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Features

Komatsu

PC3000-6 Loading Shovel/ Backhoe

Loading 85 to 165 ton trucks using a backhoe or shovel productively requires speed, good dump height and reach, and power. The PC3000-6, whether in backhoe or shovel configurations, is the right fit for this loading class.

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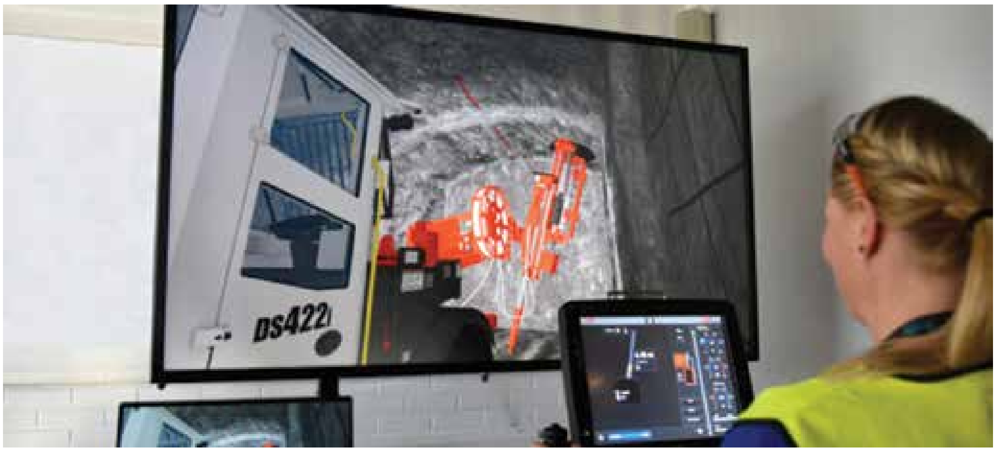
Sandvik's Digital Driller simulator provides convenient learning



Sandvik has updated its operator training simulator Digital Driller to safely train underground drill operators and maintenance teams.

The upgraded Digital Driller family now includes DS422i to provide advanced simulator training to improve operator and drilling performance, to deliver benefits for underground operations.

This new version of Digital Driller features the latest developments in underground drilling simulation, providing a low weight, safe and portable training tool.

Updates to Digital Driller include improved mining and tunnelling underground hard rock performance, purpose designed to simulate a real drill rig and using the same control system software as found on Sandvik drills, including the iSURE tunnel and drill management tool.

The Digital Driller provides comprehensive technical training for developing or refreshing operator skills and allowing operators to develop their skills progressively. With three levels of training available – Drillmaster, Professional and Beginner – it enables training for novices through to conducting the entire drilling cycle.

This allows for a shorter transition to operators using the real equipment and no risk of equipment damage, zero consumption of rock tools and fuel, zero emissions and no exposure to risk for trainees and reduces training costs by up to 35%

With its small and compact size, weighing just 20 kg, Digital Driller can be transported easily by a single person to wherever on the site is needed, thanks to its convenient wheeled carrying case.

Also featuring a short set up time, operator training can begin before a rig has been delivered to site. It significantly reduces the drill start up period, ensuring rig operators are familiar with the features and capabilities of Sandvik rigs from day one.

NSW endorses Russell Vale mine expansion

The New South Wales Government has recommended that the Independent Planning Commission approves the expansion of Wollongong Coal's Russell Vale mine.

The state's Department of Planning, Industry and Environment believes that the economic benefits of Wollongong Coal's revised underground expansion project outweigh its residual costs, deeming the project "approvable, subject to strict conditions of consent".

Under the revised project plan, the Russell Vale operations will only involve non-caving board and pillar mining techniques, to ensure long term stable operations with minimal subsidence impacts.

The Department of Planning, Industry and Environment believes that this revised plan has addressed key issues raised by the IPC when the first plan was submitted.

"Based on a detailed assessment, the department and relevant government agencies consider the revised underground expansion plan board and pillar mining method has addressed key issues raised by the IPC, particularly in relation to the uncertainty associated with subsidence and groundwater impacts," the department said in its statement to the IPC.

The department has recommended conditions to address the residual impacts and risks of the revised underground expansion plan, in consultation with government agencies and Wollongong City Council.

Minister for Planning and Public Spaces Rob Stokes has also requested that the IPC holds a public hearing with regard to the project, determining the development application.

Once complete, the expansion will allow Wollongong Coal to extract up to 3.7 million tonnes of run-of-mine coal over five years from the Russell Vale mine.

Greece sets ambitious targets for phase out by 2028

Greece has committed €5 billion ($5.9 billion) to phase-out the use of coal by 80% by 2023, to reduce the nation's carbon footprint.

According to data and analytics company GlobalData, these are highly optimistic targets, particularly considering the fact that back in 2015 coal-based generation formed 41.6% of the generation mix. In 2019 this reduced to 27.9% which, by 2030, if not entirely staged out, is predicted to have a small share of less than 10%, anticipates GlobalData.

Somik Das, senior power analyst at GlobalData, comments: "This has been one of the largest investments the nation has made in recent times, which is strong enough to leave coal-based generation behind, subsequently bolstering renewables in the power segment. The renewable energy (RE) segment (including small hydro) is expected to get a boost over the decade, where 85-90% of the total new capacity addition between 2019-30 will be renewable in nature. Almost 55-60% of the RE capacity is anticipated to encompass solar PV."

The country's proposed green investment is expected to help create more than 8,000 employment opportunities in the western Macedonia and Megalopoli regions. Investments incorporate a strategy by the state's Public Power Corporation (PPC) to build solar PV parks in Western Macedonia with a generating capacity of 2.3GW, and a €130 million ($154 million) solar PV venture by Hellenic Petroleum in the same region.

Das adds: "Across the country, there are more than 25GW of solar PV and wind ventures, which are either announced, in the permitting phase or are under construction. The growing renewable space attractiveness is likely to captivate global investments in the sector as the country plans to have new initiatives such as a reduction in application time for renewable projects, introduction of renewable energy certificates, energy efficiency measures, and planning offshore wind ventures. These bold steps make up the ideal ingredients for the country to leverage the green potential that lies largely unexplored currently."

BHP optimises NSW assets despite sale rumours

BHP is continuing to review its Mt Arthur coal operations, including making moves to lower operating costs, despite the mining giant expressing its interest to sell the New South Wales mine.

In its annual report, BHP reflected on transitioning its New South Wales Energy Coal (NSWEC) assets, including Mt Arthur, with a new product quality strategy, resulting in reduced volumes but an increased product quality.

BHP is expecting NSWEC unit costs to be between $US55 and $US59 ($75-$80) per tonne, based on an exchange rate from Australian to United States dollars of 70 cents in the 2021 financial year.

The company is working to reduce costs at its NSWEC assets in the short term, to ensure viable and resilient long term mining options.

"Work is underway to review mine planning and operating alternatives to structurally reduce costs in the near term and ensure a viable mining operation which is resilient during low price cycles," BHP stated in the report.

"New South Wales Energy Coal continues to plan for the most productive path through steeply dipping resources and securing the required regulatory approval to continue operations post financial year 2026.

"In financial year 2020, BHP completed an optimisation of the New South Wales Energy Coal outbound supply chain commercial arrangements through a partial divestment of shares and stapled capacity at the Newcastle Coal Infrastructure Group terminal.

"The total export capacity of the asset remains unchanged and the transaction has facilitated a more competitive cost position."

During the 2020 financial year, BHP's underlying earnings before interest, taxes, depreciation and amortisation (EBITDA) dropped by $US2.4 billion ($3.3 billion) to $US1.6 billion ($2.2 billion), reflective of the lower prices.

This included an underlying EDITDA decrease of $US374 million ($514 million) for NSWEC as a result of its focus on producing higher-quality products and unfavourable weather.

Czech utility EPH to close 600MW plant in France

Czech utility EPH is to shut down its 600MW Provence coal power plant in France only a year after buying it.

The firm has announced it will close the facility at the end of 2020, two years earlier than initially planned.

The announcement of its early closure follows Vattenfall recently revealing it would be shutting down its five-year-old Moorburg coal power plant early.

EPH says it plans to "extend industrial activity at the site", with one option being to convert the facility to biomass.

France has set a 2022 coal phase-out legislated through the country's energy and climate laws.

Exposure to harmful dusts on fully powered longwall coal mines in Poland

Dust suppression in longwall mines has always been problematic. Coal International presents this article that looks at the results of tests carried out on 5 longwalls in Poland.

The mining production process is exposed to a series of different hazards. One of them is the accumulation of dust which can pose a serious threat to the life and health of mine workers. The analysis of dust hazard in hard coal mining should include two aspects. One is the risk of coal dust explosions, which poses a direct risk of injury or even loss of life, the second is the risk of harmful dust, associated with the possibility of negative health effects as a result of long-term exposure to dust in the worker's body. The technologies currently applied in underground mining produce large amounts of coal and stone dust. Long-term exposure to dust and crystalline silica may cause chronic respiratory disease. The article presents the results of tests on the dust levels in the area of a fully powered longwall. The tests were conducted for five longwalls from different hard coal mines. In each of them, the average values of inhalable and respirable dust as well as the percentage content of free silica in the dust were determined in ten selected working positions. Additionally, for the longwall with the highest dust concentration, the levels of dust were determined for the basic activities related to the phases of the technological cycle. The comparative analysis conducted, and the results obtained demonstrate large variations in the dust levels in the different areas. The permissible values were significantly exceeded in a number of cases. This poses a great threat to the health of Polish miners. The results obtained indicate that it is necessary to undertake more effective measures in order to improve the working environment of the crew in hard coal mines.

INTRODUCTION

The mining production process is accompanied by a series of different hazards. They constitute a significant source of risks and may pose a threat to the life and health of employees and large material losses. Risk is the chance or probability that a person will be harmed or experience an adverse health effect if exposed to a hazard[1], while a hazard is any source of potential damage, harm or adverse health effects to something or someone under certain conditions at work. Basically, a hazard can cause harm or adverse effects (to individuals as health effects or to organizations as property or equipment losses)[1].

For this reason, one of the most important tasks during this process is to ensure adequate levels of safety for mine workers. Depending on the type of hazard, relevant steps are taken to reduce the likelihood or consequences of its occurrence. The presence of hazards in underground mining is related to the ongoing exploitation activity and results from the disruption of the initial balance in the intact rock mass. One of such hazards that poses a significant threat to the life and health of mine workers is harmful dust[2,3,4,5,6]. It results from the widespread presence of coal and stone dust in mine headings (as a mixture of silica, aluminosilicates and other elements, including trace metals), which is generated in the mining and transportation of the excavated coal material[2,3,4,6].

The dust formed during the mining of the body of coal and the dredging of dog headings, fills the mining atmosphere and travels through the ventilation system into most of the mine headings, including those located far away from the source, thereby causing their contamination[7,8,9].

It is estimated that the amount of dust generated may be as much as approx. 3% of the mass of the whole excavated coal material[2,10]. Assuming that the average daily mining output in a mine is approximately 10,000 Mg, the quantity of dust generated may amount to approx. 300 kg/day. The quantity of dust generated during exploitation depends on a number of factors, which can be divided into mining/technical and geological[11,12,13,14,15,16,17,18]. The former include the manner in which the rock mass is mined (including the type of the cutting machine), the type and condition of the cutting knives, the type and condition of the sprinkling system, as well as the speed of exploitation (including the speed of cutting), the thickness of the seam exploited[11,12,13].

The concentration of dust that rises in mining excavations is influenced by such factors as the velocity of air flowing through the excavations (high velocities can carry particles of dust previously settled on the floor, side walls and devices) and the number of free surfaces in the wall cross-section[7].

The second one includes the type of rock mined and its mineralogical composition, cleavage, compactness (easily mineable coals cause greater dusting than coals with higher mineability rates), hardness, volatile matter content in the coal (the higher this content in coal, the higher dust concentrations are generated during the mining process) and the ash content in the coal. Increased ash content leads to increased total dusting. However, it does not result in higher concentrations of the respirable fraction that is primarily responsible for the detrimental effects of dust on the human body. This results from the physical structure of coal components that form the ash and the coal moisture content (dust levels are lower when the moistness of the seam ranges from 0 to 10%, yet increase when the moistness exceeds 10%)[14,15,16,17,18].

The specific risk for the health of workers exposed to the impact of dust arising from the exploitation of hard coal is created by the free crystalline silica contained in this dust. The specific risk for the health of workers exposed to the impact of dust arising from the exploitation of hard coal is created by the free crystalline silica contained in this dust[19,20,21,22]. Prolonged work in a high-dust environment, where dust levels exceed the permissible values and where, additionally, the free crystalline silica fraction is present, may cause serious lung diseases (mainly pneumoconiosis) in miners.

This condition is the most commonly reported occupational disease amongst current and former hard coal miners, not only in Poland but also in China and the USA. According to the statistics released by the Ministry of Health in China, more than 300,000 coal miners suffered from pneumoconiosis by the end of 2007, accounting for 50% of the total number of pneumoconiosis patients in China. Every year, more than 10,000 people who work in major state-owned coal mines are added to the list of pneumoconiosis patients, and on an average, 2500 Chinese miners die from this disease[4].

In the USA a gradual increase in the prevalence of pneumoconiosis amongst miners has also been recorded since the late 1990s. The most disconcerting trends have been observed in parts of central Appalachia (e.g., MSHA districts 4 and 12), and many new cases of CWP (pneumoconiosis-CWP, sometimes referred to as “black lung”) and/or silicosis appear to be advanced or presenting in younger miners[23].

In Poland, in the years 2000-2017, there was a total of as many as 7340 diagnosed cases of pneumoconiosis amongst current and former mine workers[24]. A comparison of the number of cases of pneumoconiosis among workers of hard coal mines is presented in **Figure 1**. The number of cases of active mine employees has also been taken into account (this is due to the data available) since 2005. It should also be emphasized that 80% of diagnosed cases of pneumoconiosis in Poland referred to active and former employees of hard coal mines. The proportions have remained at a similar level for many years. At the same time, it should be emphasized that the pneumoconiosis accounts for as much as 76.6% of all occupational diseases among miners of Polish hard coal mines[24].

Pneumoconiosis is an incurable diseases which can still progress even after cessation of harmful dust exposure. A crucial impact on the development of this disease is exerted by the particle size and the type of dust inhaled. Irreversible health changes, on the other hand, are caused by respirable dust penetrating into the human body. Respirable dust is the fraction of airborne particulates that can be deposited anywhere in the lung gas-exchange region[25].

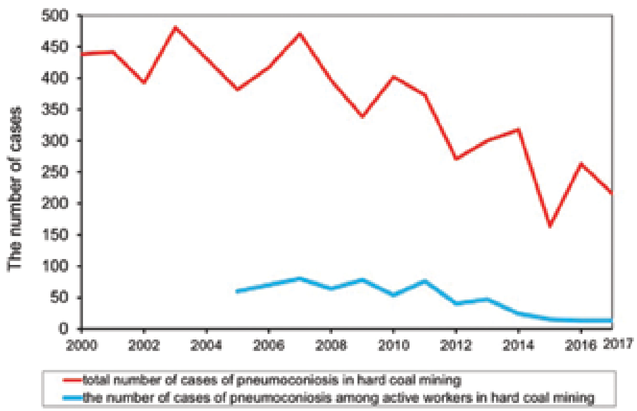


Figure 1: The number of reported cases of pneumoconiosis among workers of hard coal mines in Poland in 2000–2017.

The dusts affecting the human body can generally be divided into collagen and non-collagen dusts. The former ones show biological activity causing focal fibrosis of the pulmonary tissue through their toxic effect on the macrophages. The latter are characterised by the fact that they accumulate only in the pulmonary alveoli, without causing permanent changes in the body. Irreversible changes caused by pneumoconiosis in the pulmonary alveoli represent the process of phagocytosis in which the collagen dust contributes to the decay of macrophages (the connective tissue cells). These dead cells form deposits which become fibrous over time and thus decrease the surface area of the pulmonary alveoli. They are replaced by newly-formed phagocytes. The process whereby irreversible lesions are becoming noticeable takes from a few to a dozen or so years and may exhibit a mild or severe course. It takes approximately 10-20 years for the lesions caused by mild simple pneumoconiosis to become visible, whereas severe pneumoconiosis, or the so-called silicosis, does not manifest itself until after approximately 5-10 years of exposure to harmful dust, please refer to references 8 to 29.

Pneumoconiosis in coal mine workers represents approximately 81% of all the registered cases of this disease in the Polish industry[24]. These unfavourable statistics lead to a series of measures being taken in mines in order to reduce the workers' exposure to dusts. Nevertheless, it is not uncommon, in practice, to record dust concentrations in the mine atmosphere which significantly exceed the permissible values. This is because it turns out that complete elimination of the harmful dust hazard in hard coal mining is currently not possible. The reason for this is that these dusts are the by-product of polydisperse processes and it is impossible to completely eradicate their formation with the technology currently in use.

Therefore, it is reasonable to conduct tests and analyses with a view to diagnosing the current levels of dust in mines, as well as to develop guidelines concerning the improvement of this situation.

The issues related to the dust hazard appear in numerous publications, please refer to references 2 to 23, but they mainly concern the methods for measuring dust levels in mine headings refer to references 2,4,11,30, the analysis of average dust values occurring in underground mining or for the particular groups of working positions[31,32,33]. As a result, the publications mostly provide a general analysis of the dust content in all the mine headings, without account being taken of the type of these headings (longwall or dog headings) and the specificity of the particular working positions. By adopting the average dust levels for the purposes of analysis, these publications make it possible to illustrate merely the general scale of the problem and are primarily of a demonstrative nature. Such an approach, however, does not reflect the actual levels of dust in the particular working positions, which are – to a large extent – location-dependent.

Therefore, it can be concluded that the presentation of differences in dustiness harmful to health occurring at the same workplace, and located in different walls, is a new approach to the problem of this threat for workers in the Polish hard coal mining industry.

Taking into account the current state of knowledge and the serious hazard to human life and health caused by dust, the authors decided to conduct tests aimed at determining the concentration levels of harmful dusts in selected working positions in hard coal mines. The tests were carried out in five mines (for five different longwalls), in ten selected working positions located in the area of longwalls, i.e., in places which were assumed to exhibit the highest concentration levels of harmful dusts.

The purpose of the tests and of the analysis conducted was to determine the actual concentration values of mining dusts in these working positions, taking into account the differences related to their location. Additional tests and analyses were performed in the working positions located in the area of the longwall with the highest dust levels. In this case, account was also taken of the activities performed by the workers during a single routine working shift which lasts 7.5 h.

The Authors related the results obtained to the regulations applicable in Poland concerning the Maximum Admissible Concentration (MAC) of dust containing free crystalline silica[34]. In the authors' opinion, the manner of assessing the harmful dust hazard in Polish coal mines, as presented in the

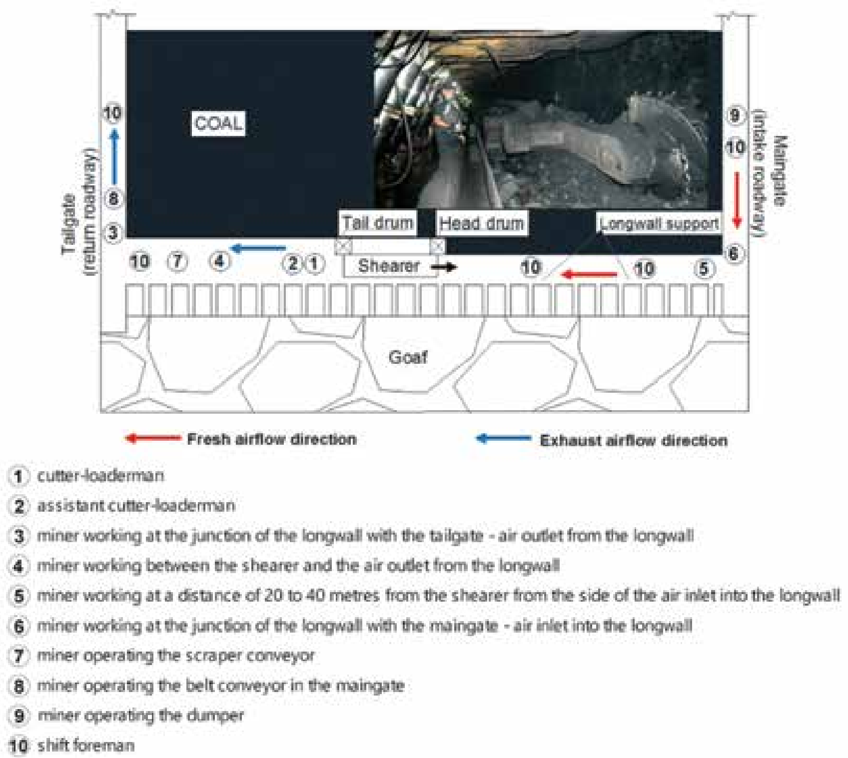


Figure 2: Typical U ventilation system in Poland with markers workplaces (modified from[35]).

paper, should be used for taking more effective prevention measures and minimising the consequences of this hazard. The related activities should therefore represent one of the pillars in the responsible and sustainable development of this industry.

MATERIALS AND METHODS – DUST MONITORING

The tests of dust levels were conducted in five hard coal mines (with one longwall analysed in each) located in the Upper Silesian Coal Basin in ten working positions. In these mines, exploitation is carried out by means of an automated longwall system. The longwalls under analysis are mined by means of longwall shearers. All the longwalls were ventilated with the U-type system (**Figure 2**).

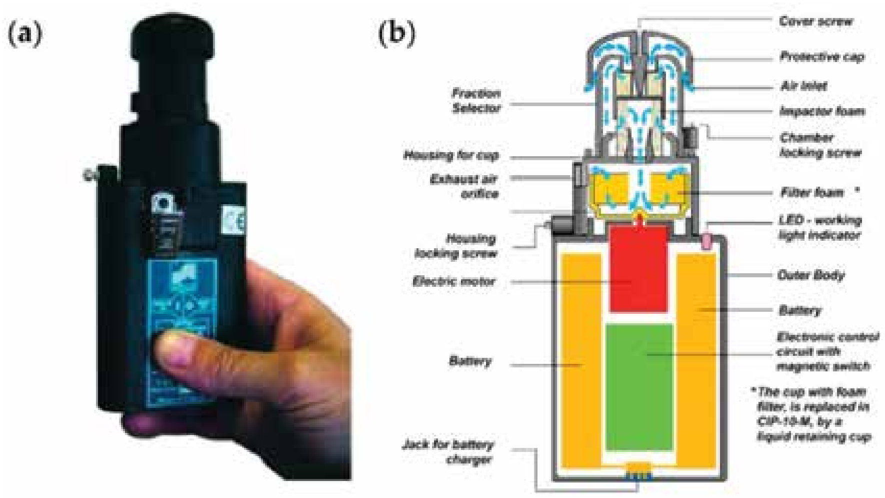


Figure 3: Personal dust sampler (a)[36] and schematic of CIP-10 Sampler (b)[37].

Measurements of dust concentration occurring at workplaces were carried out in longwalls run with a system based on caving of roof rocks into the empty space left after extracted coal (goaf), in the case of filling the empty space with other material, however, we deal with backfill.

The measurements in each of the mines were conducted in the following working positions located in the area of the longwall and its adjacent headings: cutter-loader man (1), assistant cutter-loader man (2), miner working at the junction of the longwall with the tailgate – air outlet from the longwall (3), miner working between the shearer and the air outlet from the longwall (4), miner working at a distance of 20 to 40 m from the shearer from the side of the air inlet into the longwall (5), miner working at the junction of the longwall with the main gate – air inlet into the longwall (6), miner operating the scraper conveyor (7), miner operating the belt conveyor in the main gate (8), miner operating the dumper (9) and shift foreman (10). The distribution of these positions in the area under analysis, with account being taken of the indications placed in brackets (in the form of consecutive numbers), has been presented in **Figure 2**. The shift foreman (marked as 10) has no permanent working zone in the heading since his obligations encompass the supervision of ongoing works in the entire longwall. Hence, it is the most mobile working position which has been marked in a few places in the figure.

The measurements of dust levels were carried out by means of CIP-10-type personal dust samplers (Arelco ARC, Fontenay Sous Bois, France) using the dosimetric-individual method (**Figure 3a**). Schematic of CIP-10 Sampler has been presented in Figure 3b.The selected workstations are typical for longwall mining. Between two or three employees usually work in one site, with the exception of a longwall shearer, a combine harvester assistant, and a shift foreman. Measurements were made for individual employees representing a given workstations.

A CIP-10-type dust sampler is a gravimetric dust meter intended for assessing individual exposure to dust in the working environment. It measures the mass of dust in air. For a dust sampler to be selected for the tests, it had to be admitted to operation in places where explosive atmospheres may occur. The measurement error of the device used for the tests was ±0.05 mg/m3. The measurement error for dust concentration for stable airflow did not exceed 3% of the measured value[38]. The content of free crystalline silica in the dust was determined on the basis of the samples collected. The measurement time encompassed one working shift (7.5 h for air temperature up to 28 °C).

In each of the longwalls, the measurement of dustiness at a given workstation included measurements of inhaled (total) and respirable fraction (dust penetrating the alveoli). Then, for the collected samples, the content of crystalline silica was determined under laboratory conditions.

Measurements were carried out for three phases of the technological cycle (access to the workstation and preparatory work, mining of coal with a combine harvester and work carried out during the technological break – no mining). The duration of the measurement series during the implementation of the individual phases of the cycle ranged from 1.5 h (access to the workstation, preparatory work in the longwall) to 3 h (mining). No disturbances in the measurements took place in any of the five longwalls during tests.

Once the tests were completed, the dust samples collected on the filter placed in the CIP-10 dust sampler were weighed. For the selector dust to calculate the dust concentration is collected on the foam filter. Calculation of dust concentration Xresp (mg/m^3) is performed according to the following equation[2]:

$$X_{resp} = \frac{\Delta m}{vt}$$

where: Δm is mass of the coal dust collected on filter, v is the flow of air through the dust sampler (for CIP-10 it is 10 L/min) and t is measurement time (min).

Based on the measurement of the dust mass and the length of the dust samplers' exposure to the dusty air, weighted average values were determined for the concentrations of total dust and respirable dust for the measurement period.

The preparation of the results involved determination of the weighted average values of the inhalable and respirable dust concentrations, the content of free crystalline silica in the dust and the exceedance rate of the MAC values (according to Polish regulations) for each of the working position under analysis[34].

Determination of free crystalline silica in total and respirable dust at workplaces is performed using the colorimetric method in compliance with the Polish standard[39]. The first phase is based on changing the crystalline free silica contained in the dust sample into soluble sodium silicate and then conducting colorimetric determination of silicate ions. This method of determination of silica in dust is currently used by the majority of laboratories in Poland[40]. Additionally, average values of the inhalable and respirable fraction concentrations were determined for each of the longwall areas under measurement, along with standard deviations and their medians.

RESULTS AND DISCUSSION

The measurements carried out were then used as a basis and allowed the determination of the average weighted total and respirable dust concentrations occurring at the tested sites and the content of free crystalline silica in this dust on the basis of the measurements carried out. These values were determined for all workstations in the tested longwalls (in each mine). A summary of the results obtained is presented in **Figure 4**, **Figure 5**, **Figure 6**, **Figure 7** and **Figure 8**.

Analysing the results obtained, one can conclude that the highest dust values were registered in the following working positions:

– The miner working between the shearer and the outlet of air from the longwall,
– The miner working at the junction of the longwall with the tailgate – air outlet from the longwall,
– The assistant cutter-loader man,
– The cutter-loader man.

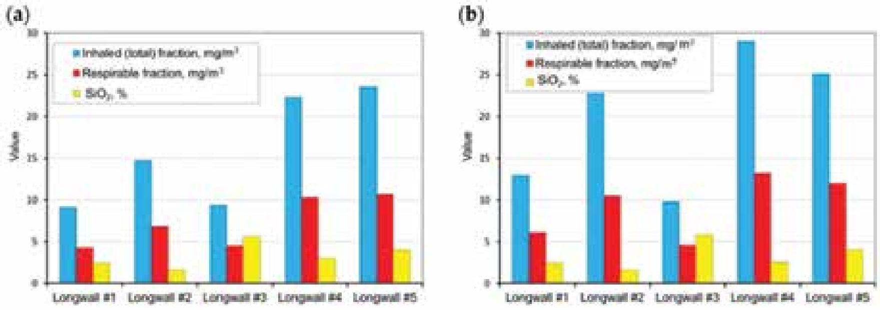


Figure 4: The measurement results for the dust levels in the positions of the cutter-loader man (a) and the miner working between the shearer and the air outlet from the longwall (b) (workplaces 1 and 4).

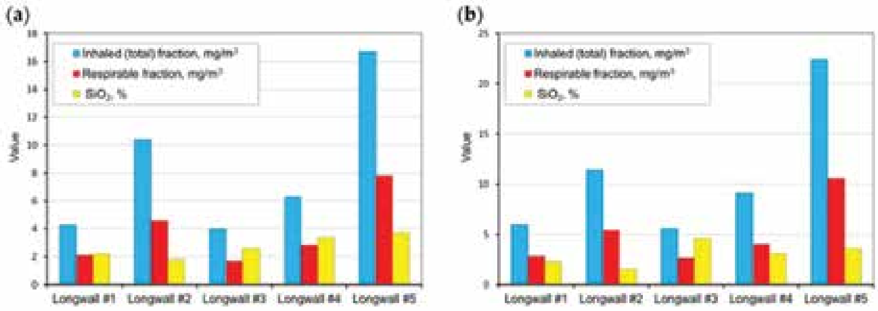


Figure 5: The measurement results for the dust levels in the positions of the miner working at the junction of the longwall with the maingate (air inlet) (a) and the shift foreman (b) (workplaces 6 and 10).

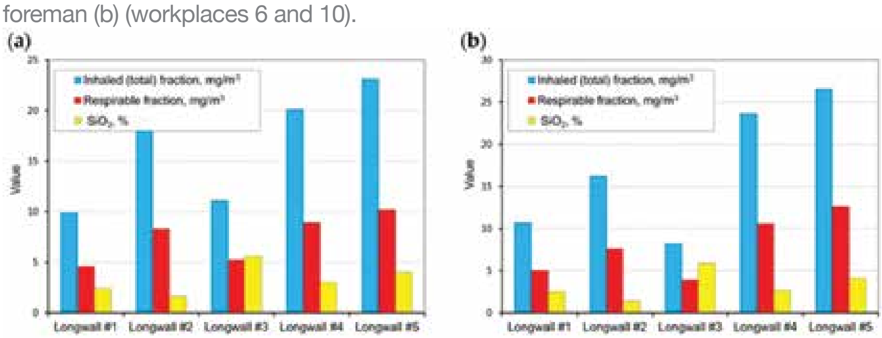


Figure 6: The measurement results for the dust levels in the positions of the assistant cutter-loader man (a) and the miner working at the junction of the longwall with the tailgate (air outlet) (b) (workplaces 2 and 3).

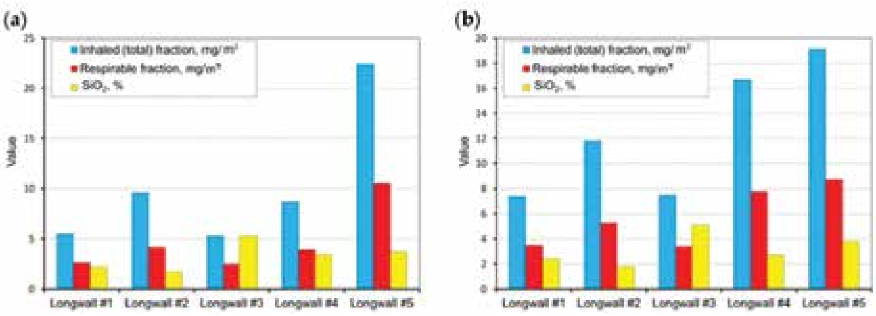


Figure 7: The measurement results for the dust levels in the positions of the miner working at a distance of 20 to 40 m from the shearer from the side of the air inlet into the longwall (a) and the miner operating the scraper conveyor (b) (workplaces 5 and 7).

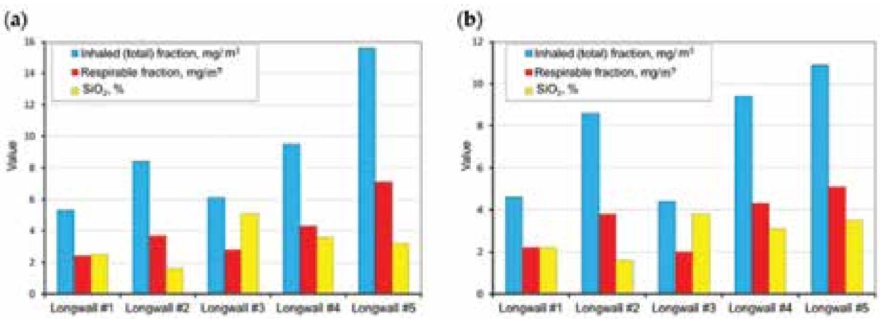


Figure 8: The measurement results for dust levels in the positions of a miner operating the belt conveyor (a) and a miner operating the dumper PZ/PT (b) (workplaces 8 and 9).

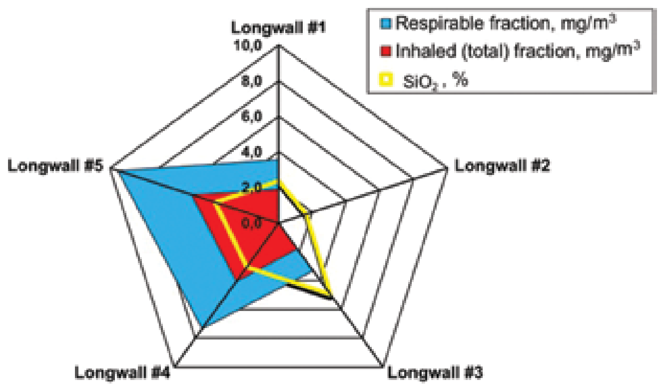


Figure 9: The average concentration of inhaled and respirable dust and the content of crystalline silica in various workstations.

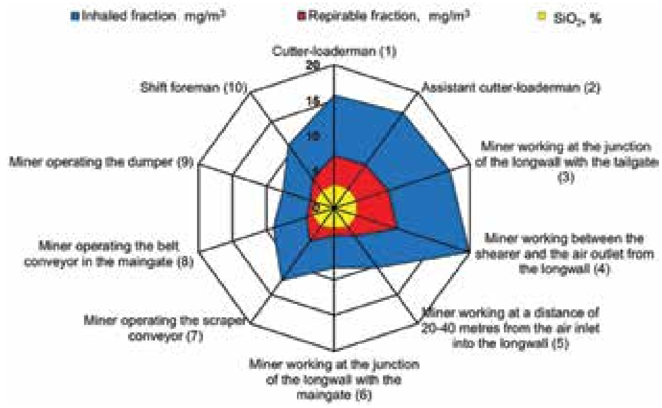


Figure 10: The average exceedance rate of the MAC values for dust in the longwalls under examination.

The lowest dust levels were registered in the position of the miner operating the dumper. This is due to the location of this position in relation to the extracting machine and the manner of longwall ventilation. This position is located not in the longwall itself but in the maingate, along which fresh air is supplied to the longwall. The dust generated during the operation of a shearer has difficulties in getting through to this region. However, it finds it easy to travel along with the stream of fresh air into the opposite heading (the tailgate).

The average concentration of inhaled and respirable dust as well as the content of crystalline silica in the various workstations in the walls are presented in **Figure 9**.

The measurement of dust levels in the working positions under analysis in the five longwalls made it possible to determine the average values of the inhalable and respirable fractions for these mines (**Table 1**). The mean content of free crystalline silica in the tested dust was also calculated (**Table 2**). The value was determined in a percentage as required by current Polish regulations. **Table 2** presents average percentages of free crystalline silica for sites located in the subject longwalls.

Based on the test results obtained, it can be concluded that the highest levels of dust occur in longwall no. 5. The workers of this coal mine have the highest exposure to the harmful effects of dust (the average value for the inhalable dust is higher than 20.0 mg/m^3, and for the respirable dust – higher than 5.0 mg/m^3). The lowest exposure to dust occurs in the area of longwall no. 1 and no. 3 (the average value for the dust inhaled is slightly higher than 7.0 mg/m^3, and for the respirable dust – 3.0 mg/m^3). The values of concentrations admissible in Poland of harmful dust containing free crystalline silica are presented in **Table 3**. These values were used to determine the fold of the MAC exceeded level of the total and respirable dust in the tested longwalls (**Figure 10**).

Table 1: A summary of the average values of measurement results for dusting in the longwalls under analysis.

	Average Concentration, mg/m^3		Standard Deviation, gm/m^3		Median, mg/m^3	
Longwall	**Inhaled fraction**	**Respirable fraction**	**Inhaled fraction**	**Respirable fraction**	**Inhaled fraction**	**Respirable fraction**
Longwall #1 (10 measurements)	7.57	3.56	2.78	1.30	6.70	3.15
Longwall #2 (10 measurements)	13.19	6.02	4.43	2.12	11.6	5.35
Longwall #3 (10 measurements)	7.14	3.32	2.31	1.13	6.80	3.10
Longwall #4 (10 measurements)	16.18	7.34	7.58	3.45	16.70	7.80
Longwall #5 (10 measurements)	20.54	9.54	4.62	2.20	22.40	10.35

Table 2: A summary of the average values of measurement results for crystalline silica in the dust.

Longwall	Average Concentration, %	Standard Deviation, %	Median, %
Longwall #1	2.35	0.12	2.4
Longwall #2	1.62	0.12	1.6
Longwall #3	4.94	1.03	5.2
Longwall #4	3.05	0.35	3.05
Longwall #5	3.76	0.28	3.75

Table 3: Limit values for dust containing free crystalline silica in compliance with Polish regulations[34].

Free Crystalline Silica Content, %	Inhalable Dust, mg/m^3	Respirable Dust, mg/m^3
over 50	2.0	0.3
2-50	4.0	1.0

The tests registered significant differences in the dust concentration values and in the percentage content of free silica during the performance of the particular activities related to the extraction cycle. These differences were dependent on the working position covered by the measurement. To illustrate these differences, an additional analysis was performed on the data for longwall no. 5 with the highest dust levels. These data were related to the activities (closely linked to the technological cycle) performed by the particular workers. They are characteristic of each working position covered by the measurement.

Implementation of the technological cycle in the exploitation wall includes the performance of a number of activities depending on the workplace, for example, works related to the mining of the combine by the combine harvester and assistant. Reconstruction of the intersections of the fail and head gates is conducted by the employees working at the inlet and outlet to the longwall. Moving the beam stage loader or the shortening of the belt conveyor are tasks performed by the miners servicing the scraper and wall conveyors. These are the main activities conducted simultaneously with the coal mining process.

When the combine moves along the wall by mining the coal, its movement should not be stopped by, for example, moving the support or other mining work. The only exception are the events resulting from the selected mining technology. This is the case, for example, when the main drive or return drive is moved. In this case, the longwall shearer is stopped for the duration of these activities. These are referred to as mining work carried out during the technological break.

The dust levels in the working positions in question were measured and analysed during three phases of the technological cycle. These phases encompassed the following:

- Reaching the workplaces and preparatory works before the exploitation process.
- Cutting of the body of coal by means of a longwall shearer;
- Mining works during a maintenance shutdown.

The results obtained are presented in **Table 4**, **Table 5**, **Table 6**, **Table 7**, **Table 8**, **Table 9**, **Table 10**, **Table 11**, **Table 12** and **Table 13**.

The results demonstrate that the location of a given working position along with the type of activities performed have a significant impact on the levels of dust. The lowest dust concentration levels were recorded in the position of the miner operating the dumper whereas the highest – in the working position located at the junction of the longwall with the tailgate (outlet). The low dust concentration levels in the position involving the operation of the dumper are due to the fact that the dumper is located at a considerable distance from the initial, direct source of the dust and lies in the inlet stream of air (with lower dust levels). Minor differences in dust levels are present in the positions of the miner working at a distance of 20 to 40 m from the shearer from the side of the air inlet into the longwall, the miner working at the junction of the longwall with the tailgate (air outlet), the cutter-loader man, the assistant cutter-loader man and the miner working between the shearer and the air outlet from the longwall. This is caused by the location

Table 4: The measurement results for the dust levels for selected activities, in the working position located at the inlet of air into the longwall.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	9.3	4.2	3.7
Support the main drive during cutting of the body of coal	20.1	9.7	
Mining works during a maintenance shutdown	19.5	8.9	

Table 5: The measurement results for the dust levels for selected activities, in the working position miner operating the scraper conveyor.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	8.9	4.1	3.8
Service of scraper conveyor	24.7	11.4	
Mining works during a maintenance shutdown	21.9	10.0	

Table 6: The measurement results for the dust levels for selected activities, in the working position the miner working at the junction of the longwall with the tailgate (air outlet).

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	10.8	5.1	3.7
Moving the roof supports during cutting of the body of coal	37.7	18.1	
Mining works during a maintenance shutdown	27.3	12.9	

Table 7: The measurement results for the dust levels for selected activities, in the working position the miner working at a distance of 20 to 40 m from the shearer from the side of the air inlet into the longwall.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	9.5	4.6	3.7
Support for auxiliary drive during cutting of the body of coal	28.7	13.4	
Mining works during a maintenance shutdown	27.0	12.6	

Table 8: The measurement results for the dust levels for selected activities, in the working position the cutter-loader man.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	9.7	4.7	4.0
Cutting of the body of coal by mining machine	33.2	15.7	
Mining works during a maintenance shutdown	24.8	10.1	

Table 9: The measurement results for the dust levels for selected activities, in the working position the assistant cutter-loader man.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	9.7	4.7	4.0
Auxiliary works during cutting of the body of coal	32.8	14.9	
Mining works during a maintenance shutdown	24.2	9.7	

Table 10: The measurement results for the dust levels for selected activities, in the working position the miner working between the shearer and the air outlet from the longwall.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	10.1	4.8	4.0
Maneuvering the roof support during cutting of the body of coal	35.4	16.8	
Mining works during a maintenance shutdown	26.2	12.7	

of these positions in a close distance to each other and the performance of similar activities.

The smallest differences in dust concentration levels during a mining cycle and a maintenance shutdown were recorded in the positions located at the air inlet to the longwall. This is closely related to the direction of the airflow in this heading. The greatest differences occur, on the other hand, between the positions located at the air inlet and outlet points of the longwall.

CONCLUSIONS

The article presents the results of the tests and analyses concerning the concentration levels of dust and the content

Table 11: The measurement results for the dust levels for selected activities, in the working position a miner operating the belt conveyor.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	8.1	3.7	3.2
Handling of conveyor and crusher	20.4	9.2	
Mining works during a maintenance shutdown	16.7	7.6	

Table 12: The measurement results for the dust levels for selected activities, in the working position a miner operating the dumper PZ/PT.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	8.2	3.8	3.5
Operating the dumper PZ/PT, cleaning the area of exploitation	13.5	6.3	

Table 13: The measurement results for the dust levels for selected activities, in the working position the shift foreman.

The Exploitation Phase	Dust concentration, mg/m^3		SiO_2, %
	Total	Respirable	
Reaching the workplaces and preparatory works before the exploitation process	9.5	4.6	3.6
Inspection works	28.1	13.0	

of free crystalline silica in the dust, as present in selected working locations in five Polish hard coal mines. The testing methodology applied made it possible to determine these values and conduct a comparative analysis. The results obtained unambiguously indicate that underground mining exploitation is characterised by extremely difficult working conditions in terms of dust levels. The process of cutting and transporting the excavated coal material leads to the formation of large amounts of harmful dust, which may have an immensely negative impact on the workers' health once it gets into the atmosphere. At the same time, the ventilation system working in the mine makes this dust spread practically all over the mine. A substantial part of this dust also reaches the surface. As a consequence, the exposure to the harmful effects of dust occurs in practically all the areas of the mine's underground infrastructure. Definitely the worst conditions in this regard, however, are present in the area of the longwall and dog headings located along the stream of used ventilation air. As was already pointed out, it is the ventilation air that transports the greatest amounts of dust.

The results obtained indicate that the atmospheric dust concentrations depend on both the location of the workplace and the working position occupied. Workers employed in the same positions yet in different areas of the mine are exposed to varying concentrations of dust with various content of free crystalline silica. The reason for this is that the silica content in dust depends on the mineralogical composition of the exploited seam, whereas the dust concentration levels are primarily dependent on the technical/mining factors. The highest dust levels, significantly exceeding the permissible values, were recorded mainly in the working positions related to the mining process.

The results obtained confirm that, despite the good recognition of the dust hazard and the application of increasingly effective measures of prevention, the exposure to the harmful effects of dust still represents a serious threat to the health of Polish workers employed in underground mine headings.

A chemical analysis of the dust present in the working positions under examination showed that, in 20% of these positions, the dust contains less than 2% of free crystalline silica, while in 80% of them – more than 2%. Significant differences were also noted in the dust concentration levels occurring within the same working position and during the performance of the same activities. Free crystalline silica has the most negative impact on the human respiratory system and determines the level of risk for the health of workers exposed to the harmful effects of dust.

Assessing the results obtained, it can be concluded that it is currently very difficult under the Polish mining/geological conditions to achieve dust concentration levels that do not exceed the MAC values, despite the application of increasingly effective measures of prevention. It is reasonable to assume that this condition will continue to negatively affect the health of workers employed in underground headings. The presented results should provide a significant source of information for the mine service departments about the actual level of dust hazards as well as encourage them to take more effective actions in order to reduce the related risk. The presented results should be used to plan preventive activities at workplaces where there is a risk of harmful dust. It should be emphasized that the problem is current and applies to the entire underground coal mining industry in Poland. In Poland, it particularly applies to employees of hard coal mines where exploitation is conducted with the use of fully powered longwalls. Similar technologies are used in other mines in the world where comparable threats occur.

The authors hope that the results obtained will be the basis of a broader discussion on the threat of dust harmful to the life and health of underground workers. This should apply to the direct protection of miners during their work underground as well as after they end working for the mining industry. A significant number of recorded cases of pneumoconiosis refers to former miners. Legislative changes also seem justified in this respect. Current regulations require to conduct measurement of the concentration of dust at workplaces at least every 12 months. Frequency of measurements seems to be insufficient, due to the changing geological and mining conditions, which have an impact on the amount of dust produced during the mining process of the body of coal. The obtained results clearly indicate that the working conditions in the mines in terms of dust hazard are still very unfavourable, and the applied solutions in the scope of limiting the dust formation are insufficient. Consequently, the threat of dust harmful to the health of Polish mine employees is a significant issue, the scale of which does not decrease.

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Fundamentals of modern ground control management in Australian underground coal mines

Underground coal mining is inherently hazardous, with uncontrolled ground failure regarded as one of only several critical risks for multiple fatality events. Development, implementation and management of overarching systems and procedures for maintaining strata control is an important step to mitigating the impact of ground failure hazards at a mine site operational level. This paper summarised the typical pro-active ground control management system (PGCMS) implemented in various Australian underground coal mines. Australia produces approximately 100 million tonnes a year of metallurgical and thermal coal from approximately 30 of the world's safest longwall mines operating in New South Wales and Queensland. The increased longwall productivity required to achieve both high levels of safety and profitability, places significant emphasis on the reliability of pro-active ground control management for longwall mining operations. Increased depths, adverse geological conditions, elevated variable stress regimes and weaker ground conditions, coupled with an industry wide need for increased development rates continue to make ground control management challenging. Ground control management is not only about ground support and pillar design though but also a structured process that requires a coordinated effort from all levels of the workforce to both minimise the occurrence of adverse geotechnical events and mitigate the potential risks when they do occur. The PGCMS presented in this paper is proven to provide both a safer and more productive mine environment through minimisation of unplanned delays. The critical elements of the method are presented in detail and demonstrate the utility and value of a ground control management system that has potential for implementation in underground coal mining globally.

INTRODUCTION

Coal has been the main source of energy in Australia for over 200 years and in more recent times a top five export commodity in terms of revenue. The Department of Industry, Innovation and Science estimates that saleable black coal production in Australia was over 440 million tonnes in 2014-2015, which is more than 50% higher than a decade ago and over 150% higher than 1990-1991[1]. Open-cut coal mining accounts for approximately 77% of coal production in Australia. Currently, approximately 100 million tonnes a year of coal is produced from 30 longwall mines operating in New South Wales and Queensland. The geological and geotechnical conditions vary significantly both between and within these operations and successfully managing such variability requires an integrated, pro-active ground control management strategy (PGCMS).

Australia's coal mining safety record outperforms that of the US, China and other major coal mining countries, with the fatal risks associated with strata failure typically well managed[2]. Large ground control failures resulting in unplanned delays to production and lost revenues do still occur relatively frequently but are often only reported on internally within organisations for commercial reasons. Over the years, many mines have developed various ground control strategies to minimise or eliminate uncontrolled strata failures as the loss of control poses significant safety and financial risks. Since the decline of coal prices from 2012, the typical internal ground control strategy has evolved to utilising in-house geotechnical engineering groups within company and/or mine site technical services departments[3]. This has resulted in most geotechnical designs being conducted by mine site geotechnical engineers whereas in times gone by this was often provided by an independent third party, or consultant engineers. Although this new approach is sound practice and acceptable if completed by suitable competent persons, it should still incorporate some form of peer and/or external third-party review. But rather than rely on the third party, the PGCMS has evolved from a desire for mining companies to continue to manage the risks on a daily basis throughout the life cycle of an operation to ensure it operates as safely and productively as possible.

The PGCMS involves many critical elements. It is not only an understanding of the impacts of the geotechnical environment on likely ground behaviour to allow the mines to extract underground reserves safely and economically, but also how to predict, communicate and escalate the expected conditions in a timely manner to the appropriate audience so its impact is suitably mitigated. This paper provides an overview of the foremost PGCMS in Australia, describes the critical elements and main features of the system, and offers guidelines for all coal mine operators to consider incorporating into a ground control management plan.

CRITICAL ELEMENTS OF A PRO-ACTIVE GROUND CONTROL MANAGEMENT STRATEGY

A PGCMS is a practical stepwise, systematic process that has evolved within the Australian coal mining industry to ensure that no person is exposed to an unacceptable level of risk from an uncontrolled strata failure. This approach is based on a standard Plan-Do-Check-Act (PDCA, **Figure 1**) methodology (or similar) originally developed as a tool to control and continuously improve processes in manufacturing, otherwise known as the Deming Wheel[4].

Figure 1: PDCA model (after[4]).

The following principles form the PDCA model in an iterative manner:

1. Plan – planning and documentation of objectives, expected outcomes, systems, processes and activities;
2. Do – acceptance and implementation of the plan;
3. Check – measurement and analysis that is understood and accepted;
4. Act – review and management follow-up, enact a response to changing conditions and implementation of improvement initiatives that are sustainable.

For implementation of a successful PGCMS, it is vital to ensure that all relevant stakeholders are involved in its formulation, and once formulated, the process is integrated into the mine's "safety and health management system" (SHMS). It is equally important that the PGCMS is communicated and made available to the entire workforce. An underlying requisite is that the process is owned not only by mine management (usually the Site Senior Executive), but the entire workforce. In many cases, implementation of a ground control management plan may be less than optimal as personnel fail to complete their roles defined within the plan. This may not be due to negligence but maybe the strategy itself is flawed, the person was unaware of their requirements in the plan or several other valid reasons not involving disregard. Involving people at all levels of the organisation in the process as required by a PGCMS, creates ownership at all levels. Hence the first point is worth iterating again; the relevant persons at the mine should all be involved in the development of the PGCMS so that they have ownership of the final document, understand its components and why they were included.

The elements of a typical ground control plan used in Australian underground coal mines can be grouped into four main categories: (1) procedures and standards, (2) data collection and design, (3) implementation and monitoring, and (4) review and investigation. The following critical elements are included in those four categories: (1) strata control principal hazard management plan, (2) ground control standards, (3) geotechnical design manuals and programs (software and calculators), (4) geotechnical design reports, (5) geotechnical data and associated calculations, (6) geological and geotechnical mapping and hazard plans, (7) investigation of incidents and hazard reports, (8) trigger action response plans specific to various stages of mining, (9) monitoring regime (observation, extensometry, powered roof support monitoring, etc.), (10) risk management and risk assessments, (11) permit to mine process (also commonly referred to as Authority to Mine Process), (12) periodic geotechnical testing and sampling maintained in an accessible database, (13) periodic third-party audits of high-risk zones (“critical areas”), (14) ground support product standard specifications, evaluation and testing procedures, (15) weekly reporting system, (16) training for employees and geotechnical engineers, (17) definition and appointment of suitably qualified geotechnical engineers, and (18) audits and reviews of ground control management process.

The above list is not exhaustive; there are other processes that Australian mines may use, and some listed that are not used. It is up to the individual mine operator to determine what is most suitable for their operation, often via a risk assessment process. While some mining companies develop the above elements as standalone processes, others combine them into an overall “ground control management plan” or into the “principal hazard management plan” in a hierarchical structure with subordinate documentation referenced in the overarching plan. The following sections present a practical guideline for the combination of those elements.

Procedures and standards

Strata control principal hazard management plan

Principal hazard management plans (PHMP), also commonly referred to as principal mining hazard management plans or ground control management plans are required by legislation in Australia as part of each mine’s safety and health management system. The PHMP must provide for the following basic elements so far as is reasonably practicable: (1) risk identification and assessment, (2) hazard analysis, (3) hazard management and control, (4) reporting and recording relevant safety and health information, and (5) recording of data and any calculations made.

It is a common practice that each principal hazard is individually risk assessed before the commencement of mining and that the principal hazard management plans are developed in accordance with that assessment to mitigate the risks identified. As discussed previously PHMPs are unique to each operating mine, involve a comprehensive and systematic investigation and analysis of all aspects of risk to health and safety associated with the principal hazard. Following commencement of mining the underlying risk assessments are reviewed periodically prior to the PHMP being reviewed, based on several criteria typically documented within the PHMP (including when a major loss of control event occurs at the mine).

The PHMP must address hazard identification, control selection, control management, review, audit and corrective action to manage risk associated with the principal hazard to within acceptable limits. In general, principal hazard management plans include: (1) legislative requirements, (2) background information about the mine including history of any significant loss of control events, (3) major strata control risks to the operation, (4) underlying risk assessments, (5) geotechnical design guidelines, (6) review requirements for the principal hazard management plan or equivalent, (7) geotechnical characterisation (domains, zones, districts etc.), (8) roles and responsibilities in accordance with the mines organisational structure, (9) all relevant Standard Operating Procedures and Standard Work Procedures, and (10) the site’s critical controls process including methods of assessing and recording the quality of implementation.

Ground control standards

The aim of ground control standards is to ensure that the ground control processes at the mines are carried out to a minimum acceptable standard to ensure safe and economic extraction of reserves and to provide a set of consistent and auditable outputs.

These standards also provide a framework for compliance with the relevant government statutory bodies and internal corporate regulations of the operator[5,6]. Regular audits are conducted at the mines to check that as a minimum these standards are met, and appropriate controls are in place. In general, the requirements of fatal risk standards, which form part of the ground control standards, are summarised in the following risk areas: (1) plant and equipment requirements, (2) system and procedural requirements, and (3) people requirements, which are regularly audited six-monthly or annually[5,6].

Geotechnical design manuals, software and calculators

Since the decline of coal prices, most coal companies in Australia employ geotechnical engineers within their technical services departments to conduct detailed geotechnical designs. These designs are often complex and require specialist skills only attained from specific training and experience. It is an important requirement of these designs that a standard process is followed, and all assumptions and calculations are transparent and auditable, as required by legislation in both New South Wales and Queensland. The internal design manuals and programs must be aligned with these requirements as the minimum unless otherwise determined by risk assessment.

Over the years, numerous pillar and roof support design methodologies have been developed in Australia and elsewhere. These methodologies are based on empirical, analytical and/or numerical methods. As the underlying principles and the databases used in the development of these methodologies vary, their applicability to geotechnical environments also vary, often requiring engineering judgement to determine their suitability. Geotechnical manuals summarise the recommended design methodologies and make recommendations on the applicability of them and the minimum design process maps as well as the acceptable standards that need to be used at mines by all personnel, both employees and contractors. In addition, design manuals make recommendations for monitoring, mapping and hazard plans to ensure that the design can be analysed, reviewed and adjusted if required in a timely manner.

A premise of having a design manual is that a standard design process is followed which is auditable and repeatable. An example of such a design and evaluation process is presented in **Figure 2**. The design manuals typically contain the following sections:

1. overall geotechnical design process (i.e. flow chart);

2. roof support design – recommendations on the input data, serviceability requirements, roof support design strategy in standard and in critical areas, e.g. longwall install and recovery roadways, design criteria, implementation and communication (i.e. support plans), review process and data storage.

3. pillar design – design process, pillar types and serviceability requirements geological and geotechnical data, design methodologies, design criteria (i.e. factor of safety and probability of failure), implementation and communication, review process and storage of data.

4. monitoring – ground deformation monitoring, roof support performance monitoring, stress measurements, critical area audits, surface subsidence monitoring, longwall powered support monitoring, implementation and communication, and monitoring data collection and analysis.

5. mapping and hazard plans – methodology, data requirements and mapping, currency of data, hazard plan presentation and communication.

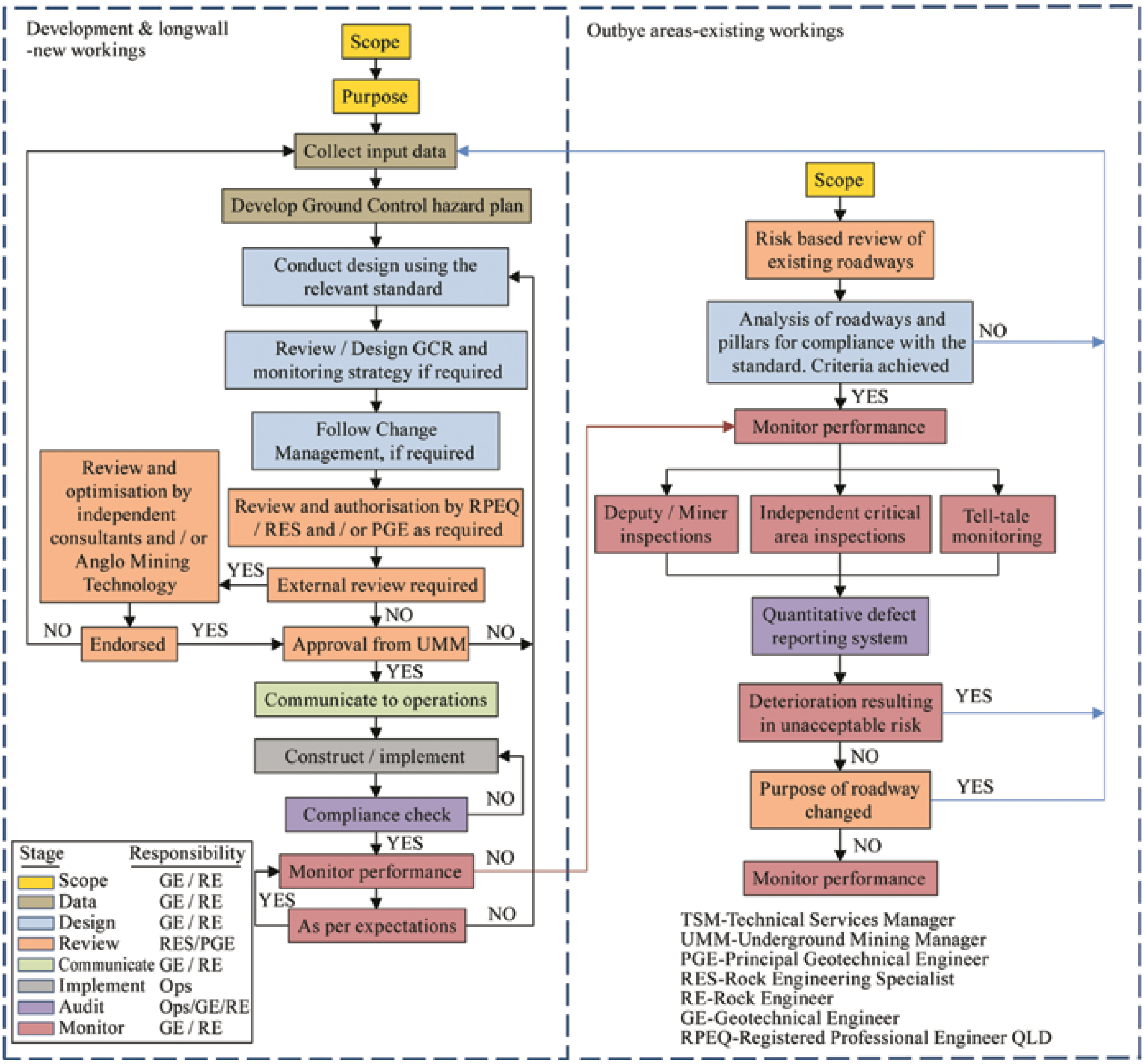


Figure 2: Ground control standard process map (after[7]).

In terms of ground support, there are no universally accepted roof and rib support design methodologies in Australia. Therefore, many mines tend to use a "combined support design methodology" which considers several methods and/or uses one method for the design and then one or more methods to back analyse and check the design. The following methods are generally used: (1) analytical methods for buckling, shear and dead-weight loading, (2) field testing and monitoring (including underground observation), (3) numerical modelling, and (4) rock mass classification and empirical analysis.

For pillar design, there is more uniformity amongst the geotechnical fraternities, who rely on the following methods to design coal pillars based on loading and serviceability requirements: (1) UNSW pillar design methodology, (2) Analysis of Longwall Pillar Stability, (3) Analysis of Longwall Tailgate Serviceability, and (4) 2D and 3D numerical modelling using both finite element method and distinct element method[8,9,10].

Worthy of special mention is the ALTS design methodology, which was initially provided to the Australian coal industry in early 1999 and over a 10 year period was continually refined and updated such the latest version, ALTS 2009 and associated software package, has grown to be the prevalent technique for chain pillar and gateroad ground (roof and rib) support design at most operating longwall mines in Australia[10,11,12].

This is largely because the outputs from ALTS 2009 most accurately reflect the design requirements to provide serviceable gateroads associated with longwall extraction. In addition, ALTS 2009 is relatively quick and straightforward to use allowing typically time poor mine site geotechnical engineers to conduct in house design work with high levels of accuracy, improving both safety and productivity at those mine sites. However, like all design methodologies the geotechnical environment needs to be properly characterised so that the data input parameters (e.g. the coal mine roof rating - CMRR and in situ stress levels) and their potential variation across the area under design consideration is well understood and therefore data input can be selected using appropriate/prudent judgement.

Geological and geotechnical mapping and hazard plans

Mapping and hazard plans are integral parts of an effective ground control management strategy requiring a consistent and standardised process integrated with the daily operations of a mine. The mapping and geotechnical hazard plan standards are usually linked to the operational and planning cycle of the mine assisting in reducing uncertainty around the nature of the rock mass and its impact upon the mine schedule.

In development sections, the geological mapping is conducted regularly immediately behind the development face, typically on a weekly basis. The longwall mapping is more variable depending on the difficulty of the seam, with some mines only mapping the gates before the panel starts while other mines map the longwall face after every shear is taken. Development mapping can be a good indicator for areas of increased risk due to geological and mining induced features but is heavily dependent on its quality and consistency. Best practice requires a second underground inspection by a geotechnical engineer to verify mapping and check for ongoing signs of deterioration prior to utilising the data for design purposes.

Development hazard plans (see **Figure 3** as an example) typically use data from mapping, borehole cores, borehole geophysics and remote sensing such as surface seismic reflection surveying and aeromagnetic surveying. In general, development hazard plans consider the following information: (1) thickness of the seam and seam split, (2) stress environment, (3) depth of cover, (4) roof competency, i.e. uniaxial comprehensive strength and coal mine roof rating, (5) floor competency, i.e. uniaxial comprehensive strength and slake durability, (6) presence, persistence and magnitude of discontinuities (faults, joints, shears, etc.), (7) presence and nature of igneous intrusions, (8) interaction between geological structures, (9) overlying competent rock thickness and strength (e.g. sandstone or basalt channels), (10) dip of the seam, and (11) water or water bearing strata.

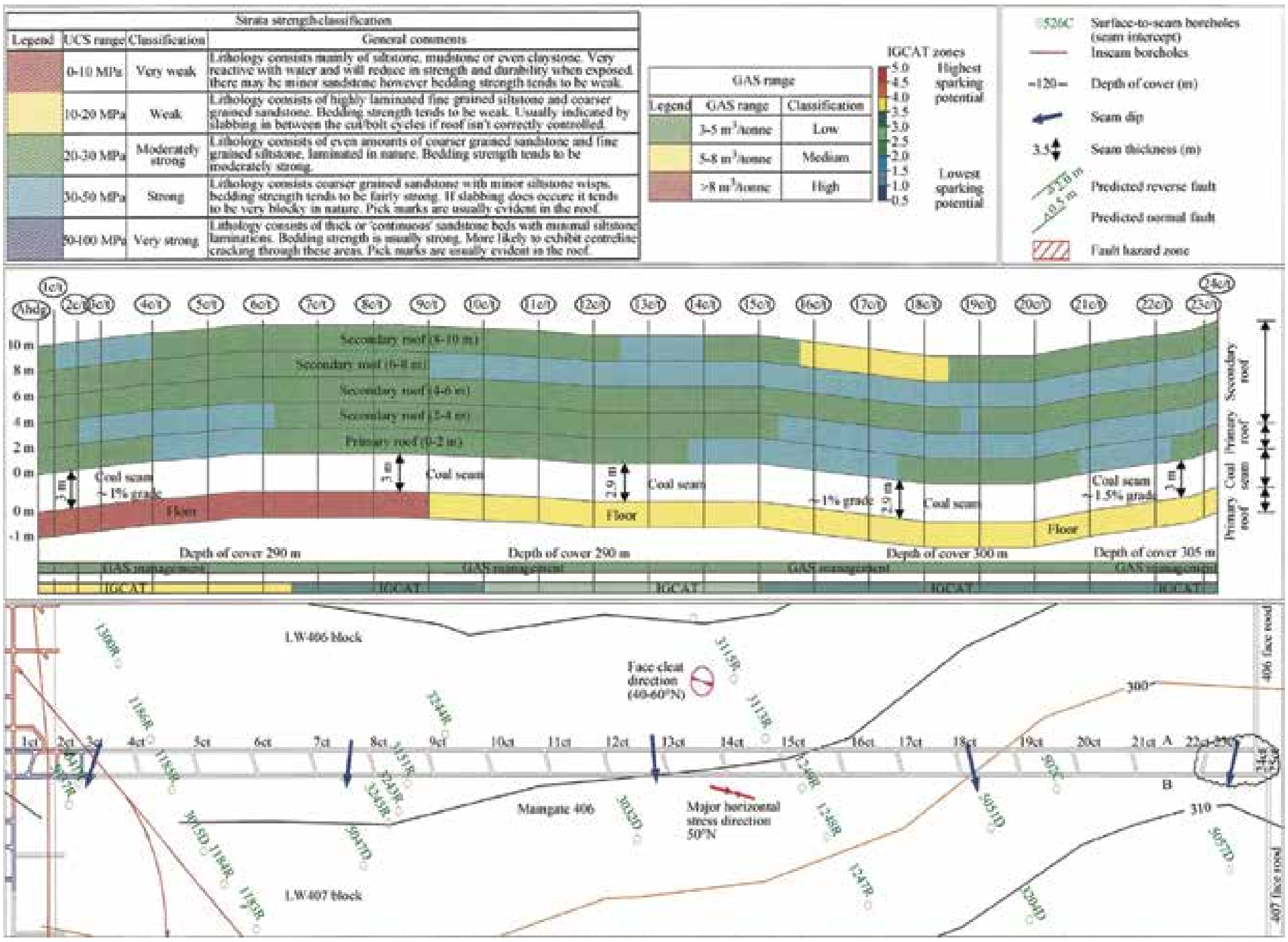
Hazard plans (see **Figure 4** as an example) for secondary extraction refer to: (1) geological structures (reverse or thrust faults, mid-angled structures, structures aligned at a shallow angle to the roadway and areas where two or more geological structures intersect), (2) direction of minor and major geological structures, (3) presence and nature of igneous intrusions, (4) roof competency, i.e. uniaxial comprehensive strength and coal mine roof rating, (5) floor competency, i.e. uniaxial comprehensive strength and slake durability, (6) roadway size, (7) roof slabbing, falls and guttering, (8) roof displacements following the development, (9) horizontal stress direction, magnitude and notch, (10) mine site-specific hazards (i.e., depth of cover, in-seam and multi-seam interactions, installed densities of support, rib spall, changes in seam dip and sandstone or conglomerate channels), (11) installed support densities, (12) off-line cut areas, and (13) installed support.

It is imperative that all available data (historical and recent) are presented on mapping and hazard plans, which are provided prior to the start of any underground development and any secondary extraction. For pre-feasibility and feasibility studies hazard plans are also provided to the project teams. It is also imperative that hazard plans are routinely updated with the most recent information.

Risk management and risk assessments

In the context of this paper, risk management refers to coordinated activities to direct and control an organisation about risk; and risk assessment is the overall process of risk identification, risk analysis and risk evaluation [14].

Overall risk management strategy of coal mines is outlined in the mines' health and risk management plans i.e. PHMP. Risk assessments are an important part of this plan and are used extensively throughout the Australian mining industry to underpin the strategy. There are many publicly



Strata strength classification			
Legend	UCS range	Classification	General comments
	0-10 MPa	Very weak	Lithology consists mainly of siltstone, mudstone or even claystone. Very reactive with water and will reduce in strength and durability when exposed. there may be minor sandstone however bedding strength tends to be weak.
	10-20 MPa	Weak	Lithology consists of highly laminated fine grained siltstone and coarser grained sandstone. Bedding strength tends to be weak. Usually indicated by slabbing in between the cut/bolt cycles if roof isn't correctly controlled.
	20-30 MPa	Moderately strong	Lithology consists of even amounts of coarser grained sandstone and fine grained siltstone, laminated in nature. Bedding strength tends to be moderately strong.
	30-50 MPa	Strong	Lithology consists coarser grained sandstone with minor siltstone wisps. bedding strength tends to be fairly strong. If slabbing does occure it tends to be very blocky in nature. Pick marks are usually evident in the roof.
	50-100 MPa	Very strong	Lithology consists of thick or 'continuous' sandstone beds with minimal siltstone laminations. Bedding strength is usually strong. More likely to exhibit centreline cracking through these areas. Pick marks are usually evident in the roof.

GAS range		
Legend	GAS range	Classification
	3-5 m^3/tonne	Low
	5-8 m^3/tonne	Medium
	>8 m^3/tonne	High

Figure 3: Development strata hazard plan example.

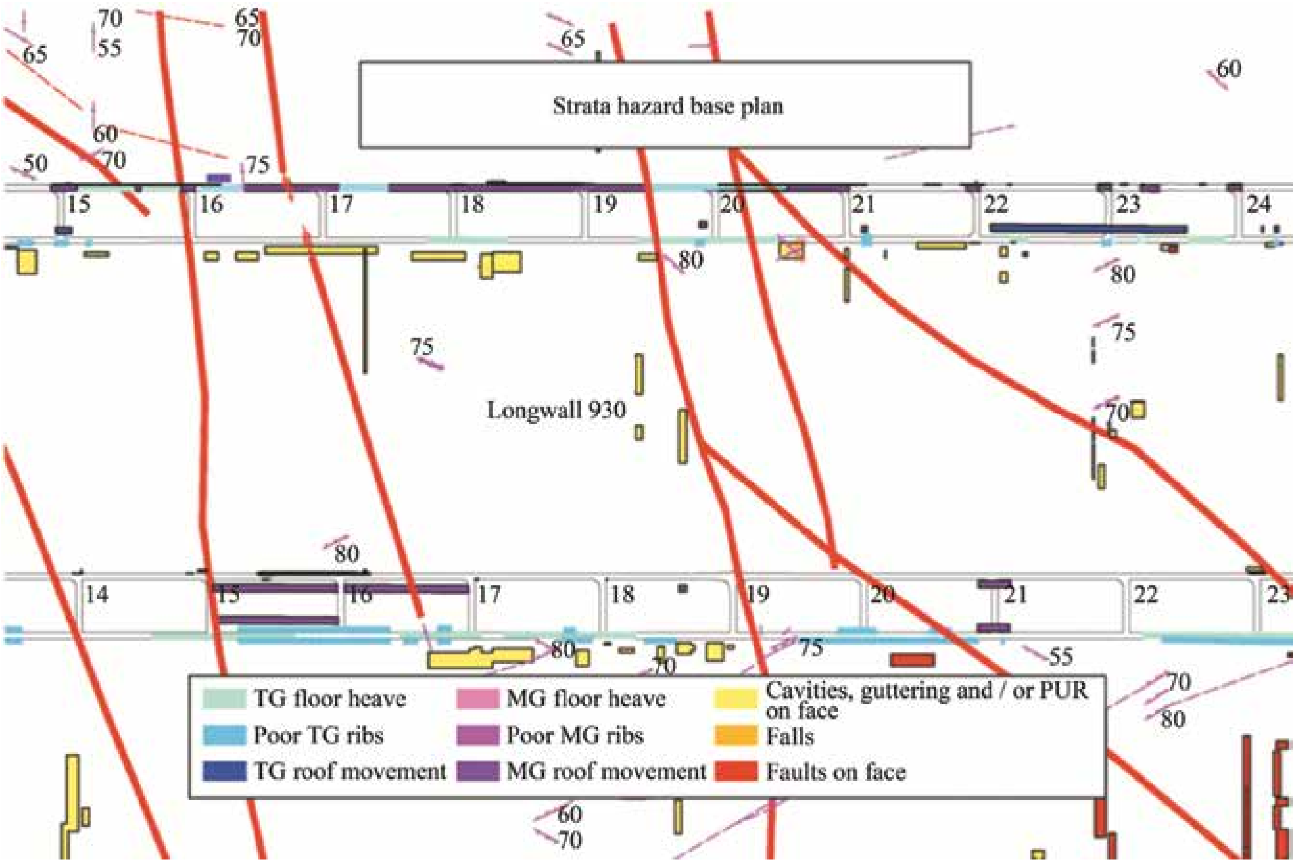


Figure 4: Longwall strata hazard plan example (after[13]).

available publications on risk management and assessment procedures and standards. This paper explains how risk assessments are used in coal mining ground control.

Risk assessments in ground control are conducted in the following stages of mining: (1) during pre-development studies, e.g., pre-feasibility, feasibility, pre-development design studies, (2) when preparing long term mine plans, (3) prior to the development of gateroads or mains sections, (4) prior to the extraction of longwall panels, and (5) in all other circumstances when a specific assessment is warranted, e.g. prior to mining through a structurally disturbed zone or following a major change in mining circumstances since a previous risk assessment.

The aim of the risk assessment is to identify all potential hazards, to rank them and implement the appropriate controls to reduce their impact on safety and productivity. The risk assessments conducted prior to start of development and longwall consider the following information: (1) geology and geotechnical – all potential structures, e.g., faults, dykes, seam thinning and thickening, seam rolls, competent layers in overburden, change in roof and floor competency, and dip of seam and potential stress environment, (2) ventilation, (3) gas, (4) spontaneous combustion, (5) surface infrastructure, cultural heritage, surface vegetation, water bearing structures (e.g. dams), (6) ground water and underground water hazards, (7) previous workings, (8) hazards associated with operating the mining equipment, and (9) other hazards as deemed appropriate.

Many of these hazards are not identified with confidence prior to the start of development. However, many are observable before the commencement of secondary extraction (i.e. longwall retreat) and should be included in the new risk assessment.

Trigger action response plan

A trigger action response plan (TARP) is an essential element of any PGCMS. A TARP is designed and implemented for a specific geotechnical area or domain to deliver a simple set of rules to provide guidance on support requirements, and other actions required as a response to specific visual and/or monitoring ground behaviour. TARPs typically categorise the geological and geotechnical conditions in a "traffic light" system to indicate different risk levels. In addition, TARPs refer to the required responses and responsibilities of all relevant people such as the deputy, mine manager, miner, geotechnical engineer, geologist, etc. This may also include the appropriate level of support to be installed. An example of a longwall strata control TARP is presented in **Figure 5**, which shows the conditions and trigger levels in different geological and geotechnical conditions for the longwall face and the gateroads

Australian coal mines use ground control TARPs for development, outbye areas, longwall face, longwall gateroads, installation roadways and longwall recovery. To be effective trigger action response plans should define: (1) different levels of ground behaviour (triggers), based on key parameters, (2) responses to triggers (changes in monitored parameters and associated actions), and (3) individual responsibilities.

The TARP should be as short and simple as possible ideally not longer than one page. The number of relevant parameters should be distilled to the minimum required to reflect the range of ground behaviour experienced locally. Ultimately, production personnel must have significant input to the documentation and the system, so that common ownership exists.

Data collection

Routine geotechnical testing and sampling

Australian mines rely on extensive geological and geotechnical data for geotechnical designs and for overall ground control management. To ensure that the required data are provided adequately and in a timely manner, mining companies developed guidelines for geotechnical testing and sampling for underground and open cut operations.

These guidelines provide a framework for collecting geotechnical data during drilling in exploration programs to ensure that geotechnical data are measured and recorded systematically to a common standard. This ensures the data can be used reliably for the assessment of rock mass and the evaluation of mine design parameters. The frequency of geotechnical testing is also specified in the guidelines.

The guidelines are usually used in conjunction with mandatory guidelines for core logging to ensure that:

1. Samples will provide adequate representation for the area of interest and/or over the entire lease.

2. Samples are taken from within the correct/target horizons and lithology.

3. The correct tests are undertaken on the samples, to allow for analysis of the conditions likely to be encountered during mining.

4. The correct number of tests are conducted.

5. All other data are recorded to allow for the assessment of the materials, and later calculation of rock mass rating systems e.g. coal mine roof rating.

6. Standard, reliable testing procedures are used in testing and data collection so that minimal uncertainties are introduced into the design during these processes.

7. Data storage is adequate.

It is well-accepted by the Australian coal mining industry that collecting adequate geological and geotechnical data through monitoring, instrumentation, drilling from the surface or underground and through 2D and 3D seismic surveys is necessary to minimise the potential for an uncontrolled fall of ground. Further the consensus is that the cost of the data acquisition, although high, is paid for many folds by improvements to productivity from reduction in uncertainty. Therefore, coal mines usually collect substantial amounts of geological and geotechnical data throughout the life of the mine operations.

<table>
<tr><th>Level</th><th colspan="3">Green-level 1</th></tr>
<tr><td>Condition</td><td colspan="2">LW face
Geology
• Minor faulting in face < 0.3 m
Roof
• Solid roof and floor
• No visible stress cracking in roof
• Break line at rear of sheilds
Face
• Face spall < 0.5 m
• Minor spall ahead of lead drum
Chocks
• No signs of chock yield
• Tip to face < 0.7 m
• Clearance spill plate to canopy > 1 m
• Creep < 500 mm off centre</td><td>Gate conditions
Roof (< 10 m OB of face)
• Minor fretting of roof / rib corner
• No additional guttering
• No signs of additional loading on support
• No roof talk
Ribs (< 10 m OB of face)
• Minor additional rib spall < 300 mm
• Installed rib support controlling rib conditions
Telltales (> 10 m from face)
Additional movement < 10 mm since stable conditions
Gateroad support
• Support installed to design > 200 m OB face</td></tr>
<tr><th>Level</th><th>Yellow-level 2</th><th>Orange-level 3</th><th>Red-level 4</th></tr>
<tr><td>Conditions – Face</td><td>Geology
• Faulting in face > 0.3 m but < 0.7 m
• Fault planes converging
Roof
• Cracking in roof
• Break line over shields
• Minor cavity not affecting chock setting (eg < 0.5 m for length of 10 chocks)
Face
• Face spall > 0.5 m bur < 1 m for length of more than 10 chocks
• Spall < 10 chocks ahead of lead drum
Chocks
• Area of chocks yielding
• Tip to face > 0.7 m but < 1 m
• Clearance spill plate to canopy < 1 m
• Creep < 500 mm off centre</td><td>Geology
• Faulting in face > 0.7 m
Roof
• Major cracking in roof
• Break line nearing face
• Cavity affecting chock setting (eg > 0.5 m for a length of 10 chocks)
Face
• Face spall > 1 m for more than 10 chocks
• Spall > 10 chocks ahead of lead drum
Chocks
• Chocks on constant yield across face
• Tip to face > 1 m but < 1.5 m
• Clearance spill plate to canopy < 0.75 m</td><td>• Major strata failure causing stoppage of production</td></tr>
<tr><td>Conditions – Cates</td><td>Roof
• Minor fresh cracking
• Additional fretting of roof / rib corner 10-20 m OB face
• Minor signs of roof loading on support 10-20 m OB face
• Additional guttering 200-300 mm < 5 m OB face on one side only
• Relatively infrequent roof talk
Ribs
• Additional rib spall 300-750 mm (>10 m but < 20 m OB face)
• Rib bolt plates beginning to buckle or break
Telltales (> 10 m from face)
• Additional movement > 10 mm but < 40 mm since stable conditions</td><td>Roof
• Fresh fracturing evident
• Additional fretting of roof / rib corner > 20 m OB face
• Roof bolts and plates deforming Tendons birdcaging or heavily deformed plates. Numerous broken bolts. Standing support showing significant loading
• Roof talk
• Additional guttering > 300 mm > 5 m OB face on one or both sides of roadway
Ribs
• Additional rib spall > 750 mm (> 20 m OB face)
• Rib bolt plates buckled and breaking
• Area more than 10 m long (> 10 m OB face) with more than half the rib bolts broken or majorly exposed
Telltales (> 10 m from face)
Additional movement > 40 mm since stable conditions</td><td>Roof
• Fall of roof ahead of face in either gate
Ribs
• Fall of rib greater than rib bolt length and further than 5 m OB face</td></tr>
</table>

Figure 5: An example of a longwall Trigger Action Response Plan. Note: the responses and the roles and responsibilities of relevant people are not included in this example.

Roof bolt and accessories – standard specifications, evaluation and testing procedures

The Australian mining industry has access to a variety of ground support products. A concern with many support products is that the mines must make sure that the products supplied meet the minimum specifications outlined in ground support selection and testing standards. Therefore, in recent years, there has been an increasing emphasis on quality assurance and quality control testing.

Ground support selection and testing standards specify the dimensional, material and testing requirements for roof bolts and cables and accessories used at mines, including the steel bars, cables, nuts, plates, resin and mesh for use in a complete assembly. These standards also specify the requirement for the suppliers to conduct routine tests to ensure that the roof support components comply with the minimum standards.

There are several factors that contribute to the under-performance of installed ground support elements. These factors should be controlled to specified tolerances by regular, systematic quality control procedures. The factors that can affect the performance of a roof bolt support system can be classified into two areas: indirect controls and direct controls.

The indirect controls are related to suppliers' quality control procedures, such as metallurgical properties of roof bolts, deformation pattern of roof bolts, chemicals used in the manufacturing process of resin capsules and the consistency of these properties. The standards require that the suppliers' quality control procedures are audited routinely and that the manufacturer's quality control procedures should comply with the relevant ISO and/or Australian Standards. All quality assurance and quality control test results are also provided to the site geotechnical engineers for random checking and record keeping.

The direct controls can be divided into three distinct groups: ground support and accessories, compliance with the design, and quality of installation.

The minimum specifications of the standards include: (1) roof bolts and cables – chemical composition, length, profile, straightness, finish, colour coding, colour coding, nut break out, mechanical performance, plates (washers), nuts, drill bits, rods and spanners, (2) resin – capsule size, shelf life, gel and setting time, bond strength and system stiffness, colour coding, packaging, uniaxial compressive strength, elastic modulus, creep, shear strength, push test, capsule diameter, capsule length, and freedom from leakage, and (3) mesh and straps – compliance with relevant Australian standards, grade, fire resistance, profile, dimensions, and yield strength and storage.

For the above specifications, regular testing and audit requirements are also prescribed in the standards. These standards are also used in supply contracts to ensure that the standards are obligatory. All required tests are usually conducted by the suppliers.

Many mines may also require on site quality assurance and quality control testing conducted by independent companies to ensure that all support products comply with the minimum standards.

Monitoring

Regular monitoring

Monitoring is probably the most important element of a PGCMS to prevent uncontrolled falls of ground. Therefore, not surprisingly, ground deformation and support effectiveness monitoring are also a requirement of Australian legislation. In Australian coal mines, strata monitoring consists of both observation (qualitative) and measurement (quantitative). Both methods are required primarily for design verification but also for identifying areas of non-conformance (such as unpredicted anomalies) so that remedial measures can be applied in a timely manner.

Monitoring provides different qualities of data to different users (i.e. operators for decision making and geotechnical engineers for design purposes). Ground monitoring is undertaken to: (1) aid in exploration, (2) establish benchmark data for environmental approval and licensing purposes, (3) determine properties for input into mine design, (4) validate mine design, (5) validate the quality of ground support hardware, (6) validate the quality of ground support installations, (7) research the unknown, (8) provide timely warning of deviation from predicted ground conditions and design performance, both in the short and long term, and (9) identify, quantify and verify mining effects, impacts and consequences[13].

Australian mines conduct extensive monitoring to understand the ground behaviour and to measure all parameters that can result in strata problems. These include: (1) pre-mining stresses, (2) stress changes, (3) displacements of roof, ribs, floor and pillars, (4) reinforcement installation procedures, (5) permeability of strata, (6) longwall shield loading, (7) performance or condition of pillars, and (8) installed support.

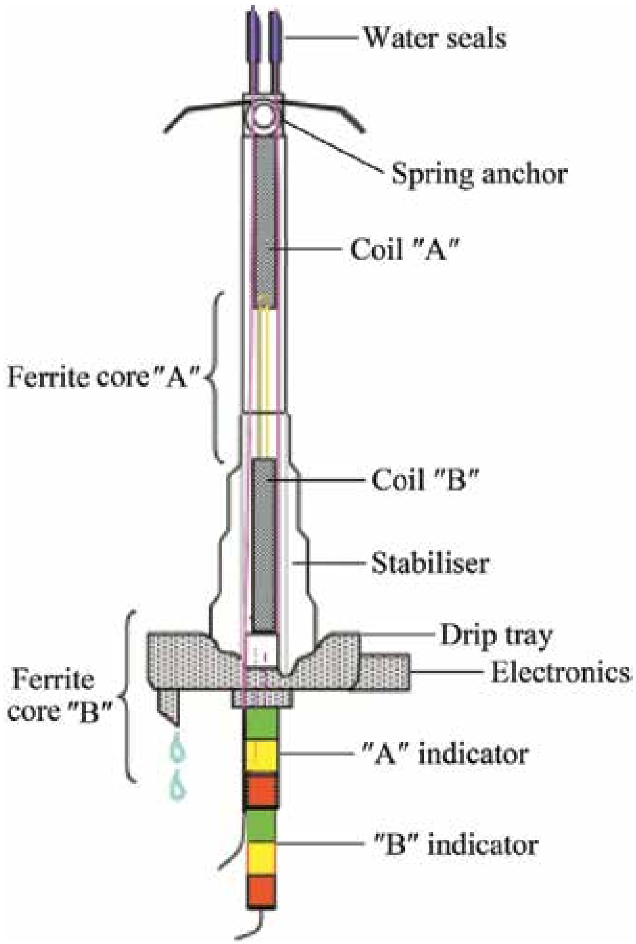


Figure 6: A Two-anchor Remote Reading Tell Tale schematic (after[15]).

Ideally, monitoring systems need to be designed and implemented to provide timely, fail-safe warning of the development of critical ground conditions so that personnel and equipment are not exposed to burial, entrapment, windblast, dust, and noxious and flammable atmospheres. Gaps in knowledge and technology currently prevent these monitoring goals from being fully achieved[13]. The use of real time strata control displacement monitoring (extensometers), as shown in **Figure 6**, is continuing to grow in popularity as the technology develops, however it is still not common or accepted practice.

Permit to mine process

A Permit to Mine (also known as Authority to Mine or PTM) is a site-based process that identifies the principal mining hazards and controls for each new mining area. This typically includes expected ground conditions, ground support requirements, gas drainage and ventilation compliance requirements, inrush potential, surface structures and restrictions. A Permit to Mine is developed before mining takes place in any area. The process originated as a tool to assist in controlling the risk of outburst but has since evolved to cover all principal hazards.

All relevant information is listed, reviewed and authorised by all parties (i.e., mine gas and ventilation engineer, geotechnical engineer, geologist, surveyor, development and/or longwall crews and mine manager) to indicate that the identified risks are considered and controlled. This allows the mine manager to make a well-informed decision on the expected hazards and ensure the appropriate controls are in place prior to approving mining to commence. If used appropriately, this system is powerful in identifying and mitigating risks before any mining takes place, hence it is universally adopted and covers all technical risk factors including ground control.

Regular geotechnical critical area inspection

Historically, strata control failures in mine access roadways that cease production are not uncommon, and often may have been preventable with earlier identification and intervention. Regular geotechnical critical area reviews ensure that certain critical areas of the mine are regularly inspected so that the ground support in those areas is in line with the mine's minimum support requirements and any ongoing deterioration is recognised. A critical area is defined as areas of the mine where strata deterioration or failure may cause process delays or expose people or equipment to potential harm.

For active panels, there are routine processes in place, as stipulated in principal hazard management plans and mine inspection regimes, to manage the hazards associated with ground control. Therefore, the critical area inspections are specifically for outbye areas of the mine where inspection is less frequent.

Critical area inspections involve simple visual observations of ground conditions and identification of deteriorating ground conditions so that those areas are included in the mine's maintenance scheme e.g. replacing corroded or damaged support elements. The inspections are usually performed by an independent third party who are not familiar with the area so that an objective and unbiased inspection of the ground conditions is conducted and recorded.

Weekly reporting system

Weekly reporting systems are another valuable tool in ground control management to ensure that all development and secondary extraction panels are inspected by geologists and/or geotechnical engineers and a standard inspection sheet is filled in. Following the inspections, a standard weekly report is produced and distributed to all relevant parties (i.e., development crews, longwall crews and management) to indicate the areas of non-compliance with ground support designs given in trigger action response plans. The frequency of response plan triggers and installed support are also mapped to ensure that the trigger levels in the response plans are appropriate for the conditions in the panel.

The geological mapping of the panel is also conducted during these inspections and a mine plan with geological structures, installed support and general geotechnical conditions in the panel are included in weekly reports.

Training

Ground control management in Australian mines involves many critical steps and requirements. To ensure that these steps are well-understood, and the requirements are met internal and external training programs are provided to geotechnical engineers as well as the workforce. A training scheme is also a requirement of Australian legislation.

All Australian mines have a system in place to ensure that all personnel working underground are competent, trained and authorised to perform the geotechnical tasks assigned to them. There is also an on-the-job training and assessment process for mine workers. All employees are also trained by a geotechnical engineer in the following areas: (1) support design principles, (2) principal hazard management plan requirements, (3) identification of geological anomalies which contribute to weaker ground conditions, and (4) trigger action response plans.

Geotechnical engineers are usually responsible for ensuring that refresher training courses are provided regularly to all employees. The training of junior geotechnical engineers involves in-house training sessions and external courses. In-house training sessions involve training in ground control management strategy and the geotechnical design processes. External courses are usually structured around new developments in geotechnical engineering. Registered professional engineers are required to complete and demonstrate continual professional development (CPD) to a level determined by the governing body to remain registered and audited regularly.

Figure 7: Typical critical controls process flow chart[16].

Critical controls

As risk management has evolved over time so have the checks and balances used to assess the health of the system. One aspect that is now elementary to a PGCMS is the implementation of a critical controls monitoring or critical control verification process. A critical control (CC) is defined as a risk control that is either crucial to preventing an event from occurring or mitigating the consequences of an event **Figure 7**[16]. Each mining company has its own slight variation on the CC process however it will typically consist of a series of verification activities to be performed periodically on the identified CCs. This verification will be completed by the risk owner within the site management team and compliance reported through to corporate. Where there are deficiencies identified action plans must be developed and assigned to relevant persons to ensure the deficiency is rectified. An example for ground control is the critical controls associated with geotechnical design. A universal critical control for geotechnical design is that each design is completed, and peer reviewed by a competent person in accordance with the PHMP and ground control design guidelines. Evidence of this process must be available for each design currently being implemented at the mine (typically a peer review sign off form). Another universal CC is the monitoring of underground excavations according to a scheduled inspection regime. Documentation must be supplied as evidence that both the monitoring, and the appropriate responses to this monitoring, are being carried out in accordance with the relevant documentation.

Review and investigation

Investigation of accidents and incidents

In ground control management, accidents and incidents are related to fall of ground which is defined as an unplanned movement of ground that results in a failure within the ground control system with the potential to affect safety and production or has a business cost.

Investigations of accidents and incidents are required by the Australian legislation. Queensland Coal Mining Safety and Health Regulation states that a coal mine's safety and health management system must provide the following: (1) the procedure for investigating accidents and incidents at the mine, (2) making the investigation findings available to the mine's workers, and (3) implementing corrective action for accidents and incidents[17].

Many accident and incident investigations involve using the Incident Cause Analysis Method, which provides a logic towards incident and accident causation and supports the notion that most incidents and accidents are rarely caused by a single act or condition, but rather by a number of factors working together[18].

Audits and review of ground control management process

A PGCMS has many elements and it is a live process. The implementation of this strategy is not a simple task; it requires resources and time. To ensure that all operations are at comparable levels in implementation of the strategy,

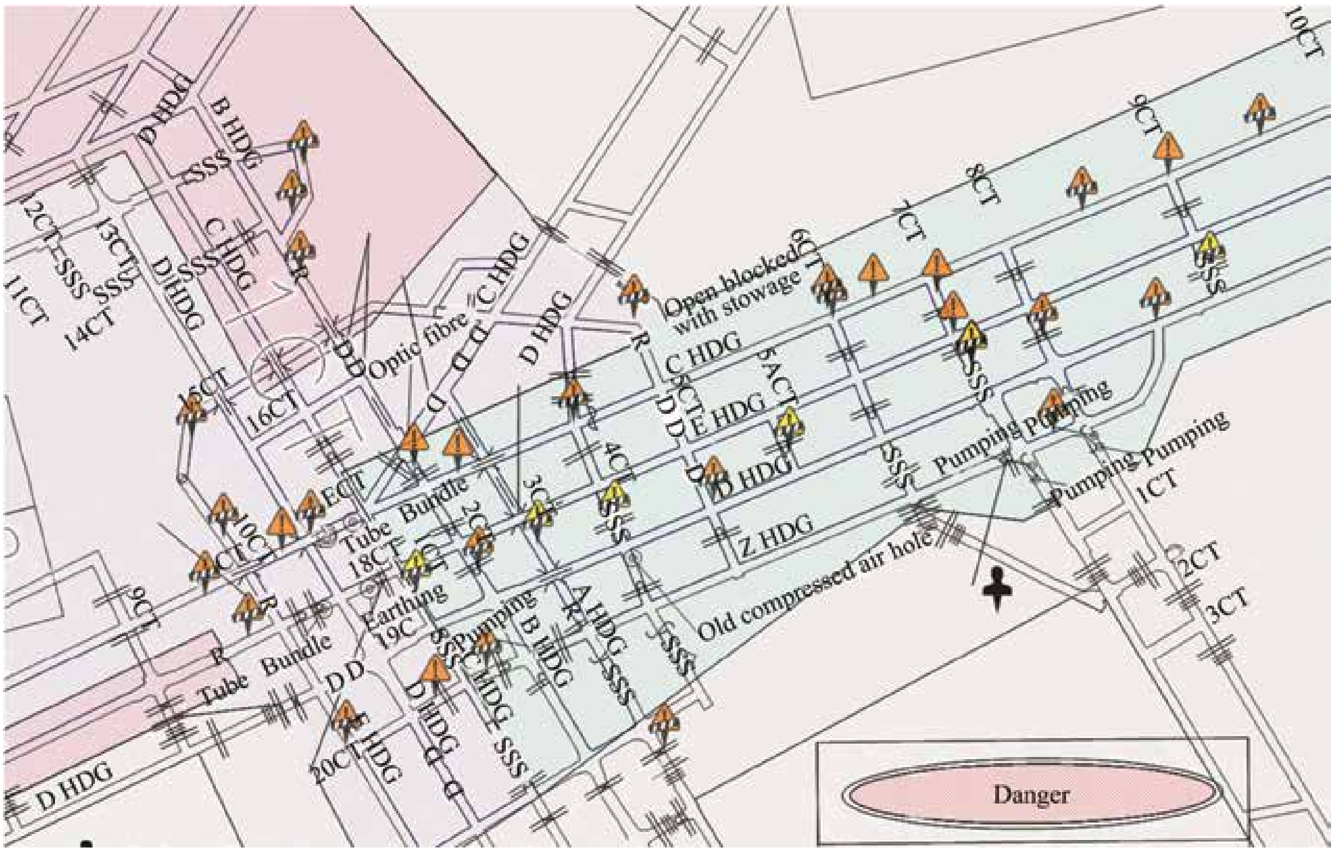


Figure 8: Example strata defect hazard map.

regular internal and external audits are conducted by mining companies.

Every element of a ground control strategy is also reviewed regularly to ensure each element is still effective and applicable to the environment the mine is operating in. In an event of a major failure (such as fall of ground), reviews of the complete process are also conducted, and this requirement is included within the ground control management plan.

Strata defect hazard register

Although the risk of ground control failure is highest when within a certain distance of the active mining face, the deterioration of ground support over time has also become a key element of a PGCMS. With large-scale modern mines in operation for many decades the deterioration of ground support and the associated conditions increases with time due to weathering of the ground, weathering of support elements and damage due to impact from mobile equipment. Due to this deterioration over time and an absence of response there have been several large failures generally in outbye areas of mines that incurred significant business losses and unacceptable levels of exposure to coal mine workers. Many operations now utilise a system that includes the regular reporting, inspection and remediation tracking for identified defective strata support in outbye areas of the mines (e.g. Figure 8). These strata defects are also tracked in global information system (GIS) enabled maps so that the defects may be identified prior to planning tasks to be undertaken in certain areas.

CONCLUSIONS

Ground failures pose a high-level risk to both individuals and production in underground coal mines. Therefore, Australian mines have developed over time what is considered the best practice pro-active ground control management strategy globally, to provide work areas both safe for employees and to minimise unplanned delays to production.

This paper summarised the typical best practice pro-active ground control management strategy used in Australian underground coal mines and detailed the critical elements. A pro-active ground control management strategy is not only about roof support and pillar designs. It involves many critical steps. Applications of these steps vary significantly by the size of a mine and the size of a mining company. Yet all ground control strategies are required to comply with and demonstrate compliance with the relevant Australian legislation. Although not all Australian coal mines apply all the elements outlined above, most of the mines do have similar systems that they utilise in daily ground control management. This requires a high level of onsite geotechnical knowledge and skills with most companies now employing several geotechnical engineers and geologists at each mine site to ensure the PGCMS is implemented effectively.

As research into ground control continues to improve, so does the application of ground control strategy and its elements with emphasis on roof and rib support designs, and technology for instrumentation and monitoring. The material presented in this paper gives guidance to mining engineers in other countries for achieving safe and productive coal extraction similar to that being achieved in Australia through a PGCMS.

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Last hopes for demand growth set to disappoint exporters

The world's largest thermal coal exporter is eyeing new markets as its largest export destinations threaten to cut imports.

Indonesia is targeting Bangladesh, Pakistan and Vietnam as China and India seek to curb thermal coal imports into the long term.

As a result, Australian and South African thermal coal exporters aren't going to have it all their own way in seeking increased deliveries to markets like Vietnam. Furthermore, it looks increasingly like Vietnam, Bangladesh and Pakistan are going to disappoint those hoping for more demand growth from these countries.

A key problem faced by thermal coal exporters in Australia, Indonesia and South Africa is that China and India are increasingly prioritising their own very large domestic coal mining industries. Both nations are keen to reduce imports as much as possible for energy security reasons and to protect domestic jobs.

Other established coal export destinations are also likely to reduce imports in the long term.

South Korea has been the third largest thermal coal export destination for Australia, Indonesia and South Africa. However, in September 2020 President Moon Jae-in announced that 30 coal-fired power plants will be closed by 2034 and wind and solar capacity tripled by 2025.

Meanwhile Japan is now planning the closure of 100 coal-fired power units by 2030 as it gears up for its own push into offshore wind.

Bangladesh set to abandon its coal push

Bangladesh's coal power capacity will likely now be limited to what is already under construction

With one of the largest coal power project pipelines in the world, Bangladesh has been considered a significant source of future demand growth for thermal coal in the Asian seaborne market.

However, that hope looks like it is about to come to an abrupt end now that the nation's power ministry has sought approval from the Prime Minister to cancel 13,000 megawatts of coal power plans.

The relative expense of coal-fired power compared to new energy technology, and increasing difficulties securing finance for coal projects are behind Bangladesh's sudden shift. Bangladesh's coal power capacity will likely now be limited to what is already under construction – a major boom in coal imports to help replace declines elsewhere won't now happen.

Vietnam prepares to dial down coal focus

Increased export focus on Vietnaam this year highlights how this nation is becoming a battleground market as the Asian seaborne thermal coal pond threatens to shrink significantly in the long term.

Although Vietnam has significantly increased coal imports recently, it now looks like nation's next long term power plan will further disappoint the thermal coal sector.

A boost in renewable energy focus along with curtailment of coal-fired power additions are set to be key features of Vietnam's soon-to-be-finalised Power Development Pan VIII.

Vietnam won't be able to replace markets the main exporters are set to lose elsewhere

This follows the Vietnamese National Steering Committee for Power Development's recommendation that 15 gigawatts of planned coal projects be scrapped as renewables get cheaper and more banks pull out of coal power financing.

Coal power plants currently under construction in Vietnam will continue, increasing the demand for thermal coal imports in the medium term, but with Indonesia, Australia and South Africa all targeting the nation, it won't be able to replace markets the main exporters are set to lose elsewhere in Asia.

The International Energy Agency's (IEA) most recent World Energy Outlook – often quoted by the coal industry – outlines a 15% decline in the global thermal coal trade by 2030 under its central Stated Policies Scenario. Under its Sustainable Development Scenario, the decline is 56% by 2030.

According to the IEA, it is increasing coal exports to Asian markets outside of Japan, China, South Korea and Taiwan that will slow thermal coal's decline under its central scenario.

With the recent moves by Vietnam, Bangladesh and Pakistan that is looking less likely and the decline of seaborne thermal coal threatens to look more like the accelerated decline in the Sustainable Development Scenario.

The thermal coal industry can no longer claim that growth markets ensure a rosy future for exports.


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Reasonable coal pillar design and remote-control mining technology for highwall residual coal resources

Highwall mining (HWM) technology is an efficient method for exploiting residual coal resources in Chinese open-pit coal mines. However, on-site personnel and equipment can be damaged by the instability of the highwall mining residual coal pillars and subsidence of final end-walls. This paper considers the geological conditions of an open-pit mine in Shendong Coal Field (China) in order to prevent overlying rock fall accidents; the Mark-Bieniawski formula and the FLAC3D numerical simulation are used to analyse reasonable coal pillar widths outside and under the road, which were determined to be 1.7 m and 1.3 m, respectively. Using the EBH132 cantilever excavator for remote control mining, the field experiment shows that the recovery ratio of highwall residual coal resources was over 67%; hence, safety, efficiency and high recovery ratio of highwall mining were achieved for the residual coal resources of an open-pit mine.

INTRODUCTION

The slope angle, mining boundary, changes of the coal seam thickness, etc. are the main reasons that many coal resources remain under the end-walls in open-pit coal mines[1-3]. Because of the undeveloped mining technology and low economic benefits, those resources were discarded or exploited using the room and pillar method with low recovery ratios. This issue causes waste of resources and introduces safety risks such as the spontaneous combustion of the coal seam, surface vegetation destruction, rock fall accidents, slope landslides and large-area collapse because of the instability of the residual pillars. Since highwall mining has the advantages of a high recovery ratio, easy manoeuvrability, safety and low production cost, states and enterprises have paid increasing attention to this method[4-6]. There is valuable and challenging research on how to recycle the residual coal under the end-walls safely and efficiently[7,8].

Highwall mining is a technique to obtain additional coal recovery after the economic strip limit is reached in surface mining. It involves the 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 maximum pulling capacity of the machine, traction of the cutting head, and material conveying, all of which limit the penetration depth. The maximum penetration is greater for flatter slopes and decreases for slopes near the threshold of the maximum machine operating angle. Most highwall mining operations are relatively flat with slight undulations in the seam; therefore, the highwall mining pillar design criteria apply 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.

To recover coal remnants around the end-walls, an underground mining system is normally adopted by excavating some adits into the end-walls[9-11]. However, because of the small area of residual coal resource and poor production conditions, it is not beneficial for the layout of the traditional longwall working face. If the full caving method is used to manage the coal roof, it will cause a larger ground subsidence, which is not conducive to the stability of the slope. Zonal mining is commonly used in regions where only minor ground placement is permissible. With this method, the main haulage and ventilation roadways are designed and excavated from the exposed position of the coal seam in the end-walls. For the irregular and small area of an end-wall residual resource, the costs remain high[12-14]. Thus, it is difficult to exploit small-area residual coal resources with conventional techniques, but remote control technology can be a good solution to this problem[15,16]. Many countries throughout the world have been investigating remote control coal mining technologies and their equipment, which is shown in **Figure 1**. This technology has been developed after much field practice[17-19]. Moreover, the mining parameters significantly affect the stability of the end-wall in the mining process of the residual coal resource. Hence, to ensure the stability of the end-wall and minimum sinkage for the upper highway, the mining parameters must be analysed.

A reasonable mining pillar for highwall mining is conducive to safe and efficient mining. The highwall mining pillar design is a direct function of the coal strength, opening height, opening width, and depth of cover. An elasto-plastic model suitable for the analysis of coal pillars has been developed and implemented in both two- and three-dimensional finite-element codes by Fama *et al*.[20]. The use of the local mine stiffness concept can provide added confidence in a highwall mining panel layout design[21]. Web and barrier pillar recommendations for close-proximity multiple-seam highwall mining were studied by Mark[22,23]. Perry *et al*.[24] studied the effect of the highwall mining progression on the web and barrier pillar stability. Using numerical modelling tools, a correction factor was suggested in the empirical pillar strength equation for slender pillars with width-to-height ratios less than unity[25]. However, current studies rarely consider the effect of the roads on the highwall and pillar design. The effect of the coal trucks on the road on the stability of coal pillars was also of less consideration.

Both theoretical analysis and numerical simulation are used in this paper to calculate the reasonable width of pillars based on the background of highwall mining in a coal mine. The EBH132 cantilever excavator was used for the remote-control mining. The safe, efficient and high-recovery-ratio highwall mining was achieved for the residual coal resources of an open-pit mine.

ENGINEERING BACKGROUND

The study coal mine is located in Inner Mongolia Autonomous Province, China. Its north western boundary borders on another coal mine, and the southwestern boundary is linked to a highway. There is a 50-80 m wide, 390 m long reservation coal resource in the north western boundary due to the design requirement for the security pillar, boundary stage and road for transportation above the end slope. In the southwestern boundary, there are highways, 11-kVA high-voltage power lines, a 35-kVA electrical substation, boundary stage and road for transportation above the end slope, which resulted in the 180 m wide residual coal resources. It is difficult to exploit these coal resources using only conventional mining techniques. Thus, highwall mining technology was used in this paper to excavate the residual coal resource and improve the recovery rate.

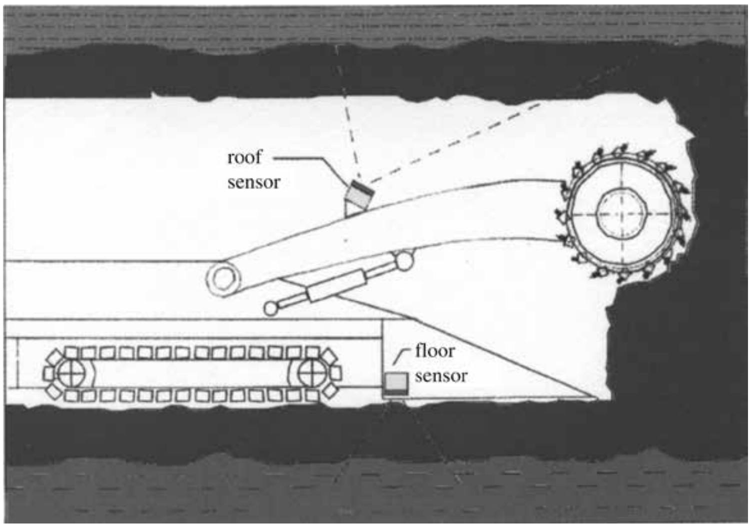

Figure 1: Horizon control problem. (Here, the underlying objective is to keep the mining machine in the coal seam to maximize the coal recovery. The black bands represent coal, and the textured zones represent the tuff.)

The area of highwall mining is located in the southwestern boundary of the 2# coal seam open pit. It was laid out along the road and 145-240 m away from the road. The surface of the mining area is the sand dunes, whose ground elevation is 1300-1330 m, with no ground water or building. The slope angle is 45°, and the detail of the topographic map is shown in **Fgure 2**.

The working face is located in the 2# coal seam, and the average thickness is 3.5 m. The dip angle is 1-2°, and the density is 1.4 t m−3. The coal seam is simple with no dirt band. The roof is sand and fine sand mud; the interbed and floor are mainly sandy mudstone with partly argillaceous siltstone.

The main problem of the highwall mining technology is the layout of roadways. In this paper, remote control technology is used, and the excavate width is determined by the size of the EBH132 cantilever

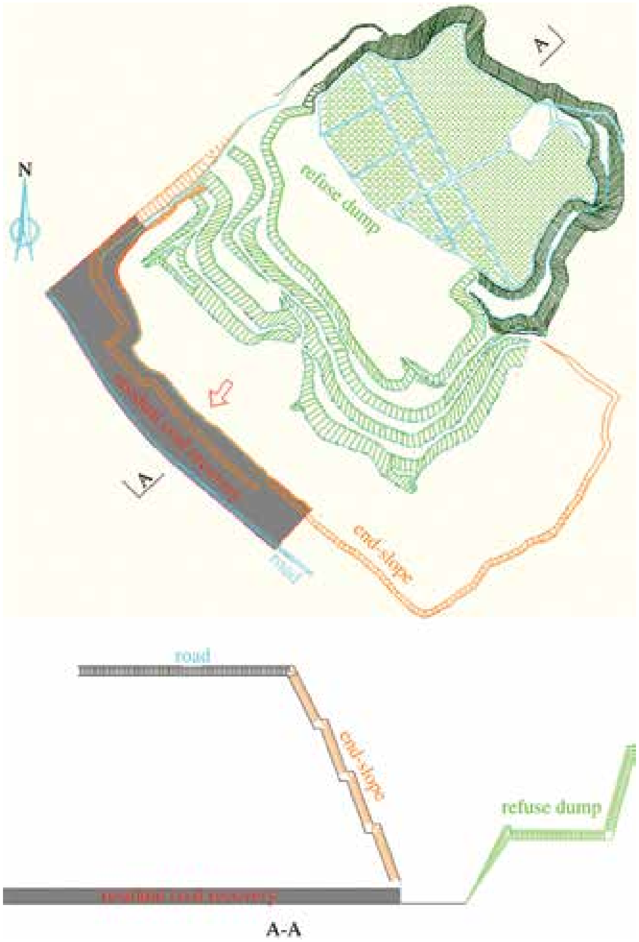

Figure 2: Topographic map of the highwall area.

excavator. There is no need to extract the roadway for humans, which can simplify the mining system to achieve efficient mining. However, there is no permanent support in the roadway for the remote-control mining. Thus, the stability of the overburden structure mainly depends on the stability of the retained coal pillars, and a reasonable pillar width ensures the safety of mining and increases the recovery ratio. Thus, the reasonable coal pillar design and remote-control technology for the HWM technology were studied in this paper.

REASONABLE COAL PILLAR DESIGN TO PREVENT ROCK FALL ACCIDENTS

Numerical simulation

Working status of the coal pillar

The working status of the coal pillar mainly includes three different situations, as shown in **Figure 3**. Mohr-Coulomb failure criteria are used in the numerical model. A more plastic zone in the pillar indicates a lower bearing strength of the pillar. However, the remaining plastic zone has the residual strength to support the elastic zone, as shown in **Figure 3b**. When the plastic zone cuts through the pillar, the plastic penetration area does not have sufficient strength to support the roadway, as shown in **Figure 3c**. Thus, the safety conditions of the pillar are that the plastic zones do not cut through the pillar. **Figure 3a,b** satisfies the requirement. However, if the coal pillar width is too large as **Figure 3a** shows, it will waste the resource and reduce the resource recovery. In general, the reasonable width of coal pillar is shown in **Figure 3b**: exploit as much coal resource as possible under the safety condition of the pillars.

Simulation model parameters

To accurately simulate the deformation and failure characteristics of coal and rock in coal mining, the paper introduces the software FLAC3D, which is based on the finite difference method, to establish the model. In this model, the rock stratum is represented by Mohr-Coulomb model, the coal seam is described by the strain softening model, the cohesion and friction angle of the coal seam degrades as the plastic shear strain increases, and these factors are assigned the residual values when the plastic shear strain reaches 0.01; the physical and mechanical parameters of the coal and rock in the model are listed in **Table 1**. The numerical model and selected properties were calibrated through comparison using the coal uniaxial compressive strength test[26]. **Figure 4** shows a good consistency between the numerical results and the laboratory test for the stress-strain curve and failure mode of the sample. The maximum shear strain in the numerical model shows an X-shaped failure mode of the sample, which is consistent with the laboratory test.

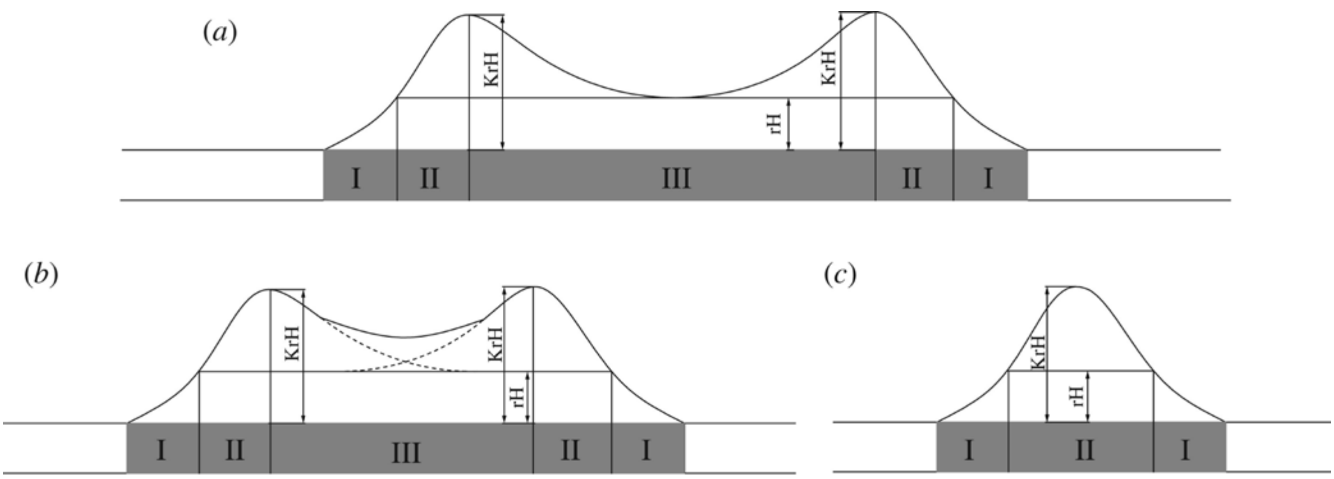

Figure 3: Plastic zone with different coal pillar widths. (a) Too large coal pillar width, (b) appropriate coal pillar width and (c) too small coal pillar width. I – fractured zone; II – plastic zone; III – elastic zone.

Table 1: Physical and mechanical parameters of the coal and rock.

no.	lithology	thickness (m)	density (kg m−3)	bulk modulus (GPa)	shear modulus (GPa)	cohesion (MPa)	internal friction angle (°)	tensile strength (MPa)
1	aeolian sand	20.0	2200	0.5	0.3	0.8	10	0.5
2	sandy mudstone and sandstone interbed	26.5	2400	6.7	2.7	2.9	28	1.3
3	2# coal seam	3.5	1400	1.2	0.7	1.1 (0.11)a	30 (20)a	1.0
4	sandy mudstone	20.0	2450	9.6	4.4	3.5	29	2.3

Numbers in the parentheses are residual values.

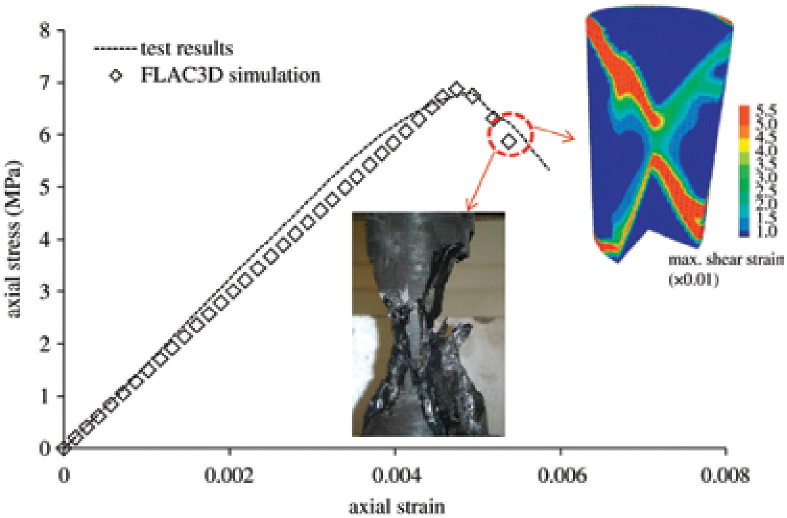

Figure 4: Stress-strain curves and failure mode of the laboratory test and numerical simulation in the coal UCS test.

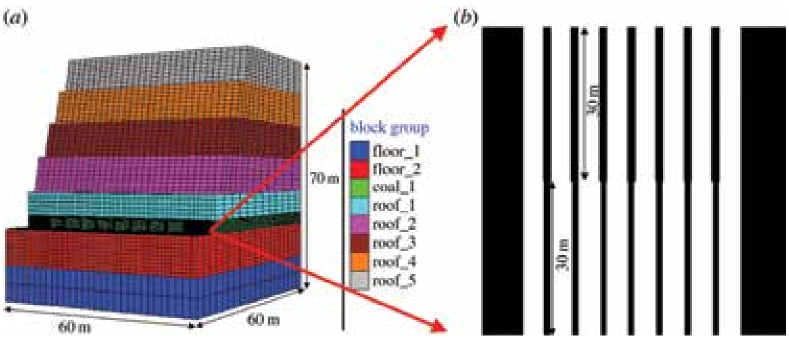

Figure 5: Three-dimensional model and sectional view of the coal seam. (a) Simulation model and (b) coal seam cross profile.

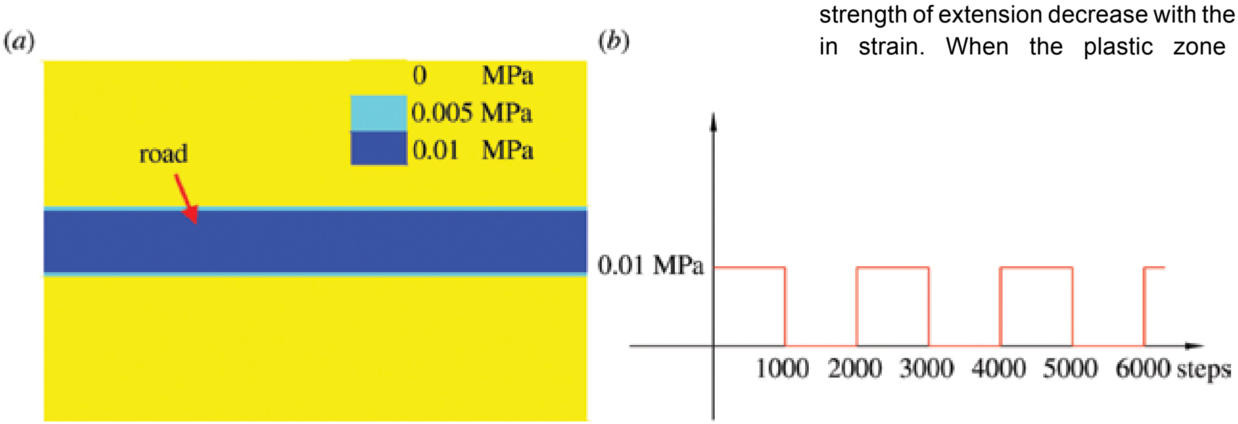

Figure 6: Dynamic load applied on the road. (a) Dynamic load on the street and (b) dynamic load path.

The pillar under the road outside was simulated to be 1.0 m, 1.3 m and 1.5 m wide, and the pillar under the road was simulated to be 1.4 m, 1.7 m and 2.0 m wide, respectively. According to the size of the residual coal and cantilever excavator, the width of each panel (sum of the pillar width and excavated width) was set to 5.5 m. Thus, the excavation width under the road outside was simulated to be 4.5 m, 4.2 m and 4 m, and the excavation width under the road was simulated to be 4.1 m, 3.8 m and 3.5 m, respectively. Both the ends and the bottom of the model were fixed. The parameters of all rocks are shown in **Table 1**, and the diagram of the model is shown in **Figure 5**.

To analyse the effect of the coal trucks on the road on the stability of the coal pillars, a dynamic load was applied in the model, as shown in **Figure 6**. The load is assumed to be 0.01 MPa intervals of 1000 steps, since the load of a truck loaded with coal is approximately 0.01 MPa, and the dynamic balancing steps are 1000.

Simulation results analysis

Because of the shallow depth of the coal seam, the vertical stress of the coal pillar is normally less than its compressive strength, so the failure mode of the coal pillar conforms to Mohr-Coulomb yielding criteria. When plastic failure occurs in the coal pillar, the ultimate bearing capacity of the coal pillar declines. Thus, the strain softening model is used for the pillar, where the internal friction angle, cohesion and strength of extension decrease with the increase in strain. When the plastic zone spreads

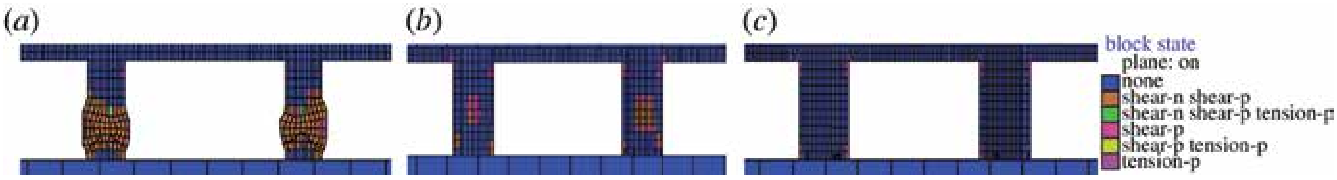

Figure 7: Plastic zone of the coal pillar under the road outside. (a) 1.0 m, (b) 1.3 m, and (c) 1.5 m.

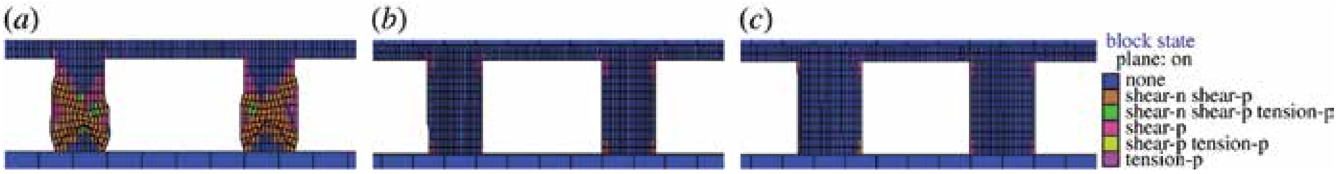

Figure 8: Plastic zone of the coal pillar under the road. (a) 1.4 m, (b) 1.7 m, and (c) 2.0 m.

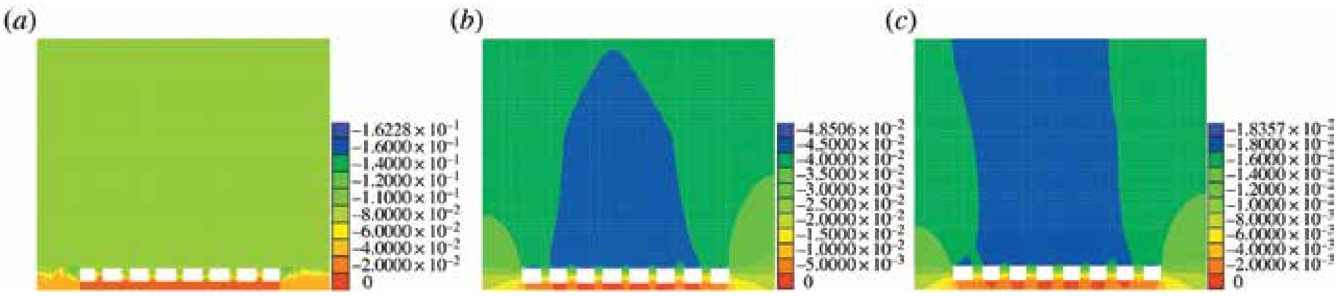

Figure 9: Vertical displacement of the road section with different pillar widths. (a) 1.4 m wide coal pillar, (b) 1.7 m wide coal pillar, and (c) 2.0 m wide coal pillar.

throughout the coal pillar, the ultimate bearing capacity will significantly decline and make the coal pillar unstable. The diagrams of plastic zones only show the partial cross-sections of coal pillars, and each grid that corresponds to the actual length is 0.1 m. **Figures 7** and **8** show the plastic zone development of the coal pillar outside and under the road, respectively, and **Figure 9** shows the vertical displacement cloud diagram of the road cross-section.

As shown in **Figure 7**, when the coal pillar outside the highway is 1.0 m, the plastic zone has spread throughout the entire pillar, and the coal pillar collapses because of its instability, which cannot satisfy the mining safety requirement; comparing the condition of 1.0 m, all plastic zones of 1.3 and 1.5 m decreased. Due to the corners of the pillar experiencing the highest stress concentration, the plastic zone priority occurs in this area. Thus, **Figure 7c** with stress concentrated only at the corners shows that the bearing capacity can satisfy the requirement. While in **Figure 7b**, after the corners of the pillar yielded, the plastic zone then occurred in the core of pillar, due to the brittleness of coal; when the stress reaches a certain strength, tensile failure will occur in the middle part, but the plastic penetration area did not appear, so both zones maintained the stability. Therefore, the width of the coal pillar under the road outside is 1.3 m due to the high resource recovery ratio.

As shown in **Figure 8**, the plastic zone spreads throughout the entire coal pillar when the width under the road was 1.4 m. As shown in **Figure 9a**, the bearing capacity of the coal pillar significantly decreased and caused the shrinkage to reach 16 cm, which is significantly more than the others. This result demonstrates that the coal pillar collapsed due to unstability. When the width increases to 1.7 m, the plastic zone is mainly distributed in the corners, but the scope is relatively small and does not spread throughout the whole pillar. The bearing capacity can satisfy the requirement. Meanwhile, the maximum shrinkage is 4.8 cm in the cross-section of the highway, which satisfies the engineering requirement. When the width reaches 2.0 m, the scope plastic zone further declines, and the maximum shrinkage is only 1.8 cm, which indicates that the coal pillar is stable. Comparing both conditions and considering the recovery rates, we obtain that the width of the pillar under the road is 1.7 m, which is 0.2 m larger on both sides than the pillar under the road outside.

The simulation result indicates that the reasonable widths of the pillar outside and under the road are 1.3 m and 1.7 m, respectively, which can satisfy the requirements for mining safety and a high recovery ratio. Therefore, this condition was considered the reasonable width of a coal pillar after comprehensive comparison.

Empirical analysis

Underground pillars are mostly square and rectangular, whereas highwall mining pillars are long and narrow because they are formed after driving parallel entries in the seam from the highwall. These pillars are called web pillars. Several empirical coal pillar strength equations, which were developed for rectangular pillars, are modified for use with web pillars. However, for the rectangle pillar with a large aspect ratio, practice shows that the Mark-Bieniawski formula is the most suitable formula[27]:

Equation 1

$$S_p = S_I(0.64 + 0.36W/H)$$

where: S_P is the coal pillar strength, MPa; S_I is the *in-situ* coal strength, MPa; W is the coal pillar width, m; H is the mining height, m.

The tributary area method is useful for estimating the vertical stress on web and barrier pillars. The average vertical stress on the pillar is[28]

Equation 2

$$S_{WP} = \frac{S_V(W_{WP}+W_E)}{W_{WP}}$$

where: S_V is the *in situ* vertical stress, MPa; W_{WP} is the room coal pillar width, m; W_E is the highwall miner hole width, m.

The overburden depth may be taken as the maximum overburden depth on a highwall mining web pillar, which is notably conservative, or as a high average value as follows[25]:

Equation 3

$$D_{Design} = 0.75 \times D_{Max} + 0.25 \times D_{Min}$$

where: D_{Max} is the maximum overburden depth, m; D_{Min} is the minimum overburden depth, m.

Neglecting the stress carried by the pillars (i.e. assuming that they have all failed), we obtain the average vertical stress on a barrier pillar[29].

Equation 4

$$S_{BP} = \frac{S_V(W_{PN}+W_{BP})}{W_{BP}}$$

where: W_{PN} is the panel width, m; W_{BP} is the barrier pillar width, m.

According to the numerical simulation result, the width of the barrier coal pillar W_{BP} is 8 m, the width of the web coal pillar under the road W_{WPI} is 1.7 m, and the width outside the road W_{WPO} is 1.3 m. The depths of the coal seam inside and outside the road were calculated using **Equation 3** ($D_{DesignI}$ = 60 m, $D_{DesignO}$ = 45 m). The panel width W_{PN} is 60 m, and the rock density r is 24 000 N m^{-3}, so the *in-situ* vertical stress S_{VI} is 1.44 MPa, and S_{VO} is 1.08 MPa. The strength of the *in-situ* coal is 6.89 MPa according to the mechanics experiment in the laboratory. Therefore, the strength of the barrier pillar is 12.91 MPa, the strength of the coal pillar under the road is 5.82 MPa, and the strength of the coal pillar outside the road is 5.48 MPa. The vertical stresses of the coal seam inside and outside the road are 4.7 MPa and 4.3 MPa according to **Equation 2**, and S_{BP} is 12.24 MPa according to **Equation 4**, all of which are less than the corresponding strength of the coal pillar calculated according to **Equation 1** and similar to the strength of the coal pillar. The result shows that the width of the coal pillar obtained from the numerical simulation can satisfy the support requirement and recover more resources. However, it also shows that the empirical formulae have a large surplus coefficient, and the effect of the dynamic load of the road is not considered. Thus, although the pillar width determination method in this paper cannot be directly applied to other geological conditions, the pillar width of other geological conditions can be obtained using the method in this paper. Moreover, the numerical simulation can be used to obtain the reasonable width coal pillar with different road cross-section widths, pillar strengths, and overburden strata thicknesses. Thus, we can obtain a normalized empirical formula for the optimal width of coal pillars by setting a safety factor according to the numerical simulation.

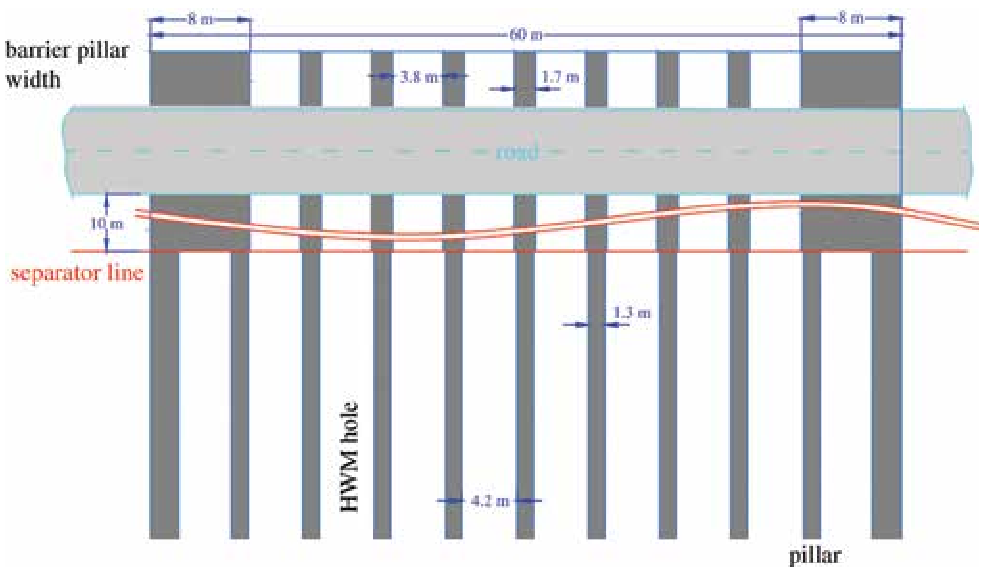

Figure 10: Highwall coal pillar width design.

According to the numerical simulation and theoretical analysis results, the design width of the coal pillar in the highwall is shown in **Figure 10**.

ROADWAY LAYOUT AND MINING DESIGN

Roadway layout

For the highwall mining working faces, the gateway and pillar method is used to exploit the 2# coal seam with the characteristics of the roadway cross-section in **Figure 11**.

In normal conditions, the roadway is 4.2 m wide, but it becomes 3.8 m wide when it develops under the road, and the heights are both 3.0 m. Because the cantilever excavator is remote controlled to mine, there is no need to excavate the roadway for humans. Although there is no

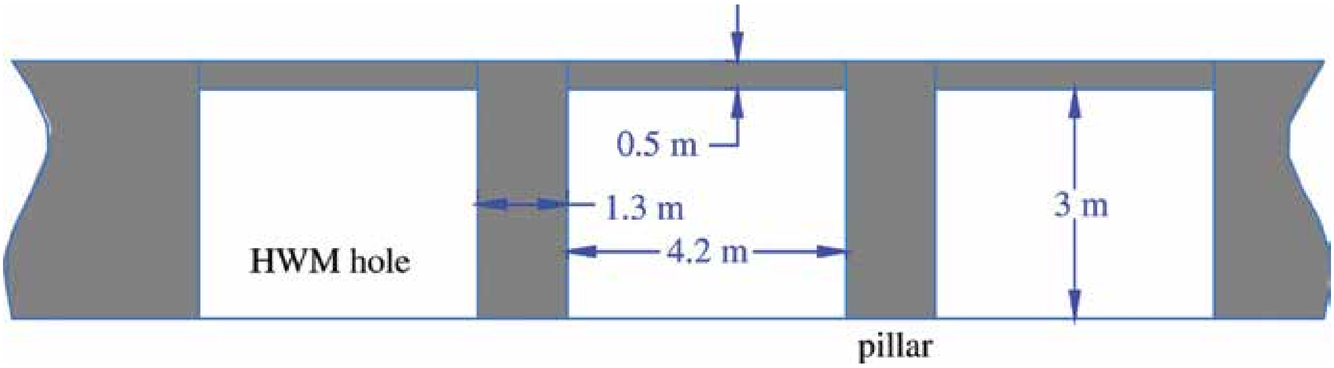

Figure 11: Excavation and residual pillar width design.

permanent support in the roadway, the 0.5 m upper coal is treated as the temporary support to prevent the roof fall from damaging the devices. On the roadway, the protective shed is set before driving. The protective sheds are made from #16 joist steel with a 6-mm steel plate, and the support is made of a 114 mm steel tube. There are 8 sheds in total, which are 4.5 m long, 1.5 m wide and 3.3-3.7 m high.

Mining design

The HTM working face is located at the north western boundary of the coal mine, which is seated in the 2# coal seam with 390 m along the strike direction and 60 m along the dip direction. The mining design is shown in **Figure 12**. This area cannot be exploited by the open-pit mining method. To improve the recovery ratio, the coal mine purchases an EBH132 cantilever excavator, which was produced and developed by NHIG, to perform the highwall mining, as shown in **Figure 12**. After the coal resource in the end slope has been mined, the inner spoil dump backfills in a timely manner.

The EBH132 cantilever excavator is used for the mining and coaling, and the 650 belt conveyer is used for transportation. The main characteristics and working principles of the EBH-132 cantilever excavator are as follows:

Man-machine separation and high safety. The system consists of a cantilever excavator and a remote-control room, which is comprised of a KXB1-200/1140 (660) ZE mining flameproof electrical control box, a cantilever excavator electrical control box and a control panel. The system has the remote cable control function and achieves man-machine separation. When mining, the workers can operate the machine in the control room, as shown in figure

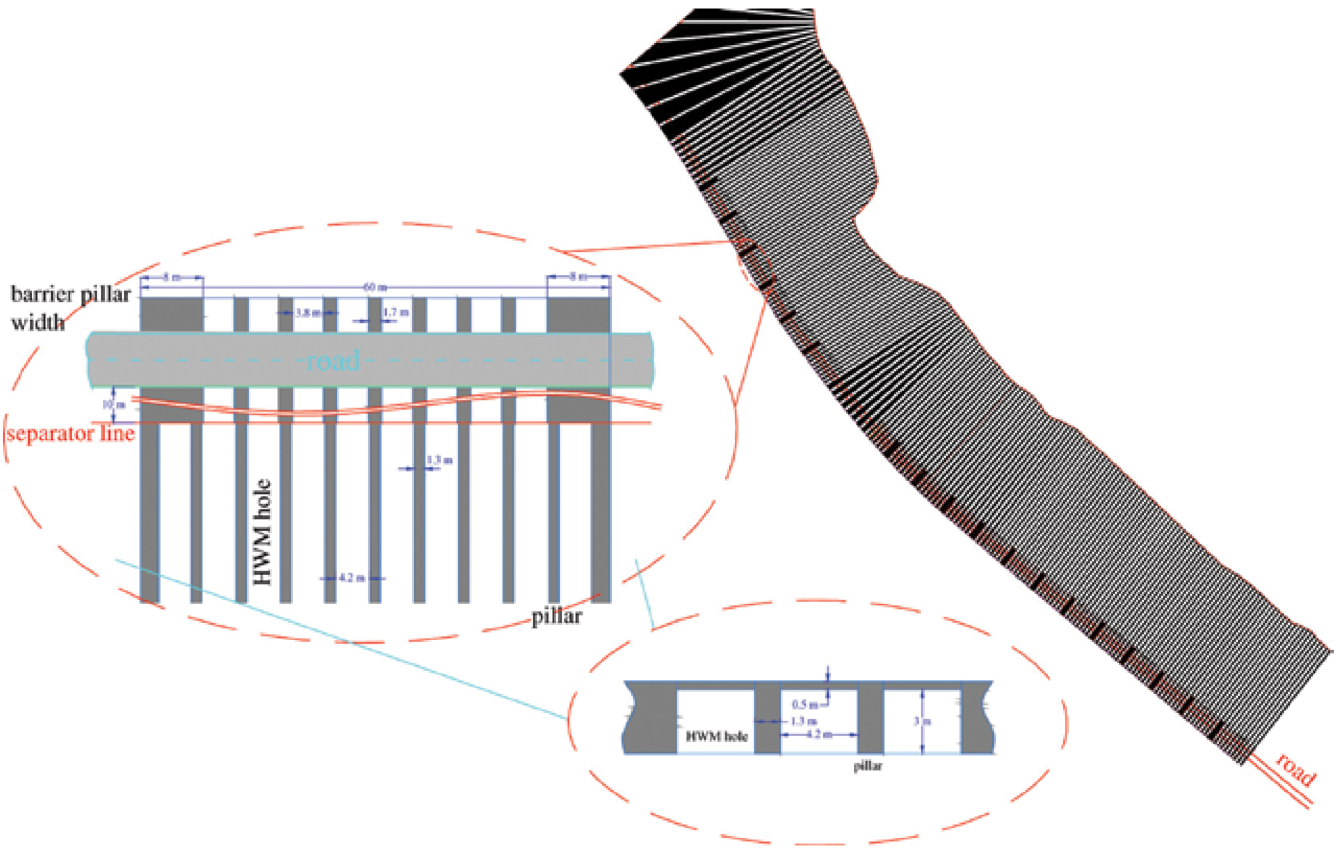

Figure 12: Highwall mining design planning map.

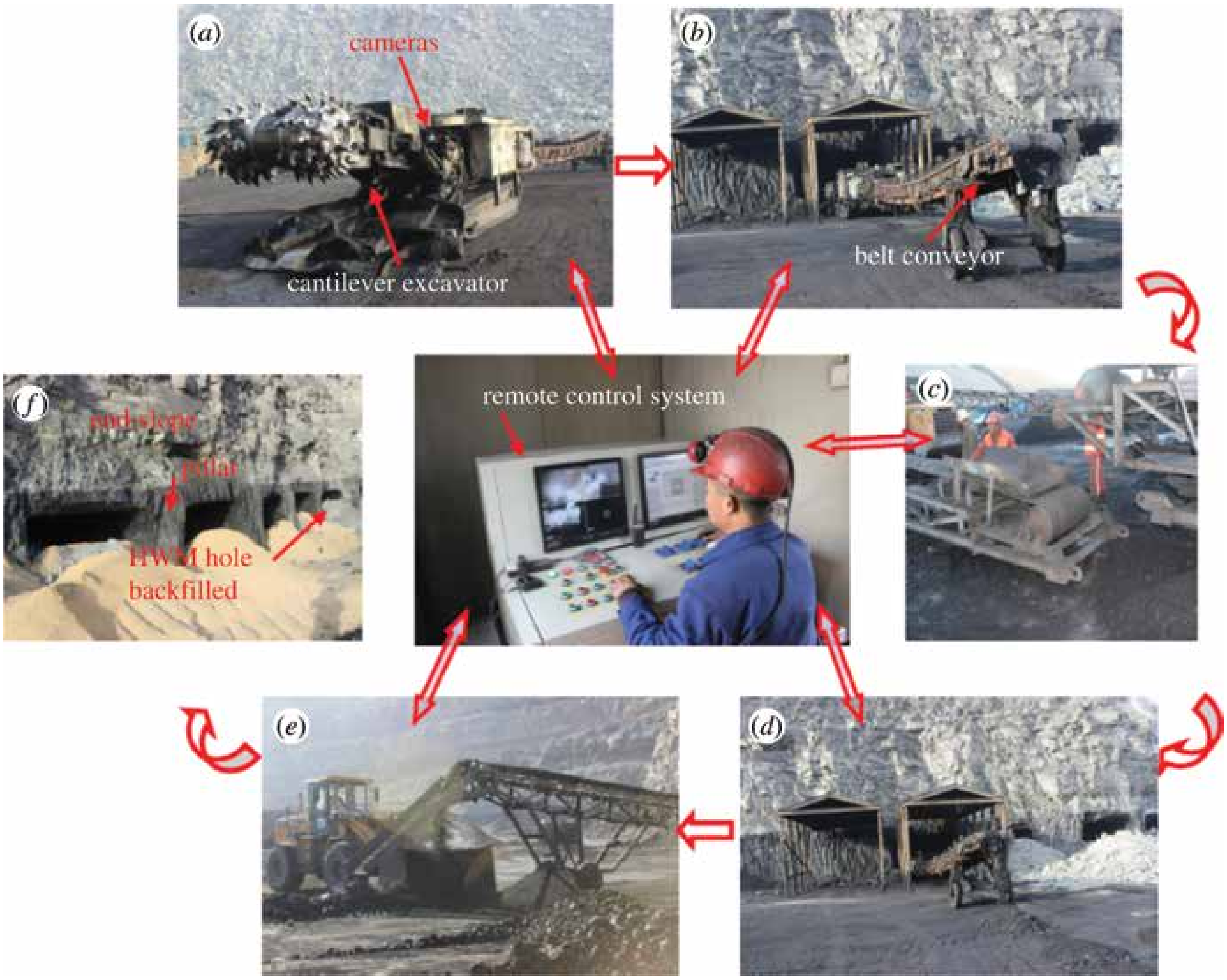

Figure 13: Simple mining process. (a) Preparatory work, (b) mining and loading, (c) installing the belt conveyer, (d) transporting by belt conveyer, (e) casting by loader and (f) HWM hole backfilled.

13. The cantilever excavator enters the highwall working face to finish mining to reduce the possibility of casualties and achieve safe mining.

Visualized operations and remote real-time control cantilever excavator. Three cameras are installed in the cantilever excavator (the back one is added by the coal mine, as shown in figure 13). Using video imaging, the operator can see the projection of the seam-rock interface and guide the progress of the continuous miner. The work image of the cantilever excavator is transmitted to the control room through the signal transmission equipment, which achieves a real-time display of the working condition of each system using the screen on the control panel. It can satisfy the requirement for the controlling protection, display, illuminating and signal receiving functions of the cantilever excavator.

Simple mining process. The main process is: preparation work → cutting → coaling → installing the belt conveyer → transporting by belt conveyer → casting by loader → digging to the design location and coming out of the roadway → equipment retracement → overhauling, as shown in **Figure 13**. In the process of cutting, the first step is to adjust the cutting head to the right bottom corner and cut into the coal body. The cutting sequence is from right to left and subsequently from the bottom to the upper position. When the cutting head reaches the roof, the cantilever excavator goes backward to cut the top coal, which makes the roof smooth. When the process completes, we adjust the cutting head to the bottom and start another circulation. The details are shown in **Figure 14**.

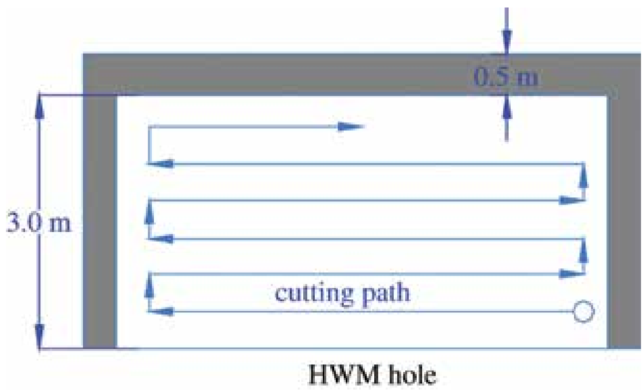

Figure 14: Excavator cutting path.

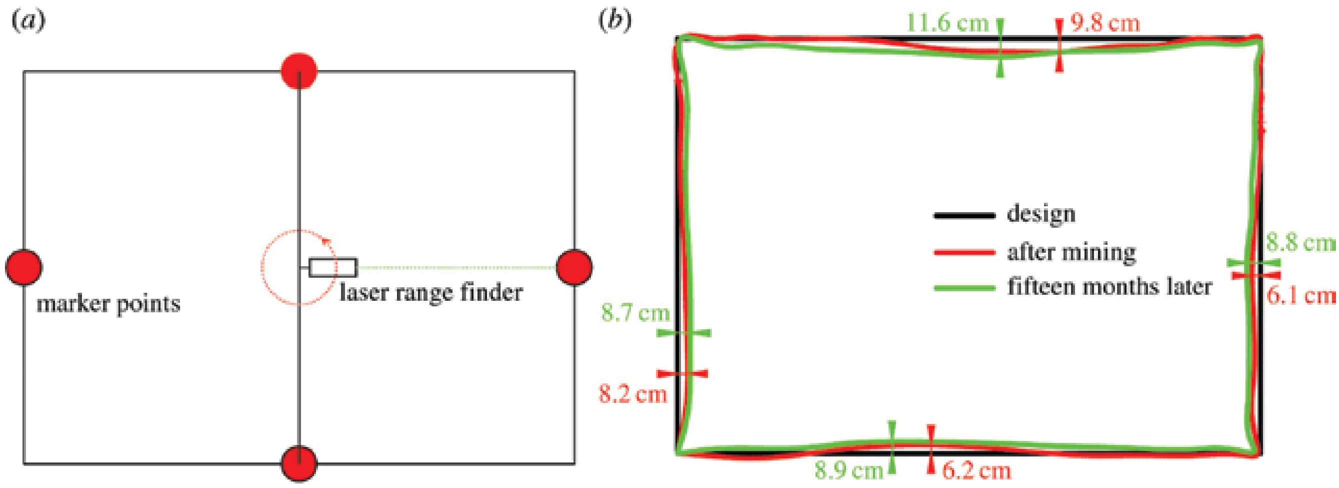

Figure 15: (a) The monitoring method and (b) the deformation monitoring results of the exposed roadway section.

FIELD APPLICATION EFFECT ANALYSES

The EBH132 cantilever excavator is used to exploit the end-wall residual coal resource in open-pit coal mines, and in comparison to other methods it is characterized by a high degree of mechanization, fewer workers, less labour density, good working environment and high efficiency. Regarding the work mode, two driving teams are required every day, and every team is composed of a team leader, a machine unit driver, a belt conveyer driver, a loading-machine runner and an electrician, totalling 10 workers who are required every day. After the team finishes mining the designed roadway and draws back the equipment, the machine should be overhauled. The length of driving is 10 m every day, and the work efficiency of each worker is 17.6 tons per day.

Within two years of mining time, 710 m of the 2# coal seam was excavated in the strike direction, the coal reserves were 435 thousand tons, and the recoverable reserves were 291 thousand tons, except for the security coal pillar of 10-kVA high-voltage power lines and 35-kVA electrical substation. The highwall mining produced 197 thousand tons of coal, and the recovery ratio was above 67%.

To further verify the stability of the selected coal pillars, the roadway section scanning analyses were carried out in the roadway under the road after highwall mining. The test method was shown in figure 15a. However, due to the fact that the HWM hole will be backfilled after highwall mining (figure 13f), it is impossible to put the equipment into the roadway for measurement. Therefore, we can only monitor the deformation of the exposed roadway section; the monitoring results are shown in **Figure 15b**. Within 15 months of mining, the maximum deformation of the roadway section is only 2.7 cm, which demonstrates that the design width of the coal pillar is sufficient to achieve reasonable stability.

CONCLUSION

To ensure the stability of coal pillars and prevent rock fall accidents in the process of HWM, the paper calculated and analysed the reasonable width of the coal pillar using numerical simulations and the strength theory. Then, the reasonable coal pillar widths outside and under the road were determined to be 1.7 m and 1.3 m, respectively, which can efficiently ensure the safety of ground facilities.

The EBH132 cantilever excavator has many advantages such as man-machine separation visualized operations and remote real-time control, which can achieve highwall mining with remote control. Hence, the safe, efficient and high-recovery-ratio highwall mining was achieved for the residual coal resources of an open-cast mine.

Regarding the open-cast mine, the remote-control cantilever excavator was used to develop the unmanned highwall mining, which created a better working environment and a higher efficiency for the single worker. The recovery ratio was over 67%, and significant technical and economic results were achieved. During the mining process, there was no collapse of the coal pillars or roof caving accident, which demonstrates that the design width of coal pillar is reasonable, and the use of the residual coal resource in the end-wall area is efficient for the open-pit coal mine.

DATA ACCESSIBILITY

Our data are from the field case. The numerical parameters are based on the geological data of a coal reservoir in Inner Mongolia Autonomous Province, China, which are listed in **Table 1**. The data of the field application effect are described in detail in the field application section (§5) of this manuscript.

AUTHORS

Fangtian Wang and Cun Zhang

AUTHORS' CONTRIBUTIONS

F.W. performed the mining parameter design, participated in the field test, and drafted the manuscript. C.Z. conducted the numerical simulation and conceived of, designed, and coordinated the study. All authors gave their final approval for publication.

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Development of automation in US longwall mining

Image courtesy of Famur

Before we start talking about automation in longwall mining, the word "automation" needs to be clarified. Adoption of automation and a longwall system does not guarantee a reduction in face personnel. The highest level automation of longwall systems today still requires 4–10 crew members at various mines. Obviously mine/crew practices and the mining and geological conditions still dictate the manpower needs at the longwall face. Managing Editor Trevor Barratt looks at how the USA has progressed to become a leader in the automation of longwalls over the last 50 years. ***Has it really reduced manpower?*** **The following article may give some answers to the question.**

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 the domestic market. Automation began with the development of electrohydraulic shields in 1984 and continue to this present day. Introduction of first generation semi-automated longwall system occurred in 1995 and step-to-step improvement continues following the development of sensor technology and internet of things (IOT). Since then, 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 the longwall system. The automation development of a longwall system is divided into three stages: shearer-initiated-shield-advance (SISA), semi-automated longwall system, and remote-control shearer.

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 centre 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 1**.

Since longwall mining consists of three major machines, shearer, AFC, and shields, there are two types of "automation", "automation" for each machine and

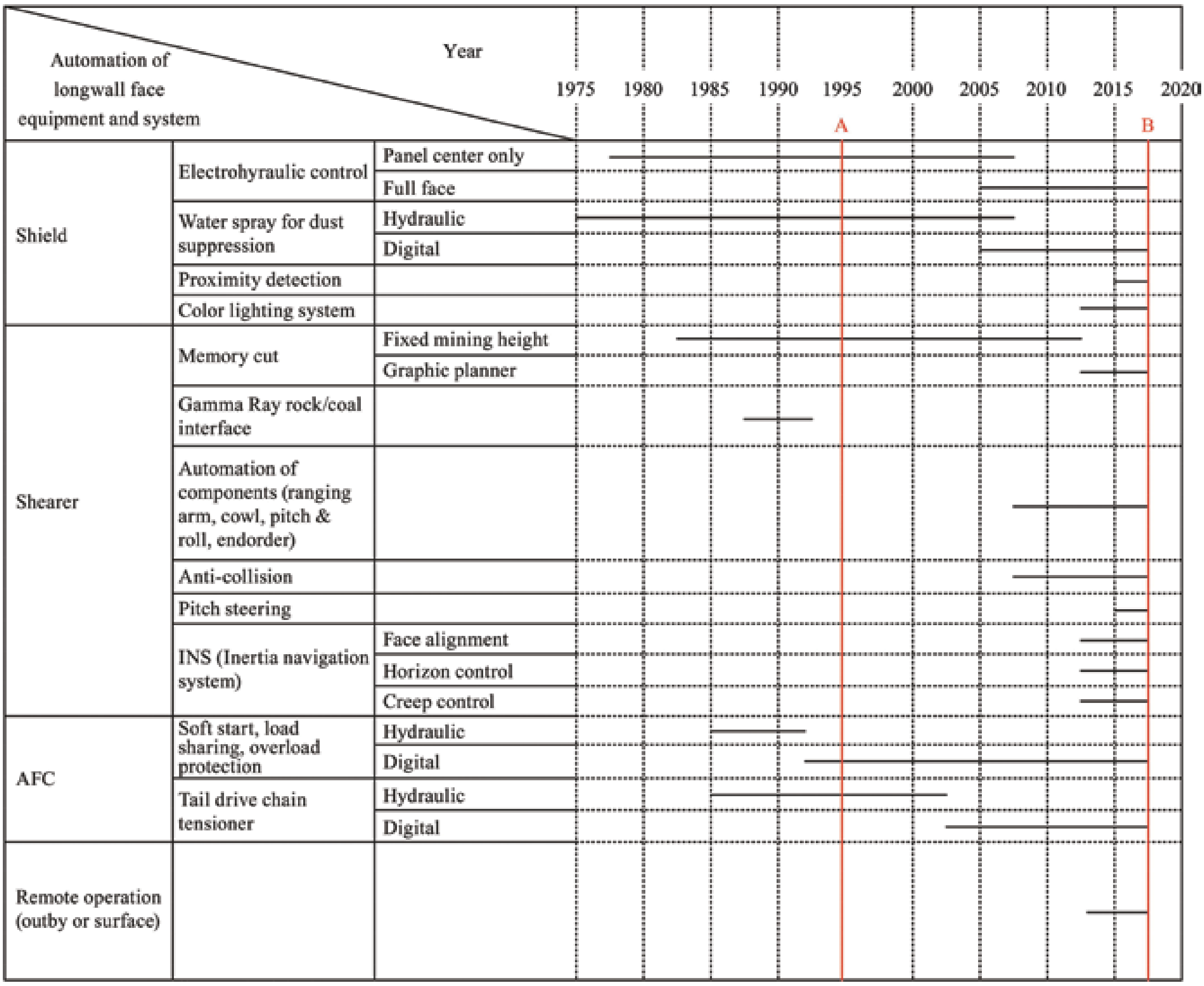

Figure 1: Development Chronology of Automation of Longwall Face Equipment and system. Automation of longwall system can be found by drawing a vertical line from the year of interest. For example, the 1995 Cumberland Mine system (A line) would have shield with electrohydraulic and hydraulic water spray, shearer with memory cut and AFC with hydraulic soft start and tail drive tensioner. B line represents state-of-the-art automation longwall system.

"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 emphasized 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 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 exceptionally reliable and gained industry's full acceptance.

A fully operational batch control electrohydraulic system that requires only two pushbuttons 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).

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 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.

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.

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. 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 it so far .

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 especially 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) shearer-initiated-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 2**) 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

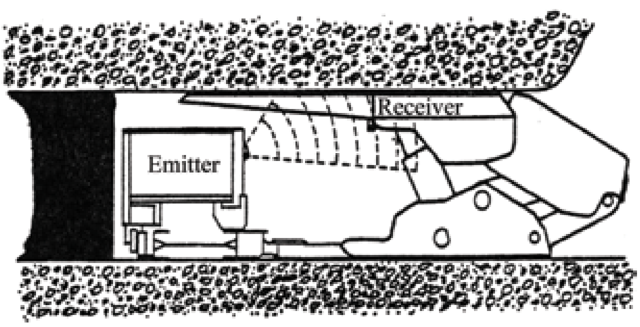

Figure 2: Principle of infrared sensor for shearer's location

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.

The system worked well for the whole panel width when the shearer employed the uni-direction cutting method. When bi-direction 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: ASA- pitch & 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 running 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

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 check-up. A surface control centre 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 sophisticated 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 centre.

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 centre away from the face.

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.

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 continues to 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 extremely 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.

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Perfect for coal storage

Hard coal is temporarily stored in stockpiles and then continuously fed to be processed, as needed. The design of the depositories must ensure constant filling and reliable emptying. The required capacity is determined based on the incoming and outgoing conveying flow. Different stacking and reclaiming options as well as various layouts for the stockpiles are also needed. BEUMER Group provides the engineering for handling stockpiles and offers the required components to coal mine operators, such as stackers and reclaimers.

BEUMER Group offers a comprehensive product and system solutions portfolio to customers from the coal mining industry. Conveying technology includes closed Pipe Conveyors and open troughed belt conveyors that can be adjusted to the respective situation. As a system supplier, BEUMER Group also provides extensive know-how and the necessary components for storing hard coal, e.g. stackers and bridge reclaimers. "We support our customers immediately from the design phase," says Andrea Prevedello, System Technology Global Sales Director, BEUMER Group, Germany. Drone technology is used more and more frequently during project planning, implementation and documentation to optimise the design phase. The recorded aerial photos are rectified with regard to their perspective and evaluated photogrammetrically. The software calculates a point cloud in order to generate 3D models from the two-dimensional views, i.e. digital terrain models. Stockpiles can now be greenfield and brownfield developments.

"We have some major customers with very interesting projects in this sector," explains Andrea Prevedello. This most certainly includes Prairie Eagle Mine in Illinois, the largest coal mine of Knight Hawk Coal. This is one of the most efficient underground mining plants in the US. They produce approximately five million tons of coal annually, of which more than 80% is processed and delivered in Prairie Eagle.

Management was looking for a more sustainable operating solution. "We provided an overland conveyor that transports the coal from the mine to the main processing plant," describes Andrea Prevedello. "Our conveyor helps the company to considerably reduce its ecological footprint. With this technology, Knight Hawk can significantly reduce its long-term environmental impact compared to using truck transportation." BEUMER Group not only supplied the conveying solution. As a system supplier, the company also supported the mining group in building a stockpile for hard coal. "The requirements for storing coal are obviously very different from other materials," explains Andrea Prevedello. Some of the important requirements are changing if the stockpile is covered and if explosion-proof specific equipment is needed. Hard coal is very susceptible to spontaneous combustion, which is why the height of the stockpile must be in certain cases limited.

CIRCULAR OR LONGITUDINAL STOCKPILE?

Depending on the customer, stockpile dimensions and design can vary. Two layouts are generally available: circular and longitudinal. "Their dimensioning and design depend on the purpose of the stockpile," describes

Andrea Prevedello. Space availability and possible future expansions are also critical factors. The application must also be considered: does the customer want to store the bulk material temporarily, then continuously feed it for further processing, like Knight Hawk? "Then longitudinal stockpiles are your best choice," says Andrea Prevedello. They can also be extended, if necessary. The irregular flow of bulk material arrives at the stockpile and can then be continuously introduced to the process. Circular stockpiles are frequently used for other bulk materials as well, e.g. for limestone, clay. This is particularly used by cement manufacturers and power plant operators.

The Prairie Eagle Mine in Illinois is the largest coal mine of Knight Hawk Coal. They produce approximately five million tons of coal annually, of which more than 80 % is processed and delivered in Prairie Eagle.

But back to coal. Once the layout of the stockpile has been decided on, the next task is to stack the bulk material efficiently. BEUMER Group also provided these necessary components such as the stacker. "Depending on its mobility, the systems can be categorised into three groups," explains Andrea Prevedello. The stacker can be stationary, travel on rails or be circular with endless movement. If the machine is circular with endless movement, it is positioned on a column in the centre of the stockpile. Over a conveyor bridge installed above the stockpile, the material is transported directly into the axis of rotation of the stacker and from there distributed centrally. Depending on the stacking method, the boom conveyor can be fixed or it can be lifted and tilted.

IT DEPENDS ON THE METHOD

The stacking method of choice depends on whether the bulk material is only temporarily stored or if it also needs to be blended. "For simple stockpiling without blending, we provided with the simple 'cone shell method'," describes Andrea Prevedello. The stacker only moves up and down, not slew. The stacker design can be more simpler. This method works for longitudinal as well as circular stockpiles. For belnding the bulk material however, the Chevron method can be used. The boom of the stacker starts in its lowest position. The first row is deposited in the centre of the stockpile, the next rows are layered on it. In longitudinal stockpiles, the stacker usually moves in a tilting and slewing motion, in circular stockpiles the stacker moves in a circulating and luffing motion.

EFFICIENT COAL MINING

"The perfect system solution is always an optimal relation between stacker and reclaimer," explains Andrea Prevedello. Reclaimers such as side reclaimers or bucket

The overland conveyor transports the coal from the mine to the main processing plant.

This conveying solution helps the company to reduce its ecological footprint because, compared to transport by truck, the environmental impact can be significantly reduced in the long term.

BEUMER Group provide the suitable stacker for stacking the coal

BEUMER Group offers a comprehensive product and system solutions portfolio to customers from the coal mining industry.

wheel remove the material, as necessary. The best option for the customer depends again on the stockpiling task at end. Side reclaimers work for both types of stockpiles, longitudinal or circular. The bulk material can be reclaimed from the front or the side. When reclaiming from the side, scraper chains move the material on a belt conveyor. Front reclaiming usually uses a rake that in small side-to-side movements pushes the material on a scraper chain to be transported further to the conveyor. The advantage is that the bulk material is reclaimed from the entire cross-sectional area. Bucket wheel are used in particular when the bulk material, especially in large quantities, needs to be blended.

Each operator has their own very specific requirements on the stockpile and stockyard machines. This is shown in a project that the BEUMER engineers are currently implementing for a customer in the energy industry. The order includes the delivery of several conveyors, including Pipe Conveyors, and a ship loader. The challenge: “On the ground where we will install our solution, there can be violent gusts of wind,” reports Andrea Prevedello. “That’s why we pay special attention to the dimensioning of the steel structure.” The system provider will thus be able to hand over a tailor-made system to the customer, with investment expenditure tailored precisely to him. The expected commissioning is scheduled for the third quarter of 2020.


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