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LIUGONG DRESSTA MACHINERY
Sp. z o.o.
ul. Kwiatkowskiego 1
37-450 Stalowa Wola
Poland
sales@dressta.com
tel: +48 501 802 802

Managing Director and Publisher: Trevor Barratt

International Sales:
Gordon Barratt +44 1909 474258 gordon.barratt@tradelinkpub.com
Gunter Schneider +49 2131 511801 info@gsm-international.eu

Graphic Designer: Sarah Beale sarah@g-s-g.co.uk

Published by: Tradelink Publications Ltd.
16 Boscombe Road, Gateford, Worksop, Nottinghamshire S81 7SB

Tel: +44 (0)1777 871007
Fax: +44 (0)1777 872271
E-mail: admin@mqworld.com
Web: www.mqworld.com

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Colombia looks to ramp up mining as economy struggles

Despite the intensifying fight against global warming and climate change, which is supported by some of the world's largest energy companies, Colombia's president Ivan Duque is determined to expand the country's coal mining. The strife-torn Andean nation is South America's largest coal producer and the national government is seeking to bolster output as part of its plans to reactivate the economy after it shrank nearly 7% during 2020 due to the COVID-19 pandemic.

Duque intends to expand Colombia's thermal coal production regardless of the environmental consequences and the government's obligations as a signatory to the 2015 Paris Agreement on Climate Change. A key component of the agreement is that the 196 signatories, including Colombia, will implement greenhouse gas emission-reducing strategies to limit global warming to well below two degrees Celsius. It is recognized that this can only be achieved if thermal coal is removed from the global energy mix because it produces more carbon emissions than any other fossil fuel. U.S. EIA data shows anthracite coal emits 228.6 million pounds of carbon dioxide per million British thermal units produced, whereas bituminous coal pumps out 205.7 pounds when burned.

Those emissions are nearly double the 117 million pounds of carbon dioxide emitted by natural gas, considered to be the cleanest of the fossil fuels, and around 40% greater than either gasoline or diesel. Bogota intends to expand coal production despite 94% of Colombia's proven coal reserves, which amount to more than 5 billion tons, according to the U.S. Geological Survey being comprised of anthracite and bituminous coal, the most polluting types of fossil fuel. This is because coal generates 85% of mining royalties, making it a key driver of government revenue, and is Colombia's second-largest export, after crude oil, accounting for 11% of export earnings.

The desperation of the Duque administration to kickstart economic growth, regardless of the cost or its international obligations, is underscored by the ongoing weakness of Colombia's economy. Despite lifting the strict lockdown instituted across Colombia in March 2020, to mitigate the spread of the pandemic and implementing a series of measures to promote growth, first-quarter 2021 GDP contracted by (Spanish) a worrying 9% compared to the previous quarter. Unemployment remains stubbornly high with the government statistics agency DANE reporting that nearly 16% (Spanish) of Colombians were unemployed at the end of May 2021. Those shocking numbers can be attributed to the impact of a third viral wave on the economy which forced many of Colombia's major cities into partial lockdowns. That makes it difficult to see the Andean country's economy expanding by 6.5% as its central bank predicts. Even the more modest 5% 2021 GDP growth forecast by the IMF appears difficult to achieve.

The nationwide anti-government protests sparked by Duque's inept attempt to hike taxes at the end of April 2021 sharply impacted the economy. Heavy-handed repression by authorities, with independent thinktank Indepaz reporting 44 protestors were killed by police and security forces, caused the protests to explode. Not only are they continuing into their third month, but anti-government protestors established roadblocks that prevented the transportation of food, water, medicines and other crucial supplies in Colombia. Those roadblocks were so significant by mid-May 2021 that Colombian onshore petroleum producers, including national oil company Ecopetrol, were forced to shut-in production.

This sharply impacted Colombia's economically crucial oil output, which is responsible for 3% of GDP, nearly a third of exports by value, and almost a fifth of fiscal income. According to data from Colombia's petroleum regulator, the National Hydrocarbon Agency (ANH – Spanish initials) petroleum output (Spanish) fell to a low of 650,884 barrels daily by 25 May 2021 and had only recovered to 696,672 barrels daily on 24 June 2021. Such a sharp decline in oil production will impact Colombia's economic recovery and Bogota's fiscal income. While most roadblocks have been lifted, Colombia's economy is struggling to reactivate because of heightened political turmoil as well as

Photo by Kelly Lacy from Pexels

insecurity along with limited protests continuing in some cities.

That is only further fueling the Duque administration's desperation to boost economic growth and increase fiscal revenue, with some analysts estimating Bogota's budget deficit could blow out to more than 9% of GDP this year. Those events further emphasize the Duque administration's desperation to reactivate the economy and spark growth by any means available, explain why boosting coal output is perceived to be an important economic lever. Bogota is making good on its plans regardless of the global fight against climate change and Colombia's obligations under the Paris Agreement.

The energy ministry reported that first quarter of 2021 coal output soared by a whopping 52% compared to the previous quarter to 13.9 million tons, although that was 28% less than the 19.4 million tons produced a year earlier. Colombia's energy minister Diego Mesa foresees increased production because of greater coal demand from China and India. This is despite globally diversified miner Glencore, through its Colombian subsidiary Prodeco, seeking to hand back the licenses for the open pit Calenturitas and La Jaguar coal mines in the department of Cesar.

Glencore determined that after mothballing operations at the mines because of the pandemic it was uneconomic to restart the mines. Initially, the miner sought to keep Calenturitas and La Jaguar on care and maintenance, a plan initially vetoed by Colombia's mining regulator the National Mining Agency (ANM – Spanish initials). So far, the regulator has rejected Glencore's requests to hand in those mining contracts, although a final decision is expected by mid-July 2021.

Mesa expects Asian mining companies to consider acquiring the licenses and investing the capital required to recommence operations at the affected coal mines after the matter is settled with Glencore. Not surprisingly other major miners are seeking to reduce their carbon footprint by divesting their coal mining assets. As part of that strategy global mining giant BHP and Anglo American each agreed to sell their 33.3% interest in Cerrejon, Colombia's largest coal mine, to Glencore for a total of $588 million. This will make Glencore sole owner of the controversial Cerrejon mine. The operation suffered a three-month work stoppage from the end of August 2020 until the start of December, sharply impacting Colombia's coal output. Earlier this year, the OECD committed to an investigation into human rights abuses and environmental damage at the Cerrejon mine. The ongoing turmoil and uncertainty surrounding Cerrejon's operations indicate that further stoppages could occur impacting Colombia's coal production.

The desperation of the Duque administration to reactivate Colombia's economy and promote growth is easy to understand considering the harsh financial impact of the pandemic, the recent protests, and a ballooning government budget deficit. Nonetheless, by furiously expanding coal production Bogota is not only investing in what is fast becoming a stranded asset, which could eventually become a costly liability, but it is working against the Paris Agreement and the global fight to prevent climate change. Any expansion in coal production will likely only deliver a short-term benefit with many countries, including those Duque's government has pinned their hopes on China and India, focused on phasing it out of their energy mix. The resources dedicated to expanding Colombia's coal production could be better used to rebuild the crisis-driven country's hydrocarbon sector which was sharply impacted by the 2020 oil price collapse, the COVID-19 pandemic, and turmoil triggered by recent anti-government protests.

Komatsu introduces replaceable trapping shoe wear inserts for Joy shearers

Engineered to help lower operating costs and can promote reduced worker exposure

Joy shearer owners will appreciate the fast replacement times and effort, low operating costs and the advancement of the journey to zero harm.

Key to the shoe inserts is their slip-in design. They efficiently slide in and out from the end of the shoe, so they can be replaced anywhere on the longwall. Plus, they're only 32 kg (70 lbs.), compared to up to 454 kg (1,000 lbs.) for a full shoe – helping to reduce effort equipment worker exposure during replacement.

"This is a major breakthrough for our longwall customers," said Shawn Franklin, product manager, longwall shearers. "Using these inserts can help them reduce downtime, maintenance and replacement costs."

Operations can also be confident in the quality and performance of the shoes, as they are covered under a Komatsu warranty. Shoes rebuilt or rewelded by third parties can shorten rack bar life or affect the fatigue life of the parent shoe.

Automation in US longwall coal mining: A state-of-the-art review

This paper reviews the development of US longwall mining from an unknown to became the world standard in the past five decades with emphasis on automation. Large scale longwall face equipment were imported from Germany and United Kingdom to increase production in the 1970s and great effort was made to improve them to suit US conditions, rather than domestic market. Automation began with the development of electro-hydraulic shields in 1984 and continue to present. Introduction of first generation semi-automated longwall system occurred in 1995 and step-to-step improvement continues to this day following the development of sensor technology and internet of things (IOT). Since then an emphasis on new development has been concentrated on the improvement of equipment reliability, miner's health and safety as well as production, including dust control techniques, proximity sensor, anti-collision and remote control.

Automation is classified into two categories: automation of individual face equipment and automation of longwall system. The automation development of longwall system is divided into three stages: shearer-initiated-shield-advance (SISA), semi-automated longwall system, and remote control shearer.

INTRODUCTION

US coal industry employs two mining methods: room and pillar mining and longwall mining. Manual room and pillar mining method was developed and used ever since coal mining began in the 18th century. Mechanized room and pillar mining began in the late 1920s and reached full mechanization in the late 1940s.

Modern mechanized longwall mining however began in the early 1950s[1]. It was imported from Germany for low coal seams in southern West Virginia. The system consisted of hook plow, chain conveyor and frame support. Due to low productivity and safety issues, it was not accepted by the industry in general. The first shearing machine in the United States was introduced at Kaiser Steel Corporation's Sunnyside No. 3 Mine in Utah in 1961, and later in mines in the East.

In 1972, the Saudi's Oil Embargo that created the ''Energy Crisis" prompted the Nixon Administration to develop the Energy Independence Policy, one of the key components of which is to increase coal production, because coal was the cheapest energy resources and US has the largest coal reserves in the world.

Since room and pillar was the dominant mining method and very few mines were able to produce more than one million tons annually, mainly due to the fact that room

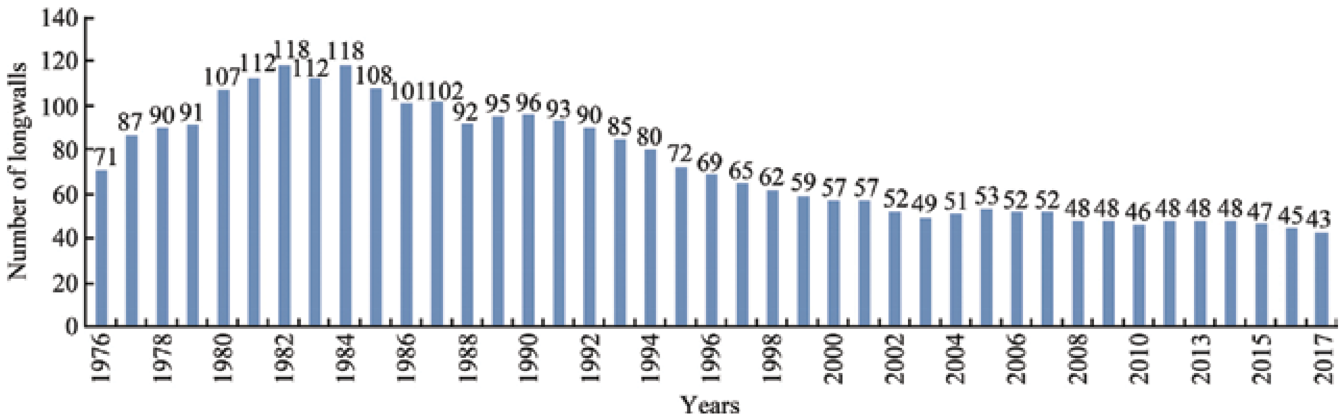


Figure 1: Annual change in number of US longwall panels (or mines) since 1976.

and pillar mining method using the continuous miner was not a continuous mining system. Conversely, the modern longwall mining as practiced in Germany and United Kingdom was truly a continuous mining system with great potential to increase production quickly to meet the nation's demand for energy.

Consequently, German and UK longwall mining equipment were introduced and adopted quickly by US coal industry during and soon after the Energy Crisis. The number of longwall mines (note due to large capital required one coal mine could only afford one longwall panel or one set of longwall equipment) increased rapidly in the 1970s, reaching a peak of 118 in 1982-1984 (**Figure 1**). Since all coal mines then were operating with room and pillar mining system that is flexible, as opposed to rigid longwall panel layout, it was easy to convert room and pillar to longwall mine. Since longwall mining consists of two parts: panel development and longwall mining (advance or retreat), in order to shorten the training period of miners, management decided to keep the room and pillar mining method for entry (gate roads) development, only adopting the longwall face equipment for mining. However, in order to meet the Coal Mine Health and Safety Act of 1969, multiple escapeways (a fresh air and a return air), plus neutral belt entry must be maintained while mining coal. Furthermore, continuous miners are not designed to cut rock, so all entries are driven in-seam in rectangular shape. As a result, US longwall mining at the outset differed from those of European in panel development in that it is multiple entry development (normally more than 3 but 2 in coal bump-prone western coalfield) with in-seam rectangular entry, which, plus two-stage entry support plan, turns out to be the best choice enabling not only rapid entry development, but also fast retreat mining, leading to high efficiency high production longwall mining system soon after its adoption.

In addition, it was also determined that in advancing longwall mining method, entries along the outby gob are not only difficult to maintain, but also unsafe for travel. Therefore, retreat mining with the gob inby is uniquely employed, not the advancing method.

Therefore, US longwall mining employed multiple entry development using continuous mining system for panel development and mining in retreat vs European's single arched entry development and advancing system. Only the face equipment was adopted outright at the outset.

It must be emphasised that all major new technologies in longwall equipment in the past 5 decades were developed by German and UK (for AFC and shield) and US (for shearer) original equipment manufacturers (OEM). In the 1970s and 1980s, there were numerous German and UK manufacturers. They were gradually merged in the 1980s and 1990s, and eventually consolidated into two large manufacturers in the 2010s, i.e., Caterpillar (USA) and Komatsu Mining (Japan). Both Caterpillar and Komatsu Mining make all face equipment and automation devices with Caterpillar in Germany and Komatsu Mining in UK except the shearer in the USA. Frequently they differ in design, but with similar functional purpose. So in this paper no attempt is to distinguish their products, only general description.

This paper describes the evolution of US longwall mining technology from an unknown to a world standard with emphasis on automation aspect of the technology.

EUROPEAN LONGWALL TO US LONGWALL SYSTEM

In the 1970s, there were many manufacturers in Germany and United Kingdom, specializing in various face equipment (shearer, AFC or armored flexible face conveyor, and self-advancing hydraulic powered support) and each type of face equipment had multiple manufacturers manufacturing for different types and models, making it confusing for selection. For instance, for shearer, there were single and double ranging drum; for the powered support, there were frame (4- or 6-leg), chock (4- or 6-leg), shield (caliper, lemniscate, 2- or 4-leg), and chock shield (vertical or V or cross legs); and for AFC, single center, twin in-board or twin center (TIB), twin out-board (TOB), and triple chain strands.

With so many different types of equipment and mixed application results in the early 1970s, it was not until the successful introduction and application of shield in a northern WV mine in 1975, demonstrating that European longwall system can be a safe and productive system that longwall mining using European longwall technology gained acceptance in US coal industry.

Although all major equipment were made in Germany and United Kingdom, their major market and clients were the US coal industry due to the rapidly expanding longwall mining (**Figure 1**). Consequently improvements in equipment were mainly at the request of US operators gained from their operational experience, rather than by and for their domestic market. It was true then and continues to present. Major improvement during the 1970s and 1980s included:

1. Shearer: from DC to AC motors, from mid motor to multi motors, from integrated to modular design, from single to double ranging drum, continuously increase in power and reliability.
2. Powered support: modifications in structure configuration and welding technology, determination of 2-leg shield as the standard, development of shield testing protocols, additions of safety features, increasing supporting capacity and reliability.
3. AFC: improvement of manufacturing process and wear resistance, increasing chain strength and haulage capacity, longer life and reliability.

Intensive research combined with mine operators' operational experience in the 1970s and 1980s resulted in the optimization and finally adoption of the following face equipment as standard for US longwall mining in the mid to late 1980s: Shearer – doubled-ended ranging drum (DERD), AFC – twin-centered chain strand with integrated head drive, cross frame, Powered support – 2-leg lemniscate shield.

By late 1980s, US longwall annual production both in terms of annual tons per longwall panel or tons per man-hour (productivity) had far exceeded those in the Europe and attracted numerous visitors worldwide wanting to find out why and how? Most of them attributed it to good geological conditions. What they did not realize was that the superior panel layout and US-developed auxiliary devices enabled the utilization of the full potential of reliable heavy face equipment then made mostly by European manufacturers.

THE ROLE OF GROUND CONTROL IN US LONGWALL MINING

Since US coal industry was a newcomer in longwall mining in the 1970s and early 1980s, there had been quite a few failures in the start-up of longwall mining, some of which resulted in closure of the mines permanently. The major reasons could be attributed to the neglect and/or ignorance of ground control issues. First and foremost was what kind of roof and floor strata were economically suitable for the face equipment then available. Those cases included for example: (1) the powered supports were not properly designed or selected both in capacity and strength of structural components as well as welds that caused numerous production delays; (2) the roof was so weak that fell off immediately before the powered support could be advanced to support it at the face; (3) the longwall face was oriented in such a way that it collapsed when the face reached a large joint set parallel to the face; (4) the immediate roof did not cave following the face advance from the set up room and overhanged for more than 100 m and finally crushed the powered supports at the face; (5) the chain pillars in the tailgate side were not properly sized such that a few panels into the district, the tailgate was badly deformed such that the AFC tail drive could not freely advance; (6) the headgate and tailgate T-junction areas were not properly supported such that either massive roof falls or excessive roof convergence or floor heave occurred hindering the movement of head and tail drives; (7) sandstone channels were found cutting into the coal seam inside the coal panel block, causing large production loss; (8) will the capacity of existing powered support be sufficient when the panel width is expanded 30 m, 61 m, 91 m.

The general experience had been that if the longwall mining project was going to succeed and continue, the first panel must be successful, i.e., no or little ground control problems that caused excessive production and safety delays.

The initiation and continuation of the annual International Conference on Ground Control in Mining (ICGCM) starting in 1981 provides an excellent forum whereby all professionals interested in ground control, in particular relating to longwall mining, gather to exchange information relating to ground control techniques in mining. They included university professors, government researchers and regulators, equipment manufacturers, mining consultants and services personnel, and above all, mine operators, not only from all parts of US, but also from all major coal producing countries. Any new techniques and products presented at the conferences were quickly adopted by the mine operators and if not successful, it was abandoned immediately. Otherwise, it would be accepted and spread industry wide soon or later. It was mainly through this process, a series of ground control problems associated with the introduction of European longwall technology were resolved one by one in a short period of time. The result was that ground control became the first part of mine design project whenever a new longwall mine or a new panel in an existing mine was planned, thereby setting up ground control as the leading factor in the success or failure of a longwall mining project.

Major ground control factors in longwall mining design include:

1. Optimum panel width (or face length) for the geological condition.
2. Type and yield capacity of hydraulic powered supports.
3. Rows and size of chain pillars in development section and number of entries.
4. Entry (gateroad) supports (primary and supplementary) including T-junctions.
5. Surface subsidence prediction and protection of structures and water bodies, both surface and underground.

DEVELOPMENT OF AUTOMATION IN US LONGWALL MINING

Before we start talking about automation in longwall mining, the word ‘‘automation” needs to be clarified.

It is reminded that the three face machines in modern longwall mining are mechanically linked. The shearer cut coal of fixed width (web) riding on the pans, and following the configuration of AFC as traveling track, while the relay bar of each shield at the center of the base is connected to the pan in front of it using the relay bar to pull itself and push the pan forward for a web distance. Each shield needs one or more shieldmen to initiate and complete the shield advance cycle, and push the panline. The shearer needs two operators, one for the lead and the other for the trailing drum. Thus in longwall mining the shearer operators are to steer the drum to cut at the coal-roof rock interface for the lead drum and coal-floor rock interface for the trailing drum (this is called "horizon control"), while the shieldmen are to advance the shield and push the pans for a web distance following the shearer's cutting. Accordingly, the AFC does not need an operator. Consequently, "automation" in longwall mining centers on equipping the shearer and shields with "intelligent' to operate by themselves doing what the human operators do. Note there is only one shearer while there are numerous shields, the motion of which fortunately is linear advancing and all shields are the same. The shearer drums however require the identification of, and follow the ever changing coal/rock interface, which obviously is much harder to automate.

Therefore, "full automation longwall mining" or "intelligent longwall mining" in this paper refers to manless mining at the face. In other words, normal coal production operations are conducted without any crew members at the face, nor at any time the production operations require crew's intervention to correct any problem. So it is the highest level of automation and based on the current technology available, it is not attainable. So "automation" in this paper refers to different levels or depth of "full automation". The year the automation technique or device was introduced was stated and improvement to rising maturity and/or power capacity wherever possible. The development chronology of automation of longwall face equipment and systems in the United States is shown in **Figure 2**.

Since longwall mining consists of three major machines, shearer, AFC, and shields, there are two types of "automation", "automation" for each machine and "automation" for the longwall system (i.e. the three machines operate automatically for coal production). Since each machine has its level of automation developed at any specific time frame, the precise level of automation for the whole system at any instant is difficult to specify. Besides, there remains many unknown events to be defined for automation for each machine and longwall system. Consequently, it is important to recognize that the word "automation" is qualitative, not quantitative in this paper.

It must be strongly emphasised that underground operational experience has demonstrated in the past 4 decades that it takes time for a new automation device/technique to reach maturity, i.e., the elapse time between introduction and becoming a reliable one can be very

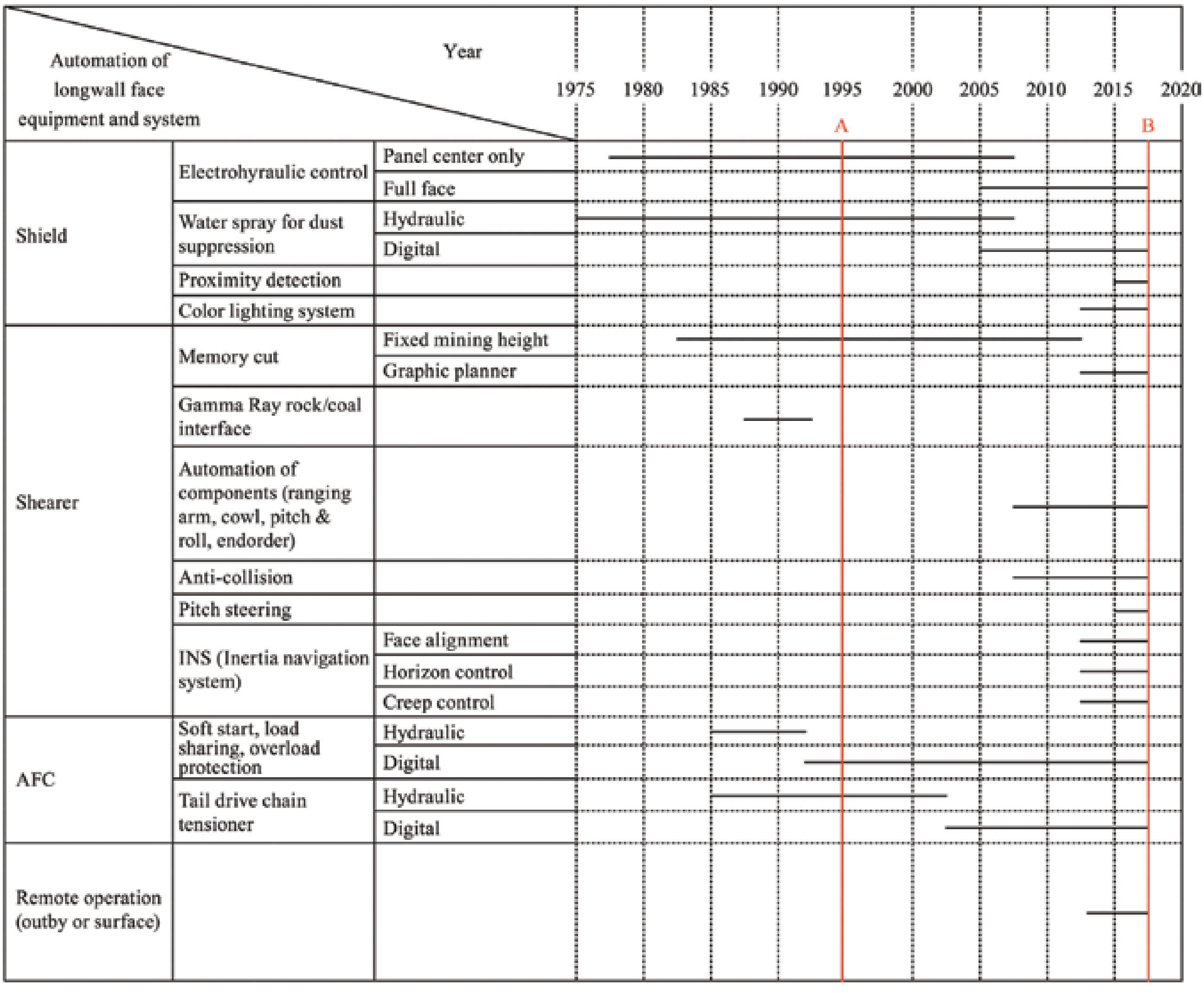


Figure 2: Development Chronology of Automation of Longwall Face Equipment and system. Automation of longwall system can be found by drawing a vertical line from the year of interest. For example, the 1995 Cumberland Mine system (A line) would have shield with electrohydraulic and hydraulic water spray, shearer with memory cut and AFC with hydraulic soft start and tail drive tensioner. B line represents state-of-the-art automation longwall system.

long due to complicated geological and mining conditions underground. Therefore, introduction of an automation device/technology does not necessarily mean it will work reliably for safe production operation, rather it requires continuing refinements and upgrades.

Automation of three face machines and automated plowing system, 1970s–1990s

Automation of shields

A shield's cyclic movement is linear and simple, i.e., lower a few inches, advance a web distance and raise a few inches to set, followed by extending the relay bar to push the pan forward for a web distance. However, it has multiple units normally 150-270 for today's panel width and thus requires sequencing. Therefore, shield is easy and most beneficial to automate among the three face machines.

Automation of shield is to add a pilot valve and an electromagnetic solenoid valve to the hydraulic valve such that with a light touch of a button, the three separate steps of shield advancing cycle is performed automatically in sequence, rather than a push button for each of the cycle step when only the manual hydraulic valve is available. This is electrohydraulic shield which is much faster and safer than only the hydraulic valve. Since there are many units of shields in a face, the shield control unit (SCU) for every shield is in fact a computer.

Electrohydraulic shields were first installed in 1984 at the Monongalia County mine about 18 miles west of Morgantown, WV. It was a landmark event in that it was the first industry ever to employ a computer in production operation because IBM's personal computer was barely in the market then. The German made one was a self-contained computer that can communicate with other shields directly, while the UK made one was not and required a headgate computer to communicate among shields. Electrohydraulic shields can be operated individually or in batch of 3-8 units in the beginning for safety reason and later expanded to the full face.

When it was first introduced in 1984, electrohydraulic control shields had many problems, the major ones of which were not resistant to underground environments (moisture, dirt, and dust), and cable faults. It was damaged quickly and required replacement frequently. At a cost of US$14000 per unit, it was not cost effective. However, the OEM's worked hard to solve the problems. By around 1990, it had become very reliable and gained industry's full acceptance.

A fully operational batch control electrohydraulic system that requires only two push-buttons enabled a faster shearer cutting, thereby higher and safer production. Specifically, (1) Shield cycle time reduced from greater than 40 sec (manual) to 12-20 sec. (2) Positive setting ensures all shields were properly set and uniform loading across the face. (3) Reduction in number of shield men.

Automation of shearer

Shearer automation began with radio remote control in 1978 (analogue) and then 1984 (digital)[3].

The "Memory Cut" technology was introduced in 1986 in which in the training run, the data on drum elevation and ranging arm inclination as well as shearer location were recorded. Those data were recalled to run the subsequent shearer's cuts. It allowed the face cutting profile to be displayed. This level of automation required ranging arm tilt sensor and shearer location sensor. An on-board central computer is also needed to process the data. The most popular "Memory Cut" system only monitor the lead drum position and the trailing floor drum was fixed by running the shearer with a constant mining height, resulting in more uniform floor horizon. The technology then was not reliable and required frequent retraining run. In the early to mid-1990s, the OEMs claimed the developed system then had more than 400 operational cutting modes available, in addition to "Memory Cut" mode.

A gamma-ray coal thickness sensor was developed in late 1989 to early 1990s[4,5] and mounted on the shearer for shearer's cutting horizon control. For this automation system the shearer was equipped with gamma-ray coal thickness sensor for coal/rock interface detection through coal thickness measurement, roll and pitch sensors for the attitude of shearer body, and tilt meters for ranging arms height and data were sampled at 0.6 m. interval. All four sensors and meter were externally mounted and bulky. Unfortunately, the gamma ray coal thickness sensor only works in shale roof. So the system never gained industry-wide acceptance and never took off.

The next level of shearer automation was the concurrent development of more sophisticated sensors to replace the old ones for more precise control of shearer operations and these include the pitch and roll sensors for shearer's body, tilt sensors for the ranging arm, and rotary encoder for shearer's location that samples every 60 ms. Other accomplishments include automation of gate end operation and individual components such as ranging arm, cowl, lump breaker, haulage and water. Software has been developed to control cutting, cowl, haulage, separate ranging arm control, and other components independently. As a result, the shearer can run automatically by itself all day long if it is not out of the seam. However, the shearer still requires one or two operators to operate the training run and constantly evaluate when a new training run is required[6].

The latest development is pitch steering in which the floor drum is positioned such that it is steered to a specific pitch angle on the following pan push in order to smooth out the floor steps[6].

From the late 1990s to 2013, Australia's CSIRO developed successfully the inertia navigation system (INS) for face alignment, shear's horizon control, and AFC creep control: In face alignment, the desired face alignment is maintained by accurately measuring the 3D position of the shearer using the latest gyroscope and motion sensor technology[7,8]. This information was used to control the movement of shields. In horizon control, thermal infrared camera is used to detect the heating of marker bands in the coal seam using it for reference for horizon control. In AFC creep control a sensor based on 2D laser scanner is installed on the AFC structure at the headgate T-junction to actively measure the closure distance per cutting cycle. INS has been widely used in Australia coal industry since 2014, but it was only recently introduced into US longwall mining and only one mine has adopted so far in early.

Automation of AFC

The AFC drive system requires static and dynamic power reserves, start-up against high loads and handling of slack chain. In the 1980s, due to smaller chain size and chain strength, sudden overload of AFC often occurred that cause chain breakage and/or drive motor burnout. So the soft start technology was developed for AFC drives in 1992 to handle heavy-load start up, to guarantee load sharing among the drive motors and avoid slack chain at the head drive frame. It will stop when a sudden overload occurs to relief the pressure and re-clutch quickly. The variable frequency drive (VFD) will slow down the conveyor speed when encountering overloads.

Maintaining a proper tension on the chain strand is very important to prolong the chain life. Since the head drive frame is used to transfer the produced raw coal to the stage loader, the tail drive frame must control the chain tension. A tensionable tail drive using a double-acting hydraulic cylinder was introduced in early 1990s for manually and later automatically adjusting the tension as needed.

A tail drive with smart chain tension self-adjusting system was installed in the 2010s. The system has a load sensor on the flight bar for measuring the chain tension in real time.

Automated plowing system

The fully automated plow face was installed at US Steel Mining Co.'s Mine 50 near Pineville, WV in 1989 and continues operation until 2018, this was the only plow face in the US

The mining height is 1.2-1.42 m, not suitable for the shearer. The record production was 32402 t/d with advance rate of 47.8 m/d in April 2014. However, due to low coal, production seldom exceeding 2.5 million clean tons per year, and due to smaller plowing capacity, it cannot cut hard partings including fault materials causing operational difficulties, although the latest model has 2 800 KW installed power and cutting speed up to 3.6 m/s.

There are only two miners at the face, one each at the head and tail ends.

Although fully automated, plow longwall never caught on in the US and US has little contribution in the development of automated plow longwall mining, and due to small market demand, the plow system at Mine 50 has few improvements over the last 3 decades, except larger plow power and speed, and higher reliability.

Automation of longwall system

According to the chronological development of automation of each of the three machines, "automation" of the US longwall system can be roughly divided into the following three stages with rising maturity of automation longwall system: (1) shearerinitiated-shield advance (SISA); (2) semi-automated longwall face; and (3) remote control of shearer.

Shearer-initiated-shield-advance (SISA), mid 1980s – present

The next step is to establish communication between automated shearer and automated shields to establish an automation longwall system. In this first step automation longwall system, as the shearer passes by, the shield behind it will automatically advances one by one, followed by the panline being push also one by one automatically. The system requires a shearer location sensor that determines the shearer's location so that the headgate computer know where the shearer is cutting and issues commands to certain shield behind it to begin advancing cycle.

An infrared emitter was developed in mid 1980s (**Figure 3**) as shearer location sensor. It is mounted on the main body of the shearer and as it passes a shield the infrared radiates toward and covers three adjacent shields that in turn relay the shearer's position and direction of travel to the headgate computer, which then commands the next shield (2-3 shield behind the trailing drum) to initiate advancing cycle. This longwall system automation requires: (1) a headgate computer, (2) a shearer mounted with an infrared emitter, (3) shields with fully operational electrohydraulic control system, (4) shearer can perform "memory cut" with fixed height full face, and (5) the face shift crew may reduce to 4 or less. Note that the first generation infrared sensor's radiation covered 3 shield wide or 4.5 m which was not as precise as in identifying the shearer's location as today's single shield coverage.

It was soon discovered that infrared sensor for shearer location had several shortcomings: In areas where the heavy dust laden air and water spray mists exist, its signal would confuse the receiver leading to missing shearer location. Also since the location is represented by shield number which was 1.5 m and now 1.75 m or 2 m wide, the shearer's location so determined was rough such that after a few web cuts, the location at both head and tail ends may misrepresent. Therefore, a serial link that count the number of the teeth of shearer's driving sprocket, much like the automobile's odometer counter and much more accurate in location identification than the infrared system, was developed and installed in the early 1990s. Today, all shearers are equipped with both systems (infrared and serial) to complement each other. In the 2000s, rotary encoder monitoring the haulage motor gear train was developed and has since been used for shearer location with an accuracy of a few mm.

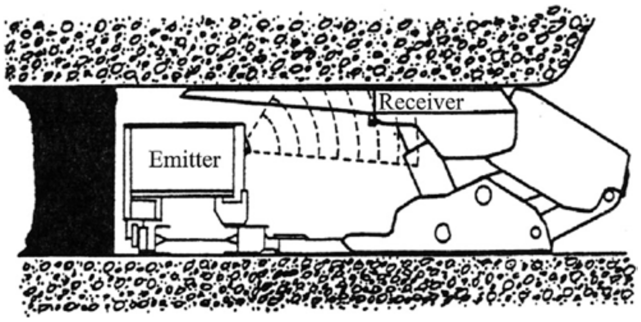


Figure 3: Principle of infrared sensor for shearer's location[9].

The system worked well for the whole panel width when the shearer employed the uni-direction cutting method. When bidirection cutting was used, the system worked well within the panel, but more often encountered difficulties in carrying out the face end operation that involves two double-backs and wedge cuts in automation mode.

The problems with face-end operation remained and then was solved in the late 2000s. Since then SISA system is fully operational full face wide.

Semi-automated longwall face, 2000s to present

This is the system most commonly used in the US longwall mines. It consists of the following equipment:

1. SISA system.
2. Shearer with ASA (advanced shearer automation) and DCM (dynamic chain control management) systems with (currently one face only) or without CSIRO's INS system: ASApitch & roll sensors, ranging arm inclinometer, serial or encoder for precise shearer location; DCM- chain tension, load on AFC; and INS (Inertia Navigation System) by Australian's CSIRO.

The shearer is operated in memory cut mode: the first cut is operated by the operator. The recorded data (drum heights, pitch and roll sensors at each shear location) are stored in the computer. Those data are recalled to run the 2nd and subsequent cuts. It is much more accurate and reliable than the initial system developed in the 1980s and 1990s. The cut profile can be adjusted anytime by the shearer operator.

The system still requires 4-8 crew members at the face depending on mining practice and mining and geological conditions: 1-2 shearer operators, 1-3 shield men, 1 electrician-mechanics, 1 utility man, and a foreman.

Remote control of shearer, 2012 – present

It began in 2012 in a coal mine in New Mexico in order to increase coal miner's safety and production[10], a shearer operator was moved to a control point outby the headgate T-junction and remotely controlled the shearer's cutting from head to tail, while the other operator controlled the shearer at the face from tail-to-head cutting.

The respirable dust standard was reduced from 2 mg/m^3 to 1.5 mg/m^3 effective August 2016. In order to meet the standard, a WV coal mine adopted similar cutting method. In these two remote control systems[11]:

1. 6-8 video cameras are mounted on the shearer at strategic locations facing different directions such that all essential views of the face in real time are visible for the remote control shearer operator. 0-4 cameras are at AFC head and tail drives.
2. A longwall control panel is installed outby at T-junction or in the mule train where the real time numbers of all monitored parameters for all sensors are available in related screens. Any time the operator finds deviation, the maintenance guys are notified immediately for checkup. A surface control center with similar function is also installed on the surface.

With complicated geological and mining conditions, so many unexpected events can occur. For a fully automated face, all of those events must be able to detect in advance and corrective actions implemented timely. It requires sophiscated sensors of proper type and correctly mounted. In the last two decades great strides have been made toward that direction. We are closer but still a long way to full automation!

At this moment, it seems remote control of face equipment, in particular, the shearer is the correct step-stone toward full automation as a remedial measure for lack of various reliable sensor technologies, i.e., move the face and equipment operators to a safe place away from the face and operate the equipment via video cameras. The problem is to find a video camera that can defy the dirty misty air around the shearer all times and transmit clear pictures to the control center.

The health and safety and production records for the two operational remote control longwall faces have been excellent, except the number of face crew did not reduce much. Perhaps over time it will see the benefits of the manpower reduction once they gain confidence in the system.

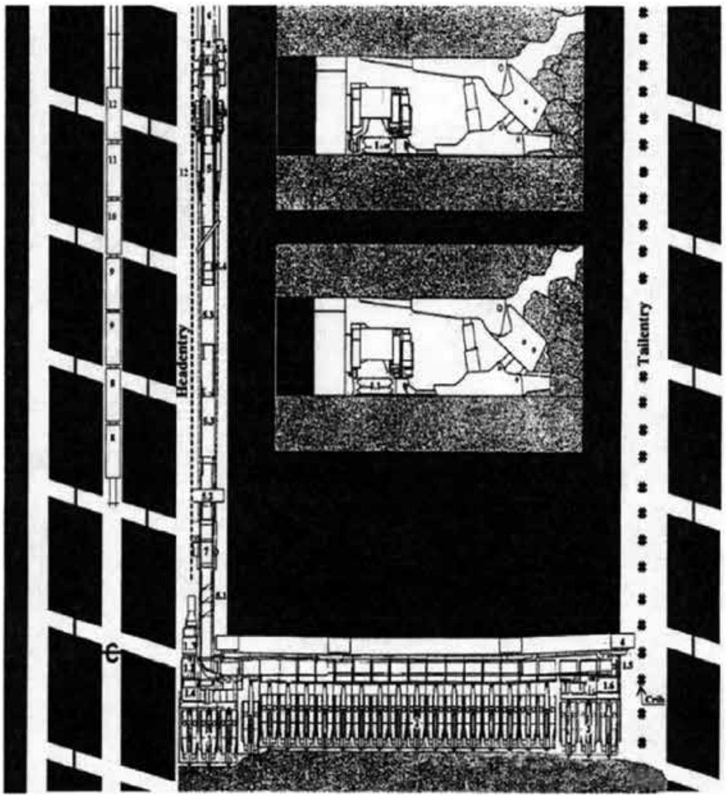
The number of face crew remains the same, except one shearer operator is at the remote control center away from the face.

THE FIRST MODERN LONGWALL MINING SYSTEM STANDARD

In 1993, the senior author designed the first semi-automated longwall for US Steel Mining Company's Cumberland Mine, Kirby, PA (**Figures 4-6**). The system began production in 1995. It consisted of: (1) electrohydraulically controlled shields; (2) the shearer with infrared emitter and serial link to enable shearer initiated shield advance (SISA), and with Memory Cut algorithm; (3) AFC with soft start and hydraulic chain tensioner. The mining height was 2.4 m. A similar system was also designed by the senior author for Shell Oil Company's Moranbah North No. 1 Mine, Moranbah, QLD, Australia in 1995. It began production in 1997. The mining height was 5.5 m.

The 1995 Cumberland Mine system routinely produced 4000-6000 clean tons per shift. For the 300-m wide longwall, shift crew = 8: shearer operator 2, shield men 3, mechanics 1, headgate 1, and foreman 1. It has the following special features[2]:

1. A crawler-mounted tail piece for the panel belt conveyor is normally employed so that it can be trammed easily and rapidly in order to keep pace with the fast advancing longwall face. The belt structure 6.1-9.1 m immediately outby the tailpiece is dismantled in advance.
2. The drive head of the stage loader, where it dumps coal into the panel belt conveyor, rides on the belt tail pieces on a dolly that can travel freely for a distance of 3.7-4.6 m. So as a new cut is made, the gate-end shields advance and push the cross frame and the whole stage loader a web distance ahead.
3. There are four to five flexible short pans, 700 mm long, between the cross frame and crusher. This arrangement



1 Face conveyor
1.1 Face convyor
1.2 Head-end drive cross-frame
1.3 Gearbox
1.4 Gearbox
1.5 Tail-end drive
1.6 Gearbox 3
2 Two-leg or four-leg shields with positive baselifting
3 Face-end shields
4 Shearer
5 Stage loader
5.1 Flex-pans and emergency pull-cord
5.2 Cable duct
5.3 Electric controls (optional)
5.4 Inspection-pan
5.5 Drive frame
5.6 Gearbox
6 Belt conveyor tail-piece
6.1 Stage loader advancing and belt tail anchoring unit
7 Crusher
8 Power center/control panel
9 Hydraulic pump and emulsion tank stations
10 Rock dust car
11 Water pumps
12 Supply/utility/parts cars
13 Monorail

Figure 4: Typical modern longwall panel layout[2].

Figure 5: Face equipment setup at the set up room, ready for panel startup mining at Cumberland Mine 1995.

can absorb the bending and creeping caused by the panline that may be pushed left and right too far out of alignment between the panline and stage loader thereby reducing damage to the coal transfer system.

4. The gate-end shields in the headgate T-junction not only protect the drive head cross frame, which is mounted on a skid, they are also used to advance the cross frame and stage loader forward cut by cut.
5. The AFC tail drive is mounted on a skid and is flush with the rib of the panel coal block without sidestepping into the tailgate. It is also easily advanced by the three gate-end shields cut by cut. This arrangement does not require dismantling the wood cribs (or other types of standing supports) in the tailgate, which are normally installed as a secondary support to cope with the incoming front and side abutment pressures, and insures rapid advance of the longwall.
6. A monorail, 305-610 m long is hung from the roof either on the pillar or panel block side of the belt conveyor in the headgate. Power supply cables and hoses between the power train and stage loader are hung and guided by dollies/trolleys. These cables and hoses stretch and coil like an accordion easily following the longwall face advance. The electrical cables, hoses, etc., are looped between bumper trolleys when the monorail is at its full length, those cables and hoses are fully stretched. As the face retreats, the push/ pull unit mounted at the inby end pushes them gradually into loops. When the cables/ hoses between all bumper trolleys are fully looped, the monorail system is ready to be moved outby. At this time, the winch mounted at the outby end is used to move and reset the monorail system.
7. The power train, which consists of power centers, hydraulic supply pump system, rock duster, water pumps, tool and supply cars, etc., is normally on a track or skids in the second entry outby the face and can easily be moved with the moving longwall face.

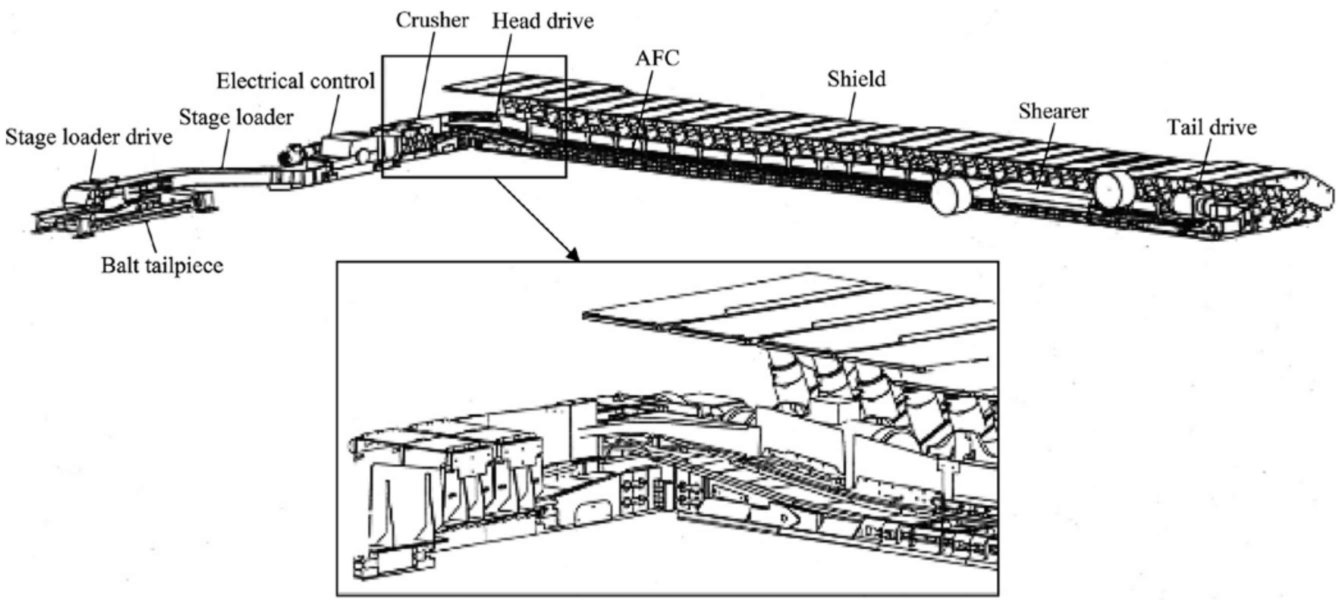


Figure 6: 3D face view of equipment layout for modern standard longwall panel.

This longwall mining set up and arrangement are designed to allow the face to advance freely without delay caused by any other auxiliary systems. Consequently, this system was quickly adopted in US coal industry and later worldwide.

REASONS FOR NO MAJOR IMPROVEMENTS IN US LONGWALL AUTOMATION TECHNOLOGY SINCE MID-1990S

1. The system works well – no need for major improvements.
2. The state-of-the-art automation system is suitable for ideal conditions.
3. Depressed market condition deprived manufacturers' incentive to innovate and large price hike in electrohydraulic shield in early 2000s caused mine operators, instead of buying new equipment, to look for alternatives, i.e. rebuilt, bought used shields, or held on to existing shields longer.
4. Miners, actually the union's reluctance to accept the technology for fear of losing jobs.
5. Successful application of the automated system did not guarantee high production. In some cases it even reduced production.
6. First and foremost, coal mining requires high production in highly safe environments. In other words, miner's health and safety is the top priority. Any application of new technology must take it into consideration. Therefore, the coal industry has been concentrating on improving miner's health and safety environments.
7. There are numerous unexpected events and conditions (mining and geological) that may be encountered any time and disrupt automation that requires miner's intervention, otherwise it may cause hazardous conditions leading to accidents, for example: (1) the shearer drum cut into roof bolt at gate-ends, (2) the shearer cut into shield canopy, (3) very large hard roof rocks fall on AFC pans blocking the shearer's advance, (4) one or more shields fail to advance as designed requiring shieldman's intervention.

IMPROVEMENTS IN HEALTH AND SAFETY OF SEMI-AUTOMATED LONGWALL SYSTEM SINCE MID-1990S

Since the first system installed in 1995, the manufacturers have been concentrating on improving the reliability and health and safety features of face equipment, including:

1. Development of various sensors to measure the ever changing geological and mining conditions for advanced control of face equipment and its operation. For example, pitch & roll controls and serial (encoder) for real time shearer attitude and location.
2. Fast moving automated heavy face equipment, especially the huge number of shields require multiple warning systems to ensure miner's safety. Developments include: proximity sensor or safety lockout, and color lighting systems to represent start of moving, fault, other activities.
3. Intelligent water spray system for shield to suppress roof dust during shield advance.
4. Refine the gate-end turn-around and wedge cut algorithm to become fully automated.
5. Refine the health and diagnostic monitoring of components of shearer, shield and AFC for preventive maintenance.
6. Development of anti-collision technology between shield and shearer.
7. Automation of shearer components: ranging arm, cowl, lump breaker, haulage & water.
8. Individual equipment reliability reaches >98.5% and whole mine system >80-85%.

DISCUSSION

The latest Coal Age longwall survey[12] showed that in 21 out of a total of 43 US longwalls, the mining height was larger than the coal seam thickness by 18-102 cm. It either cut into the roof or floor or both. With today's longwall mining technology, the minimum mining height was 165 cm (machine height) with the optimum operation height (crew's working height) around 178-203 cm. In Central and Southern Appalachian coal field where met coal seams are thinner, they can afford larger amount of rejects in coal preparation plant and still make profits due to higher met coal price. Consequently their cutting heights are 64-102 cm larger than the seam thickness. In Northern Appalachian coal field, the immediate roof, slate, of Pittsburgh seam, 15-51 cm thick, tends to fall off immediately after shearer cutting, making the cutting height larger than the seam thickness.

For those seams where mining height is larger than seam thickness, shearer's horizon control may not be a critical factor in that it does not need to follow the coal/rock interfaces. Rather the important factor is to maintain uniform flat roof and floor surfaces for smooth operation of shield and AFC pans. With the ever increasing shearer's cutting power, cutting a shale or weaker roof or floor does not seem to present any problem, except it presents a more challenging job for coal cleaning and surface waste disposal which are much easier to solve. It follows that horizon control for cutting at coal/rock interfaces is not as critical as conventional thinking.

Cutting in the rock consistently causes faster bit wear and more frequent bit change, perhaps requiring one change per cut in today's 366 m wide or wider panels. For current drum bit layout, each bit change needs and causes 20-30 min production delay.

Therefore, development of higher wear-resistant bits will greatly help, but not eliminating miners to change bits when required.

If horizon control for coal/rock interface is not a critical factor, then horizon control for shearer drum's colliding with shield canopy and/or shield flipper if exists is the most critical factor, if the shearer operators are to be eliminated. Although there are anti-collision technologies reported in the literature[6,13], their reliability in production operation is unknown. Therefore, further research is required. The anti-collision technique essentially requires the exact special location of the shearer's leading cutting drum and the canopy, and flipper if equipped, of each shield at any instant.

CONCLUSIONS

Automation of longwall machine and system follows the advancement of sensor technology and big data analysis and transmission capability. As the sensors are getting more sophisticated, longwall automation will improve more.

The state-of-the-art semi-automated longwall system consists of: Electrohydraulic control shields and doubled-ended ranging drum shearer operated in shearer-initiated-shield advance (SISA) mode with the memory cut cutting algorithm.

Miner's health and safety and company's coal production concern slowed down further development toward full automation.

Recent adoption and application of remote control of shearer is the correct approach from miner's health and safety as well as coal production points of view considering the complicated mining and geological conditions where many unexpected events can occur suddenly.

The pre-requisite for longwall automation is very high equipment reliability, i.e., 100% reliable if a man-less system is to be adopted.

Various sensors for detecting unexpected underground events as well as their software control algorithm due to complicated mining and geological conditions must be identified, developed and tested successfully in advance before full automation is feasible.

Adoption of automation of longwall system does not guarantee reduction in face crew. The highest level automation of longwall system today requires 4–10 crew members at various mines. Obviously mine/crew practices and mining and geological conditions still dictate the manpower need at the longwall face.

ACKNOWLEDGMENTS

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Syd S. Peng[1,2,3], **Feng Du**[2], **Jingyi Cheng**[3], **Yang Li**[4]

1 West Virginia University, Morgantown, WV 26505, USA

2 School of Energy Science and Engineering, Henan Polytechnic University, Jiaozuo, Henan 454000, China

3 School of Mines, China University of Mining and Technology, Xuzhou, Jiangsu 221116, China

4 College of Resource and Safety Engineering, China University of Mining and Technology (Beijing), Beijing 100083, China


DSI
UNDERGROUND
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Progress

Justifying conveyor system equipment upgrades

Safe, efficient conveying systems are the lifeblood of many large coal mining operations.

From mining to biomass, industries that handle bulk materials depend on intelligent, hardworking individuals who can be trained and promoted to positions from which they go on to make experience-based decisions. Using their expertise, they are often tasked with identifying conveyor system issues and proposing critical changes to improve production, safety and efficiency. These projects typically require capital investments, and convincing management to earmark budgets for improvements requires supporting data, solid ROI projections, thoughtful persuasion and good timing.

"Collecting the proper data and presenting convincing arguments is almost an art form," said Dan Marshall, Process Engineer at Martin Engineering. "The first few times you do it can be frustrating and tedious. But reviewing some of the company's past proposals – including those that were rejected – is always educational and working with the manufacturer of the proposed equipment can be helpful."

CHOOSING THE RIGHT KPIS

Measuring performance requires data, so determining the most relevant Key Performance Indicators (KPIs) is important. These measurements help create evidence for stakeholders so they can make informed budget decisions. From a single piece of equipment to an entire project involving multiple components, KPIs should be part of any strategic process to assess performance and help set objectives. Often displayed in graphs or charts for visual effect, performance measurements relay trends and progress related to a goal that can be easily recognised and absorbed[1].

There are two types of KPIs, *leading* and *lagging*. Leading KPIs are those that indicate future problems which can cause expensive unscheduled downtime, such as Mean Time Between Failure (MTBF). Lagging KPIs are those that happen during or after downtime, such as "reactive maintenance." Keep in mind that KPIs require a reasonable period to collect the data, sometimes stretching across an entire year or more. Benchmarks by which to measure failure or success of the performance metrics are essential.

Common Types of Bulk Handling KPIs:

1. *Unscheduled Downtime* – Labor and servicing during an emergency shutdown are estimated to be three to seven times more expensive than scheduled downtime when workers are not pulled from other essential duties and contractors have time to offer competitive estimates. For example, just a one-percent difference in system

availability for a coal-fired power plant could be worth one to two million US dollars in annual revenue. The cost of even the shortest unscheduled outage is prohibitive.

When calculating the cost of downtime, common expenses to include are:

A. Lost opportunity cost (missed sales, supply line impact, etc.)
B. Purchase of replacement components
C. Maintenance labor
D. Subcontractor labor
E. Consulting and engineering fees
F. Testing and analysis

2. *Labor Costs and Fees* – Although these are included in determining the cost of unscheduled downtime, they are both leading and lagging KPIs, essential budget line items to determine the viability of any pending project. All maintenance related to the targeted project component(s) should be logged, including servicing of the system leading to and from the component(s).

3. *Direct and Indirect Costs* – Direct costs can include labor, but generally also cover replacement equipment, contractor costs, production losses and injuries. Indirect costs are investigations and settlements as a result of injuries or accidents, increased energy usage, increases in insurance premiums, MSHA or OSHA fines and qualitative costs like poor morale, etc.

4. *MTBF* – Mean time between failures is the average uptime between unscheduled outages. It is a vital performance metric to measure safety and equipment design and aids in determining new equipment's return on prevention (ROP) as compared to existing equipment[2]. ROP is an abstract representation of the potential economic success of occupational safety and health. Equipment with a better ROP is generally higher quality, with less maintenance required. It can be expected to carry a somewhat higher purchase price, so MTBF is key to justifying the cost and safety benefits.

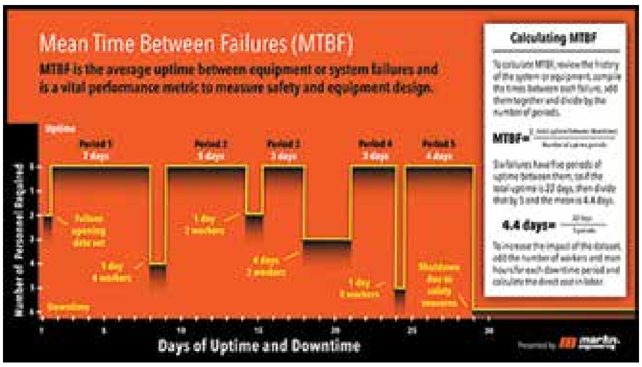

Figure 1: MTBF calculated over a sample period.

To calculate MTBF, review the history of the system or equipment, compile the times between each failure, add them together and divide by the number of periods. For example, six failures have five periods of uptime between, so if the total uptime is 22 days, dividing that by five makes the mean 4.4 days. To increase the impact of the dataset, add the number of workers and man-hours for each downtime period and calculate the direct cost in labor. [**Figure 1**]

5. *Opportunity Cost* – Opportunity cost is the value of production lost due to unscheduled events such as machine breakdowns, shutting down to clean up fugitive material or safety incidents. The concept is that if the product is not available for processing, and therefore sale, a profit opportunity is lost. [**Figure 2**]

MAKING THE CASE

"As technical people who work with the equipment day in and day out, perhaps the most difficult part of this process is having to justify or 'sell' it to management," said Marshall. "To do this, operators need a good narrative, solid data, reasonable cost projections and a convincing ROI (return on investment)."

Stakeholders will typically visit the area when the system is working well, so photos and video bolster the narrative and help with visualization. More is better, and quality matters. Graphs are also invaluable for visualization, so plan KPIs with a clear X & Y axis that will reveal evident "differences over time" or "costs per unit," etc.

ROI is extremely important in any equipment purchase but calculating it can be tricky. That is why all direct and indirect costs need to be applied. The goal for many smaller projects such as belt cleaner upgrades is to get the payback period to 1 year or less. [**Figure 3**] Categorise all possible causes of increased costs and then figure out the costs associated with each category.

For example, calculating ROI to upgrade belt cleaners starts first with isolating a cleaner, then identifying the challenges associated with it. Likely one category will be spillage from carryback. Some of the common costs associated with spillage are cleanup time/labor, low air quality, safety (lockout/tagout, PPE, etc.), replacement parts (fouled rollers and machinery) and unscheduled downtime. [**Figure 4**]

Although ROI is a focus for management, Return on Prevention is arguably just as important. Staying with the example above, lower quality equipment may offer a quicker ROI but might only clean 80% of material from the belt and deliver a shorter service life before unscheduled downtime starts all over again due to dust and spillage. Higher quality equipment with proven performance may be a higher cost with a slightly extended ROI, but the cost is generally justified over the long term. Reviewing equipment

$$\text{Opportunity Cost} = \frac{\text{tons}}{\text{hour}} \times \text{Unplanned Downtime (hours)} \times \left[\frac{\text{Sales(\$)}}{\text{ton}} - \frac{\text{Cost of Sales(\$)}}{\text{ton}} \right]$$

Figure 2: Opportunity cost calculation[2].

RDI conversions

ROI	Payback years	Payback months
10%	10.0	120.0
20%	5.0	60.0
30%	3.3	40.0
40%	2.5	30.0
50%	2.0	24.0
60%	1.7	20.0
70%	1.4	17.1
80%	1.3	15.0
90%	1.1	13.3
100%	1.0	12.0

Figure 3: ROI payback over the specified time[1].

Data used in ROI calculations

Data	Units
Administrative/operating	
Cost of compliance: record keeping and reporting	currency
Health and liability insurance premiums increase	currency
Reduced life of equipment	currency
Safety/environmental fines	currency
Legal costs	currency
Energy costs	currency
Waste disposal costs	currency
Production	
Throughput: per hour, day, week, or month	tons (st)
Production time	hours
Cost per ton of bulk material	currency/ton (st)
Cost of down time	currency/hour
Cleanup manual (1 ton per hour is avarage)	labor cost/hour
Cleanup machine (5 tons per hour is average)	labor and machine cost/ hour
Lost product due to dust and spillage	0.5% to 3% of production rate is typical
Safety *(Reference 31.2)*	
Cost of recordable incident	currency
Cost of lost-time incident	currency
Maintenance	
New installation: *estimated cost of labor and materials*	currency
Adjustment: *estimated labor cost per adjustment*	currency
Replacement Parts: *cost of parts and labor*	currency
Equipment wear: *cost of belt and wear-resistant mateials*	currency

Figure 4: ROI categories for a belt cleaner replacement[1].

Factory-trained personnel help ensure that projects will meet government-mandated safety standards.

specs, examining the construction and evaluating case studies from similar applications can help determine ROP.

Successful proposals generally offer a direct line to a solution and the next steps for implementation. Make sure the intent of the project is clear, the bottom line is as close to the real outcome as possible and that all project variables are considered (downtime, labor, installation obstacles, special equipment such as cranes and any associated safety regulations or certifications).

To ensure that projects will meet government-mandated safety standards, insist on factory-trained technicians with certifications from OSHA, MSHA and other industry-recognised organizations. Many equipment suppliers contract their installation and service functions to outside firms, which often represent dozens of different product lines. Personnel trained by the equipment manufacturer and dedicated solely to its proper care will have greater knowledge and experience, ultimately delivering superior results over the long term.

DETERMINING THE INVESTMENT STRENGTH

One of the most anxiety-inducing aspects of this process is determining how to make the best financial decision on equipment. Luckily, there are the general calculations of net present value (NPV) and internal rate of return (IRR) to help with this endeavor. These are financial tools that can be used to compare investment options, including safety investments.

NPV is a financial measurement of life cycle costing where two or more options are evaluated based on initial price, annual costs and expected life as expressed in terms of today's currency. Generally, the option with the highest NPV would be the wisest choice. IRR shows the annual compounded rate of return on an investment and is defined as the interest (or discount) rate that makes the NPV equal to zero.

NPV and IRR are calculated in **Figure 5**. The calculations are linked to:

- Cash Flow = the expected savings for a specific year minus the costs of operating and maintaining the project

$$\text{Net Present Value} = -\text{Initial Investment} + \sum_{i=1}^{I} \frac{\text{Annual Cash Flows}}{(1+R)^i}$$

$$\text{NPV} = -\text{Initial Investment} + \frac{\text{Cash Flow Year 1}}{(1+R)^1} + \frac{\text{Cash Flow Year 2}}{(1+R)^2} + \frac{\text{Cash Flow Year 3}}{(1+R)^3} \ldots$$

Internal Rate of Return = What Rate R will Make NPV = 0?

$$0 = -\text{Initial Investment} + \frac{\text{Cash Flow Year 1}}{(1+IRR)^1} + \frac{\text{Cash Flow Year 2}}{(1+IRR)^2} + \frac{\text{Cash Flow Year 3}}{(1+IRR)^3} \ldots$$

Figure 5: NPV and IRR are common industry-wide tools used to approximate investment strength[2].

in that year.

- I = The total number of periods (usually years) used in the analysis.
- Initial Investment = the initial purchase, delivery and installation costs of the project.
- R = the weighted cost of money for the company from all sources: borrowing, selling stock, etc. Expressed as a decimal and often called the discount rate, this can also be thought of as the inflation rate.
- IRR = the discount rate that makes the NPV equal to zero.

HALF MEASURES OFTEN ACHIEVE LESS THAN HALF RESULTS

Purchasing decisions are often based more on price and what's in the budget than on achieving performance (ROP) and reducing costs. A common question is: "This is what I have in the budget, what can you do for that?" The correct answer is often, "Nothing." That's because taking half measures often only temporarily treats the symptoms of conveying problems and doesn't address the root causes. To illustrate the point, a belt cleaning case study[3] is analyzed using actual customer data and making some assumptions based on industry averages[2]. The installation and maintenance costs consider that the conveyor is a reversing design and dual belt cleaners were installed at both ends. It's critical to specify equipment that is designed for safety and ease of service, rather than just seeking the lowest-cost options. These components may carry a slightly higher initial price, but they will pay off over the life of the equipment and ultimately result in lower life cycle costs.

Belt Cleaning effectiveness is the % of material the cleaner removes from the belt and is measured by the grams per square meter (g/m^2) that the cleaner removes from the dirty portion of the belt. Many manufacturers claim 98% or more cleaning efficiency without specifying 98% of what: 98% of 500 g/m^2 or 98% of 100 g/m^2 of carryback? The desired result is not cleaning efficiency, but the effectiveness in reducing carryback – expressed in the tons of fugitive material that have to be cleaned up. In this study the carryback levels were measured by a technician using a standardised test method. Equipment design and effective maintenance are keys long term safety and cost control. Components that are engineered with these priorities will deliver longer service life and reduce maintenance costs, while minimising the risks inherent to bulk conveying. In this analysis, the effectiveness is assumed to be 50% for the precleaner and 55% for the secondary. It was assumed the cleanup was done manually by shoveling at a rate of ½ a ton per hour and labor cost is $25/hour.

The 5-year time frame was chosen as a reasonable life for this type of equipment. Doing nothing is costing $800,800 in discounted cash flow over 5 years. For spending an additional $10,000 up front on equipment and $5,000 a year in maintenance, the additional cash flow for the full solution (installing two cleaners on each end of the reversing conveyor) compared to the half solution is $201,700 on labor alone for the dual cleaning system vs. a single belt cleaner on each end of the conveyor and $578,000 compared to doing nothing.

Customer data		Assumptions	
Material	Frac Sand	Initial Installation Cost	$20,000
Carryback Before	4,225 tons/y	Annual Maintenance Cost	$7,000
Carryback After	930 tons/y	Cost of Money	10%
Additional Sales	$400,000	Evaluation Time Frame	5 years
Downtime Reduced	$?	Cleanup Rate per Hour	0.5 t/h Shoveling
Cleanup Reduced	$?	Belt Cleaner Effectiveness	50% & 55%
Safety Savings	$?		

Figure 6: Belt Cleaning Case Study Data.

	Cleaner Effect.	Carryback Clean Up	Labor Cost/y @ 0.5 t/h Shoveling	Initial Installation	Annual Maint.	NPV: 5 years @ 10%
Before Upgrade	0%	4225 t/y	$211,250	$0	$0	$800,800
NPV of Cash Flows from Labor Savings						
Half Solution 2 Precleaners	50%	2113 t/y	$105,650	$10,000	$3,500	$377,300
Full Solution 2 Precleaners & 2 Secondaries	77.5%[a]	950 t/y	$46,500	$20,000	$7,000	$578,000
[a] Assume the dirty belt has 100 g/m^2 of carryback. Effectiveness = 100 g/m^2 x [(1-50%) x (1-55%)] = 22.5 g/m^2 remaining on belt after cleaning or (100g/m^2 - 22.5 g/m^2)/100 g/m^2 x 100% = 77.5% effective.						

Figure 7: NPV of Cleanup Labor Savings for Half and Full Solutions

If the one-year ROI on the initial investment for the full solution compared to the half solution is considered as savings divided by costs, it would be ($211,250 – 46,500)/$20,000 = 1.76 or 176%, which is very good. But ROI doesn't tell the whole story, and that's why the NPV method should be used. One could also consider adding tertiary cleaners, but at some point there is a diminishing return, as it's not possible to clean a conveyor belt 100% consistently over time.

A company's cost of money may be different, or it may have a different labor rate. Once the NPV spreadsheet is set up, it's very easy to change assumptions, costs and savings to compare the results. If the cash flow from added sales and reduced accident exposure and other identifiable costs are included, it becomes even more clear that best financial, safety and production is the full solution. As is the case of most upgrades for the control of fugitive materials, the ROP is so great that the Internal Rate of Return is off the charts.

PROJECT MANAGEMENT

The success or failure of a project can come down to good project managers. They manage the schedule and budget to ensure that work is completed on time, and on budget. Establishing reasonable and clear expectations for co-workers, vendors and subcontractors helps ensure the quality of the finished product. Some manufacturers offer conveyor inspections and cleaner maintenance as part of a managed service relationship. Their monitoring systems can track component wear and update the service technician and/or operations personnel via wi-fi or cell phone on upcoming service needs. Some new systems can even adjust belt cleaner tension automatically, and the technology will also send an alert through a mobile app in the event of upset conditions.

Clear scope, budget and timeline management are critical to a successful project.

Factory-trained service technicians provide an added set of eyes on the conveyors, travelling to and from the equipment to be serviced and logging details in their service reports. Because they see so many different applications, they can often alert on problems that maintenance personnel don't see or have become accustomed to ignoring. With factory-direct managed service, the responsibility for maintenance falls on the manufacturer, allowing the staff to focus on other priorities.

At first glance it may seem that a plant has the in-house capacity to maintain belt cleaners, and hiring a managed service provider doesn't make sense. The reality is a conveyor will run with a belt, a head and tail pulley and a drive – maintaining everything else can be put off (and often is) for production at any cost. A "run until broken" philosophy means more than non-functioning equipment – it can increase unplanned downtime, exacerbate financial issues and affect worker morale, too. Then, in the rush to patch things together, maintenance workers are tempted to take shortcuts and work around established procedures, exposing them to greater potential for injury. In contrast, a service contract that employs factory-trained technicians will often result in problems being identified before they become catastrophic failures, reducing downtime and further equipment damage.

Factory-trained direct service personnel and replacement parts are key to obtaining expert maintenance for optimum performance and component life, leading to on-time deliveries and high customer satisfaction. Some manufacturers will even supply free remote monitoring and reporting equipment that's accessible by wi-fi or cell phone. These managed service technicians, supported by a financially stable, well-established manufacturer and armed with the specific knowledge and equipment to do the job, are often the answer to common belt cleaning

A properly configured conveyor controls emissions for improved safety and easier maintenance.

problems. For these technicians, who spend every day assessing and servicing belt conveyors, maintenance and repairs become more of a precise science than a judgement by rule of thumb.

PRIORITIZING SAFETY JUSTIFIES THE COST

Often issues like excessive dust, mistracking, spillage, carryback, etc. are considered commonplace and "the cost of doing business." In reality, they are extremely unsafe, costly and easily remedied with modern equipment. A common injury for cleaning or maintenance personnel is a muscle strain. A common injury for cleaning or maintenance personnel is a muscle strain. The OSHA Safety Pays Calculator[4] estimates the cost of a single lost time muscle strain injury at $32,023 in direct and $35,225

Regular inspections by factory-direct professionals help minimise downtime and improve efficiency.

in indirect costs for a total of $67,248. If there is a history of safety incidents, improvements can often be justified on safety alone. Identifying that an issue exists is the first hurdle; another is asking for help collecting data and making sure it's recorded correctly. Keeping the project and equipment decisions simple and safety-focused is the best approach.

"The earlier service technicians are brought into the process, the more they can assist," Marshall added. "We often walk the belt and inspect conveyor systems along with operators to find practical solutions that can help define their KPIs, narrow the scope of data collection and get them to their goal faster and more safely. Regularly-scheduled reviews of conveyor belts, cleaners, tracking, chutes, dust control and other components from experienced specialists with extensive training and expertise will help conveyor operators maximise productivity and reduce downtime.

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R. Todd Swinderman
CEO Emeritus / Martin Engineering

An innovative non-pillar coal mining technology with automatically formed entry

A non-pillar coal-mining technology with an automatically formed entry is proposed, which reduces the waste of coal resources and the underground entry drivage workload. Three key techniques in this technology cooperate to achieve automatic formation and retaining of the gob-side entry, and to realize non-pillar mining. Constant-resistance large deformation (CRLD) support ensures the stability of the entry roof; directional presplitting blasting (DPB) separates the entry roof and the gob roof; and a blocking-gangue support system (BGSS) integrates the caved rock material as an effective entry rib. An industrial test was conducted to verify the engineering effects of these key techniques. The field application results showed that the retained entry was under the pressure-relief zone due to the broken-expansion nature of the caved rock mass within the DPB height. After going through a provisional dynamic pressure-bearing zone, the retained entry entered the stability zone. The final stable entry meets the requirements of safety and production. The research results demonstrate the good engineering applicability of this technology. By taking the framework of the technology design principles into consideration and adjusting the measures according to different site conditions, it is expected that the proposed non-pillar coal-mining technology can be popularized on a large scale.

INTRODUCTION

China continues to dominate the global market for coal, and the outlook for the next 20 years indicates that China will remain the world's largest consumer of coal[1]. Based on insights into China's energy consumption, coal consumption will still account for over 50% of primary energy by 2030[2,3]. Meanwhile, coal demand within India and other emerging Asian economies will increase, since coal will be used to meet robust growth in power demand as these economies grow and their prosperity increases[4]. China is the biggest coal-mining country and a global pioneer in mining technology, and its longwall mining technology development is greatly improving coal production[5]. However, China faces a significant problem of coal resource waste during exploitation.

The mine average recovery rate is only about 50%, as many coal pillars acting as shields are left underground and cannot be reclaimed[6,7]. In **Figure 1(a)**[8], a long coal pillar is set between mining panels I and II, and the pillar width

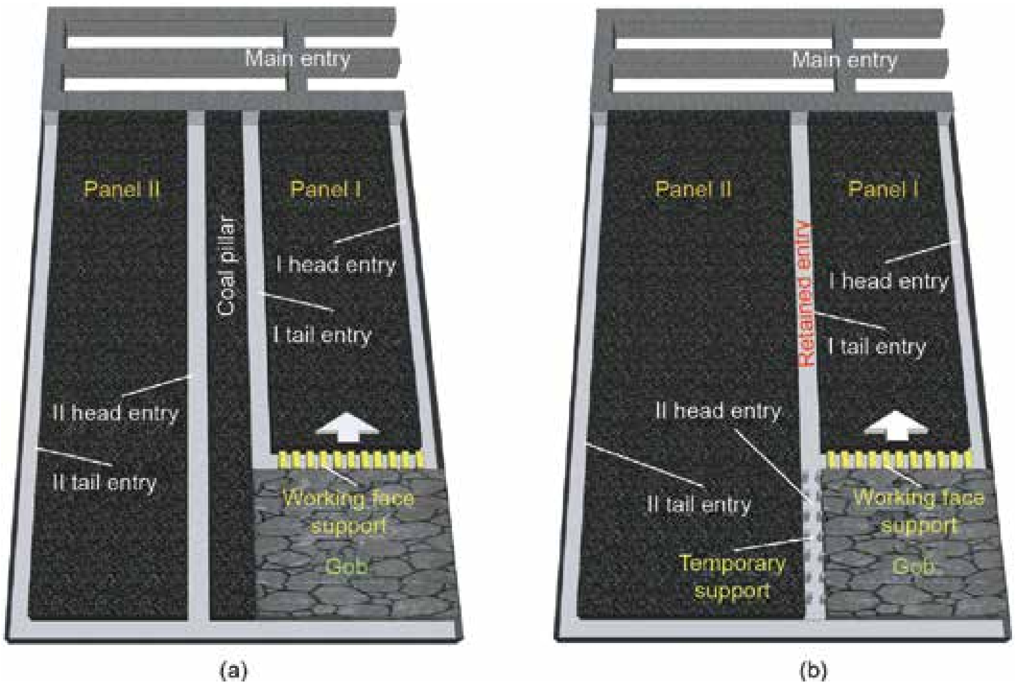

Figure 1: Schematic diagram of the conventional and non-pillar mining approaches. (a) Conventional entry layout; (b) entry layout for non-pillar mining. Reproduced from Ref.[8] with permission of Elsevier Ltd., ©2019.

would generally increase as mining activity continues to a deeper level[9]. Furthermore, the width design of the entry pillar is a complicated issue that has come under continual study around the world[10-16].

Therefore, this study presents a non-pillar longwall mining technology that does not require the entry pillar setting. As shown in **Figure 1(b)**[8], the II head entry will be automatically formed from the I tail entry during panel I advances. Using the technology presented here, this automatically formed entry can be safely and stably retained for subsequent panel mining. In this way, no coal pillars are set between the mining panels, and coal resources can be extracted to the greatest extent, allowing engineers to avoid the protective coal pillar retention problem. Furthermore, the head entry of the next mining panel is automatically formed during the mining of the previous panel. The preparation entry drivage work is cut in half compared with the conventional mining method, significantly decreasing the workload and thus reducing the potential drivage hazard[17-20].

In recent years, industrial experiments on non-pillar mining at pilot sites have been successfully conducted, led by the State Key Laboratory for Geomechanics and Deep Underground Engineering, Beijing, China. Guo *et al.*[21] examined the feasibility of this technology in thin coal-seam mining, and He *et al.*[22] studied the adaptability of this technology for medium-thickness coal-seam mining. He *et al.*[23] conducted an industrial test of this technology in thick coal-seam mining with a fast mining rate and achieved a satisfying effect. He *et al.*[24] also successfully applied this technology to deep coal mining.

In this paper, we systematically summarize this non-pillar mining technology. The principles behind the technology are analyzed first. The key techniques involved in the technology are then introduced; their design methods and application effects are studied separately in combination with their field application. Finally, the retained entry stability and engineering effect are discussed.

2. PRINCIPLES OF NON-PILLAR MINING

The automatically retained entry is utilized by the next panel, and no coal pillar is left during the mining. Due to the strata movement, the retained entry faces intense mining pressure. In this technology, we use three key techniques to ensure entry stability: constant-resistance large deformation (CRLD) anchors, directional presplitting blasting (DPB), and a blocking-gangue support system (BGSS). As shown in **Figure 2**, CRLD anchors are first applied to support the entry roof. DPB is then applied in the roof on the mining side. A smooth fracture face (presplitting plane) is formed under the DPB effect. As the coal seam is mined out, the roof within the DPB range caves under the mine pressure. The caved rock mass expands because of the broken-expansion nature of the rock. The expanded rock material compensates for the coal-extraction space. Therefore, the upper roof movement is restricted. Meanwhile, the caved rock mass in the gob becomes the natural rib of the entry. The BGSS is set at the gob side in the entry to mold the integrated rib. This entry is automatically formed and safely retained to serve the next panel. This method allows the maximum extraction of coal resources in the mining area, and reduces the entry excavation by half in the subsequent mining. In the practice of longwall mining, entries are excavated in advance to prepare the mining face. Therefore, CRLD support and DPB can well be pre-implemented after the entry excavation. When the mining starts, the BGSS is installed behind the mining face and is implemented simultaneously with the advance of the mining face. Therefore, the implementation of these three techniques can be prepared well and the techniques do not interfere with each other, ensuring mining efficiency.

(a)

(b)

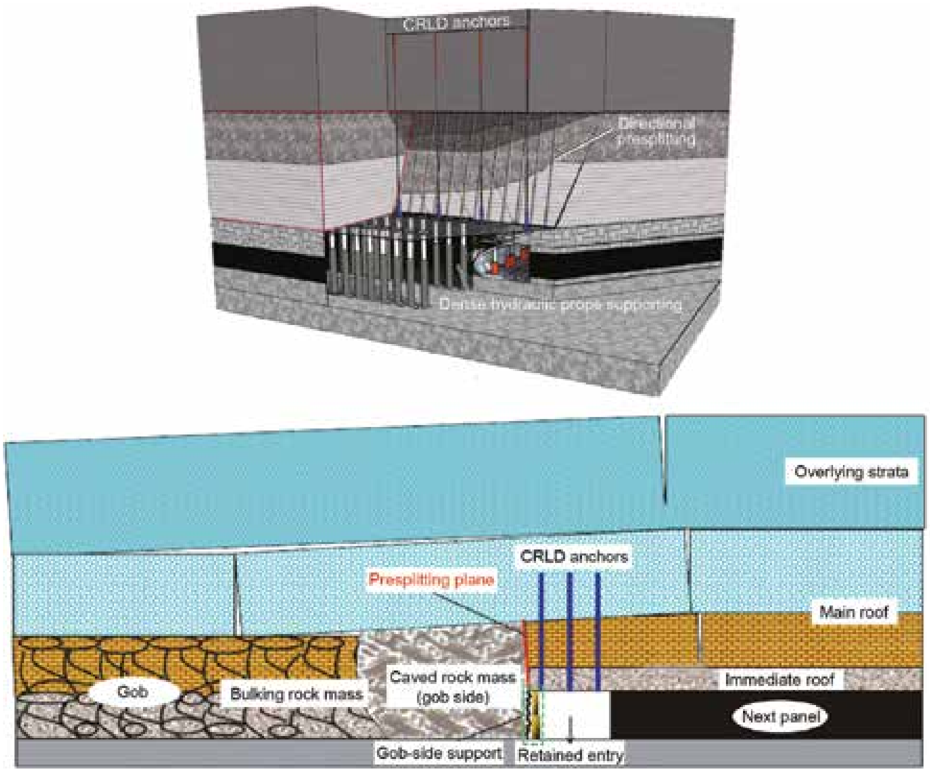

Figure 2: Schematic diagram of non-pillar coal mining with an automatically formed gob-side entry. (a) Three-dimensional (3D) view of the retained entry; (b) stratigraphic model of non-pillar mining with a retained entry.

(a)

(b)

(c)

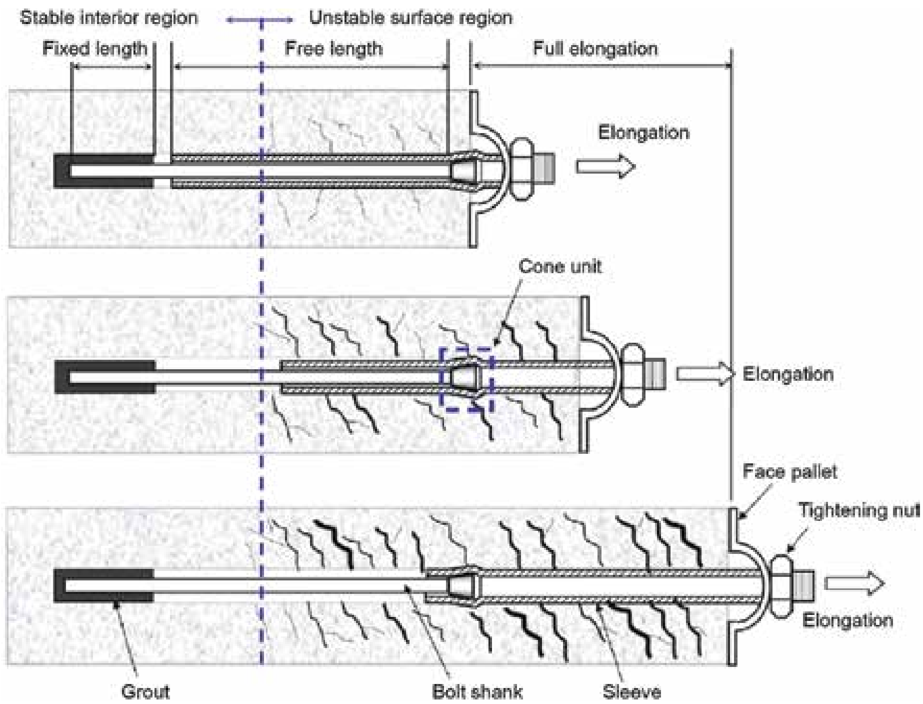

Figure 3: Working principle of the CRLD anchor. (a) Elastic deformation stage; (b) constant-resistance deformation stage; (c) ultimate deformation stage. Reproduced from Ref. [25] with permission of Elsevier Ltd., ©2014.

3. KEY TECHNIQUES

3.1 CRLD support

The CRLD anchor cable was developed and patented by Manchao He and his research team[25,26]. In actual engineering application, CRLD anchors can accommodate large deformation of the adjoining rock mass at a great depth in response to external force. The anchor consists of two parts: the constant-resistance body and the bolt shank. As shown in **Figure 3**[25], the constant-resistance body is composed of a cone unit and a sleeve. The sleeve acts as a slide track for the cone unit. The shank is anchored by grouting in the depth of the rock mass – that is, in the fixed stable region. On the surface of the anchored mass, a combination of a face pallet and tightening nut is used to fix the free end. When the rock mass deforms under external disturbances, the constant-

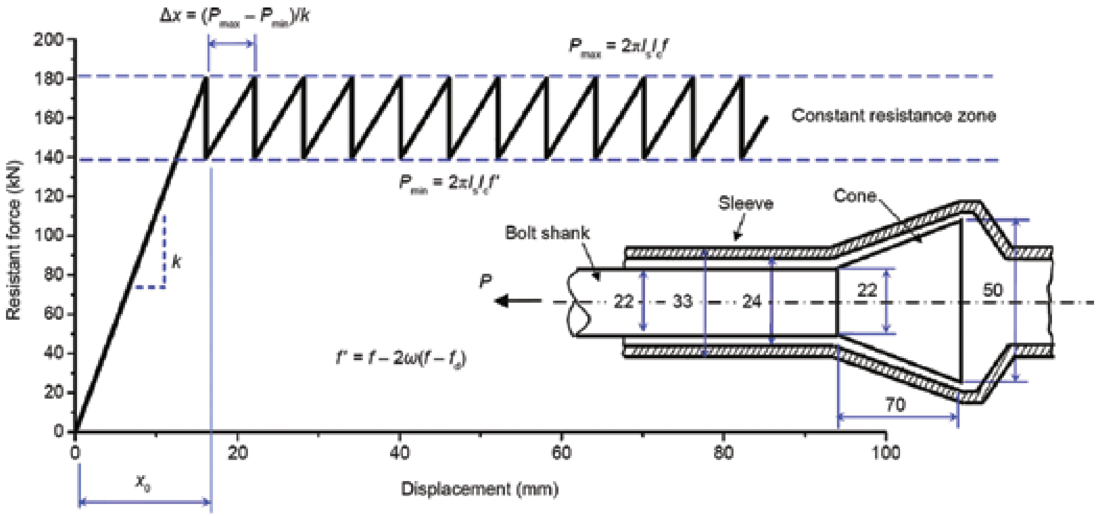

Figure 4: Analytical load-elongation curve of the CRLD anchor. The anchor's size is given in the sketch (unit: mm). x: Undamped natural frequency; f: static frictional coefficient; fd: dynamic frictional coefficient; f0: equivalent frictional coefficient; k: shank stiffness; Is: sleeve elastic constant; Ic: cone geometrical constant; x0: elastic displacement; Dx: cycled displacement. Reproduced from Ref. [25] with permission of Elsevier Ltd., ©2014.

resistance body generates an internal slide, and the sliding distance depends on the free length of the CRLD anchor. At present, the free length is from 300 to 2000 mm of the different CRLD specifications[27]. Three stages of the sliding movement are illustrated in **Figure 3**[25]: the elastic deformation stage (**Figure 3(a)**), constant-resistance deformation stage (**Figure 3(b)**), and ultimate deformation stage (**Figure 3(c)**). At the elastic deformation stage, the axial force caused by the rock deformation is less than the constant resistance of the CRLD anchor, which is not enough to activate the cone unit sliding in the sleeve. The elastic deformation is tiny and occurs within the constant-resistance body and bolt shank themselves; the bolt does not elongate substantially. As the axial force increases to the constant force, the CRLD anchor enters the constant-resistance deformation stage. The CRLD anchor maintains high constant resistance during bolt elongation (i.e., the sliding movement of the cone unit). This resistance is predefined by the function of the cone unit and sleeve. At present, the successfully tested resistance is up to 850 kN[27]. Therefore, the CRLD anchor absorbs a massive amount of energy to resist the consistent deformation and failure of the country rock mass in the constant-resistance deformation stage. The elongation will eventually stop after the energy is fully released; at that moment, the external force will be smaller than the constant resistance. The rock mass within the anchored range will achieve a new stable state after the strong disturbance.

An analytical load-elongation relation was established for the CRLD anchor according to its constitutive relation[25]. **Figure 4**[25] shows several typical cycles of the analytical load-elongation curve for the CRLD anchor based on the calculation of a 16 t CRLD anchor. In the initial stage (elastic deformation stage), the resistance elastically increases with a tiny displacement of less than 20 mm. The curve of the elastic deformation stage is in accordance with Hooke's law, P = kx, where P is static tensile load, k is the stiffness of the bolt shank, and x is the displacement or elongation. When the increased force achieves the predesigned constant resistance, the curve oscillates periodically in the constant-resistance zone with the continuously increasing displacement. The calculated maximum and minimum forces are 180 and 140 kN, respectively. These two limit values remain stable while the CRLD anchor is elongating. Related laboratory tests were conducted and developed to observe the CRLD performance, and the test results verified the ability of the anchor to accommodate a large deformation with a high constant resistance[28].

3.2 Directional presplitting blasting

The DPB technique applied in the new non-pillar mining approach is based on the bilateral cumulative explosion technology presented and developed by Manchao He and his research team[29,30]. This technology is aimed at directionally blasting a material that has a high compression resistance and low tension resistance. This technology makes use of a bilateral energy-gathering device. The explosive blasts in this device and the blasting energy are converted into point-strip energy flow via energy-gathering holes. As shown in **Figures 5(a)** and **(b)**[31], the ejected point-strip energy flow applies the cumulative tension on the local area of the borehole (i.e., the area of the energy-gathering holes) while the remaining area of the borehole is uniformly compressed due to the protective function of the energy-gathering device. Therefore, a directional crack can be developed in material that is good at resisting compression but fails under tension. The rock itself possesses this mechanical property. A blasting test was conducted in the rock mass, and the application effect is illustrated in **Figure 5(c)**[31]. A line of boreholes using the bilateral cumulative explosion technology were blasted together in the rock mass. A directional crack connected these boreholes along the energy-gathering direction, and no other visible cracks were generated in other directions.

DPB is used to generate a smooth structural surface between the retained entry roof and the gob roof before the mining activity arrives. As shown in **Figure 6**[8], bilateral energy-gathering devices with the explosive are installed into boreholes that are designed in the retained entry roof on the mining side (i.e., the gob side after mining, as shown in **Figure 1**[8]). Rows of the energy-gathering holes are aligned along the roadway strike direction. By setting a

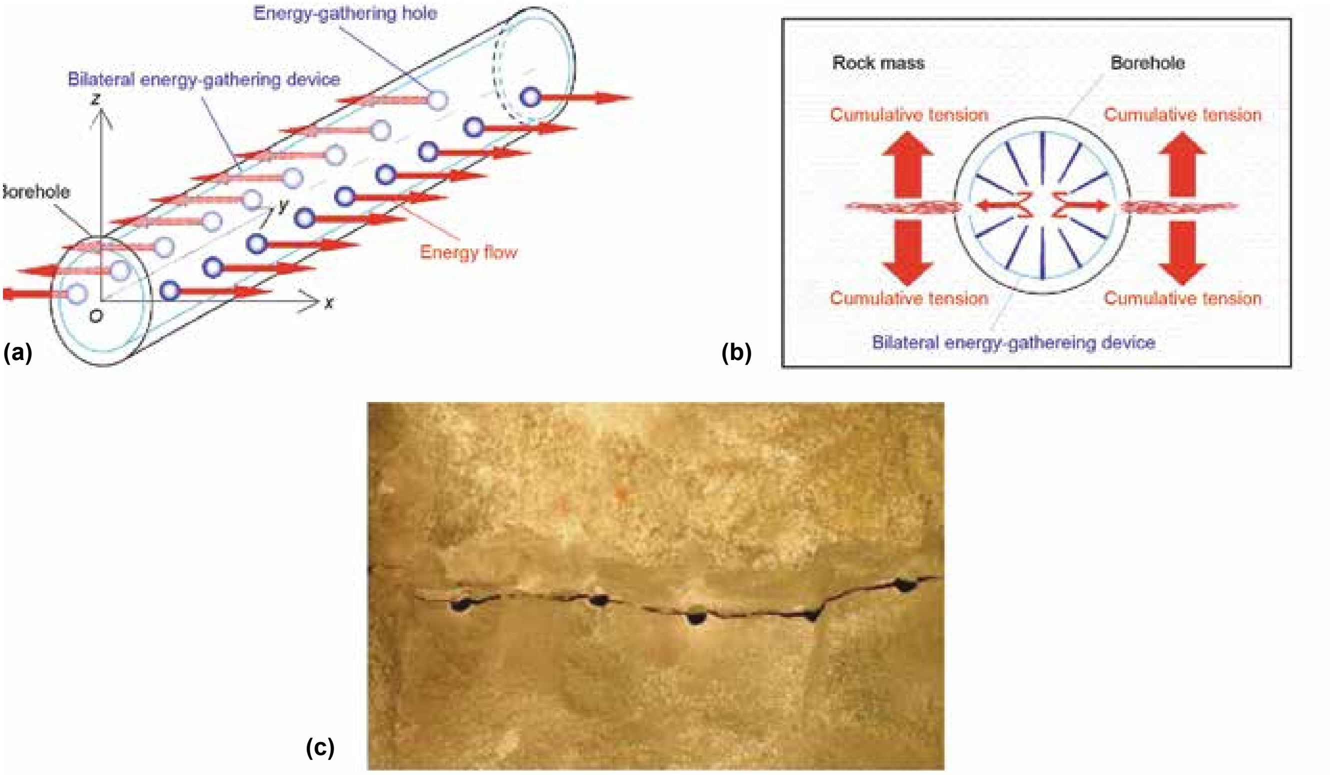

Figure 5: Mechanical model of the directional blasting and its application effect. (a) Cumulative blasting diorama; (b) cumulative blasting effect in the view of the x-z plane; (c) multi-hole blasting effect in rock mass. Reproduced from Ref. [31] with permission of Elsevier Ltd., ©2020.

specific interval among these devices, a presplitting plane is generated in the energy-gathering direction by means of the bilateral cumulative explosion technology. DPB realizes the separation between the retained entry roof and the gob roof, which artificially controls the caving position of the gangue on the entry side. This makes it possible for the caved gob roof to turn into the rib of the retained entry. Moreover, as a refined blasting technique, DPB will not damage the original roof integrity of the retained entry.

3.3 Blocking-gangue support system

To integrate the caved rock material on the gob side into an effective entry rib, the gob-side support technique was studied. In terms of the space-time relation, the dynamic course of the caved material exists in two forms: the caving process and the compacting process. The falling rock material first causes an instantaneous impact on the gob-side support in the caving process and then causes a lateral extrusion to the gob-side support in the compacting process. The BGSS is accordingly designed with three major parts: an anti-impact self-advancing structure, a sliding-yield structure, and an auxiliary supporting structure. The structure layout of the BGSS is shown in **Figure 7**. The anti-impact self-advancing structure is located right behind the face-end support and is connected to it; a metal mesh is set to segregate the gangue; and the sliding-yield structure and auxiliary supporting structure are spaced

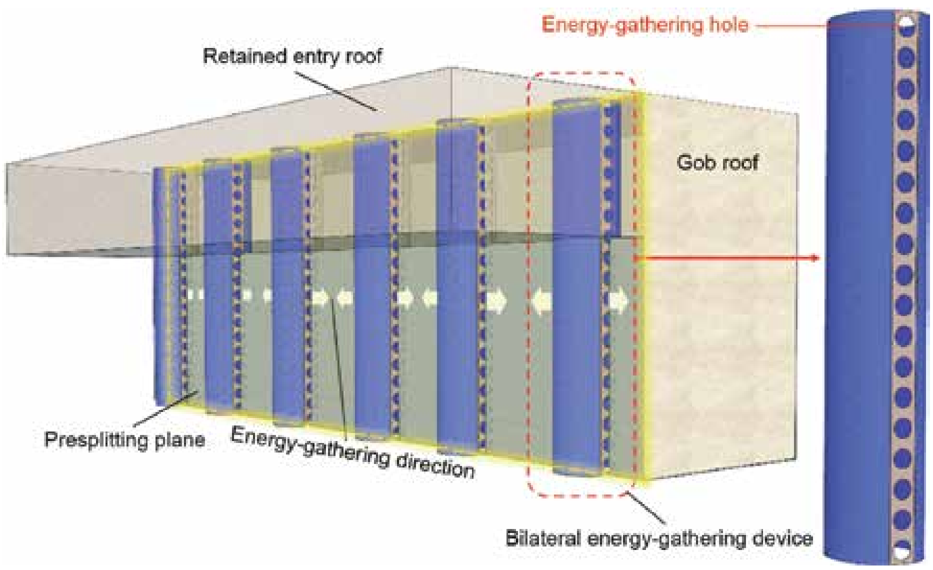

Figure 6: DPB application in a retained entry roof. Reproduced from Ref. 8 with permission of Elsevier Ltd., ©2019.

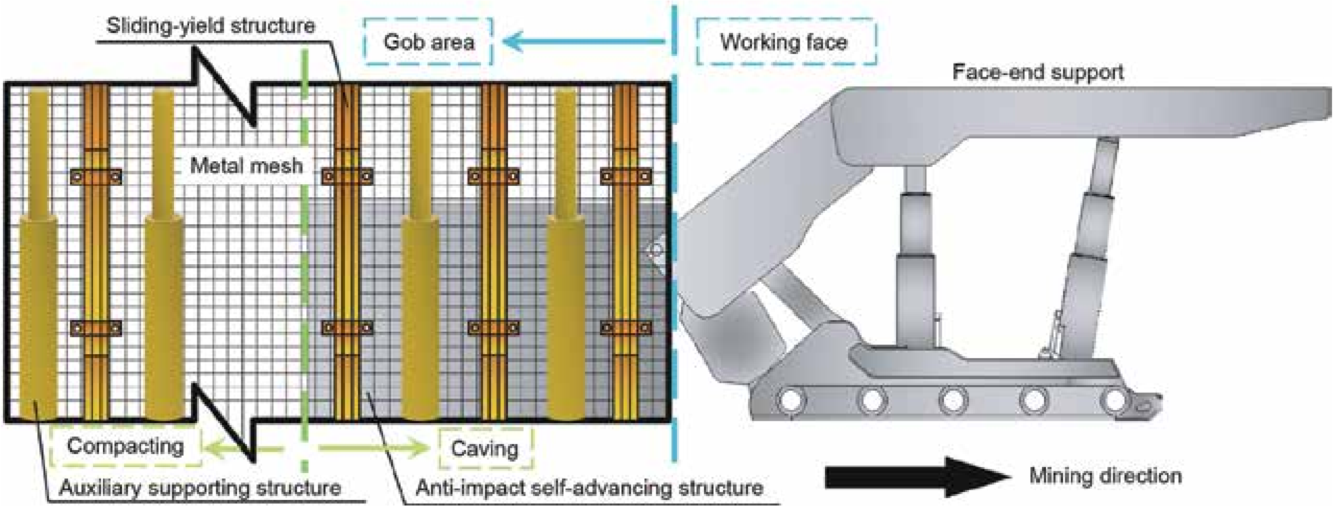

Figure 7: Layout of the BGSS structures.

reciprocally outside the metal mesh. First, the anti-impact self-advancing structure in the rock caving area converts the local impact into integral load-bearing by increasing the force area with the gangue and the contact area with other structures; this reduces the impact on the individual blocking-gangue structure. Moreover, this structure realizes self-advancement by connecting with the face-end support for timely resistance of the instantaneous impact in the caving process. The sliding-yield structure is composed of overlapped U steel, which possesses excellent resistance against bending. The sliding-yield structure can also slip appropriately to accommodate vertical deformation due to roof pressure during the compacting process. Adjusting the torque of the clips can strengthen the structure axial bearing capacity. These performances ensure the integrity and reusability of the sliding-yield structure. The auxiliary supporting structure is used to resist the roof pressure on the presplitting side, thus reducing the axial load on the sliding-yield structure. An excessive axial load would cause the structure to bend locally and would influence the structural resistance to lateral deformation. Therefore, the setup of the auxiliary supporting structure potentially maximizes the resistance to the lateral deformation of the sliding-yield structure in this circumstance. In addition, the substantial contact between the sliding-yield structure and the entry roof and floor generates resistance friction to control the gangue lateral deformation cooperatively.

According to different geological mining conditions, we designed and adopted a matched auxiliary supporting structure. The hydraulic prop is applicable to coal-seam mining of thin and medium thickness, while the unit support is applicable to thick coal-seam mining, whose rock pressure phenomenon is more violent. The constructions of the BGSS structures are shown in **Figure 8**.

4. FIELD APPLICATION

4.1 Site conditions

The Baoshan coalmine, located in Inner Mongolia, China, was selected for the field application of this technology. As shown in **Figure 9(a)**, panel 6301 became the gob after mining, and the coal pillar was left between the 6301 and 6302 mining panels. The proposed non-pillar mining technology was adopted during the mining of panel 6302, so the 6302 tail entry was retained automatically as the 6303 head entry for the next panel; thus no coal pillar was left there. The roof lithology 10m above the 6302 tail entry and the lithology of the entry were investigated, as shown in **Figure 9(b)**. The entry was 2.45m in height and was a half-coal and half-rock tunnel excavated along the top of the coal seam; its average buried depth was 60m. The mean thickness and inclination of the mined coal seam were 1.56m and 2°, respectively; thus, it was a medium-thickness and near-horizontal coal seam. The immediate entry roof was fine

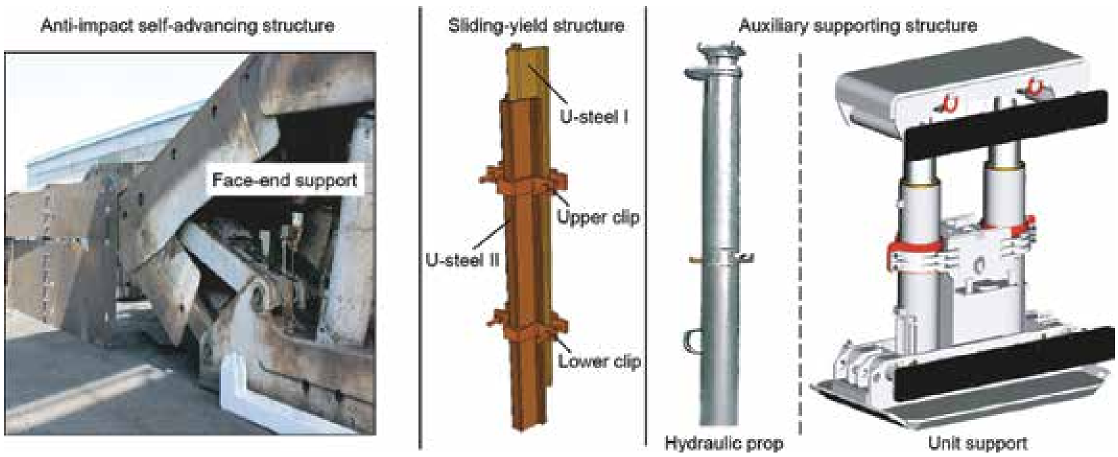

Figure 8: Classification and construction of the BGSS structures.

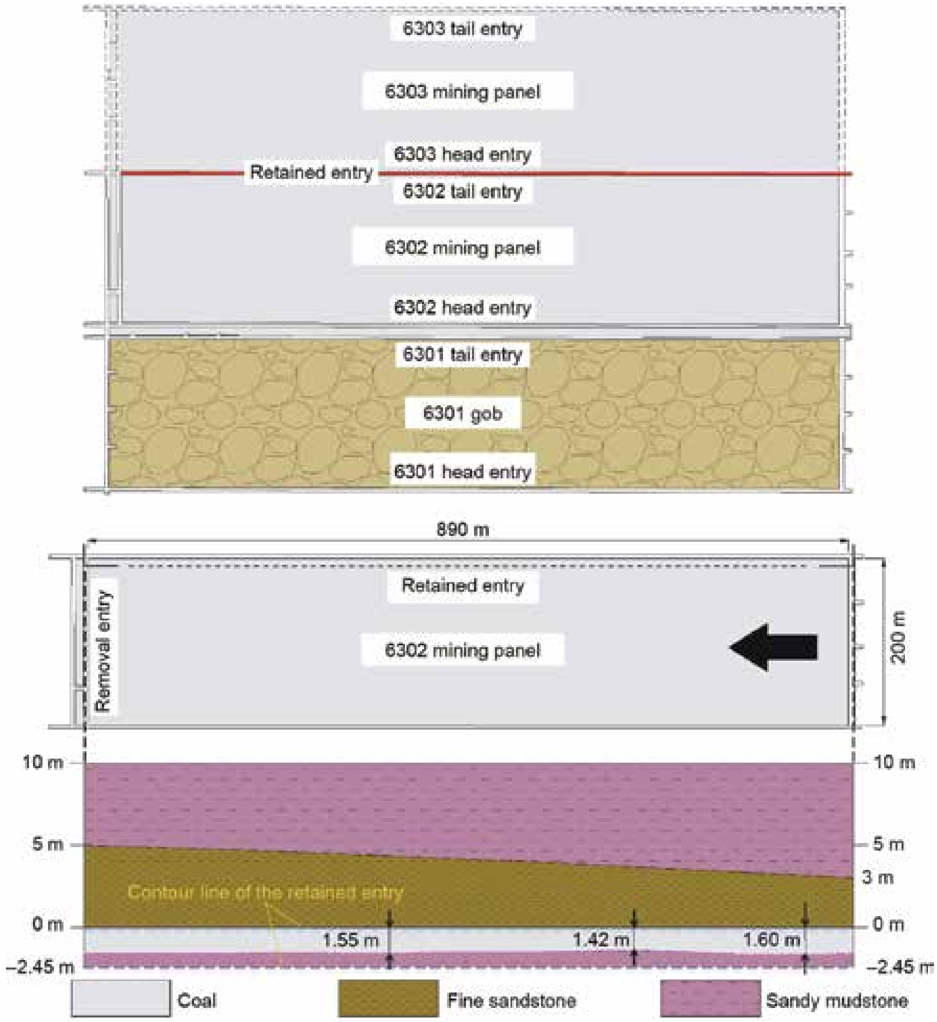

Figure 9: Mining panel layout and geological conditions. (a) Layouts of the 6302 mining panel and its adjacent panels; (b) roof lithology for the retained entry.

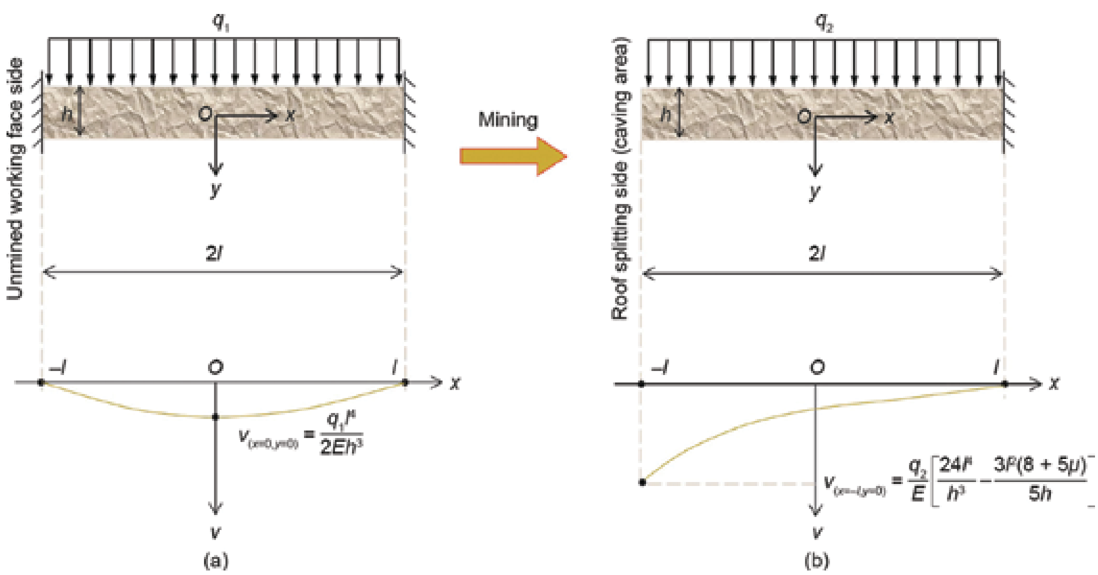

Figure 10: Mechanical state evolution of the retained entry roof. (a) Initial state before mining; (b) working face roof-caving state after mining. q1 and q2: the uniform loads on the fixed rock beam and the cantilever beam, respectively; E: the elastic modulus of the rock mass; l: Poisson's ratio of the roof rock mass; h: the height of the rock beam; v: the axis deflection of the rock beam.

sandstone; it was thus a hard rock roof. The upper roof and the floor of the entry were sandy mudstone with medium strength. The mining panel was 200m wide along the dip direction and 890m long along the strike direction, so the retained length of the 6302 tail entry was 890m.

4.2. Field methods and designs

4.2.1 CRLD support design

During the course of the application of this technology, the structural conditions of the retained entry roof varied considerably as the mining panel advanced. The initial roof state, which was free from any mining disturbance, was the most stable. According to the principles of this technology (Section 2), DPB was applied before mining, and divided the entry roof and the gob roof. As the coal was extracted, the retained entry roof lost the mining side support and hung temporarily, due to the spatiotemporal behavior of the gob roof caving. After the broken rock expanded sufficiently, the entry roof touched the gangue and acquired natural support. Therefore, the entry roof in the gob roof-caving area was the least stable over the entire process. To facilitate analysis of the mechanical roof states, we established rock beam models without considering the support conditions, as shown in **Figure 10**. In its most stable state, the rock beam is regarded as a fixed beam; in its least stable state, it is regarded as a cantilever beam. Taking the rock roof dimensions as *2l* long, *h* deep, and 1m wide, the elasticity solutions of the axis deflection (*v*) for both models were respectively calculated as follows:

Equation 1

$$v_1 = \frac{q_1}{2Eh^3}(x^2 - l^2)^2$$

Equation 2

$$v_2 = \frac{q_2}{E}\left[\frac{6}{h^3}\left(\frac{x^4}{12} + \frac{x^2 l^2}{2} + \frac{x^3 l}{3}\right) - \frac{3}{20h}(8+5\mu)(x-l)^2 - \frac{14xl^3}{h^3} + \frac{17l^4}{2h^3}\right]$$

where *q*1 and *q*2 are the uniform loads on the fixed rock beam and the cantilever beam, respectively; *E* is the elastic modulus of the rock mass; and *μ* is Poisson's ratio of the roof rock mass.

In the initial state before mining, maximum deformation of the entry roof occurred in the middle. Shortly after mining, the maxi mum deformation was transferred to the edge on the roof splitting side; the configurational freedom of the rock roof increased.

Based on the above analysis, CRLD anchors were first installed to protect the entry roof. A row of CRLD anchors at intervals of 1m was installed on the roof splitting side, and the anchors were connected with W-steel belts for cooperative control. In addition, a row of CRLD anchors at intervals of 3m was installed on the middle of the roof to reinforce the original support. As shown in **Figure 11**, two rows of CRLD anchors and W-steel belts were added based on the original support. Since the burial depth of the entry was comparatively shallow, the mine pressure was not great. The specifications of the CRLD anchor were 300 mm free length and 25 t constant resistance.

4.2.2. DPB design

DPB in the field is designed to separate the roof and make the caved gob roof compensate for the mining void. Therefore, the DPB height and angle in the roof should be specifically designed based on the conditions of the site. First, the DPB height H should satisfy the following:

Equation 3

$$H \geq \frac{m}{(k_b - 1)\cos\theta}$$

where m is the mining height, kb is the bulking factor of the rock roof, and h is the DPB angle.

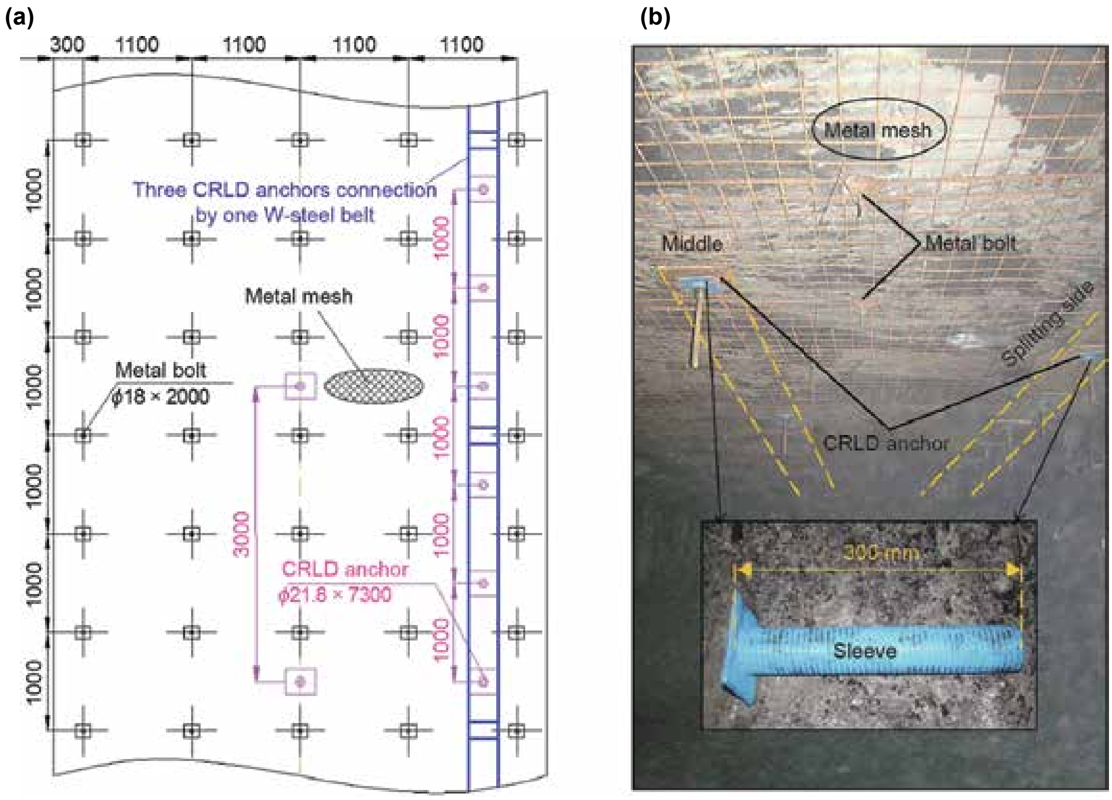

Figure 11: Retained entry roof support design. (a) Unfolded drawing of the roof support (unit: mm); (b) field scene. W-steel specification: 2400 mm x 280 mm x 4 mm. U is the diameter of the steel strand of bolt or anchor.

The DPB angle is set to make the rock roof within the DPB range collapse effectively and rapidly, so that the caved rock material becomes the entry rib and expands quickly to support the cantilever rock beam. The caved rock material within the DPB range is located on the lower roof and fails in a sliding manner. According to the instability principle of a voussoir beam[32], the sliding instability of the interacted rock beams occurs when

Equation 4

$$\theta \geq \varphi_{\mathrm{f}} - \arctan\frac{2(h_0 - \Delta S)}{L}$$

where φ_f is the rock friction angle, h_0 is the rock block height, DS is the rotational subsidence of the rock block, and L is the rock block length.

Substituting the related geological parameters of the field surrounding rock into **Equation 4**, where φ_f = 30°, h_0 = 3.78m, ΔS = 1.6m and L = 15.5m, we obtained θ 14.28°. The greater the angle is, the higher the DPB length will need to be. The practical DPB angle was determined to be 15°. Therefore, $H \geq$ 4.73m was obtained according to **Equation 3**, where m = 1.6m and k_b = 1.35. Considering the roof strata relation from **Figure 9** and the operability, the practical DPB height was determined to be 5m. After determining the DPB borehole length, the charging parameters and the hole distance needed to be designed. The charging parameters are generally determined by site tests for the optimal charge quantity. The ultimate charge structure was tested to be a "3 + 2" pattern, in which emulsion explosives with a unit length of 300mm were arranged in a decoupling air-spacing way; the detonation mode was serial blasting. The distance between bore-holes can be derived by the following[33]:

Equation 5

$$d \leq 2r_{\mathrm{b}} + 2r_{\mathrm{b}} \left(\left[\frac{\rho_0 D_{\mathrm{j}}^2 \lambda n \xi}{2(1-D)(\gamma_{\mathrm{b}}+1)(\sigma_{\mathrm{t}} + \lambda \gamma H_{\mathrm{b}})} \right] \left(\frac{l_{\mathrm{e}}}{l_{\mathrm{b}} c^2} \right)^{\gamma_{\mathrm{b}}} \right)^{\frac{1}{2+\lambda}}$$

where d is the hole distance; r_b is the hole radius; ρ_0 is the explosive density; D_j is the detonation velocity; λ is the coefficient of the side pressure; n is the enhancement coefficient of detonation products; ξ is the energy-focusing blasting coefficient; D is the damage variable of rock mass; γ_b is a constant related to explosive property and charging density; σ_t is the rock static tensile strength; γ is the rock density; H_b is the buried depth of the rock; l_e is the summation length of explosives; l_b is the length of charging segments; c is the ratio of the diameters of the borehole and the explosive.

Fine sandstone accounted for the majority of the rock within the 5m DPB height (**Figure 9**), and the tensile strength of the sand-stone was greater than that of the sandy mudstone (the remainder of the rock mass within the DPB range). Therefore, according to the physical-mechanical properties of the sandstone and the used explosive specification, $d \leq$ 518.77mm was calculated by substituting the following parameters into **Equation 5**: r_b = 24mm, ρ_0 = 1200 kg·m^{-3}, D_j = 3600 m·s^{-3}, λ = 2.6, n = 10, ξ = 2, D = 0.7, γ_b = 3, σ_t = 2.6 MPa, γ = 25 kN·m^{-3}, H_b = 60 m, l_e = 1.5 m, l_b = 3.0 m, and c = 0.75.

The DPB design overview is shown in **Figure 12**. Note that the designed height of the CRLD anchor should be greater than the DPB height, and the difference is generally 2-3 m, which allows the fixed length (**Figure 3**[25]) to be free from the blasting influence. In addition, the CRLD anchor can firmly hang the immediate roof of the retained entry on the thick and strong upper rock formation.

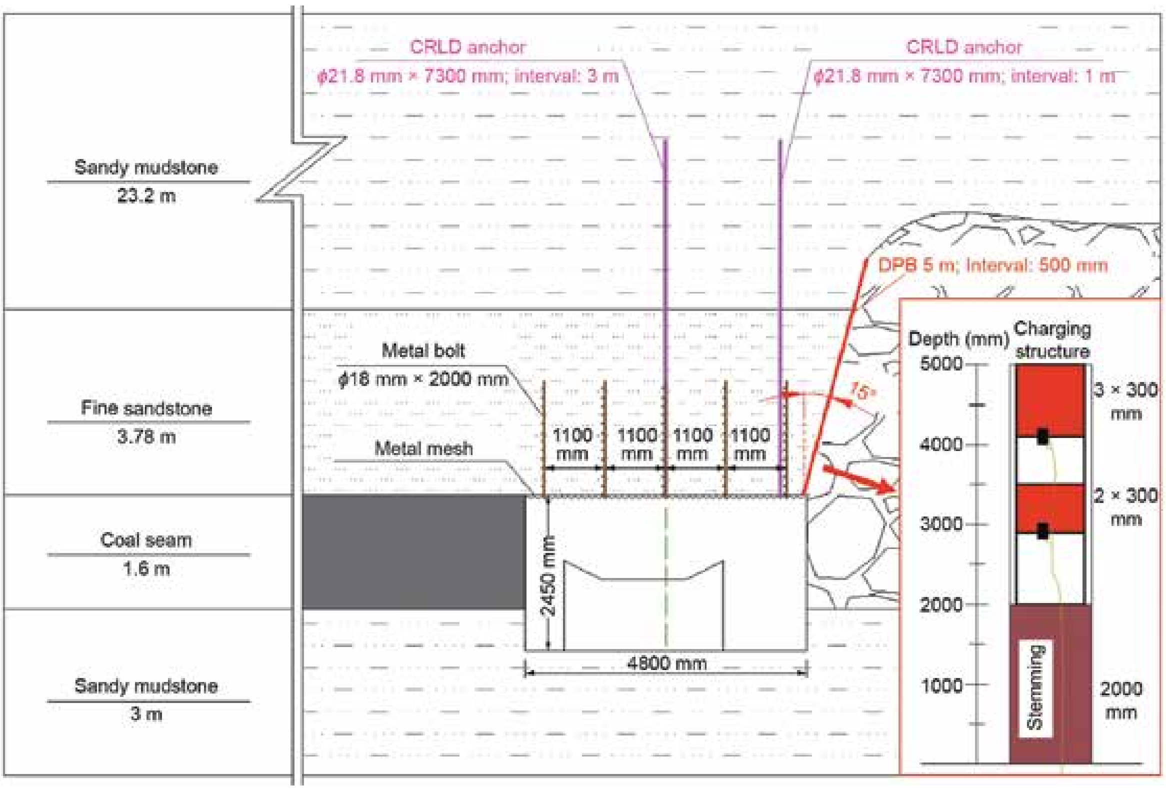

Figure 12: Section diagram of the DPB retained entry and DPB design.

4.2.3. BGSS design

The anti-impact self-advancing structure connected with the face-end support was made of deformable steel with a length of 6 m and a height of 1.5m. This structure was behind the metal mesh and prevented the metal mesh from being damaged by the caving gangue. The 100mm x 100mm metal mesh was used to integrate the gangue wall and prevent the small gangue from thrusting into the entry. The sliding-yield structure was made of overlapped double U steel, which nicely adapted to the vertical deformation and withstood the horizontal deformation. Therefore, the double U steel used for long-term support could be used again to serve the next entry after the retained entry was abandoned. The auxiliary supporting structure comprised a hydraulic prop. As the temporary support, the hydraulic props were set in the dynamic pressure-bearing zone, which stretched for an empirical length of 150-200m behind the working face. Thus, the props were used and reused as the working face advanced. The hydraulic prop and double U steel were placed in a staggered arrangement at intervals of 500 mm. The BGSS design for the field is shown in **Figure 13**.

4.3. Field monitoring

4.3.1. CRLD support effect

As discussed in Section 4.2.1, the entry roof would deflect the most on the roof-splitting side. Choosing to place the CRLD anchor on the splitting side, we monitored the CRLD anchor stress and its retraction value during the face mining. The stress was monitored in real time by a YAD-200 vibrating-string cable dynamometer that is manufactured by Shandong University of Technology Zhongtian Safety Control Technology Co., Ltd., China, and the retraction value was measured continuously with a Vernier caliper. The dynamometer outputted the anchor loads R_i based on the calculation formula without regard to temperature change:

Equation 6

$$R_i = G(f_0 - f_i)$$

where G is the apparatus coefficient, f_0 is the initial frequency modulus of the vibrating string, and f_i is the real-time recorded frequency modulus of the vibrating string.

The comprehensive performance of the CRLD anchor is shown in **Figure 14**. A preloading force no less than 250 kN was first applied to the CRLD anchor during the installation. At the initial stage of the monitoring area (-20 to 3m), when the position of the CRLD anchor ranged from 20m ahead of the mining face to about 3m behind the mining face, the CRLD anchor was in the constant-resistance state and no retraction occurred; the output stress fluctuated smoothly. After that, the stress vibrated dramatically, and the vibration amplitude decreased as the mining face advanced. Meanwhile, the retraction value increased rapidly; the constant-resistance body slid accordingly. When the mining face was about 52m ahead of the CRLD anchor, the anchor stress leveled off again; the corresponding retraction value no longer increased significantly. The recorded ultimate retraction value stabilized at 28 mm. As seen from the above phenomena, the most intense activity period of the retained entry roof occurred within 60m behind the working face. The CRLD anchor was well adapted to the large deformation of the entry roof and displayed superior energy absorption while resisting roof sagging.

4.3.2. DPB effect

The CXK-6 borehole imager, which is manufactured by Wuhan Conourish Coalmine Safety Technology Co., Ltd., China was used to observe the formation effect of blasting cracks to optimize the charging parameters and supervise the blasting quality. As shown in **Figure 15(a)**, two clear directional cracks

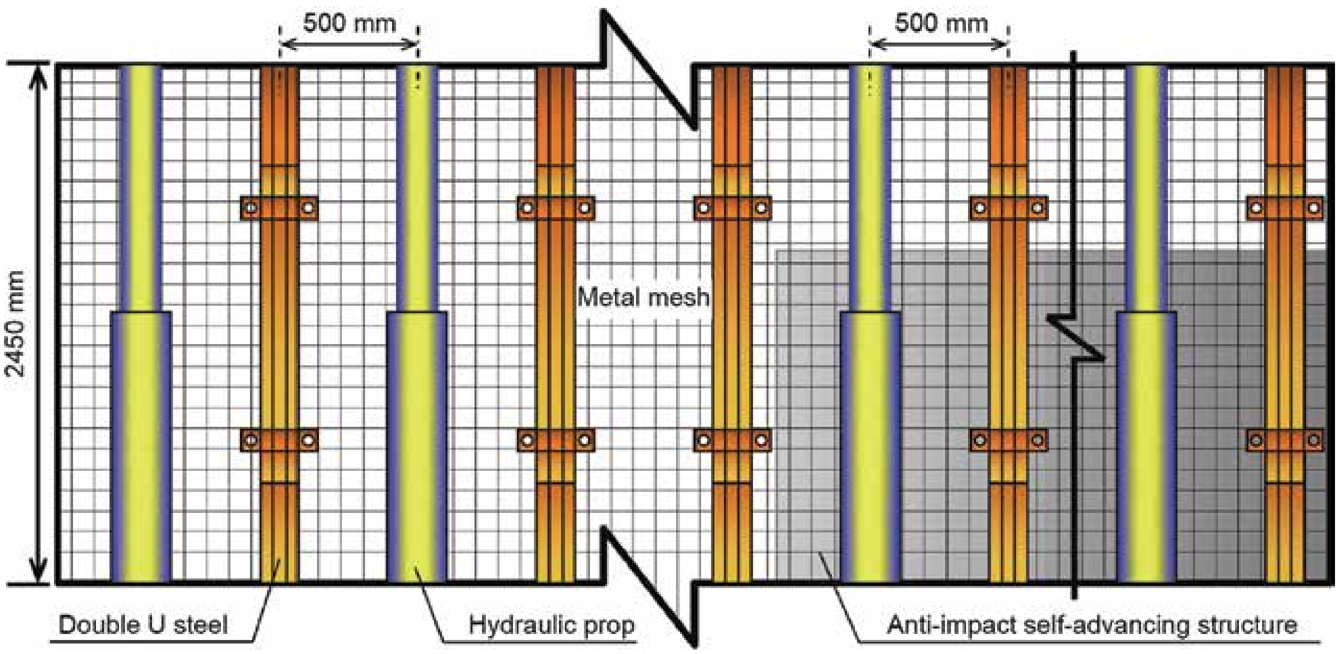

Figure 13: BGSS design in the field.

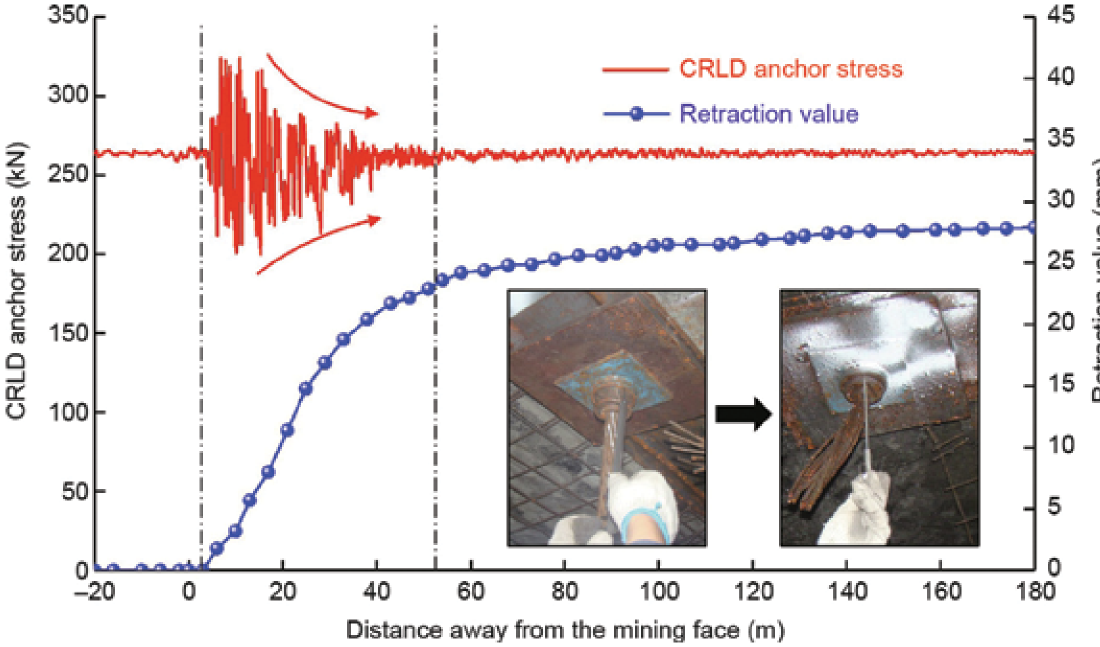

Figure 14: CRLD anchor support performance.

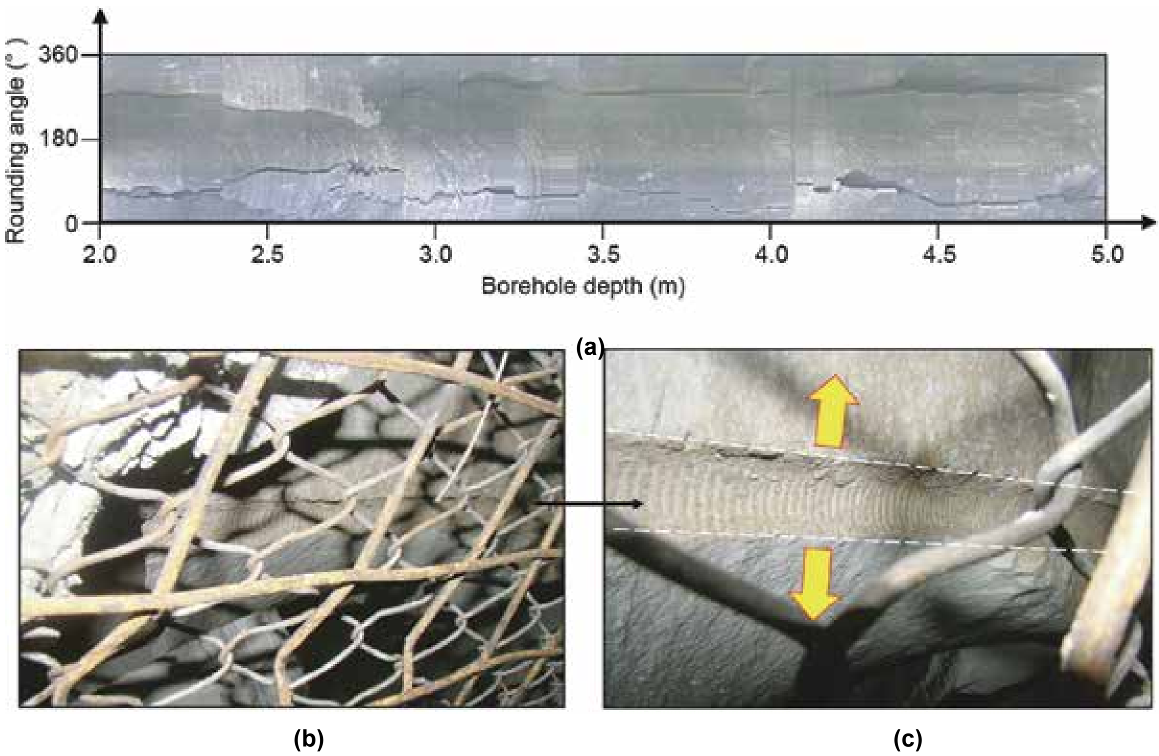

Figure 15: Directional blasting effect. (a) Borehole imaging; (b) half-hole in the caved zone; (c) enlarged view of the half-hole.

were generated during the charging passage (for a borehole depth of 2-5 m) by using the "3 + 2" pattern described in Section 4.2.2. In addition, the DPB boreholes should be subjected to a spot check of the crack ratio (the crack length divides the borehole length), which is expected to be higher than 60%. The crack ratio in the field was 74% under a random check every 200 boreholes (i.e., every 100m along the entry strike direction). As shown in **Figures 15(b)** and **(c)**, the DPB effect could be observed from the automatically formed entry rib. DPB separated the roof along the borehole line, and the half-hole on the gob roof side would cave in after mining. We captured the rock fracture plane with the blasting hole. The half-hole was left in the gob area; no other apparent cracks were formed on the borehole surface, and the fracture plane was smooth. These occurrences demonstrated the expected DPB application effect.

4.3.3. The BGSS effect

To ascertain the change in lateral gangue pressure applied on the BGSS as the working face advanced, we set the pressure gauge behind the double U steel to record the pressure change, as shown in **Figure 16**. In the rising stage, the pressure appeared after the working face had advanced by 4.2m. The pressure then slightly increased from 0.33 to 0.38 MPa between the advanced distances of 4.2-8.2m, while the pressure climbed fast from the distance of 8.2m; the maximum pressure was 1.63 MPa at 45.2m. These results indicated that the hard roof caving had an obvious space lag when the working face was pushed away. The anti-impact structure worked and decomposed the impact force from the gob roof first caving, which corresponded to the early slow increase period. After this period, the anti-impact structure moved forward and the upper roof collapsed in layers, which led to a rapid pressure increase. In the falling stage, the pressure declined after the impact motion of the upper main roof, and then gradually became stable at about 1.22 MPa at around 96m. At this point, the compacting gangue was basically stable.

The retained entry segment behind the working face was divided into two parts: the dynamic pressure-bearing zone and the entry stabilization zone. In the dynamic pressure-bearing zone, the double U steel and the hydraulic prop were alternately spaced, as shown in **Figure 17(a)**. The caved rock material was blocked to form the integrated entry rib by the BGSS. As the working face advanced, the segment that was previously in the dynamic pressure-bearing zone would enter the entry stabilization zone, and the props therein would be retracted to support the next dynamic pressure-bearing zone. As shown in **Figure 17(b)**, the retained entry in the stabilization zone was already steady, and the double U steel did not have apparent lateral deformation after the prop retraction. The final automatically formed entry rib met the production and safety requirements.

4.3.4. Mining pressure

By processing the pressure data of the working face supports, we obtained the three-dimensional (3D) distribution of the working face mining pressure, as shown in **Figure 18**. The mining pressure was minimal on the retained entry side. The pressure increased along the dip direction of the working face, and the maximum pressure was on the non-retained entry side. These results demonstrated that, due to the DPB design and construction, the caved rock material fully expanded to become a natural supporting body and restricted the increase of mining pressure. This pressure reduction effect decreased along with the increasing distance away from the DPB position. Therefore, the maximum pressure was located on the non-retained entry side, where the entry was under the conventional mining way. Thus, the retained entry faced less mining pressure in this non-pillar coal-mining technology than it would have under traditional mining technology, which was very beneficial to the entry stability.

4.3.5. Retained entry stability

The roof-to-floor displacement of the retained entry is a direct index reflecting the entry stabilization. We set displacement-monitoring points in the retained entry in order to observe the roof-to-floor convergence change. As shown in **Figure 19**, the displacement gradually went from rising to stable as the working face advanced away from the measuring point. The displacement rising

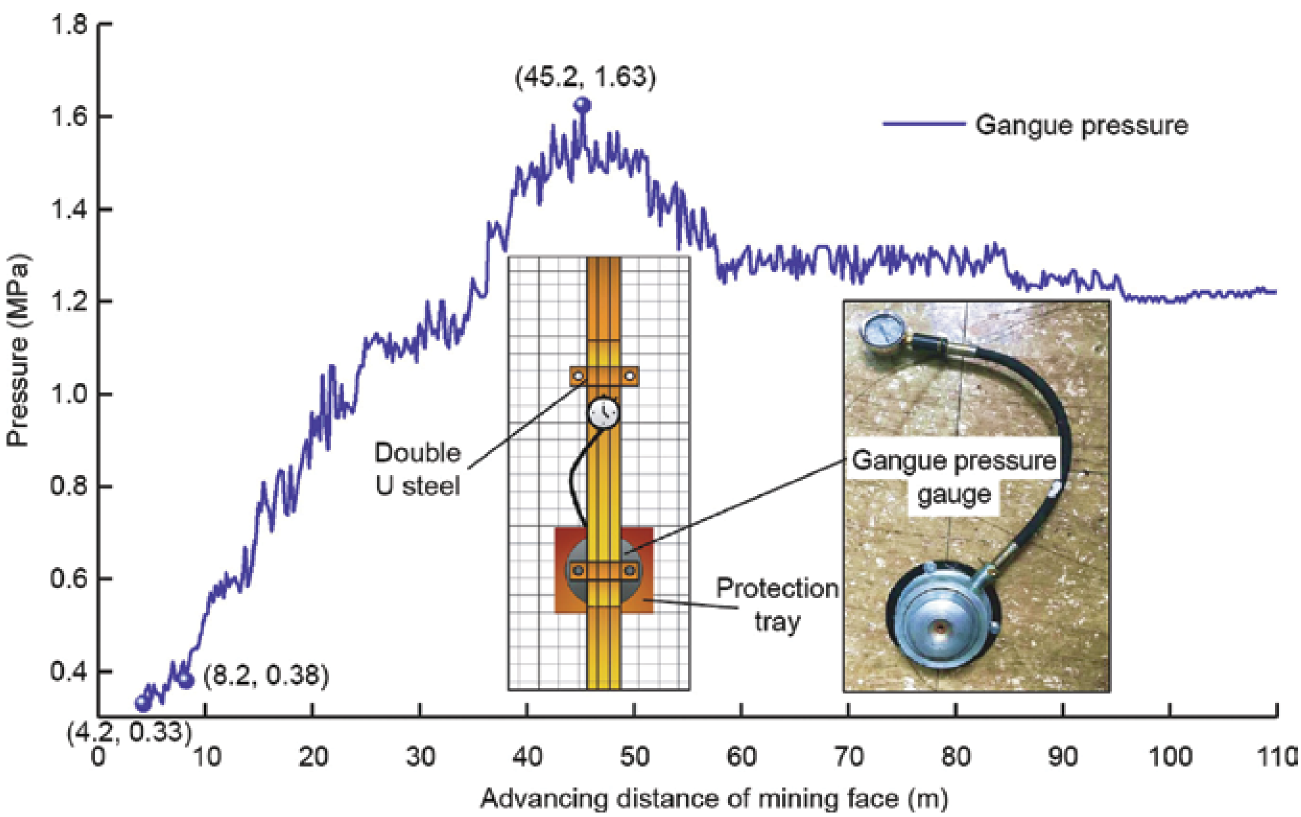

Figure 16: Gangue pressure monitoring curve.

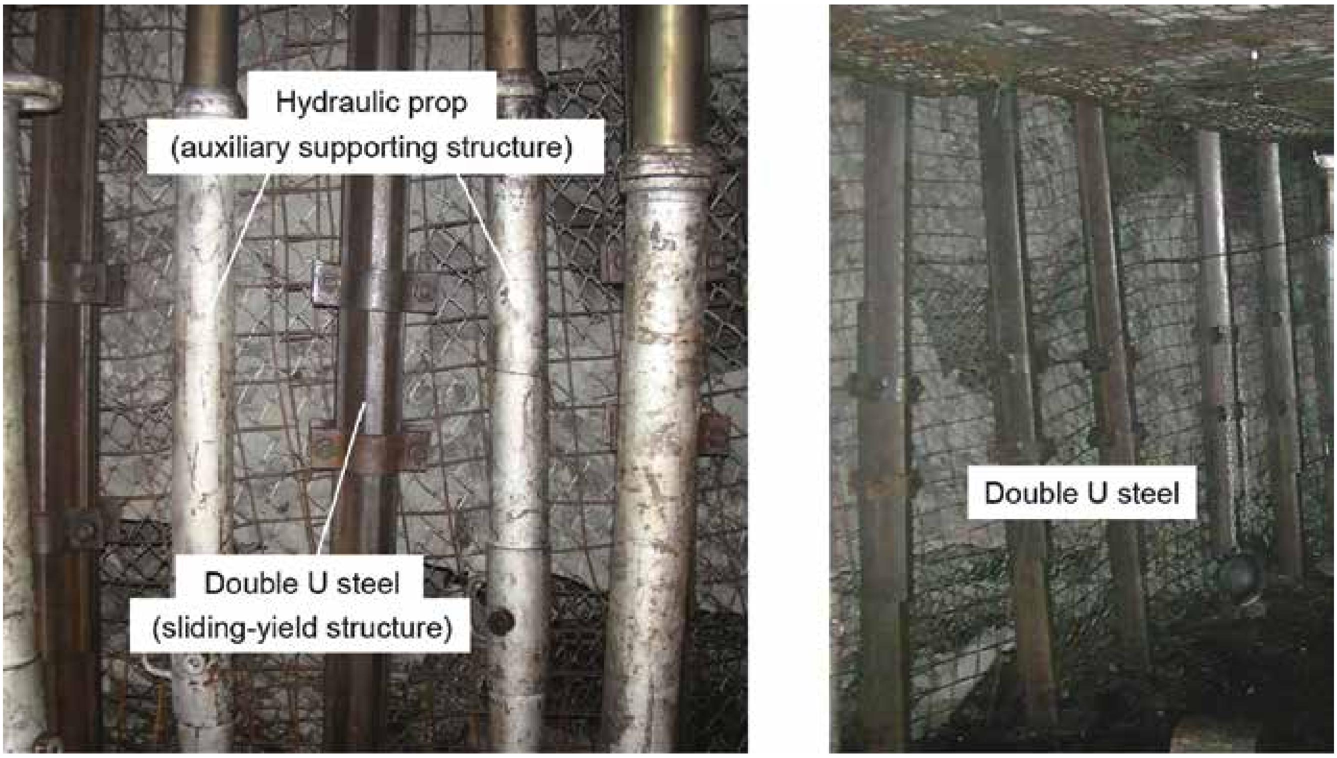

Figure 17: Field application effect of the BGSS. (a) Before the prop retraction; (b) after the prop retraction.

phase was defined as the dynamic pressure-bearing zone, and the phase when the displacement tended to be stable was defined as the entry stabilization zone. In the dynamic pressure-bearing zone, the retained entry was close behind the mining face. Because of the mining disturbance and the gob upper roof movement, the retained entry was subjected to converging forces and generated convergence. Therefore, we set the temporary support (hydraulic props matching up with the lace girder) in this zone to reduce the entry displacement and promote a transition to the stable stage. The monitoring result showed that the stabilization distance was around 148m on site. When the entry was entering the stable stage, the props could be withdrawn and utilized to support the newly formed dynamic pressure-bearing zone. In the following field operation, the prop retraction distance at the site was set at 160m to be on the safe side. With such a retraction circulation, the whole 890m entry in the field was retained successfully. The roof-to-floor displacement stabilized at 212mm, which met the entry retaining and mining production requirements.

4.4. Problems of retained entry

During the field application of the technology for a medium-thickness coal seam with a hard roof, a few sections along the retained entry showed a coal rib spalling and a rock arch hanging in the gangue rib, as shown in **Figure 20**. First, the coal rib spalling of the retained entry increased the entry span, which led to an increase of the unsupported area of the entry roof and a decrease of the roof safety factor due to the long service life of the entry. From the investigation and analysis, we considered that, in this segment, the poor connectivity of the directional cracks caused the entry rib to bear more load from the movement of the cantilever rock beam. When the gob roof caved along the poor presplitting plane, the entry roof had to overcome the cohesive force in the uncut positions. Because the roof rock was hard, the entry roof deflected more and thus squeezed the coal rib, causing the spalling. Another problem of the hanging rock arch was a potential threat to entry safety. We believed that there were two reasons behind that: First, DPB did not cut apart the roof effectively; and second, the fractured lumpiness of the rock roof was large and the rotary space was small due to the site conditions of a hard roof and a medium-thickness coal seam. When the inside edge of the big rock touched the floor after its small rotation, this rock was balanced by another edge of the friction on the presplitting plane. This balance would be easily broken when the next mining face approached. Once the big rock fell, the regional gob-side support might be destroyed, which would be a threat to the moving workers. However, adjusting the measures to the site conditions solved these problems: The connectivity rate between the boreholes was elevated, the entry coal rib was supported, and the loose blasting boreholes next to the DPB boreholes were increased in order to decrease the rock lumpiness.

5. DISCUSSION

Pillar-less longwall mining first began to appear in the 1950s. The conventional method is to build the gob-side pack; this method has been tested and employed under many geological conditions for different mining depths[34,35], coal-seam thickness[36,37], and roof lithology[38-40]. As a pioneering approach in pillar-less longwall mining, this conventional approach has been developed considerably over the past decades. However, some inherent disadvantages have emerged. Long-term roof movement disturbances make the retained entry difficult to maintain under the conventional approach[41]. Furthermore, the application conditions are

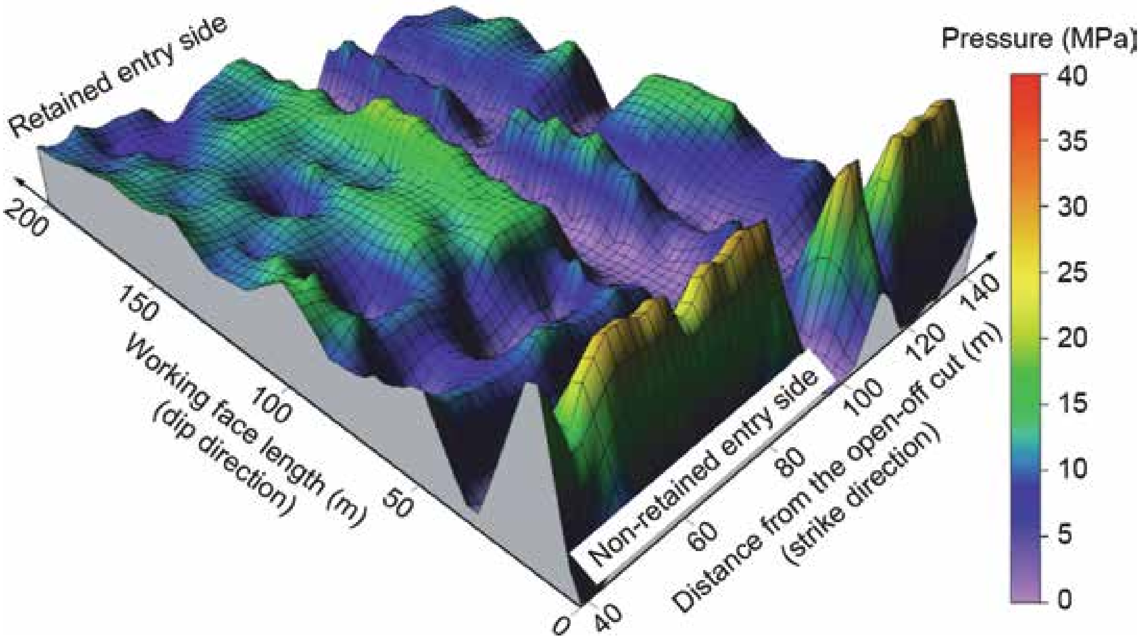

Figure 18: 3D nephogram of the mining pressure.

limited; for example, this method has bad adaptability in the case of a hard roof[42]. These problems are becoming intractable as mining depths increase[43]. Therefore, it is complicated to apply the conventional approach in practice[44]. In addition, building the gob-side pack by means of construction or material filling has a high cost in both workforce and material resources, and introduces potential delay that reduces the efficiency of longwall mining.

This study presents the innovative technology of non-pillar longwall mining with an automatically formed entry, which uses the self-bearing ability of the gangue to relieve the mining pressure and form a natural entry rib. Based on the proposed design principles, three key techniques (CRLD, DPB, and BGSS) cooperate to retain the gob-side entry efficiently and safely. Some problems may be encountered during actual application due to different geological mining conditions. The ideal application effect can be achieved by adjusting the measures to the specific site conditions. Coal distribution and mining conditions are complicated[45].

For potentially disastrous mines (e.g., those with risk of rockburst, coal and gas outburst, and mine water disaster), this innovative technology has good application potential, although it still needs to be further studied and tested in the field.

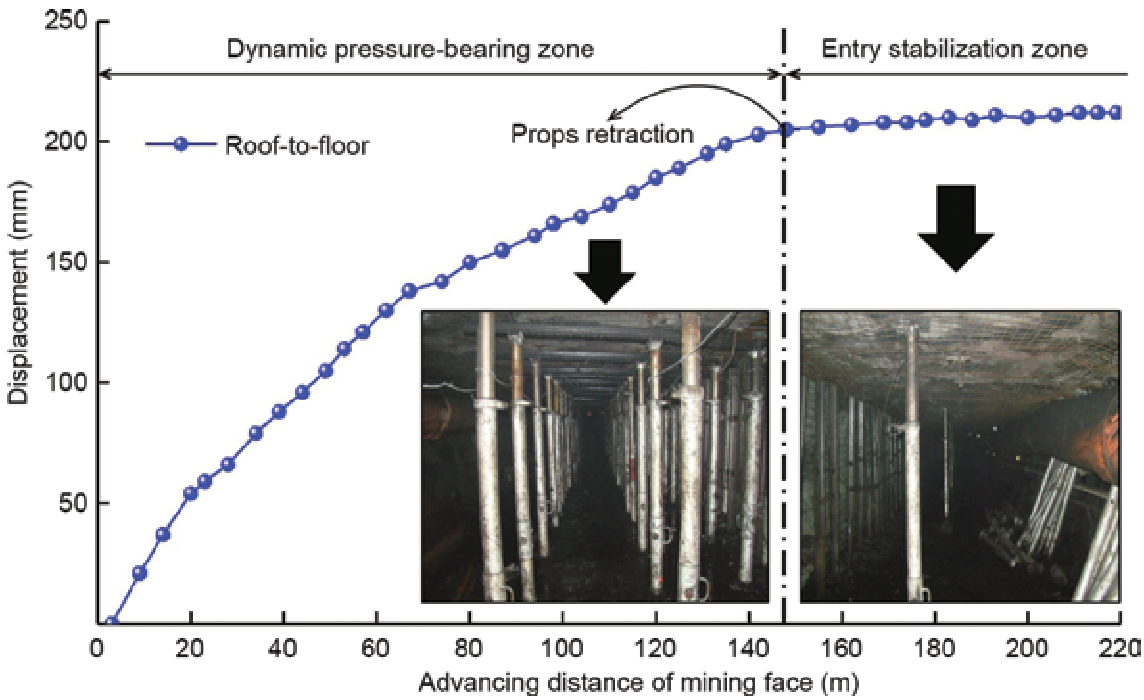

Figure 19: Roof-to-floor displacement and retained entry effect.

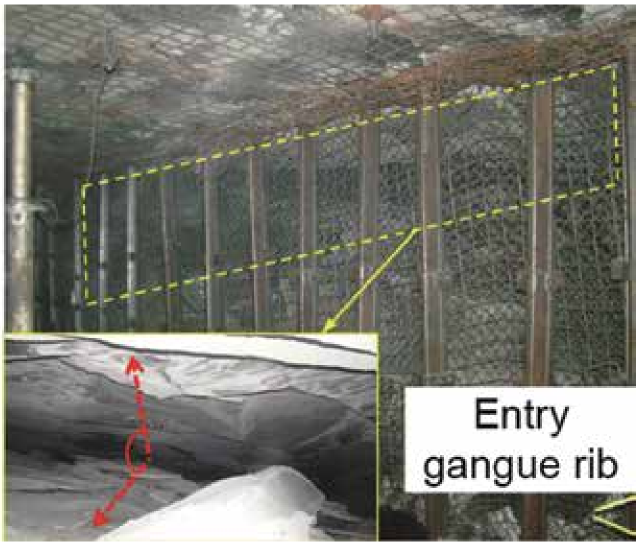

Figure 20: Problems during the entry retaining. (a) Coal rib spalling; (b) rock arch hanging in the gangue rib.

This innovative non-pillar coal-mining technology takes the proposed principles as the framework and requires the adjustment of some measures to different site conditions to ensure the quality and safety of the retained entry. This technology has good engineering applicability and promotion value.

6. CONCLUSIONS

A non-pillar coal-mining technology with an automatically formed entry was studied, which reduces the waste of coal resources and entry excavation. Three key techniques involved in the technology were introduced, namely: CRLD support, DPB, and BGSS. These three designs and their effects were investigated by means of field application. Entry in the field was retained successfully, validating the engineering applicability and promotion value of this technology. The primary conclusions are as follows.

The CRLD support accommodated the large roof deformation with high resistance during the retained entry service life. The roof on the DPB side was the focal supporting area, and the CRLD anchor there had an outstanding effect within 60m behind the working face. DPB needed to be designed at a certain height and angle to make the gob roof collapse quickly and effectively. DPB application generated a directional crack and separated the roof between the retained entry and the gob. When the working face moved away, the gob roof caved in along the design position. The BGSS was designed into the anti-impact self-advancing structure, the sliding-yield structure, and the auxiliary supporting structure, which were well adapted to the mining pressure. The BGSS application integrated the gangue into an effective entry rib. The monitoring result showed that the gangue rib tended to be stable when the mining face advanced to a distance of 9m.

The stable expanded gangue became the natural supporting body and decreased the mining pressure on the gob side. The retained entry under this pressure-relief circumstance did not deform much; it then entered the entry stabilization zone, where temporary support could be withdrawn. The onsite stabilization distance was 148m. The gob-side entry was successfully retained in the field and non-pillar mining was realized. The retained entry quality is the critical factor in non-pillar mining technology. Adjusting the measures according to different mining geological conditions can improve the retained entry quality and make this technology universally applicable.

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COMPLIANCE WITH ETHICS GUIDELINES

Xingyu Zhang, Manchao He, Jun Yang, Eryu Wang, Jiabin Zhang, and Yue Sun declare that they have no conflict of interest or financial conflicts to disclose.

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Xingyu Zhang[1,2,3], Manchao He[1], Jun Yang[1,2], Eryu Wang[4], Jiabin Zhang[1,2], Yue Sun[1,2]

1 *State Key Laboratory for Geomechanics and Deep Underground Engineering, China University of Mining and Technology, Beijing 100083, China*

2 *School of Mechanics and Civil Engineering, China University of Mining and Technology, Beijing 100083, China*

3 *Department of Civil, Environmental and Architectural Engineering, University of Colorado, Boulder, CO 80309, USA*

4 *Institute of Mining Research, Inner Mongolia University of Science and Technology, Baotou 014010, China*

Asia could be where the future of coal is decided

Recent figures have not been kind to the coal industry, which in many countries has been in consistent decline over the last few years. Cheaply priced competitor energy sources such as natural gas as well as subsidised solar and wind energy boosted by growing global concern over coal's significant impact on climate change have led to coal falling out of favour as a natural resource across much of the industrialised world. And with the onset of the coronavirus only expediting coal's decline, many are left wondering whether it has now entered its final days as a viable source of energy.

USA

The United States is among those countries experiencing such a decline. With warm weather pushing natural-gas prices lower, oil prices hitting historic lows and the concerted shift towards wind and solar power gaining steam, coal has become gradually more excluded from the mix of fuels that generate energy in the country. Indeed, the US recently saw the demand for electricity from coal-fired power plants drop to new lows. According to a recent analysis by consultancy Rhodium Group, coal's contribution to total US power generation has fallen to just 15 percent – the lowest level in modern history. And for the first time, Rhodium noted, wind and solar together generated more electricity than coal on certain days.

IT IS ECONOMICS, NOT POLICY, THAT IS HURTING THE INDUSTRY

Despite promises from President Donald Trump's administration to rescue the US coal industry, mainly through the reversal of environmental regulations implemented by the previous administration and by taking a distinctly more conservative position towards climate-change action, 39,000 MW of coal-fired power-plant capacity has closed down. According to Reuters, if this trend continues, more coal plants will have shut their doors during Trump's first term (2017-20) – around 46,600 MW – than during President Barack Obama's second term (2013-16 – approximately 43,100 MW. This trend is expected to continue as the economy worsens. It will simply be that renewables and gas will keep their market, and coal, being the more expensive fuel, is going to get pushed out even more than it would have liked.

The industry is not also likely to prosper under the Biden administration, but it was not exactly gang busters under the previous ones despite all the promises. There is a real gap between rhetoric and reality in the industry. The rhetoric was that the administration was going to help the industry, The reality was that by pursuing what you might call an all-of -the-above energy strategy, in particular one focused on oil and gas, that actually hurt the industry

SAME TRENDS ALL OVER THE WORLD

The same trends are broadly observed across much of the rest of the world, moreover. The Paris-based International Energy Agency (IEA) puts coal demand at 8 percent less during this year's first quarter 2000 compared to first-quarter 2019, with such a significant drop attributed to lower demand in the electricity sector, in which two-thirds of coal is consumed. With governments globally imposing social-distancing and stay-at-home directives to combat the spread of the coronavirus, coal demand from power plants has been waning. In China, where more than half of the world's coal is consumed, the COVID 19 outbreak triggered a marked decline in coal demand because coal supplies 60 percent of primary energy and an even higher share of electricity

Australia, the world's second-biggest thermal-coal exporter, meanwhile, has experienced a particularly challenging environment of late, especially when it has come to obtaining funding for coal projects. With financial institutions under increasing pressure to divest from fossil-fuel activity all around the world, Australian banks have drastically reduced their exposures to the domestic coal industry. Most recently, Westpac, one of the country's biggest lenders, announced that it will exit the industry by 2030, following in the footsteps of fellow banking majors Commonwealth Bank of Australia (CBA) and National Australia Bank (NAB), which will cease all thermal-coal financing activity by 2030 and 2035, respectively. Westpac has already reduced its coal exposure to AUD$700 million while aiming to lend AUD$3.5 billion to new climate-change solutions over the coming three years. Among the country's banking conglomerates, only ANZ (Australia and New Zealand) Banking Group remains willing to maintain investments in the coal industry. And although it has not set a formal date to exit the sector, it has signalled that it will reduce its involvement in the sector.

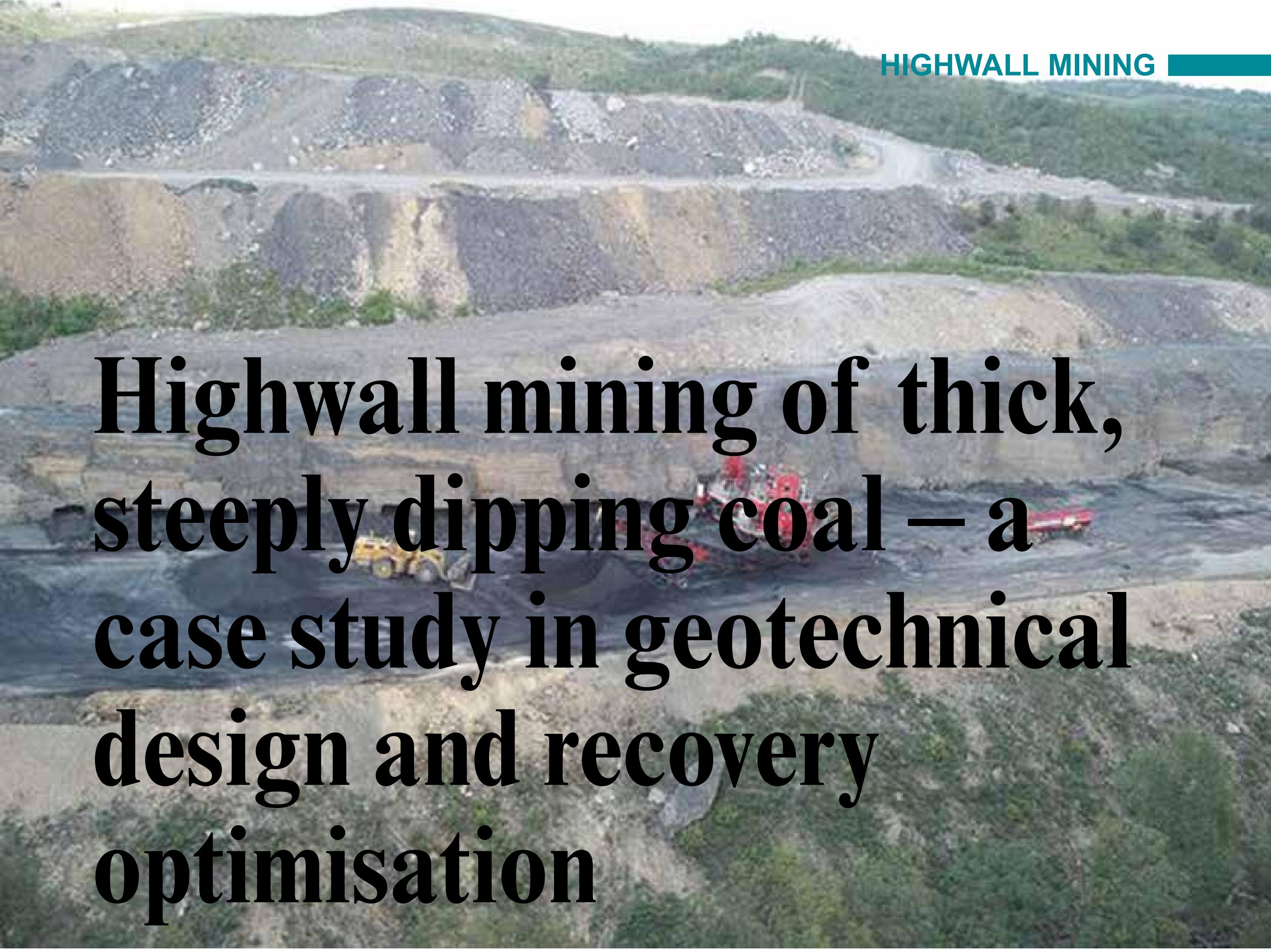
The most immediate question, however, is whether the coronavirus represents a temporary problem for coal, one from which it can even modestly recover as the economy rebounds, or if it is another nail in the coffin of this long-declining industry. The current downturn could slow coal's projected turnaround, That said, many are expecting natural-gas prices to increase this year on the back of slowing oil drilling, which should cut the amount of gas produced from oil wells. If this materialises, demand for coal may well turn a corner once more. While the writing may well be increasingly on the wall for coal, the commodity remains crucial for power generation across other major economies in the world. And that situation will not be changing for the foreseeable future and Asia could be where the future of coal is decided.

ASIA

Coal remains a cornerstone of electricity generation in China, India, and other Asian nations, which together account for around 75% of global coal demand. As more players in the EU and US switch away from coal, their behaviour will have less of an impact on the coal industry's future. Left unchecked, Asia's coal consumption growth could have serious implications for the planet. There remains regional imbalances, with economies like China and India offsetting declines in the United States, the European Union, and other regions.

Whilst these imbalances remains there will always be a demand for coal. Getting the whole world to agree on fossil fuel reductions is a highly unlikely scenario.

Miners in the USA just want to be part of the conversation.



Highwall mining of thick, steeply dipping coal – a case study in geotechnical design and recovery optimisation

Highwall mining of thick (up to 30.48 m) steeply dipping (20° or more) coal seams provides many challenges, both geotechnically and operationally, as seam dips near or in excess of highwall mining machine capabilities are encountered. Maximizing coal recovery while maintaining highwall stability requires innovative techniques with regard to web and barrier pillar layout, depth of penetration, and choice of mining horizon within the seam. Stability of highwall mining slopes, openings, and pillars are typically analysed using the ARMPS-HWM program, as well as LAMODEL, UDEC and Slope-W modelling. Highwall stability can be maintained, and highwall mining production optimized by applying design criteria in creative ways, including alternating miner penetration depths, and initiating mining of thick seams toward the bottom of the seam. Highwall mining of thick, steeply dipping coal requires careful planning and execution, including close cooperation between geotechnical design engineers, the mining company, and the highwall mining contractor. This paper describes the application of creative design techniques to a specific pit arrangement at the Westmoreland Kemmerer Mine, Kemmerer, Wyoming. Highwall mining was accomplished by UGM ADDCAR Systems, LLC on a contract basis.

INTRODUCTION

Highwall mining is a technique for attaining additional coal recovery after the economic strip limit is reached in surface mining. It involves remote deployment of a continuous miner in openings beneath the final highwall, with no personnel entry. Many candidate areas for highwall mining have thick and steeply dipping seams. Mining down dip presents challenges related to the machine's maximum pulling capacity, traction of the cutting head, and material conveying, all of which limit penetration depth. Maximum penetration is greater for flatter slopes and decreases for slopes nearing the threshold of the maximum machine operating angle. Most highwall mining operations are relatively flat with slight undulations within the seam; therefore, the highwall mining pillar design criteria applies fairly equally to the entire mining area. However, in steeply dipping deposits, design criteria based on higher overburden loads at the far end of the penetration are excessively conservative for the shallower portions of the openings near the highwall.

Thick seams allow for more flexibility when designing for maximum recovery. Holes can be angled across the seam dip, thereby reducing the gradient of the openings and increasing the maximum penetration. Multiple passes from the same hole can increase the mining height as long as they remain aligned and do not encroach into the web pillars. If the seam is thick enough, multiple penetrations can be made in different elevations while leaving a sill pillar between the excavations. These down dip operations are susceptible to water inflow, both from underground and surface sources, requiring pumping and/or other control measures.

ANALYSIS METHODS

Highwall mining pillar design is a direct function of coal strength, opening height, opening width, and depth of cover. These inputs are used to create design curves unique to each mine's geologic environment to specify the web and barrier pillar widths necessary to achieve the desired safety factor (SF). Agapito Associates, Inc. (AAI) has been involved with most western United States highwall mining projects performed since the late-1990s. Design techniques have evolved over the years; currently, web and barrier pillar designs are based on the National Institute of Occupational Safety and Health (NIOSH) ARMPS-HWM program[1]. The program uses a modified tributary area loading model and an empirical estimation of coal strength to determine appropriate pillar widths for both web and barrier pillars. AAI then uses the LAMODEL program as a check of the ARMPS-HWM designs and to predict the potential for cascading pillar failure should one or more pillars fail. Typically, one complete panel and portions of the adjacent panels are modelled in LAMODEL with and without a failed central pillar[2]. UDEC is used primarily to model roof conditions, but also provides insight into pillar and floor conditions[3]. UDEC is also used in multi-level designs to evaluate the interaction between stacked openings and the adequacy of sill pillars. Highwall mining has the potential to reduce highwall stability by weakening and raising the stress levels in the mined coal seam, usually at the base of the highwall. AAI uses Slope-W to evaluate overall highwall stability before and after highwall mining by simulating the weakening of the coal seam based on the extraction ratio[4].

The following are the SFs and design criteria typically used in designing a highwall mining excavation area:

A highwall mining panel should not exceed 20 openings between barrier pillars.

Web pillar *SF* ≥ 1.6 for normal operations
Web pillar *SF* ≥ 2.0 for protection of critical structures
Minimum 0.8 web pillar width-to-height (*w:h*) ratio
Barrier pillar *SF* of 1.5 with w:h > 4, 2.0 with *w:h* < 4
Overall panel *SF* (webs and barriers) ≥2.0.

PRODUCTION OPTIMIZATION

There are many challenges in maximizing recovery for very steep and thick seams with moderately dipping highwalls. Penetration of the mining machine is typically limited for steeper operating slopes. For example, the ADDCAR miner routinely achieves maximum penetrations of 365.76 m or more when working on grades of less than 16°[5]. As the grade increases past 16°, the maximum penetration decreases, to an ultimate limit of 182.88 m at a 20° slope.

For thick seams, production may be increased by mining cross-seam, cutting vertically across the seam at a flatter dip, to obtain greater penetration depth. The machine may ultimately contact the seam roof before maximum penetration is achieved; therefore, the opening should be initiated at the base of the seam to optimize penetration. If the seam is thick enough, the machine operating angle may be reduced sufficiently to permit up to twice the penetration depth, resulting in twice the production. **Figure 1** illustrates the expected maximum penetration depths that might be achieved for a 6.10-m excavation height in a 20° seam for different machine inclination sand seam thicknesses. At some inclination, the penetration at which the machine contacts the roof coincides with the limiting penetration for the given inclination and represents the maximum possible penetration (optimum case). At steeper inclinations, the penetration is reduced, due to machine limitations, resulting in less production.

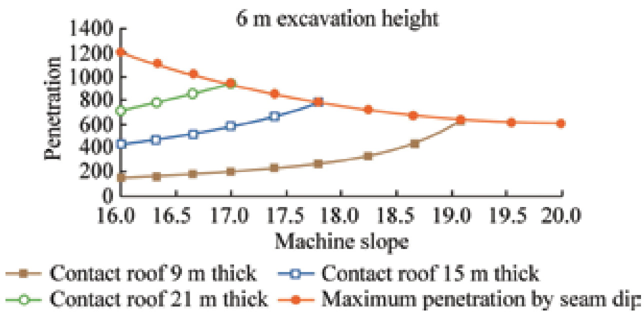


Figure 1: Maximum penetration depths in a 20° maximum for different machine inclinations and seam thickness.

Highwalls are normally mined along the seam strike and therefore, highwall mining openings are oriented and mined downdip, perpendicular to the face. For steep seams, an angled-hole technique permits increasing penetration, and thus production. By orienting the openings at an angle to the highwall, instead of mining directly downdip, the mining gradient can be decreased somewhat resulting in a greater penetration depth. However, the holes must be reoriented at a fairly large angle to realize any significant benefit in a steeply dipping environment, which may not be practical operationally. Additionally, to maintain the required web pillar width, fewer openings are possible for a given pit width, thereby reducing the overall production. For the maximum practical orientation of 15° from the perpendicular, the effective dip for a 20° seam dip is reduced to 18°, permitting approximately 30.48 m of increased penetration. Angled holes also result in a wider roof exposure at the collar, potentially resulting in decreased stability.

Production can also be increased in thick seams by making multiple passes in a single opening to increase the effective mining height or to mine multiple stacked openings. Multi-pass mining has been accomplished for heights of about 8.53 m, and even greater heights are operationally possible. Overall production with a greater mining height is somewhat offset by the requirement for wider web and barrier pillars to maintain stability. For example, overall production for 8.53 m openings would be about 47.8% greater than the production using 4.26 m openings. The

ARMPS-HWM design formula includes a w:h term which reflects the decreased strength of taller (slenderer) pillars. Also, a minimum w:h ratio of 0.8 is normally imposed, based on past experience with instability of pillars having low w:h ratios. With taller openings, the potential for rib spalling also increases, and needs to be considered.

Multi-lift mining requires very thick seams in order to accommodate two (or more) openings and the intermediate sill pillar(s). For single-pass openings, a rule of thumb that AAI has applied is that the sill pillar thickness should be at least two times the height of the openings. As the opening height increases however, this guideline is likely conservative, and sill pillar thickness should be determined through numerical analysis. Depending on the seam thickness, multi-pass mining can be combined with multi-lift mining to increase production. Normally, the recovery lost from having to increase pillar widths associated with higher openings is offset by the increased recovery from the higher openings. Since web pillar widths required for multi-lift mining are only slightly increased versus those of single-lift mining, production from multi-lift mining could double or more, depending on the number of lifts.

For moderately sloping highwalls, production can be increased by implementing an alternate depth mining method. In this method, every other hole is mined to the design penetration, while the holes between the full penetration openings are stopped short. **Figure 2** shows a plan view of the hole layout that was implemented at the mine. For the shorter holes, the depth of cover under the highwall is less than that at full penetration, permitting the use of narrower web pillars at the highwall. The pillar between the ends of the full penetration holes is typically wider than necessary as it is composed of the widths of two shorter penetration web pillars plus the opening width. Although the coal produced from the shorter holes is reduced, the narrower web pillars at the highwall allow more openings to be mined for a given pit width. This increases overall recovery versus a layout in which all holes are mined to the same penetration. AAI has developed algorithms for determining the shorter hole penetration that optimizes recovery for a specific highwall profile.

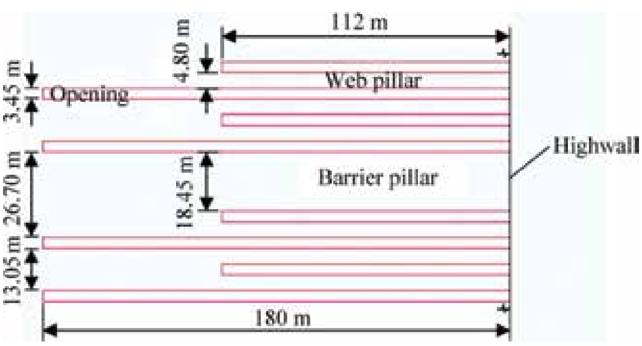


Figure 2: HWM alternate-depth hole layout geometry.

FIELD EXPERIENCE

Using design guidelines developed by AAI and approved by the Mine Safety and Health Administration (MSHA), highwall mining at Westmoreland's Kemmerer Mine began in January 2017 in the 475 Seam in Pit 5 of the 2UD mining area. The 475 Seam has an average dip to the west of approximately 20°, with a maximum dip greater than 24°. The seam thickness in this area averages approximately 7.32 m.

ADDCAR was contracted by Westmoreland to perform the highwall mining excavation in the 2UD area. **Figure 3** shows the ADDCAR launch vehicle stationed at a highwall mining opening. A conventional coal continuous miner excavates the opening and deposits the cut coal onto a series of linked conveyor cars that are positioned and removed from the launch vehicle using a front-end loader. Coal discharged from the cars is conveyed along the base of the launch vehicle to a side-stacking conveyor at the rear of the vehicle. The system is monitored and controlled from an operator room mounted at the rear of the launch vehicle. A number of modifications were made to the standard highwall mining system to allow it to work at the 20° slope present at Kemmerer. These modification were primarily related to strengthening components to handle the weight of the miner and train of conveyor cars. Additionally, the cutter head was replaced with a higher power, heavy duty unit that yields a higher cutting height (3.35 m) and higher mining rates than the standard cutter head.

Figure 3: ADDCAR launch vehicle in operation.

Since the potential highwall mining recovery of the 475 Seam is beyond the economic limits of surface mining, exploration data for the seam beyond the final highwall were sparse. Therefore, additional exploration holes were drilled to support detailed highwall mining planning. The depth of cover at the highwall mining machine's limits was crucial to the design, and based on the expected seam dip of 20°, 182.88 m was the maximum penetration that could be achieved. Seven holes were drilled from the surface above the highwall: four corresponding to a penetration depth of 91.44 m from the highwall, and three near the penetration limit of 182.88 m. This additional drilling improved the seam model beyond the final highwall and confirmed the applicability of the ADDCAR highwall mining system. Below are the seam variables and highwall mining design criteria used at this location, with holes alternating between full and reduced penetration: maximum mining height is 6.10 m (two 3.05 m passes), maximum penetration depth 182.88 m, depth of cover at maximum penetration 131.06 m, reduced penetration depth 114.30 m, depth of cover at reduced penetration depth 95.10, maximum penetration web pillar width 13.26 m (7.41 m required), reduced penetration web pillar width 4.88 m, maximum penetration barrier pillar width 27.13 m (24.99 m required), and reduced penetration barrier pillar width 18.75 m.

Figure 4 shows a typical highwall cross section for Pit 5, where the highwall mining took place. If all holes were mined to the full penetration depth of 182.88 m with an excavation height of 6.10 m, the cover depth would be 131.06 m, the required web pillar would be 7.41 m, and the overall production would be 419.36 tons/m along the highwall. If the plan was optimized to include alternate holes with a penetration depth of 114.3 m, the cover depth over the shorter holes would be 95.10 m, the required web pillar width at the highwall would be reduced to 4.88 m (0.8 w:h ratio), and the overall production would increase to 442.58 tons/m. Note that the web pillar width at maximum penetration is 13.26 m (twice the 4.88-m web pillar at the highwall, plus the opening width of 3.51 m), which is greater than the 7.41 m required (**Figure 2**).

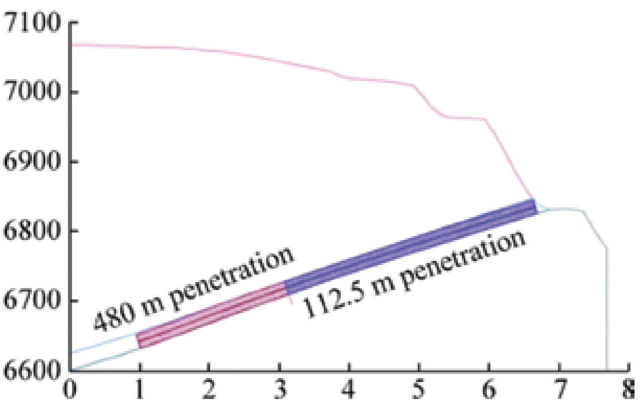


Figure 4: Typical profile of the 6.10-m seam dipping 20°.

As an added degree of conservatism to the geotechnical design, AAI recommended that the first highwall panel be limited to 10 openings. AAI also recommended that web pillars in this initial panel be increased by at least 0.61 m until the alignment of multi-pass openings could be confirmed. Data from ADDCAR's on-board monitoring systems confirmed that the alignment was adequate, and subsequent panels were mined with designed pillar widths and 20 openings per panel.

Highwall mining at the Kemmerer Mine is ongoing. To date, mining has been successful, but not without challenges. The system has proven capable of operating at inclinations greater than the design limit of 20°; in some areas, it has successfully mined at an angle greater than 25°. ADDCAR also successfully verified that alignment could be adequately maintained for two-pass mining.

During the highwall mining design stage, it was recognized that water could be an issue because certain areas on the property have historically produced water. However, the location of water sources is inconsistent, and inflow volumes could not be predicted with accuracy. Owing to the inclination of the openings, water that drains into the excavation (surface or groundwater) accumulates at the face. Groundwater was encountered in the 475 Seam at penetration depths between 64.00 and 182.88 m. Most of the initial holes in the southern portion of Pit 5 encountered water. While the highwall miner is operating, the water has minimal effect on production. However, if mining is interrupted due to mechanical or operational delays, the back of the hole tends to flood, making reentry problematic. Although some holes had to be abandoned due to flooding, the next adjacent holes benefited from the dewatering of the coal seam. By taking advantage of the dewatering, improving machine availability, and using operator experience to allow more of the water to be conveyed out of the hole with the coal, ADDCAR was able to incrementally increase penetration and ultimately achieve design penetration on a consistent basis. ADDCAR plans to design a pumping system for the highwall mining machine to help reduce the amount of water at the back of the hole, allowing the miner to reenter flooded holes.

Westmoreland, ADDCAR, and AAI are currently in the process of designing another area for highwall mining production, which should be ready to begin production later this year.

CONCLUSIONS

The restrictions imposed by steep-dip mining substantially reduce production and recovery as compared to flat-seam mining. Creative mining methods have been evaluated to optimize production in steep seams, and thick seams in particular. These methods involve mining the openings at shallower gradients by mining cross-seam or at an angle to the highwall. Other techniques applicable to increase production include multi-pass, multi-lift, and alternate depth methods. Angled openings, multi-pass mining, and multi-lift mining can increase production, but have some geotechnical risk associated with them. Cross-seam and alternate penetration methods can increase production with very little additional risk. Highwall mining of thick, steeply dipping coal requires careful planning and execution, including close cooperation between those responsible for geotechnical design, the mining company, and the highwall mining contractor.

AUTHORS

Chris Ross, David Conovera

Agapito Associate, Inc., Golden, CO, USA

Jake Baineb

Westmoreland Kemmerer, LLC Kemmerer Mine, Kemmerer, USA

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Fax : +852 2890 2657
Email : info@together-expo.com
marjorie@together-expo.com
katherinelee@together-expo.com

Beijing Office:
Room 12A11, Building A, Kunsha Center, 16 Xinyuanli,
Chaoyang District, Beijing 100027, P.R. China
Tel : +86 10 8451 0286 / 8451 0267
Fax : +86 10 8451 0263
Email : info@together-expo.com.cn
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