Analysis of Highwall Mining Stability - The Effect of Multiple Seams and Prior Auger Mining on Design

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ABSTRACT

The objective of this is paper is to review the current state of knowledge and practice in highwall mining (HWM). HWM has become a widely-applied method in surface mining, commonly used alone or in conjunction with contour or slot mining. It provides 800-feet to 1,200-feet of additional recovery when the economic stripping ratio is reached in contour mining or in slot mining when surface access to a reserve is limited. A significant attribute of the highwall miner is its versatility. HWM has been used successfully to mine

- abandoned pre-reclamation law highwalls,
- points or ridges uneconomic to mine by underground or other surface methods,
- outcrop barriers left adjacent to underground mines,
- separate benches of the same seam where the parting thickness or quality differences between benches render complete extraction uneconomic,
- previously augered areas containing otherwise inaccessible additional reserves and
- close or widely spaced multiple seams.

The theory and design methods to assess roof, pillar, and floor stability are presented followed by three case histories. Simple design charts for sizing HWM web and barrier pillars are also presented. A recommended web pillar width may be obtained from the design charts given the overburden depth, the HWM cut width, and the mining height. Given the depth and panel width for a set of HWM cuts, another set of charts gives a suggested barrier pillar width.

The case histories, from Northern and Southern Appalachia are used to illustrate the application of rock mechanics to quantify the stability of the highwall, roof, web pillars, and floor. The case histories involve 1) mining through a previously augered highwall, 2) mining under back-stacked spoil and 3) selective mining of closely spaced benches of the same seam.

Because each site is unique, the appropriate pre-mining geotechnical analyses range from the calculation of roof, web pillar, and floor bearing capacity stability factors to detailed numerical modeling of the auger and underground mine workings. When operating in the vicinity of existing underground mine or auger workings, the determination of ground deformation and strains resulting from highwall mining is a necessary facet of a ground control investigation.

INTRODUCTION

Auger and highwall mining has evolved from a secondary production method to become an integral part of many surface mining operations throughout the U.S. and the world. Auger mining in the Appalachians began sometime in the mid 1940s when surface miners turned vertical blast hole drills horizontal at the economic limit of a surface pit to recover more coal from the highwall. Modern auger and highwall mining has evolved from this humble beginning into a highly productive, sophisticated, high-technology, surface coal mining technique requiring four to five miners per shift.

Volkwein, et al. (*1*) review the evolution of auger and highwall mining systems in the U.S. including the earliest augers dating from the mid 1940s, early highwall mining concepts such as the "Carbide Miner[1]", the "Push-button Miner" (1970s), the "Edna Miner" and the Metec miner (1980s) and finally several continuous haulage concepts in the 1990s including CONSOL's "Tramveyor" and Arch Coal's "Archveyor." Arrowsmith (*2*) and Fiscor (*3*) discuss recent developments in highwall mining techniques. Currently, two manufacturers dominate the market for highwall mining systems with each having about 30 systems in operation. The Superior Highwall Mining Company (SHM) (*4*) developed the Superior Highwall Miner, based upon the Metec design. Mining Technologies Inc., now International Coal Group, Inc. – ADDCAR Systems, LLC (*5*) developed the ADDCAR system.

The two systems are similar in that a cutter head, consisting of a continuous miner head and gathering arms is attached to a series of coal transport modules that are added as the cutter head is advanced into the coal seam. The machines are operated from an enclosed, climate controlled cab, located at the rear of the unit. The SHM, shown in photograph 1 uses 20-foot long rectangular "push beams" that are dropped in place as the cutter head is advanced. Twin spiral augers are fully enclosed within the "push beams" and are used to move the coal away from the face.

The ADDCAR system relies upon a series of conveyor cars that utilize conveyor belts to transport coal from the face, as shown in photograph 2. Another significant difference between the two HWM systems is the guidance and monitoring systems utilized to

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[1]Mention of specific product or trade names does not imply endorsement by NIOSH.

direct and monitor the HWM while cutting to reduce out-of-seam dilution and to maintain a constant web width. The guidance and monitoring system technology, which is outside of the scope of this paper, uses a combination of sophisticated PLC control, video cameras, and methane monitors, to remotely control the HWM operation.

Recent estimates shown in Table 1 (6) suggest that auger and highwall mining may account for around 45 million tons of clean coal production representing about 4% of the U.S. total production. In addition to being a very productive and economic coal production method, analysis shows that auger and highwall mining appears to be as safe as surface coal mining (6). However, there are geotechnical considerations with highwall mining that require careful engineering to realize the full economic and safe potential of highwall mining.

Photograph 1. "Push beam" being loaded into a SHM.

Photograph 2. Conveyor car on the launch vehicle of an ADDCAR Systems Highwall Miner.

Highwall stability is the overriding concern in highwall mining. Zipf and Mark (7) discuss the major factors affecting highwall stability, namely, geologic structure (i.e., hill-seams and mud-seams) and web pillar stability. Unfortunately, little can be done to control the location of hill-seams and mud-seams in a highwall, and they are difficult to detect reliably. Daily inspection of benches above an active highwall mining operation to look for cracks and signs of movement is one recommended standard operating procedure, easily integrated into the standard pre-shift inspection done by a knowledgeable, experienced company official. High technology tools such as GroundProbe Inc.'s Slope Stability Radar may also prove useful in early detection of movement and potential instability in coal mine highwalls (8). Slope Stability Radar is presently demonstrating its potential at several open pit copper mines in the U.S. (9). Using highwall slope angles in the 70 to 80 degree range could aid in eliminating much of the stability hazard from hill-seams (7).

This paper concentrates on web pillar stability, a major factor controlling highwall stability. The basic equations for computing pillar strength and applied stress are presented. By making certain assumptions, simple design charts for estimating minimum web and barrier pillar widths have been derived, and they are presented herein and compared to case history data. These charts apply to many routine highwall mining situations. However, exceptions are frequently encountered. The case histories are used to illustrate many common exceptions, namely 1) the presence of old auger holes, 2) highwall mining under back-stacked spoil and finally, 3) closely-spaced multiple seam mining.

Auger and highwall mining are safe and productive surface coal mining methods provided that proper geotechnical engineering is conducted. Failure to properly engineer web and barrier pillars can result in highwall failure, trapped highwall mining equipment, and significant economic impact as most of the equipment lies under the highwall. These failures result in lost reserves, low productivity and potential miner safety issues as personnel work in close proximity to the highwall. Recovery of a stuck miner can be inherently hazardous since it involves operating under extra-ordinary conditions.

ANALYSIS AND DESIGN OF HIGHWALL MINING WEB AND BARRIER PILLARS

Prior work by Zipf (10, 11) derived simple design equations for web and barrier pillars. The stability factor for a web pillar is

$$SF_{WP} = \frac{S_I [\ 0.64 + 0.54\ W_{WP}/H\ ]}{[S_V (W_{WP} + W_E)/W_{WP}]} \qquad (1)$$

Table 1 - Estimated Auger and Highwall Mining Production for 2003

Machine	Approximate number in operation	Productivity (raw tons per year)	Production (raw tons)
Superior Highwall Miners	30	650,000	20,000,000
ADDCAR Highwall Miners	30	1,000,000	30,000,000
Augers	150	100,000	15,000,000
		TOTAL (raw tons)	**65,000,000**
		TOTAL (clean tons)	**45,000,000**

Where: S_I = in situ coal strength,
$\quad\quad\quad$ S_V = in situ vertical stress,
$\quad\quad\quad$ W_{WP} = web pillar width,
$\quad\quad\quad$ W_E = highwall miner cut width, and
$\quad\quad\quad$ H = mining height.

In situ coal strength is normally assumed as 900 lb/in^2, unless laboratory testing, previous web pillar behavior, coal cleat spacing, or site-specific field data suggests a different value to be appropriate. Equation 1 is sensitive to in-situ coal strength so care must be exercised in assuming a value with site specific testing. The mining height is the height of the HMW cut which may be greater or less than the seam thickness, dependent upon out-of-seam dilution or if "head" coal is left in the immediate roof to improve roof conditions. The type of HWM equipment dictates the cut width that varies from 9-ft to 12-ft. In situ vertical stress depends on the overlying rock density and overburden depth. Vertical stress gradient is typically 1.1 lb/in^2/ft. The high average overburden depth can be used to calculate web pillar stress. This is the overburden under which 80% of the HWM panel exists or alternatively may be calculated using equation 2 where a continuous slope is present.

$$OB_{Design} = 0.75 * OB_{MAX} + 0.25 * OB_{MIN} \quad\quad (2)$$

Where: $\quad$ OB_{MAX} = maximum overburden depth
$\quad\quad\quad\quad$ OB_{MIN} = minimum overburden depth.

For design purposes, the stability factor for web pillars typically ranges from 1.30 to 2.00 dependent upon the subsidence constraints imposed by current and future surface usage. Based on data in HWM ground control plans submitted by coal mine operators to MSHA, studies (6) found that the stability factor for web pillars ranged from 1.30 to 1.60 in about 30% of the plans and exceeded 1.60 in 45% of the plans. The stability factor range obtained from the ground control plans is based on statistical information provided where the appropriate stability factor is unique to each mine site. This survey also found that the width-to-height (W/H) ratio of web pillars exceeded 1.00 in 75% of the cases examined. In general, keeping the web pillar W/H ratio above 1.00 is desirable to maintain web pillar integrity. Narrow web pillars increase the demand on surveying accuracy as an orientation alignment error of about 4 minutes results in a 1-foot deviation in

web pillar width in an 800-ft-long cut. For this reason, a minimum web width is typically specified irrespective of overburden depth.

If the number of web pillars in a panel is selected as "N", then the panel width is given by

$$W_{PN} = N (W_{WP} + W_E) + W_E \quad\quad (3)$$

A barrier pillar is commonly used to separate adjacent panels and prevent ground control problems from cascading along the entire length of highwall. Neglecting the stress carried by the web pillars (i.e., assuming that they have all failed), the stability factor for a barrier pillar is determined as

$$SF_{BP} = \frac{S_I [\, 0.64 + 0.54\, W_{BP} / H \,]}{[S_V (W_{PN} + W_{BP}) / W_{BP}]} \quad\quad (4)$$

Where: S_I = in situ coal strength,
$\quad\quad\quad$ S_V = in situ vertical stress,
$\quad\quad\quad$ W_{PN} = panel width,
$\quad\quad\quad$ W_{BP} = barrier pillar width, and
$\quad\quad\quad$ H = mining height.

Because the stress carried by web pillars within a panel is neglected, the stability factor for barrier pillars can be as low as 1.00. Studies (6) found that the width of barrier pillars exceeded 16-ft in more than half the cases examined and more important, the W/H ratio for barrier pillars exceeded 3 in 66% of the cases. Barrier pillars with a W/H ratio greater than 3 are superior for sound geomechanics reasons.

Based on equation 1, web pillar design charts are developed and presented in figures 1 and 2. Figure 1 applies to a 9-ft-wide highwall miner cut, while figure 2 applies to a 12-ft-wide cut. In figures 1 and 2, options a and b apply to stability factors of 1.3 and 1.6, respectively. Similarly, based on equation 4, design charts for barrier pillars are presented in figure 3. Options a, b and c apply to panel widths of 100, 200 and 400 ft, respectively. Note that this design chart assumes a barrier pillar stability factor of 1.0 and it neglects any load carrying capacity of the web pillars within a panel. To use figures 1, 2 or 3, the user begins with the design depth on the x-axis, moves up vertically to the applicable mining height and then moves left horizontally to the y-axis where the suggested web (or barrier) pillar width is read.
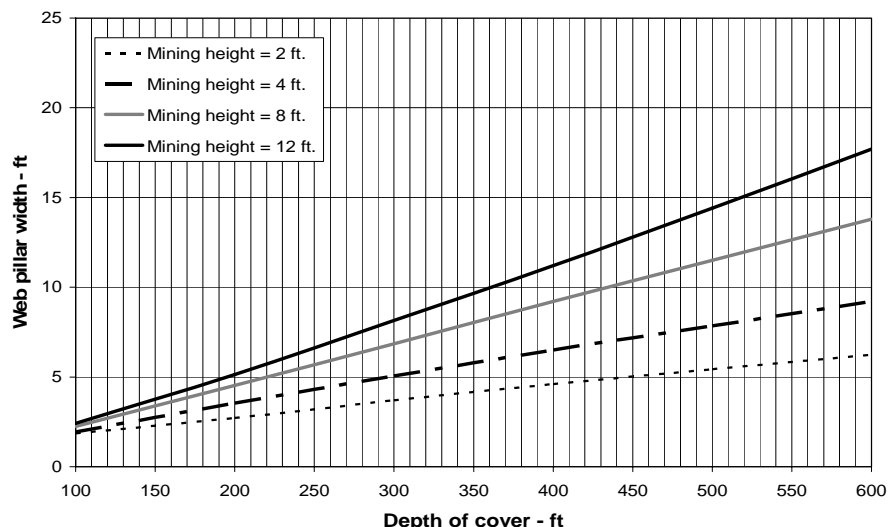

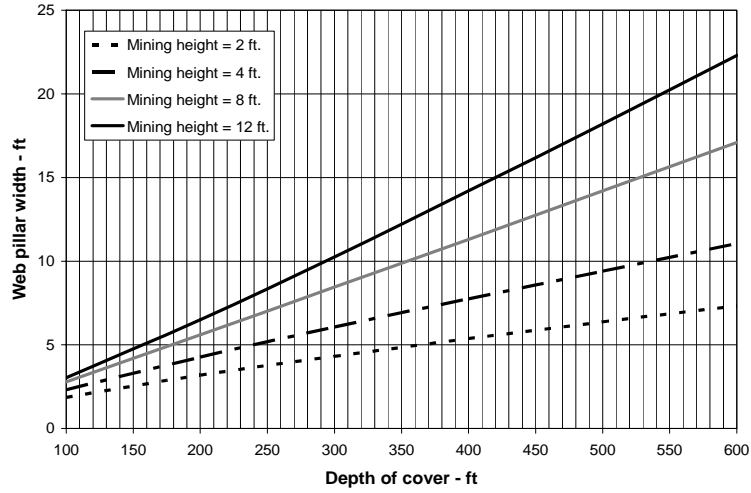Figure 1A – Suggested web pillar width with stability factor of 1.3, coal strength of 900 lb/in^2

Figure 1B – Suggested web pillar width with stability factor of 1.6, coal strength of 900 lb/in^2 and 9-ft wide cut.
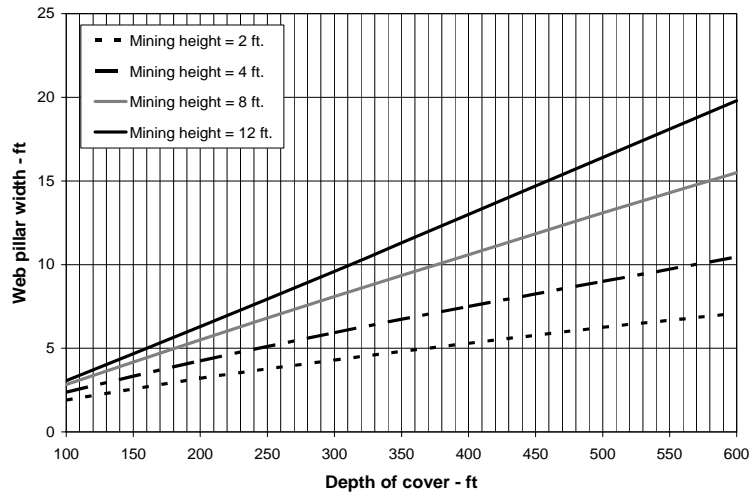

Figure 2A – Suggested web pillar width with stability factor of 1.3, coal strength of 900 lb/in^2 and 12-ft wide cut.
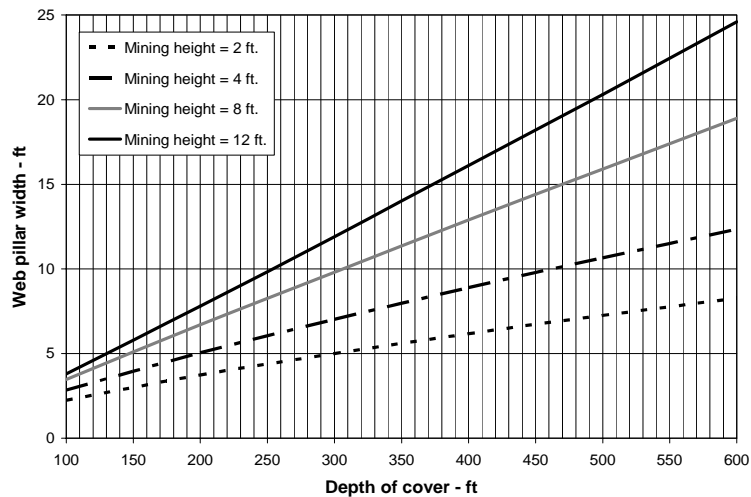

Figure 2B – Suggested web pillar width with stability factor of 1.6, coal strength of 900 lb/in^2 and 12-ft wide cut.
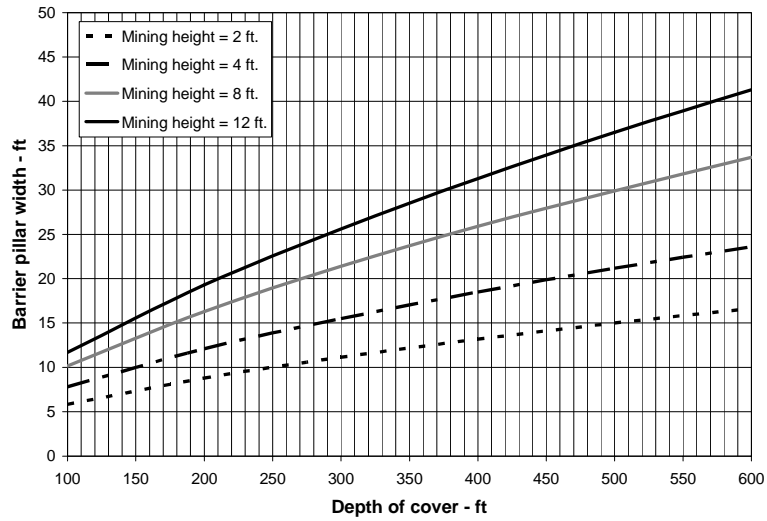
Figure 3A – Suggested barrier pillar width for 100-ft-wide panel assuming coal strength of 900 lb/in^2 and stability factor of 1.0.
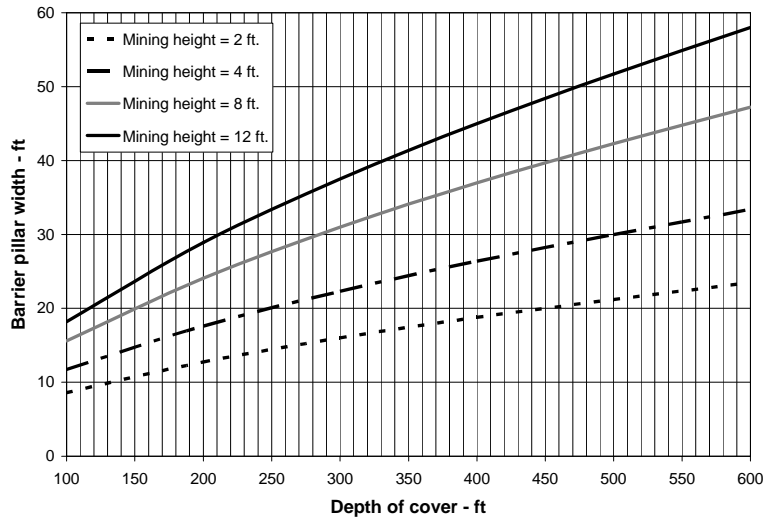

Figure 3B – Suggested barrier pillar width for 200-ft-wide panel assuming coal strength of 900 lb/in^2 and stability factor of 1.0.
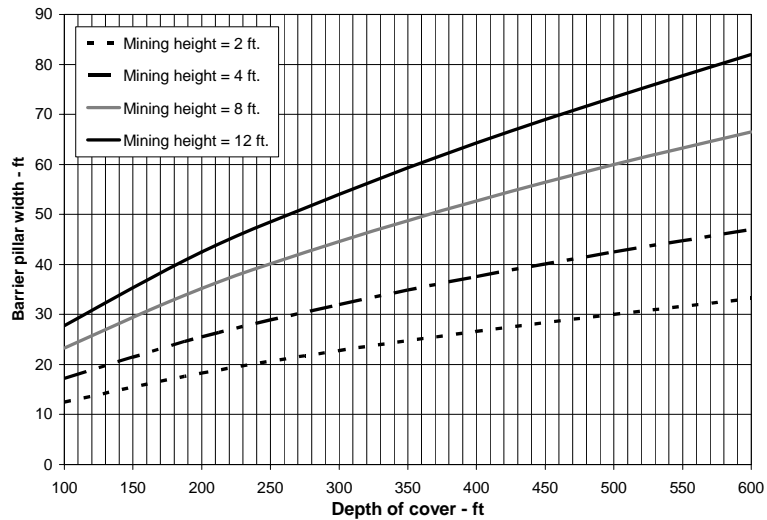

Figure 3C – Suggested barrier pillar width for 400-ft-wide panel assuming coal strength of 900 lb/in^2 and stability factor of 1.0.

HIGHWALL MINING CASE HISTORIES

The application of web pillar design is examined in the discussion of three case histories. The intent of the case histories is to illustrate different circumstances under which HWM is conducted and the required changes in engineering design to accommodate unique mining circumstances.

Case Study A – Mining Into Old Auger Holes

In Appalachia, HWM is often conducted in previously augered highwalls. Although inexpensive and efficient, auger mining sterilizes coal reserves past the typical 50-ft to 150-ft penetration depth. The greater 800-ft to 1,200-ft penetration of a highwall miner permits recovery of the "island" of coal left in the center of a ridge when the perimeter has been augered.

The concern in HWM through a previously augered highwall is that web stability in the initial veneer of augering is a function of the auger web widths. The auger webs were either pre-defined in the case of multiple head machines or reduced to the minimum required to maintain highwall stability during augering. The initial challenge is to determine the number of auger webs to be skipped between adjacent highwall miner cuts.

This case history involves re-mining a previously augered Stockton seam highwall in Southern West Virginia. The auger mining was done using a single head 42-in (3.50-ft) diameter auger in the bottom split of the Stockton seam to produce a clean, direct ship product. This resulted in leaving a substantial amount of head coal given the 7.32-ft to 4.30-ft range of mining height in the proposed re-mining area. A portion of the previously auger highwall is shown in Photograph 3.

Photograph 3. A Previously Augered Stockton Seam Highwall.

The auger-web-width ranged between 14-in and 22.5-in with an average width of 17.6-in. Although the depth of the auger holes is unknown, the company engineer said that based upon similar mining on the property 150-ft is probably the deepest penetration. The stability of the existing auger webs left between the 42-in diameter auger holes was examined assuming an average 17.6-in width and using in-situ coal strength of 676 lb/in^2 based on prior projects in the area.

Auger web stability was evaluated by considering each web as a long, narrow pillar and calculating pillar strength using the well accepted Mark/Bieniawski pillar strength formula that defaults to equation 1 with the long pillar lengths of HWM. When calculating the auger web strength, the 42-in (3.50-ft) auger diameter was used for the pillar height although significant "head" coal is present above the auger hole.

However, when calculating auger web stability to determine the number of auger webs to skip between adjacent HWM cuts, the full seam thickness was used for the mining height. Because the HWM cuts the entire seam, it leaves a pillar of the same height as the seam. Web pillar stress is calculated using the tributary area approach shown in equation 5. Recognizing that a portion of the overburden consists of spoil from surface mining of the overlying No. 5 Block seam, the overburden density was derated from the standard value of 160 to 150 lb/ft^3 or 1.1 to 1.04 lb/in^2/ft. The actual height of spoil was found to be highly variable and therefore difficult to use in the overburden stress calculation. The reduced overburden density is a compromise between using the standard rock density and the spoil and intact overburden heights.

$$S_p = \frac{1.04(H)(w + B)(l + B)}{(w)(l)} \qquad (5)$$

Where: S_p = pillar stress (lb/in^2),
H = overburden depth (ft),
w = pillar width (ft),
B = entry or crosscut width (ft), and
L = pillar length (ft).

The auger hole depth was not accurately recorded at the time of mining. Consequently, an auger web length of 150-ft was used since this is likely the greatest penetration achieved with a single head auger. The previously augered highwall was divided into individual sub-areas based upon seam thickness and overburden depth prior to conducting the auger web stability calculations. The auger web stability calculations were initially performed by considering only the auger web and therefore reported in terms of lb/in^2. When calculating the number of web pillars to skip between adjacent HWM cuts, it was easier to determine the tons of overburden load placed on both the HWM cut and the augered area. Similarly, the bearing capacity of the aggregate of the auger webs was reported in tons.

The planned re-mining was done using a HWM machine that makes an 11.50-ft-wide cut into the highwall. The 11.50-ft-wide cut encompassed two 42-in-diameter auger holes and a 17.6-in average auger web, plus it exposed the next auger hole leaving a void space of 13.43-feet. Therefore, all HWM calculations were done assuming a 13.43-ft-wide void and not the standard 11.50-ft-wide HWM cut.

The web pillar stability factors for each sub-area were presented in table 2, which documents the stability factors for the existing auger webs from a nominal 20-ft of overburden to the maximum overburden depth present within the assumed 150-ft-penetration depth of the auger hole. Table 3 shows the number of auger webs that should be skipped between adjacent HWM cuts. The boldface and shaded areas of tables 2 and 3 are those combinations of web width and overburden thickness that are not sufficiently stable to insure the web stability.

Table 2. Auger Web Width Stability.

Auger hole diameter B (feet)	Web length l (feet)	Web width w (feet)	Coal seam height h (feet)	Web pillar height h (feet)	Overburden depth (feet)	Overburden Stress (lb/in²)	Bieniawski pillar strength formula (lb/in²)	Bieniawski safety factor	Mark/ Bieniawski strength formula (lb/in²)	Mark/ Bieniawski safety factor
Stockton Seam - Pillar Safety Factors - Existing Single Head 42 Inch Diameter Auger Holes Bieniawski and Mark/Bieniawski Pillar Strength Formulae										
Auger Hole Stability - Areas 50-4, 27-3, & 27-2 - 17.58 Inch Web Thickness Between 42 Inch Diameter Auger Holes										
3.50	150	1.47	5.00	3.50	20	70	535	7.58	585	8.30
3.50	150	1.47	5.00	0.50	40	141	535	3.79	585	4.15
3.50	150	1.47	5.00	3.50	60	211	535	2.53	585	2.77
3.50	150	1.47	5.00	3.50	80	282	535	1.90	585	2.07
3.50	150	1.47	5.00	3.50	100	352	535	1.52	585	1.66
3.50	150	1.47	5.00	3.50	120	423	535	1.26	585	1.38
3.50	150	1.47	5.00	3.50	140	493	535	1.08	585	1.19
Marginal Safety Factor										

Table 3. HWM Web Pillar Stability and the Number of Auger Webs Skipped Between HWM Cuts.

Highwall miner cut b' (feet)	Auger hole diameter b (feet)	Highwall miner cut length l (feet)	Web width w (feet)	Coal seam height h (feet)	Number of web pillars between HWM cuts	Overburden depth (feet)	Overburden load (tons)	Bieniawski pillar strength formula (tons)	Bieniawski safety factor	Mark/ Bieniawski strength formula (tons)	Mark/ Bieniawski safety factor
Stockton Seam - Pillar Safety Factors - Number of Skipped Auger Webs Between Highwall Miner Cuts Bieniawski and Mark/Bieniawski Pillar Strength Formulae											
Area 50-4, 27-3, & 27-2 - Number of Auger Webs Between Each HWM Cut											
13.43	3.50	150	1.47	5.00	4	20	7,491	31,902	4.26	34,137	4.56
13.43	3.50	150	1.47	5.00	4	40	14,981	31,902	2.13	34,137	2.28
13.43	3.50	150	1.47	5.00	5	60	25,823	39,877	1.54	42,671	1.65
13.43	3.50	150	1.47	5.00	8	80	47,837	63,804	1.33	68,274	1.43
13.43	3.50	150	1.47	5.00	15	100	98,898	119,632	1.21	128,014	1.29
13.43	3.50	150	1.47	5.00	15	120	118,677	119,632	1.01	128,014	1.08
13.43	3.50	150	1.47	5.00	15	140	138,457	119,632	0.86	128,014	0.92
Marginal Safety Factor											

By reviewing Table 3 for each sub-area, it is readily apparent that the ability to re-mine the existing auger holes is limited by the stability of the existing auger web pillars. A threshold SF of 1.30 is used to separate mineable from marginal areas. It should be noted that a stability factor of 1.30 is low and is for short-term stability only. It is clear from Table 3 that the ability to skip auger webs to maintain highwall stability rapidly diminishes at greater than 100-ft of overburden. A skip of one HMW cut is recommended between each group of 10 HWM cuts to provide a barrier pillar and prevent a squeeze in previously mined HWM cuts and auger webs from cascading onto the active group of HWM cuts.

The mining was successfully done by a contractor who did initially adhere to the recommended number of auger web skips between adjacent HWM cuts. As the job progressed and the contractor gained experience with the behavior and response of the auger webs to HWM, the number of skipped auger webs was selected based upon a combination of engineering calculations and site conditions.

Case Study B - Mining Under Back-Stacked Spoil

A highwall web pillar collapse occurred in the 11th cut of a peninsula-shaped remnant left after Pittsburgh No. 8 seam was contour surface mined in the 1970's. The reclaimed surface mine was re-opened for HWM. The spoil was removed and a second contour cut was taken to provide a clean, stable highwall under which to face up the Southwest side of the peninsula and operate a HWM. HWM web pillar widths were designed using the anticipated overburden range, mining height, and a 900-lb/in² in-situ coal strength. A 75-ft to 80-ft high spoil pile from the second cut was placed on top of the peninsula, directly above where the highwall miner was to begin cutting. Starting from the Northwest side, eleven highwall miner cuts were driven into the peninsula from the new face-up.

In cuts 1 and 2, the highwall miner operator attempted to stay below the Pittsburgh No. 8 Rider seam that is approximately 1-ft to 1.5-ft above the Pittsburgh No. 8 seam main bench. The interburden is very weak and the immediate roof strata fell in upon cutting. In subsequent cuts (3 through 7 and 9 though 10), the interburden was taken to the height of the Pittsburgh No. 8 Rider

seam. Intermittent falls occurred in the Pittsburgh No. 8 Rider seam exposing a weak overlying gray shale unit. Once the gray shale is exposed to the atmosphere, the shale degrades and roof falls are propagated upward as an arch and terminated against a competent brown sandstone bed.

Full penetration of 800 ft was achieved in one cut because roof falls of various magnitudes occurred in each cut. The eleven cuts ranged in depth between 362 and 810 ft. Difficult roof conditions were routinely encountered in each cut within a zone from 400-ft to 600-ft deep. Cut 8 was skipped because of a highwall slide. Cut 10 reached a depth of 810-ft when a roof fall terminated further advance. The highwall miner head had advanced to a depth of 710-ft in cut 11 when a web pillar collapse and roof fall trapped the machine.

The layout of the eleven cuts is shown above in figure 4. The arched roof falls are clearly seen in Photograph 4 that shows the condition of highwall miner cuts 9, 10, and 11. The tail end of the push beams can be seen in cut 11 on the right side of the photograph.
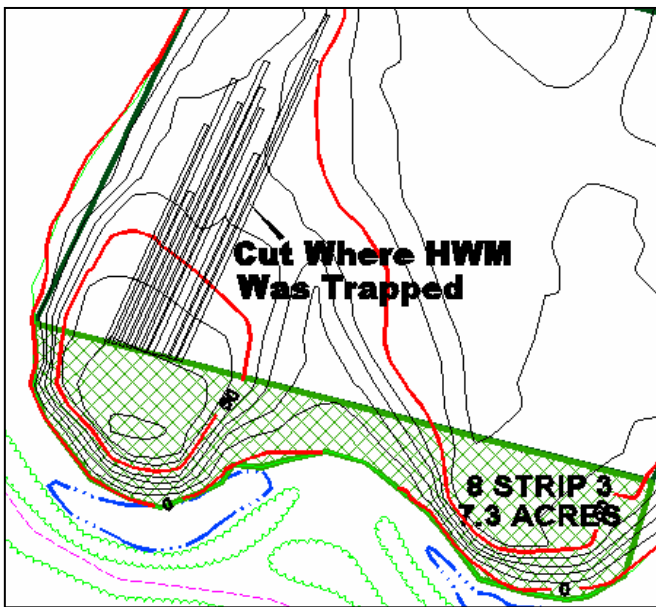

Figure 4. Mine Layout Where HWM was Trapped.

Photograph 4. Roof Falls and Web Collapse

The cause of the web pillar collapse was

❑ back-stacked spoil which increased the web pillar loading not being incorporated into the web pillar design and
❑ an increase in mining height attributable to the roof falls that reduced the width/height ratio of the web pillar and ultimately the web pillar strength

The following recommendations were offered to reduce the likelihood of web pillar failures in subsequent mining.

❑ Mining from a fresh second contour strip cut and scaled face-up with a minimum height of 60 ft,
❑ Leaving sufficient "head" coal from the main bench of the Pittsburgh No. 8 seam to provide support for the overlying interburden, Pittsburgh No. 8 Rider, and immediate roof strata,
❑ Not placing spoil or unconsolidated material on the highwall above areas where mine personnel work or travel,
❑ Avoid mining in areas of low overburden (<60 ft) thickness,
❑ Orienting the highwall miner cuts to start from highest overburden and mine to the lowest overburden, and
❑ Maintaining a standard 6-ft-wide web in the Pittsburgh No. 8 seam, unless highwall mining is or will be present in the Meigs Creek No. 9 seam, at which point the web width should be increased to 8.75 ft.

Case Study C – Closely Spaced Multiple Bench Mining

This case history concerns a contract HWM operator mining the Stockton coal seam from surface contour pits developed by the parent mining company. In the active mining area, the Stockton seam occurs in two splits, ranging between 42-in to 45-in thick, separated by an in-seam parting. The in-seam parting, as seen in the surface mine pits, is of variable thickness and consists of a competent sandy fireclay (Ferm No. 327) and weak shaley coal. Samples of the upper split of the Stockton seam and the sandy fireclay parting were recovered as part of the site inspection. The uniaxial compressive strength of the coal and the tensile strength of the in-seam parting are the basis for web pillar and in-seam parting stability factor calculations.

In-seam parting stability was analyzed using the simply supported beam equation (6) to determine the required in-seam parting thickness so that the in-seam parting will remain stable while the lower split is safely extracted. The simply supported beam equation was selected because it assumes that the beam ends can flex and is appropriate for lower (<300 ft) overburden depths Peng, (*12*). No strength was assigned to the shaley coal unit because of the variability of this stratum. The in-seam parting strength and stability factor calculations are based on the sandy fireclay. The tensile strength and density of the sandy fireclay is 590 lb/in^2 and 169.50 lb/ft^3.

$$\sigma_t = \frac{3\gamma L^2}{4t} \qquad (6)$$

Where: σ_t = tensile strength (lb/in^2),
γ = density (lb/ft^3),
L = opening width (ft), and
t = beam thickness of individual stratum (ft).

Table 4. Simply Supported Beam Stability factors for in-seam parting between upper and lower splits of the Stockton seam

HWM hole number	Parting thickness (feet)	Tensile strength (lb/in²)	Density (lb/ft³)	Simply supported beam safety factor 10.4 ft. Span	Simply support beam calculations		
					Maximum roof span for a S.F. = 1.00	Maximum roof span for a S.F. = 4.00	Maximum roof span for a S.F. = 8.00
Safety Factors for Actual Sandy Fireclay In-Seam Parting Thickness in the Previous Pit							
16	1.35	520	169.50	7.35	28.19	14.09	9.97
17	1.06	520	169.50	5.77	24.98	12.49	8.83
18	1.30	520	169.50	7.08	27.66	13.83	9.78
19	1.20	520	169.50	6.53	26.58	13.29	9.40
20	1.20	520	169.50	6.53	26.58	13.29	9.40
21	1.09	520	169.50	5.93	25.33	12.67	8.96
22	0.84	520	169.50	4.57	22.24	11.12	7.86
23	0.90	520	169.50	4.90	23.02	11.51	8.14
24	0.60	520	169.50	3.27	18.79	9.40	6.64
Safety Factors for a Range of In-Seam Parting Thickness							
	3.00	520	169.50	16.33	42.02	21.01	14.86
	2.50	520	169.50	13.61	38.36	19.18	13.56
	2.00	520	169.50	10.88	34.31	17.16	12.13
	1.75	520	169.50	9.52	32.10	16.05	11.35
	1.50	520	169.50	8.16	29.71	14.86	10.51
	1.25	520	169.50	6.80	27.13	13.56	9.59
	1.00	520	169.50	5.44	24.26	12.13	8.58
	0.75	520	169.50	4.08	21.01	10.51	7.43
	0.50	520	169.50	2.72	17.16	8.58	6.07
	0.25	520	169.50	1.36	12.13	6.07	4.29

In the upper section of table 4, the stability factor for a standard 10.40-foot wide opening is shown along with the maximum allowable highwall miner opening for stability factors of 1.00 (marginal stability), 4.00 (short-term stability), and 8.00 (long-term stability) given the actual in-seam parting thickness, and the tensile strength and density of the parting material.

The lower portion of the table is a sensitivity analysis where the same analysis is conducted for a range of in-seam parting. Obert and Duvall (13) define the stability factor criterion of 4.00 and 8.00 "for members in tension such as bedded roof." Those combinations that do not meet this criterion are marginal and are shown as shaded.

It is clear, based upon Table 4, that the combination of in-seam parting thickness and the 10.4-ft HWM cut width present in the previous pit do not satisfy the long-term stability criterion and in one instance does not meet the short-term requirements. *A minimum sandy fireclay parting thickness of 1.50 ft is necessary to provide long-term stability when mining both splits of the Stockton seam.*

Since the parting lithology is subject to change across the property, the minimum thickness applies to competent shales (Ferm No. 124), sandy shales, or sandstones. Caution was strongly recommended if the parting consists of weak strata, for example, fireclay (Ferm No. 127), laminated sandstone, shale with coal stringers, or fractured strata.

The obvious question from Table 4 is why did the failure occur at HWM cut number 16 when the in-seam parting stability factor was greatest and appears to have exceeded the short-term stability threshold. In conversation with the operator during the site inspection, he believed that the HWM cut orientation deviated so

the openings in the upper and lower split were not columnized. Stability in multiple seam or multiple split mining is dependent upon the transfer of overburden stress through the upper web pillars to the lower web pillars. If a HWM cut is not perfectly aligned with the overlying cut, the overburden stress is transferred onto the in-seam parting. In this situation, failure is likely to occur as the web punches through the in-seam parting. The stability factors in table 4 reflect only the ability of the in-seam parting to remain self-supporting, not its ability to resist vertical loads from an overlying web. When there is any chance that the webs will not be perfectly columnized, a tensile failure analysis similar to one described by Zipf (11) is recommended.

CONCLUSIONS

Highwall mining is an efficient and economic means of surface mining. In contrast to auger mining, HWM requires a thorough knowledge of the strength and physical properties of the immediate roof, coal, and immediate floor strata. Engineering design and surveying the orientation of each HWM cut is necessary to avoid ground control problems. Ground control is critical in HWM because a significant portion of the $5MM to $6MM HWM machine lies beneath the hillside with virtually no easy access to resolve a roof fall or pillar squeeze.

The analytical design equations provided in this paper present the base from which web pillar design should begin. However, as illustrated by the case histories, site conditions and characteristics of the coal seam and immediate roof strata frequently require deviation from the web pillar width calculated using the Mark/Bieniawski equation.

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