple seam highwall mining situations.

2.6 Empirical Highwall Mining System Design

Highwall mining should follow a detailed plan to control the hazards which may happen during highwall mining operations. When designing a highwall mining layout, the mining designer must determine 1) web pillar width, 2) the number of web pillars between barrier pillars and 3) the barrier pillar width. The design parameters are determined by the highwall miner entry width, the mining height and the overburden depth (Zipf, 2005). In addition, the pillar strength, the applied stress on pillars, and the pillar stability factor should be estimated by the mine designer.

2.6.1 Coal Pillar Strength

The Mark-Bieniawski formula applies best for web pillars, which are very long, narrow rectangular pillars. For long pillars whose strength length is much greater than their width, the Mark-Bieniawski formula can be reduced to the Equation 2-1. In-situ coal strength is normally selected as 900 psi. Mining height depends on the seam thickness.

$$S_p = S_i \cdot \left[0.64 + \frac{0.54W}{H} \right] \quad (2 - 1)$$

Where: S_p —web or barrier pillar strength

W — web or barrier pillar width

S_i —in-situ coal strength

H —mining height

2.6.2 Coal Pillar Stress

For the web pillar system, the tributary area method is used for calculating the coal seam vertical stress on web and barrier pillars. Average vertical stress on a web pillar is shown in Equation 2-2. The highwall mining equipment dictates the entry width which is either 9.5 or 11.5 ft. In-situ vertical stress is determined by two factors, namely overburden depth and the overlying rock density. Vertical stress gradient is typically 1.1psi/ft. Overburden depth may be selected as the maximum overburden depth on a highwall mining web pillar, which is very conservative and can be computed using a high average value shown in Equation 2-3. Finally, the stability factor for a web pillar is shown in Equation 2-4. For design purposes, the stability factor for a web pillar typically ranges from 1.3 to 1.6.

$$\sigma_{wp} = \sigma_i \cdot \frac{W_{wp} + W_e}{W_{wp}} \quad (2 - 2)$$

$$D_{Design} = 0.75 \times D_{max} + 0.25 \times D_{min} \quad (2 - 3)$$

$$SF_{wp} = \frac{S_{wp}}{\sigma_{wp}} \quad (2 - 4)$$

Where: W_e —highwall miner entry width

W_{wp} —web pillar width

σ_i — in-situ vertical stress

D_{max} — maximum overburden depth

D_{min} —minimum overburden depth

For the barrier pillar system, when the number of web pillars in a panel is N , the panel width can be computed by Equation 2-5. The average vertical stress on a barrier pillar can be determined by Equation 2-6. Similarly, the stability factor for a barrier pillar is shown in Equation 2-7. Since the stress undertaken by web pillars is neglected, the stability factor for a barrier pillar can be as low as 1.

$$W_p = N(W_{wp} + W_e) + W_e \quad (2 - 5)$$

$$\sigma_{bp} = \sigma_i \cdot \frac{W_p + W_{bp}}{W_{bp}} \quad (2 - 6)$$

$$SF_{bp} = \frac{S_{bp}}{\sigma_{bp}} \quad (2 - 7)$$

Where: W_p — panel width

W_{bp} —barrier pillar width

2.7 Introduction of Examine2D and FLAC

2.7.1 Methods of Numerical Modeling

Numerical modeling could be used to design and evaluate mine structures and support systems. The advantage of numerical modeling is that model construction is much easier and quicker than the physical modeling, and the design parameters can be changed easily to evaluate the effect of parameters' change on the overall design. The most commonly applied numerical methods for rock mechanics problems are (Jing, 2002):

- (1) *Continuum methods* including: the finite difference method (FDM), the finite element method (FEM), and the boundary element method (BEM)
- (2) *Discrete methods* including: the discrete element method (DEM) and the discrete fracture network method (DFN)

For finite element method (FEM), the basic principal is that a solution region can be analytically or approximately modeled by replacing it with an assemblage of discrete elements. Because these elements can be permuted in a variety of ways, they can be used to represent exceedingly complex shapes. For finite difference method (FDM), numerical techniques are used

to approximate the solutions of differential equations by replacing the partial derivatives with finite difference approximations defined at neighboring grid points. The advantage for FEM and FDM is that these methods allow material to deform and fail, and thus these methods are capable of modeling complex behavior. And simple structures can be simulated with interfaces, but not suitable for highly jointed-blocky media. Also, there are some limitations should be noticed including effects of mesh size, boundaries, symmetry restrictions and data input limitations (such as effects of variation of critical input parameters). In the boundary element method (BEM), the governing differential equations are transformed into integral variables, which can be used over the boundary surface of the region. The advantages are able to do elastic and rapid assessment of designs and stress concentrations (Peng, 2008).

In the discrete element methods, the blocks are in mutual contact and the contact is represented by springs in both normal and tangential directions. Due to the fact that natural characteristics of rock mass are comprised of blocks bounded by joints, there is a large applicability in the field of geomechanics. However, the drawback is that it is difficult to obtain reliable data on location, orientation and persistence of the discontinuities. For discrete fracture network (DFN), it is most suitable for the study of fluid flow and mass transport in fractured rocks for which an equivalent continuum model is difficult to model or establish (Jing, 2003).

2.7.2 Introduction of Examine2D

Examine 2D is designed to be a quick and simple to use parametric analysis tool for investigating influence of geometry and in-situ stress variability on the stress changes in rock due to excavations. The induced stresses can be analyzed by means of stress contour patterns in the region surrounding the excavations. As a tool for interpreting the amount of deviating overstress (principal stress difference) around openings, strength factor contours give a quantitative measure of strength over induced stress according to a user defined failure criterion for the rock mass (Rocscience, 2012).

Some important limitations of the program should be considered when interpreting Examine2D output. First of all, the program is based on the assumption of plane strain, which means that the modeled excavation is of infinite length normal to the plane section of the analysis. In practice, the out-of-plane excavation length should be larger than five times the largest cross-sectional dimension. Second, the material being modeled in the Examine2D is assumed to be homogenous, isotropic or transversely isotropic, and linearly elastic. Third, the

displacements shown by Examine2D are meant to qualitatively illustrate regional deformation trends only. In general, the induced stresses calculated and displaced by Examine2D can usually prove useful and may still yield useful insight in the analysis.

2.7.3 Introduction of FLAC

FLAC is a two dimensional finite difference program for engineering mechanics computation, which is capable of simulating the behavior of structures built of soil, rock or other materials. In FLAC, materials are represented by elements or zones, which is able to form a grid to fit the shape of the object to be modeled. With the purpose of responding to the applied forces or boundary restrains, these elements would behave by following the prescribed linear or nonlinear stress/strain law. FLAC also consists of built-in programming language FISH. Because of FISH, the user can write functions to extend FLAC's usefulness. At the same time, FLAC can be operated as either a menu-driven or a command-driven computer program (Itasca, 2011).

With the purpose of setting up a model with FLAC, three fundamental components of a problem must be specified: (1) a finite difference grid; (2) constitutive behavior and material properties; and (3) boundary and initial conditions. Among them, the grid defines the geometry of the problem, the constitutive behavior and associated material properties dictate the type of response the model will display, and boundary and initial conditions define the in-situ state. In FLAC, the time-marching method is adopted to solve the algebraic equations and the solutions are reached after a series of computation steps. In the meantime, the number of steps required to reach a solution can be controlled by code or manually by the user. The generation solution procedure is illustrated in Figure 2-5, which represents the sequences of processes that occurs in the physical environment.

In FLAC, there are twelve built-in constitutive material models: null model; elastic, isotropic model; elastic, orthotropic model; elastic, transversely isotropic model; Drucker-Prager model; Mohr-Coulomb model; ubiquitous-joint model; strain-hardening/softening model; bilinear strain-hardening/softening ubiquitous-joint model; double-yield model; modified Cam-clay model and Hoek-Brown model. In this thesis, the Mohr-Coulomb material is used to simulate the real behavior of the highwall structure.

Figure 2 - 5 General FLAC Solution Procedure (Itasca, 2011)

2.7.4 Mohr-Coulomb Model

Mohr-Coulomb Model is used in this model construction to simulate the real behavior of highwall structures. The reason to choose this model is that the Mohr-Coulomb failure criterion is frequently used to assess the state of failure in the study of soil and rock mechanics. However, the drawback is that the effect of intermediate principal stress will not be taken into consideration. In FLAC, the material properties that must be defined for a Mohr-Coulomb material include: density, bulk modulus, shear modulus, friction angle, cohesion, dilation angle, and tensile strength.

The failure envelop for this model corresponds to a Mohr-Coulomb criterion (shear yield function) with tension cutoff (tension yield function). The tensile flow rule is associated and the shear flow rule is non-associated. In the FLAC model, principal stresses $\sigma_1, \sigma_2, \sigma_3$ are used, the out-of-plane stress, σ_{zz} , being recognized as one of these. The principal stresses and principal directions are evaluated from the stresses' tensor components and ordered so that $\sigma_1 \ll \sigma_2 \ll \sigma_3$. The corresponding principal strain increments $\Delta e_1, \Delta e_2, \Delta e_3$ are decomposed as follows:

$$\Delta e_i = \Delta e_i^e + \Delta e_i^p \quad i = 1, 3 \quad (2 - 8)$$

Where the superscripts e and p refer to elastic and plastic parts, respectively, and the plastic components are nonzero only during plastic flow.

With the ordering convention of equation 2-8, the failure criterion may be signified in the plane (σ_1, σ_3) as illustrated in Figure 2-6.

The failure envelop is defined from point A to point B by the Mohr-Coulomb yield function 2-9:

$$f^s = \sigma_1 - \sigma_3 N_\phi + 2c\sqrt{N_\phi} \quad (2 - 9)$$

From B to C by a tension yield function 2-10:

$$f^t = \sigma^t - \sigma_3 \quad (2 - 10)$$

Where ϕ is the friction angle, c , the cohesion, σ^t , the tensile strength is N_ϕ .

$$N_\phi = \frac{1 + \sin\phi}{1 - \sin\phi} \quad (2 - 11)$$

Figure 2 - 6 Mohr-Coulomb Failure Criterion in FLAC (Itasca, 2011)

In FLAC, only the major and minor principal stresses are effective in the shear yield formulation, which means the intermediate principal stress has no effect. For a material with friction, $\phi \neq 0$ and the tensile strength of the material cannot surpass the value σ_{max}^t given by equation 2-12.

$$\sigma_{max}^t = \frac{c}{\tan \phi} \quad (2 - 12)$$

Chapter 3 Application of the Highwall Mining Method

3.1 Suitability of Highwall Mining in West Virginia

There are several mining methods that are popularly used in recovering coal seams in West Virginia, such as underground mining, mountain-top-removal mining, contour mining, auger mining, and highwall mining. Many factors should be taken into consideration when selecting the most suitable mining method from above list. The most significant factors should be taken into account are geology and topography. Most of the high-quality and high-value coal in the state of West Virginia is produced in the mountainous areas of the central Appalachian coalfields. In most cases, the multiple coal seams are formed in close proximity as shown in Figure 3-1. From this figure, it is easy to find that some of the coal seams recovered by the mountain-top-removal method would often result in too much damage to the surface plants, disturbance of streams, and movement of overburden rock and surface soil.

Figure 3 - 1 Geological Column at a Southern WV Mine Site (Kitts, 2009)

3.1.1 Requirements for Underground Mining

If coal seams are thick enough (e.g., >3ft in height) that could be recovered by underground mining methods, the suitability of adopting underground mining methods is restricted by coal seams' width between the outcrops on both mountain sides as shown in Figure 3-2. A minimum width is required to accommodate two outcrop barrier pillars on the mountain sides, and a 5-entry mains system is placed between the outcrop barriers (Kitts, 2009).

Based on underground mining regulations, it is required that overburden depth should be larger than 100 ft within the inner edge of an outcrop barrier. Therefore, Equation 3-1 can be applied to determine the minimum coal seam width. However, in order to ensure a profitable and efficient mining operation, the coal seam width should be much larger than the minimum width derived from Equation 3-1 (Luo, 2014).

$$W_{min} = 2 \times \frac{100}{\tan \alpha} + NW_e + (N - 1)W_p \quad (3 - 1)$$

Where: W_{min} – minimum width of the coal seam between the outcrops

α —angle of the mountain slope

N—number of entries required in the mains

W_e —mine entry width, typically 20 ft

W_p —rib-to-rib width of the pillars in the mains system

For example, in order to construct an underground coal mine under a ridge top with a typical surface slope of 30°, 30 ft pillar width, and 20 ft entry width, the minimum coal seam width should be about 570 ft. Only a small portion of the coal reserves in the example in the Figure 3-2 can be recovered using an underground mining method.

Figure 3 - 2 Typical Appalachian Ridge Top Coal Reserve (Kitts, 2009)

3.1.2 Limitations of Contour Mining

The contour mining method can only recover a restricted amount of the coal reserves from the outcrop location. With the purpose of extracting more coal from a coal seam, it is required to increase the width of the contour bench. However, the amount of overburden to be removed for creating the coal bench and the increase of production cost are in proportion to the square of the bench width. Therefore, the market price of the coal and economics of production cost would determine the bench width. The maximum bench width can be yielded by carrying out an economic zero profit analysis between the value of the coal mined and the cost to remove the overburden. The dollar per cubic yard is usually used to present the unit cost to remove the overburden strata. The total volume of overburden to be removed for exposing the contour bench width of W_b and one-foot development along the longitudinal direction of the contour bench could be derived from the equation shown in Equation 3-2 (Luo, 2014).

Figure 3 - 3 Contour Mine Bench Design

$$V = \frac{1}{54} W_b^2 \frac{\sin \alpha \times \cos(\beta - 90)}{\sin(180 - \alpha - \beta)} yd^3 \quad (3 - 2)$$

Where: V–volume of overburden rock to be removed, yd³

W_b – bench width, ft

α –slope angle, degrees

β –highwall angle, degrees

The maximum bench width based on an economic break-even analysis for a contour mining operation can be determined by:

$$W_{b,max} = 0.027 \times \frac{C_{coal}}{C_{OR}} \times \frac{m \cdot \gamma \cdot \eta \cdot \sin(180 - \alpha - \beta)}{\sin \alpha \cdot \cos(\beta - 90)} ft \quad (3 - 3)$$

Where: $W_{b,max}$ –the maximum bench width, ft

C_{coal} –coal price, \$/ton

C_{OR} –unit cost of overburden removal, \$/yd³

m –coal thickness, ft

γ –coal density, lbs/ft³

η —coal wash recovery rate

For example, a contour mining operation has the following design and operating parameters:

Surface slope: 30°

Highwall angle: 95°

Coal thickness: 36 inches

Coal wash recovery rate: 95%

Coal density: $85\text{lbs}/\text{ft}^3$

Overburden removal cost: $\$3/\text{yd}^3$

Coal price: $\$40/\text{ton}$

Using Equation 3-3, the maximum bench width derived from the break-even analysis should be smaller than 143.4ft. Therefore, a significant volume of coal reserves could be left unmined if only the contour mining method is applied to recover those coal seams presented in the Figure 3-2 case.

3.1.3 Auger Mining

Prior to the development of the highwall mining system, the auger mining method is widely used to recover the coal resources left by contour mining. As the cutter heads cut through the coal seam, auger drills and flights are inserted behind the cutter head. The cutter head on the auger excavates a number of entries into the seam, which operates very similarly to how a wood drill produces wood shavings. The coal is then recovered and moved back to the surface through the spiral action of flights. However, as the depth of the bored entry is extended, the drilling power is diminishing and the coal production is decreased. When the auger reaches the maximum torque, the maximum length of a highwall entry is determined. There are also some other disadvantages associated with augers that restricted the use of auger mining. Due to the fixed cutting height of auger mining, it tends to reduce the recovery ratio of the coal resources significantly, which only recovers 30% to 40% of the coal within that coal seam to that depth. It also has no ability to negotiate dips and rolls in the coal seam because of the rigid structure of the auger flights. (http://www.ritchiewiki.com/wiki/index.php/Auger_Mining#ixzz3RM9tPgum)

3.1.4 Suitability of Highwall Mining

Because of the special geological and topographical structures in the Appalachian coalfields, especially in central Appalachia region, all four of the popular coal mining methods (i.e., mountain-top-removal, underground, auger and contour) have their restrictions to recover some of the high-quality and high-value coal seams in this area. However, as for highwall mining, many of the unsatisfactory aspects of other coal mining methods have been improved. With its capacity to penetrate into the coal seams for a great distance with almost constant power, it can extract a large amount of the coal reserves. With a highwall mining system equipped with adjustable continuous miner cutting head, it can recover variable mining heights which increases the recovery of these coal reserves to a great extent. Simultaneously, no coal size is necessarily degraded with the increasing mining depth. Since there is no rigid structure of the push beams, the dips and rolls can be negotiated in the coal seam, which is capable of advancing the production rate greatly. Although highwall mining is a hybrid of surface and underground mining technologies, it is much safer compared to the underground mining method because no in-seam support or transport systems are required, and ventilation measures are negligible. In fact, the highwall mining method combined with contour mining would become the most preferred mining method for recovering many of the coal seams exposed on the mountainsides in the Appalachian coalfields.

3.2 Main Challenges with Highwall Mining

3.2.1 Types of Highwall

According to Gardner and Wu (2002), there are three types of highwalls in highwall mining, namely unreclaimed highwalls, surface mining highwalls, and highwall mining highwalls. Unreclaimed highwalls are abandoned highwalls from previous contour mining, including auger mining. In this case, the highwalls are most likely not appropriately tilted and the highwalls are not stable. Surface mining highwalls are operated on the final bench of surface mining operation. Highwall mining highwalls are the most favorable ones due to the fact that they are usually well located and designed to enhance stability and productivity. Among the three types of highwalls, unreclaimed highwalls demand the most attention. This type of highwall may have been degraded due to weathering, ground water flow, and failure of web pillars. Therefore, mining operators should examine this type of highwall very carefully and take methods to avoid any potential failure of the highwall before putting them into use.

3.2.2 MSHA Incident Statistics for Highwall Mining

Highwall mining is essentially a hybrid of surface and underground mining technologies. Although all highwall mining workers stay in the surface, the highwall mining production is still operated in the underground environment. Therefore, highwall mining is capable of resulting in the stress redistribution in the overburden strata, which poses potential hazards to mine workers as well as highwall mining machines.

Based on the distribution of fatalities classified by the Mine Safety and Health Administration (MSHA), ground control, particularly the highwall stability, is the principal source of highwall accidents. Powered haulage and machinery accidents are other prominent sources (Zipf, 2005). At most times, slope stability accidents and highwall failures not only pose serious risks to coal mine workers, but also result in at least a week off from work to recover mining machines and resume highwall operations. In most cases, such accidents occurred because mine management failed to recognize a geologic abnormality and failed to adopt rational mining design to ensure highwall stability. In order to prevent the accidents happening, MSHA has proposed some best practices in the following (MSHA, 2015):

- (a) Train all highwall workers to recognize the highwall hazards.
- (b) Highwall needs to be inspected before, during, and after every rain, freeze, or thaw. Examine the face of the highwall, benches, and areas with cracks and loose rocks. Sloughing over hangs, ground, and large rocks that could cause potential safety hazards to highwall mining operations.
- (c) Pay attention to loose highwall material and never work under them. Cut down loose precarious material on a safe position. When dangerous situations fail to be modified, barricade and post signs to stop entering the working area. Be cautious to the loose highwall material especially when working close to the corners of highwalls. Points and outside corners of highwalls are naturally unstable due to weathering and erosion and toppling failures are likely to occur under this geological situation.
- (d) Specify adequate size of the highwall benches in the highwall mine design.
- (e) Clear all the gatherings of fallen rock from benches before conducting operations in the protected area.

(f) Convey dangerous situations to other workers and equipment operators. Notify them of hazardous highwall situations with the help of radios or cell phones.

3.2.3 Types of Highwall Failures

Failure of highwall and highwall falls have been the most serious safety hazards in highwall mining operations. In general, a highwall failure is the unintended loss of material from a highwall. There are many factors that contribute to highwall instability, including rock mass properties, highwall geometry, face orientation, precipitation, groundwater, freeze, equipment vibrations, blasting, and so on. On the whole, there are two types of highwall failures, namely rock mass failures and rock falls.

3.2.3.1 Rock Mass Failure

Rock mass failures generally involve a relatively large amount of material on a large portion of a highwall and it is important to make sure the material or structures are controlled. There are four types of rock mass failure, namely planar, wedge, toppling, and circular. Toppling is the most common failure mode, circular and wedge follow, and the possibility for the planar failure to occur is the lowest (MSHA, 2014).

3.2.3.1.1 Planar Failures

The planar failure mode refers to a situation in which the sliding movement occurs along a single discontinuity surface, shown in Figure 3-4. Planar failures require an adverse potential failure plane striking subparallel to the face and release surfaces at the top and both ends (Bullock et al., 1993). In general, a tension crack would appear on the upper slope surface and the failed block would detach from the slope and slide down along the plane of failure when planar failure occurred. When heavy rains occur, water tends to flow into the tension crack and the friction resistance is reduced to a great extent, which also accounts for the main factor causing planar failure. Therefore, the location and severity of the tension crack deserves more attention in order to prevent this type of failure.

Figure 3 - 4 Planar Type of Highwall Failure (MSHA, 2014)

3.2.3.1.2 Wedge Failure

The wedge failure occurs when sliding movement along two involved discontinuity surfaces that intersect at an angle forming a wedge shaped block in the highwall face, shown in Figure 3-5. Typically, a bedding plane forms on the upper surface of the wedge and sliding occurs along the intersection or on one of the two discontinuity surfaces. There are many factors that can trigger the wedge type of highwall failure, such as mining activities or water flow, which are capable of reducing the friction resistance of the potential failure planes.

3.2.3.1.3 Toppling Failures

The toppling failure mode refers to a situation in which bulking or rotational movement occurs around the base of a slab or column, shown in Figure 3-6. In a highwall, due to the fact that the stress component normal to the highwall face does not exist, the highwall tends to expand toward the free face and splits off parallel or subparallel to the highwall face.

Figure 3 - 5 Wedge Type of Highwall Failure (MSHA, 2014)

Figure 3 - 6 Toppling Type of Highwall Failure (MSHA, 2014)

3.2.3.1.4 Circular Failures

In general, rotational and sling movement along a failure surface are involved in circular failures, shown in Figure 3-7. This type of failure usually occurs along numerous discontinuities and the shape of this type of failure is often like the arc of a circle. Typically, circular failures are the least favorable type of failures, which may extend from the crest to the toe of the slope.

Figure 3 - 7 Circular Type of Highwall Failure (MSHA, 2014)

3.2.3.2 Rock Falls

Rock falls are a type of failure where intact blocks of rock on the fragmented highwall fall down because they are unconfined. There are several critical factors in evaluating the rock fall hazards, namely exposure, block weight, drop height, and highwall geometry. Among these factors, block weight and drop height play an important role in determining the extent of damage of a falling rock, and geometry of the highwall will affect how a rock falls and where it lands (MSHA, 2014).

3.2.3.3 Highwall Failure Examples

One highwall mining accident example occurred in Martin County, Kentucky on May 24th, 2000 (MSHA, 2000). One front-end loader operator was fatally injured by the collapse of the highwall (Figure 3-8) when he was moving coal from the stockpile in the highwall miner. An

unexpected fall of a larger vertical plate of overburden strata was involved in this accident, roughly 12 ft in depth and up to 264 ft in length. The fall of the plate was generated by the extensive failure of the mine pillars under the highwall. Apparently, the primary reason of this accident was the inadequate loading capacity of the highwall pillars for the extreme load of detached rock plate.

Figure 3 - 8 Photo and Site Map of a Highwall Collapse in a Highwall Mining Operation (MSHA, 2000)

Figure 3-9 shows that one highwall fall accident occurred leading to serious damage to the highwall mining machines. Again, the fall occurred along the nearly right angle rock joint interface. The kind of failure is most likely generated because of insufficient loading capability of the web and barrier pillars at the mouth section of the highwall entries.

Figure 3 - 9 Highwall Fall along Joint Interface in a Highwall Mining Operation

Figure 3-10 shows the highwall mining machines are damaged by the falling rocks from the top soil and weathered rock zone. It is quite easy to find that the highwall structures under the failed top zone has become instable first.

Figure 3 - 10 Highwall Failure Cases in Highwall Mining Operations

3.2.4 Factors Affecting Highwall Stability

Overall, highwall mining seems to be a very safe mining method, based on the analysis of MSHA accident and injury statistics. Since highwall mining incorporates elements of both surface as well as underground mining, it can avoid many of the safety and stability concerns which surface or underground mining may encounter during operation. However, some unique safety concerns associated with highwall mining need to be addressed. Generally, the factors affecting highwall stability during highwall mining operations can be summarized into two categories, namely geologic structure and highwall structure stability (Zipf, 2005).

3.2.4.1 Geologic Structure

The principal geologic structures affecting highwall stability in the Appalachians coalfields are hillseams. Hillseams are almost perpendicular to the bedding planes and provide ground water channels to permeate downward from the surface, and correspondingly the weathering process is accelerated in the fracture walls (Sames, G. P. & Moebs, N. N., 1989). Hillseams commonly contain a joint with right angles or closely spaced joints that are weathered, as indicated by mud or softening of the neighboring rock. They extend several hundred feet down from the surface and their orientation is nearly parallel to the hillside on the whole. In general, a secondary set of vertical fractures to the main fracture may exist with these hillseams. Hillseams may give rise to long rectangular slabs or vertical wedges to separate from the highwall. There may exist a highwall stability safety hazard leading to large rock falls from the highwall when rock slabs that form along the hillseams detach and fall away from the highwall face. For the sake of avoiding large rock falling from highwall, many highwall mining operators would choose to skip an entry where a hillseam enters the highwall. A highwall containing hillseams example is shown in Figure 3-11 (Zipf, 2005).

Unfortunately, it is out of the question to control the locations of hillseams and detecting their presence within a highwall is not reliable. However, a number of measures can be taken to reduce the potential failure associated with hillseams. First, those areas of the highwall where a prominent hillseam enters highwall can be skipped when planning. In this way, the layout of highwall mining panels is adjusted with the purpose of locating barrier pillars away from the unstable areas. Second, it is essential to inspect and monitor the benches above the active highwall mining area every day. Third, it is practical to reduce the slope angle of highwall from 90° to 70°- 80°, which is favorable in reducing the hillseam hazards. Some other methods such as performing a good blasting practice are also very helpful in decreasing the damage to the highwall structures.

The discussions above make clear the importance of performing geologic mapping to identify geologic structures that may be naturally occurring or induced by blasting or stress-relaxation during highwall mining operations. The orientation and extent of these geologic defects should be mapped in advance. Attention should be paid to identifying hazards, which can give the operator some suggestions to the expected conditions and avoid some potential hazards such as equipment entrapment. Since groundwater pressure has the function of destabilizing a highwall and accelerating weathering, seepage should be effectively controlled. It is quite

important to control the water rush into the highwall face on rainy days in highwall mining. Moreover, some minerals that are often associated with coal such as shale, siltstone, and mudstone show an excessive slaking behavior when they come into contact with water, leading to a severe deterioration of their properties (Matsui, et al., 2004). Therefore, particular attention should be given to mine water problems that have a great chance of interfering with extraction work and decreasing the stability of highwall opening.

Figure 3 - 11 Hillseams Indicated by Arrows in Contour Mine Highwall. (Note That Weathering along Hillseam Can Extend Several Hundred Feet or More below the Surface Zipf, 2009)

3.2.4.2 Highwall Structure Stability

Another significant concern related to highwall mining is the stability of highwall structure. Because no artificial roof support is installed during the entire highwall mining operation (unless excavated entries are backfilled), the ground must be designed to be self-supporting. Large roof falls in a highwall entry could generate substantial stability problems to a highwall mining operation. It usually takes a long time to recover the entrapped equipment and

then as a result, the mining operation has to be interrupted. Pillars are left between the highwall entries to support the overburden during and after highwall mining. If pillars collapse during the highwall mining operation, it is quite likely for the overburden to cave into the entries, resulting in equipment entrapment as well as significant loss of mineable resources. Even worse, if one pillar fails the neighboring pillars are immediately overloaded, causing them to fail and so forth until all pillars in the layout area have failed. This domino-type failure posts a great hazard to all highwall miners (Zipf, 1999). Since a highwall mining system is extremely expensive, any kinds of failure of highwall structures that could make the highwall mining equipment entrapped would lead to a significant economic loss to the mining company. In addition, it is quite dangerous to recover the entrapped highwall mining equipment underground. Therefore, design and operational efforts should emphasize maintaining the stability of the highwall structures for the sake of a successful highwall mining operation (Shen, 2001).

Figure 3 - 12 Highwall Collapse in Multiple Seam Mining Area (Zipf, 2009)

It is fairly common to conduct highwall mining operations in multiple seams, especially in the central Appalachian coalfields. However, serious ground control problems could affect highwall mining operations when encountering multiple coal seams in very close proximity, especially in the situation that the thickness of interburden is less than the width of the highwall entry (Zipf, 2009). Figure 3-12 shows one example where highwall mining was conducted in two close coal seams. The upper seam was 3ft thick and the lower seam was 3 ft thick. The two seams

were separated by a weak, laminated interburden with a thickness from 4 to 10 ft. The extensive highwall failure has occurred due to the domino failure of the web pillars.

Therefore, proper layout of these web and barrier pillars has a significant effect on coal resources recovery and is very essential for highwall stability as well. The highwall structures could be destabilized by the failure of web pillars and the following subsidence of overburden strata. Collapse of pillars can also lead to entrapment of highwall mining machines. It is quite dangerous to recover the entrapped highwall mining equipment. Because of these reasons, it can be established that pillar design is the key factor to highwall mining layout. An over-optimal design may pose too much risk for mining personnel and equipment, while over preservative design causes unnecessary loss of resources.

Chapter 4 Design Methods for Highwall Mining Operations

Although highwall miners have been widely used to recover the coal resources from previous mining operations, the methods that can be used to systematically design a highwall mine and satisfy the requirement of safety and operational challenges are still evolving. Highwall mine structure design methods will be introduced in this section that are critical for the safety of miners and mining operations. The design methods and the stability analysis methods for highwall mining operations are applicable to both single coal seam and closely spaced multiple coal seams, respectively. With the purpose of achieving high recovery ratio, and avoiding highwall mine structure failure as well as surface subsidence, an optimization design process is established as well.

4.1 Stability of the Highwall Top Surface

It is not difficult to find apparent landslide of the intact blocks of rock on the top of the fragmented highwalls in the middle part of Figure 3-10. Due to the relatively large elevation difference between the top and bottom of the highwall, any sliding of the top soil and debris from the top surface of the highwall will put the workers and mining equipment on the working bench at risk. However, since a highwall mining operation normally starts from the foot of an open-pit mine or from the previous contour mine benches, the stability of the highwall top surface should have been dealt with before the highwall mining operations. Even in such a case, the highwall mining operators should examine the top surface of the highwall very carefully and take precautionary actions to avoid any potential land sliding at the highwall top.

4.2 Stability of the Highwall

Due to the close distances between the highwall and miners, and the highwall and surface equipment, the stability of the highwall itself is a potential safety threat for a highwall mining operation as well. Just like the stability of the highwall top surface, the highwall is generally stable. This is because highwalls are created with the rational design during the contour and open-pit mining operations. However, when sufficient movements and deformations occur at the bottom of the highwall due to highwall mining operations, it is quite likely for the highwall itself to become unstable. Since the fundamental cause for such a highwall instability problem is the instability of pillars at the mouth section, rational design of the pillar system at the entry mouth

section is the primary method to stop the highwall failure. Maintaining highwall stability through pillar design will be discussed later.

4.3 Stability of the Entry Roof

Large roof falls in a mine entry could generate substantial stability problems to a highwall mining operation. It usually takes a long time to recover the entrapped equipment and then as a result, the mining operation has to be interrupted. It is quite likely for roof falls to occur when encountering weak and thinly bedded immediate roof strata. There are generally two forms of roof falls: (1) tensile failure at the middle part of the mine entry due to excessive roof sag and bed separations, and (2) cutter roofs at the corner of the entry. It should be noted that the width of a mine entry created by a continuous miner in a highwall mining operation is either 9.5 or 11.5 ft (2.9 or 3.5 m) wide depending on the highwall miner being used, which is much smaller than the regular width of the mine entries and crosscuts in underground coal mines. However, since no artificial roof support is installed during the entire highwall mining operation, the bed separation from the overlying layers and sagging of thin and weak rock layer in the immediate roof should be paid great attention. In order to assess the stability of the mine roof, a rock layer can be treated as a beam with fixed ends once it detaches from its overlying strata as shown in Figure4-1.

Figure 4 - 1 A Beam with Fixed Ends for Assessing Mine Roof Stability in Highwall Mining

Using the beam theory, the deflection of the detached roof layer at a given point of interest can be determined by using the following equation:

$$S(x) = \frac{wx^2}{24EI} (W_e - x)^2 \quad (4 - 1)$$

Where: $S(x)$ – beam deflection at the point of interest, inches

x – distance from the pillar edge, inches

w – load per unit length of a 1-inch thick beam (in entry axial direction), lbs/in

$$w = \frac{b\rho}{1728}$$

b – layer thickness, inches

ρ – density of the rock, lbs/ft³

E – modulus of elasticity, psi

I – moment of inertia, inch⁴

$$I = \frac{b^3}{12}$$

W_e – width of the entry, inches

Through combining the moment of inertia and load w into the equation, the roof deflection can be expressed by Equation 4-2a. The maximum deflection at the entry center (S_{max}) is shown in Equation 4-2b and it can be used as an indicator for roof stability. Once the layer sags significantly, it could increase the likelihood of roof failure considerably. From Equation 4-2b, it is not difficult to note that the maximum deflection of the sagging layer is proportional to the rock density as well as the entry width, and inversely proportional to the elastic modulus and the layer thickness. Especially, the entry width and the thickness of the rock layer will play an important role in determining the maximum deflection. This means that if the rock layer is relatively strong and the layer thickness is relatively large, the chance for this type of bed separation from the overlying layers to happen is relatively small. The roof deflection at a given point can also be expressed in the term of the maximum roof deflection (S_{max}) in Equation 4-2c.

$$S(x) = \frac{\rho x^2}{3,456Eb^2} (W_e - x)^2 \quad (4 - 2a)$$

$$S_{max} = \frac{\rho W_e^4}{55,296Eb^2} \quad (4 - 2b)$$

$$S(x) = \frac{16S_{max} x^2}{W_e^4} (W_e - x)^2 \quad (4 - 2c)$$

If the immediate roof is a soft rock layer, the thickness of the rock layer above a pillar will be shrunk by the additional vertical compressive stress after the entries adjacent to the web pillar are mined. For a web pillar, before extraction, the load supported by a web pillar is equal to the roof area of the web pillar. However, after extraction, the load supported by a web pillar equals the total area of the entry and the web pillar. The incremental load, originally carried by the entry, will also be undertaken by the web pillar. Therefore, the vertical compressive strain in the roof layer at that time can be determined by Equation 4-3.

$$\Delta\varepsilon = \frac{1.1 \cdot h \cdot W_e}{E \cdot W_w} \quad (4 - 3)$$

Where: h – overburden depth, ft

W_w – width of the web pillar, ft

Figure 4 - 2 Deformation of the Soft Roof Layer above a Highwall Mine Pillar

As the thickness of the soft roof layer is shrunk in the vertical direction by an amount of $b \times \Delta\varepsilon$, it will be laterally squeezed into the mine entry by an amount of Δl on each side of the web pillar as shown in Figure 4-2. Under a reasonable assumption that the volume of the roof layer before and after the deformation remains the same, the total lateral elongation can be determined by using the following equation:

$$2\Delta l = \frac{1.1hW_eW_w}{EW_w - 1.1hW_e} \quad (4 - 4)$$

The elongation of the section of the roof layer above the web pillar ($2\Delta l$) will induce an additional sagging of the roof in the entry. The maximum roof deflection after considering the lateral squeezing effect of the soft roof layer (S'_{max}) will be:

$$S'_{max} = \sqrt{\left[\sqrt{\left(\frac{W_e}{2}\right)^2 + S_{max}^2} + \frac{\Delta l}{1.219} \right]^2 - \left(\frac{W_e}{2}\right)^2} \quad (4-5)$$

Apparently, the squeezing effect on roof deflection depends on overburden depth, Young's modulus, entry width, pillar width and the thickness of the roof layer. Among them, the Young's modulus most heavily affects the maximum deflection. When the pillar squeezing effect is considered, the determined S'_{max} using Equation 4-5 can replace S_{max} in Equation 4-2c for the purpose of determining the roof deflection at a given point $S(x)$.

In general, each layer in the roof strata is subjected to not only the layer weight itself, but the load as a result of interaction effect between layers. Therefore, the interaction effect between layers should be taken into consideration when assessing the stability of the roof entry. In order to figure out the interaction effect on the stability of the rock layer in the immediate roof, the key factor is to determine the load undertaken by the first immediate roof layer, which is defined as q (Qian, et al., 2003). On the whole, it is assumed that there are still several thin and weak layers above the first immediate roof layer. Additionally, the load exerted on layers is distributed unevenly. However, in order to analyze the problems conveniently, it is assumed that the stress exerted on each layer is distributed uniformly. Now take the first layer as an example to illustrate the method of calculation of load q .

It is assumed that there are total of $(m+1)$ layers in the roof strata. The thickness of each layer is h_i ($i=1,2, \dots, m$), the unit weight of rock layer is γ_i ($i=1,2, \dots, m$), and the elastic modulus is E_i ($i=1,2, \dots, m$). The number of layers that deform with the first layer is represented by n , as shown in Figure 4-3. The first layer and the other n layers will deform simultaneously, forming a composite beam. According to the theory of composite beams, the shear stress (Q) and bending moment (M) of each cross section of a composite beam are the summation of the shear stress and bending moment of each small cross section, shown in Equation 4-6 and 4-7. According to the theory of material mechanics, the curvature is $k_i = 1/\rho_i$ (ρ_i is the radius of curvature), and the relationship between the bending moment and curvature is shown in Equation 4-8.

$$Q = Q_1 + Q_2 + \cdots + Q_n \quad (4-6)$$

$$M = M_1 + M_2 + \cdots M_n \quad (4-7)$$

$$k_i = \frac{1}{\rho_i} = \frac{(M_i)_x}{E_i I_i} \quad (4-8)$$

Figure 4 - 3 Load Calculation of Roof Layers above a Highwall Entry

Due to the reason that these layers are combined together, the curvature between layers should be same, leading to the redistribution of the bending moment of each layer. The relationship between each layer is shown in Equation 4-9. The procedure to calculate the magnitude of q is shown in the following equations from 4-10a to 4-10d. Finally, the load undertaken by the first layer is derived, which is shown in Equation 4-10.

$$\frac{M_1}{E_1 I_1} = \frac{M_2}{E_2 I_2} = \cdots = \frac{M_n}{E_n I_n} \quad (4-9)$$

$$M_x = (M_1)_x + (M_2)_x + \cdots (M_n)_x \quad (4-10a)$$

$$M_x = (M_1)_x \left(1 + \frac{E_2 I_2 + E_3 I_3 + \cdots E_n I_n}{E_1 I_1} \right) \quad (4-10b)$$

$$(M_1)_x = \frac{E_1 I_1 \cdot M_x}{E_1 I_1 + E_2 I_2 + \cdots E_n I_n} \quad (4-10c)$$

Due to $\frac{dM}{dx} = Q, \frac{dQ}{dx} = q,$

$$(q_1)_x = \frac{E_1 I_1 \cdot q_x}{E_1 I_1 + E_2 I_2 + \dots E_n I_n} \quad (4 - 10d)$$

Where $q_x = \gamma_1 h_1 + \gamma_2 h_2 + \dots + \gamma_n h_n;$

$$I_1 = \frac{b h_1^3}{12}, I_2 = \frac{b h_2^3}{12}, \dots I_n = \frac{b h_n^3}{12}$$

After plugging $q_x, I_1, I_2 \dots I_n$ into Equation 4-10d, the load undertaken by the first layer is derived, as shown in Equation 4-10.

$$(q_n)_1 = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2 + \dots + \gamma_n h_n)}{E_1 h_1^3 + E_2 h_2^3 + \dots + E_n h_n^3} \quad (4 - 10)$$

Table 4 - 1 Calculation Example Table

Layer	Strata	Density, lbs/ft ³	Thickness, ft	Young's Modulus, psi	Tensile Strength, psi
1	Mudstone	120	0.2	7.50E+05	725
2	Mudstone	120	0.1	7.50E+05	725
3	Mudstone	120	0.1	7.50E+05	725
4	Mudstone	120	0.2	7.50E+05	725

The load undertaken by the first immediate roof layer depends on Young's modulus, unit weight of each rock layer, as well as thickness of each rock layer. Therefore, when the interaction effect between rock layers is considered, the determined $(q_n)_1$ using Equation 4-10 can replace w in Equation 4-1 for the purpose of determining the roof deflection. Table 4-1 provides parameters to calculate the load undertaken by the first layer.

Load undertaken by the first layer:

$$q_1 = \gamma_1 \times h_1 = 120 \times 0.2/144 = 0.167psi$$

Load undertaken by the first layer considering the interaction of second layer to the first layer:

$$(q_2)_1 = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2)}{E_1 h_1^3 + E_2 h_2^3} = 0.222 \text{psi}$$

Load undertaken by the first layer considering the interaction of second and third layers to the first layer:

$$(q_3)_1 = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2 + \gamma_3 h_3)}{E_1 h_1^3 + E_2 h_2^3 + E_3 h_3^3} = 0.267 \text{psi}$$

Load undertaken by the first layer considering the interaction of second, third, and fourth layers to the first layer:

$$(q_4)_1 = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2 + \gamma_3 h_3 + \gamma_4 h_4)}{E_1 h_1^3 + E_2 h_2^3 + E_3 h_3^3 + E_4 h_4^3} = 0.222 \text{psi} < (q_3)_1$$

The above calculation shows that the interaction of the second and third layers to the first layer should be taken into consideration. Due to the relatively larger thickness of the fourth layer, it is not likely to cause any impact to the first layer. Therefore, the load undertaken by the first layer is 0.267 psi. If the fourth layer is sandstone, the load transfer process will be terminated as well because of its larger Young's modulus. Therefore, when encountering the relatively stronger and thicker layer, the stress exerted on the first layer will be terminated by that layer, which means that the failure process tends to be ended by that layer. It can also be concluded that if the rock in the interburden is soft and the thickness of each layer is relatively thin, then it is quite likely for the roof layers directly above the highwall mine entry to separate and sag sequentially until they encounter relatively stronger and thicker layers.

Now, change the property of strata to sandstone. The density is 160 lbs/ft³, Young's modulus is 5.0E+06 psi, tensile strength is 600 psi, and the other conditions remain the same. Following the same calculation procedure, the load undertaken by the first layer is 0.356 psi. Under the same geometry condition, the load undertaken by the sandstone layer in the immediate roof is larger than the load undertaken by the mudstone layer in the immediate roof. This is because the density of sandstone is larger than that of mudstone. For the above example of mudstone, if the layer thickness is changed to 0.05, 0.025, 0.025, and 0.05 ft respectively, the load undertaken by the first layer is 0.067 psi. For the above example of sandstone, if the layer thickness is also changed to 0.05, 0.025, 0.025, and 0.05 ft respectively, the load undertaken by

the first layer is 0.089 psi. From this, it should also be noted that the thinner the layer, the less load would be undertaken by the first layer.

For coal measure rocks, the tensile strain should be more critically examined than the compressive strain. In order to assess the tensile strain of the immediate entry roof, the critical tensile strain values deducted from the subsidence monitoring programs performed on various structures affected by mining operation subsidence are used. Based on the published literature and our own subsidence studies, the critical tensile strain values for sandstone and mudstone are 2.0×10^{-3} ft/ft and 1.5×10^{-3} ft/ft, respectively.

For quick reference, Table 4-2 shows the physical and mechanical properties of a number of common coal measure rocks in dry conditions. However, it should be pointed out that the mechanical properties and strength of mudstone could be greatly affected by its moisture content.

**Table 4 - 2 Physical and Mechanical Properties of Coal Measure Rocks in Dry Condition
(Zhao)**

Rock Type	Dry Density, lbs/ft ³		UC Strength, psi		Tensile Strength, psi		Young's Modulus, psi		Poisson's Ratio	Strain at Failure, %
	Min	Max	Min	Max	Min	Max	Min	Max		
Sandstone	119	161	2,900	24,650	580	3,625	2.175E+06	7.250E+06	0.14	0.20
Shale	125	150	725	14,500	290	1,450	7.250E+05	4.350E+06	0.10	
Mudstone	113	168	1,450	14,500	725	4,350	7.250E+05	1.015E+07	0.15	0.15
Limestone	167	170	4,350	36,250	870	3,625	2.900E+06	1.015E+07	0.30	

A computer program has been developed in MS Excel based on the roof beam model. The program can be used to assess the stability of an entry roof when it consists of soft and thin rock layers. The input and the primary calculation page of the program is shown in Figure 4-4. In the data input sections, the user should enter the geometric information of the highwall mine. The ranges of the mechanical and physical properties of common coal measure rocks are given in Table 4-2. The basic derived information is listed in the bottom portion of the table.

Data Input Section		
Project ID:	Program Test	
Max. Overburden Depth (h):	170	ft
Entry Width (W):	11.5	ft
Web Pillar Width (W_w):	4.0	ft
Rock Type of Roof Layer:	Mudstone	
Layer Thickness (b):	0.2	ft
Layer Young's Modulus (E):	7.500E+05	psi
Rock Density (ρ):	120	lbs/ft ³
Tensile Strength:	725	psi
Basic Calculations		
Load/in Length, 1 in deep beam (w):	0.267	lbs/in
Moment of Inertia (I):	1.152	in ⁴
In-Situ Stress (σ_i):	187	psi
Web Pillar Stress (σ):	725	psi
Incremental Stress ($\Delta\sigma$):	538	psi
Incremental Strain ($\Delta\epsilon$):	7.168E-04	ft/ft
Layer Elongation ($2\Delta l$):	2.869E-03	ft
Strain due to Roof Elongation (Tensile):	6.867E-07	ft/ft
Strain due to Pillar Squeezing (Compressive):	2.495E-04	ft/ft

Figure 4 - 4 Input and Primary Calculation of the Roof Stability Assessment Program

Based on Equation 4-2c, the roof deflection profiles of 0.2-ft thick mudstone immediate roof layer ($E = 7.5 \times 10^5$ psi) are derived and shown in Figure 4-5. In the example, the overburden depth, entry width, and web pillar width are 150, 11.5 and 4 ft, respectively. The rock density is 120 lbs/ft³. After taking interaction effect between layers into consideration, the load undertaken by per unit length of a 1-inch thick layer is 0.267 psi. When only the gravity effect is considered, the maximum roof deflection is 0.182 inches. When the pillar squeezing effect as well as interaction effect between two layers are considered, the maximum roof deflection is about 0.374 inches. Therefore, the squeezing effect as well as interaction effect between layers should be taken into consideration when evaluating the stability of the entry roof.

Figure 4 - 5 Roof Deflection Profiles for 0.2-ft Thick Mudstone Layer with and without Lateral Squeezing Effect and Layers Interaction Effect

Figure 4 - 6 Roof Deflection Profiles for 0.2-ft Thick Mudstone and Sandstone Layers

Figure 4-6 shows the roof deflection profiles of 0.2-ft thick mudstone and sandstone layers. The Young's Modulus for sandstone layer is 5.0×10^6 psi, the density is 160 lbs/ft^3 , and the load undertaken by per unit length of a 1-inch thick layer is 0.356 psi based on the calculation shown in the above example. Since the entry width, pillar width, and the thickness of each layer for sandstone as well as mudstone layer are the same, the Young's modulus, as well as the density of rock layers, make big differences in this case. However, for a relatively thick layer spanning over a relatively narrow entry, the estimate maximum deflections are 0.374 and 0.107 inches for mudstone and sandstone respectively, which tends not to cause any stability problems for the entry roof.

Figure 4-7 shows the roof deflection profiles for 0.05-ft thick mudstone and sandstone layers. The load carried by per unit length of a 1-inch thick layer is 0.067 psi and 0.089 psi for mudstone and sandstone, respectively, based on the above calculation analysis. From this figure, the estimate maximum deflections are 4.703 and 0.939 inches for mudstone and sandstone, respectively. It is easy to find that the roof deflection for mudstone is much higher than that of sandstone, and such roof layer deflection could quite likely cause the stability problem for the entry roof. Therefore, the existence of the relatively thin thickness and weak strata in the immediate roof could cause potential stability problems for the entry roof.

Figure 4 - 7 Roof Deflection Profiles for 0.05-ft Thick Mudstone and Sandstone Layers

As it is difficult to precisely figure out the tensile strength of rock materials, tensile strain is often used as a failure criterion, as shown in Table 4-2. Therefore, it is important to determine the strain distribution on the top and bottom surfaces of the roof layer. Equation 4-11 can be used to determine the surface strains on a sagging roof layer. In this equation, the first term is the tensile strain caused by the beam elongation due to sagging, the second is the compressive strain due to pillar squeezing, and the third term is due to beam bending. The positive sign (+) in front of the third term represents the strain on the top surface of the layer and the negative sign (-) represents the bottom surface. If the resulting strain is a positive one, it is in tension and otherwise in compression. For coal measure rocks, the tensile strain should be more critically examined than the compressive strain.

$$\begin{aligned} \varepsilon(x) = & [1.219 \times (\sqrt{1 + 4 \left(\frac{S'_{max}}{l} \right)^2} - 1)] - \left(\frac{1.1hW_eW_w}{EW_w - 1.1hW_e} \right) \\ & \pm \left[\frac{\rho}{6912Eb} (2W_e^2 - 12W_ex + 12x^2) \right] \end{aligned} \quad (4-11)$$

As indicated by Equation 4-11, the thickness of the rock layer will greatly affect the maximum strains at the pillar edge and at the center. The strain profiles of a 0.05-ft (0.6-inch) thick mudstone and sandstone roof layers are shown in Figure 4-8 and Figure 4-9. The maximum tensile strains on the top surface at the entry corner are 0.493% and 0.055% for mudstone and sandstone respectively. The tensile strain for mudstone is larger than the failure strain shown in Table 4-2 while the tensile strain for sandstone is smaller than the failure strain. Failure is most likely to start at the entry corner, which is usually referred to as cutter roof phenomena. The maximum tensile strains on the bottom surface at the center are 0.375% and 0.031% for mudstone and sandstone, respectively. For mudstone, it also has the potential to cause roof failure. Therefore, if the immediate entry roof strata consists of thinly bedded weak rock layers, progressive and upward failure of the immediate mine roof is more likely to occur. This kind of failure is not likely to terminate before encountering the relatively stronger and thicker layer. Then the debris of failed roof layers, if in a large volume, can greatly hinder the mining operations and potentially entrap the underground mining equipment. Therefore, for highwall mining operations to be conducted in coal seams with thinly bedded roof strata, a correct decision to cut some of the thin weak roof rock layers with the main coal seam can be greatly beneficial to the mining operations.

Figure 4 - 8 Surface Strain Profiles of a 0.05-ft Mudstone Layer

Figure 4 - 9 Surface Strain Profiles of a 0.05-ft Sandstone Layer

Figure 4 - 10 Strain Profiles on a 0.2-ft Dry Mudstone Roof Layer

Figure 4-10 shows the predicted strain profile on the top and bottom surfaces of 0.2-ft thick mudstone layer above an 11.5-ft wide entry. A positive strain means the roof layer is in tension while a negative strain means the roof layer is in compression. Apparently, the maximum tensile strains are located directly above the pillar edge. On the top surface, the maximum tensile strain above the pillar edge is 0.0516%. On the bottom surface, the maximum tensile strain is 0.0139%, and occurs at the center of the entry. In comparison to the critical tensile strains in the range from 0.15% to 0.20% (Table 4-2), the tensile strains are still too small to cause tensile failure of these rock layers above the entry roof. It shows the maximum tensile strains directly above the pillar edge are significantly smaller than those of 0.05-ft (2.4 inches) thick layers. The Figure also shows that the magnitudes of the maximum tensile strain on the top surface at the entry corner is much larger than that on the bottom surface at the entry center. Therefore, it is quite likely that the layer begins to fail at the entry corner.

The stresses on the top and bottom surfaces can be determined by using the Equation 4-11.

$$\sigma_{(x)} = E \cdot \varepsilon(x) \quad (4 - 12)$$

Figure 4 - 11 Stress Profiles on the Top and Bottom Surfaces of 0.05-ft Mudstone Roof Layer

Figure 4 - 12 Stress Profiles on the Top and Bottom Surfaces of 0.05-ft Sandstone Roof Layer

Figure 4-11 and Figure 4-12 show the stress profiles on the top and bottom surfaces of 0.05-ft thick mudstone and sandstone layers. The tensile strength of the mudstone (725 psi) and sandstone (600psi) are also plotted in the figure. Figure 4-11 shows that both the maximum tensile stresses on the top surface at the edges of the pillar (3,698 psi) and that on the bottom surface at the entry center (2,809 psi) are larger than the critical mudstone tensile strength. Figure 4-12 shows that both the maximum tensile stresses on the top surface at the edges of the pillar (2,738 psi) and that on the bottom surface at the entry center (1,554 psi) are larger than the critical sandstone tensile strength. However, as the roof sagging is a gradual process, it is more likely for the thinly bedded immediate roof rock layers to fail at the pillar edge in the form of cutter roof than at the center of the entry in the form of tension cracks. This phenomenon of roof failure has been frequently observed in underground coal mines. Therefore, the cutter roof in the highwall mining is caused by tensile failure and not by shear failure as is commonly believed.

Figure 4 - 13 Stress Profiles on the Top and Bottom Surfaces of 0.2-ft Mudstone Roof Layer

The stress profile on the top and bottom surfaces of a 0.2-ft thick mudstone layer is shown in Figure 4-13. For a 0.2-ft thick immediate roof layer, the maximum tensile stresses on the top and bottom surfaces of the layer for mudstone are 387 and 104 psi. In dry conditions, this

thicker mudstone layer is not likely to fail. However, when the mudstone becomes wet, its tensile strength will be greatly reduced and it is possible to fail.

It should be noted that it is hard to keep a mine in dry conditions. When the mudstone is wet, the mechanical properties of coal rocks, especially mudstone, claystone, and shale, could change considerably. First, the uniaxial compressive strength (UCS) decreases as the moisture content increases (Lashkaripour and Ajalloeian, 2000). Since the elastic modulus of sedimentary rocks decreases with the uniaxial compressive strength (Chang, et al., 2006), the wetted rocks are more deformable under the same loading condition. The rock tensile strength is generally linearly proportional to its uniaxial compressive strength (Nazir, et al., 2013). Therefore, the tensile strength of the roof rock layer at wet conditions should be much smaller than that at dry conditions.

4.4 Pillar Stability for Highwall Mining in Single Coal Seam

As mentioned previously, the most important design task for a highwall mining operation is to ensure that the pillars will not fail during and after the mining operation. Highwall pillars, which are formed after driving parallel entries in the seam from the highwall, are long, narrow, rectangular coal pillars left to support the overlying strata and the highwall. Since no artificial supports are provided in the entries, stable pillars are essential for a successful highwall mining operation. If the pillars collapse during active mining operations, it is quite likely for overburden to cave into the mining entries and endanger the underground mining machines. This often involves high risk of destabilizing the slope, burial of highwall cutting equipment, and a loss of minable resources. Most seriously, large deformation or failure of the pillars at the mouth could cause failure of the highwall. Pillar failure in a large contiguous area could also cause chimney or trough types of subsidence events on the ground surface.

In order to avoid these adverse conditions for highwall mining operations, adequately sized web and barrier pillars are the most fundamental to the overall stability and safety of the highwall mining operations (Zipf, 2009). A systematic design of a highwall mining panel, consisting of a number of web pillars bounded by two barrier pillars as shown in Figure 4-14, is employed to achieve the required stability of the mine pillars and to maintain an acceptable recovery ratio of coal reserves. The web pillar is relatively small and is only responsible for carrying the tributary load. The barrier pillar should undertake the entire overburden load, assuming that all of the web pillars in the two adjacent panels have failed completely in an

extreme case. In the design, the stability factors of 1.3 and 1.6 for the web and barrier pillars under normal tributary load conditions are used, respectively. It should be noted that the overburden depth at the mouth section of a highwall mine is normally much smaller than the overburden depth along the remaining length. The pillar widths designed according to the average or maximum overburden depths along the length of the entries should ensure pillar stability at the mouth section. Under the extreme loading condition, when all web pillars in the adjacent panels fail, the stability factor for the barrier pillars is set as 1.0.

Since the height of the web pillar is determined by the seam thickness, the width of the web pillar needs to be designed rationally. From Equation 4-4 in the last section, it can be found that the width of the web pillars plays an important role in affecting the elongation of the roof layer above the web pillar. Then, the maximum deflection, strain distribution, and stress distribution of the immediate entry roof strata will be influenced by the width of the web pillar. Therefore, after determining the width of the web pillar, a stability assessment of the entry roof should be conducted based on the methods proposed in the previous section.

Figure 4 - 14 a Schematic for Highwall Mining Design (Luo, 2014)

4.5 Influence of Multi-Seam Mining Operations

It should be noted that between 20 and 40 percent of highwall mining operations are carried out in closely spaced coal seams. The strata deformations and stress concentrations as a result of past and current mining activities in the underlying or overlying coal seams could impact the stability of the mine structures in the active highwall mine. When the thickness of the interburden strata is smaller than one entry width, the multi-seam mining (MSM) interactions could be potentially strong enough to impact the mining operations (Zipf, 2009). In such scenarios, the MSM interactions have to be cautiously considered in the design and operations of highwall mines.

In order to reduce the undesirable MSM interactions when the coal seams are closely spaced, it is suggested to vertically align the highwall miner entries and pillars as shown in Figure 4-15. As such, if the soft and thin interburden is unable to carry its own weight and the weight of the mining machine, downward mining sequence would be favored. In this scenario, water accumulated in the previous mine entries in the upper seam(s) could potentially create some issues for the mining operations in the underlying coal seam. Though, most of the industry practices in ultra-close, multiple-seam highwalls mine the lower seam first, followed by the upper seam (Peng, 2008). In this manner, the pit would be backfilled to build a platform to approach the upper workable seam.

With the purpose of preventing web pillars from collapsing and highwall failures, it is suggested that reducing the number of highwall mining entries between barrier pillars to five. And if the mining heights in the seams are significantly different, the resulting sizes of the web and barrier pillars for one seam could differ considerably from those in the other seams as well. The larger pillar sizes should be adopted in the design for the pillar alignment vertically as it is recommended in Figure 4-15. Additionally, if the interburden thickness is less than one highwall miner entry width, stacked pillars are effectively a tall pillar with its height equal to the sum of the height of the upper seam pillar, the lower seam pillar, and the interburden thickness (Peng, 2008).

When the interburden strata are adequately competent, the design of the web and barrier pillars can be carried out provided that the highwall mining is conducted in each of the individual coal seams alone. In the following part of this research, two models are constructed with

Examine2D and FLAC to provide examples to find the minimum thickness of interburden where no interaction between the two coal seams is expected.

Figure 4 - 15 Suggested Layout for Highwall Mining in Closely Spaced Multiple Coal Seam (Luo, 2014)

4.6 Model Development in Examine2D

In the model of Examine2D, the material being modeled is based on the following assumptions:

- (a) The geo-materials are homogenous, isotropic and linearly elastic;
- (b) The whole model is established based on the assumption of hydrostatic situation;

(c) The whole model in Examine2D is constructed based on the assumption of plane strain. This means that the modeled excavation is of infinite length normal to the plane section of the analysis, which corresponds well with the highwall mining situation that the web and barrier pillars are very long compared to their cross section dimensions.

The model is constructed based on the following mining and geological conditions:

Overburden depth for the lower coal seam: 170 ft

Mining height in both coal seams: 6 ft

Entry width: 11.5 ft (Entries in both seams are vertically aligned)

Number of the entries in a panel: 5

Multi-seam mining sequence: overmining

Interburden thickness: 11.5 ft

Web pillar thickness: 4 ft

Barrier pillar thickness: 10.6 ft

Density: 160 lbf/ft³

Poisson's Ratio: 0.25

Young's modulus: 2×10^6 psi,

Tensile strength: 122 psi

Cohesion: 183.3 psi

Internal friction angle: 28°

The failure trajectories and strength factor contours for 11.5 ft interburden thickness are shown in Figure 4-16. In Examine2D, if the strength factor is less than one, this indicates the material would fail under given stress conditions. In Figure 4-16, it is easy to note that there is possibility of failure around the entries indicated by the orange regions according to the strength factors in the upper corner of this figure. It is also easy to find that almost all the web pillars in the panels are overstressed, and it is quite likely for them to fail under such loading conditions. In order to determine the failure area of this model, failure trajectories are displayed at grid points where the induced elastic stresses exceed the strength envelop of the material. From Figure 4-16, two intersecting lines (an "X") are found in the web pillars between entries. This means that the overstress phenomenon has occurred and there are potential failures in the web pillars. Figure 4-16 also shows a sign of the pressure arch effect that there are more failure trajectories for inner panel web pillars than the pillars in the outer part of the panel which means that the possibility for inner panel web pillars to fail is much higher than outer panel web pillars. This is because more

stress would be transferred to barrier pillars for the outer panel web pillars than the inner panel web pillars. In particular, most of the web pillars in the inner part of panels are subjected to extreme loading conditions, and they would fail as indicated by red color zones. However, due to the relatively large size of barrier pillars, there should be no stability problems, which reduces the potential for cascading pillar failure.

Figure 4 - 16 Failure Trajectories and Strength Factor Contours for 11.5ft Interburden Thickness

The results derived from Figure 4-16 also verify the conclusion of Zip (2009) that when the thickness of interburden is smaller than one entry width, the interaction caused by multi-seam mining is capable of causing stability issues for highwall mining structures. In order to find out the stable interburden thickness, the interburden thickness is changed in this model until no failure trajectories are found around the web pillars. Finally, the stable interburden thickness of 20 ft is derived, as shown in Figure 4-17. The mining activities in the overlying coal seam will not affect the stability of the underlying coal seam. Under such mining and geological conditions, the pillar designs in each seam could be treated independently. However, it is still required to keep the pillars in both seams as vertically aligned as possible. This is because when upper and lower seam pillars are offset horizontally relative to one another, the interburden is also loaded by stress concentrations under the upper pillar offset edge (Zhou and Haycocks, 1989). The high

concentration stresses developed on the offset rib-side have great potential to lead to cutter roof failure in the interburden. In the Figure 4-18, the pillars in the upper seam are not columnized with the pillars in the lower seam. It can be noted that the interaction between two seams under offset pillar condition is much higher than that under columnized pillar condition. For these geological and mining conditions, the interburden is stable and the mining activities in two seams can be treated independently.

Figure 4 - 17 Failure Trajectories and Strength Factor Contours for 20ft Interburden Thickness

Figure 4 - 18 Failure Trajectories and Strength Factor Contours for 20ft Interburden Thickness under Offset Upper Pillars Condition

4.7 Model Development in FLAC

In this section, with the purpose of further evaluating the stability of interburden between two seams, a FLAC model is constructed to explore whether the 20 ft thickness of interburden is stable. In the meantime, the FLAC model is preformed to check out the stability of the entry roofs and pillars and detect other potential failure mechanisms.

4.7.1 Model Assumption

The whole model is established based on the following assumptions:

- (a) The model is set up based on a lithostatic stress field: vertical and horizontal stresses are equal and based on overburden load (Vandergrift, et al., 2004);
- (b) Both rock and coal are elastic and isotropic material; the calculated in-situ vertical stress is equal to

$$\sigma_z = \gamma H \quad (4 - 13)$$

Where γ - the unit weight of the overlying rock

H - the depth below surface;

(c) To simulate the ground pressure of the miner, appropriate stress is applied in the last entry in the upper seam based on the weight of the highwall miner.

4.7.2 Mining and Geology Conditions

In this model, two seams are extracted by highwall mining. Both coal seams are 6ft thick, and the upper seam entries and lower seam entries are vertically lined up. The geometry of this model is shown in Figure 4-19. The rock mechanics input parameters in the model are shown in the following Table 4-3.

This model is constructed based on the following mining conditions:

Overburden depth: 170 ft

Entry width: 11.5 ft

Web pillar width: 4 ft

Number of entries: 5

Multi-seam mining sequence: overmining

Interburden thickness: 20 ft (Based on the results derived from Exaime2D)

Figure 4 - 19 Geometry of the Model

Table 4 - 3 Rock Mechanics Input Parameters in the Model

Inputs	Coal Seam	Weak Shale
Density (lb/ft ³)	85	162
K (Bulk Modulus) (lbf/ft ²)	3.60E+07	1.92E+08
G (Shear Modulus) (lbf/ft ²)	1.66E+07	1.15E+08
Cohesion (lbf/ft ²)	3.89E+04	2.64E+04
Friction Angle (degree)	28	28
Tensile Strength (lbf/ft ²)	5760	17568
Young's Modulus (lbf/ft ²)	4.32×10 ⁷	2.88×10 ⁸
Poisson's Ratio	0.3	0.25

4.7.3 Boundary Conditions and Constitutive Model

Roller boundaries are applied along the sides and fixed boundaries are applied at the bottom of the model. The left and right side of the model will be fixed in X direction, and the bottom of the model is fixed both in X and Y directions. Gravitational forces are applied to the zones, and allow the in-situ stresses to develop as they occur in nature.

The whole model is constructed with Mohr-Coulomb material, which exhibits an elastic-plastic behavior. And the whole model created in FLAC is a plane-strain model in which the block is considered to have an infinite length normal to the plane section of the analysis.

4.7.4 Model Development

The model is solved in three steps, which is similar to the highwall mining practice. In the first step, the model is generated based on the simplified geological conditions and then run to initial equilibrium state, which simulates the stress state of the virgin coal seam without mining disturbance. In the second step, the entries in the lower seam are extracted. In the last step, the entries in the upper seam are developed and appropriate stress is applied in the last entry in the upper seam.

4.7.5 Calculation Analysis and Result Discussion

In order to analyze the initial equilibrium state, the σ_{yy} stress contours and unbalanced force figures are generated, as shown in Figure 4-20 and 4-21. From Figure 4-20, it is shown that the σ_{yy} stress increases with the increasing depth, which corresponds to the basic theory $\sigma_z = \gamma H$. The vertical stress is increasing linearly, and the value of the vertical stress at the surface is zero, corresponding to the real situation. The largest unbalanced force is decreasing with the increase of computation steps. When the steps are close to 5000, the largest unbalanced force is close to zero, which means that it reaches the equilibrium state. After analyzing these two figures, it can be concluded that the model is under initial equilibrium state, thus it is practical to perform this model.

Figure 4 - 20 the Geostatic State of YY-Stress Contour

Figure 4 - 21 Maximum Unbalanced Force at Initial Equilibrium

The maximum and minimum principal stress distributions around highwall entries are shown in Figures 4-22 and 4-23. In the lower seam, the largest value of the maximum principal stress is $8.00\text{E}+04 \text{ lbf/ft}^2$, and the largest value of the minimum principal stress is $4.00\text{E}+04 \text{ lbf/ft}^2$. In the upper seam, the largest value of the maximum principal stress is $6.00\text{E}+04 \text{ lbf/ft}^2$, and the largest value of the minimum principal stress is $3.00\text{E}+04 \text{ lbf/ft}^2$. Therefore, both the pillars in the upper and lower seams are stable. Through monitoring the stress around the last entry in the upper seam, it is found that the effect of ground pressure of the miner is not apparent, and the ground pressure of the miner causes no concern to the entry stability. It should also be noted that the stress transfer effect between two seams is not serious. Thus, the interburden strata will maintain stability during mining activities. Since the coal seams are stable after mining activities, the mining of the upper seam should have little effect on the stability of the lower seam. The minimum interburden thickness of 20 ft derived from Examine2D is verified. The seam interaction will not pose potential stability problems to these multiple seam mining activities. Through examining the stress above the entry roofs in the Figures 4-22 and 4-23, the relatively low stresses are not likely to cause any stability problems. Therefore, there are no stability problems for this highwall design under such geological and mining conditions.

Figure 4 - 22 Maximum Principal Stress around the Entries

Figure 4 - 23 Minimum Principal Stress around the Entries

Figure 4 - 24 Stress State around the Entries

Figure 4-24 shows the stress state around the highwall entries. In the beginning, the excavation of entries will disturb the initial geostatic state of the virgin area. Because of the close distance of the highwall entries, the induced stresses around each entry would interact with each other. Then after all the entries are excavated, the induced stresses around the highwall entries will have reached a new equilibrium state. From Figure 4-24, it can be found that all the pillars and entry roofs are in elastic state except the web pillar in the middle of the panel in the upper seam. This web pillar's rib is in tension, which has potential for failure. There is a tendency for rib spalling, but not to the degree that would suggest instability of the whole web pillar. From the above discussions, this highwall mining design is rational and there should be no stability problem for the whole highwall structure.

4.8 Highwall Mining Design Optimization

Like any other mining operations, enhancing the recovery ratio of coal reserves without sacrificing mine safety is the primary goal for highwall mining design and operations. When using the tributary design method, the recovery ratio of coal resources within a production panel is restricted by many factors. Among these factors, the overburden depth is the main factor to restrict the recovery ratio of coal reserves because the web pillars are required to carry the entire tributary overburden load, namely half-way to each adjacent pillar and all the way to the surface, with an acceptable safety factor. When the rock strata in the overburden are thick and competent, the pressure arch theory is recommended for use in the systematic design of the highwall mine panels, as shown in Figure 4-15. In this design system, the web pillars in the production panel are left to only carry the overburden load under the pressure arch. Through this design system, a high recovery ratio will be accomplished within the panel. The large barrier pillar, separating the adjacent production panels, is designed to take on the extreme loading condition when the web pillars in the adjacent production panel failed totally with the purpose of avoiding the cascading pillars phenomena. Using a design optimization process, the overall recovery of coal reserves from a highwall mining panel system can be increased without hindering the production practices.

In this suggested design method, the production panel and two adjacent barrier pillars are combined as a system. In order to ensure the existence of a pressure arch within the competent overburden strata, it is required to carefully select the width of the production panel. Under the pressure arch loading condition, the web pillars in the panels just need to carry the overburden load under the pressure arch. As a result, these pillars can be designed smaller than those designed using traditional methods.

In this design system, the pressure arch concept is adopted in the panel pillars design. When an opening is excavated, the weight of the ground above the opening will be transferred outward to the strata around the opening, forming a distressed zone under the pressure arch (IME, 1936). The overburden strata under the pressure arch bend slightly and no longer undertake the super-incumbent mass of strata. The pressure arch is considered to exist in every mining excavation's roof and the load of the superincumbent strata would be transferred to the pressure arch two abutments (IME, 1949). Based on this pressure arch concept, the web pillars only need to undertake the overburden load up to the pressure arch, and barrier pillars will absorb the load from web pillars. The pressure arch formed over a production panel is depicted in Figure 4-25. An ellipse function can be used to mathematically define the pressure arch, as shown in Equation 4-14.

$$\frac{x^2}{a^2} + \frac{y^2}{b^2} = 1 \quad (4 - 14)$$

In this method, with the angle of abutment pressure, α , the semi-minor and major axes of the ellipses, a and b can be correlated, as shown in the Equation 4-15. Based on NIOSH research, the abutment angle is selected as 21° , which is appropriate for US coal mines (Mark and Chase, 1997). Within the equation, a is half width of the pressure arch production panel. The height (b) of the pressure arch can also be determined by the Equation 4-15.

$$b = \frac{a}{\tan \alpha} = \frac{W_p}{2 \tan \alpha} \quad (4 - 15)$$

In the meantime, the thickness of the competent strata (h_c), within which the pressure arch is still able to exist, can be used to determine the maximum allowable panel width (W_{max}). From Figure 4-25, the term h_c can be determined by subtracting the thickness of unconsolidated materials near the ground surface from the overburden depth. The height (b) of the pressure arch located x distance away from the center of the production panel is then determined, as shown in Equation 4-16.

$$y = \frac{\sqrt{a^2 - x^2}}{\tan \alpha} \quad (4 - 16)$$

Figure 4 - 25 Pressure Arch in Highwall Mine Design (Luo, 2014)

The transverse cross-section area under the pressure arch can be analytically determined using Equation 4-14. The total overburden load carried by the web pillars per foot length can be determined by Equation 4-17, considering the average density of the overburden strata to be γ .

$$P = \frac{\gamma \pi W_p^2}{8 \tan \alpha} \quad (4 - 17)$$

The actual panel width should be chosen based on the Equation 4-18. When a proper panel width, W_p , is selected, N entries and $(N-1)$ web pillars can be arranged within the panel.

$$W_p = N \cdot W_e + (N - 1)W_w \quad (4 - 18)$$

The load-carrying capacity of each web pillar can be determined by using Mark-Bieniawski pillar strength formula (Mark, 1995).

$$C_p = 144 \cdot W_w \cdot \sigma_i \left(0.64 + 0.54 \frac{W_w}{m} \right) \quad (4 - 19)$$

In the above equation, m is the mining height of the coal seam. In the Equation 4-19, 900 psi should be selected for the inside strength of the coal (σ_i) for US coal mines based on Mark and Chase's suggestion (1997). However, if clay-rich rock strata, capable of being weakened with great extent when encountering water, are presented in the immediate floor and roof, a reduced strength ($\sigma_i = 600 \text{ psi}$) of the coal can be chosen. The total load capacity of the web pillars in a highwall mining production panel under the pressure arch can be determined by substituting the Equation 4-19 into Equation 4-20.

$$C = (N - 1)C_p \quad (4 - 20)$$

In order to ensure the stability of the production panel, it is very essential to select an adequate average safety factor (SF) for web pillars during the design process, as shown in Equation 4-21.

$$SF = \frac{C}{P} \quad (4 - 21)$$

Since the entry width (W_e) is either 9.5 or 11.5ft depending on the highwall miner being used, the width (W_w) of web pillars is the only unknown variable that needs to be determined from Equation 4-21 after substituting in the corresponding equations (Equations 4-17, 4-18, and 4-19). Once W_w is determined, the recovery ratio of the highwall mining production panel can be determined as:

$$\eta_p = \frac{N \cdot W_e}{W_p} \quad (4 - 22)$$

It should be pointed out that the design procedure for web pillars in a production panel within the pressure arch should be an iterative process. First of all, a rational panel width (W_p) should be selected, which is capable of fitting N entries and $(N-1)$ web pillars within the panel as shown in Figure 4-15. In the meantime, with the purpose of ensuring that the top of the pressure arch still exists within the competent strata, the panel width should be maintained smaller than the maximum allowable panel width (W_{max}). With the purpose of increasing the recovery ratio under the given conditions, the selection of the panel width should comply with a practical range for highwall mining operations.

When the width of web pillars in the production panel are determined, the safety factor for each web pillar should also be examined. The safety factor for each web pillar from the left edge to the right edge of the production panel can be determined by using Equation 4-23. The integral part in the following equation determines the size of the shaded area under the pressure arch in Figure 4-25.

$$SF_i = \frac{C_p}{\gamma} \frac{1}{\int_{(i-\frac{1}{2})W_e + (i-1)W - \frac{W_p}{2}}^{(i+\frac{1}{2})W_e + iW - \frac{W_p}{2}} \frac{\sqrt{(\frac{W_p}{2})^2 - x^2}}{\tan \alpha} dx} \quad i = 1, 2, \dots, N - 1 \quad (4 - 23)$$

It is clear that the safety factor for each web pillars differs with the distance away from the panel center. Figure 4-26 shows the resulting safety factors for each web pillar in a production panel in which six entries and five web pillars are contained with an average web pillar safety factor of 1.3 is selected. Apparently, the web pillars adjacent to the barrier pillars on both sides of the production panel have larger safety factors than those near the center. If the resulting safety factor is significantly lower than 1.0, a higher average safety factor for web pillars should be selected and the whole highwall mining production panel should be redesigned.

Figure 4 - 26 Resulting Safety Factors for Web Pillars in a Production Panel

The barrier pillar should be designed with a much larger width than the web pillars in the production panels. It should be designed with a safety factor of 1.0 to undertake the entire overburden load under the worst loading condition when all the web pillars in the adjacent

highwall mining panels have failed, as presented by Zipf (2005). The load to be undertaken by the barrier pillar is determined as the entire overburden depth within a distance of the summation of the width of barrier pillar and the highwall mining panel (Equation 4-24). The load undertaking capacity for the barrier pillars is determined using Equation 4-19 after replacing W_w with the barrier pillar width (W_b).

$$P_b = h \cdot (W_p + W_b) \cdot \gamma \quad (4 - 24)$$

The overall recovery ratio of the production panel (η) can be determined by Equation 4-25.

$$\eta = \frac{N \cdot W_e}{W_p + W_b} \quad (4 - 25)$$

Equation 4-25 shows that the overall recovery ratio of a highwall mining panel is a function of the production panel width, the barrier pillar width, and the web pillars width. In any mining operations, one of the principal goals is to increase the recovery ratio of the resources with the prerequisite of ensuring safe operation. Because of the relatively complicated equations and some practical constraint (e.g., N has to be an integer number) presented in this section, it is necessary to carry out the optimization design process with a designed program in order to maximize the recovery ratio of coal resources. The optimization process begins with a panel width that ensures the existence of the top of the pressure arch within the competent strata. In the meantime, N highwall entries and $(N-1)$ properly sized web pillars should be able to fit into the panel. In order to satisfy the average safety factor for the web pillars, the determination of the width of the production panel, number of entries, and width of web pillars should go through an iterative process. Then, the barrier pillar is designed based on the determined panel width. In the end, the overall recovery ratio can be determined. There could be many practical plans to design the production panel if a large thickness of the competent overburden strata exist over the production panel. In this case, the design with the largest recovery ratio of coal resource can be discovered.

4.9 Benefit of Using the Highwall Mining Design Method

Similar to any other pillar design tasks, selecting a proper overburden depth is very crucial to a highwall mine design. Since most highwall mining operations would start from the benches left by contour mining operations, the overburden depth along the length of the

production entries could vary significantly. Zipf (2009) has suggested a high average depth (h) to be computed from the design depth from the minimum (most likely at mouth section, h_{min}) and maximum (h_{max}) overburden depths in the following formula.

$$h = 0.75 \cdot h_{max} + 0.25 \cdot h_{min} \quad (4 - 26)$$

It should be noted that a very high weight in Equation 4-26 is assigned for the maximum overburden depth and all sizes of pillars are designed based on this average design depth. It is quite likely that the determined average depth is much larger than the overburden depth at the highwall mouth section. Therefore, the selected safety factors for the highwall mine design would be much smaller than the safety factors of pillars at the entry mouth section, which make the pillars at the highwall mouth section much more stable than the remaining length of the highwall entries. Higher stability of pillars at the mouth section is critical to prevent any highwall failures.

It is quite unlikely for web pillars to fail when designing the web and barrier pillars of a highwall mining production panel according to the pressure arch concept and the above design procedure with a safety factor of 1.3. Even if all the web pillars in a highwall mine panel have collapsed, the stable barrier pillars adjacent to the panel can still allow the pressure arch to exist. In this manner, any strata movements would be controlled under the pressure arch. Therefore, the fundamental cause for surface subsidence, another principal safety issue for highwall mining operations, has been solved.

4.10 Highwall Mine Design Programs

Three spreadsheet programs have been developed to design highwall mine pillars with the proposed design methodologies and concepts. In this section, the detailed descriptions for the highwall mine design programs are presented.

The first design program can be used to assess the roof stability if the thinly bedded weak layer exists in the immediate roof of the coal seam. As proposed in the above section, it is quite likely for the thinly bedded weak layer in the immediate entry roof to sag significantly under the gravity effect, the pillar's squeezing effect, as well as the interaction effect between rock layers. It is possible for the tensile failures to occur either at the corner of the entry or at the middle of the entry if the roof sagging is too large. The roof layers directly above the highwall mine entry would separate and sag layer by layer until encountering the relatively stronger and thicker layers, which tends to result in serious problems to highwall mining operations. Under such situations, it

is necessary to take some approaches to mine the thin and weak rock layers with the main coal seam together. Therefore, in order to terminate such progressive roof failure, it is very important to make correct decisions when selecting the particular rock layer as the immediate roof of the highwall entries. In this design program, the maximum overburden depth along the highwall entries should be selected with the purpose of ensuring the stability of the entry roof even under the extreme loading condition.

In order to assess the pillar stability and panel design under single coal seam extraction condition, the second program is developed based on the tributary area method. A screen capture of the highwall mine pillar design program is shown in Figure 4-27. In this program, the width of the web pillars is designed in following two sections: (1) the section at the entry mouth, and (2) the remaining length of a production panel. At the entry mouth section, the height of the highwall is selected as the overburden depth. Due to weathering, the in-situ strength of coal at the mouth section should be lower than the in-situ strength of coal in deeper locations. By selecting a lower coal strength at the mouth (800 psi) section for the highwall mine design, the pillar will be designed more conservatively and then the stability of the highwall will be strengthened. For the remaining length of the highwall mine entries, the design depth is an average depth calculated by Equation 4-26. The normal in-situ strength (900 psi) should be selected in the design for the remaining length of the highwall entries. The 800 psi coal strength can also be selected for highwall mining operations to be conducted under shallow covers (e.g., less than 300 ft). The pillar strength is computed using the Bieniawski formula (Equation 4-19) which takes the pillar width-to-height ratio into consideration.

The results of this design program are shown in Figure 4-27. In the case of this figure, the highwall height is 70 ft and the maximum depth for overburden is 146 ft. The mining height is 6.0 ft. The in-situ coal strengths in deeper locations and the mouth section are 800 and 700 psi, respectively. The program suggests that the web and barrier pillars be 3.6 and 13.2 ft wide, respectively. All the pillar safety factors meet the requirements and list in the bottom portion of the figure. The safety factor for the web pillars at the entry mouth section is 1.6, which is capable of ensuring the stability of the highwall operations. The overall coal recovery for a highwall mine panel is about 69.0%.

In order to maximize the recovery ratio, a panel design optimization process program is developed. Figure 4-28 shows a screen capture of the design program. In the design optimization process, the critical parameter is the thickness of the competent strata above the mined coal seam.

This program requires the user to input trial values for the width of the production panel and the maximum number of entries within one production panel. Through an optimization process, a feasible design plane that is able to provide the best recovery ratio will be achieved. The Figure 4-28 example adopts the same input parameters from those parameters in Figure 4-27. After running the optimization program, it is shown that the width of the panel is 83.3ft, the number of entries is six, and the width of the five web pillars is 2.9 ft. The width of the barrier pillar is 10.5 ft and the overall recovery ratio of the highwall mine panel is 73.6%, which is 4.6% higher than that derived through the traditional design process shown in Figure 4-27.

As for multi-seam highwall mining operations, especially those operations conducted in close proximity, it is suggested to vertically align the highwall entries and pillars, as shown in Figure 4-15, with the purpose of reducing the adverse multi-seam mining interactions. If the interburden strata are thick and competent, the pillar design in each seam can be treated independently. This is due to the fact that the seam thickness and overburden depth are different leading to different designed sizes of pillars. The design plan for a particular coal seam yielding the largest web and barrier pillars should be selected for the multi-seam highwall mining operations.

Highwall Mining Design Program		
Developed by Yi Luo, Ph.D., P.E., Dept of Mining Engineering, WVU		
Based on R. Karl Zipf, Jr., NOISH, Pittsburgh, PA		
Input Information		
Mining Height, m :	6.0	ft
Average Rock Density of Overburden Strata, γ :	160.0	lbs/ft ³
In-Situ Coal Strength, σ_i :	800.0	psi
In-Situ Coal Strength near Outcrop, σ_{io} :	700.0	psi
Highwall Miner Width, W_h :	11.5	ft
Maximum Overburden Depth, h_{max} :	146.0	ft
Thickness of Competent Strata, h_c :	130.0	ft
Highwall Height, h_{min} :	70.0	ft
Web Pillar Design Safety Factor:	1.3	
Barrier Pillar Design Safety Factor:	1.0	
Number of Entries Between Barrier Pillars, N :	6	
Resulting Design Parameters		
Design Depth, h_d :	127.0	ft
Rib-to-Rib Panel Width, W_{panel} :	86.8	ft
Web Pillar Width for Design Depth, W_{web1} :	3.6	ft
Web Pillar Width at Mouth, W_{web2} :	2.8	ft
Barrier Pillar Width, $W_{barrier}$:	13.2	ft
Design Web Width, W_{web} :	3.6	ft
Panel Overall Recovery Ratio:	69.0%	
Determined Pillar Stresses and Strengths		
Web Pillar Stress at Design Depth:	591	psi
Web Pillar Strength at Design Depth:	768	psi
Web Pillar Stress at Mouth:	391	psi
Web Pillar Strength at Mouth:	626	psi
Barrier Pillar Stress:	1,227,575	lb
Barrier Pillar Strength:	1,227,576	lb
Resulting Pillar Stability Factors		
Stability Factor for Web Pillar at Design Depth:	1.30	
Stability Factor for Web Pillar at Mouth in Design:	1.60	
Actual Stability Factor for Web Pillar at Mouth:	1.90	
Stability Factor for Barrier Pillar (Assuming all web pillars failed)	1.00	

Figure 4 - 27 Screen Capture of the Highwall Mine Pillar Design Program

Input Obtained from Previous Page		
Thickness of Competent Strata, h_c :	130	ft
Max. Panel Width for Pressure Arch to Exist, W_{max} :	99.8	ft
Input Trial Values (Only in this section)		
Panel Width in Design, W_p :	83.3	ft
Max. Number Entries Allowed in a Panel, N_{max} :	6	
Optimization Design Output		
Actual Number in Design, N :	6	
Web Pillar Width, W :	2.9	ft
Total Load under Pressure Arch, P :	1,135,263	lbs/ft
Web Pillar Load Capacity, C_w :	295,168	lbs/ft
Average Web Pillar Safety Factor:	1.30	
Required Barrier Pillar Width, W_b :	10.5	ft
Barrier Pillar Safety Factor:	1.00	
Recovery Ratio in Production Panel:	82.9%	
Overall Recovery Ratio, h :	73.6%	
Web Pillar Safety Factor at Mouth Section:	1.61	
Max. Height of Pressure Arch, h_{arch} :	108.5	ft

Figure 4 - 28 Design Optimization for Highwall Mining Panel Based on Pressure Arch Concept

Chapter 5 Conclusions and Recommendations

5.1 Summary

Highwall mining is a relatively new mining method, which is the preferred, and often the only feasible method to recover the coal reserves in the central Appalachian coalfields. These coal seams used to be mined by mountain-top-removal, underground, contour, and auger mining methods. Compared to the mountain-top removal mining method, highwall mining method could significantly reduce environmental impacts. Compared to the contour and auger mining methods, the new method can considerably increase productivity and the recovery ratio of coal resources. Since it is a hybrid of surface and underground mining methods, it is safer and much more productive than the underground mining method. Therefore, it is probable that more coal operators will choose the highwall mining method to recover the high-value coals in Appalachian coalfields.

By far, the greatest ground control safety concerns in highwall mining operations are rock falls from the highwall and mining equipment entrapment underground. Generally, there are two factors affecting the highwall mining operations, namely geologic structure and highwall structure stability. As for geological constraints, particular precautions should be considered to minimize the risk of failure associated with hillseams. In the highwall mine design, more efforts should emphasize maintaining the stability of the mine roof and pillars to avoid safety hazards to both personnel and mining equipment, ensuring a successful highwall mining operation.

The method to assess the stability of the entry roof is proposed by applying the beam theory. When evaluating the stability of the entry roof, the gravity effect, the squeezing effect, and the interaction effect between layers are taken into consideration. After comparing the deflection, stress, and strain profiles of 0.05 ft sandstone and mudstone roof layers, it can be concluded that the existence of relatively thin and weak layer in the immediate roof could cause potential stability problems for the entry roof. Therefore, for highwall mining operations to be conducted in coal seams with thinly bedded roof strata, a correct decision to cut some of the thin weak roof rock layers with the main coal seam can be greatly beneficial to mining operations. Then, the pressure arch concept is applied for the systematic design of the highwall mining operation. Within this theory, the web pillars in the production panel are designed to only carry the overburden load under the pressure arch. Then, barrier pillars are designed to undertake the extreme loading condition when the web pillars in the adjacent production panel fail completely.

Through a design optimization process, the overall recovery of the coal reserves can be greatly increased. For multi-seam highwall mining operations, the number of entries is recommended to reduce to five and the largest web and barrier pillar sizes should be adopted in the design and vertically aligned into seams.

Numerical modeling results obtained from Examine2D and FLAC models show that, under given conditions in the thesis, the stable interburden thickness is 20 ft where no interaction between the two coal seams is expected. The Examine2D and FLAC models also support the claim that when the thickness of the interburden strata is smaller than one entry width, the multi-seam mining (MSM) interactions could be potentially strong enough to influence the mining operations. Furthermore, both the Examine2D and FLAC modeling efforts show that the pillars, the roof, and the interburden would remain stable under these geology and mining conditions. Additionally, the effect of ground pressure of the miner is not apparent, and it causes no concern to stability.

In the end, three spreadsheet programs are developed for assessment of highwall mine structures, for the design of the web and barrier pillars, and for the optimization design process, based on the proposed design concepts and methodologies. In these programs, the pillars are designed at the mouth section and the remaining length of the entry, respectively. Therefore, the safety factors for the web and barrier pillars at the mouth section would be much higher than the selected safety factors in the design, making the pillars at the mouth section much more stable than the remaining length of the entries. Stable pillars will provide conditions for smooth mining operations and avoid immediate and long-term mine subsidence on the ground surface.

5.2 Future Research Recommendations

Based on the investigations conducted in this research, the following work needs to be updated:

(1) Future work should find the stable interburden under different overburden depth while considering different properties of interburden.

(2) FLAC2D model should be modified to better simulate the practical geological and mining conditions.

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